Vladimir Litvinenko *Editor*

# XVIII International Coal Preparation Congress

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# Foreword

Dear Ladies and Gentlemen,

In the last 15 years in Russia, there has been a rethinking of attitudes towards the coal industry. In Soviet Union it was one of the primary branches of industry and there was a ministry responsible for this, a network of research and engineering institutes and companies developed infrastructure of specialized machine-building factories producing equipment for these aims. In the late XX century the situation changed. Liquid and gaseous hydrocarbons have significantly changed the raw materials structure of energy producing in our country. For quite a long period of time, coal manufacturers had certain difficulties.

It is pleasant to point out that now the situation is changing substantially. In recent years coal has steadily returned to the structure of energy balance in Russia; the state goes through the effort of supporting research in the field of coal processing and coal preparation. Russian scientists have achieved serious results in the areas of new kinds of treatment, diversification of the use of coal, that is its use not only for producing energy but for new products.

Papers published in the proceedings indicate that residents in cities have no reason to worry about the problems caused by coal combustion for power generation, that they will breathe clean air, that we have a respected place in what is called 'clean coal energy', that we supply with new products those who work in high technologies and in particular that we can extract rare earth elements from the coal feedstock.

I would like to thank all authors of scientific papers, all participants of the XVIII International Coal Preparation Congress. I am sure that the Congress will work in warm atmosphere and will be effective and fruitful.

Leonid Weisberg
Chairperson of the XVIII International Coal Preparation Congress

# Preface

Dear Congress participants,

Changes in economic and market conditions of mineral raw materials in recent years have greatly increased demands on the efficiency of mining production. This is certainly true of the coal industry. World coal consumption is growing faster than other types of fuel and in the past year it exceeded 7.6 billion tons. Coal extraction and processing technology are continuously evolving, becoming more economical and environmentally friendly. "Clean coal" technology is becoming increasingly popular. Coal chemistry, production of new materials and pharmacology are now added to the traditional use areas—power industry and metallurgy. The leading role in the development of new areas of coal use belongs to preparation technology and advanced coal processing. Hi-tech modern technology and the increasing international demand for its effectiveness and efficiency put completely new goals for the University. Our main task is to develop a new generation of workforce capacity and research in line with global trends in the development of science and technology to address critical industry issues.

Today Russia, like the rest of the world faces rapid and profound changes affecting all spheres of life. The defining feature of modern era has been a rapid development of high technology, intellectual capital being its main asset and resource. The dynamics of scientific and technological development requires activation of University research activities. The University must be a generator of ideas to meet the needs of the economy and national development. Due to the high intellectual potential, University expert mission becomes more and more called for and is capable of providing professional assessment and building science-based predictions in various fields.

Coal industry, as well as the whole fuel and energy sector of the global economy is growing fast. Global multinational energy companies are less likely to be under state influence and will soon become the main mechanism for the rapid spread of technologies based on new knowledge. Mineral resources will have an even greater impact on the stability of the economies of many countries. Current progress in the technology of coal-based gas synthesis is not just a change in the traditional energy markets, but the emergence of new products of direct consumption, obtained from coal, such as synthetic fuels, chemicals and agrochemical products. All this requires a revision of the value of coal in the modern world economy.

I believe that the Congress ICPC-2016 made a great contribution in promoting new knowledge, breakthrough technologies in a field of coal preparation and time-sensitive approach to resource conservation of the world.

Vladimir Litvinenko
Rector of Saint-Petersburg Mining University

# Welcome Speech

Dear participants,

For the first time in 60 years the Russian Federation is hosting the International Coal Preparation Congress. For the coal industry the Congress is an event of true international significance and it is especially pleasant for me that it is held with the full support of the Ministry of Energy.

Coal industry is an exceptional sector for Russia. The significance of the XVIII International Coal Preparation Congress is confirmed by the establishing in the country of a National Organizing Committee which includes the leaders of all major coal companies, and also representatives of scientific organizations and universities.

In 2014 the Ministry of Energy worked out and approved a strategic program for development of coal industry up to 2030. The program is designed to meet stable and reliable internal demands of Russia for coal and coal products and solution of social problems, and it takes into account the development of the internal market due to advanced coal processing, coal chemistry products and comprehensive use of coal.

The main goal of the Congress is to assist scientific and technological cooperation for the benefit of achieving progress in coal preparation and to find solutions of environmental issues, directly correlating with the strategic program of development of the coal industry, which once more underlines the importance of the Congress for our country.

I would like to point out that, according to the traditions of the Congress, an exhibition on coal preparation equipment will be held. This is a unique opportunity for all companies and organizations working in the mining sector to demonstrate their scientific and technical achievements, to strengthen existing business contacts, and to develop new ones.

I wish all the best to the participants of the Congress!

Alexander Novak

# International Organizing Committee

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# Sponsors

Siberian Coal Energy Company (SUEK)

SUEK is one of the world's leading coal mining companies and Russia's largest coal producing corporation. It provides about 30 % of power-generating coal in the internal market and about 20 % of Russian coal exports. SUEK is the only Russian coal company, which is in the list of world's top ten leading coal market companies. It has affiliates and subsidiaries in Krasnoyarsk oblast, Primorsky krai, and Khabarovsk krai, Irkutsk, Chita oblast, Kemerovo oblast, Buryatia and Khakassia. SUEK is the second largest shareholder of several Siberian and Far-Eastern power companies after RAO UES.

En+ Group

"En+ group" is a leading Russia-based natural resources industrial group, which unites companies working in the areas of metallurgy, mining, energy and other business related fields. Currently the company total stock of power-generating and coking coal is more than 4 billion tons and annual output is over 50 billion tons. The key coal mining regions are East Siberia and Kazakhstan which are situated close to developing Asian markets.

Severstal Resources

Severstal

Severstal Resources is one of the Russia's largest mining companies forming the mining division Severstal. Severstal Resources is comprised of two iron ore mining plants "Karelsky okatysh" and "Olkon", ferroniobium manufacturer "Stalmag" (Krasnoyarsky krai), coal mining company PBS Coals (USA), geological exploration project Putu Range (Liberia), 100 % of Nkout (Cameroon), gold division (includes projects in Liberia, Ndablama, Weaju, Goldoja, Silver Hills), 22 % of diamond producer Stellar Diamonds Plc shares, mining company design enterprise "SPb-Giproshakht", and a significant part of gold mining sector which assets are located in the Eastern part of the RF, Kazakhstan, Burkina Faso and Guinea.

Mechel Mining

MECHEL

Mechel Mining is a leading Russian coking coal producer as well as one of the world's major coking coal concentrate producers. The Group controls 25 % of Russia's coking coal washing facilities. In 2013 the company produced 27.5 million tons of coal.

Siberian Business Union

ХОЛДИНГОВАЯ КОМПАНИЯ

СДС

СИБИРСКИЙ ДЕЛОВОЙ СОЮЗ

Siberian Business Union is a holding company employing about 50 thousand people of enterprises situated in Kemerovskaya oblast, Altaysky krai and other regions of Russia and the CIS. Its companies are active in coal mining and processing, energy and agrobusiness sectors including machine building, car-repairing and chemical plants, railway transportation and construction companies.

EVRAZ

EVRAZ is a vertically integrated steel and mining business with operations in the Russian Federation, Ukraine, the USA, Canada, the Czech Republic, Italy, Kazakhstan and South Africa. A significant portion of the company's internal consumption of iron ore and coking coal is covered by its mining operations. The Group is listed on the London Stock Exchange and is a constituent of the FTSE 250. EVRAZ employs approximately 100,000 people.

UK Kuzbassrazrezugol

UK Kuzbassrazrezugol is the largest Russia-based coal company situated in Kemerovskaya oblast and the RF specializing in open-pit coal mining. Commercial coal reserves are over 2 billion tons. Annual fuel output is more than 45 million tons. The main coal ranks are D, DG, G, SS, T, KO, KS. About 50 % of produced coal is exported.

Tyva Energy Industrial Corporation (TEIC)

TEIC is a company implementing large-scale infrastructure projects in Krasnoyarsky krai, the Tyva Republic and Far-Eastern Federal District. Currently it carries out a complex of investment project involving development of Elegestskoe deposit of Ulug-Khemsky coal basin in the Tyva Republic and construction of mineral processing plant.

UK Kolmar

UK Kolmar (coal mining company) is a group uniting coal mining, trading and logistics companies employing more than 1400 people. It implements a range of large-scale projects in the field of coal industry such as construction of Inaglinsky coal complex (mine and processing plant) at Chulmakanskoe deposit and Denisovsky complex (mine and processing plant) at Denisovskoe deposit.

# Media Sponsors

«GORNYI ZHURNAL» Mining Journal

The oldest Russian monthly scientific, technical and industrial journal about all the issues of mining and extraction of minerals. Experience of mining ores, precious stones, building materials is highlighted in the journal. The most advanced ideas in the domain of mining as well as novel scientific development, new designs of mining machines, achievements in environmental protection are published in the journal. Published since 1825. Circulation—3000 copies.
The journal «Gornyi Zhurnal» is included into the international bibliographic and abstract database Scopus, Web of Science which is an instrument for tracking scientific articles citedness. Ministry of Education and Science of the Russian Federation considers the Scopus database as a criterion of estimation of activity efficiency of higher education institutes.

"UGOL" (Coal) Journal

It is a monthly scientific-technical and industrial-economic periodical. It was founded in October, 1925. It covers current situation and outlook for the development of coal industry, operation of enterprises, news of mining engineering and coal mining, preparation and usage technology, issues of labor and industrial safety, ecology, social topics, re-organization, economic data, and coal market. It publishes articles from regions, chronicles, materials of mining exhibitions, conferences, congresses, official documents, materials on history of mining, and best foreign practices.

"Marksheideriya i nedropolzovanie" Journal

"Marksheideriya i nedropolzovanie" (Surveying and Use of Natural Resources) is a journal for specialists working in the field of mining, geology, ecology; as well as services of mining companies, managers and senior officials of government bodies of federal and local levels; scientists and employees of higher education institutions, R&D and design institutes.

World Coal Journal

World Coal is a leading journal and web-site in the field of world coal industry. The journal covers economic, political, environmental and technical trends of coal industry from deposits to final customers.

Mineral Resources of Russia. Economics and Management Journal

The journal has been published since 1991 (six issues per year) by the informational publishing center of Geology and subsoil resources management of OOO "Geoinformmark". It publishes materials on all types of mineral resources: oil, gas, coal, ferrous, non-ferrous, precious and rare metals, agrochemical and mining-chemical raw materials, etc. The main sections of the journal are the following: the current state and development prospects of the petroleum and mining industries (geological exploration and raw material base); economics and management of mineral sector; legal support of natural recourses management; companies and projects; national and world mineral markets, and foreign experience and international cooperation.

"Dalniy Vostok" (Far East) Business Media

Publishing company Ltd. Business Media "Dalniy Vostok" releases unique information products for business. Such as Business newspaper "Our region—Far East"—the newspaper is a publication for executives and business people, Business directories in the leading sectors of the Far East and Siberia economics and Regional portal of Mining and Subsoil Use, includes more than 3,500 mineral deposits of the Far East.
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Webs: http://nedradv.ru, http://www.biznes-gazeta.ru

Bulletin Subsurface Use in Russia

It has been published since 1992 (24 issues per year) by the informational publishing center of Geology and subsoil resources management of OOO "Geoinformmark". It is an All-Russian and official publication of the Federal Agency for Mineral Resources (Rosnedra). It publishes invitations to tenders and auctions for the right to use a site of subsurface resources of all types of commercial minerals. The invitations contain the full text of a tender/auction notice approved by Rosnedra or its local agency, including tender/auction terms and conditions, a request to participate, details of an applicant, a prepayment contract, requisites, with the exception of information on the composition of an Auction Committee.


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# Part I
# Resources of the Coal Industry and Their Features

# Technological Progress Having Impact on Coal Demand Growth

V.S. Litvinenko, National
Saint-Petersburg Mining University,
DScTech, professor

**Abstract**

In the context of global cross-industry and interdisciplinary integration technological and techno-economic changes in related sectors of power resources will affect growing demand for coaI.

The following arguments are brought forward in this article:

factors affecting changes in the structure of power resources consumption, greening of economy and toughening of ecological requirements in power sector. Priority orientations in on quality improvement and enhancement of coal competitiveness as well as expansion of the scope of its application are defined. Advanced studies of technologies of deep-processing of coal for the near future are also specified.

**Key words**: coal, power resources, greening of economy, coal-to-gas conversion, synthesis gas, coal slurries.

## 1. Introduction

It is well known that coal is a unique energy resource combustion of which leads to emission of a large amount of $CO_2$. As a comparison, it is twice as much as when combusting natural gas on re-count to heat cost. Coal negative impact on the biosphere is the main problem during its usage as energy resource. As of today the rate of coal share reduction in power balance of many countries having the richest coal deposits

 
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depends, in general, on effectiveness of government policies focused on control over a pollutant waste and greenhouse gases emissions.

For example, energy balance structure impact analysis of APAC countries shows that during next 15 years overall energy resources consumption of regional countries will grow by almost 30%, and energy balance structure of these countries will shift towards ecologically large energy resources. Despite negative impact on the environment the coal will remain a key fuel type in many countries.

Considering huge world coal reserves humanity faces the task – how to convert coal into energy using effectively safe way in the conditions of escalating struggle against polluting emissions and greenhouse gases. In order to solve range of problems related with hazardous waste reduction when using the coal in traditional technologies providing coal phase transition from solid state into required energy fundamentally new approaches shall be used.

The recent developments in the area of power plants construction and operation using coal gasification technology are hold back mainly due to imperfection of international polluting emissions control system and economic methods. On the one hand it is ecologically advantageous, on the other hand it is more expensive, and that is the problem of the progress.

History shows that when humanity faces serious problems the inventions that can resolve the problem appear. For instance, the discovery made in Germany by the Director of Coal Research Institute F. Fisher in 1926 led to creation of hydrocarbon synthesis from carbon monoxide. It was fundamentally new technology that allowed hydrogen and carbon monoxide mixture obtaining in controlled proportions not only from coal but also from other carbon-containing raw materials.

Coal share increase during its conversion into the energy depends on technological progress ensuring introduction of new technologies that solve both ecological and economical tasks providing profitability in general.

The entire process chain including extraction, storage, transportation, processing and generation of coal requires complex of technical and technological measures providing protection against harmful effects on the biosphere.

Furthermore, scientific progress in the area of hydrocarbons obtainment from schist, methane from low calorie hydrocarbon-containing resources also has major impact on coal market demand.

In this regard coal shall be considered as an energy resource both from current technological paradigm prospective and considering general energy market condition and current international control over harmful emissions protection process using economic methods. When creating international conditions providing business motivation for environmentally friendly technologies implementation the role of coal can be significantly changed and its share can be increased in the following industries such as agrochemicals, biotechnology and others.

In this regard coal demand for the coming 15 years shall be considered subject to:

1. Factors affecting the change of primary energy consumption structure.

2. Greening the economy and tightening of environmental requirements for energy.

3. Coal quality and competitive performance improvement, expansion of its usage spheres.

4. Creation of innovative products and services having efficient impact on production performance, usage of fuel and

energy resources.

## 2. Factors affecting the change of primary energy consumption structure.

In the next 15 years total world energy consumption will increase by 30-40%. In addition to the above primary energy consumption structure change will occur.

Shift towards the usage of more environmentally friendly energy sources including natural gas will be specific feature in the dynamics of energy balance structure change of many countries, especially APAC countries.

Coal remains as the main fuel type in many countries; however its consumption growth is forecasted to be much less compared to other energy resources. In Asia-Pacific countries coal share will decrease in energy balance up to 40%, natural gas consumption growth – up to 14%. Nuclear energy share will increase up to 5%. Oil, biomass, hydroelectric power and renewable energy sources share will change slightly.

In many respects coal share reduction in region energy balance will be associated with government policies of these countries in the sphere of control over pollutant and greenhouse gases emissions.

Coal is an energy resource with special environmental properties. During its usage cost growth due to environmental expenses increase shall be considered. Increase of environmental expenses is a global movement. Currently environmental protection measures expense is from 0.2 up to 2% of GDP in the world, in other countries – 3-4% of GDP.

Reduction of environmental load and environmentally-oriented economies development becomes a priority in the area of technology policy of leading countries. This paper is based on the priority of carbon dioxide zero emissions achievement due to economy energy efficiency reduction, the changes in waste management, ecologically-oriented lifestyle. In the EU environmental innovation industry has become important economy sector annual turnover of which is estimated in 319 billion Euros which already amounts more than 2.5% GDP.

Extraction and transportation of solid fuel (coal, peat, schist) is more complicated than oil and gas extraction and transportation; therefore research and development of conversion technologies of solid fuel into gaseous or liquid ones production of which shall be preferably located near oilfield being developed.

According to expert analysis [1,2,3,4,9] in the period till 2030 the following innovative products and services that will cause significant effect on coal demand will be distributed on energy resources extraction, processing and usage markets:

- Equipment and materials for minerals extraction efficiency improvement: field development system based on the combination of physical and technical, physical and chemical technologies; enhanced bed recovery systems and methods including directed change of their reservoir properties;
- Special-purpose equipment (including equipment designed for reduction of negative impact on the environment), as well as oilfield services in the area of non-traditional and scavenger oil extraction (heavy and extra-heavy oil, oil sands and bitumen, oil from low-permeable rocks including schist, oil of Bazhenov formation, kerogen);

- Natural gas of unconventional fields: coal bed methane, shale gas, gas of low-permeable rocks, deep layer gas, gas hydrates;
- Innovative equipment for natural compressed gas production: floating gas liquefaction factories, methane carriers, floating regasification terminals;
- Alternative motor fuels: synthetic motor fuels of natural gas, coal or biomass; hydrogen for energy generation in fuel cells;
- Information systems for mineral extraction efficiency improvement: algorithms and software for formalization and knowledge extraction from semi-structured and unstructured information, geologic survey systems in complicated climatic and geological conditions, supplementary exploration of produced and generated fields, geological survey of unconventional energy sources;
- Innovation materials and equipment on their basis for oil refining and petrochemistry: fuel cells, catalysts for innovative energy sources obtainment; gas-separating membranous nanomaterials; nanostructured materials for chemical current source; heat-resistant nanostructured composite, ceramic and metallic materials; nanostructured composite and ceramic materials and coverings with special thermal properties (heat-conducting, thermostatic); nanostructured anticorrosion coating; nanostructured antifriction and adhesive materials;
- Innovation materials and special equipment on its basis for mining work: new types of light and high-strength materials; heat-resistant nanostructured composite, ceramic and metallic materials; radiation resistant and radioprotective nanostructured composite materials and coatings; gas separation membrane nanomaterials;
- Industrial bioproducts: industrial enzymes and biocatalyst carriers; chemicals including monomer, for biodegradable polymers (organic acids, alcohols, diols, hydrocarbons).

## 3. Greening the Global Economy and tightening of environmental requirements for energy

Fuel and energy resources (FER) extraction has significant environment damage caused by dislocation of surface areas of rock massif, trap-downs and blows in mine openings and on ground surface of harmful gaseous, liquid and solid substances, change of natural physical fields and processes in rock massifs and far beyond its borders (hydrological, gas and heat).

Mining and processing mineral industries are among the most active and powerful man-caused environment conversion sources that often have both regional and global scope.

According to expert analysis [10] annually the large amounts of rocks are moved during mining operations, and natural landscapes have been changed by man for more than 50 % of globe territory.

Many mining regions both in Russia (Kuzbass, Norilsk, Donbass, Ural, etc.) and in other countries (Ruhr and Saar in Germany, Witwatersrand in RSA, Colorado, Wyoming in the United States, etc.) are an artificial environment that is often unsuitable for human habitation. Approximately 3 t of waste is formed during extraction of 1 t of coal and 0.2-0.3 t is formed during consumption. During extraction of 1 t of steel – 5-6 t waste and 0.5-0.7 t during processing, for 1 t of nonferrous metals – 100-150 t of waste during extraction and more 50-60 t during processing, for 1 t of rare, precious and radioactive metals – up to 5-10 thousand tonnes of waste during extraction and from 10 to 100 thousand tonnes during processing. So far waste disposal is not more than 6-10 %. Total number of all types of mining unutilized waste in Russia is about 45 billion tonnes. For their storage 250 thousand hectares of plot lands are used.

As a result of human impact on environment the total loss of living substance on the planet is more than 5 billion tonnes per year including those related with forests destruction – 4 billion tonnes per year, with soil erosion – 0.5 billion tonnes per year. In Germany about 30 % of forests are desperately diseased, at distances up to several hundred kilometres within cities and big agglomerations, as well as in their adjacent areas biomes are almost changed completely, animals, birds and insects that once inhabited these territories are disappeared, and they are replaced by parasitizing species.

Regions with intensive development of the mining and processing industries which are city-forming give rise to concern. Weak regulatory and legal framework, lack of strict control functions and organizations, low effectiveness of penalties facilitate the growth of downturns in natural environment exponentially.

## 4. Aspects for coal multipurpose utilization

The analysis of world technological trends shows that coal industries are appear to be on the brink of transition from industrial development to post-industrial one that ensures coal advanced processing and multipurpose utilization technologies implementation.

In particular, it is scheduled to provide “world standards in the area of environmental safety during coal extraction and enrichment; industrial recovery of coal advanced processing products (synthetic liquid fuel, ethanol, etc.) and related resources (methane, underground water, building materials)”.

Coal extraction capacities are kept ahead of demand for traditional coal market to a significant extent i.e. domestic energy coal market. Its capacity increase shall be expected only in the longer term due to new coal power stations implementation, as well as during large consumers’ transition from natural gas to coal. For the short-term the only option left for coal producers is a fierce competition with each other, at that the main mechanism of competitive struggle is price reduction.

There is a burning need in coal processing development which can provide qualitative change in products consumer performance and increase its market price respecevly, as well as allow exceeding the bounds of energy coal market.

Coal quality and competitiveness improvement, expansion of its field of usage and reduction of environment pollution with waste and polluting emissions can be achieved based on implementation of below indicated technologies (Fig. 1):

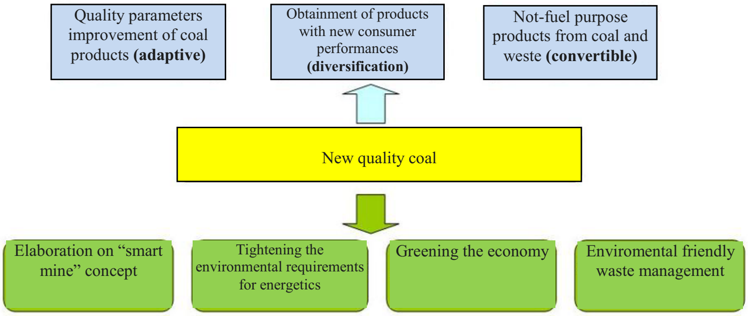


Fig.1 Coal processing technologies with solid part multipurpose utilization

- Foremost, technologies providing maximum gratification of increasing requirements of traditional customers (thermal power stations, metallurgy, public utility service, etc.) by means of quality parameters improvement of coal products. They allow to increase coal products quality burning of which is attended with efficiency growth of thermal power facilities and fuel saving. The following is included in the list of such technologies: coal enrichment, briquetting, water-coal slurry usage, pulverized fuel.

- Secondly, technologies providing products manufacturing with new customer performances. These should include: heat treatment (low-temperature carbonisation), coal gasification and hydrogenation. These technologies help to expand existing coal markets and create new ones.

- Thirdly, technologies providing coal processing (and coal waste) into non-fuel purpose products to be in certain demand among diferent industries. The following shall be included into the list of such products: adsorbents, humic fertilizers, montan wax, coal-alkali reagents, microspheres, etc. Usage of technologies of this group can improve environmental component of coal production due to partial waste management.

As of today experimental-industrial and research works regarding technological advancement and performance improvement of separate stages of coal and liquefaction products processing that significantly increases method efficiency in general is carried out in many countries of the world. Mostly major developments are carried out for:

- Coal gasification and hydrogenation, synthesis gas and liquid fuels generation;
- Production technologies of coal-water slurry fuels.

### 4.1. Coal gasification and hydrogenation, synthesis gas and liquid fuels generation

*Coal liquefaction technology* is a generation of oil products from coal by means of liquefaction. This technology allows generating about 1 tonne of oil products from 4 tonnes of coal.

Gasification is a high-temperature interacting process of fuel carbon with oxidants carried out in order to generate combustion gases ($H_2$, CO, $CH_4$).

*Synthesis gas* is an intermediate product which is obtained as a result of raw hydrocarbons processing.

One should emphasize that synthesis gas generation is a first stage of coal and natural gas conversion into chemical products and liquid fuels (diesel fuel, gasoline, etc.). The following can be generated from synthesis gas: methanol, aromatic compounds, ammonia, acetic acid, olefins and other compounds which are used as raw materials in the chemical industry (Fig.2).

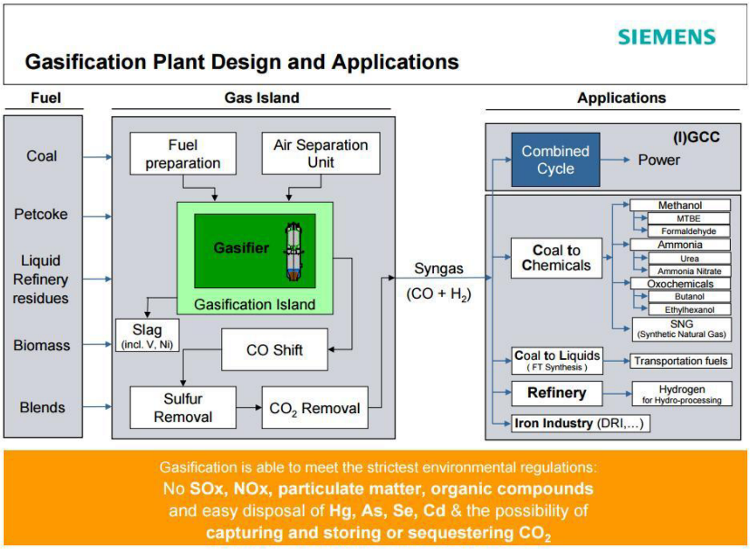

**Fig.2 Gasification plant arrangement and product scope of application**

Synthesis gas is a mix of hydrogen and carbon monoxide. There are many methods of synthesis gas generation, e.g., by means of methane oxidation or vapour convertion. Distinctive feature of these methods is a high cost of the final product.

Widespread occurrence of synthesis gas generation is explained due to environmental harmlessness of this fuel generation method. Comparing to traditional combustion of natural fossil synthesis gas generated from coal allows to reach 20 % fuel saving. Another positive and important advantage of synthesis gas generaion from coal is that coal gas generator allows emissions regulation and actually reduces environmental pollution to zero. Using coal gas generator one can generate $H_2$, CO, $CH_4$, etc. from coal, peat and wood. Coal gas generator does not produce soot, smoke and has a high efficiency.

Water combined with nonequilibrium plasma is used as hydrogen donor for *coal hydrogenation*. The process proceeds at about atmospheric pressure and moderate temperatures. Raw material is lignite fine and cheap culm fractions. In addition to coal one can use bituminous coal of low-grade metamorphism, as well as peat after partial thermal degradation stage to be performed in order to remove oxygen-containing compound excesses. Conversion degree of raw material combustion part into liquid fuel is 90-98.5 % depending on raw material type. It is possible to carry out raw material processing with higher ash content rather than raw material processed with other hydrogenation methods.

Currently coal processing researches are widely conducted in Australia, the Great Britain, Germany, Spain, Indonesia, the Philippines, China, Pakistan, the USA and Japan.

### 4.2. Production technologies of coal-water slurry fuels

*Coal water slurry fuel*, coal-water (abbreviation: CWS, CWSM, CWM) is a liquid fuel which is obtained by mixing crushed coal, water and plasticizer. It is used at heat-generating facilities primarily as an alternative to natural gas and fuel oil. It can significantly reduce costs in heat and electricity production.

*Coal-water* has set rheologic (viscosity, bias voltage), sedimentation (preservation of uniformity in static of circulated conditions) and fuel (energy potential, completeness of organic compounds combustion) characteristics. Coal-water parameters are strictly defined by national standards which can be used as reference. Also the following properties are typical for coal-water slurry fuel: ignition temperature – 800-850 °C, combustion temperature – 950-1150 °C, calorific efficiency – 3700 … 4700 kcal; carbon combustion degree – more than 99 %. Coal-water is fire and explosion-proof.

The basic principle in coal-water slurry fuel preparation is to provide coal grinding stability with set parameters and strict compliance with auxiliary material concentration resulting in improvement of rheological properties and combustion process stability.

As of today there are different coal grinding methods, but the most proven and studied method is to use ball mills of continuous wet grinding.

*Fuel coal.* As is seen from classification and quality requirements for coal-water slurry fuel only high quality thermal coal with low sulphur and ash content shall be used for preparation.

*Water.* During fuel preparation significant attention to elemental composition control shall be paid. This is due to the necessity to meet environmental regulations, and also it can extend equipment lifetime. Therefore in order to prepare the fuel one shall use only prepared and treated water.

*Plasticizers.* The usage of plasticizers in coal-water slurry fuel is driven by the need to provide special characteristics: low viscosity, good fluidity, long-term stability of suspended coal particles. Impurities on the basis of technical lignosulfonates, humic agents (sodium salts of humic acids of various fractions), polyphosphates which are effective in alkaline medium (at pH = 9 ÷ 13 at 40 % water in the fuel) are commonly used.

Usage restrictions:

- *Moisture content.* It is obvious that moisture which can make up to 40 % CWS is inert, and energy part from coal combustion is consumed on phase transfer energy of water from liquid to gaseous state. Exact value of energy consumed shall be considered based on coal calorific efficiency. For the most grades of coal one can assume that for every 10 % of moister is spent on 1 % of coal calorific efficiency. It is important to note that when comparing coal and CWS one should compare CWS moisture content and moisture content of initial coal which can be up to 25 %. For example, if moisture content of initial coal is 15 %, and moisture content of CWS generated is 38 %, then additionally one shall evaporate 38 % - 15 % = 23 % of moisture. Despite its obviousness this fact is often ignored during preliminary calculations.

• *CWS stability*. Standard CWS generated in the majority of facilities preserves its stability (it does not breakdown) within day or two suggesting the usage of specially-designed additives-plasticizers.

• *Injectors abrasive wear*. In the early stages of CWS application high abrasion wear of the injectors had occurred. For example, the first injectors operated less than 40 hours at Novosibirsk TPP-5. Under current conditions these problems have been fixed which includes release of serial burner units for CWS particularly made by Chine.

## 5. Conclusions

1. Advanced coal processing provides qualitative change of products consumer performance, increases its competitive capability and market value, allows reducing environmental pollution with west and polluting emissions.

2. Coal gasification and synthetic liquid fuel (SLF) generation technologies provide means for production of 1 tonne of oil products from 4-5 tonnes of coal.

3. Approximate specific construction capital intensity is estimated of about 1200 US dollars per 1 tonne/year of production capacity for synthetic liquid fuels.

4. For the short-term high-priority researches of advanced coal processing technologies with solid part multipurpose utilization will be:

• Development of new solid fuel gasification technologies with synthesis gas production, selection of optimal parameters and main equipment designs;

- Development of new solid fuel gasification and pyrolysis technologies, selection of optimal parameters and main equipment designs;
- Development of manufacturing methods for wide range of products from synthesis gas, selection of optimal parameters and main equipment designs;
- Development of optimal process diagrams of electro technical units on the basis of solid fuel advanced processing with high-quality fuels, electric power energy and chemical products generation;
- Long-term perspective research of large-scale solid fossil fuels processing and assessment of its impact on energy markets.

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# NEW APPROACHES TO CREATION OF LOW-TEMPERATURE COMPOSITIONS FOR PREVENTION OF COAL CONGEALING

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Kuzbass increases production and transportation of black coal. During the winter period there is congealing of coal mass and its freezing to the walls and the bottom of rail cars and dump trucks. Preventive liquids forming the low-freezing membranes used to be the main means of control.

New ideas of hydrogen bond (A.A. Grishayev) in water responsible for converging of water into ice and congealing of loads which are based on dynamic balance of long and short H-bond, allow to consider ice, water, and water solutions as dynamic systems with lifetime of $10^{-5}$-$10^{-12}$ seconds. In such systems the power centers of crystallization and structural waves due to long H-bonds, heteroatoms and other active particles are created. Blocking or destruction of such centers interrupts formation of ice or makes it inhomogeneous and fragile. This is affected by the introduction of alkyl, alkoxy, or acetyl radicals, Cl-anion, and the use of mixes of chemical compounds.

The example of a technical mix which forms a basis for creation of a preventive for the coal, walls and the bottom of rail cars is caprolactam production waste - the alkaline concentrate of caprolactam production (ACCP) on STO 05761637-003-2012 with a freezing temperature minus 37°C. Surfactants allow to regulate wetting, spreading and other indicators of a reagent.

**Keywords: congealing of coal, striking toughness, strength of ice, SAW surfactants, reagents, freight, hydrophobic membranes, polymers of water.**

Kuzbass is ramping up production and transportation of coal. Winter is the period of coal mass congealing; its adfreezing to the walls and bottom of the cars and trucks. The main means of combating these phenomena is to use preventive liquids, which form low-freezing membranes.

We have reviewed the proposals of the leading foreign companies Nalco, American Chemical Services Co and the American Chemical Services Associates, as well as Rosneftehim, Bashneftehimsnab and local producers supplying reagents for coal treatment against congealing. Today they use blue oils based on naphta (niogrin, severin, etc.), a mixture of alcohols (bottoms liquid, ketgol), aqueous solutions of glycols (ethylene and propylene) and salts (chlorides, acetates), solutions of solid paraffins in hydrocarbons, and al.

Most of the hydrocarbons compositions, alcohols, and aqueous solutions of glycols create antifreeze with a pour point of minus (40 - 60) °C. Some aqueous solutions have moderate freezing temperatures minus (5 - 30) °C, but they form a fragile structure of ice, which is easily destroyed during unloading. Paraffins form hydrophobic membranes on the surface of the coal and car walls.

New ideas of the hydrogen bonds (A.A. Grishaev) in water, responsible for the transformation of water into ice and congealing loads, which are based on the dynamic equilibrium of the long and short H-bonds,



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allow us to consider ice, water, and aqueous solutions as dynamic systems with lifetime of 10-5 – 10-12 seconds. These systems provide power centers of crystallization and structural waves due to long H-bonds, heteroatoms and other active particles. Blocking or destruction of such centers interrupts the formation of ice, or makes it uneven and fragile. The agents would be alkyl, alkoxy or acetyl radicals, anion Cl-, and mixtures of chemical compounds.

The advantages of preventive agents include not only the prevention of coal congealing and its adfreezing to the walls of the cars, but also reduced strength of the formed ice.

Ice and snow used to be polymers of water; they are in complex interaction with liquid water and monomer - H2O vapor. Ice has a heterogeneous structure, it includes several types of crystals, bubbles of gas (usually air), solids (minerals, organic compounds), and surfactants adsorbed on the boundaries of the phases. The pattern of this ice may serve frozen air foam of low expansion or foam plastic based on oligomers. Foam freezing slows down the processes of foam inner destruction, in foam plastic this process is not practically going on. The frozen foam is characterized by the intense sublimation of ice from the warmer layers into the cold ones. Creating hydrophobic membranes (partition) to the movement of water vapor inhibits this process. In some cases even monolayer membranes are effective: lauryl alcohol or polietilgidrosiloksan (NGL-94). Additives NGL-94 (up to 0.1 %) decrease the rate of ice sublimation 4 fold. Mixtures of surfactants are even more effective. Volgonat (0.1 %) and NGL-94 (0.006 %) reduce the flow of water vapor in 8.7 times [3].

The adsorption of surfactants saturation of the reaction mixture of plastic foams results in the mechanically fragile membranes and foam plastics. A similar phenomenon is observed in solidification of foam concrete at the excess of the surfactants in the solidifying suspension.

We have studied the effect of anionic surfactants of caprolactam alkaline concentrate (CAC) and added to it nonionic surfactant - PolyPnB on the ice strength. Along with the main components - sodium adipate CAC contains sodium salts of carboxylic acids, which are the typical anionic surfactants [1] and [2].

One of the mechanical strength characteristics under dynamic loads is resilience, which is identified by pendulum type device (Fig. 1 and Fig. 2) [4] and [5].

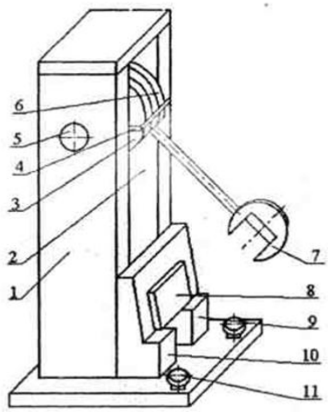


**Fig.1.** A device for determining resilience

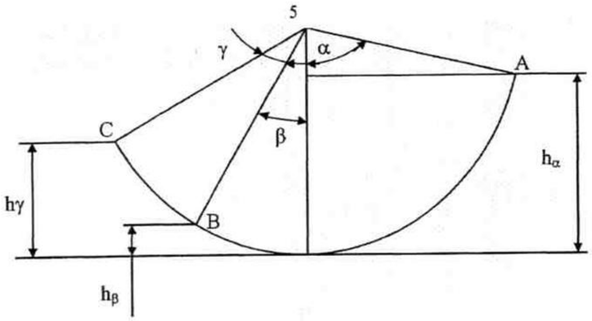


**Fig. 2.** A diagram of the pendulum impact.

Specific resilience $a$ is equal to the energy U, spent on the destruction of the sample and divided by the cross-sectional area of the sample $A = h \times B$.

$$a = \frac{U}{A}, N \cdot m/m^2, \quad (1)$$

$$U = mgl\left[\cos\beta - \cos\alpha - (\cos\gamma - \cos\alpha)\frac{\alpha+\beta}{\alpha+\gamma}\right], \quad (2)$$

where:

m - mass, kg;
g - acceleration of free fall;
l - pendulum equivalent length;
α - pendulum angle of ascent;
β - the angle of the pendulum take-off after the sample destruction of (Fig. 2)
γ – the angle of pendulum in idling;
The scale unit is graduated in dimension U.

For the ice production we used: distilled water and 10 % solution of CAC with additives. Ice samples were prepared in metal molds of 100 × 26 × 23 mm, which, after filling the liquid were kept in a freezer at -21 °C for 4 - 24 hours. The test results are shown in Table 1.

**Table 1** - The strength of ice

| Samples of ice on the basis of | Cross-section h×B, mm | U, Nm | a, $Nm/m^2$ |
|---|---|---|---|
| Distilled water | 26,0×15,0 | 0,433 | 1110 |
| Distil. water + 10% CAC | 26,0×16,0 | 0,360 | 865 |
| Distil. water + 10% CAC, soda 1% surfactant | 26,0×15,0 | 0,263 | 653 |
| Distil. water + 10% CAC, soda 0,5% surfactant and 0,5% oil | 26,5×17,0 | 0,275 | 610 |

Anionic surfactants - CAC at a concentration of 10 % by weight reduces the strength of ice by 22.4 %. The presence of nonionic surfactant – PolyPnB, the content of which in the solution does not exceed 0.1%, the strength decreases even more – by 41.2 %. If the mixture of PolyPnB and oil is used as an additive to CAC, the result is improved by 45.09 %.

During this experiment, we studied coal congealing and adfreezing it to the metal walls under the laboratory conditions.

Experimental conditions: the temperature was minus 21°C. The coal of 0-3 mm with humidity of 3.4 % kept for a long time in the open air was not subject of congealing when loaded into a refrigerator compartment. The coal was humidified to 15 %, thoroughly mixed and kept in a sealed flask for 24 hours.

A portion of wet coal was placed in a flask with a stopper, an additive was put in an amount of 0.4 % of wt., and the content was thoroughly mixed. Then the samples were placed into a metal container and put in a freezer for 8-12 hours. The test results are given in the Table 2.

**Table 2** - Coal congealing by humidity of 15% by weight.

| № | Reagent | Observations |
|---|---|---|
| 1 | control (untreated) | Coal congealed, did not spill out of the container |
| 2 | Nalco | Coal spilled out, small agglomerates of coal observed |
| 3 | Polyglycols (DPG) | Coal spilled out of the container after three times shaking |
| 4 | Polyglycols + surfactants | Coal spilled out of the container after three times shaking |
| 5 | Polyglycol aqueous solution (VRPG) | Coal congealed, but in 0.5 of an hour spilled out without shaking |

Samples 1,2,4,5 were kept in the open air for 24 hours at a temperature of minus 31-35 °C. The control sample 1 congealed, did not spill out. The results of treatment by reagents Nalco, polyglycols and VRPG were similar. Coal spilled out of metal containers, but we observed fragile agglomerates and light adfreezing to walls.

The second series of experiments was devoted to the study of brown coal with humidity of 30%. Coal is ground into powder for analysis. Adfreezing of treated or untreated by reagents brown coal to the walls and bottom of the container and congealing of the coal mass occurs at a temperature of minus 20 °C, provided its compaction. Without compaction the coal also congeals in mass but less adfreezes to the bottom and walls, if they are lubricated with the reagent. The observation results are shown in Table 3.

**Table 3** - Coal congealing of 30 % wt. humidity

| № | Reagent | Observations |
|---|---|---|
| 1 | control (untreated) | Coal froze on the walls and the bottom, did not spill out after 5 strokes |
| 2 | CAC ρ 1,210 | Coal froze on the walls and the bottom, spilled out after 5 strokes |
| 3 | CAC ρ 1,210 + 1 % surfactant | Coal froze on the walls and the bottom, spilled out after 5 strokes, the surface is smoother |
| 4 | RPS-2, grade A | Coal froze on the walls and the bottom, spilled out after 5 strokes |
| 5 | RPS-2, grade B | Coal froze on the walls and the bottom, spilled out after 4 strokes |

**The study of coal congealing at minus 40 °C (climate chamber)**

We used a metal model of a train car in scale 1:40. Load was 1.5 kg of steam coal, humidity 10.5 % by weight. There were three types of treatment: the surface of the car, the car surface + coal in bulk. Exposure in a chamber at -40 °C continued for 24 hours. The test results are shown in Table 4. The table shows characteristics of the reagents in accordance with the test procedure.

**- Reagent CAC (production - "Azot", Kemerovo)**

Reagent is a turbid viscous dark brown liquid with solids in the form of off-white flakes. It is easily sprayed, forms a stable membrane on the surface of the car. The reagent has an unpleasant characteristic smell. It preserves a limited fluidity at - 40 °C. The reagent showed the worse effectiveness then the control in all methods of treatment (-5 % to -21.4 %).

Coal from the train cars was unloaded in lumps from tight-loose to dense consistency. In the combined method of treatment the coal in train cars congealed into a monolith and could not be unloaded even while the car was under the conditions of room temperature for 15-20 minutes.

**Table 4**- Test results

| Reagent | Type of treatment | The amount of the remaining coal in the train car | | | | |
|---|---|---|---|---|---|---|
| | | Without stroke | After the 1 stroke | After the 2 stroke | After the 3 stroke | After the 6 stroke |
| CAC | The control car (no treatment) | 99,7 | 91,7 | 82,5 | 74,9 | 51,4 |
| | Treated surface of the car | 99,8 | 92,1 | 83,6 | 78,2 | 56,4 |
| | Treated bulk of coal | 99,9 | 91,9 | 85,1 | 82,4 | 65,5 |
| | Treated surfaces and the bulk | 100 | 96,8 | 93,8 | 90,0 | 77,8 |

| | | | | | | |
|---|---|---|---|---|---|---|
| | of coal | | | | | |
| RPS-2 Grade B | The control car (no treatment) | 99,6 | 80,4 | 63,2 | 27,1 | 12,1 |
| | Treated surface of the car | 99,6 | 81,3 | 68,3 | 60,7 | 0,0 |
| | Treated bulk of coal | 99,7 | 83,2 | 58,3 | 26,6 | 0,0 |
| | Treated surfaces and the bulk of coal | 99,8 | 86,1 | 74,5 | 64,4 | 15,0 |
| RPS-2 Grade A | The control car (no treatment) | 100 | 88,3 | 79,2 | 64,2 | 24,4 |
| | Treated surface of the car | 100 | 91,9 | 86,4 | 68,0 | 31,4 |
| | Treated bulk of coal | 99,8 | 89,8 | 81,2 | 76,4 | 20,5 |
| | Treated surfaces and the bulk of coal | 99,9 | 84,3 | 74,2 | 55,5 | 4,8 |

**- Reagent RPS-2 grade B (experimental sample, KemTIPP, Kemerovo)**

The reagent is turbid dark brown liquid with a specific smell. It is well sprayed, forms a membrane, some quantity of the reagent is dripping from the walls of cars on the bottom. It is fluid at -40 °C.

The reagent showed better effectiveness then the control in treating coal in the bulk and on the car surface (in both cases + 12.1 %). In the combined method of treatment the effectiveness was a little worse than the control (-2.9 %).

The coal is unloaded from the car in lumps of tight-loose consistency.

**- Reagent RPS-2 grade (experimental sample KemTIPP, Kemerovo)**

The reagent is a clear, dark brown liquid with a characteristic smell. Sprayed well, but during the surface treatment the reagent flows down to the bottom of the car due to the low viscosity. At -40 $^{O}$C it remains fluid. In the treatment of surfaces of cars the effectiveness was worse than the control (-7.0%). During treatment of coal in bulk and combined treatment method the reagent showed better efficiency then control (+ 3.9 % and + 19.6 % respectively).

The coal unloaded from the cars was in the form of loose lumps.

**The development of improved reagent based on CAC.**

The results of laboratory and production tests suggest RPS-2 grade A and grade B as the reagents against congealing. The indicators of these reagents are presented in Table 5.

**Table 5** - Indicators of RPS-2 grade A and B

| Indicator | Standard | | Method of treatment |
|---|---|---|---|
| | Grade A | Grade B | |
| 1.Visual look | Liquid from brown to dark brown color, opaque, with no visible mechanical impurities | Liquid from light yellow to brown color without mechanical impurities | In para. 4.3 of current specifications |
| 2. Density at 20 ° C, g / cm$^3$ in the range | 1,152 - 1,185 | 1,070 - 1,120 | According to GOST 18995.1 Section 1 |
| 3. Congealing temperature, °C in the range | minus 30 - minus 35 | minus 40 - minus 45 | According to GOST 20287 p. 2 and Method B and p. 4.4 of current specifications |
| 4. Hydrogen ions activity, pH units, in the range | 8 - 10 | 8 - 10 | According to GOST R 50550 and paragraph 4.5 of current specifications |

Grade A provides the operating temperature of the reagent to minus 30 - 35 °C by applying a dilute solution of CAC with 1,152 - 1,185 density. The temperature range of minus 40 - 45 °C is achieved by adding a water-glycol mixture of HCV to CAC. Grades A and B have lower indicators of viscosity, and surface tension than CAC, which provides good dispersion through the nozzles under experimental and industrial conditions.

The reagent has been subject of patent application number 201403821.

## Conclusion.

1. A reagent against congealing RPS-2 has been developed, which comprises a combination of the following factors.

RPS-2 Grade A has a density of 1,152-1,185 and pour point temperature of minus 30-35 °C.

RPS-2 Grade B has a density of 1,070-1,120 and pour point temperature of minus 40-45 °C.

2. Reagent RPS-2 has been tested in the laboratory and partially in an industrial environment (Borodinsky open cast - CAC + 0.08 surfactants), showed good dispersion via nozzles.

3. Reagent RPS-2 has advantages over CAC on the reagent consumption per $m^2$ of the surface and on the reduction of the ice strength (if any).

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# Various potential uses of coal from Apsatsky coalfield

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**Summary**

Additional geological exploration and studies of balance reserves of Apsatsky hard coal deposit have shown that these belong to valuable coking coal grades – "Zh", "KZh" "K", "KO", and "OC". Only a small proportion of coals located near the surface at the depth of up to 20 metres is partly oxidised and belongs to the "CC" grade of steaming coal.

The coal ash has low slagging properties and medium ability to pollute convective heating surfaces. In terms of fusibility, the ash belongs to medium-melting: 1200ºC<$t_b$<1350ºC. The main harmful components in the coal seams are sulfur ($S_t^d$) whose content is below 1.0% and phosphorus ($P^d$) whose weighted average content is 0.003% and potential maximum content is 0.025%.

Thus, coals of Apsatsky deposit are recommended for use in the layered coking process to produce high quality metallurgical coke as well as liquid and gaseous products. At the same time, the oxidised coals are a valuable raw material for burning at modern combined heat and power plants and state district power stations. So one of the main challenges related to operation of Apsatsky coalfield is to produce concentrate with ash ($A^d$) content below 9.6%.

The results of pilot benefication of Apsatsky coals at the leading CPPs in Russia showed that in the course of transportation, storage, coal handling and processing there was a noticeable change in particle size distribution. The percentage of sizes above 25 mm decreased by 2.2-2.5 times and came to less than 30.0% while the total percentage of fine sizes of less than 13 mm increased by more than 30.0%.

Most of the concentrate yield belonged to small sizes when washing coals of 0.2-3 mm size while the concentrate ash content was less than 9.0%. Crushing of middlings (fraction of 1550-1800 kg/m$^3$) of plus 25 mm size to 0-3 mm size confirmed feasibility of liberation and additional recovery of up to 15% of concentrate with ash content below 9.6%.

Apsatsky coals demonstrate high coking properties. The coefficient of technological value calculated using VUHIN Institute's methodology has a high rate – 0.97. Mechanical strength of coke in case of individual coking was M40=74%, M25=88%, and M10=8.5%. Coke strength in a hot condition in case of individual coking of coal seams varied between CSR=34.0%-52.0% at CRI=40.0%-54.0%.

Commercial blend of Apsatsky coals with other specially selected coking coals has shown significant improvement of the CSR value. Coking properties of Apsatsky coals also significantly improved once concentrate ash content was reduced from 9.6% to 8.0%. All of these factors allow us to place concentrates of Apsatsky coals into the semi-hard coking coal category.

In the course of tests and studies of Apsatsky coals it was demonstrated that their dense medium benefication using flotation enables us to produce high quality concentrate at more than 74.0% yield and less than 9.6% ash. When Apsatsky coal concentrates were used at Russian coking plants we managed to obtain blast-furnace and foundry coke of export quality with high performance capability.

**Keywords**

Valuable coking coal grades, concentrate yield and quality, physical strength of coal in cold and hot condition.

JSC SUEK has started developing a new Apsatsky hard coal deposit located in the southern part of Kodar Ridge 33 km north of the Baikal-Amur railway in Chita region. Development of new mineral reserves in remote locations is always associated with overcoming additional difficulties and finding solutions to various engineering challenges.



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Among the major factors which complicate construction and operation of new coal mines in Russia are: [1] remoteness of coalfields from sales markets and well-developed industrial centres; [2] poor infrastructure of the coal-bearing region and lack of engineering services of the required capacity; [3] poor exploration of deposits and incomplete geological, mining and technical and economic data; [4] need for additional exploration of the deposit to specify geologic structure of coal measures, quality and properties of coals and the strata, and feasibility of their integrated use; and [5] insufficient technical and economic validity of the proposed technological and engineering solutions.

Apsatsky coalfield has been developed in view of its specific features, namely:

- complex geologic structure of Srednechepinsky subsuite of Upper Jurassic age;
- highly faulted coal seams and strata;
- permafrost extending to the depth of 200 metres;
- large number of closely spaced coal seams;
- considerable variability of in-situ and operational characteristics of coals along strike;
- major variability of seam thickness and gradients;
- low as-received total moisture content of coals because of their freeze-out within the extent of permafrost;
- very low, fourth (IV) category washability of coals;
- high ratio (more than 30%) of fine sizes below 2.0 mm in ROM coals;
- high degree of breakage and slime generation of coals during storage and transportation;
- tendency of coals to oxidise in case of prolonged storage and transportation due to high proportion of fine sizes of coal, low moisture content, and presence of iron and aluminium oxides in mineral impurities.

Due to the lack of complete geological and mining data additional exploration and studies of Apsatsky's balance reserves have been carried out. By drilling additional exploratory boreholes and taking seam and operational samples we have established that these coals belong to a group of coal grades which are in short supply in Russia – "Zh", "KZh", "K", "KO", and "OC". Only a small proportion of coals located near the surface at the depth of up to 20 metres is partly oxidised and belongs to the "CC" grade of steaming coal.

Balance reserves of Apsatsky open pit are located in the southeast area of the deposit. In the original mining area there are five extractable seams (B1, B2+4, B5, B6, B7) and two split seams (B2+4lower and B2+4upper). The average major quality parameters of ROM coals are shown in Table 1.

The quality parameters of ROM Apsatsky coals show medium ash content, high calorific value and low sulfur ($S_t^d$) and phosphorus ($P^d$) content – below 0.6% and below 0.025%, respectively. The HGI parameter indicates that these coals have low resistance to physical impact and are prone to degradation in size. The coal ash has low slagging properties and medium ability to pollute convective heating surfaces. In terms of fusibility, the ash belongs to medium-melting: 1200ºC< $t_b$<1350ºC.

**Table 1: ROM coal quality**

| Quality parameters | Units | Coal Seam | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | B1 | B2+4lower | B2+4 | B2+4upper | B5 | B6 | B7 |
| Seam thickness | m | 6.5 | 9.3 | 14.3 | 12.7 | 11.3 | 2.3 | 6.7 |
| Moisture in the analysis sample ($W^a$) | % | 0.5 | 0.5 | 0.7 | 0.7 | 0.7 | 0.8 | 0.8 |
| Ash ($A^d$) | % | 15.7 | 19.5 | 18.5 | 16.4 | 16.2 | 18.2 | 20.5 |
| Gross calorific value ($Q_s^{daf}$) | MJ/kg | 36.1 | 36.2 | 36.0 | 35.9 | 36.0 | 36.2 | 36.1 |
| Net calorific value ($Q_t^r$) | kcal/kg | 6850 | 6520 | 6430 | 6680 | 6700 | 6510 | 6350 |
| Volatile content ($V^{daf}$) | % | 27.3 | 27.6 | 26.2 | 26.4 | 28.1 | 28.4 | 29.0 |
| Sulfur content ($S_t^d$) | % | 0.35 | 0.42 | 0.30 | 0.30 | 0.34 | 0.35 | 0.45 |
| Phosphorus content ($P^d$) | % | 0.006 | 0.006 | 0.025 | 0.004 | 0.025 | 0.002 | 0.004 |
| Absolute density ($d_r^d$) | $g/cm^3$ | 1.38 | 1.48 | 1.42 | 1.40 | 1.42 | 1.42 | 1.48 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Apparent density ($d_a^d$) | g/cm$^3$ | 1.27 | 1.37 | 1.31 | 1.29 | 1.30 | 1.32 | 1.35 |
| Hardgrove grindability index (HGI) | units | 87 | 84 | 85 | 86 | 85 | 84 | 84 |
| Thickness of plastic layer (Y) | mm | 17 | 16 | 13 | 14 | 16 | 17 | 17 |
| Free swelling index (FSI) | units | 5.5 | 5.5 | 4.5 | 4 | 5.5 | 5.5 | 5.5 |
| Gray-King coke type (G) | type | G5 | G4 | G2 | G3 | G4 | G5 | G5 |
| Gieseler max fluidity ($F_{max}$) | ddpm | 57 | 40 | 26 | 28 | 45 | 145 | 280 |
| Roga index (IR) | units | 30 | 35 | 24 | 25 | 40 | 55 | 58 |
| Carbon ($C^{daf}$) | % | 90.5 | 91.3 | 90.6 | 90.8 | 90.6 | 90.5 | 90.9 |
| Coal grade as per GOST standard 25543-88 | grade | K | K | KO | KO | K | K | KZh |

Thus, coals of Apsatsky deposit are recommended for use in the layered coking process to produce high quality metallurgical coke as well as liquid and gaseous products. At the same time, the oxidised coals are a valuable raw material for burning at modern combined heat and power plants and state district power stations. So one of the main challenges related to operation of Apsatsky coalfield is to produce concentrate with ash ($A^d$) content below 9.6%.

For the purpose of studying peculiar features of Apsatsky coal washability trials have been held at the leading Russian CPPs. The results showed that in the course of transportation, storage, coal handling and processing there was a noticeable change in particle size distribution. The percentage of sizes above 25 mm decreased by 2.2-2.5 times and came to less than 30.0% while the total percentage of fine sizes of less than 13 mm increased by more than 30.0%.

Studies of breakage and slime generation immediately during benefication of coals showed a decrease in the percentage of medium and coarse sizes by 10.0%-12.0% while there was more muddy slime of 0-0.05 mm generated, which increased its share to 17.0%-19.0%.

The theoretical optimal separation density varied seam by seam from 1350 kg/m$^3$ to 1650 kg/m$^3$, which presented technological challenges in terms of blending coal from different seams. The mineral impurities which were hard and had low swelling ability mostly belonged to the +50 mm and -0.05 mm sizes. The concentrate yield varied by seam from 72% to 79% with ash content below 9.6%.

For coarse and medium sizes the optimal density exceeded 1500 kg/m$^3$ while the ash content of the concentrate was around 11.0% and its yield up to 70.0%. The share of coarse and medium sizes in particle size distribution of the concentrate was under 20.0%.

Most of the concentrate yield belonged to small sizes when washing coals of 0.2-3 mm size while the concentrate ash content was less than 9.0%. Crushing of middlings (fraction of 1550-1800 kg/m$^3$) of plus 25 mm size to 0-3 mm size confirmed feasibility of liberation and additional recovery of up to 15% of concentrate with ash content below 9.6%.

Comprehensive research of the quality and washability of Apsatsky coals conducted by SibNIIUgleobogashenie LLC in cooperation with CSJC SGS Vostok Limited, an independent body, and OJSC VUHIN (East Scientific and Research Coal Chemical Institute) identified a number of preferable technological solutions for processing coals from various coal seams and obtained estimated coal processing product balance. The estimated coal processing product balance for the seams which are mined by Apsatsky open pit is shown in Table 2.

**Table 2: Estimated coal processing product balance by seam**

| Seam | Grade | Washability | Products | Density (kg/m$^3$) | Yield (%) | Ash (%) | CV (kcal/kg) |
|---|---|---|---|---|---|---|---|
| B1 | "K" | low | concentrate | 1500 | 81.2 | 9.6 | 7430 |
| | | | middlings | 1800 | 11.6 | 27.2 | 5690 |
| | | | refuse | | 7.2 | 65.9 | |
| B2+4lower | "K" | very low | concentrate | 1400 | 34.9 | 9.6 | 7540 |
| | | | middlings | 1800 | 38.2 | 30.0 | 5725 |
| | | | refuse | | 26.9 | 77.3 | |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| B2+4 | "KO" | low | concentrate | 1500 | 81.9 | 9.6 | 7400 |
| | | | middlings | 1800 | 11.5 | 49.3 | 5725 |
| | | | refuse | | 6.7 | 66.8 | |
| B2+4upper | "KO" | medium | concentrate | 1500 | 82.5 | 9.6 | 7450 |
| | | | middlings | 1800 | 10.5 | 24.6 | 5865 |
| | | | refuse | | 7.0 | 66.6 | |
| B5 | "K" | medium | concentrate | 1650 | 83.2 | 9.6 | 7460 |
| | | | middlings | 1800 | 9.0 | 22.3 | 6055 |
| | | | refuse | | 7.8 | 65.0 | |
| B6 | "K" | low | concentrate | 1450 | 50.2 | 9.6 | 7400 |
| | | | middlings | 1800 | 22.3 | 23.1 | 6005 |
| | | | refuse | | 27.5 | 77.3 | |
| B7 | "KZh" | low | concentrate | 1400 | 52.0 | 9.6 | 7355 |
| | | | middlings | 1800 | 38.6 | 24.5 | 5885 |
| | | | refuse | | 9.4 | 71.4 | |

Besides the abovementioned peculiar features of Apsatsky coalfield, additional ones include: [1] significant variation of coal grades both by seam and along strike and to the dip in each particular seam; [2] different optimal separation density for different seams, which is in the 1350-1650 kg/m$^3$ range; and [3] different optimal separation settings for machine size grades for subsequent processing. All of these present additional challenges during mixing of coals from different seams and selection of an appropriate processing flowsheet.

At the same time, it is already clear that for processing Apsatsky coals we will need a two-section coal preparation plant with gravity dense medium benefication method and flotation. The CPP will process coals of three machine size grades.

**Process flow diagram for processing Apsatsky coals**

The surface coal handling facility for receiving ROM coal of 0-1200 mm size is located at the CPP and includes an open stockpile of at least 50,000 tonne capacity. There are four coal piles for each group of grades: "Zh+KZh", "K", "KO-OC", and "CC". Partly oxidised coals are not processed – instead they are transported to a mobile crushing and sizing facility where they are divided by size for selling to customers for power generation.

The preparation stage for ROM coal which consists of coals suitable for coking includes production of ROM coals within the 0-250 mm size range. For this purpose the ROM coal is loaded with a front-end loader into a 100 m$^3$ hopper which includes a 600 mm static grizzly and a vibrating feeder. A scraper feeder 1.6 m wide and 50.0 m long with 600 mm static grizzlies may be used instead of a hopper with a static grizzly. The scraper feeder can be loaded as coal is unloaded from 110 tonne trucks.

The 0-600 mm size is transported to a cylindrical screen/crusher while the 600-1200 mm size is crushed at the stockpile using mechanical means. The cylindrical screen/crusher ensures the following: [1] preliminary sizing of ROM coal at 250 mm; [2] removal of foreign objects (wood, steel, other materials) from the ROM flow; [3] mechanical retrieval of waste rock in the 250-600 mm size thus reducing its ash content; and [4] crushing of the 250-600 mm size coal to 0-250 mm.

Then ROM coal of 0-250 mm size is transported on a belt conveyor which includes devices for removing any residual ferromagnetic objects to screens for preliminary sizing for dry sieving into 50-250, 1-50 and 0-1 mm machine size grades. These machine size grades are conveyed to the main CPP building. The 0-250 mm ROM coal is delivered to a double deck screen for wet desliming where it is re-sorted into 50-250, 1-50 and 0-1 mm sizes in a more assured way and wetted for subsequent processing.

Removal of waste rock and production of mix from the 50-250 mm size requires a double product dense medium separator operating at 1800 kg/m$^3$ separation density. The mix will be sized and crushed to the 1-25 mm size to ensure liberation in coal and rock and subsequent increase of concentrate yield. The crushed 1-25 mm size mix goes to a vibrating screen for desliming, after which it is washed in a double product dense medium cyclone at 1400-1650 kg/m$^3$ separation density producing concentrate and middlings.

Removal of waste rock and production of mix from the 1-50 mm size requires a double product dense medium cyclone operating at 1750 kg/m$^3$ separation density. The mix will be sized and crushed to 1-13 mm size to ensure liberation in coal and rock and subsequent increase of concentrate yield. The crushed 1-13 mm size mix goes to a vibrating screen for desliming, after which it is washed in a double product dense medium cyclone at 1350-1500 kg/m$^3$ separation density producing concentrate and middlings.

Three-product flotation is required for washing the 0-1 mm size. The raw flotation feed is supplied from the overflow of dense medium cyclones, after desliming and sizing into machine size grades. The speed and efficiency of the flotation process largely depend on selection of flotation unit and optimal reagents for various grades and seams of Apsatsky coals.

The belt filter press section and the fines circuit are conventional for Russian coking coals. The concentrate stockpile is enclosed with at least 20,000 tonne capacity allowing coal to be piled according to the grade groups. The middlings stockpile is of open type and has capacity of at least 10,000 tonnes.

Apsatsky coals demonstrate high coking properties. The coefficient of technological value calculated using VUHIN Institute's methodology has a high rate – 0.97. Mechanical strength of coke in case of individual coking was M40=74%, M25=88%, and M10=8.5%. Coke strength in a hot condition in case of individual coking of coal seams varied between CSR=34.0%-52.0% at CRI=40.0%-54.0%.

VUHIN's experiments associated with trial coking of a blend produced by various Russian coking plants, which is based on Apsatsky coal concentrates and specially selected coking coals from other suppliers have shown significant improvement in the CSR value. Preliminary studies held by independent coal chemical laboratories have confirmed that reduced ash content of Apsatsky coals and concentrates leads to their increased coking properties (see Table 3). This enables us to place concentrates of Apsatsky coals into the semi-hard coking coal category.

**Table 3: Effect of ash content on coking properties**

| Quality parameters | Units | Coal Seam | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | B1 | B2+4lower | B2+4 | B2+4upper | B5 | B6 | B7 |
| Average values for ROM coals | | | | | | | | |
| Ash ($A^d$) | % | 15.7 | 19.5 | 18.5 | 16.4 | 16.2 | 18.2 | 20.5 |
| Thickness of plastic layer (Y) | mm | 17 | 16 | 13 | 14 | 16 | 17 | 17 |
| Free swelling index (FSI) | units | 5.5 | 5.5 | 4.5 | 4 | 5.5 | 5.5 | 5.5 |
| Gray-King coke type (G) | type | G5 | G4 | G2 | G3 | G4 | G5 | G5 |
| Gieseler max fluidity ($F_{max}$) | ddpm | 57 | 40 | 26 | 28 | 45 | 145 | 280 |
| Roga index (IR) | units | 30 | 35 | 24 | 25 | 40 | 55 | 58 |
| Low-ash ROM coals | | | | | | | | |
| Ash ($A^d$) | % | 11.0 | 12.0 | 12.0 | 11.5 | 11.5 | 10.5 | 10.5 |
| Thickness of plastic layer (Y) | mm | 19 | 18 | 15 | 16 | 18 | 19 | 20 |
| Free swelling index (FSI) | units | 6 | 6 | 5 | 5 | 6 | 6 | 7 |
| Gray-King coke type (G) | type | G6 | G5 | G4 | G4 | G5 | G6 | G7 |
| Gieseler max fluidity ($F_{max}$) | ddpm | 85 | 60 | 50 | 55 | 65 | 200 | 1150 |
| Roga index (IR) | units | 42 | 47 | 38 | 35 | 52 | 60 | 60 |
| Concentrate | | | | | | | | |
| Ash ($A^d$) | % | 9.6 | 9.6 | 9.6 | 9.6 | 9.6 | 9.6 | 9.6 |
| Thickness of plastic layer (Y) | mm | 20 | 19 | 17 | 17 | 19 | 21 | 21 |
| Free swelling index (FSI) | units | 6 | 6 | 5 | 5 | 6 | 6 | 7 |
| Gray-King coke type (G) | type | G6 | G5 | G4 | G4 | G5 | G6 | G7 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Gieseler max fluidity ($F_{max}$) | ddpm | 120 | 80 | 90 | 70 | 80 | 250 | 1300 |
| Roga index (IR) | units | 45 | 50 | 40 | 40 | 55 | 60 | 65 |
| Low-ash concentrate | | | | | | | | |
| Ash ($A^d$) | % | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 |
| Thickness of plastic layer (Y) | mm | 22 | 20 | 18 | 18 | 19 | 22 | 23 |
| Free swelling index (FSI) | units | 6.5 | 6.5 | 5.5 | 5.5 | 6 | 6.5 | 7.5 |
| Gray-King coke type (G) | type | G6 | G5 | G4 | G4 | G5 | G6 | G7 |
| Gieseler max fluidity ($F_{max}$) | ddpm | 250 | 100 | 100 | 100 | 150 | 300 | 1400 |
| Roga index (IR) | units | 50 | 50 | 45 | 45 | 50 | 55 | 65 |

In the course of tests and studies of Apsatsky coals it was demonstrated that their dense medium benefication using flotation enables us to produce high quality concentrate at more than 74.0% yield and less than 9.6% ash. When Apsatsky coal concentrates were used at Russian coking plants we managed to obtain blast-furnace and foundry coke of export quality with high performance capability.

## COMBINED CHEMICAL-BENEFICIATION PROCESSES OF VALUABLE COMPONENTS EXTRACTION FROM COAL BURNING WASTES

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**ABSTRACT**

This paper demonstrates the variety of the composition and properties of the ash and slag combustion products depending on coal type, grade, and conditions of combustion. Typical integrated processing methods of coal ash from Russia's different regions are presented based on concentration technology: collection of aluminosilicate microspheres; flotation of the unburnt coal organic mass; magnetic separation to produce iron concentrate; flotation of the aluminosilicate product of non-magnetic fraction; gravity concentration recovering valuable trace elements into the heavy fraction and the aluminosilicate product into the light fraction; chemical treatment to recover various metals (germanium, gallium, scandium, gold, etc.). To recover gold from stack emissions and ash, an experimental process is proposed for liquid-phase condensation of volatile gold from stack emissions by live steaming followed by purification of the solution and sorption of gold by activated carbon. Process for gold capture from stack emissions is proposed using solid-phase condensation.

**Keywords:** coal, combustion, ash and slag waste, fly ash, recovery of valuable components, gold, chemical concentration processes

### 1. Introduction

Russia's energy industry is becoming increasingly focused on the use of lignite and coal both as an energy source and as a multiple use resource. Integrated use of mineral raw materials and the related problems involved in creating environmentally friendly technologies for combined processing of solid fossil fuels are most relevant and extremely challenging in practice [1]. The use of combined chemical concentration processes for recovery of valuable components from coal combustion products can simultaneously solve waste disposal and environmental protection problems, satisfy demand for raw materials in a number of industries, and reduce the unit cost of production.

The majority of waste management approaches still insufficiently utilize the significant energy and chemical potential of the solid wastes of coal production and preparation and the mineral components of coal processing waste. Many coal deposits in Russia have elevated concentrations of various valuable elements, such as gallium, titanium, zirconium, hafnium, rubidium, niobium, gold, silver, rare earths, etc. Coal from such deposits is typically used for power generation.

To date, no coal deposits have been found where valuable elements (except for uranium) can be cost-effectively recovered without the coal's energy potential being exploited first. Consequently, burning of coal to produce energy, fly ash, and slag rich in valuable elements is one of the methods of combined thermochemical beneficiation of coal allowing to concentrate in the respective processing products the rare, precious, and other metals contained in the coal.

Ash and slag waste and stack emissions formed during combustion are a major environmental hazard. However, they are at the same time a non-conventional mineral and raw material base of rare and trace elements, noble and ferrous metals, as well as raw materials for the construction industry and other products, as repeatedly demonstrated in studies by Russian researchers.



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Efficient recovery of precious metal compounds, rare elements, and other valuable components from coal combustion waste products and preventing the release of atmospheric pollutants requires a scientific approach to selecting efficient gas dynamics and temperature conditions of coal combustion in order to maximize the concentration of mineral components in the respective products of thermochemical processing, as well as efficient methods of the subsequent recovery of valuable components.

## 2. Recovery of valuable components from the ash and slag waste

Russia's coal-fired thermal power plants (TPP) have accumulated in their ash dumps more than 1.7 bn tons of ash and slag waste and produce 30-40 mn tons annually. Until recently, they were viewed primarily as a source of environmental pollution, despite the fact that ash and slag waste is a valuable and renewable anthropogenic resource. Aluminosilicate microspheres (ASM), silica ($SiO_2$), alumina ($Al_2O_3$), ferrous, rare, rare earth, and noble metals, as well as the organic unburnt coal fraction are of greatest commercial relevance.

Analysis of the morphostructural characteristics and chemical composition of the mineral phase indicates that most of the minerals pass into ash without undergoing significant changes [2]. Ash and slag waste is composed of crystalline, glassy, and organic material. The crystalline material is represented by both primary and newly formed minerals. The newly formed phases are predominantly concentrated in the fine fraction of ash and represented by oxide and sulfate compounds of the main elements of ash Si, Al, Ca, Fe, Mg, Na, K, S.

Commercial concentrations of various valuable elements have been in the ash and slag waste in all major coal mining regions of Russia. For example, in the Kuznetsk basin region, 10 elements with following concentrations (%) in the ash and slag waste are considered to be the most promising for the initial assessment purposes: Al – 12.0-19.3; Ti – 0.3-9.7; Fe – 6.0-25.3; Y – 0.01-0.07; Zr – 0.1-1.8; Nb – 0.01-0.03; rare earths total – 0.03-0.3 and U – 0.003-0.01; Au – 0.2-27.2 (g/t); Ag – 10-387 (g/t) [3]. Given the fact that Al, Fe, and rare elements can be recovered jointly, 25-35% of mineral components can be recovered from the anthropogenic raw material stock at a high commercial performance level. Recovery of iron from ash and slag waste can be of commercial relevance at grades of 7.5% or higher.

In Russia, management of ash and slag waste involves their partial use, mainly in the construction industry. In 2000-2011, annual utilization of ash and slag waste never exceeded 7-10% of production. In recent years, there has been a tendency to higher utilization. More than 300 processes for ash and slag waste recycling and utilization are known, mainly as a direct replacement of natural materials in construction and as a raw material in the production of building materials. Direct utilization of ash and slag waste in construction is prevented by the high content of unburnt carbon (5%) and the complex particle size distribution.

The majority of other countries recycle a large share of the ash and slag waste, classified as a by-product of thermal power plants. For example, Denmark has achieved 100% utilization of ash and slag waste, Poland, Germany, China, and UK recycle up to 70-85%, while India recycles 50%.

In the future, the problem of ash and slag waste recycling can be solved before it arises - for example, if in coal preparation metals and sulfur components are recovered from the mineral fraction.

Irrespective of how the mineral fraction of coal is utilized, the diversity of its components should be considered. Ash and slag waste contains up to 150 minerals. As a consequence, the chemical nature, physical, and therefore, technical properties of ashes may differ. For example, fly ash resulting from the combustion of lignite from the Kansk-Achinsk basin has following variations in the chemical composition: $SiO_2$ - 2-40 %; $Al_2O_3$ - 8-11 %; $Fe_2O_3$ - 10-15 %; $TiO_2$ - 0.6-0.8 %; CaO - 25-50 %; MgO - 2-4 %; $SO_3$ - 1-3 %; alkali - up to 2 %. These ashes have a great potential in the construction industry, because they contain a high share (50%) of ash binder.

Ash resulting from the combustion of coal, especially coal from the Kuznetsk basin, has a completely different composition and products. Analysis of the available research shows that the ash from the ash dump site of Kazan TPP-2 resulting from the combustion of Kuznetsk coal grade T is a silica (acidic) ash with a high content of $Al_2O_3$ and $Fe_2O_3$, therefore using it in the production of aluminum sulfate and alumina can be an option. [3]

Using coal ash in the construction industry without pre-processing is an unaffordable luxury. This is due to the more complicated chemical composition and particle size distribution of coal fly ash, which, in addition to rare, noble, and ferrous metals, contains microspheres of different densities, sizes, and color. Coal ash composition may include: lightweight ASM - 1-2%; basic microspheres ($SiO_2$) - 60%; microspheres containing mullite - 25%; coke - 1 to 7%; magnetite in the spherical form - 3-5%; other minerals - 2%.

Valuable components are recovered from ash and slag waste using various combined processes. Conventionally, depending on the ash composition and goals pursued, ash and slag waste processing may include following processes sequenced appropriately:

- collection of ASM floating in an aqueous medium;
- separation of the unburnt coal organic material by classified ash flotation (separation in air stream or electrostatic separation may be applied);
- magnetic separation to produce iron-containing concentrate;
- recovery of the aluminosilicate product from the non-magnetic fraction by flotation;
- gravity concentration (screw separator, concentration table, etc.) with rare elements in the heavy fraction and the aluminosilicate product in the light fraction;
- chemical treatment to produce rare and trace elements and noble metals (germanium, gallium, scandium, yttrium, rhenium, gold, etc.).

Numerous studies have shown the good prospects of recovering unburnt coal from the ash and slag waste by flotation with preliminary recovery of ASM. Flowsheets have been designed, equipment selected, and optimal recovery conditions proposed for the fly ashes from a number of Russian thermal power plants [4]. Mineral fraction with a carbon content of under 5% and coal concentrate (with a calorific value of 4000-4500 Kcal/kg) are recovered. When gold-bearing coal is burnt, the concentrate may have an elevated gold content. For example, coal concentrates produced by flotation of ash left after combustion of coal from deposits in the Far East had a gold content of 0.6-4.4 g/t. [5]. These coals should be burnt in special furnaces to recover the gold.

To process fly ash after wet ash handling at TPP-9 operated by OAO Irkutskenergo owned by OOO Spirit, a flowsheet was proposed that included wet magnetic separation, screw separation of the magnetic separation tailings, gravity concentrate finishing on a concentration table and in an electromagnetic separator. Heavy fraction containing valuable trace elements and light fraction containing aluminosilicate product (feedstock in alumina production) were obtained. High iron recoveries were achieved - 83.07% at $Fe_{total}$ grade of 60.83%.

Ash from Kashira TPP burning coal from the Kuznetsk basin, containing more than 8% of iron, was used to demonstrate the possibility of producing iron concentrate in one process stage with recovery over 50% at iron grade of 60-70%. That high performance was achieved by applying a combined process in an electrodynamic belt separator with rotating drum magnetic system of permanent magnets with alternating poles [6].

Based on the principle of deep and complete processing of the ash to produce a wide range of commercial products, the Joint Institute for High Temperatures, Russian Academy of Sciences, developed an innovative technology for processing high alumina ash resulting from the combustion of Ekibastuz coal at Troitsk TPP (South Urals) composed of following stages: classification by size, flotation of the unburnt coal in size fraction +45-200 microns, magnetic separation of the flotation tailings, hydrothermochemical processing of the nonmagnetic ash residue - alumina concentrate. Aluminosilicate product (70-80-% $Al_2O_3$) is treated with an alkali to produce alumina and sodium silicate solution. The latter is treated with lime to produce white slurry - belite (Class A) - and an alkali solution. Alumina is sintered with limestone and then leached with sodium carbonate to produce aluminate solution and gray sludge - belite (Class B) - for use in the cement industry. After desilication, the aluminate solution is fed to the carbonation plant to produce aluminum hydroxide using the conventional alumina process. [7].

Any examination of the process of recovery of potentially valuable elements from Russian thermal coal should be guided by the following basic principles:

- Rare element content in ash and slag must not be lower than in the conventional head ore;
- Commercial rare element compounds should be recovered from the coal with the maximum possible utilization of the energy content of the coal;
- Ash and slag resulting from coal combustion and containing rare elements should be used as a standalone raw material or a component of the head ore charge, where possible, at existing plants producing commercial rare element compounds.

Studies by many researchers have shown that fly ash has a 2-5 times (depending on the furnace's slag separation ratio) higher concentration of sublimated rare element compounds (Ge, Ga, Mo, W, Re, Nb, Hf, Zr, etc.) and noble metals (Au, Ag) forming gaseous compounds under oxidizing conditions at high temperatures. Concentration of rare elements not forming volatile compounds increases in the slag 1.2-1.8 times. Valuable elements in the fly ash are potentially present as oxygenates. Seredin showed the correlation between rare earth elements and humic substances in coal [8]. When coal is burned, rare earth elements in the ash and slag waste may be present in a dispersed state and their concentration is 2.5-4 times higher than in the coal itself. At concentration, rare earth elements and rare elements accumulate in the heavy fraction.

A distinguishing features of ash and slag waste is the low content of rare elements contrasted with the high content of silicon, aluminum, iron, and calcium compounds formed at high temperatures (1200-1700°C) and therefore chemically passive. This makes it necessary to use acidic reagents, longer residence times, higher temperatures, and special technology to recover rare elements.

Recovery of germanium from coal is the subject of many studies [8]. To achieve a high germanium content in the product, it is recommended to burn coal in furnaces with a high slag separation ratio (layered, cyclone) and gasification of germanium. At low metal grades, secondary concentration of the captured fly ash is recommended. It is smelted with addition of fuel and limestone at 1500-1600°C. Secondary gasification of GeO occurs and the concentration of the metal in the fly ash increases 30-40 times compared to coal. Concentrate containing more than 0.2% of the metal can be treated with a mixture of hydrochloric and sulfuric acids. Gallium can be recovered in parallel in the same process. However, studies have shown that direct hydrochloric acid leaching of gallium from ash and slag waste is more efficient.

In direct liberation with hydrochloric acid, up to 75-85% of scandium can be recovered. However, problems arise with corrosion of process equipment and the need for removal of impurities from process solutions containing scandium. Therefore the possibility was investigated of sorption leaching of scandium from the ash of burning at 750-800°C coal from Irsha-Borodino coal mine. It was found that recovery of scandium by sorption using cation exchanger KU-2-8 is up to 85% [9].

**3. Recovery of gold from the combustion products of coal**

Considering recovery of gold from coal combustion waste products, it should be noted that their content in coal coming from same deposit varies very widely [10]. Many believe that the average gold grade of coal is underestimated because special preparation of samples for analysis is required to collect reliable data.

Kuzminykh at AmurNC developed an assay method that preserves gold in the sample [11]. Using that method, gold grades higher by 1-2 orders of magnitude were identified in more than 3000 coal samples. Research Institute of Comprehensive Exploitation of Mineral Resources, Russian Academy of Sciences, also proposed a method of microwave decomposition of coal samples under pressure producing reliable data. [12]. Using neutron activation analysis, Seredin measured the content of Au and Ag in the ash left after burning coal from Pavlovsk deposit, ranging from 3-4 to 50 g/ton of ash. In all tests, the results were influenced by representativeness of the sample used.

Numerous studies have found that coals can be involved as feedstock in commercial production of gold, whose content in coals from individual regions of occurrence (Far East, Kansk-Achinsk basin, Kuznetsk basin, etc.) ranges from one to several grams per ton. Thermal lignite has, as a rule, a higher gold grade. It was found that in the Far East lignites gold is represented by mineral particles of native high-grade gold and solid solutions of Au-Ag, Au-Ag-Cu, Au-Cu (Ni, Zn) [2]. In Kansk-Achinsk lignite, gold is also found as micro and nanoparticles of solid solutions of Au-Cu-Zn, Au-Cu-Ag, Au-Cu-Zn-Pb with a gold grade of more than 80 wt%. Due to this kind of deportment, recovering gold from the coal by conventional methods is technologically impractical and uneconomical. Using direct cyanidation, especially in lignite, is complicated by the interaction of the coal in the solution with the alkali and active sorption by coal of the forming cyanide complex.

Currently, two approaches to the behavior of gold under combustion of coal are discussed in the literature. The first one is based on accumulation of the metal in the ash and slag waste. The second one is based is based on the fact that gold is easily carried away with flue gas. Results of experiments conducted jointly by Research Institute of Comprehensive Exploitation of Mineral Resources, Russian Academy of Sciences, AmurNC, Far East Branch, Russian Academy of Sciences, and OAO SUEK on lignites from Kansk-Achinsk basin showed that in combustion of coal more than 80% of gold passes into the flue gas. Evidence that gold particles found in the flue gas are not redeposited is their morphostructural characteristics and chemical composition identical to those of the particles found in the coal. Size, loose spongy structure of the gold particles, and bonds between the gold particle and the organic matter contributes to their entrainment in the fly ash [2]. The rest of the gold can completely pass into gaseous state at a temperature above $1000^{o}$, as evidenced by the thermodynamic analysis undertaken by the Research Institute of Comprehensive Exploitation of Mineral Resources, Russian Academy of Sciences [13].

The second approach is supported by the findings of the joint study by Research Institute of Comprehensive Exploitation of Mineral Resources, Russian Academy of Sciences, and AmurNC, Far East Branch, Russian Academy of Sciences, showing recovery of gold from the stack emissions from the combustion of Kansk-Achinsk coal in a lab furnace at AmurNC. Studies have demonstrated the possibility of recovery of gold averaging approximately 40% [14]. A pilot process was developed and a pilot plant was built for liquid-phase condensation of volatile gold from stack emissions by live steam followed by purification of solutions and sorption of gold by activated carbon [15].

Research Institute of Comprehensive Exploitation of Mineral Resources, Russian Academy of Sciences, is developing a process for capturing noble metals from stack emissions using solid-phase condensation, avoiding the high-cost liquid-phase condensation of gold [16]. The gold content of the leach cycle feed is 5-8 times higher than that of the head coal.

**Conclusions**

Thus, it can be concluded that the determinant factors of valuable element concentration into solid or volatile compounds and their subsequent efficient conversion to condensed state, controlling downstream commercial products, are the optimal combinations of gas dynamics, temperature, and redox conditions of the thermal treatment of coal.

It is shown that various combined processes have been proposed and developed in Russia for recovery from ash and slag waste of microspheres, iron and aluminosilicate products, rare and noble metals, building materials, and other valuable components, whose properties are determined by the composition of the coal combustion waste products.

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# Coal in the economy of Russia

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**Key words**: Russian coal industry, deep processing

## Summary

The report analyses the situation, issues and directions for development of Russian coal industry from the position of understanding of its role and meaning in the country economy as well as with consideration of problems of improvement of consumed coal quality including such processes as coal preparation and deep coal processing.

In general the future of coal in the world is defined by its competitiveness. Being the cheapest type of fuel (in comparable terms) coal is surely going to be on demand, especially when we talk about developing economies.

The future of coal in Russia is connected with modernization of physical resources not only in the industry but first of all in coal consumption. Development of coal consumption technologies, which includes expansion of high quality coal, adoption of coal gasification (technological restructuring), is one of the main tasks in the foreseeable future.

Improvement of quality of coal that the markets are supplied with shall contribute to upturn in coal demand both in the world and in Russia. Development of coal preparation, new technologies introduction of its deep processing will help neutralize risks associated with high cost of its transportation and negative effect that coal consumption causes to the environment.

The issue of extension of local use of coal is quite challenging, especially by means of development of chain of small municipal coal thermal power plants (regionalization of coal).

The economic recovery for Russia should be closely connected to optimization of country fuel balance, while commitment to one fuel resource (gas) is jeopardizing the energy security of the country.

## 1. Introduction

At the moment the world energy market has faced the period of economic glut which is naturally followed by fall in prices. The competitive struggle between certain types of energy resources escalates and that in turn leads to worsening of the political environment; the strategical goals of the major regional association of states are being reassessed. Both the regions that possess valuable energy resources and importing countries revise their taxation and stimulating programs.

At this level any reference points or predictive assessments of the demand for coal become ambiguous. Even more uncertainty is caused by such events that have been going on in the leading coal mining countries as 'shale revolution' in the USA [1] and 'demonization' of coal in Australia [2].

International Energy Agency (IEA) gives two polar scenarios of the change of coal demand. According to the «scenario of current policy (SCP)»[1] by 2040 the demand for it can grow by over than 50 %, and according to the «scenario 450»[2] it shrinks by 33 % [3].

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[1] The scenario is based on prolongation of the trends of fuel and energy resources consumption
[2] The scenario foresees principal reduction of greenhouse gases concentration in the atmosphere to the level of 450 particles per million (ppm) of $CO_2$ equivalent

 
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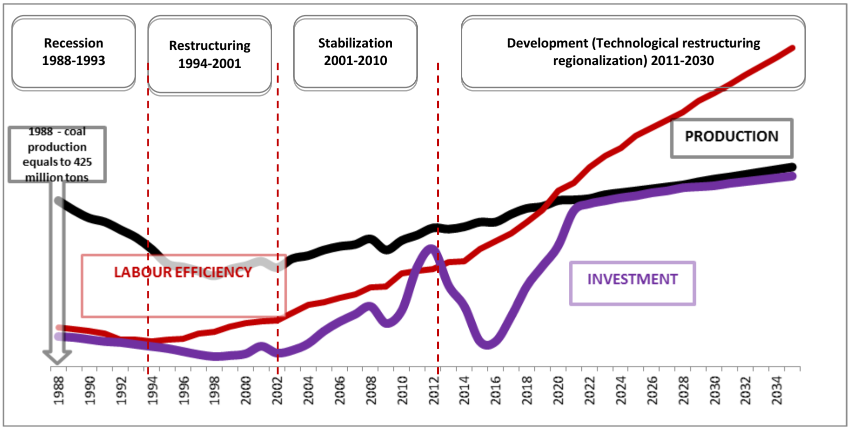
## 2. Stages of coal industry development in Russia.

For Russia, which is the major coal developing industry in the world, it is crucial to pick up the right strategy of further coal industry development.

To that end it is instructive to recall the four stages of coal industry development during the post-Soviet period [4]. The most important of them is the restructuring period.




Picture 1 – Stages of coal industry development in Russia

Coal industry is the only economic branch of Russia that has undergone restructuring according to the standards of the World Bank and International Monetary Fund through the offices of the 'Know-How' fund (Great Britain) and the Government of the Russian Federation during the last 25 years of its contemporary history [5].

As a result in the period of 1995 - 2000, i.e. during 6 years, 188 mines and 15 opencast mines were closed, 100 million tons of ineffective coal depths were dropped out. All the collieries were reincorporated as joint-stock companies and privatized.

After structural reforms the industry stepped into the stage of steady development. During the last 10 years the volume of coal production has increased by about one-fourth and by the end of 2015 is going to exceed the level of 370 million tons per year. At that due to the structure of reforms and introduction of new technologies the labor efficiency of the miners has grown fourfold. The proportion of open-cut mining, which is considered to be cheaper and safer, new has exceeded 70%.

At the moment Russian coal industry counts 124 opencast mines, 70 mines with total productive capacity of over 400 million tons of coal production per annum.

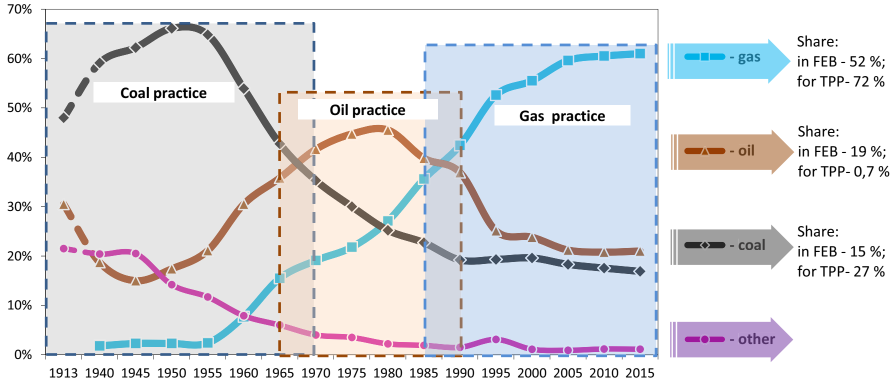
The processing segment of the industry consists of 66 coal-preparation plants and hundreds of siftings. The coal- preparation plants manufacture over 200 million tons of product per annum.

Most part of the assets in the coal industry are managed by 20 major holding companies providing 85 % of total production. The fields of the most of holdings and asset management companies are situated in Kuzbass.

## 3. Coal in the economy and fuel and energy budget of Russia.

The role of coal industry in the country GDP cannot be named significant. The annual volume of the commodity output produced by the branch equals to only 0,6 % of Russia`s GDP and to 1 % of the industrial production. At that the share of the branch in these macroeconomic results does not grow, which is quite understandable for Russia - the largest producer of the most effective gas fuel.

The domestic market is not the growth driver for the coal branch, though 53 % of Russian coal is used by the domestic market. I would like to remind how the consumption of coal is formed in Russia. During the last hundred years there`s been three fuel practices – coal, oil and gas [6]. Starting from the 1980s coal in Russia has been the closing type of fuel in the fuel and energy balance (FEB) of the country. In the beginning of the 21st century in the circumstances, when gas fuel has taken leading position because of low regulated gas prices, coal does not find market domestically and is being exported in increasing volumes.




Picture 2 – Change of practices in the fuel and energy balance of Russia

In the structure of the fuel and energy balance of Russia the shares of gas, nuclear and hydro power are growing, while the share of oil and coal is shrinking. Thus the share of coal in the structure of consumption of fuel by Russian thermal power plants (TPP) now equals to 27% (to compare with 30,5% in 2000). At that the share of gas has grown from 64 to 72% within the same period.

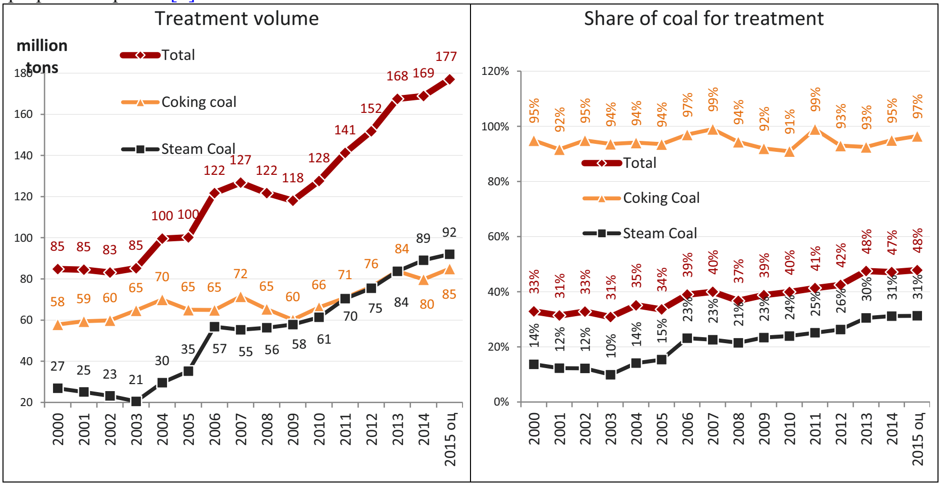
Substantial reserves of gas and favorable prices of it contribute to the fact that this type of fuel is predominantly used nowadays. One should note that since 2005 gas in Russia (in comparable units) has been more expensive than coal and for now the gas/coal price ratio equals 1,7 to 1,0. For the coal to become competitive this ratio should exceed 2,5 to 1,0.

## 4. Export market as the main driver of coal preparation growth

Export of competitive at the world market Russian coal has become the only direction of forming a sustainable model of development of Russian coal industry. For the period from 1995 to 2015 the export flows have increased from 21 million tons to 159 million tons. From the beginning of 2000th coal companies demonstrated high rate of consolidated profit formation which lead to increase of investments in fixed capital. The coal mines ‘arose like phoenix from ashes’ (from 267 million tons to 408 million tons).

The trend for export of the coal business has formed commercial interest for coal processing (especially steam coal). High demands to the quality of products on the export market influenced the pre-sales preparation of coal. Selective extraction, sorting and preparation of coal became an indispensable part of improvement of consumer properties of the coal for export. At the moment Russian miners use modern coal preparation plants, the major 10 of the plants treat 44 % of all coal in the country. Moreover Russian exporters established themselves as reliable suppliers.

Since 2000 the volumes of preparation of Russian coal have grown more than twofold (from 84 million tons to 177 million tons). At the moment Russian coal industry treats 47 % of all the run-of-mine coal. On average there are 2-3 preparation plants put into operation, there has been 12 preparation plants put into operation during the last 5 years. Most part of the new facilities are the power plant coal preparation plants [7].



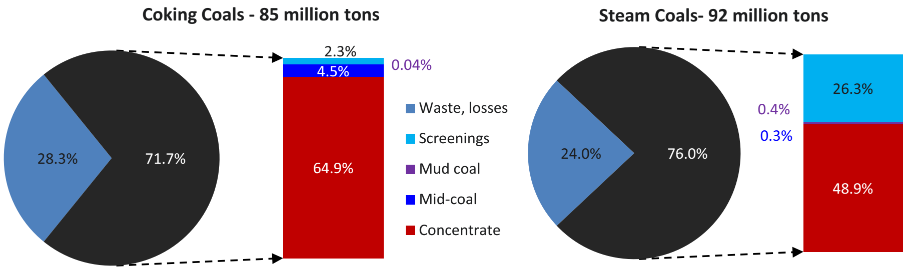
Picture 3 – Dynamics of coal treatment in the coal treatment plants of Russia

Almost all types of coking coals undergo treatment and only 31 % of steam coal undergoes treatment. More than half of coking coal concentrate is consumed by the domestic market, the rest is exported.

Unfortunately the equipment of domestic power plants is not designed for usage of prepared coal. As a result the domestic market (thermal power plants) uses mostly brown coal (about 60 %), common black coal and low quality washings, i.e. mid-coal, screenings and mud-coal [8].

Still Russian colliers are facing a task of further facility building for sorting and preparation of steam coal. Not only will this help to keep, but also to expand the niche of Russian coal on the external market, first of all on the market of the Pacific Rim, where our share can be increased by 2030 from 7% that Russia holds now up to 10%.

Still there is a big internal issue Russia faces concerning sales of illiquid products and utilization of coal prepared wastes the annual volume of which reaches over 40 million tons. During the coming 15 years the wastes of coal preparation are going to increase at least twofold and equal 82 million tons. The problem of usage of the illiquid products of coal preparation can be solved by using coal gasification technologies and the scope of coal chemistry.



Picture 4 – Treatment and coal-preparation production in Russia for 2015 (assessment), million tons

It is necessary to stimulate introduction of progressive technologies of coal generation, for that will contribute to the increased use of high-quality coal fuel in the functioning thermal power plants.

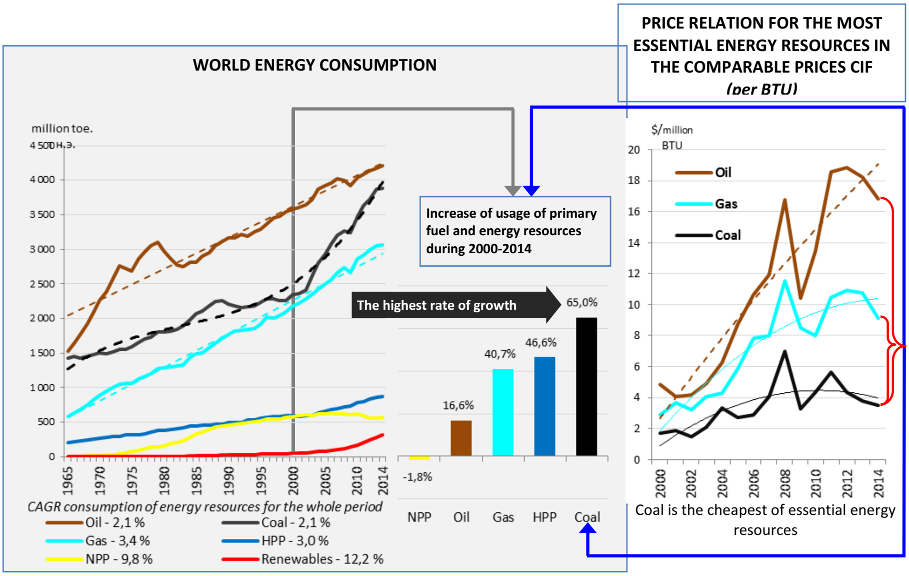
## 5. Perspective of Russian coal on export market

Today coal is still the cheapest and most demanded fuel. From 2000 the volumes of coal production have grown 1,7 times. The volume of international coal trade has grown rapidly (by 233%) and now equals to 1,4 billion tons (of which 300 million is formed by coking coal) [9].

The Pacific Rim countries are still dominating here. The top coal mining nations today are: China with over 3,7 billion tons, India - over 670 million tons, Australia – 490 million tons, Indonesia – 470 million tons per year.

Russian coal companies do not stay on the sidelines. Several large scale projects aimed at the development of new coal mining areas have been started in Sakha Republic (Yakutia) and Zabaikalskiy region. The Program for development of coal industry of Russia approved by the Government of the Russian Federation submits 48 investment projects to the implementation, of which 21 projects concern the regions situated in East Siberia and Far East [10]. The maximum amount of run-of-mine coal in these regions can be reached by 2030 and equal 150 million tons. The largest new coal mining centers are Elginsky coal mine (30 million tons) and coal mine «Inaglinsky» (12 million tons) in Sakha Republic (Yakutia). Apsatsky black coal deposit (Zabaikalskiy region) is under development now, there are plans for complex development of Erkovetsky coal deposit (29 million tons according to intentions of the investor «Inter RAO») and Gerbikano-Ogodzhinsky coal bearing district (30 million tons – «RT-Globalnye resursy") in Amur region. These projects are directly connected with coal mining and will be provided with enrichment facilities.

The adoption of the federal law aimed at creation of favorable tax conditions to carry out investment activity on the Far East and in East Siberia is to ensure the intended shift of coal mining center to Far East and East Siberia and enhance economic efficiency of coal export.




Picture 5 – Coal in worldwide energy consumption

As a result the export of coal from Russia can increase by 2035 up to 225-270 million tons. At that the correlation between western and eastern directions will change. The eastern direction will show growth, while the supply to the Atlantic market will either stay on the current level or even shrink. Only the corresponding development of coal preparation can ensure such growth.

## 6. Conclusion

In general the future of coal in the world is defined by its competitiveness. Being the cheapest type of fuel (in comparable terms) coal is surely going to be on demand, especially when we talk about developing economies.

The future of coal in Russia is connected with modernization of physical resources not only in the industry but first of all in coal consumption.

The problem of further growth of coal demand both in Russian and abroad can be solved by increasing of the quality of coal supplied on the market. Development of coal preparation, introduction of new technologies of its deep processing will help neutralize risks associated with high cost of its transportation and negative effect that coal consumption causes to the environment.

The perspectives of the use of coal on Russian domestic market are connected with improvement of the technologies of coal consumption, including wider usage of high quality coal, implementation of coal gasification technologies (technological restructuring).

The issue of extension of local use of coal is quite challenging, especially by means of development of chain of small municipal coal thermal power plants (regionalization of coal).

The economic recovery for Russia should be closely connected to optimization of country fuel balance, while commitment to one fuel resource (gas) is jeopardizing the energy security of the country.

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# Desulphurization of high-sulfur coking coal by microwave irradiation assisted with alkali solution

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**Abstract:** A desulfurization method that by using microwave radiation assisted with alkali solution was applied to a high-sulfur coking coal collected from Shanxi Province, north of China. First, the single factor experiments were involved to get the optimal microwave desulfurization conditions. Then, several methods including X-ray photoelectron spectroscopy (XPS), sulfur K-edge X-ray absorption near edge spectroscopy (XANES) and Fourier transform infrared spectroscopy (FTIR) were used to investigate the changes in the relative contents of sulfur forms and main organic groups in coal before and after desulfurization process. The single-factor experiment results showed that the optimal microwave desulfurization conditions were: particle size of -0.074 mm, additive concentration of 6mol/L, microwave power of 900 W, irradiation time of 8 min and solid-to-liquid ratio of 1:4. And the total desulfurization rate was 25.23% under the optimum experimental conditions. XPS and XANES analysis showed the decline of mercapten (sulfother), thiophene and sulfides, while the contents of sulfone, sulfoxide and sulfates increased. The FTIR spectra demonstrated that the overall organic structure of coal hardly got destruction after microwave desulfurization. In addition, the caking index (*G*) declined from 76 to 62, which suggested the microwave radiation combined with NaOH solution weakened the cokability of coking coal slightly.
**Key words:** Coal desulfurization; Microwave; Alkali solution; Sulfur form; Caking property; XPS; XANES; FTIR

## 1 Introduction

Coal is the most economical source of energy in China. However the emission of $SO_X$ after coal combustion is not only toxic to the human body, but also the cause of acid rain. So the removal of sulfur in coal has recently become even more critically valuable, especially considering the most of workable coal deposit is low-grade coal with high ash and high sulfur.

With the particular work mechanism, microwave irradiation has been widely used in fields such as organic synthesis[1], environmental engineering[2,3], mineral processing[4,5] and petroleum upgrading[6]. There have been several attempts to remove the sulfur in coal via microwave irradiation[7,8]. And it is believed that the use of microwave in the coal desulfurization has tremendous potential and significant prospects in the future[9-11].

In present paper, a high-sulfur coking coal was adopted in the desulfurization experiment, which was carried out by using microwave radiation combined with NaOH solution. Several variables, including particle size of coal, microwaving conditions, concentration of alkali solution and solid-to-liquid ratio, were optimized. The transformation of the chemical forms of sulfur before and after the process was detected via XPS, sulfur K-edge XANES. In addition, the FTIR test was manipulated to investigate the impact of the desulfurization method on the structure of coal. And its effect on the caking properties of coal was investigated by comparing the caking parameters before and after microwave treatment.

## 2 Experimental

### 2.1 Material

The high-sulfur coal sample was collected from Xinyu mine, Shanxi province, northern China. After ground to -1 mm at nitrogen atmosphere and stored in a sealed bag. The proximate analysis, ultimate analysis and sulfur speciation determination was carried out, according to national standards of China (GB/T 212-2008, GB/T 476-2001 and GB/T 215-2003). And the results are shown in Table 1.

### 2.2 Desulfurization experiments



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The desulfurization experiments were performed in the microwave reactor (MAS-II, operated at frequency 2.45 GHz, maximum power 1000W) manufactured by Sineo Microwave Chemistry Technology (Shanghai) Company.

**Table 1** Proximate analysis, ultimate analysis and sulfur distribution of coal samples

| Coal sample | Proximate analysis | | | | Ultimate analysis | | | | | Sulfur distribution | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | $M_{ad}$,% | $A_d$,% | $V_{daf}$,% | $FC_{daf}$,% | $C_{daf}$,% | $H_{daf}$,% | $N_{daf}$,% | $S_{daf}$,% | $O_{daf}$,% | $S_{s,d}$,% | $S_{p,d}$,% | $S_{o,d}$,% | $S_{t,d}$,% |
| Raw coal | 1.33 | 10.74 | 27.50 | 72.50 | 82.49 | 4.24 | 1.52 | 2.60 | 9.15 | 0.22 | 0.29 | 1.87 | 2.38 |
| Raw coal -0.074 mm | 1.58 | 11.29 | 28.46 | 71.54 | 80.83 | 4.95 | 1.69 | 2.93 | 9.60 | 0.20 | 0.47 | 1.91 | 2.58 |
| Treated coal | 1.65 | 10.56 | 26.67 | 73.33 | 78.51 | 3.46 | 1.68 | 2.15 | 14.20 | 0.42 | 0.11 | 1.45 | 1.98 |

For each experiment, a certain amount of coal and alkali solution is loaded in a round-bottom flask (250mL) at first. Then put the flask into microwave reaction device and regulates irradiation time and power. At the end of each experiment, the reacted coal samples are cooled and washed with 80°C deionized water repeatedly in a filtration, then recovers, dries in vacuum oven for 4h in ca.105 °C and conserves for further analysis. The desulfurization experiment flow scheme is shown in Fig.1.

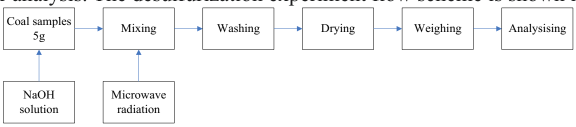


**Fig.1** Experiment flow scheme of microwave desulfurization

## 2.3 Instrumental analysis on coal

The XPS analysis was operated on X-ray photoelectron spectrometer (ESCALAB250Xi, America) that equipped with Al Kα (1486.6 eV) source at room temperature. The calibration for binding energy was carried out to the main C1s peak at 284.80 eV. The peak curves were resolved using Gaussian line shape with XPSpeak4.1 software.

Sulfur K-edge XANES analyses were accomplished on beamline 4B7A at Beijing Synchrotron Radiation Facility (BSRF). The sulfur XANES spectra of all coal samples were measured using the fluorescence yield mode, whereas those of reference samples were measured in the total electron yield mode. The absolute energy position of the sulfur K-edge white line of calcium sulfate was calibrated to 2481.8 eV, and the energy resolutions was 0.20 eV.

Specimens for FTIR were prepared using the potassium bromide (KBr) pellet technique. FTIR spectroscopy analysis was carried out on a Nicolet 380 spectrometer, equipped with a DTGS detector. Spectra were recorded by coadding 32 scans at a resolution of 4 $cm^{-1}$. The infrared signal was recorded in the range from 400 to 4000 $cm^{-1}$ wavenumber.

# 3 Results and discussion

### 3.1 Effects of several factors on coal desulfurization rate

General conditions applied in microwave desulfurization in coal were used for reference from earlier investigation[10]. Based on the earlier research, the desulfurization experiments of coal samples with particle size of -0.074mm, 0.074~0.125mm, 0.125~0.250mm, 0.250~0.500mm and 0.500~1.000mm performed at first, and the results are presented in Table 2.

The improvement of desulfurization performance came into being with the decrease in the coal granularity. One might have expected that the evolution of desulfurization performance with the decrease of coal granularity because of the extend of the surface area per unit mass of coal particles and the progress of the motility of sulfur-containing chemicals generated in the desulfurization process or existing in raw coal inherently. The effect of the extent of surface area on the microwave desulfurization is the acceleration to the sulfur-containing compounds in coal to contact NaOH solution, and raise the probability of reaction between them. Additionally, the small particle size may be in favor of the emitting of desulfurization resultants, such as gaseous $H_2S$ and COS, even $SO_2$.

Concentrations of NaOH solution were regulated to 2 mol/L, 3 mol/L, 4 mol/L, 5 mol/L and 6 mol/L, and the rest of conditions were particle size -0.074 mm, applied power 900 W, irradiation time 8 min and solid-to- liquid ratio 1:4. The experiment results are shown in Table 2.

It is clearly to be seen that the general rule of the effect of the NaOH solution concentration on the desulfurization performance in coal. That is, the desulfurization rate increased with the increasing of the alkali liquor concentration. In detail, the desulfurization rate was 5.87% with the sulfur content in clean coal was 2.47% when the 2 mol/L NaOH solution was used in the experiment. As the concentration of alkali solution increased from 2 mol/L to 5 mol/L, almost a linear increase trend was observed in the desulfurization rate. According to the development trends in the desulfurization rate in various alkali solution concentration, a conjecture that the reduction of sulfur in coal may not get obviously enhancement by the increase of the concentration after it exceed 6 mol/L, was brought about.

**Table 2** Experimental results of microwave desulfurization under various conditions

| Variable | Level | Sulfur content of untreated coal, % | Sulfur content of clean coal, % | Clean coal yield, % | Desulfurization rate, % |
|---|---|---|---|---|---|
| Particle size of coal, mm | 0.500-1.000 | 2.19 | 1.89 | 98.73 | 14.74 |
| | 0.250-0.500 | 2.33 | 1.98 | 98.12 | 16.61 |
| | 0.125-0.250 | 2.40 | 2.01 | 97.56 | 18.16 |
| | 0.074-0.125 | 2.45 | 1.94 | 99.02 | 22.43 |
| | <0.074 | 2.58 | 1.92 | 98.52 | 26.83 |
| Concentration of alkali solution, mol/L | 2 | 2.58 | 2.49 | 97.58 | 5.87 |
| | 3 | | 2.34 | 98.32 | 10.94 |
| | 4 | | 2.14 | 98.24 | 18.38 |
| | 5 | | 2.02 | 98.59 | 22.98 |
| | 6 | | 1.95 | 97.96 | 25.83 |
| Solid-to-liquid ratio, g/mL | 1;2 | 2.58 | 2.15 | 98.38 | 18.00 |
| | 1;3 | | 2.00 | 97.56 | 24.20 |
| | 1;4 | | 1.93 | 98.18 | 26.67 |
| | 1;5 | | 2.02 | 97.89 | 23.25 |
| | 1;6 | | 2.11 | 98.61 | 19.24 |
| Microwave power, W | 200 | 2.58 | 2.46 | 99.11 | 5.43 |
| | 400 | | 2.24 | 98.69 | 14.19 |
| | 600 | | 2.11 | 98.24 | 19.68 |
| | 800 | | 1.95 | 98.35 | 25.57 |
| | 900 | | 1.94 | 97.59 | 26.66 |
| Irradiation time, min | 2 | 2.58 | 2.63 | 99.08 | -0.88 |
| | 4 | | 2.30 | 99.26 | 11.36 |
| | 6 | | 2.15 | 98.69 | 17.78 |
| | 8 | | 2.01 | 98.58 | 23.37 |
| | 9 | | 1.95 | 98.84 | 25.16 |

*Yield of clean coal* ( $Y_j$ )= $m_j / m_y$ * 100, %

*Desulfurization rate* =(100* $S_y$ -$R_j$* $S_j$) / $S_y$,%

Where $S_y$ is the sulfur content of untreated coal /%, $S_j$ the sulfur content of clean coal /%,

$Y_j$ the yield of clean coal /%, $m_y$ the mass of original dry coal/g, $m_j$ the mass of clean dry coal/g.

To evaluate the effect of solid-to-liquid ratio on coal desulfurization, a series of tests were implemented with the solid-to-liquid ratio 1:2, 1:3, 1:4, 1:5 and 1:6, respectively.

It can be obtained from the Table 2 that the sulfur content in clean coal decreased with the increase of the solid-to-liquid ratio range from 1:2 to 1:4. And, the optimum solid-to-liquid ratio was 1:4 with the sulfur content in clean coal is 1.93%.

Based on the exploration results above, a study was conducted for effect of variations in microwave powers on coal desulfurization and the microwave powers were 200W, 400W, 600W, 800W and 900W, respectively.

What can be observed from Table 2 is that the desulfurization rate was increasing steadily with the increase of microwave power at beginning stage, and the increase of desulfurization rate was not so remarkable when the power got enough high. To particular, the desulfurization rate was only 5.43% under the power of 200 W. Even there was an increase of sulfur content in treated coal obtained after microwave radiation under the power of 200W in earlier experiment. The reason can be inferred from the content of ash decreased. The minerals in coal can react with the NaOH solution to generate soluble substances, thus lowering the content of ash, while the desulfurization reaction between the solution and sulfur-bearing compounds in raw coal cannot be triggered under microwave irradiation power of 200W. Therefore the extended enlarge of irradiation power is necessary for enhancing the degree of desulfurization. As shown in Table 2, the desulfurization rate increase from 5.43% to 26.66% with the radiation power enlarged from 200W to 900W. Besides, the promotion of the extension of microwave power on microwave desulfurization may not express so noticeable when the power reached so high that made the most of microwave radiation.

The desulfurization results of the experiments that accomplished for 2 min, 4 min, 6 min 8 min and 9 min with the microwave power of 900 W are as shown in Table 2 .

It can be seen that the irradiation time has a convincing effect on the reduction of sulfur in coal by using microwave radiation. The desulfurization rate was a negative number when the irradiation time was set to 2 min. But the desulfurization performance would be improved with the increase of irradiation time, and the desulfurization rate became a positive number and got bigger. It is impractical for energy consumption and economic efficiency that made the reaction time longer, considering a long-time microwave radiation would result in the destruction to coal basis structure and the impairments of coal properties.

**3.2 Analysis of coal samples characteristics before and after desulfurization**

**3.2.1 XPS analysis**

In general, the sulfur in coal was classified into 4 or 5 types for XPS spectra peak fitting[13-15]. The results of S2p XPS peak fitting before and after desulfurization of coal samples are shown in Fig.2.

The sulfur in raw coal mainly existed in the forms of mercaptan, sulfoether and thiophene, which accounted for ac.80% of total sulfur. In addition, 7.52% of sulfur in raw coal also contained as sulfoxide and sulfone, and inorganic sulfates accounted for 12.42%.

All S2p binding energy peak positions show in Fig.2 had not distinct changes after microwave treatment, which verified the fact that the chemical forms of sulfur in coal, especially organic sulfur had not been changed. In contrast, the intensity of S2p peaks reduced remarkably, indicating good performance of microwave with alkaline additive on coal desulfurization. Specifically, the contents of sulfur in the forms of mercaptan and thiophene significantly decreased to 4.64% and 49.48% from 17.91% and 62.16% of total sulfur, respectively, and the proportion of sulfones and sulfoxides increased to 14.70% from 7.52% with the signal of inorganic sulfate strengthening. Most of organic sulfur atoms in raw coal present in the form of $S^{2-}$, containing two lone electron pair and leading to strong electronagativity, and shows intense reducibility so that induces oxidation reduction reaction with oxidizing agent. Mercaptan sulfur and sulfide sulfur (both with low polarity) were easily oxidized to sulfatic sulfur, and the relative content of sulfatic sulfur increasing from 12.42% to 31.18% sharply confirmed the transformation. The relative content of thiophenic sulfur decreased by 12.68% might be attributed to the oxidization of sulfur atom after the breakage of C-S in thiophenic structure caused by microwave process.

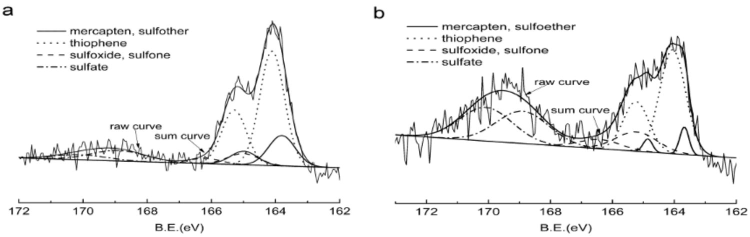


**Fig.2** Fitting XPS S2p spectrum of raw coal (a) and desulfurized coal (b)

**3.2.2 XANES analysis**

In this case, each type of functionality should, in principle, be modeled with one Gaussian centered at the energy of the corresponding s→p photoelectron transmission, one or several broad Gaussian peaks representing the resonance scattering that reflects the average binding environment of sulfur in this functionality. The spectra were normalized and then quantified by the least squares fit using a linear combination of the spectra of reference compounds with the use of Athena program package[16]. And the original testing spectra and corresponding fitting curve of coal samples were delineated in Fig.3.

It can be seen that sulfate, pyrite, thiol, elemental sulfur, thiophene, and sulfoxide accounted for 15.27%, 7.48%, 14.29%,.2.06 %, 55.48%, and 5.32% of the total sulfur in coal before desulfurization, respectively. After desulfurization, the relative content of ferric sulfide sulfur decreased significantly; the sulfur-containing structures in coal were oxidized to the sulfur in oxidation state with a higher valence state, which agreed with the result of XPS analysis. The final product was hexavalent sulfur compound so as to facilitate sulfur removal.

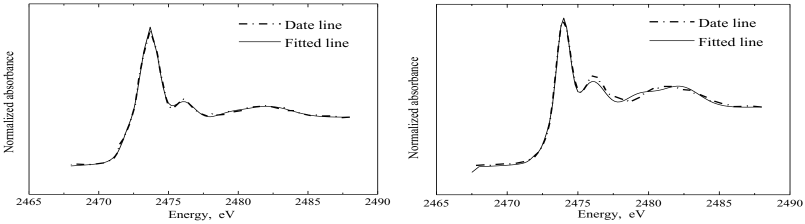


**Fig.3** Sulfur K-edge XANES spectra of coal samples before (left) and after (right) treatment

### 3.2.3 FTIR analysis

Transformation of coal structure taking places in microwaved sample was studied by FTIR spectroscopy as Fig.4. Compared with the raw coal, the main characteristics of the coal structure seemed unchanged after microwaving treatments. The peaks at 3600~3200 $cm^{-1}$ interpreted as O-H bonds in the mineral and organic matter. Intensities of the bands at 3000~2700 $cm^{-1}$ are corresponding to the stretching vibration of aliphatic C–H bonds. The peaks near 1600 $cm^{-1}$ and 1400 $cm^{-1}$ can be attributed to skeletal C-C stretching vibration of polycyclic materials. Signals located at 1200~1000 $cm^{-1}$ are resulted from the absorption of mineral matters, and the faintness of the intensity of that peaks may due to the dissolution of minerals disseminated in coal after microwave treatment with NaOH solution.

Moreover, the spectral range of 750~400 $cm^{-1}$ also contains signals coming from the mineral components, and the intensity of these peaks decreased obviously after treatment. The peak at approx 425 $cm^{-1}$ is assigned to the absorption of pyrite, which shown the reduction of pyritic sulfur. In general, IR bands of organic sulfur groups are very weak and hardly to be observed even in the case of high-sulfur coals, but the IR absorptions of some of them was studied in previous work[17]. The absorption bands in the FTIR spectra of the desulfurized coal near 475 $cm^{-1}$ became illegible are due to the removal of disulfides and mercaptens from coal.

**Fig.4** FTIR spectra of raw and desulfurized coal

### 3.3 The influence of microwave desulfurization treatment on the caking properties of the coal

As we know that cokability of coal relates to the quality of colloids present in course of coal become cake and the colloids are comprised of side chains and fragments derived from the decomposition of coal molecules under thermal. During the process, the oxygen atoms attacked the free radicals contribute by the cleavage of bridge bonds between the structure units of coal macromolecules. Afterwards, the products with small molecular weight increased. In contrast, the products with slightly bigger molecular weight decreased, which contributed to the decrease of colloids during coke-making. The results of Gieseler test shown in Table 3 transfer the viewpoint that the fluidity of the coal gets deterioration as well as the quality of the cokes after microwave desulfurization.

In comparison with the indices adopted in plants, the caking index ($G$) of desulfurized coal is 62, which is higher than the industry standard of 58 at least; the Plastic layer index ($Y$) is 11, which shows a little disabled as the parent coal during coke-making.

**Table 3** The caking properties of the coal before and after microwave process

| Coal samples | Caking index (*G*) | Plastic layer index (*Y*) | Free Swelling Index (*FSI*) | Gieseler test | | | | |
|---|---|---|---|---|---|---|---|---|
| | | | | $Ts$, °C | $T_f$, °C | $T_r$, °C | *MF*, ddpm | $T\Delta$, °C |
| Raw coal | 76 | 19 | 7 1/2 | 403 | 443 | 469 | 70 | 66 |
| Desulfurized coal | 62 | 11 | 5 | 417 | 440 | 463 | 14 | 46 |

## 4 Conclusions

(1) Under the optimal experimental conditions that 5g high-sulfur coal with particle size of -0.074mm mixed up with 20ml NaOH solution with concentration of 6mol/L at solid-to-liquid ratio of 1:4 then was subjected to microwave radiation at the power of 900W for 8min, the content of sulfur in coal decreased from 2.58% to 1.98% with desulfurization rate of 25.23%.

(2) The relative contens of the sulfur existing in forms of mercaptan (sulfoether) and thiophene went down while these of sulfur occurring with sulfone (sulfoxide) and sulfate increased obviously after process. In addition, XANES analysis showed that the pyritic and a small amount of aliphatic hydrocarbon sulfur decreased while sulfate sulfur increased after desulfurization. That meant the sulfur in coal transformed to oxidized states from reduction state in microwave desulfurization procedure.

(3) FTIR spectrums of the coal samples illustrated that the overall structure could get negligible destruction during microwave desulfurization, but the sulfur removal and deash effect. Simultaneously, slight impact on the caking properties of coal sample adopted was tested by the measurements of the caking properties.

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# Country report Mongolia

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**Abstract**

The extremely low coal prices of the last years had huge impact on Mongolia´s coal industry. New processing plants have not been implemented. However, several projects are in initial planning phase and are planned to be realized within the next years.

The coal characteristics of the deposits are differing a lot – this paper will give an overview about statistic figures such as reserves, production, product quality, the mining method of the various deposits and also a brief description of the special requirements for the planned coal processing projects against the background of the specific climatic conditions in Mongolia.

Mongolia´s huge coal reserves and its dependence to fuel import are leading to intensive research and studies for coal gasification/liquefaction processes. Upgrading of raw coal and production of clean coal concentrate as a basis for higher efficiency of CTL plants is a significant issue.

**Key words**: Mongolia, mining industry, raw coal, deposits, preparation technology, BATAC® jig, jigging process, sorting.

## 1. GENERAL INFORMATION

Mongolia is very rich on various natural resources. The coal basins consist of lignite, sub-bituminous, bituminous and coking coal with high grade coal qualities, are distributed all over the country. The beneficial geological conditions allow open pit mining for the huge deposits, mainly conducted by truck & shovel, resulting in enormous cost reduction. Additional advantages are the low stripping rates as well as the dry weather conditions.


Figure 1. Distribution of Mongolia's large coal bearing regions



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The mines are located in very remote areas and transportation of the coal products to the customers is a real challenge due to the underdeveloped infrastructure, i.e. roads, railway system. Another difficulty is water shortage and lack of power supply. Foreign investment is required to realize new coal mine projects. However, changing tax regulations and environmental stipulations are big issues to be considered.

Over 300 coal deposits are identified in Mongolia – the coal resources are estimated to be approx. 173,5 billion tons; proven resources approx. 31,7 billion tons. In total 47 coal mines were in operation in 2015: 29-bituminous coal mines and 18-lignite coal mines, operated by 43 companies.

| Region | Deposits |
|---|---|
| West | Nuursthotgor, Khartarvagatai, Khuden, Yavar, Khushuut, Olonbulag, Zeegt, Khurengol |
| Khangai | Alagtsakhir, Uvurchuluut, Shinejinst, Bayanteeg, MogoinGol, Jilchigbulag, Ereen, Bayanduurkh |
| Central | Uvdugkhudag, Tevshiingovi, Khuutiinkhonkhor, Ulaan-Ovoo, Baganuur, Tsaidamnuur, Tugrugnuur, Bayanjargalan, Shivee-Ovoo, Olongiinukhaa, Khashaatkhudag, Khamriinshural, Tavantolgoi, Nariinsukhait, Gurvantes, Tsagaantolgoi, BaruunNaran |
| East | Talbulag, Bayantsogt, Aduunchuluun, Chandgana |

Table 1. Mongolian Main coal deposits

| Company | Deposits | Resource (M&I) | Reserve |
|---|---|---|---|
| MMC | UkhaaKhudag, BaruunNaran | 890.1 | 471 |
| SouthGobi | Ovoot Tolgoi, Soumber, ZagSuuj, Tsagaan Tolgoi | 470 | 175.7 |
| MEC | Khushuut | 149.2 (Indicated) | NA |
| Aspire | Ovoot | 252 | 147 (Probable) |
| Xanadu | Nuurstei, KharTarvaga | 527.0 | NA |
| Prophecy Coal | Chandgana, UlaanOvoo | 1398.2 | NA |
| SharynGol | SharynGol | 251.604 | NA |

Table 2. Mongolian coal resources example

## 2. BRIEF HISTORY OF MONGOLIAN COAL INDUSTRY

The country has a relatively new coal industry – only few mines are in operation:
1922 - NALAIKH underground mine, taken to the state ownership
1965 - opening of SHARYNGOL open pit coal mine
1966 - opening of TAVAN TOLGOI open pit coal mine
1978 - opening of BAGANUUR open pit coal mine
1992 - opening of SHIVEE OVOO open pit coal mine.
2002 - opening of MAK Naryn-Sukhait open pit coal mine.
2008 - opening of ENERGYSOURCES Ukhaa-Khudag open pit coal mine
2011 - opening of ERDENES TAVAN TOLGOI open pit coal mine, East Tsankhi

| Balance | 2011 | 2012 | 2013 | 2014 | 2015 | 2020 | 2025 |
|---|---|---|---|---|---|---|---|
| Power stations | 5349.5 | 5910.5 | 7061.5 | 106360.0 | 11717.0 | 13000.0 | 15000.0 |
| Industry, construction | 299.2 | 324.0 | 348.4 | 373.0 | 397.6 | 521.0 | 630.0 |
| Transport, communications | 144.0 | 147.0 | 150.0 | 153.0 | 156.0 | 171.0 | 210.0 |
| Agriculture | 16.0 | 20.0 | 24.0 | 28.0 | 32.0 | 52.0 | 60.0 |
| Household, utility services | 520.0 | 525.0 | 530.0 | 535.0 | 540.0 | 565.0 | 610.5 |
| Processing industry | 500.0 | 600.0 | 700.0 | 800.0 | 900.0 | 1400.0 | 1600.0 |
| Central region for heating use | 6828.7 | 7526.5 | 8813.9 | 12525.0 | 13742.6 | 15709.0 | 18110.5 |
| Rural demand | 806.0 | 814.0 | 822.0 | 830.0 | 838.0 | 878.0 | 900.0 |
| Export demand | 25000.0 | 30000.0 | 33000.0 | 40000.0 | 50000.0 | 65000.0 | 75000.0 |
| TOTAL | 32634.7 | 38340.0 | 42635.0 | 53355.0 | **64580.0** | 81587.0 | 94010.5 |

Table 3. Forecast of Mongolian coal consumption 2010- 2025 in thousand tons

Mongolia's main potential customer for the coal products is China, followed by Russia and South Korea.

| Total | thousand tons | 10,458.50 |
|---|---|---|
| Korea | thousand tons | 1.10 |
| China | thousand tons | 10,127.10 |
| Russia | thousand tons | 76.80 |
| Great Britain | thousand tons | 103.20 |
| Singapore | thousand tons | 150.40 |

Table 4. Coal exports Mongolia for hard coal 2015

## 3. PRESENT SITUATION OF MONGOLIAN COAL INDUSTRY

The fall in demand of Mongolia´s biggest customer China as well as the steep reduction of the coal world market prices had huge impact on the coal industry. The enormous decrease of coal exports in 2015 resulting big troubles and losses for the industry.

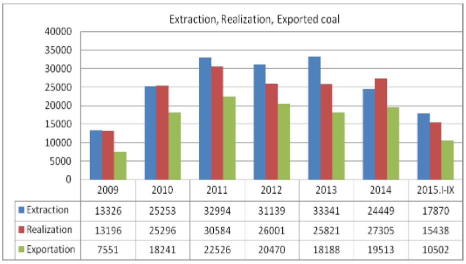


Table 5. Development of Mongolian Coal industry, 2009 – 2015 in thousand tons

| Type of coal | World market price | 2012 | 2013 | 2014 | 2015.IX | 2015.IX/2014% |
|---|---|---|---|---|---|---|
| Thermal coal | US $/t | 75.0 | 60.0 | 56.6 | 46.5 | 82.10% |
| Coking coal | US $/t | 200.0 | 150.0 | 150.0 | 99.0 | 66% |

Table 6. Development of coal world market prices

New mines or processing plants have not been opened and put into operation within the last 2 years. However, long-term plans for development of the Mongolian coal industry consider the rising local demand and expect a stabilized market demand in the future especially in respect to coking coal. Due to the present situation of the industry, a prediction in respect to a time frame is impossible.

| Mine name | Forecast of production in 2025 (1000t) | | Remarks |
|---|---|---|---|
| | ROM | Clean coal | |
| Erdenes Tavan Tolgoi (East Tsankhi) | 20.000 | 14.000 | Coal Handling & Processing Plant planned |
| Tavan Tolgoi (West Tsankhi) | 20.000 | 14.000 | Coal Handling & Processing Plant planned |
| UHG (Ukhaa-Khudag) | 15.000 | 11.000 | Coal Handling & Processing Plant consisting of 3 modules, each 5 mio. tpa = 15 mio. tpa |
| MAK Naryn Sukhait | 14.000 | 5.000 | Coal Handling & Processing Plant planned for approx. 4 mio. tpa |
| Ovoot Tolgoi | 8.000 | 5.000 | Dry and wet Coal Handling & Processing Plant planned |
| Baruun Naran | 7.000 | - | - |
| Tsant Uul | 2.000 | - | - |
| Soumber | 5.000 | - | - |
| Khushuut | 5.000 | - | - |
| Maanit | 2.000 | - | - |
| Huren Gol | 3.000 | - | - |
| Total | 101.000 | 49.000 | |

Table 7. Forecast coal production Mongolia, 2025 in thousand tons

## 4. PREPARATION TECHNOLOGY

### 4.1. Dry coal preparation technology

Due to lack of water in many coal mines, dry coal processing technology would be an appreciated way for upgrading of the clean coal concentrate qualities. On the other hand, its performance is much lower compared to the state of the art wet coal processing technologies in achieving the comparable product qualities; especially for coking coal export will be difficult.

Intensive investigations, analysis and maybe test works with representative coal samples have to be conducted to achieve comprehensive knowledge about the option as well as a proper basis for the decision about the applicable processing technology. Special focus also has to be given to safety aspects due to the risk of fine coal dust ignitions in correlation with dry coal processing.

Dry coal processing facilities have been implemented in some mines already, however, with relatively low success.

| Action plan | Short term (2015) | Intermediate (2020) | Long term (2025) |
|---|---|---|---|
| Dry coal preparation validation facilities | | | |
| Construction of commercial plant | | | |
| | | | |

Table 8. Action plan dry coal processing plants up to 2025

### 4.2. Wet coal preparation technology

Coal processing plants always have to be adapted according to the raw coal features, to the product requirements as well as to the environmental conditions on site.

Water-, other consumables-, energy- and space requirement has to be minimized as much as possible under the conditions of the particular application to achieve reasonable CAPEX and low OPEX. The gentlest processing system should be considered, avoiding additional self-comminuting of the raw coal and by that to keep the slurry circuit as small as possible, resulting in reduction of investment cost. Closed water circuits using thickeners for water clarification and recovery has to be implied to recycle the recovered water back into the plant system. This is also to avoid water losses to the environment. Proper dewatering system for coal processing plant's product has to be implemented. It is especially necessary for plant operations in Mongolia with its extreme climatic conditions in arid regions.

A max. product moisture content of 7-8% $H_2O$ has to be guaranteed throughout the year, ensuring smooth transportability of the products also in winter time at temperatures up to max. -45°C. Installation of coal processing plants in Mongolia in closed, heated and insulated buildings to ensure continuous operation all over the year is necessary. Dust emissions have to be reduced to a minimum, especially at raw coal receiving stations, stockpiles, at belt conveyor discharges and of course also in the processing plant as far as necessary for environmental protection.

The common two types of wet coal preparation processes - Heavy-Media-System (HMS) and the jigging technology – are very different, however both technologies have pros and cons and it is a matter of checking, comparing and calculating of the achievable yield, ash content, CAPEX and OPEX to come to the decision for the most beneficial, cost efficient, economical as well as environmental friendly processing technology for the particular application.

The Heavy-Media-System (HMS) is especially applied for coking coal and for raw coal with high near-gravity-content >20%. The system is featured by a big number of equipment and magnetite requirement as consumable for the process, leading to high CAPEX and OPEX. On the other hand highest yield can be achieved, resulting in sales benefit and positive effect to the balance sheet.

The jigging system, with the advanced BATAC-jig technology needs less equipment - a water-only process and doesn´t require additional consumables. The jigging system is featured by simple design and a plant availability of > 98% is guaranteed – slightly lower yield is achieved, but much less CAPEX and OPEX.

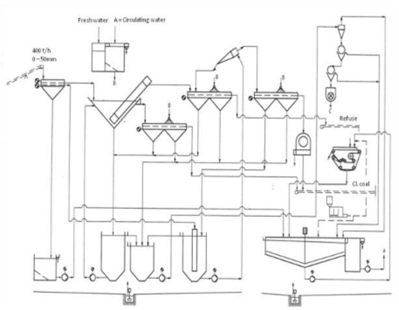


Figure 3. Flow sheet of Heavy-Media-System (HMS)

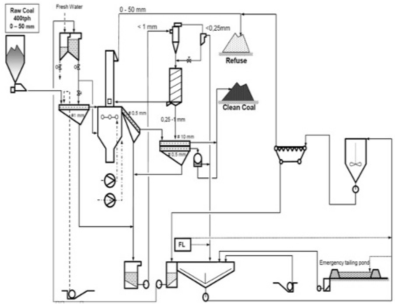


Figure 4. Flow sheet jigging process

Checking, comparing and calculating of every issue and aspect of both processing technologies has to be done very carefully in advance. Although the yield achieved by a HMS-plant is slightly higher than the BATAC-jig plant, the much higher CAPEX is a disadvantage and additionally also the requirement for very fine and high quality magnetite ($Fe_3O_4$, 100% < 0.1 mm), which has to be imported to Mongolia and has huge impact to the OPEX of the system.

However, the raw coal features determine the applied process technology. Combining of both technologies is always possible, resulting in an optimized and simplified flow sheet, reasonable investment cost and huge benefit for operation process and operation cost of a plant. Advanced flotation systems like the PNEUFLOT pneumatic flotation can be supplemented for processing and recovery of the fine coal <0,5mm in order to achieve the best and most economical products.

The requirement to process the coal products of Mongolian mines can be seen on example of OVOOT TOLGOI coal mine, entering in an agreement with a Chinese company to upgrade 3,5 Mil. tpa in its wet coal processing plant located in China, approx. 10km from Mongolians border. However, up to date the commercial operations of the wet coal processing plant have not been commenced.

The existing UKHAA KHUDAG coal processing plant of ENERGY RESOURCES is a Heavy-Media-System plant, designed in 3 identical modules. 2 stages were set into operation in 2012, the total capacity of 15 Mil. tpa. was achieved with finalizing the 3rd module in 2014. The dewatering system for fine tailings has been installed in a modification step, in order to recover the maximum of the process water, to improve the water balance of the plant, to release the tailings pond and also to improve the environmental protection of the surroundings. However, investigations for processing of the tailings pond are ongoing in order also to recover the fine coal from this secondary resource.

## 5. COAL GASIFICATION / LIQUIFICATION SYSTEMS

Mongolia´s huge coal reserves and its dependence to import fuel, diesel, kerosene etc. are leading to intensive research and studies for coal gasification / liquefaction processes. Upgrading of raw coal and production of clean coal concentrate as a basis for higher efficiency of CTL plants is beneficial and can simplify the further refinement.

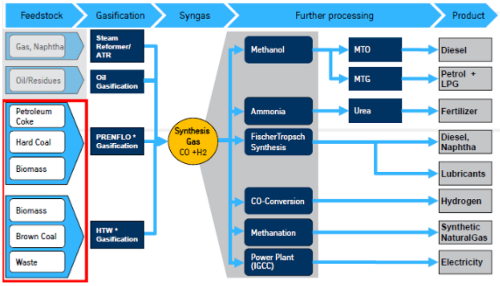


Figure 5. General system Coal refinement

## 6. LEGISLATION DEVELOPMENT OF MONGOLIAN MINING INDUSTRY

The Mongolian Mining Law introduced in 2006 is under revision. Significant reforms on the existing legal frame work are expected, but also of the environmental regulations – especially in respect to water consumption / pollution etc. However, a big impact to the foreign investors is also expected.

| | |
|---|---|
| Government Policy on Mineral sector (2014/01/16) | • To develop transparent and responsible mining for competitive capability based on sector activity. |
| Investment law (2013/10/03) | • To stabilize and provide investment environment.<br>• No difference between Foreign and National investment |
| Law of Common minerals (2014/01/09) | • License issue is authorized to Provincial administration<br>• No tender/bid for license<br>• Set Royalty 2.5 % |
| Redemption of VAT and Custom's tax law (2013/06/07) | • Crude oil, coal, oil shale refinery project equipments and machinery's VAT and Custom taxes free until 2018. |
| Law of border ports (2013/12/26) | • Border port's management system improved<br>• Continuous operating and unified Administration less bureaucracy |
| Amendment on Minerals law (2014/07/01) | • Exploration license begins to issue |
| Law on Prohibiting Mineral Exploration and Extraction Near Water Sources, Protected Areas and Forests | • Issued licenses are valid for operation under Agreement |

Figure 6. Governmental regulations for mining industry

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# Indonesia economic output and employment impact of low rank coal utilization: An input output analysis

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Indonesia is the largest exporter of steam coal. Domestic coal utilization still limited, and even for low rank coal (LRC) has not been used. On the other hand, the need for energy is increasing and now, Indonesia is an oil importer. Through Presidential Decree No. 5/2006 on National Energy Policy, Indonesia will take advantage of 33% coal and coal liquefaction by 2% of the total energy required in 2025. This policy will also reduce the use of oil to only 20%.

In government policies, the LRC utilization is used as a source of energy for domestic purposes. Based on economic calculations, the LRC can be used as briquettes, electricity, coal water mixture or even LRC liquefaction. Nevertheless, the calculation of the economic benefits of widespread the utilization of the LRC has not been done. This paper aims to evaluate the economic impact of output and employment from the LRC utilization using input-output analysis by the latest data of The Indonesian Input Output Table 2008. The calculations show, all types of LRC utilization give high output effect among 9 other economic sectors. While the use of briquettes will provide the output, income and employment effects for the highest compared with the utilization of the power plant, brown coal liquefaction and coal water mixture.

Keywords: LRC, energy, economic impact, input-output, output effect, income effect, employment effect

## 1. Introduction

By coal resources of 161 billion tons and reserves of 28 billion tons, or 3.1% of world reserves (Geology Agency, 2012; BP, 2015), Indonesia produced at 7.2% of total world coal production (statistica, 2014). Coal production of 523 million tons (BP, 2015) and exports amounted to 410 million tons in 2014 (worldcoal, 2015) puts Indonesia as the world's largest coal exporter. Export coal is still needed in Indonesia as a source of the state revenue. Coal mined today that have calorie above 5,100 kcal / kg, while low caloric value (less than 5,100 kcal/kg or LRC) has not been mined.

Energy needs in Indonesia continues to increase, from 777.9 Million barrel oil equivalent (BOE) (2000) to 1,211 Million BOE (2013), with the use of oil is still dominant (MEMR, 2014). Oil demand continues to increase with production continues to decline, forcing Indonesia to become oil importer. The use of biomass in the form of firewood is still high (23.5%), almost half of the timber harvested is used as energy (Mathews, 2009), and this led to excessive deforestation. Then it is proper the energy potential used optimally, one of which is the LRC. When compared with in other places/countries, the quality of the LRC in Indonesia is relatively better with a low moisture content, low ash content and calorie content of a relatively higher (Energy and Resources, 2014).

Because it has a high moisture content and high reactivity, it requires the utilization of the LRC in the mouth of mine, but can also be upgraded to obtain value-added products to improve transport safety and economy (Allardice, et.al, 2001). The main of LRC utilization is a power plant, although a number of novel alternative fuel provide value-added potential like briquetting, coal water mixture/fluid (CWM/CWF), or even LRC liquefaction. Assuming an economic growth of 5.5% / y, Indonesian energy demand will reach 2,658 million BOE in 2025. It would make an interesting LRC to be used as a source of domestic energy according to government policy.

Some research and development has long been conducted utilization of the LRC in Indonesia, although until now has not utilized commercially. LRC utilization program is conducted in cooperation with Japan



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(NEDO) includes LRC liquefaction program, Upgrading Brown Coal (UBC) and CWM / CWF. Of a technical aspect, all utilization can be done, but based on the calculation of the economic feasibility of assuming at this time, LRC feasible to be used as briquettes, power plant/generation, CWM / CWF and even LRC liquefaction.

## 2. Research Methodology

In this study will be analyzed the impact of the LRC utilization on the Indonesian economy. The analytical methods used are the Input Output analysis, with the last IO Table, namely Indonesia IO Table Updated 2008. This analysis was first developed by Wassily Leontief in the 1930s, which was then supplemented by Miller and Blair (1985, 2009), and other researchers. The IO table is a statistical description of the form of a matrix that presents information about the transactions of goods and services as well as interconnections between the units of economic activity (sector) in a region at a certain period. IO models are widely used to analyze the effects of investment-intensive industries to the national and regional economies (Qi, et.al, 2012). In addition, the IO can also describe an indirect effect on the demand for goods and services in other sectors, which is caused by a change in the initial sector. With these IO models, the impact of the use of the LRC on output, incomes and employment can be determined based on Leontief inverse matrix. Some of the assumptions built into these IO models such as homogeneity, proportionality and additivity.

IO basis model consists of rows that show "who gives to whom" and columns showing "who receives from whom" in the economic sectors (Christian and Klaus, 2009; Xu, et.al, 2011). The key parameters in IO analysis are :

- Technical coefficient $a_{ij}$ and matrix A
- Leontief inverse matrix M and output multiplier Oj
- Income technical coefficient and income multiplier $H_j$

In the rows of IO table, can be formed the following equation :

$$x_{i1} + x_{i2} + \ldots + x_{ij} \ldots + x_{in} + Y_i = X_i \qquad \text{(1)}$$

or in general this equation can be written :

$$\sum_{j=1}^{i} Xij + Yi = Xi \qquad \text{(2)}$$

where $X_{ij}$ = purchases by sector j of the goods/services produced by sector i
$Y_i$ = sales from sector i to final demand,
$X_i$ = total output of sector i

Technical coefficient $a_{ij}$ calculated by the formula:

$$a_{ij} = x_{ij}/X_j \qquad \text{(3)}$$

where $X_j$ = total output of sector j

then, we can write again :

$$a_{i1}X_1 + a_{i2}X_2 + \ldots + a_{ij}X_{ij} \ldots + a_{in}X_n + Y_i = X_i \qquad \text{(4)}$$

in the form of a matrix, the above equation can be written :

$$\begin{bmatrix} a_{11} & a_{12} & \ldots & a_{1n} \\ a_{21} & a_{22} & \ldots & a_{2n} \\ & \ldots & & \\ a_{n1} & a_{n2} & \ldots & a_{nn} \end{bmatrix} \begin{bmatrix} X_1 \\ X_2 \\ \ldots \\ X_n \end{bmatrix} + \begin{bmatrix} Y_1 \\ Y_2 \\ \ldots \\ Y_n \end{bmatrix} = \begin{bmatrix} X_1 \\ X_2 \\ \ldots \\ X_n \end{bmatrix} \qquad \text{(5)}$$

If it is written in the form of the equation, it will be :

$$AX + Y = X \qquad \text{(6)}$$
$$\text{or } (I - A)\,X = Y \qquad \text{(7)}$$

$X = (I – A)^{-1} Y$ ……………………………………………………….. (8)

where : I = identity matrix
(I-A) = Leontief matrix
$(I – A)^{-1}$ = Leontief inverse matrix (M)

Leontief inverse matrix may reflect an increase in output was driven by both an increase in directly and indirectly demand. Multiplier output, Oj is the overall increase in total output in the economy as a result of investment in the sector j. Calculations can be done by summing the Leontief inverse matrix column.

$O_j = \sum_i^n M$ ……………………………………………………………… (9)

where M is an element of the Leontief inverse matrix.

Household income multiplier or income effect Hj, shows the total amount of household income that is created by the addition of one unit of final demand money in that sector.

If $v_j = Vj / Xj$ ……………………………………………………………… (10)

then $Hj = \Sigma$ vj. M ……………………………………………………… (11)

where, $v_j$ = labor income coefficient sector j
Vj = value added of labor income sector j
Xj = total input/output sector j

While the employment multiplier or employment effect Ij, is the total effect of changes in employment in the economy due to the presence of one unit of money changes in final demand in a particular sector. Analysis of employment multiplier is used to look at the role of a sector in terms of improving the large number of workers absorbed by the economy. If the employment multiplier or employment effect in the sector greater than one indicates the absorption of labor/employment in this sector is quite high.

If : wj = Lj / Xj ……………………………………………………….. (12)

then lj = $\Sigma$ wj. M ……………………………………………………….. (13)

where, wj = employment coefficien (man/output) sector j
Lj = total employment of sector j

**3. Result**

LRC utilization types are determined after the economic calculation that includes NPV, IRR and Payback Period. This calculation is based on a literature review and then made modifications to the price of the LRC, WACC and the selling price of products, adapted to current conditions. From these results demonstrate the use of briquette, power plant; CWM and liquefaction are economically feasible.

Calculations based on capacity: briquettes with a production capacity of 45,000 tons / year, power plant with a capacity of 1,000 MW (7 Million MWh) (PLN 2010), CWM with a capacity of 1 million tons and BCL with a capacity of 26,900 barrels of oil per day (bopd) or 8,34 million barrels a year (NEDO, 2008).In further calculations (above), then assumed in the same capacity equal to 1 million BOE per year. Economic calculation results then summed to obtain data inputs needed to produce the above production capacity.

From the economic calculation data, then will be obtained for inputs to production, and distributed to the input of Table IO. While in the output, assuming equal distribution of BCL utilization by manufacturing industry, briquetting together with mining and quarrying, CWM together with industry and power plant manufacturing the same with electricity.

Based on the calculation of output and income multiplier of 9 sectors IO Table 2008, shows that the electricity sector gave the highest output multiplier, amounting to 1.883. As for the highest income multiplier is the services sector by 0.609. These numbers shows that the electricity sector can create the highest overall output in the economy. While the number of high income multiplier is in the services sector shows that the sector gave the highest amount of household income.

Table 1. Corresponding sector in IO Table

| Sector | Cost | Briquetting | BCL | CWF | Power Plant |
|---|---|---|---|---|---|
| Agriculture | | | | | |
| Mining and Quarrying | Low rank coal | 8,395,160 | 16,164,166 | 7,843,295 | 22,976,744 |
| | Clay | 337,838 | - | - | - |
| Industry and Manufacture | Catalyst | 3,292,380 | 1,117,556 | - | - |
| | Chemical | 1,228,500 | 1,638,377 | - | - |
| | Spare part | 191,100 | - | - | - |
| | Diesel Fuel | 277,778 | - | - | - |
| | Maintenance | 245,748 | 3,514,912 | 1,092,657 | 5,093,142 |
| Electricity, gas and water | | 54,600 | 2,585 | - | - |
| Construction | | - | - | - | - |
| Trade, hotel and restaurant | | - | - | - | - |
| Transport & Communication | Suppl & Transp | 4,914,000 | - | - | - |
| Finance and business serv | Debt | 2,300,986 | 21,073,452 | 8,169,898 | 30,535,636 |
| | Insurance | 245,748 | 3,514,912 | 1,092,657 | 5,093,142 |
| Services | Packg & Markg | 3,685,500 | - | - | - |
| | Taxes | 4,387,571 | 7,999,272 | 7,347,160 | 28,795,176 |

Table 2. Output and Income Multiplier of the 9 sectors of IO Table Update 2008

| Sector | Multiplier | |
|---|---|---|
| | Output | Income |
| Agriculture | 1.4476735 | 0.226304 |
| Mining & Quarrying | 1.2421247 | 0.144556 |
| Industry Manufacture | 1.7717305 | 0.192653 |
| Electricity, gas and water | 1.9332732 | 0.490276 |
| Construction | 1.8832259 | 0.254113 |
| Trade, hotel & Restaurant | 1.7470745 | 0.267994 |
| Transportation and Communication | 1.6835537 | 0.273350 |
| Finance, RE & Business Services | 1.4598985 | 0.219796 |
| Services | 1.6721904 | 0.609284 |

In the calculation of the economic impact of the LRC utilization with the same production capacity of 1 million BOE per year showed that the briquettes give the highest output multiplier affect an output of 2,034; followed by BCL for 2.033. These numbers shows a creation indicator output. The above calculation shows that due to the addition of one unit of money demand at the end of the briquette sector, the overall total output created in the economy amounted to 2,034 units of money. Sometimes the output multiplier is used as a basis in determining the main sectors in the economy, although it should not happen exaggerated interpretation in these calculations.

LRC utilization as briquette also provide the highest numbers of household income, or the income multiplier effect by 0.084; followed by the use of the LRC through liquefaction that is equal to 0.018. This means that if an increase in final demand by 1 unit of money in briquette sector, household income will increase by 0.084 units of money.

While for employment multiplier or employment effect, the briquette sector also provide the highest number by 0.0015. These numbers is actually almost the same as the meaning of the income effect, the difference lies in the unit, which is expressed in units of jobs. Probably, employment multiplier looks very small, but keep in mind that these multipliers show the number of jobs created. In the above example, if there is an increase in the final demand of 10,000 units of money, there will be 15 new jobs in the economy.

Table 3. The effect of four types of LRC utilization

| Utilization type | Item | Value |
|---|---|---|
| Briquetting | Output effect | 2.033978065 |
| | Income effect | 0.084221758 |
| | Employment effect | 0.001468018 |
| BCL | Output effect | 2.03355608 |
| | Income effect | 0.017675993 |
| | Employment effect | 0.000144159 |
| CWM | Output effect | 1.597718137 |
| | Income effect | 0.04038376 |
| | Employment effect | 3.26027E-05 |
| Power plant | Output effect | 1.835618317 |
| | Income effect | 0.002351687 |
| | Employment effect | 1.60342E-05 |

## 4. Conclusion

As a whole, the LRC utilization provide high multiplier output when compared with other 9 sectors, which shows it can be used as a major economic driver. While income multipliers and employment multiplier from this sector is lower than the 9 other sectors shows that the LRC utilization industry have high productivity in creating output.

Calculation of the economic impact of the LRC utilization itself shows that utilization in the form of briquettes provides the highest impact compared with other utilization. However, because of the limitations of the briquettes themselves as less practical, startup and is only effective if used more than two hours, making the briquettes are still not popular in Indonesia. The BCL sector also results in high output effect. This coupled with the investment cost of the BCL is high; making the resulting output will be more than doubled.

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# From Mining to Refining - Contribution of selective Mining to Value Chain optimisation

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The target of extraction of minerals from the earth crust is the creation of final products. On the way of the mineral from the earth crust to the final product are many steps: exploration, extraction, transportation, storage, processing, refining … In a successful operation, the steps shall be optimized, to minimize the energy and material consumption, environmental impacts, costs …. So one action in a prior step can influence the following. Coal is a unique commodity and can be used for different targets: heat, electricity, metallurgy, chemistry … but this targets need special quality of coal. In coal deposits often can be exist different qualities of coal from point of views like heating value, ash content, humidity, sulphur, bitumen and other minerals. In a cheap mass excavation the coal can be extracted like a mix and then later separated and processed for the different targets. But knowing the distribution of the different coal qualities in the coal seam, a selective mining is possible; the coal qualities are separated on face and delivered to the next process step. From the higher strains in mining profit the following steps: less material must be treated, higher concentration of useful and less concentration of disturbing elements. The paper presents a case study from the middle German mining district for the selective mining of coal for extraction and catalysis of bitumen rich coal. There was developed a concept of real time mining, based on interaction of the excavation data with the data of the geological model to control the excavation process.

Key words: coal, selective coal mining, real time mining, sensors

## 1 Introduction

With over 180 million tons per year, Germany is by far the largest lignite producer in the world. But only a small part of the German lignite is used for material applications. Therefore the Technical University Bergakademie Freiberg worked on a case study with partners from the industry at the ibi-project (innovative brown-coal integration). The aim of the project was the interdisciplinary research on the material use of lignite in Central Germany. The main development goals were an operating diagram for selective mining of quality lignite as well as the development of a mining concept, which uses sensor technology for material and boundary layer detection. The article gives an introduction how different sensors like georadar, XRF, as well as optical and infrared systems were tested and used to achieve these goals. Therefore the main working principles of these sensors will be explained besides the idea how to attach one of these measuring devices directly at the boom of a bucket-wheel excavator to measure and recognize the underneath lying surfaces or guiding layers. In the end the consideration and results how these technologies could work as an aid and the dividing layer recognition and selective mining process should occur is also examined.

## 2 Actual state of that in selective mining

The mining technology in the current situation is designed for a high output, in which the selectivity plays a minor role. But only selective mining guarantees a material application with a maximum product yield. Selective mining yet again is dependent on a precise material and boundary detection, which sets high level of sensory detection ahead. Furthermore in case of the actual production technology used in German opencast mines, like bucket-wheel excavators, bucket-chain excavators, Continuous Surface Miners and Mobile Technology, it is to mention that the machines with a high selective digging height own only a low dividing sharpness. Nevertheless primarily for reasons of the existing production technology in the Central German mining region and the economic efficiency as

 
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well as the demanded performances, the consideration of selective mining by bucket-wheel excavator was made as a result of suitable consultations. Therefore the bucket-wheel excavator was used as basic production technology in this case study on which the operations of different sensor systems were tested [1, 2].

A selective mining of horizontally stored layers cab be carried out with the bucket-wheel excavator in the side block cutting effectively. Thereby the excavator will proceed beside the opencast mining-sided embankment of the passageway and can remove so bigger bench parts in slices. Condition for this is that the thickness of the single layer to be separated of each other is largely enough to get still a sufficient production amount with the bucket-wheel. The separation of the layers can be carried out in the terrace cut in best when the bucket-wheel proceed after a slewing process (cut) several times in cutting direction and the height position of the bucket-wheel is only reasonable changed. With the terrace cut the dividing surface can be laid between the single terraces in such a way that they correspond to the dividing surface between the single layers. However, the uppermost dividing surface may not lie higher than the lower edge of the bucket-wheel in his highest position. This is the selective digging height of the excavator ($h_a$, figure 1). For the operation of the excavator it is important that the maximum height position of the bucket-wheel can be reached in every position of the excavator, so also under a certain excavator slope [3, 4].

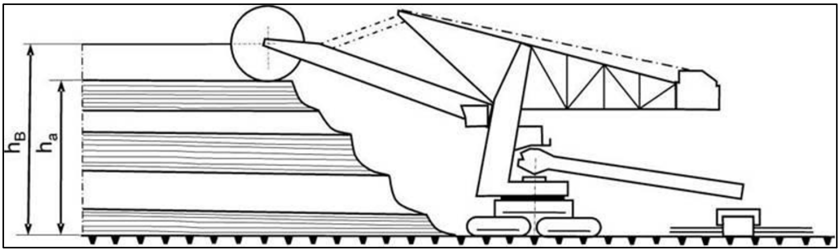


***Fig. 1*** *Selective mining with bucket-wheel excavator ($h_B$-cutting height, $h_a$-selective digging height) [3]*

### 3 Optimization approaches

To optimize selective mining a high level of sensory detection is very important; whereby the fact here is that it needs to be sensors suitable for mining technology. The number of the possible sensor systems was limited after detailed search, considering the requirements for the sensors and on account of the past research results by the numerous procedures with which a dividing surfaces recognition is possible. While looking for potential solution attempts in the area of sensor technology it is a matter of considering that with a choice of a suitable measuring method no direct contact of the sensor with the coal face is aimed to bend forward oversized wear and damages of the sensor technology. Because in addition measuring procedures working with contact are often very sensitive and complicated, therefore as a result the decision was made to investigate measuring procedures working without contact. These offer the advantage that the sensors must be hardly protected against wear by abrasion. An overview of the possible measuring procedures is shown in figure 2. Hereby the technologies of georadar, X-ray fluorescence (XRF) as well as infrared detection or optical sensors have been identified as especially promising. Therefore these sensor technologies and the way they were tested will be explained in the following passage [5].

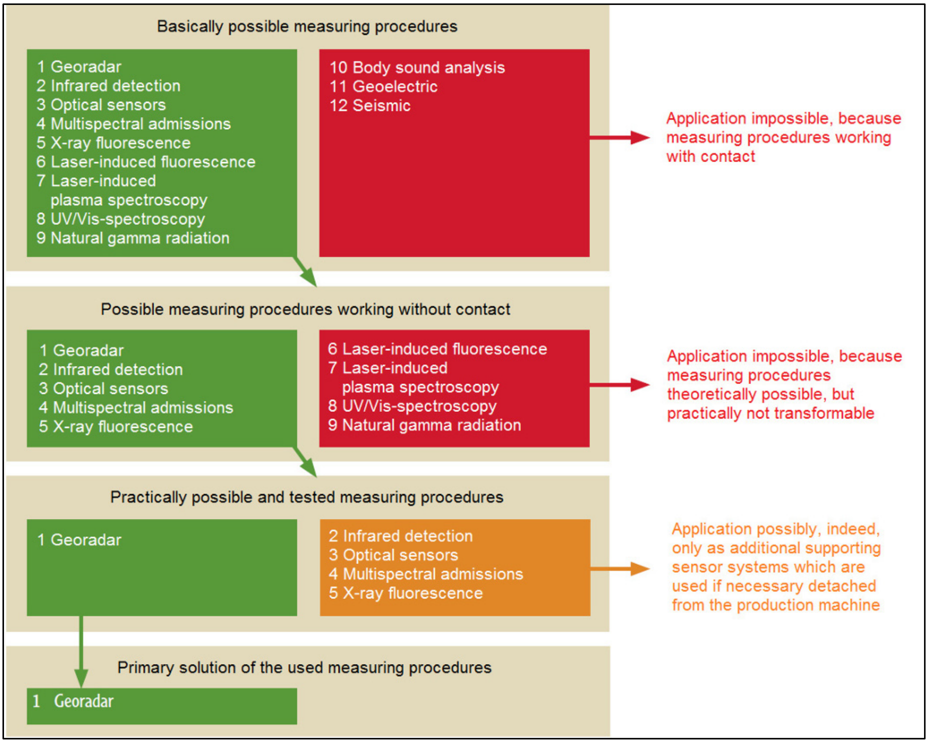


***Fig. 2*** *Selection procedures for suitable sensor systems [Graph: M. Pfütze]*

In the case of the sensor systems suitable for wall face mapping it is basically important to compare the data with the geologic model and to have suitable clues for the part of the layer recognition. IN this area the technologies of optical detection, infrared sensors and XRF have given helpful and promising results. For optical sensors or picture processing programs as well as infrared cameras the consideration consisted in using different software systems to analyse pictures and in extracting certain signs. The experiments have proved that layer courses are indeed recognizable; however, certain conditions like the right day light are needed. Figure 3 shows the result of an infrared measurement at a coal seam. In the case of X-ray fluorescence (XRF) it concerns an optical procedure that with the help of typical fluorescence signals recognizes qualitative differences between significant material types. In different field tests carried out 60 point measurements were taken at a coal seam in an open pit mine in Central Germany. In the analysis by which it was concentrated on the elements iron, titanium, calcium, sulphur and silicon it has become clear that the layer sequence of the coal seam can be exactly defined and understood on account of measured values.

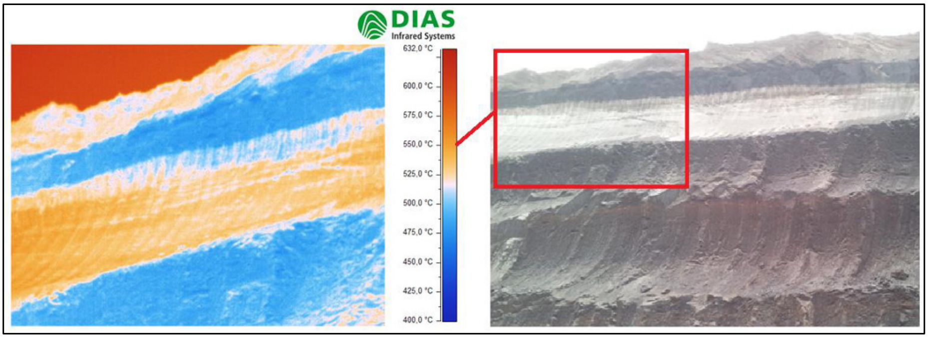


***Fig. 3*** *Infrared picture which was taken at a coal seam and shows the difference between overburden and coal as well as the interburden in the coal seam [Pictures: M. Pfütze]*

In the case of the application of georadar, the measuring instrument sends out electromagnetic waves which are reflected of heterogeneities in the ground and in the loose rock (areas in which the electric conductivity and the relative permittivity change be leaps and bounds). Transitions are mostly geologic border layers, changes of the mineralogical composition or the moist salary. The reflected waves will be conceived with an aerial and the term, the phase and the amplitude will be taped (figure 4), whereby the water content of the ground or loose rock influences the measuring result substantially.

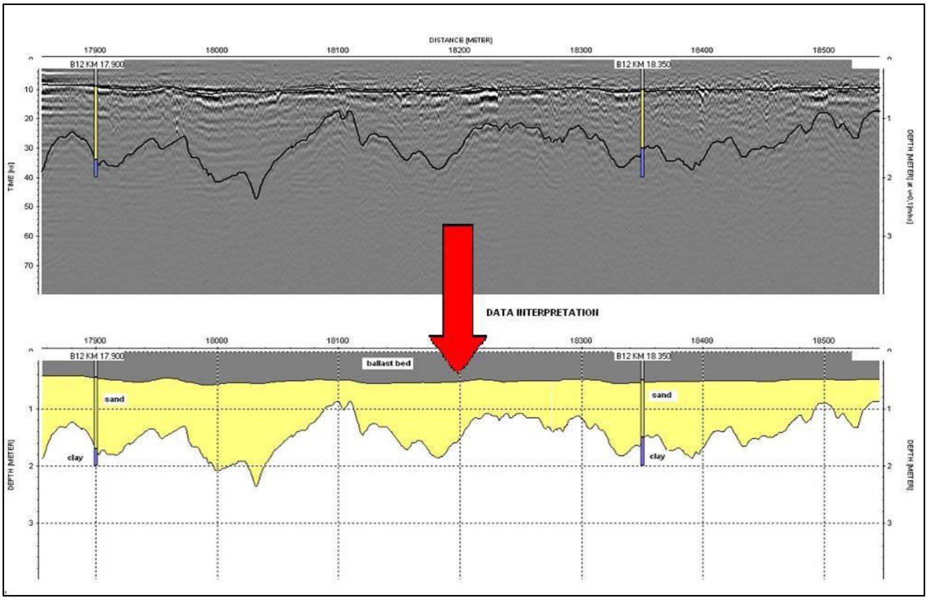


***Fig. 4*** *Example fort he operation of georadar (The layer difference between the both boreholes is clearly recognizable) [6]*

Specific experiments which were carried out on freely accessible seam surfaces as well as directly at a bucket-wheel excavator proceeded very promisingly. Thus it was possible to recognize clay layers as well as other distinguishable layers. Furthermore the idea exists to attach the measuring device for the georadar directly at the boom of the excavator parallel to the bucket-wheel to measure and recognize the underneath lying surfaces or layers (figure 5).

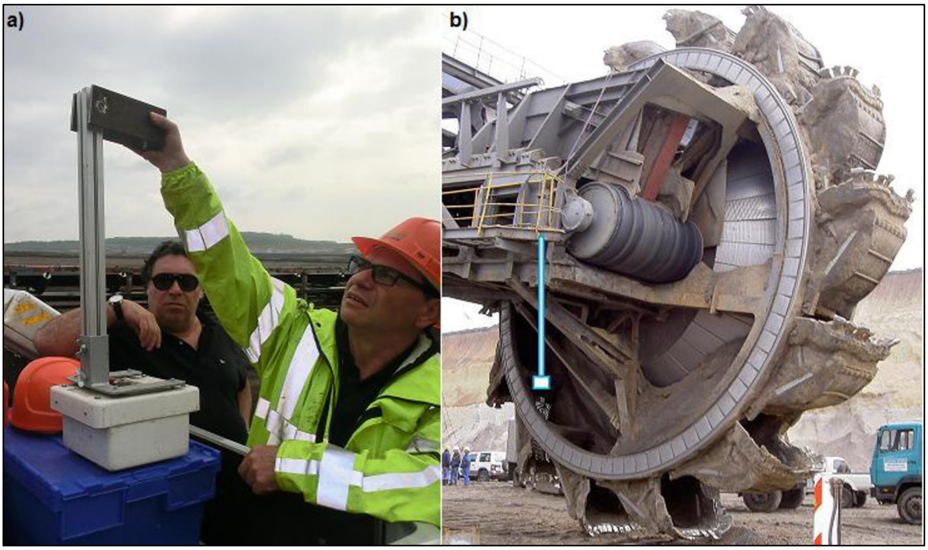


*Fig. 5 Construction-ready measuring device for the georadar (a) and idea-image of the position of the georadar at the end of the boom of an excavator parallel tot he bucket-wheel (b) [Pictures: M. Pfütze]*

Thus in particular the consideration how the sensor technology with georadar could work as an aid and the dividing layer recognition should occur will be further examined. Besides it was a matter of analysing also which dividing sharpness and exactness is accessible according to today´s state and which dividing sharpness is necessary now for a selective production and should be transformed by sensor technology with the georadar. This was also valid for the technical transformation on the machine (figure 6) [5].

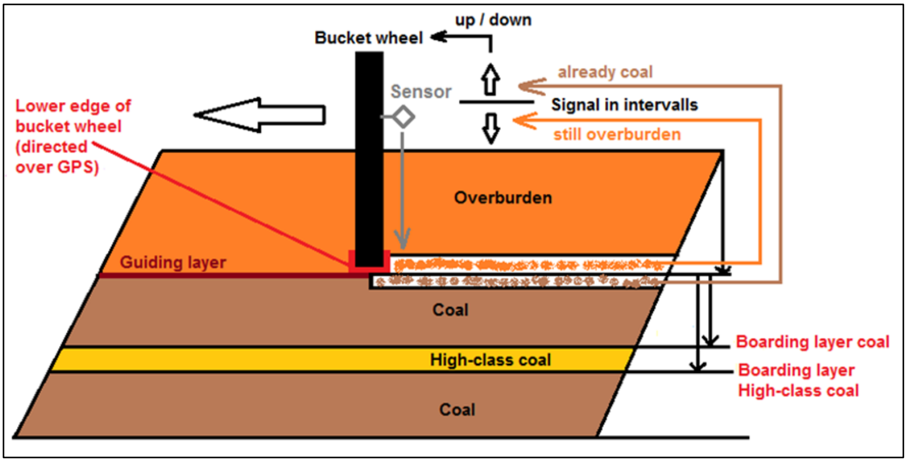


***Fig. 6** Control oft he bucket-wheel by sensor technology with recognition oft he guiding layer [Graph: M. Pfütze]*

## 4 Summary and outlook

During appropriate measurement tests in the field work the aim was to characterize different sensor technologies as well as to recognize the relevant border dimensions and the adaptation to the working conditions. The technologies of georadar, X-ray fluorescence (XRF) as well as infrared detection or optical sensors have given first promising results to define different layers in the coal seam. However these technologies need to be divided into systems suitable for wall face mapping (optical procedures, infrared sensors, XRF) and a system which can be used for layer recognition in the direction of advance on the seam surface as well as directly at the excavator (georadar).

Furthermore as a result of the differentiation a suitable approach for the development of a procedure for sensor recognition of different coal quality layers was defined. In the 1$^{st}$ step suitable data over the respective layer courses are gained by the different procedures which are suited for the wall face mapping. Afterwards the data are transmitted as a 2D-representation of the coal face in the existing geologic model where each single layer can be assigned an x- and z-coordinate (for the direction of working and the dismantling height). A sequence of the representations of the respective coal faces originates from the repetition of the measuring process for every new dismantling block which leads in their sum to an increasing prediction exactness of the single layer courses. In the 2$^{nd}$ step the seam surface is measured by the georadar and therefore data is gained about how the layer courses develop in direction of advance. Afterwards the data will be transferred into the already revised and refined geologic model to receive a detailed 3D-representation of the upcoming dismantling blocks. Thereby the respective layers become next to the already determined x- and z-coordinate from the wall face mapping one more y-coordinate assigned (for the direction of advance). Finally in the 3$^{rd}$ step additional data of the single layer courses is registered and taken by a georadar which is directly at the boom of a bucket-wheel excavator and works continuously during the dismantling process. Again the gained data should serve to refine the geologic model and to define the authoritative predictions for the right cut division for selective mining. An overview of the approach of the single steps is shown in figure 7.

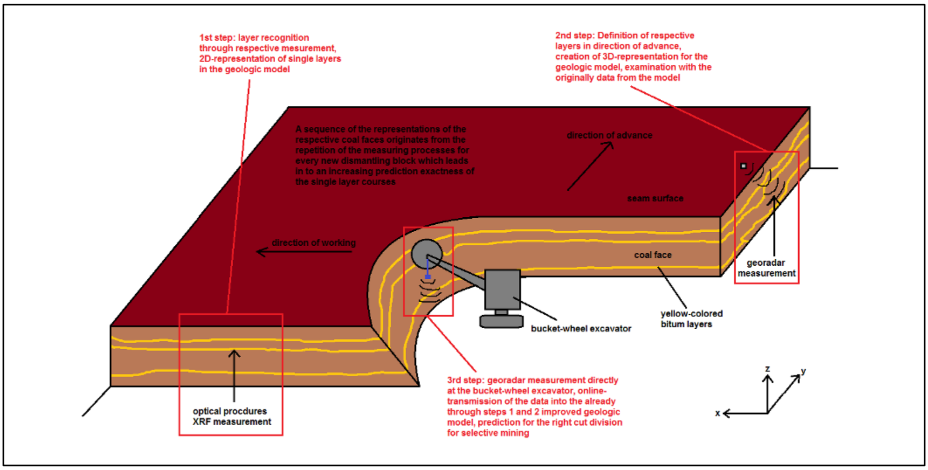


*Fig. 7 Overview of gradual approach for the development of a procedure of sensor recognition of different coal quality layers [7]*

For the upcoming work mainly the technical implementation on the bucket-wheel excavator will play a great role, because the production equipment with sensor will be used for tracking the deposit model and the determination of the distribution of the coal quality in the open pit.

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# STATUS OF COAL MINING AND COAL PREPARATION IN POLAND

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**ABSTRACT**

The Report presents information on the occurrence of hard coal in Poland. Two coal basins are described, in which underground coal mining is currently taking place: the Lublin Coal Basin (LZW) and the Upper Silesian Coal Basin (GZW). For these two coal basins, general information is given concerning geological resources and their structure, as well as the types and average quality properties of the extracted coal.

The Polish hard coal mining industry is currently in crisis due to the decrease in coal prices on international markets. Now (2015) a new restructuring programme of coal companies operating on the Polish market is being implemented. The programme assumes significant changes in the ownership and organisational structure.

On the background of this information some issues of hard coal preparation in Poland are presented (Blaschke Z. 2001). Technological layouts of typical preparation plants are shown (in Poland, different technological systems are used for preparation of steam coal and coking coal). Beneficiation of large, medium and fine coal is discussed. Specification of various machines for coal beneficiation is given, as well as a description of equipment for auxiliary operations. The systems for slurry-water management are also described (Blaschke W.S. 2010, 2013).

**KEY WORDS:** coal production, coal resources, steam coal, coking coal, preparation, washing coarse coal, washing fine product, coal resources, jigs, hydrocyclones, flotation.

## 1. INTRODUCTION

Poland is the largest coal producer in Europe. In 20l4 the output of steam coal amounted to 60,225 million tonnes and 12,288 million tonnes of coking coal. Poland also has the largest coal resources. They amount to 52.0 billion tonnes (geological resources in place), but the amount of economic reserve base is deemed to be 3.7 billion tonnes.

The developed deposits amount to 19.8 billion tonnes (geological resources in place) including 3.7 billion tonnes deemed to be economic reserve base. Steam coal resources amount to 11.6 billion tonnes (geological resources in place), and 2.2 billion tonnes are deemed to be economic reserve base. However, coking coal resources amount to 8.1 billion tonnes geological resources in place, of which 1.5 billion tonnes are deemed to be economic reserve base (Bilans zasobów 2015).



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Recoverable resources of Poland are located up to the depth of 1250-1300 m. Coal extraction is conducted at depths from 400 to 1050 m.

## 2. THE OCCURRENCE OF HARD COAL IN POLAND

Coal can be found in two regions in Poland. These are the Upper Silesian Basin and the Lublin Basin (Blians gospodarki 2015; Blaschke W.S. 2013; Blaschke Z. 2001). The Upper Silesian Coal Basin (GZW) is the largest coal mining centre in Poland. The most productive coal deposits are within mine-field boundary of the coal mines. Coal has been mined in this area for over 300 years. Coal seams with thickness most convenient for mining and located in good mining and geological conditions have already been mostly mined. The Basin constitutes a single entity together with the Ostrava-Karvina area (in the Czech Republic). The entire surface area of the basin is about 5,400 km$^2$ of which 4,450 km$^2$ lies in Poland. In the productive series, the seams are 1.0 – 1.5 m thick, but some seams reach thickness ranging from a few meters to more than a dozen meters. The average ash content in the seams ranges from several to thirty percent. The sulphur content of coal ranges from several per mill to 6.5 percent, and occasionally even more. The net calorific value varies significantly: from 17.0 to 34.4 MJ/kg. All of Poland's coal mines (with the exception of one) are located in this basin.

The Lublin Coal Basin (LZW), located in the east of Poland, covers an area of 4,630 km$^2$. It is about 180 km long and 20 – 40 km wide. The depth of bedding of seams does not exceed 750 m. The productive series contains more than ten seams (up to 18). The thickness of the seams ranges from 0.8 to 2.5 m (and rarely more than 3 m). This Basin contains essentially only steam coal. The ash content varies from 9 to 30%, but 91% of the seams contain up to 15% of ash. The sulphur content ranges from 0.8 to about 3.7%, although some (unmined) deposits contain between 10 and 20% of sulphur. The net calorific value ranges from 25.6 MJ/kg to 28.4 MJ/kg. There is only one operating mine in this Basin (Bogdanka) and its yearly output is almost 10 million tonnes.

In the Upper Silesian Coal Basin the following coal companies operate (the status as in a mid of 2015):

- Kompania Węglowa (Towarzystwo Finansowe Silesia Sp. z o.o.) – with its coal mines: „Jankowice", „Chwałowice" „Rydułtowy – Anna", „Marcel", „Bolesław Śmiały", „Piast", „Halemba – Wirek", „Bielszowice", „Ziemowit", „Pokój", „Sośnica";
- Węglokoks Kraj Sp. z o.o. - with coal mines: „Piekary", „Bobrek";
- Katowicki Holding Węglowy S.A. – with coal mines: „Mysłowice-Wesoła", „Murcki-Staszic", „Wieczorek", „Wujek";
- Jastrzębska Spółka Węglowa S A. – with coal mines: „Borynia-Zofiówka-Jastrzębie", „Budryk", „Krupiński", „Pniówek", „Knurów-Szczygłowice";
- Turon Wydobycie S.A. – operating on three-shafts mine:„Sobieski", „Janina", „Brzeszcze";

There exist also the following small mines:

- Siltech Sp z o.o. – private mine;
- ECO-PLUS Sp. z o.o. – private mine;
- PG„Silesia"- property of a Czech coal company;
- Spółka Restrukturyzacji Kopalń S.A. – preliminarilly aimed for liquidation of unprofitable coal mines, nevertheless currently operates 4 mines: „Centrum", „Makoszowy", „Mysłowice", „Kazimierz Juliusz", for which there is no final decision about their future.

## 3. COAL PREPARATION

### 3.1. WASHING COARSE STEAM COAL

The process of washing + 20 mm coal is conducted in dense medium separators. Magnetite is used as a thickener. A typical flow sheet of washing +20 mm steam coal is presented in fig. 1.

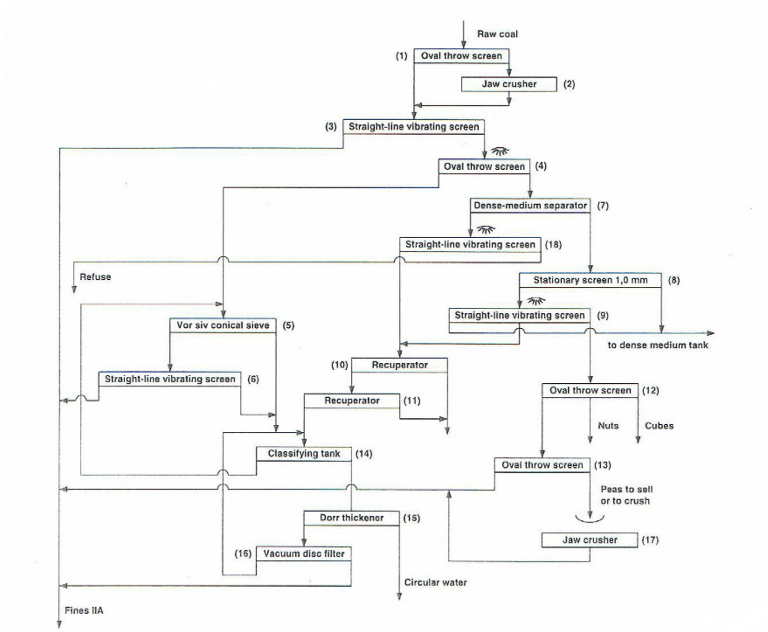


Fig. 1. Flowsheet of a Polish steam coal preparation plant (Blaschke W.S. 2006; Blaschke Z. 2000)

The raw coal is directed for pre-classification at 200 mm vibrating screens PWK-1 (1) with a rotary movement. Particles greater than 200 mm are directed for crushing at jaw crushers KWK- 200U (2). Then the coal moves on to PZ 2675 (3) screen equipped with slotted sieves with the slot width of 20 mm. The coal class +20 mm to -200 mm moves on for de-sliming at the rotary motion screen WK-1 (4) and then for treatment in classifiers DISA - 2S-300 (7). The adjustment and control of dense medium thickness is done automatically. The product is dewatered and the heavy medium/magnetite is separated on table screens (8) and screens with water sprays (9). Filtrate is directed to magnetic recuperator MR - 9/204 (10, 11) where the magnetite is recovered. Dewatered concentrate is sent for final classification (12, 13) into size grades: cobbles, nuts, peas. In case that there is a lack of demand for coarse or middle size grades, coal can be crushed (17) and sent to raw coal bins. The rejects that had been dewatered and stripped of magnetite are sent to waste bins and transported onto waste dumps.

The plant operates with a closed water circulation circuit. The filtrate from the de-sliming screen (4) is sent to a sieve (5). The filtrate from the sieve (5), together with the non-magnetic product from the separator (11) is directed to a classification sump (14) from which the overflow is sent back to the sieve (5). The product, initially dewatered on the sieve (5) is then dewatered on vibrating screens with linear motion (6). The dewatered product is directed to the fine coal bin. The overflow from the classification sump is sent for thickening and filtration. The thickening process is done in radial thickeners and filtration at disc vacuum filters (16). Filtration cake is directed to raw coal bin. The control of commercial products is carried out using a continuous system of coal quality measurement „AIFA-03".

Technological circuits for treatment of coal with particle size +20 mm are practically identical in all coal washing plants. Particular plants differ one from another in technical equipment. They have, for instance, installed different types of DISA separators, screens or filters. The quality of commercial products in coarse and medium particle sizes is very good. The average ash content is 6%, sulphur below 0.64% and calorific value is about 28 MJ/kg (Blaschke W.S. 2010; Blaschke Z. 2001).

### 3.2. WASHING FINES

Considerable differentiation of technologies used for washing fine coal has taken place. In most cases fines are washed in pulsation jigs both of Polish production and in imported ones (Baum, Humboldt, Allmineral). Also there are used water-only hydrocyclones (4 plants), hydrocyclones with dense medium (1 plant) and barrel separators (2 plants). One plant uses vibration-fluidal washers. Fig. 2 presents the flowsheet of a modern plant for washing and desulfurization of fines.

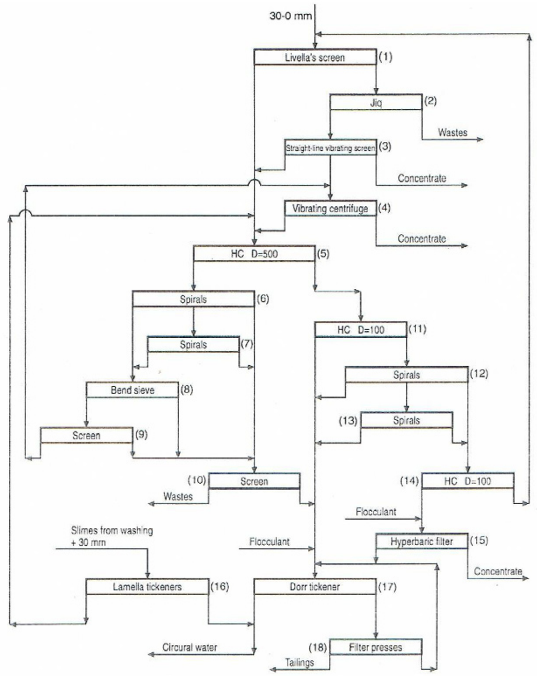


Fig. 2. Flowsheet of coal fines preparation and desulphurization process (Blaschke W.S. 2006; Blaschke Z. 2000)

Fine coal less than 30 mm in size, separated during initial classification, is directed to Livell (1) screen where two size classes are separated: +2.0 to 30 mm and -2.0 mm. The class from +2 mm to -30 mm is washed in Allmineral jigs. Although this jig is a three product machine it currently operates as a two-product one. Reject from the jig is sent to a waste pile. Concentrate from the jig is de-watered at the double deck vibrating screen (diameters 10 mm and 2 mm) (3). The class + 10 to -30 mm is directed to a concentrate tank. The class +2 to -10 mm is forwarded for dewatering by a vibrating centrifuge.

The class -2.0 mm together with filtrate from the screen (1) is treated in Holter hydrocyclones D = 500 mm, with the separation limit particle size set to 0.5 mm. The overflow is treated in hydrocyclones D = 150 mm with the separation limit particle size of 0.04 mm. Classes from +0.5 to -2.0 mm and from +0.04

to -0.5 mm are treated at spirals. Concentrate of class +0.5 to -2.0 mm is dewatered successively at bend sieve (8), screen (9) and centrifuges (4). Rejects +0.5 mm to -2.0 mm are dewatered at a screen and directed to a waste pile.

Concentrate of class +0.04 mm to -0.5 mm is thickened in hydrocyclones D = 100 mm and dewatered at hyper-baric filters by Andritz. Overflow from hydrocyclones D = 100 mm, rejects from the spirals (11) and (12) and filtrate from the screen (10) are sent to Dorr radial thickeners. The sediment is dewatered at filter presses (18) (Blaschke W.S. 2010; Blaschke Z. 2001).

### 3.3. COKING COAL PREPARATION

Coking coal is treated in full range of particle sizes. Coal with particle size +20 mm is treated in grain jigs or DISA separators. Fine coa1 +0.5 to 20 mm is washed in jigs. In the process of gravity separation three products are separated: concentrate, middlings and rejects. Middlings are used as steam coal. There is also an option of crushing the middlings and their re-treatment.

Slimes +0.5 mm, (sometimes 0.3 mm) are washed by foam flotation methods in flotation cells of Polish manufacture (PA-38, PA-6X, IŻ-5, IŻ-12, FLOKOB-12, FLOKOB-24, FLOKOB-40) or in imported ones (Denver, Allmineral). Flotation concentrates are de-watered in vacuum filters. At present imported centrifuges are replacing the vacuum filters in several plants. Modernization of the dewatering process is intended, targeted at elimination of the coal drying process.

Flotation tailings are thickened in Dorr radial thickeners. The sediment is dewatered at Polish made filter presses or at belt filter presses. A typical technological flow sheet of coking coal washing with is presented in Fig. 3.

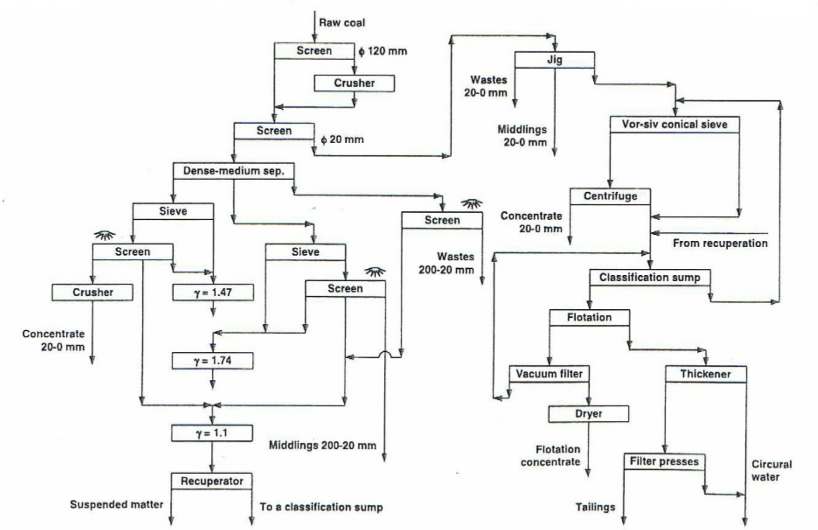


Fig. 3. Flowsheet of the coking coal preparation plant (Blaschke W.S. 2006; Blaschke Z. 2001)

## 4. MACHINERY AND EQUIPMENT USED IN POLISH COAL PREPARATION PLANTS

Raw coal is taken from the output shaft of the mine to preliminary preparation before coal washing. Here wood, metal, rubber and other rubbish are removed and lumps of coal too large for the cleaning plant are crushed. Then the coal is preliminary classified into size classes appropriate to the applied technology of washing. Here usually domestic screens (WK, PWK, PWP, PZ, PWE, ZDR, RT) or imported ones (Schenk, Siebtechnik, Don Valley, Livell, Allis) are applied. In Polish coal preparation plants 332 screens are used, out of which 273 are of domestic construction and 59 are from abroad.

The washing of coal sizes above 20 to 10 mm is done in Polish dense media separators type Disa (Disa 1, Disa 2S, Disa KU, Disa KR) or in water jigs – also produced in Poland, like (OBZ, OZ, OZL, OS). One Drew-Boy separator from the Denver firm is used in one of plants. It washes coal sizes 600 - 6 mm and is very effective.

There are many types of machines for washing fine grains. Classes between 20 or 10 mm to between 0.5 or 0.2 are usually washed in Polish jigs (OBM, OM, OS) and domestic hydrocyclones (HWO, HKZ). In many coal preparation plants foreign machines are installed: Allmineral and Batac jigs, Parnaby and AKW hydrocyclones, Krebs dense media cyclones, Reichert spirals. There are also some barrel concentrators with natural dense media. Altogether, 427 concentrators operate in Poland, out of which 351 are of Polish production and 76 are imported.

Flotation is used to wash coking coal. Also, in two steam coal preparation plants flotation is used experimentally. Polish steam coals are difficult to float. The basic flotation machines are mechanical flotation machines produced in Poland (IZ5, IZ12) and column flotation machines (FLOKOB 12, FLOKOB 24, FLOKOB 40). Also, imported flotation machines from Allminerall (Allflot) and Denver are used.

Many different methods are used in coal preparation plants for de-watering, as the process is difficult. De-watering of coarse grains of concentrate takes place in vibrating screens produced in Poland (WP-2, PWP-1, PWE, PWL-Z). Concentrates of grain sizes below 20 mm are de-watered in the set: screen – de-watering sieve (OSO) – vibration centrifuge (WOW) – de-watering machine (Nael). In some plants Humbold centrifuges (HSG) work, as well as Wemco (H-900) or Humbold de-watering machines (Konturbex). Slimes (grains below 0.5 mm) are de-watered in the set: de-watering sieve – hydraulic classifier – radial thickener Dorra – vacuum filter. Vacuum and disc filters are used (FTB, FTC, FTBO) along with domestic filter presses (PF- 570, PF 1.2, PF 1.5) and hydrocyclones (HKZ). As concerns imported machines, pressure filters Andritz (HBF-96) and belt filter presses (CPF-2200, PL-2200) can be found. Sieve-sedimentary centrifuges and sedimentary centrifuges are not produced in Poland. They are imported from BIRD.

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# Further Developments in Washability Prediction Using CGA

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**Abstract**

Coal Grain Analysis (CGA) measures the maceral grain types in individual particles, not just as an overall maceral distribution. Such information is extremely informative to understanding how individual particles behave in coal preparation, and provides a fundamental understanding of practical requirements for liberation of 'clean' coal. Atkinson et al (2012) reported on a new project that utilised CGA to predict CPP feed washability at 50 mm topsize. At that stage, only two coal types had been evaluated.

The results obtained by this project show that the technique is, as a minimum, successful for predicting CPP feed washability for each coal type (each mine lease).

This paper provides an update on the CGA Washability Prediction project, including information on reproducibility of the CGA analysis itself and its impact on CGA washability prediction. CGA reproducibility is contrasted with published reproducibility for float and sink determinations.

Keywords: coal grain analysis, CGA, washability, borecore testing, float and sink analysis, coal preparation, automated image analysis, relative density

## Introduction

The current status of resource assessment is that borecores (or alternative raw coal samples) are subjected to some form of laboratory pre-treatment, generally different for each site, to fragment the coal into different particle sizes followed by float and sink testing to determine the washability distribution. Several issues arise and consequently, a new and more fundamental means of characterising coal is necessary to facilitate greater flexibility in using the resource data long after they are generated, even when potential uses for the coal, or available means of beneficiation, change.

CGA maintains all of the advantages of conventional maceral analysis, in that the absolute proportions of coal matter and mineral matter are measured. The distinct advantage of CGA is that it measures this information on a particle-by-particle basis.

The downside of CGA is that it relates to minus 1 mm particles. The analysis uses approximately 10 g of material at 1 mm topsize, and the particles are mounted in a resin block, then sectioned and polished.

The image analysis process involves classifying each grain type according to the volumetric proportions of vitrinite, inertinite, liptinite and minerals. An example image is shown in Figure 1.

The automated image analysis software interprets (classifies) each and every individual pixel. This project uses the CGA result in its standard form. The unique aspect of this project is that it has determined the CGA of individual relative density (RD) fractions of individual size fractions of various coals.

The CGA method allows the RD of each individual particle to be estimated with a reasonable level of accuracy (O'Brien et al. 2013a). Beneficiation of raw coals in preparation processes largely depends upon particle separation by density. Flotation is also employed in coal preparation, and that process relies on particle surface characteristics rather than density. CGA provides invaluable information for flotation design since the flotation characteristics of a coal are directly related to its coal grain makeup (Ofori et al, 2013).

 
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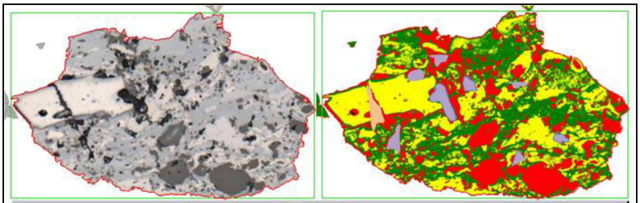

| Volume% | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| % vitrinite | % inertinite | %Dark Minerals | | | | %Bright Minerals | | % Liptinite |
| 38.0 | 32.1 | 30.0 | | | | 0.0 | | 0.0 |
| | | Clay minerals | Quartz | Carbonate minerals | Other | Pyrite | Iron oxide | |

**Figure 1: Example CGA image of an individual particle (grain)**

***Reproducibility of CGA***

Atkinson (2015) reported on the reproducibility of the CGA technique. The project involved distributing four carefully subdivided samples at 4 mm topsize, of each of four different coal types, to each of four different laboratories. Each laboratory was then required to undertake petrographics sample preparation in accordance with AS2856.1 for subsequent CGA. The project evaluated the total CGA result variability for each coal type including all sample preparation errors from 4 mm topsize.

The results are summarised in Table 1. AS2856.2 publishes reproducibility data that varies according to the volume percentage of each maceral component.

CGA uses automated image analysis software to interpret the optical images. The basis for comparison between CGA and manual point counts (AS2856.2-1998) was to consider one pixel to be equivalent to one point count (one cross-hair intersection). In a typical CGA determination, the average ‘analysis area’ of a grain is typically of the order of 1,000 pixels. Since the CGA maceral count is undertaken on each grain individually, that is an appropriate comparison basis. An overall CGA result is simply a sum of the individual grain analyses.

The take-home message is that CGA has been found to offer a reproducibility level that is superior to that for conventional petrographics assessment. This is despite the fact that the ‘practical’ CGA reproducibility figures quoted in Table 1 include a wider range of error-inducing activities when compared to the Standard, noting the inclusion of sample subdivision at 4 mm topsize, followed by size reduction to 1 mm topsize and subsequent further subdivision.

**Characterisation as the First Step**

Characterisation has been based on detailed analysis of borecores or CPP feed samples (representative raw coal) for each coal type.

Figure 2 shows the overall coal grain distribution for coal QLD3, being Leichardt seam coal from the Rangal measures. Table 2 shows an example coal grain partition distribution, in this case for the vitrinite component.

**Table 1: Reproducibility of CGA compared to AS2856.2-1998 (Atkinson, 2015)**

| Component | CGA Sample 1 | | | CGA Sample 2 | | |
|---|---|---|---|---|---|---|
| Coal Grain Classification | Average composition %v/v | AS2856.2-1998 Reproducibility (1,000 point count) %v/v | Measured Reproducibility CGA %v/v | Average composition %v/v | AS2856.2-1998 Reproducibility (1,000 point count) %v/v | Measured Reproducibility CGA %v/v |
| Vitrite (>95% vitrinite) | 5.9 | 5.4 | 1.6 | 10.8 | 7.2 | 4.2 |
| Inertite (>95% inertinite) | 3.3 | 3.9 | 1.5 | 3.3 | 3.9 | 0.6 |
| Minerite (>95% minls) | 0.2 | 3.9 | 0.1 | 0.1 | 3.9 | 0.1 |
| Vit dom (95%>V>65%) | 15.5 | 7.2 | 4.3 | 19.4 | 7.2 | 3.1 |
| Inert dom (95%>I>65%) | 35.1 | 8.8 | 5.1 | 28.5 | 8.2 | 3.8 |
| Minl dom (95%>M>65%) | 0.6 | 3.9 | 1.3 | 0.3 | 3.9 | 0.3 |
| Vit rich (V > I ,L, minls) | 18.1 | 7.2 | 5.0 | 22.3 | 8.2 | 1.2 |
| Inert rich (I > V, L, minls) | 20.7 | 8.2 | 5.7 | 14.9 | 7.2 | 2.9 |
| Mineral rich (M > V, I, L) | 0.7 | 3.9 | 0.9 | 0.3 | 3.9 | 0.2 |

**Results from Predictive Washability Work**

Further to Atkinson et al (2012), validation results for washability prediction are now available for additional coals of different rank and type collected from different coal measures.

The washability predictions were undertaken by using secondary ('validation') samples, each of the same seam section but from different locations on each lease. For some coals, data were generated from 100 mm and 200 mm diameter borecores. For other coals, data originated from representative CPP feed samples each of approximately 600 kg. For the CPP feed samples, the validation samples were collected more than six months after the characterisation samples, demonstrating significant change in geographical location due to the advance in mining over that period.

Each validation (and characterisation) borecore sample was subjected to conventional pre-treatment (eg. drop-shatter) and all samples were subjected to wet tumble, sizing and float/sink. Only the raw coal composite for each validation sample was subjected to CGA. Thus each validation sample used only one single raw CGA analysis to cover the working section, compared to the characterisation samples which used up to 45 CGA analyses (every size and RD fraction).

By applying the relevant partition table for each grain type to the amount of each grain type in the raw coal validation sample, the overall size and density distributions were predicted. The method of applying the partition tables is described in Atkinson et al (2012). Those predictions were then compared to the measured size and washability data for each validation sample.

Figure 3 shows the comparisons between the predicted and measured washabilities for coal QLD2, being Lilyvale seam from the German Creek coal measures. The results demonstrate excellent agreement. In Figure 3, a third washability is also shown – that of the original characterisation sample. The result demonstrates that the prediction successfully accounts for changes in washability. In that case, the washability of the characterisation sample was substantially different to the validation sample.

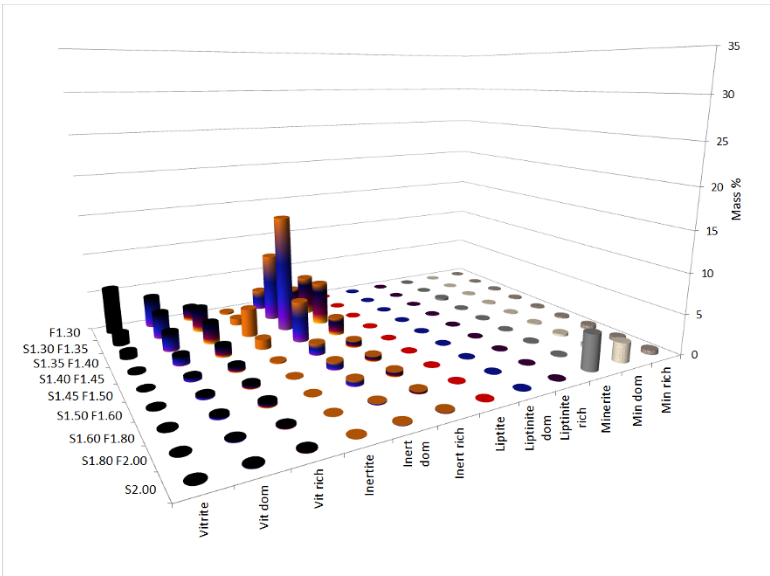


**Figure 2: Overall CGA distributions – Coal QLD3 +0.038mm (94.9% of raw coal)**

**Table 2: Example vitrinite grain partition table for coal QLD2**

| | **Mass partition by size and RD fraction (%) – Vitrinite** | | | | |
|---|---|---|---|---|---|
| **RD Fraction** | **+31.5 mm** | **-31.5 +8 mm** | **-8 +2 mm** | **-2 +1 mm** | **-1 +0.038 mm** |
| F1.30 | 4.06 | 15.89 | 26.78 | 10.76 | 28.19 |
| S1.30 F1.35 | 1.18 | 4.95 | 2.27 | 0.54 | 2.61 |
| S1.35 F1.40 | 0.14 | 0.65 | 0.52 | 0.12 | 0.44 |
| S1.40 F1.45 | 0.02 | 0.15 | 0.10 | 0.04 | 0.19 |
| S1.45 F1.50 | 0.00 | 0.05 | 0.06 | 0.01 | 0.08 |
| S1.50 F1.60 | 0.00 | 0.02 | 0.03 | 0.01 | 0.09 |
| S1.60 F1.80 | 0.00 | 0.01 | 0.01 | 0.00 | 0.00 |
| S1.80 F2.00 | 0.00 | 0.00 | 0.01 | 0.00 | 0.00 |
| S2.00 | 0.00 | 0.00 | 0.01 | 0.00 | 0.00 |

Figure 4 shows the impact of CGA reproducibility (Atkinson, 2015) on predicted washability. The range of predicted washabilities falls well within the range that would be expected due to reproducibility of float and sink alone.

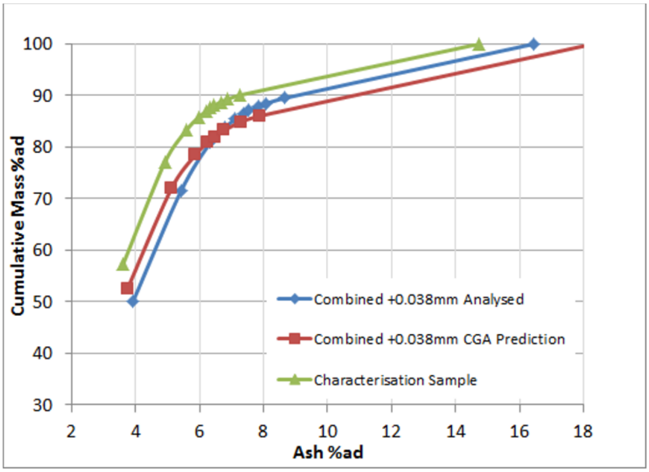


**Figure 3: Measured v predicted washability for coal QLD2**

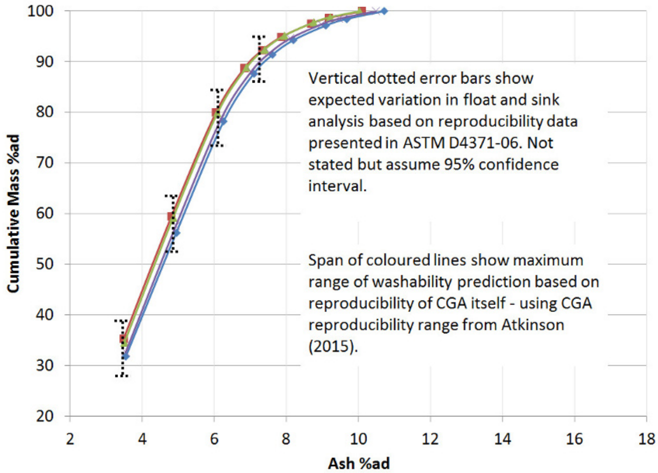


**Figure 4: CGA vs float/sink reproducibility**

**Conclusions**

The reproducibility of the CGA method is such that it should provide a tighter range of washability prediction compared to float and sink analysis.

All coal types which have so far been subjected to validation have demonstrated the value of the CGA partition table approach. It is evident that the CGA washability prediction approach has substantial merit, especially considering the flow-on benefits of being able to consequently predict product coal quality parameters on a fundamental basis according to the product grain composition.

The small sample size requirement (10 grams for CGA) means that slimcore samples are all that are required to quantify (recoverable and saleable) coal reserves within a resource. This will have a marked impact on resource assessment by reducing the associated time and cost.

Further CGA data are required to progressively build up a substantial body of knowledge on the fundamental grain distributions by size and RD.

The data so far available demonstrate that CGA characterisation by size and RD is invaluable for the purpose of a fundamental understanding where the coal values lie, and what may be able to be done to optimise their recovery.

The predictability of size and washability, based on raw coal CGA alone, is very promising, and CGA in combination with slim cores, offers a new and fundamental approach to comprehensive resource assessment and coal preparation feed characterisation.

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# New type of noble metal mineralization in the graphite formation of the Khanka terrane, Russian Southeast

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**ABSTRACT**

**N**oble metal–graphite mineralization has been identified in the Riphean–Cambrian metamorphic complexes of the northern Khanka terrane, Russia. The graphite mineralization is hosted in magmatic and sedimentary rocks metamorphosed under greenschist to granulite facies conditions. This paper provides the results of our study of the Turgenevo–Tamga graphite deposits. This study analyzes the geochemistry of the noble metals with the aim of determining the spatial relationships between noble metals and graphite. The graphitized rocks, analyzed by various geochemical methods, show a wide range of noble metal concentrations (ppm): Pt (0.02–62.13), Au (0.02–26), Ag (0.56–4.41), Pd (0.003–5.67). Crystallization from gas-condensates is indicated by the relationships between the noble metal mineralization and the graphite, and in particular the inhomogeneous distribution of graphite in the rocks, the inhomogeneous distribution of metals in the graphite, the microglobular graphite structures, and the carbon isotopic compositions. Our conclusion is that the noble metals and graphite mainly originated from magmatic fluids, but that some material was derived from exogenic and metamorphic sources.

Keywords: Noble metals, Carbon, Graphite, Metamorphism, Fluids

The participation of endogenic carbon during ore formation has been supported by the discovery of a new type of noble-metal mineralization in the Riphean–Cambrian metamorphic complexes of the northern Khanka terrane of Russian Primorye [1, 2, 3]. The graphitization here is of regional extent and affects rocks that have undergone greenschist to granulite-facies metamorphism, including schists, gneisses, granite gneisses, marbles, phyllites, and lamprophyre dikes. The graphitization is quite unlike the mineralization found in typical stratiform black-shales. In the Khanka terrane, exploration for graphite took place prior to 1950, particularly in the large deposits at Tamga (400 $km^2$) and Turgenevo (225 $km^2$) in the Lesozavodsk district of Primorye.

The determination of noble-metal concentrations in graphite-bearing rocks is problematic because graphite is inert to almost all chemicals. The quantitative physicochemical methods used in this study include atomic emission spectrometry (AES), inductively coupled plasma–mass spectrometry (ICP-MS), and atomic absorption spectrophotometry by thermoelectric atomization with prior extract concentration (AAS-TEA), all of which have been previously used by other workers to analyze noble metals in carbonaceous terrigenous rocks [4, 8]. To determine the concentrations of Pt and Au in the graphite-bearing rocks from the Khanka terrane, we mostly used AAS-TEA with a sensitivity of $10^{-8}$ wt%. Chemical decomposition was performed as follows. The samples were successively dissolved in mixtures of concentrated acids in the proportions $HF{:}HNO_3 = 2{:}1$, → $HCl{:}HNO_3 = 3{:}1$, → HCl. Next, the insoluble graphite residue was filtered and dissolved in $HClO_4$, and both solutions were combined and converted into 2N HCl. Finally, noble metals were extracted with alkylaniline, and concentrations were determined by AAS-TEA on a Shimadzu AA-6800 spectrophotometer. The AAS-TEA method provides more reproducible results compared with other techniques, but is time consuming. The results obtained by AAS analysis for Au, Pt, and Pd concentrations in soluble silicate fractions and in the graphite residue are listed in Table 1.

High Au and Pt contents were first discovered in the graphite-bearing rocks of the Khanka terrane by ion mass spectrometry (IMS) analyses in 2004 performed by scientists of the Institute of Microelectronics and High-Purity Compounds, Russian Academy of Sciences, Chernogolovka. This method of analysis has

 
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an advantage in that it allows the high-sensitive detection of elements and noble metals in solid samples, and it does not require the preliminary dissolution of samples in strongly oxidizing reactants. The IMS analyses were conducted using an Element 2 ICP mass spectrometer manufactured by Thermo Electron Corporation. The spectrometer was equipped with a glow-discharge ion source based on a hollow cathode installed instead of an ICP-source sampler [10]. Application of a new source of ions allowed us to analyze solid, non-conducting powder samples, and the high sensitivity of the method made it possible to analyze the powdered carbonaceous rock samples without pre-concentration of the metals. In addition, IMS allows the imaging of spectra for a wide range of metals and their isotopes. In earlier methods of analysis, the dissolution of samples at high temperatures often led to a drop in the detected values of the metals due to the emission of some carbon-metallic volatile complexes; the IMS method avoids this problem. The value of the IMS method is supported by comparisons of analytical results produced by both the AAS and IMS methods for the same samples (Table 2).

Table 1: Noble metal contents (ppm) in selected rock samples from the Khanka terrane, as determined using soluble silicate fractions and graphite residues.

| Sample | Au⁄ | Graphite | | | PAu | C (wt%) | Rock |
|---|---|---|---|---|---|---|---|
| | | Au | Pt | Pd | | | |
| 02/1 | 0.73 | 16.68 | 8.68 | 5.67 | 17.41 | 35 | Graphitic metasomatic rock |
| 02/3 | 0.56 | 2.83 | 2.15 | 0.99 | 3.39 | 4.7 | Plagiogneiss with graphite |
| 02/4 | 0.61 | 4.18 | 2.39 | 1.23 | 4.79 | 6.3 | Granite gneiss |
| 03/1a | – | 2.56 | 4.14 | 3.31 | 2.56 | 5.6 | Graphitic metasomatic rock |
| 03/3 | 0.1 | 5.37 | 14.15 | 7.31 | 5.47 | 30 | Garnet–biotite–graphite schist |
| 03/5 | 1.26 | 0.04 | 4.46 | 1.24 | 1.30 | 29 | Lamprophyre |

Note: Au⁄ = gold in solution after the initial treatment of samples with aqua-regia; Au, Pt, and Pd were measured in the graphite residue after its decomposition in HF and HClO4 (analyses were performed using AAS).

The rock samples in polished sections, fresh rock chips, and separate grains were also studied with a scanning electron microscope EVO-50XVP equipped with an energy dispersive X-ray spectrometer INCA Energy-350 (SEM EDA analysis) at the Analytical Centre of FEGI FEB RAS. It is noteworthy that the identification of native metal phases in graphite-bearing rocks is a difficult task. In polished sections, the soft graphite spreads over the surface of the section, and minute hard grains of metal are crumbled out. For this reason we used fresh chips of rocks without polishing. In some cases the surface of a chip was etched with strong acids, which made it relatively easy to find microcrystals of native noble metals. Moreover, electron energy loss spectroscopy (EELS) was used to detect platinum inclusions in graphite flakes from the Turgenevo deposit. The EELS allowed us to visualize the distribution of noble metals in any matrix on the nanometer scale [6].

In all the varieties of rocks present in the study area, a high degree of graphitization, including graphitic metasomatism, is ideal for studying the graphite. Graphite occurs as monomineralic veins and lenticular inclusions in the igneous protoliths and skarns, and it is also widespread as dispersed lamellae, ranging in size from 200 nm to 2 mm, that are oriented along the foliation of the gneisses and schists. The relationships with other minerals in thin section indicate that it crystallized at the same time as the other metamorphic minerals, forming mutual intergrowths with biotite, sericite, chlorite, and quartz, or as inclusions in feldspar and pyroxene. This suggests that the regional carbonization coincided in time with the regional metamorphism.

The isotopic compositions of the carbon in rocks metamorphosed under the amphibolite and greenschist facies also illustrate the inhomogeneity of the carbonaceous matter (Table 4). Graphite in the core of the Ruzhino metamorphic dome has stable carbon isotopic compositions with δ13C = –8.7 ± 0.1‰, inherent in all rocks of the amphibolite facies. It is known that values of δ13C close to –7‰ characterize an endogenous crustal source of carbon. The carbon isotopic compositions in greenschist-facies graphite–sericite–quartz schists have lower δ13C values that range from –19.3‰ to –26.6‰, which is typical of organic carbon in marine sediments [3].

Table 2: Gold and platinum contents (ppm) in rocks from the Turgenevo deposit, as determined by IMS and AAS (one analytical run).

| Sample | Au | Pt | Method | Rock |
|---|---|---|---|---|
| 02/3 | 13 | 4 | IMS | Granite gneiss |
| | 3.39 | 2.15 | AAS | |
| 03/1a | 5 | 16 | IMS | Granite gneiss |
| | 2.56 | 4.14 | AAS | |
| 03/3 | 3 | 6.7 | IMS | Garnet–biotite–graphite schist |
| | 5.47 | 14.15 | AAS | |
| 03/5 | 5 | 52 | IMS | Lamprophyre |
| | 1.30 | 4.46 | AAS | |
| 04/7a | 12 | 20 | IMS | Skarn |
| | 1.04 | 1.15 | AAS | |
| 04/7, | 12 | 14 | IMS | Skarn |
| | 0.16 | 1.51 | AAS | |
| 04/17 | 7.2 | 5 | IMS | Graphite–sericite–quartz shale |
| | 0.66 | 1.30 | AAS | |
| 04/29 | 15 | 18 | IMS | Lamprophyre |
| | 0.46 | 1.28 | AAS | |
| 04/40 | 17 | 24 | IMS | Graphite–sericite–quartz shale |
| | 0.18 | 1.29 | AAS | |
| 04/9 | 2.2 | 3.3 | IMS | Graphitic metasomatic rock |
| | 0.14 | 0.82 | AAS | |

The forms of the noble metal–graphite mineralization were studied mainly with SEM and a Zeiss Libra-120 transmission electron microscope (TEM) with a HADDF detector and δ-filter. The TEM method involves electron energy-loss spectroscopy, providing an EELS image, which represents the qualitative distribution of elements with high sensitivity. It also allowed us to detect the characteristic losses of the $O_{2.3}$ and $M_{4.5}$ energy of platinum (Fig. 1 a, b), which is an indicator of Pt in graphite.

SEM analyses show that thin-prismatic crystals of isoferroplatinum (Pt = 90.36 wt%, Fe = 9.64 wt%) exist in the graphite–sericite–quartz schist (Fig. 2). Subprismatic microcrystals (up to 2–3 μm) containing platinum and an admixture of other elements (Pt = 79.31 wt%, C = 8.65 wt%, Cu = 2.09 wt%, Si = 1.05 wt%, and O = 8.93 wt%) were also found in graphite-bearing metasomatic rocks in the Tamga and Turgenevo deposits. The noted admixture of C, O, and Si may be spurious and a consequence of the minute dimensions of the platinum crystals being set in a silicate–carbonaceous matrix.

Inclusions of globular gold in the graphite-bearing rocks are common, and an example of a spheroidal aggregate of gold with inclusions of thin graphite flakes is shown in Fig. 3. The SEM analyses revealed significant compositional variations in this gold grain with Au = 100.0–79.3 at%, Ag = 0.0–22.02 at%, and Cu = 0.0–2.2 at%. The flakes of graphite enclosed in the gold also have heterogeneous compositions according to the measurements at three points (C = 57.92–71.25 at%, Au = 0.46–17.4 at%, O = 28.2–30.3 at%, Cl = 0.25–2.06 at%, K = 0.0–2.05 at%, Ca, Si, and Al = 0.0–1.7 at%). A carbonaceous nanofilm (100–200 nm thick), revealed by SEM on the gold spheroid, contains an admixture of Cl, S, Ca, Al, and Fe up to 1–2 wt% in addition to the carbon (56–60 wt%) and oxygen (19–33 wt%). It is another example of the co-precipitation of gold and carbonaceous matter from an ore-bearing metamorphic fluid. Native gold in the skarns differs from that in the graphite-bearing rocks, and is characterized by its crystal form, compositions with higher contents of silver (up to 10 wt%), and larger-sized spongy and lamellar crystals (Fig. 3). Gold in the quartz veins contains admixtures of W (1.26 wt%), F (up to 34.66 wt%), and U (3.42 wt%), and these admixtures, including the fluorine, increase in amount from the centers of the grains outwards. The grain margins of the gold are coated with a fragmentary carbonaceous nanofilm ranging from 100 to 200 nm thick.

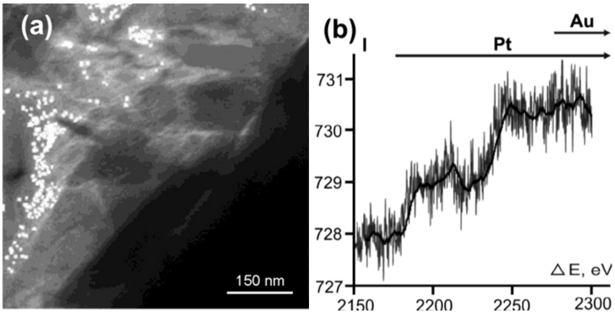


Fig. 1. (a) EELS image of graphite flake including an assemblage of dispersed nano- sized crystals of natural platinum. (b) Spectrum of characteristic losses of M4.5.

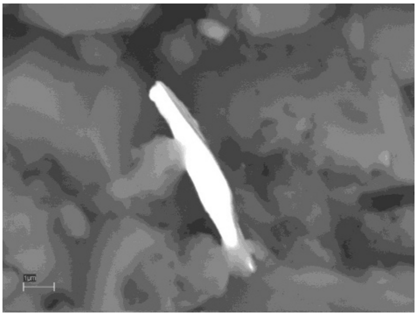

Fig. 2. SEM image of a thin prismatic microcrystal of isoferroplatinum set in a matrix of graphite and silicate material.

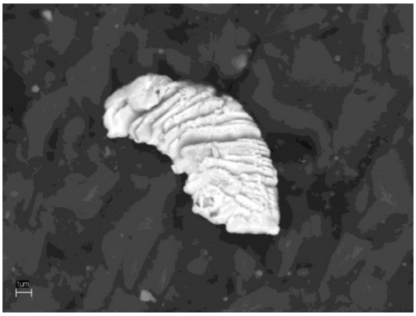

Fig.3. SEM image of gold lamellar crystals in the silicate–carbonaceous matrix of a graphite–sericite–quartz shale.

Along with platinum and gold, graphite-bearing rocks of the Khanka terrane contain many inclusions of native copper, zinc, nickel, bismuth, Ir–REE, Fe–Cr, Cu–Sn, Cu–Sn–Fe intermetallides, Ag minerals (AgI, AgCl, AgBr, Ag(Cl, Br, I), and $Hg_3Ag_2$), oxides (such as $Fe_3O_4$, $SnO_2$, and $YPO_4$), monazite, and other REE minerals [2,3]. Sulfides, such as pyrite, arsenopyrite, galena, and argentite, occur only as accessory, exotic minerals. This is an essential difference between the studied rocks and deposits of the black shale type that normally contain appreciable amounts of sulfides [5].

The quantitative determinations of gold and PGE, reported in Table 2, show that the contents of Au and PGE are elevated in all lithological varieties of the graphite-bearing metamorphic rocks. These data were further supported by AAS-TEA analyses of Pt, Pd, Au, and Ag in graphite-bearing lamprophyre, schist, granite gneiss, skarn, mica–quartz shale, phyllite-like shale, and an amphibolite (Table 3). The AAS-TEA method proved to be the most accurate, as the samples with extremely high Pt contents reveal the presence of multiple nanometric grains of platinum under the electronic microscope. The noble metal contents in the rocks vary significantly, so that Pt concentrations range from 0.33 to 62.63 ppm, with the highest amounts found in lamprophyres and the amphibolite. Au concentrations range from 0.001 to 4.75

ppm, Ag contents from 0.17 to 4.41 ppm, and Pd from 0.01 to 3.85 ppm. These data reflect the irregular distribution of noble metals in the rocks studied.

About 40 samples from the Khanka terrane were first analyzed by the oxidative fluoridation method. The contents (ppm) range as follows: Au (0.02–6.73), Ag (0.2–4.41), Pt (0.014–0.116), Pd (0.02–0.55), Ir (0.002–0.055), Os (0.011–0.09), Ru (0.007–0.2), and Rh (0.001–0.74). The results of the noble metal analyses using this method show that all Rh and most of the Au, Pt, and Pd passed into solution during the process of a single fluorine oxidation, whereas Ru, Os, and Ir, which were analyzed after repeated dissolutions, were completely retained in the insoluble graphitic residue of the first dissolution. The results of the stepwise decomposition of graphite-bearing samples confirm the incorporation of some of the noble metals into graphite, which suggests that the noble metals and the graphite were formed contemporaneously [3].

Table 3: Noble metal contents (ppm) in graphitic rocks of the Khanka terrane.

| Sample | Pt | Pd | Au | Ag | Rock |
|---|---|---|---|---|---|
| AP-1(1) | 57.28 | 1.30 | 2.08 | 3.37 | Lamprophyre |
| AP-1(2) | 16.86 | 1.29 | 0.28 | 1.35 | |
| AP-1(3) | 0.70 | 0.14 | 1.33 | 1.03 | Crystalline schist |
| AP-15 | 0.33 | 0.01 | 1.90 | 0.70 | Granite |
| AP-15/4A | 2.14 | - | 0.07 | 0.17 | Skarn |
| AP-15/11 | 1.83 | 0.31 | 0.15 | 0.34 | |
| AP-18/1 | 0.41 | 0.12 | 0.41 | 0.85 | Biotite–feldspar schist |
| AP-22/3 | 4.02 | 0.06 | 0.06 | 0.56 | Muscovite–biotite schist |
| AP-22/4 | 6.31 | 0.30 | 2.19 | 1.57 | |
| AP-24 | 16.05 | 0.51 | 0.15 | 0.65 | Biotite–graphite–sericite schist |
| AP-35/5 | 6.93 | 0.04 | 4.75 | 0.48 | Graphite–sericite–quartz shale |
| AP-36/2 | 62.63 | 0.98 | 0.11 | 4.41 | Amphibolite |
| AP-36/2a | 56.88 | 3.85 | 0.41 | 2.93 | |
| AP-36/4 | 3.29 | 0.18 | 0.05 | 0.50 | Phyllite |
| AP-38/8 | 1.40 | 1.19 | 0.001 | 0.63 | Phyllite |
| AP-40/4 | 1.62 | 1.00 | 0.06 | 3.80 | |

Note: analyses were performed using AAS TEA, and the samples were chemically decomposed by a mixture of concentrated acids.

Table 4: Trace element contents (ppm) of rocks from the Ussury and Mitrofanovka formations.

| Sample | V | La | Ce | Pr | Nd | Sm | Eu | Gd | Tb | Dy | Ho | Er | Yb | Th | U |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 02/3 | 4.20 | 22.81 | 45.15 | 5.70 | 20.18 | 4.92 | 0.79 | 4.68 | 0.80 | 4.49 | 0.89 | 2.18 | 2.12 | 25.80 | 8.09 |
| 03/3 | 170.8 | 41.80 | 92.63 | 11.15 | 45.60 | 9.00 | 1.99 | 9.24 | 1.27 | 9.22 | 1.80 | 4.73 | 5.00 | 12.96 | 2.92 |
| 04/8 | 196.9 | 47.08 | 101.5 | 12.17 | 48.64 | 8.65 | 1.46 | 6.78 | 0.90 | 5.21 | 1.08 | 3.16 | 3.96 | 14.16 | 2.29 |
| 04/40 | 351.9 | 54.87 | 86.83 | 13.65 | 53.78 | 12.17 | 2.47 | 12.79 | 1.93 | 12.06 | 2.78 | 7.98 | 8.33 | 12.07 | 8.71 |
| 04/101 | 91.03 | 47.81 | 105.2 | 12.26 | 45.05 | 10.48 | 1.27 | 10.51 | 1.62 | 9.81 | 1.69 | 4.41 | 3.84 | 39.47 | 13.80 |

Note: sample 02/3 is granite-gneisses; 03/3 is schist; 04/8 is amphibolite; 04/40 is graphite–sericite–quartz shales; 04/101 is lamprophyre. ICP–MS analyses were performed at the Analytical Center of the Far East Geological Institute, Russian Academy of Sciences, Vladivostok, Russia

The current analytical results, as well as the data we obtained by X-ray and Raman spectroscopy, provide grounds for concluding that there are at least two generations of graphite in the graphite-bearing rocks we have studied. The first generation is represented by nanocrystalline graphite with relatively low temperatures of exothermal maxima (860–900 °C), and the second by crystallites that are larger than 100 nm in size and which have relatively high temperatures of exothermal maxima (950–1000 °C), according to the thermogravimetric data [9].

Discrimination of the two graphite varieties on the basis of their physical parameters is consistent with the data obtained from their chemical decomposition (Table 1). AAS analyses demonstrate that the noble metals are mostly associated with graphite. Micro-flakes of graphite pass into solution during the first decomposition by hydrofluoric acid, while the large-flakes are more resistant to decomposition. The second-generation graphite concentrates all the silver, ruthenium, osmium, and iridium, while graphite of the first generation concentrates the gold, rhodium, and platinum (Table 3). The association of the second-generation graphite with metamorphic recrystallization is illustrated by the presence of graphite veins in biotite, and of biotite inclusions in graphite. Thus, the graphitization in the Tamga and Turgenevo deposits, as well as the associated noble-metal mineralization, was a result of polychronous and polygenetic processes.

A wide range of REE and trace element contents were determined with ICP-MS using an Agilent 7500 spectrometer (some of them are listed in Table 4).

In total, three sources of noble-metal mineralization can be distinguished: (1) an endogenic magmatic fluid related to mantle–crustal magma chambers, (2) an exogenic sedimentary–chemogenetic source related to the formation of the graphite–sericite–quartz shales of the Mitrofanovka Formation, and (3) a metamorphic source related to the transfer of metals during regional metamorphism.

For the first time this research demonstrates possibility for useful components extraction from graphite ore and specific distribution of gold in course of preliminary concentration, as well as at various stages of chemical processing of graphite-bearing rocks. It also demonstrates that application of hydrodiflouridation technique to graphite-bearing rocks containing gold allows not only gold concentration, but useful by-components extraction as well [7]. Rare elements uranium and thorium can be extracted with complex processing. The technology for platinum extraction has not been invented yet.

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# Part II
# Coal Preparation Plants Design

# Design of coal preparation plants: problems and solutions

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Annotation.

With the increase of coal production, the requirements to its quality are also increasing. Improvement of coal quality is only possible with the use of coal preparation methods. The main objective is to achieve 100% treatment volume of the mined coal.

In recent years, only in the Kuznetsk Basin 15 coal preparation plants have been built.

Addressing coal research and design is an important stage in construction and operation of coal preparation plants.

At present time the calculation of flow sheets is based on the initial data of the study of operational samples (industrial) samples, taken directly from the mine (open cast) without taking into account breakage on handling of coal from the mine (open cast) to the coal preparation plant. Analysis of CPPs operation has shown that the design data do not meet the actual performance data. One of the reasons is the wrong approach to flow sheets calculations and to the choice of equipment.

The report presents the design problems and their solutions.

During the process of coal transportation from the mine (open cast) to the CPP the coal grinding occurs. Its composition varies in both size and quality of individual size grades.

Size grades washability changes, thus affecting the choice of preparation methods and equipment.

The report presents examples of qualitative changes of the mined coal during transportation and recommendations on the correct choice of input data for the design of new CPPs and reconstruction of the existing ones.

**Keywords: coal preparation plants, particle size, grinding, designing, problems, solutions, balance.**

Coal mining industry is one of the most important branches of the fuel and energy complex.

Production levels have increased by more than 350 million tons per year over the past 10 years.

Much attention is paid to both coal mining and its preparation.

Recently only in Kuznetsk Basin 15 coal preparation plants of new generation have been built.

A significant progress in design, construction and operation of coal preparation systems resulted in the increase of overall efficiency in processes, productivity, and reductions of negative costs per ton of clean coal. The design of a secure, cost-effective and efficient system of coal preparation requires taking into account such factors as the initial data for flowsheets, the choice of equipment and the equipment arrangement.

The flowsheet design is based on the data of complex research of washability and the quality characteristics of raw coal fed for washing. Complex investigations of raw coal include the study of an in-mine sample, a coal sample after grinding and sludge formation in the washing process. Typically, design institutes use the data of particle size and float/sink sampling taken from the mine (open cast) stock dump for the flowsheet design.

Practically after start-up of a new coal preparation plant, design data do not correspond to the actual ones obtained during preparation.



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This happens due to the following factors.

First - the lack of ROM stockpiles. Coal receiving to a preparation plant is carried out without a proper charge, corresponding to the design specifications.

Second – size degradation is not taken into account during the delivery from the mine (open cast) stock dump to the coal preparation plant.

Raw coal breakage takes place at the mine (open pit) stock dump on the way from the mine (open pit) to the coal preparation plant.

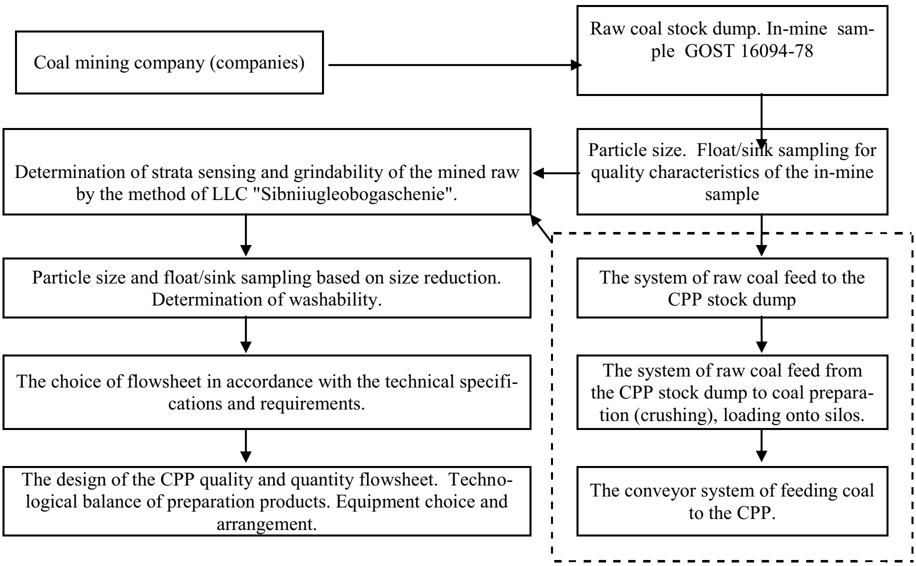
Size degradation occurs during loading coal into rail cars, dump trucks, transportation from the preparation plant stock dump and loading onto silos, at grinding of coarse coal over +200 (100) mm, during supplying coal to the preparation plant by a belt conveyor.

Coal size degradation depends on strata sensing and grindability.

Figure 1 shows a diagram of the main stages of a coal sample preparation for the study and providing the initial data for the design of a coal preparation plant.

Figure 1. Diagram of data preparation for CPP design.

As you can see on Figure 1, samples taken directly from the mine, open cast stock dump differ by particle size and, consequently, float/sink sampling changes during transportation to the CPP.




Coal grinding is paid great attention in the world [1, 2, 3].

Size reduction takes place during extraction and transportation of coal to the surface and the movement of coal on the surface.

It is specified [3] that the relative size degradation of raw coal of +13 mm is:

- when loading onto the silo (height difference - 1 m, silo height - 6 m, rail car height - 1.5-2 m) - 10.1%;
- when loading via feeding cone (height difference on inclined part - 6m, vertical - 4m, the cone height - 3 m) - 9%;
- when loading onto the rail car - 1.6%.

The increase of fines of 0-13 mm size makes respectively 6.1%; 4.1% and 0.9%.

The institute "Sibniiugleobogaschenie" ("Kuzniiugleobogaschenie") has recently performed many studies devoted to simulation of coal grinding processes to obtain reliable data for CPP design.

We offer methods of determination of coal size degradation during delivery from mine, open pit stock dumps to the preparation plant. [4,5]

The root of the method of determination of coal size degradation during delivery from mine, open cast stock dumps to the preparation plant is in mechanical impact on the coal in a revolving closed drum and then measuring the amount and quality of standard size grades.

Evaluation criterion of simulated industrial grinding in a laboratory is the amount and quality of the resulting fines under 0.5 mm size for fixed time. Simulation parameter determining the coal grinding is equivalent to the time the in-mine sample spent in the drum having the diameter of 1000 mm, and the length of 500 mm. The inner surface of the cylinder has three welded strips of steel of 5 mm thick, with a rib height of 250 mm, arranged at equal distances from each other. Rotational speed is 25 ± 1 rpm (GOST 15490-70). A DC motor with a thyristor driver allows regulating the number of revolutions. The equivalent time of grinding simulation in a drum in the study of in-mine coal samples with the existing technology of mining mechanization is as follows:

- underground mining and transportation from the face to the preparation plant, including conveyors, rail cars, recrushing and storage - 3 min.;
- open-pit mining - 5 min.

Table 1 and Figure 2 show the results of the screen analysis of in-mine samples and samples after grindability analysis.

Table 1

Coal particle size after grinding simulation

| Grades, mm | Raw coal | | After grinding | |
|---|---|---|---|---|
| | yield % | ash % | yield % | ash % |
| 150-200 | 7,4 | 11,8 | 4,3 | 14,6 |
| 100-150 | 24,0 | 21,3 | 13,8 | 26,4 |
| **Over 100** | **31,4** | **19,0** | **18,1** | **23,6** |
| 75-100 | 16,8 | 28,7 | 9,7 | 35,6 |
| 50-75 | 23,3 | 21,1 | 13,4 | 26,2 |
| 40-50 | 4,8 | 21,5 | 2,8 | 26,7 |
| **Over 40** | **76,3** | **22,0** | **44,0** | **27,3** |
| 25-40 | 9,1 | 19,4 | 15,5 | 20,9 |
| 13-25 | 7,9 | 19,9 | 12,1 | 15,5 |
| 6-13 | 2,9 | 19,1 | 8,8 | 15,1 |
| 3-6 | 1,6 | 17,3 | 6,3 | 13,7 |
| 2-3 | 0,5 | 15,5 | 2,2 | 12,5 |
| 1-2 | 0,6 | 17,0 | 3,9 | 15,3 |
| 0,5-1 | 0,4 | 17,5 | 2,6 | 14,6 |
| 0-0,5 | 0,7 | 22,4 | 4,6 | 16,2 |
| **0-40** | **23,7** | **19,3** | **56,0** | **16,6** |
| **Total** | **100,0** | **21,3** | **100,0** | **21,3** |

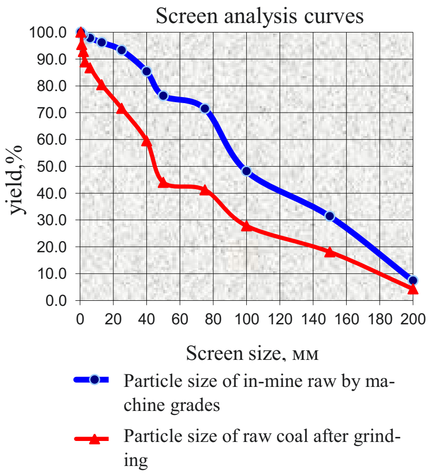



Fig. 2 Screen analysis curves

As can be seen from the table and the figure after simulation of raw coal samples grindability the yield of size grades varies with the decrease in the yield of large grades (+40 mm) from 76.3% to 44.0% and an increase in small grades. The yield of 0-0.5% grade rose from 0.7% to 4.6%.

Ash content also varies by size grades depending on particle size of coal and rock.

Tables 2, 3 and Figures 3 and 4 show float/sink sampling of in-mine coal samples and post-simulation of grindability by machine grades (GOST 10100-84)

Due to changes in particle size washability varies by machine size grades. (GOST 10100-84)

Table 2

Data for the construction of Mayer washability curve (grade over 40 mm)

Without grinding

| Fraction density kg/m³ | Yield, % | Ash, % | $\gamma \cdot A^d/100$ | Coordinates | |
|---|---|---|---|---|---|
| | | | | $\Sigma\gamma \cdot A^d/100$ | $\Sigma\, \gamma \cdot$ |
| Under 1300 | 55,57 | 4,9 | 2,7 | 2,7 | 55,57 |
| 1300 – 1400 | 10,92 | 15,3 | 1,7 | 4,4 | 66,49 |
| 1400 – 1500 | 6,55 | 23,1 | 1,5 | 5,9 | 73,04 |
| 1500 – 1600 | 2,43 | 35,0 | 1,8 | 6,8 | 75,47 |
| 1600 – 1800 | 4,39 | 41,7 | 13,4 | 8,6 | 79,86 |
| Over1800 | 20,14 | 66,5 | 22,0 | 22,0 | 100,0 |
| **Total:** | **100,0** | **22,0** | **22,0** | **22,0** | **100,0** |
| With grinding | | | | | |
| Under 1300 | 47,4 | 4,6 | 2,2 | 2,2 | 47,4 |
| 1300 – 1400 | 9,3 | 14,4 | 1,3 | 3,5 | 56,7 |
| 1400 – 1500 | 5,6 | 21,7 | 1,2 | 4,7 | 62,3 |
| 1500 – 1600 | 3,1 | 35,7 | 1,1 | 5,8 | 65,4 |
| 1600 – 1800 | 5,6 | 42,5 | 2,4 | 8,2 | 71,0 |
| Over 1800 | 29,0 | 65,8 | 19,1 | 27,3 | 100,0 |
| **Total:** | **100,0** | **27,3** | **27,3** | **27,3** | **100,0** |

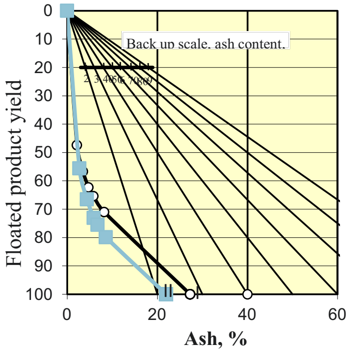
Table 3

Data for the construction of Mayer washability curve (grade 0 – 40mm)

Without grinding

| Fraction density kg/m³ | Yield, % | Ash, % | $\gamma \cdot A^d/100$ | Coordinates | |
|---|---|---|---|---|---|
| | | | | $\Sigma\gamma \cdot A^d/100$ | $\Sigma\, \gamma \cdot$ |
| Under 1300 | 58,3 | 3,7 | 2,2 | 2,2 | 58,3 |
| 1300 - 1400 | 11,3 | 11,5 | 1,3 | 3,5 | 69,6 |
| 1400 - 1500 | 5,1 | 19,7 | 1,0 | 4,5 | 74,7 |
| 1500 - 1600 | 3,2 | 29,3 | 0,9 | 5,4 | 77,9 |
| 1600 - 1800 | 4,4 | 42,3 | 1,9 | 7,3 | 82,3 |
| Over 1800 | 17,7 | 68,0 | 12,0 | 19,3 | 100,0 |
| **Total:** | **100,0** | **19,3** | **19,3** | **19,3** | **100,0** |
| With grinding | | | | | |
| Under 1300 | 58,7 | 3,3 | 1,9 | 1,9 | 58,7 |
| 1300 - 1400 | 14,6 | 8,9 | 1,3 | 3,2 | 73,3 |
| 1400 - 1500 | 6,0 | 16,9 | 1,0 | 4,3 | 79,3 |
| 1500 - 1600 | 2,9 | 30,1 | 0,9 | 5,1 | 82,2 |
| 1600 - 1800 | 3,8 | 44,0 | 1,7 | 6,8 | 86,0 |
| Over 1800 | 14,0 | 69,8 | 9,8 | 16,6 | 100,0 |
| **Total:** | **100,0** | **16,6** | **16,6** | **16,6** | **100,0** |

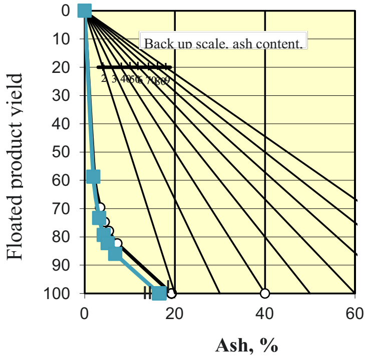
Mayer washability curves over +40 mm grade




I- In-mine raw coal washability curve
II- Washability curve after grinding simu

Fig. 3

Mayer washability curves 0- 40 mm




I - In-mine-raw coal washability curve
II - Washability curve after grindability simulation

Fig. 4

If before grindability simulation a grade over 40 mm had an intermediate category of washability, after – the washability the category is difficult. The washability category affects the choice of coal washing methods.

Tables 4 and 5 show the fractional analysis of machine grades with and without grinding

Table 4

Fractional analysis of raw coal by machine grades without grinding

| Fraction density kg/m³ | Grade over 40 mm | | | Grade 0 - 40 mm | | |
|---|---|---|---|---|---|---|
| | Yield, % to | | Ash, % | Yield, % к | | Ash, % |
| | grade | raw | | grade | raw | |
| under 1300 | 55,6 | 42,4 | 4,9 | 58,3 | 13,8 | 3,7 |
| 1300 - 1400 | 10,9 | 8,3 | 15,3 | 11,3 | 2,7 | 11,5 |
| 1400 - 1500 | 6,6 | 5,0 | 23,1 | 5,1 | 1,2 | 19,7 |
| 1500 - 1600 | 2,4 | 1,9 | 35,0 | 3,2 | 0,8 | 29,3 |
| 1600 - 1800 | 4,4 | 3,4 | 41,7 | 4,4 | 1,0 | 42,3 |
| over 1800 | 20,1 | 15,4 | 66,5 | 17,7 | 4,2 | 68,0 |
| Total | 100,00 | 76,30 | 22,0 | 100,00 | 23,70 | 19,3 |

Table 5

Fractional analysis of raw coal by machine grades with grinding

| Fraction density kg/m³ | Grade over 40 mm | | | Grade 0 - 40 mm | | |
|---|---|---|---|---|---|---|
| | Yield, % к | | Ash, % | Yield, % к | | Ash, % |
| | grade | raw | | grade | raw | |
| under 1300 | 47,4 | 20,9 | 4,6 | 58,7 | 32,9 | 3,3 |
| 1300 - 1400 | 9,3 | 4,1 | 14,4 | 14,6 | 8,2 | 8,9 |
| 1400 - 1500 | 5,6 | 2,5 | 21,7 | 6,0 | 3,4 | 16,9 |
| 1500 - 1600 | 3,1 | 1,4 | 35,7 | 2,9 | 1,6 | 30,1 |
| 1600 - 1800 | 5,6 | 2,5 | 42,5 | 3,8 | 2,1 | 44,0 |
| over 1800 | 29,0 | 12,8 | 65,8 | 14,0 | 7,8 | 69,8 |
| Total | 100,00 | 44,00 | 27,3 | 100,00 | 56,00 | 16,6 |

The tables show that the large grade yield over 40 mm decreased from 76.3% to 44%, and vice versa the grade 0 - 40 mm increased from 23.7% to 56%. Almost twice. Naturally, the calculation of the quality-quantity and water - sludge flow sheets should provide for the increased amount of small grades washing. Without taking into account this circumstance, the design does not provide complete coverage of preparation.

As a result, the technological efficiency of preparation is reduced, which leads to considerable losses of raw coal with tails.

Based on fractional analysis of raw coal for the machine grades without grinding and with grinding, we present a theoretical balance of preparation (tables 6, 7)

Table 6

Theoretical balance of preparation products of grade over 40 mm

| Preparation product | Without grinding | | With grinding | |
|---|---|---|---|---|
| | Yield, % | Ash, % | Yield, % | Ash, % |
| Concentrate | 66,5 | 6,6 | 56,7 | 6,2 |
| Intermediate product | 13,4 | 31,4 | 14,3 | 32,9 |
| Waste | 20,1 | 66,1 | 29,0 | 65,8 |
| Total | 100,0 | 22,0 | 100,0 | 27,3 |
| | Washability T=16,7% | | Washability T=20,1% | |

Table 7

Theoretical balance of preparation products of grade 0 - 40 mm

| Preparation product | Without grinding | | With grinding | |
|---|---|---|---|---|
| | Yield, % | Ash, % | Yield, % | Ash, % |
| Concentrate | 69,6 | 5,0 | 73,3 | 4,4 |
| Intermediate product | 12,7 | 29,9 | 12,7 | 28,0 |
| Waste | 17,7 | 68,0 | 14,0 | 69,8 |
| Total | 100,0 | 19,3 | 100,0 | 27,3 |
| | Washability T=16,7% | | Washability T=20,1% | |

The choice flowsheet and equipment depends on coal washability.

Coal washability is characterized by its ability to be divided into components by density: concentrate, intermediate product and waste.

The method of washability determination is based on the results of fractional analysis. Fractional analysis is performed in accordance with GOST 4790-93 [6 - 10].

Thus,

Inconsistency of data obtained for the design of a preparation plant and the actual data obtained during its operation can be explained primarily by the fact that the quality- quantity and water - sludge calculations are based on the particle size distribution and float/sink sampling analysis of the in-mine samples taken directly from the mine, open pit stock dumps without taking into account coal crushing during handling before it gets the preparation plant.

The example shows that breakage produces an effect on the category of coal washability and, consequently, on the choice of flowsheet, equipment and its arrangement. Flowsheet design requires consideration of sludge formation during washing with water, a so-called wet washing method. A comprehensive study of raw coals besides grindability includes sludge formation. Sludge formation is provided in the temporary engineering standard (VNTP-3-92), which is usually taken into account at the design of water-slurry circuits. Lack of stockpiles for obtaining proper charge is a disadvantage in the designs of coal preparation plants.

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# Implementation of modern coal processing solutions in the project of "Elegest" coal preparation plant

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**Abstract**
This paper describes the 18.6 million tons per annum (MTPA) raw coking coal preparation plant "Elegest", which is part of an underground mining and coal processing complex of the same name (MPC "Elegest"). The following problems are solved in the project: to minimize the losses of clean coal during the preparation, to ensure the possibility of obtaining two types of clean coal product with the different ash content to meet market needs, to ensure the standard product moisture in the winter time due to the application of special coal slime dewatering circuits and high-performance mechanical dewatering equipment, to minimize the amount of thermal drying. To this end the range of product fractions is optimized, so as a result, it became possible to dry only clean coal 0-1.5 mm using the innovative vertical pipe jet-dryers. As a result of complex solutions for technological tasks the raw coal losses with waste meet the standards of design of new plants. In addition, technological processing solutions include the joining of the special ash additives into low-ash clean coal to adjust the chemical composition of the ash in order to improve CSR and CRI indicators.
**Keywords:** clean coal, ash content, moisture, coal preparation, dewatering, drying, flowsheet, heavy medium cyclone, spiral, flotation, waste, ash-additives

## Introduction

In 2014 JSC "Tuva Energy Industrial Corporation" ordered a construction project for mining and processing complex (MPC) "Elegest", which provides for underground mining and preparation of coking coal of Western wing of Elegest coal deposit. The raw coal base for the preparation plant is coke fat coal from "Ulug" seam. Currently, the company "Coralina Engineering" completed the project of coal preparation plant (the General Designer of the mining and processing complex "Elegest" - LLC "SGP").

The project has the following tasks:

- To minimize the loss of produced coal, optimize layout solutions that allow a smaller area and less equipment to process a large volume of raw coal;
- To take into account strong enough fluctuations in the characteristics of raw coal produced from different coal seams;
- To provide production of high quality clean coal of different ash contents to best meet the needs of the market;
- To maximize the effectiveness of coal preparation and dewatering, to provide a closed water circuit, to minimize harmful emissions into the environment.

## Estimation of the complex capacity

Ulug-Khem coal basin is located in the central part of the Republic of Tuva. Stock of the Western part of the deposit is estimated at 1 billion tons of coal. To operate MPC "Elegest" it is constructing railways "Elegest-Kyzyl-Kuragino" length of 440 km and a coal handling terminal in the port of "Vanino" (Khabarovsk region). The clean coal produced by CPP will be supplied to Japan, China, South Korea and India.

The construction of the coal preparation plant (CPP) will be realized in two stages. After the commissioning of the first stage starting complex the capacity of the CPP will be 6.0 MTPA (840 TPH) of raw coal. After the commissioning of the second starting complexes the final capacity of the CPP will be 18.6 MTPA (2600 TPH) of raw coal. The ash content of raw coal may be varying - average from 14 to 24%.



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The production of two salable grades of clean coal - of 8.0% and 6.5% ash content will be up to 15 million tons per year. The designed capacity is planned for 2019.

**The coal preparation plant description**

The CPP layout has a three-section structure. Technological processing equipment located in independent pre-washing buildings of raw coal crushing and one (main) building of wet washing processing. Each of the three sections is further divided into two sub-sections with individual closed water circuit. For four of six sub-sections available provides for the construction of a separate drying compartment building for thermal drying the concentrate of the slime size 0-1.5 mm. The final concentrate after mixing large and dried concentrates is transferred in the storing and loading units of finished products by two conveyors. There is an ability of separate loading of 8.0% and 6.5% ash content concentrates into railway carriages. To minimize the facilities area and operating costs at the site will be installed a high-efficiency and high-performance equipment of large sizes. Moreover, given the trend towards import substitution, equipment of domestic production will reach half of the total supply.

The large volume of raw coal, possible fluctuations in his quality, work, both at low and at high density separation, determined the choice of the main method of coal preparation - wet washing in heavy-medium cyclones of large diameter - 1150 mm [1-4]. Stable operation and separation efficiency is achieved by using an automated control system of magnetite suspension density [5].

The flowsheet suggests that the run of mine coal of the average grain size 0-200 mm is crushed to 0-75 mm and divided into three machine size:

- Coarse coal of 1,5-75 mm - is washed in heavy medium cyclones;
- Fine coal of 0.3 (0.04)-1.5 mm - is washed in seven-turn spirals;
- Ultrafine coal of 0-0.3 (0.04) mm - is processed by flotation.

The processing plant can operate in two modes:

- Mode #1 - coal preparation is carried out in a single stage, in HM cyclones at a high 2.0 specific density of separation to produce the clean coal of 8% ash content;
- Mode #2 - coal preparation is carried out in two stages. In the first stage - in HM cyclones at a low 1.35 specific density of separation to produce low ash clean coal and mixed product. Then mixed product is crushed and washed in HM cyclones of second stage separation at a high 2.0 specific density of separation to produce middlings and wastes. Middlings is subject to grain open by grinding in a 4-drum crusher and further separation in spirals. The ash content of the clean coal when plant is operating at Mode #2 will be 6.5%.

In winter, the clean coal is produced with a total moisture of 7.5%, in the summer - with 9.5% total moisture. Part of the clean coal of 0-1.5 mm is dried in advanced technology pipe-dryers with gases recirculation.

The plant water-slurry circuit is closed, with high- and low-ash pulps thicken separately in radial thickeners and further dewatering of thickened products and full clarification of recycled water, which minimizes the loss of a fine low-ash coal and significantly reduces the need in additional water resources.

**The coarse coal washing circuit**

Raw coal of size 0-200 mm is classified in the building of dry pre-washing preparation by size of 75 mm on the two inclined vibrating screens. Oversize product of 75-200 mm is crushed to the size of 0-75 mm, and transferred to the main building for wet desliming by cut size of 1.5 mm.

Oversize material of size +75 mm from desliming screens top sieves is largely performed by foreign objects coming from the mine, which are sent to waste. Given the low structural strength of coal and using of roll crushers with 3-D crushing technology, with high probability can be assumed that in oversize product of +75 mm there will not be any coal, and that in this product will get such foreign objects like pieces of wood, cable, wire, rubber, plastic and so on. Oversize deslimed material of size 1,5-75 mm from bottom sieve is sent to the sump feeding heavy medium cyclones of the first stage of washing. The slime of particle size 0-1.5 mm is transferred for hydraulic classification to the feed sump of classifying cyclones.

The coarse coal is preparation in one stage at a 2.0 specific gravity of separation in heavy-medium cyclones of 1150 mm diameter with producing of waste and concentrate of 1,5-75 mm. Concentrate from heavy-medium cyclones is washed from magnetite and classified by cut size 25 mm at the double-deck drain & rinse "banana-type" screens. Oversize material from top sieves, which is the finished clean coal of size 25-75 mm - is sent to the clean coal conveyor, and transfer to the clean coal warehouse. Oversize material from bottom screens sieves, which is the concentrate of size 1,5-25 mm, finally dewatered in horizontal filtering centrifuges.

The flow-sheet of the coarse washing circuit is shown in Figure 1.

**Fine coal circuit**

Raw coal slime of 0-1.5 mm size is fed for the hydraulic classification into four blocks of 610 mm diameter cyclones, where classified by the cut size of 0.3 mm, or fed into four blocks of 150 mm diameter cyclone, where classified by the cut size of 0.04 mm, with selection the classification underflow products of size 0.3-1.5 mm and 0.04-1.5 mm, respectively. Underflow products of hydraulic classification of different sizes are washed separately in high-performance 7-turn spirals to obtain a clean coal and waste of size 0.04-1.5 mm [6]. There is an ability of co-operation of blocks of 610 mm and blocks of 150 mm cyclones. This method of classification of fine slimes can significantly reduce the amount of material being fed to flotation without reducing its effectiveness. Classification cyclones overflow products are sent to the flotation circuit.

Clean coal of spirals of size less 1.5 mm, after preliminary dumping excess water on the sieve bend, is fed for dewatering into screen-bowl centrifuges. Dewatered concentrate via a separate belt conveyor is sent for the thermal dryers. Effluents of centrifuges, as well as undersize products of sieve-bends are pumped into the low-ash slimes radial thickeners.

Waste of spirals of size less 1.5 mm is drifted by gravity for dewatering at high frequency screens with mesh size of 0.3 mm, and then wastes of 0.3-1.5 mm size are sent to the waste conveyor.

High frequency screens undersize product enters the high-ash slime collectors and further, to prevent enter of large particles of waste in radial thickeners, - for thickening in cyclones diameter of 150 mm. Underflow products of cyclones are sent back into high-frequency screens and a thin cyclones overflow is drifted in radial thickeners.

The flowsheet of fine coal washing circuit is shown in Figure 2.

**Ultrafine coal circuit**

Classification cyclones overflow is fed for flotation into special pneumatic-mechanical flotation machines and traditional six-chamber mechanical machines, which allow successfully process diluted pulp to produce flotation concentrate and chamber product [7,8]. The chamber product is sent for further flotation into six-chamber mechanical flotation machines. From flotation machines foam catchers the flotation concentrate enters the de-aerating column, of which via special foam pumps is pumped into screenbowl centrifuges of spirals concentrate dewatering and, in part, to the feed sumps of chamber filter-presses. Flotation tailings by gravity drifted into the high-ash slime radial thickeners.

Thickening of low-ash slimes is carried out in radial thickeners with the use of flocculants. Underflow is dewatered in a chamber filter-press. The resulting low-ash cake-1 combined with the slime size concentrate from screenbowl centrifuges and is sent to the drying compartment. The filtrate of chamber filter-presses is sent into the high-ash slime radial thickeners. High-ash slimes are thickened in central drive radial thickeners using anionic and cationic flocculants. Underflow product is dewatered on belt filter presses with flocculants pre-treatment.

The high-ash cake-2 from belt filter-presses is transported into the cake storage. Filtrates of belt filter-presses enter by gravity into the high-ash slime radial thickener. Radial thickeners overflow is clarified circulating water, and through the sump of circulating water is sent back into the technological process.

**Innovations**

In project there was proposed to implement one of the most important innovations of our company - a method for improving the properties of coking coal, in particular, such indicators of coke like CSR and CRI [9,10]. The essence of development - in joining of the special ash additives to low-ash clean coal on

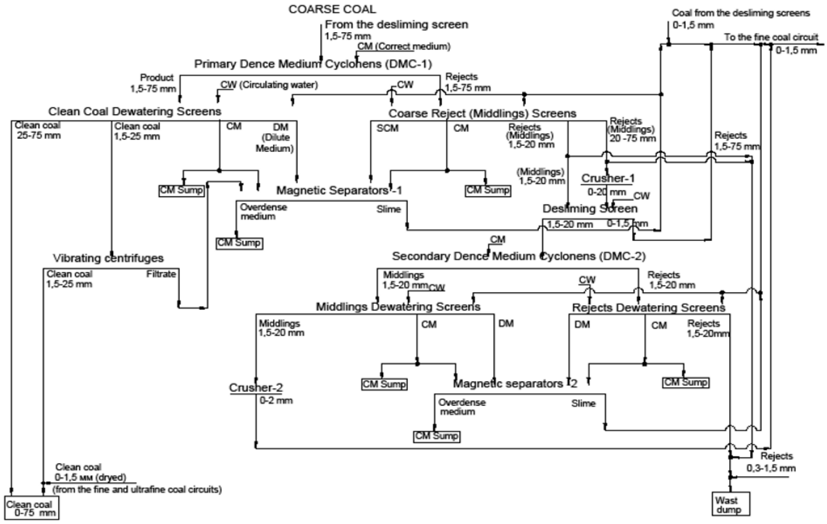


**Figure 1. Coarse coal circuit**

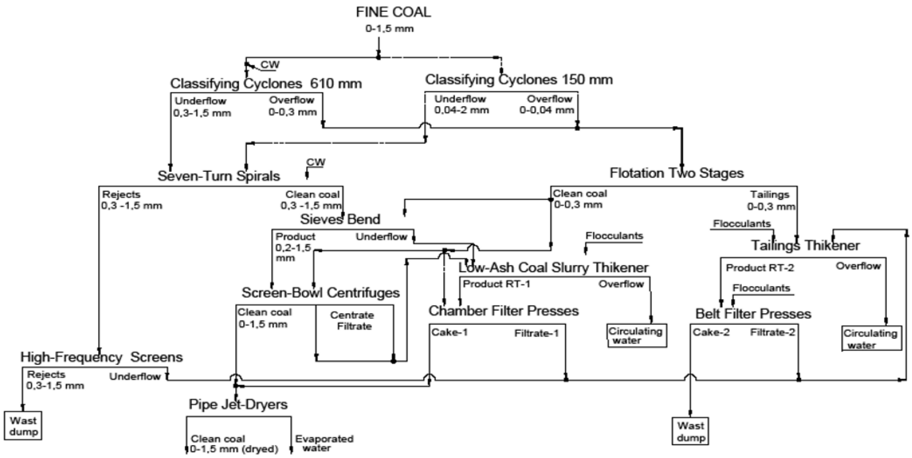


**Figure 2. Fine and ultrafine coal circuit**

the exit of the preparation plant. In order to implement improvements in coke CRI and CSR was developed a Mode-2 of the washing plant operation to obtain a clean coal with the lowest possible achievable ash content of 6.5%.

The use of ash additives to Elegest coal was tested in VUHIN Institute in parallel with the blend tests of Elegest coal and low-caking coal in proportion 60/40. The tests have shown that the addition of ash additives in the amount of 2.5-3% of the Elegest clean coal gives a result almost equivalent addition to 40% of the volume of low-caking clean coal from other deposits. It speaks about the prospects of the use of ash-additives on production stage and will contribute to a significant increase in competitiveness of the Elegest preparation plant production and an increase in sales of clean coal volume.

Another promising area is the use of innovative vertical pipe jet-dryers with an outside diameter of 2700 mm for the drying the clean coal of 0-1.5 mm size [11,12]. Pipe-dryer is operated by an advanced technology of gas recirculation and use as fuel a coal dust. The drying agent is a gas resulting from the combustion of coal dust in the low-inertia gasifiers. The required amount of coal dust is taken from the dry dust collection system of the dryer.

**Conclusions**

The adopted technological solutions at the stage of the project enabled the obtaining of competitive quality products. Project losses of coal in tailings were less than 1%, which corresponds to modern design standards of coal preparation plants.

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# SLUDGE FORMATION COEFFICIENT FOR A GIVEN PROCESS FLOW OF A COAL PREPARATION PLANT

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**Abstract**

Planning of quality and quantity indicators of run-of-mine coals at coal preparation factories requires taking into account the process flow specific sludge formation coefficient. Only in this case the difference between planned and real output of coal products will be minimal.

We suggest determining the sludge formation coefficient for a given process flow from changes in ratio of 1+ mm grade contained in coal preparation products before and after preparation on the basis of material balance of inputs and outputs.

The calculation of quality and quantity indicators of coal preparation for coal sourced from a number of coal mines at a typical coal preparation plant with and without taking into account the sludge formation coefficient (assumed as 10) showed that the planning error (overestimation of commercial product volume) is from 1 to 9% with the average of 2.8% for the 29 considered collieries.

Thus, a calculation for concentrate output from preparation of run-of-mine coal coming from Mezhirichanskaya mine at Chervonogradskaya coal preparation plant with and without accounting for sludge formation showed that output indicator reduced to ash content of 25% was overestimated in planning by 1.42% [8].

**Key words**

Coal, plant, process diagram, sludge formation, factor, technique.

When coal plants operate on client-supplied raw materials the role of correct calculations of expected quality and quantity indicators of coal preparation products increases. Sludge formation accounting may become an issue between producers and processors at the settlement stage.

It is widely known that efficiency of the preparation process decreases with lowering of fragmentation class. Thus, $E_{pm}$ for preparation of coal with aggregate size of 13–100 mm in float-and-sink separators is 60 kg/m$^3$, while for preparation of coal with aggregate size of 0.5–1 mm in a spiral separator it reaches 200–300 kg/m$^3$ [1, 2]. This is the evidence that loss of pure coal substance in preparation is five time higher for coals with aggregate size of 0.5–1 mm than for those of 13–100 mm. The demonstrated trend has a significant influence over the commercial yield. However, Ukrainian standard SOU 10.1.00185755.002-2004 *Coal Benefication Products. Method to Calculate Quality Indicators* [3] does not take into account the sludge formation thus overestimating planned quality and quantity indicators of commercial product and making them practically unreachable for existing coal preparation plants.

Let us consider a typical coal preparation plant that prepares run-of-mine coal by heavy medium separation and hydraulic jigging. To simplify the calculations let us assume that the preparation processes result in just two products: coal concentrate and refuse coal. At that the separation density is 1800 kg/m$^3$ for both large (13+ mm) and small (1-13 mm) fragmentation class. The volumes of the preparation process products are calculated in accordance with SOU [3]. Sludge collection is done following the method described in [5].

Calculation of preparation products was carried out twice: with and without taking into account the sludge formation, at that sludge formation coefficient for the process flow of the coal preparation plant was taken as 10%. Calculation of granulometric and fractional content of the run-of-mine coal with sludge formation coefficient was carried out in accordance with [4].

Granulometric and fractional content of run-of-mine coals from a number of coal mines is taken from data in the possession of the authors.

Table 1 shows balance of run-of-mine preparation products for the coal mines in question and a typical coal preparation plant and Table 2 shows results of planning error calculation. In the calculations of error



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the ash content obtained in plant's operation with sludge formation was expressed through the concentrate ash content without sludge formation.

*Table 1.*

**Expected balance of preparation products from run-of-mine coals from selected mines at a typical coal preparation plant**

| No. | Mine name | Without sludge formation | | | | | | With sludge formation | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Coal Concentrate | | Refuse Coal | | Total | | Coal Concentrate | | Refuse Coal | | Total | |
| | | $\gamma'_{к.}$ | $A^{d'}_{к}$ | $\gamma'_{отх.}$ | $A^{d'}_{отх.}$ | $\gamma'_{р.у.}$ | $A^{d'}_{р.у.}$ | $\gamma'_{к.}$ | $A^{d'}_{к}$ | $\gamma'_{отх.}$ | $A^{d'}_{отх.}$ | $\gamma'_{р.у.}$ | $A^{d'}_{р.у.}$ |
| 1 | Rodinskaya | 55.0 | 12.3 | 45.0 | 70.3 | 100.0 | 38.4 | 56.4 | 16.0 | 43.6 | 67.4 | 100.0 | 38.4 |
| 2 | Krasnolimanskaya | 63.2 | 9.7 | 36.8 | 71.7 | 100.0 | 32.5 | 62.1 | 11.3 | 37.9 | 67.2 | 100.0 | 32.5 |
| 3 | Im. Dzerzhinskogo | 56.0 | 11.4 | 44.0 | 64.4 | 100.0 | 34.7 | 63.2 | 20.8 | 36.8 | 58.6 | 100.0 | 34.7 |
| 4 | Severnaya | 62.9 | 19.7 | 37.1 | 61.3 | 100.0 | 35.1 | 62.0 | 20.6 | 38.0 | 58.7 | 100.0 | 35.1 |
| 5 | Torezskaya | 56.4 | 11.3 | 43.6 | 67.0 | 100.0 | 35.6 | 56.0 | 13.2 | 44.0 | 64.2 | 100.0 | 35.6 |
| 6 | No. 4 VM | 32.4 | 19.2 | 67.6 | 73.8 | 100.0 | 56.1 | 34.6 | 23.7 | 65.4 | 73.2 | 100.0 | 56.1 |
| 7 | No. 6 VM | 42.9 | 22.7 | 57.1 | 71.0 | 100.0 | 50.3 | 44.0 | 25.4 | 56.0 | 69.9 | 100.0 | 50.3 |
| 8 | No. 7 VM | 45.4 | 29.5 | 54.6 | 70.9 | 100.0 | 52.1 | 40.9 | 25.7 | 59.1 | 70.4 | 100.0 | 52.1 |
| 9 | No. 8 VM | 40.9 | 18.7 | 59.1 | 70.0 | 100.0 | 49.0 | 42.1 | 21.7 | 57.9 | 68.9 | 100.0 | 49.0 |
| 10 | Kirovskaya | 61.4 | 7.7 | 38.6 | 67.4 | 100.0 | 30.7 | 60.7 | 9.4 | 39.3 | 63.7 | 100.0 | 30.7 |
| 11 | Im. Skochinskogo | 52.1 | 8.0 | 47.9 | 64.6 | 100.0 | 35.1 | 52.2 | 10.4 | 47.8 | 62.1 | 100.0 | 35.1 |
| 12 | Im. Zasyadko | 60.8 | 11.2 | 39.2 | 66.0 | 100.0 | 32.7 | 60.0 | 12.7 | 40.0 | 62.8 | 100.0 | 32.7 |
| 13 | Pochenkova | 59.9 | 6.8 | 40.1 | 69.1 | 100.0 | 31.8 | 59.3 | 8.8 | 40.7 | 65.3 | 100.0 | 31.8 |
| 14 | Chaikino | 59.4 | 5.3 | 40.6 | 57.7 | 100.0 | 26.6 | 58.7 | 6.9 | 41.3 | 54.5 | 100.0 | 26.6 |
| 15 | Im. Бажанова | 56.1 | 6.8 | 43.9 | 58.7 | 100.0 | 29.6 | 55.8 | 8.6 | 44.2 | 56.1 | 100.0 | 29.6 |
| 16 | Im. Voroshilova | 56.1 | 10.7 | 43.9 | 64.4 | 100.0 | 34.3 | 55.9 | 12.5 | 44.1 | 61.9 | 100.0 | 34.3 |
| 17 | Im. Artyoma | 63.6 | 10.4 | 36.4 | 50.6 | 100.0 | 25.0 | 62.5 | 11.3 | 37.5 | 47.8 | 100.0 | 25.0 |
| 18 | Toretskaya | 54.5 | 8.9 | 45.5 | 70.0 | 100.0 | 36.0 | 54.3 | 13.5 | 45.7 | 62.7 | 100.0 | 36.0 |
| 19 | Novodzerzhinskaya | 64.2 | 9.1 | 35.8 | 56.6 | 100.0 | 26.1 | 63.1 | 10.2 | 36.9 | 53.3 | 100.0 | 26.1 |
| 20 | Novaya | 59.7 | 12.3 | 40.3 | 63.9 | 100.0 | 33.1 | 59.1 | 14.1 | 40.9 | 60.6 | 100.0 | 33.1 |
| 21 | Komsomolskaya | 55.8 | 10.1 | 44.2 | 62.1 | 100.0 | 33.1 | 55.4 | 11.9 | 44.6 | 59.4 | 100.0 | 33.1 |
| 22 | Im. Lenina | 61.9 | 10.9 | 38.1 | 64.7 | 100.0 | 31.4 | 61.0 | 12.7 | 39.0 | 60.6 | 100.0 | 31.4 |
| 23 | Izotova | 56.3 | 11.7 | 43.7 | 61.1 | 100.0 | 33.3 | 56.0 | 13.3 | 44.0 | 58.8 | 100.0 | 33.3 |
| 24 | Kochegarka | 59.8 | 11.1 | 40.2 | 61.6 | 100.0 | 31.4 | 59.2 | 12.9 | 40.8 | 58.7 | 100.0 | 31.4 |
| 25 | Im. Gagarina | 67.9 | 8.9 | 32.1 | 60.6 | 100.0 | 25.5 | 66.4 | 10.0 | 33.6 | 56.1 | 100.0 | 25.5 |
| 26 | Molodogvardeiskaya | 58.4 | 16.6 | 41.6 | 67.3 | 100.0 | 37.7 | 57.8 | 18.0 | 42.2 | 64.7 | 100.0 | 37.7 |
| 27 | Orekhovskaya | 51.5 | 15.5 | 48.5 | 70.8 | 100.0 | 42.3 | 51.5 | 17.6 | 48.5 | 68.5 | 100.0 | 42.3 |
| 28 | Barakova | 66.9 | 10.3 | 33.1 | 65.0 | 100.0 | 28.4 | 65.6 | 11.6 | 34.4 | 60.4 | 100.0 | 28.4 |
| 29 | Petrovskaya | 63.6 | 17.3 | 36.4 | 70.9 | 100.0 | 36.8 | 62.5 | 18.5 | 37.5 | 67.3 | 100.0 | 36.8 |

The table shows that the error in planning of qualitative and quantitative parameters (i.e. overestimation of commercial yield) constitutes from 1 to 9%, while an average value for the 29 mentioned mines is 2.8%.

Thus, it is necessary to take sludge formation into account to get more realistic figures in calculation of quality and quantity indicators of run-of-mine coal preparation.

Coal preparation practice estimates additional volumes of sludge depending on the grade of coal, process and formulations used [2, 6]. However, the most correct method would be determination of the sludge formation coefficient for a process scheme from analysis of its inputs and outputs [7]. At that, it is proposed to perform calculation of the coefficient of sludge formation from the changes in the content of 1+ mm grade particles in these products, because it is defined more precisely both in quantitative and qualitative terms (in comparison with -1 mm grade).

*Table 2.*

**Results of calculation of error in planning**

| No. | Coal Mines | Coefficient of reduction of concentrate yield to change in its ash content for 1% | Concentrate yield, % | | Error in planning (output overestimation) % |
|---|---|---|---|---|---|
| | | | Not considering sludge formation | Considering sludge formation and reduced to the concentrate ash content without sludge formation | |
| 1 | Rodinskaya | 1.11 | 55.0 | 52.3 | 2.7 |
| 2 | Krasnolimanskaya | 1.13 | 63.2 | 60.3 | 2.9 |
| 3 | Im. Dzerzhinskogo | 1.72 | 56.0 | 47.0 | 9.0 |
| 4 | Severnaya | 1.67 | 62.9 | 60.5 | 2.4 |
| 5 | Torezskaya | 1.13 | 56.4 | 53.9 | 2.5 |
| 6 | No. 4 VM | 0.71 | 32.4 | 31.4 | 1.0 |
| 7 | No. 6 VM | 1.01 | 42.9 | 41.3 | 1.6 |
| 8 | No. 7 VM | 0.92 | 45.4 | 44.4 | 1.0 |
| 9 | No. 8 VM | 0.91 | 40.9 | 39.4 | 1.5 |
| 10 | Kirovskaya | 1.14 | 61.4 | 58.8 | 2.6 |
| 11 | Im. Skochinskogo | 1.03 | 52.1 | 49.7 | 2.4 |
| 12 | Im. Zasyadko | 1.23 | 60.8 | 58.2 | 2.6 |
| 13 | Pochenkova | 1.07 | 59.9 | 57.2 | 2.7 |
| 14 | Chaikino | 1.26 | 59.4 | 56.7 | 2.7 |
| 15 | Im. Бажанова | 1.20 | 56.1 | 53.6 | 2.5 |
| 16 | Im. Voroshilova | 1.15 | 56.1 | 53.8 | 2.3 |
| 17 | Im. Artyoma | 1.76 | 63.6 | 60.9 | 2.7 |
| 18 | Toretskaya | 1.13 | 54.5 | 49.1 | 5.4 |
| 19 | Novodzerzhinskaya | 1.50 | 64.2 | 61.5 | 2.7 |
| 20 | Novaya | 1.30 | 59.7 | 56.8 | 2.9 |
| 21 | Komsomolskaya | 1.19 | 55.8 | 53.3 | 2.5 |
| 22 | Im. Lenina | 1.30 | 61.9 | 58.7 | 3.2 |
| 23 | Izotova | 1.26 | 56.3 | 54.0 | 2.3 |
| 24 | Kochegarka | 1.33 | 59.8 | 56.8 | 3.0 |
| 25 | Im. Gagarina | 1.47 | 67.9 | 64.8 | 3.1 |
| 26 | Molodogvardeiskaya | 1.27 | 58.4 | 56.0 | 2.4 |
| 27 | Orekhovskaya | 1.03 | 51.5 | 49.3 | 2.2 |
| 28 | Barakova | 1.37 | 66.9 | 63.8 | 3.1 |
| 29 | Petrovskaya | 1.32 | 63.6 | 60.9 | 2.7 |
| | AVERAGE | | | | 2.8 |

The method for definition of coefficient of sludge formation is based on the equation of material balance of the preparation products, according to which the quantity of run-of-mine coal incoming for preparation is equal to the sum of the products obtained as a result of its conversion. Taking the run-of-mine coal dry mass as 100% in the material balance equation we may represent it as a sum of outputs, that of 1+ mm fineness $\left(\gamma_{p.y.+1}\right)$ and sludge (defined as a fineness class of under 1 mm $\gamma_{p.y.-1}$):

$$\gamma_{p.y.+1} + \gamma_{p.y.-1} = 100\% \qquad (1)$$

Preparation of the run-of-mine coal at the plant involves additional sludge formation, thus the sum of preparation process outputs is:

$$\gamma_{p.y.+1} + \gamma_{p.y.-1} = \gamma_{n.o.+1} + \gamma_{p.y.-1} + \gamma_{n.o.-1} \qquad (2)$$

where $\gamma_{n.o.+1}$, $\gamma_{n.o.-1}$ are the percentage contents of fineness grade "1+ mm" and fineness grade "under 1 mm" in the preparation product.

From equation (2) we may determine decrease in output of 1+ mm grade in the preparation products in comparison to content of such grade in the run-of-mine coal, this decrease is caused by additional sludge formation:

$$\gamma_{n.o.-1} = \gamma_{p.y.+1} - \gamma_{n.o.+1}. \qquad (3)$$

Equation (3) shows that sludge (1- mm grade) that is already present in the run-of-mine coal does not influence the formation of additional sludge.

Dividing both sides of the equation (3) by the output of 1+ mm grade in the run-of-mine coal $\gamma_{p.y.+1}$

$$\frac{\gamma_{n.o.-1}}{\gamma_{p.y.+1}} = \frac{\gamma_{p.y.+1} - \gamma_{n.o.+1}}{\gamma_{p.y.+1}} \qquad (4)$$

and designating the ratio on the left side as $K_{ш}$, we obtain the value of relative decrease for 1+ mm grade due to additional formation of sludge which is the sludge formation coefficient for the plant's process flow:

$$K_{ш} = \frac{\gamma_{p.y.+1} - \gamma_{n.o.+1}}{\gamma_{p.y.+1}} \cdot 100\%. \qquad (5)$$

Base values calculate the sludge formation coefficient are taken from sampling and analysis of incoming run-of-mine coals and all the products of the preparation process as well as from material balance of the prerapation process products.

Thus, a calculation for concentrate output from preparation of run-of-mine coal coming from Mezhirichanskaya mine at Chervonogradskaya coal preparation plant with and without taking the sludge formation into account showed that the output indicator reduced to ash content of 25% was overestimated by 1.42% in planning [8].

Table 3 shows the sludge formation coefficients for process flows of a number of existing coal preparation plants. From the Table 3 it is evident that more process operations and higher ash content of the run-of-mine coal lead to a significant increase of the plant's sludge formation coefficient. If the first cause of increase may be explained by a higher number of differentials, pumps and extended process time, the second one is due to increased mass of rock which is easier ground and saturate.

*Table 3.*

**Sludge formation coefficient for process flow in selected coal preparation factories**

| Names of Coal Preparation Plant | Grade of Coal | Ash content for run-of-mine coal, % | Preparation processes* in four machine classes** | $K_{Sl}$, % | Year of $K_{Sl}$ determination |
|---|---|---|---|---|---|
| Kurakhovskaya | G | 23.2 | HM (+13 мм) | 1.6 | 1974 .9 |
| Kurakhovskaya | G | 45.8 | HM+J+O+O | 13.3 | 2011 |
| Dobropolskaya | G | 32.7 | J+J+O+O | 4.1 | 1974 [9] |
| Dobropolskaya | G | 45.4 | J+J+O+F | 12.0 | 2010 [10] |

| | | | | | |
|---|---|---|---|---|---|
| Krasnolimanskaya | G | 32.0 | J+J+O+O | 8.1 | 1974 [9] |
| Komendantskaya | A | 22.4 | HM+J+TC+F | 3.9 | 1974 [9] |
| Sverdlovskaya | A | 39.9 | HM+J+O+O | 2.4 | 2010 |
| Chumakovskaya | OC | 11.5 | HM+J+O+F | 12.8 | 1974 [9] |
| Chumakovskaya | G | 42.7 | HM+J+SS+F | 8.2 | 2012 |
| Kalininskaya | G | 29.5 | HM+J+O+F | 8.5 | 1974 [9] |
| Krasnoarmeyskaya | G | 39.2 | J+J+O+O | 5.8 | 1974 [9] |
| Novopavlovskaya | A | 26.0 | J (6+ mm) | 3.4 | 1974 [9] |
| Gorlovskaya | G | 18.4 | J+J+TC+F | 20.9 | 1974 [9] |
| Krivorozhskaya | K | 11.1 | J+J+O+F | 17.2 | 1974 [9] |
| Bryankovskaya | K | 19.8 | J+J+O+F | 19.0 | 1974 [9] |
| Zaporizhsky Coke Plant | coal charge | 18.3 | J+J+O+F | 9.9 | 1974 [9] |
| Sudzhendskaya | K | | | 12.8 | 1974 [9] |
| Adzherskaya | OC | | | 4.6 | 1974 [9] |
| Beryozovskaya | K | | | 18.3 | 1974 [9] |
| Belovskaya | G | | | 6.9 | 1974 [9] |
| Chertinskaya | G | | | 11.1 | 1974 [9] |
| Taybanskaya | K | | | 6.9 | 1974 [9] |
| Ziminka | K | | | 7.5 | 1974 [9] |
| Koksovaya | K | | | 6.7 | 1974 [9] |
| Abashevskaya | G | | | 12.5 | 1974 [9] |
| Tomusinskaya | K | | | 10.0 | 1974 [9] |
| Mospinskaya | G | 40.7 | HM+J+O+O | 9.8 | 2013 |
| Pavlovogradskaya | G | 45.7 | HM+J+O+O | 8.8 | 2012 |
| Oktyabrskaya | G | 44.5 | HM+J+O+F | 11.3 | 2011 [12] |
| Rovenkovskaya | A | 37.8 | J+J+O+O | 1.8 | 2009 [11] |
| Chervonogradskaya | G | 53.9 | HM+J+O+O | 8.7 | 2009 [8] |

* Large fineness grade 13+ mm; small fineness grade 1-13 mm; non-floatation sludge 0.5-1 mm; floatation sludge 0-0.5 mm.
** HM – heavy medium separators; J – jig; TC – table concentrator; SS – spiral separator; H – hydrosizers; F – floatation; O – no process.

**Conclusions**

1 It is necessary to take the sludge formation from run-of-mine coal into account while planning the preparation indicators for a given process flow used in the plant.

2 Sludge formation of run-of-mine coal should be accounted for with a process flow-dependent sludge formation coefficient defined by changes in content of 1+ mm graded in the run-of-mine coal and products of its preparation.

3 Sludge formation coefficient for a process flow of coal preparation plant is determined from the results of analysis of inputs and outputs of the process.

4 Value of the sludge formation coefficient for the process flow shall be included into the plant's process procedure.

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**Criteria of Engineering Efficiency of Thickening Flowcharts**

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**Abstract**

Thickening and water circulation are important processes in coal beneficiation cycles. Rather frequently reasonable arrangement of these processes influences not only on plant production rate and quality of final product but on its overall operating capability as well. As demonstrated in practice, only some beneficiation plants erected or retrofitted recently in Russia were commissioned in due time and reached rated performances. Statistical analysis of engineering audits of more than a hundred of existing and green beneficiation plants in the last fifteen years demonstrated that in two thirds of all situations the problems are related with incorrect arrangements of thickening and water circulation flowcharts of the plants.

The main reasons of failures in designing and subsequent inoperability of plants are as follows: disagreement between actual and rated properties of feedstock, mistakes in designing, wrong selected equipment for thickening and water circulation flowcharts.

An important issue in designing and estimation of thickening operability is meeting of criteria of process applicability. Sometimes initially wrong process variables of water and slurry flowchart makes it impossible to operate any apparatus.

The article discusses main criteria of operability estimation of thickening.

**Key words:** thickening, water circulation, slurry, radial thickener, rapid thickener, paste thickener, thin-layer thickener, flocculating agent.

Thickening and clarification are widely applied in coal beneficiation. They are related both with the issue of internal and external water circulation and with engineering efficiency of the production. Usually these processes are based on either radial or thin-layer thickeners. In actual production certain criteria are required in order to estimate engineering efficiency of thickener, enabling selection of apparatus type and to determine in the first approximation its operability in certain situations.

While classifying and defining liquid-solid systems, none of the processes seems to be so straightforward, at the same time creating numerous questions and misconceptions as thickening and clarification. Description of the processes is generally limited to the well-known Navier-Stokes equation for deposition of a single globular particle, or is related to private cases of investigations into separate factors influencing on deposition velocity. Numerous guide and reference books on development of thickeners were published more than 20-30 years ago, only slightly mentioning possibility of application of reagents for intensification of the process, or this issue is not considered at all [1-4]. Meanwhile, application of coagulants and flocculants in thickening requires for appropriate apparatus design, stipulated by properties of these reagents and appropriate approaches to arrangement and operation of technology [5].

As any of the processes of chemical industry and beneficiation, thickening and apparatuses thereof have quite definite range of applicability. Quite often the provision of optimum operation of thickening and water circulation stages is related not with successful or unsuccessful thickener design but rather with optimization of flowcharts, without significant modification of existing apparatuses or purchasing of new ones [6].

Multi-year experience of designing, commissioning and the acquired data on actual performances of numerous types of thickeners for various types of material in beneficiation and chemical technologies formed background of procedure for preliminary estimation of efficiency and applicability of thickening and thickeners. The procedure stipulates for consideration of two aspects of approved engineering



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flowcharts, determined by technological conditions and apparatus designs. The main estimation criteria are summarized below in the table.

Table. Criteria of applicability of thickening

| Criterion | Working range | Critical range | Non-applicable |
|---|---|---|---|
| Solid content in feedstock, vol. % | Less than 4 | 4-10 | More than 10 |
| Hydraulic particle size, m/h | Less than 5 (thin-layer thickening is preferable) | 5-10 | More than 10 (thickening in radial thickeners is preferable) |
| Planned solid content in slurry, wt. % | 20-60 | 40-80 | Более 80 |
| Specific loading referred to overflow surface area in terms of solids, $t/m^2h$ | Less than rated value, [8] | Close to rated value, [8] | More than rated value, [8] |
| Working volume of apparatus referred to feedstock volume, $Q_п$, $m^3/h$ | More than 1.0 $Q_п$ | 0.35-1.0 $Q_п$ | Less than 0.35$Q_п$ |
| Coagulants and flocculants in the process (yes/no) | yes | no | no |

The mentioned above ranges of applicability are not stringent. This or that criterion in critical range or inapplicability implies only special attention which should be given to scheduling and designing of thickening for a given technological task and desirability of performance of laboratory or pilot experiments at designing stage. There are certain distinctions in the procedures of development of radial and thin-layer thickeners. Lower values of parameter ranges in the table are referred to thin-layer thickening and higher values to radial thickeners.

The data of solid content in initial slurry summarized in the table are generalized for wide variety of tasks and materials. With increase in volumetric content of solids in feedstock the rate of clarification decreases up to complete termination of achievement of the slurry aggregate steady state. Restrictions on solid content in feedstock constitute one of important factors, limiting efficient application of thickeners in engineering processes of existing plants.

Hydraulic particle size is determined according to regular procedures given in reference books on beneficiation and chemical technologies [7]. The data on hydraulic particle sizes for the applicability ranges in the table are given not with regard to engineering efficiency for apparatuses of certain type but in terms of minimization of capital expenditures for equipment procurement. Practical calculations performed by us in more than decade for numerous thickening embodiments revealed that at hydraulic particle sizes more than 10 m/h the application of radial thickness is more cost efficient. The cost of apparatuses in this case would be at least twofold as lower as upon application of thin-layer thickeners at equal footprint areas. For the velocities of free particle deposition lower than 5 m/h, thin-layer thickening is generally more preferable.

The solid content in thickened slurry depends on sedimentation properties, applied reagent, slurry holding time in thickener. However, this value for actual slurry and upon application of certain flocculant (coagulant) is always maximum, this also should be considered in designing of slurry flowcharts and determination of engineering sizes of thickeners.

Specific loading, referred to overflow surface area upon feeding of solids, $\gamma$ - $t/m^2h$, is obligatory for attention of producers. Unfortunately, some statements, related mainly with marketing tricks of equipment manufacturers, and contradicting to the theory are taken by designers for granted. Either an apparatus is called "rapid' or not, physical backgrounds of the process do not vary. Permitted specific loadings for thickening are stipulated by the physical essence of the process and, to significantly lower extent, by apparatus design. This is related with conditions and possibilities of densification and discharge of mud from slurry via apparatus bottom. The permissible loadings in the first approximation should be determined by reference information given elsewhere [8].

Restrictions in terms of minimum possible working capacity of thickener in portions of hourly volume of slurry fed to the apparatus are related with necessity to dissipate the energy of inlet flow and to provide the condition of laminar flows in the clarification area. In the cases when increased requirements are set to slurry moisture with regard to decrease its content the apparatus capacity should be increased because of the possibility to held slurry in the apparatus for its thickening up to the required state. The holding time is determined from the curves of kinetics of slurry sedimentation by the slurry thickening rate.

The main property enabling estimation of the process efficiency is the hydraulic size of solid phase or clarification rate with selected reagent and established parameters of feedstock slurry. The aspects of laboratory experiments of slurry sedimentation and hydraulic size of solid phase are described in detail elsewhere [9]. While transferring the laboratory experimental data to industrial environment it is required to consider for difficulties and peculiarities of scaling-up upon simulation of hydromechanical processes [10]. The existing theory of hydrodynamic similarity provides the procedure of selection of similarity criteria and values of scaling-up upon development and investigation into process variables and apparatuses. However, in most cases implementation of the conditions of the theory application is either impossible in practice or hindered, and if the influence of chemical processes on the system prevails, then it cannot be applied at all. Under such conditions practical simulation usually applies partial similarity of flows, when the condition of similarity of main forces is valid, these forces are the most significant for the considered hydrodynamic phenomenon.

As confirmed by practical simulation of the processes with complex hydrodynamic environment, a method with partial simulation of processes can be applied with the use of system models, each being described with sufficient accuracy by means of only one similarity law. In this case regularities of each process element are discovered with obtaining of concepts of interrelation between separate parts. Splitting of laboratory investigations into subprocesses is stipulated by impossibility of simultaneous simulation of two processes in one assembly: for instance, thickening and agitation, which obey different laws.

Taking into consideration high cost portion of equipment for dewatering (thickening, filtration) in general cost of equipment of beneficiation plants, prior to adoption of decision to purchase equipment and to start designing activity it would be reasonable to carry out independent expert evaluation of technological schedules of the stages of thickening and water circulation. Expert evaluation at the designing stage would make it possible to save significant assets and to avoid subsequent expenditures for retrofitting of technological flowchart. Selection of thickeners for certain technological flowchart should be based on process approach which implies first of all revealing of determining factors in technological process, influencing on operation of water circulation flowchart. Sometimes initial wrong input process variables, determined by slurry and water circulation flowchart, prevent operation of any apparatus.

In general case expert evaluation should provide analysis of adopted slurry and water circulation flowchart, development of measures aimed at coordinated operation of process chain, - preparation of flocculant (coagulant) – preparation (conditioning) of slurry - mixing of reagent with initial slurry – flocculation – thickening (clarification) [11]. These are low-cost measures, both in terms of time and financial expenses. However, as confirmed by practical experience, already at designing stage the expert valuation sufficiently often provides possibility to make adjustments, resulting in saving of material resources and avoiding unjustified expenses for subsequent retrofitting of technological flowchart and adaptation of thickener.

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# Sandwich Belt High Angle Conveyors
# Coal Mine to Prep Plant and Beyond – 2016

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**Abstract**

Developed between 1979 and 1981 Dos Santos Sandwich Belt high angle conveyors began commercial operation in 1983. Since then some 150 systems have gone into successful operation throughout the world with most still in productive operation many years later. They have been successful with many, varied bulk materials from light, friable wood chips and grains to heavy, coarse copper and iron ores. Throughput rates have varied widely, from 272 kgs/hr to 6000 t/h. These systems have been especially successful in coal handling applications from the mine to preparation to transfer and export. Sandwich Belt high angle conveyors also found use in the disposal of refuse from coal preparation plants.

This writing presents the technical aspects of the system then describes the important sandwich belt high angle conveyor installations including vertical continuous haulage from underground mining and direct haulage from open pit mines, yard handling and prep plant feed, coal prep refuse disposal, and coal transfer, to market, domestic and export..

Also, this writing places special emphasis on noting the systems suitability for handling large size materials which are typical in haulage from underground and open pit mines, as part of IPCC (In-Pit Crushing and Conveying) systems.

**Key words**

Sandwich belt, radial hugging pressure, high angle conveyor

## Introduction

Development of the sandwich belt high angle conveyor concept has come a long way since its first introduction in the early 1950s. Over the approximate thirty year period until 1979, significant advances were few and only came in spurts. Such advances did not build on previous developments rather they were independent developments which soon reached their technical limitations.

The latest significant development of this technology, beginning in 1979, is the first to take a broad view of the industries to benefit from high angle conveying and of all previous developments. As a result these latest developments know few technical limitations, address a broad range of applications and offer a forum for further logical development or evolution.

## Sandwich Belt Principle

Dos Santos Sandwich Belt high angle conveyors represent logical evolution and optimization of the sandwich belt concept. The sandwich belt approach employs two ordinary rubber belts which sandwich the conveyed material. Additional distributed force on the belt provides hugging pressure to the conveyed material in order to develop sufficient friction at the material to belt and material to material interface to prevent sliding back at the design conveying angle.

 
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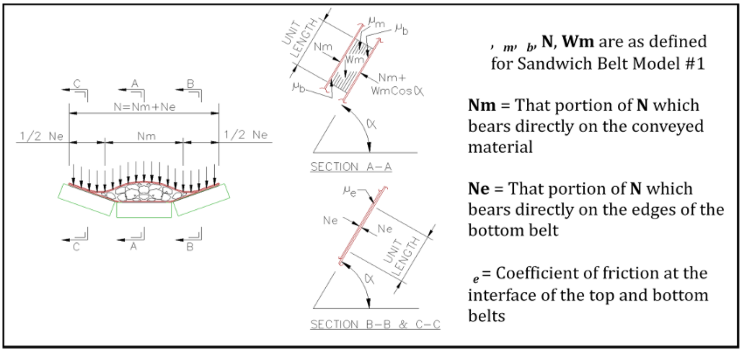

Fig. 1: Sandwich Belt Model

Figure 1 is a realistic model of the belt sandwich. An ample belt edge distance assures a sealed material package during operation even if belt misalignment occurs. This model also illustrates the interaction of forces within the sandwich. The applied or induced hugging load is distributed across and along the carrying belt sandwich. Of that hugging pressure only the middle pressure hugs the material load while the outer pressure merely bears against the material free edges of the belt. Both belt surfaces apply their frictional traction on the material. From this model one can calculate the required material hugging pressure that will ensure the material does not slide back due to the tangential gravity loads. This is expressed by Equation 1:

Equn. 1. $$Nm \geq \frac{Wm}{2}\left(\frac{\sin\alpha}{\mu} - \cos\alpha\right)$$

Where: $\mu = \mu_m$ or $\mu = \mu_b$, whichever is the smaller

**Dos Santos Sandwich Belt High Angle Conveyors**

When investigated anew in the late 1970's it was clear that the sandwich belt concept offered the greatest potential for a cost effective, operationally appropriate high angle conveying system to address the broad needs of the mining and materials handling industries.

Following the extensive study of past sandwich belt conveyors, the governing theory and constraints, and development of the governing design criteria, a broader scope effort was undertaken in 1982 to develop the first sandwich belt high angle conveyor to meet these needs. The resulting Dos Santos Sandwich Belt high angle conveyors are truly evolutionary in judiciously selecting and advancing desirable features while avoiding the pitfalls of the past. They conform entirely to the governing theory, to the constraint equations and to the development criteria.

Dos Santos Sandwich Belts fulfill all established operational requirements. The profiles can conform to a wide variety of applications.

**Advantages of Dos Santos Sandwich Belts**

Dos Santos Sandwich Belt high angle conveyors offer many advantages over other systems including:

Simplicity of Approach

The use of all conventional conveyor hardware, for very high availability and low maintenance costs.

Virtually Unlimited in Capacity

The use of all conventional conveyor components permits high conveying speeds. Available belts and hardware up to 3000 mm wide make capacities greater than 10000 t/h possible.

High Lifts and High Conveying Angles

High lifts to 300 m are possible with standard fabric. Much higher single run lifts are possible with steel cord or aramid fiber belts. High angles up to 90 degrees are possible.

Flexibility in Planning and in Operation

Dos Santos Sandwich Belts lend themselves to multi-flight conveying systems with self-contained units or to single run systems using externally anchored high angle conveyors. The system may be shortened or lengthened or the angle may be altered for the requirements of a new location.

Belts are Easily Cleaned and Quickly Repaired

Smooth surfaced rubber belts allow continuous cleaning by belt scrapers or plows. Smooth surfaced belts present no obstruction to quick repair by hot or cold vulcanizing.

Spillage Free Operation

During operation the material is contained within the belt sandwich from loading to discharge. Well centered loading and ample belt edge distance result in no spillage along the conveyor length.

**Dos Santos Installations - General**

Dos Santos Sandwich Belt high angle conveyors are well established in the industry with the first commercial unit beginning operation in 1984. Since then more than 100 units have gone into operation throughout the world. Table 1 lists only the latest DSI sandwich belt installations since 2010.

Materials handled vary from various grades of coal to coal refuse to coarse copper ore, excavated earth, dewatered sludge, wood chips to blast furnace slag, gypsum, sulfur to various grains.

Conveying rates vary from a low of 272 kg/h to a high of 6000 t/h

Conveying angles vary from 35 degrees to 90 degrees.

Elevating heights are as low as 3.66 m and as high as 175 m. These same units are respectively the shortest and the longest, at 8.6 m and 455 m respectively.

TABLE 1. Latest DSI Sandwich Belt High Angle Conveyor Installations since 2010

| DS # | Location | Material/ Rate (t/h) | Ang (º) | Elev. (m) | Lgth (m) | Width (mm) | Speed (m/s) | Top/Bot (kW) | Year |
|---|---|---|---|---|---|---|---|---|---|
| 098 | Refinery/ Muzkiz, Spain | Pet Coke/ 475 | 90 | 21.2 | 32.3 | 1400 | 3.5 | 45/ 45 | 2012 |
| 099 | Refinery/ Cartagena, Spain | Sulfur/ 40 | 90 | 10.5 | 20.6 | 600 | 2.0 | 7.5/ 7.5 | 2012 |
| 100 | Copper Mine/ Balmecera, Chile | Filter Cake/ 77 | 64 | 7.2 | 14.8 | 762 | 1.3 | 7.5/ 7.5 | 2011 |
| 101 | Pulp & Paper Mill/ Maine, USA | Hog Fuel/ 54.4 | 90 | 7.7 | 16.6 | 1067 | 1.52 | 7.5/ 7.5 | 2012 |
| 102 | Cu-Au Mine/ BC, CN | Pebble Ore/ 340 | 67 | 24.2 | 49.8 | 914 | 2.1 | 29.8/ 29.8 | 2015 |
| 103 | Pastil Plant/ | Sulfur Pastils/ | 50.5 | 19.3 | 55.8 | 600 | 0.4 | 3.7/ | 2015 |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | South England | 20 | | | | | | 3.7 | |
| 104 | Cement Plant/ Paraiba, Brazil | Raw Feed/ 720 | 60 | 20.3 | 61.5 | 1220 | 3.1 | 56/ 56 | 2015 |
| 105 | Export Terminal/ South LA, USA | Coal/ 3629 | 52 | 18.5 | 45.9 | 2438 | 4.32 | 224/ 224 | 2016 |

**Mine to Prep Plant and Beyond**

Having noted the versatility of Dos Santos Sandwich Belt Systems we now consider applications from coal mining to prep plant and beyond, to clean coal load-out then to market and to refuse disposal. Dos Santos Sandwich Belt installations have fulfilled the elevating functions at all of the stages en route.

Haulage From Underground

The wide use of longwall systems in the 1980s required upgrading all of the underground conveyors to larger belt widths that could keep up with the longwall production. In deep coal mines this resulted in choking the flow at the main haulage shafts where skip hoist systems could not meet the increased production requirements. This created opportunities for vertical high angle conveyors in the 1990s. Studies of that time developed single and multi-flight systems as alternates to the traditional skip hoist systems. These proved to easily handle the large throughput rates continuously through a mere conveyor to conveyor transfer chute precluding the large terminal storage and feeding systems that are required for the skip hoist batch haulage systems. The economics are overwhelmingly in favor of the continuous haulage systems with Sandwich Belt high angle conveyors along the vertical shafts from underground.

Figure 2 shows two variations of continuous vertical haulage from underground, Scheme-A and Scheme-B. The original basis for these schemes is an underground nickel/copper mine with a net vertical lift to surface of 1381 m.

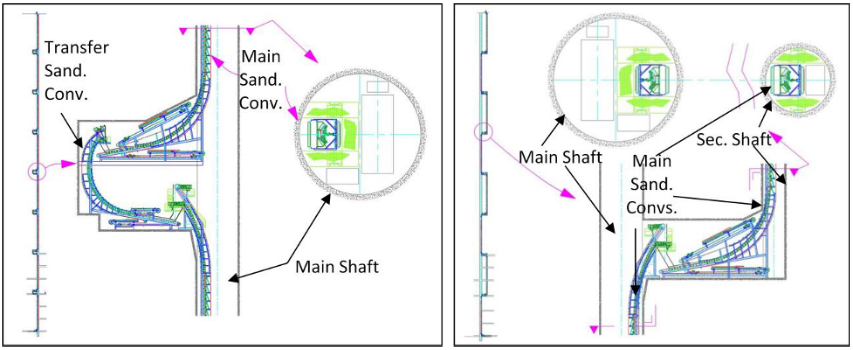


Fig. 2: Scheme-A and Scheme-B for Continuous Vertical Haulage from Underground

Scheme-A consists of eight (8) main sandwich conveyors along the main vertical shaft and seven (7) small connecting sandwich conveyors at excavated pockets. A main shaft of 7 meters finished diameter is able to accommodate the haulage system as well as all mine equipment access. The continuous vertical haulage system occupies half of the main shaft while the travel path of the 1.8 m x 4.9 m mine equipment cage occupies the other half.

Scheme-B consists of only (8) eight main sandwich conveyors, four (4) along the main vertical shaft and four (4) along independent vertical shafts that are solely dedicated to each sandwich conveyor. The four (4) independent shafts are of 3.7 m finished diameter as this accommodates the haulage system as well as a 1.1 m x 1.7 m service cage. Scheme-B requires the additional local excavation between the main shaft and the ends of each independent shaft in order to accommodate the transfers between the alternating sandwich belt conveyors.

Such ambitious multi-flight systems as described above are yet to be realized but single flight vertical sandwich belt high angle conveyors, the basis for these systems, were commercially utilized at vertical shafts from underground coal mining, gypsum mining and tunneling projects. The most impressive these went into operation at a USA Midwestern coal mine. It featured 1524 mm wide belts running at 4.57 m/s and elevated 1361 t/h of coal vertically along the mine shaft 102 meters.

To Prep Plant Feed and Clean Coal Storage

Numerous Dos Santos Sandwich Belt high angle conveyors are in operation at storage facilities that feed the coal preparation plants and at clean coal storage facilities after the prep plants. Vertical systems were built in North America to storage silos of up to 76 meters height.

Refuse Disposal

A second product of coal preparation plants, mine refuse must be hauled to waste dumps. In the USA states of Virginia and West Virginia, often the disposal sites are at high elevations between mountain ridges. The refuse haulage system must scale steep mountain slopes. The Sandwich Belt high angle conveyor system of Figure 3 served such a function for 22 years in Virginia, USA. This system is of major proportions. Though modest in tonnage rate at 454 t/h the 914 mm wide belts elevate the coal refuse 175 meters of lift, 455 meters along the mountain slope to a 272 t truck loading bin at the top. Trucks then haul the refuse into the valley where it is spread and compacted. To dramatize the dimensions Figure 3 also shows a to-scale depiction of the Washington monument, a well known USA landmark. The system replaced two skip type aerial tramways which were supplemented by truck haulage. The original economic study compared the options of constructing another identical dual tramway to achieve the needed haulage rate against complete replacement by the Sandwich Belt high angle conveyor. The economics was strongly in favor of the high angle conveyor even in this case.

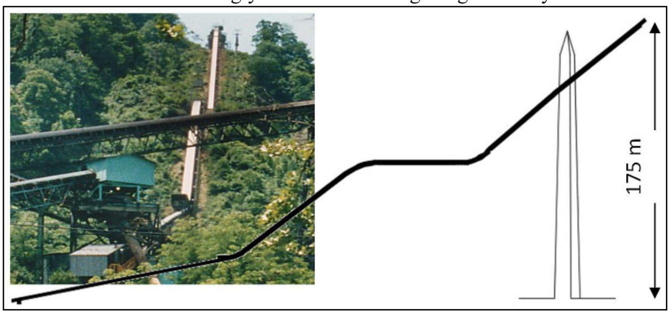


Fig. 3: Refuse Disposal Sandwich Belt High Angle Conveyor

Coal Transfer to Market

Numerous Dos Santos Sandwich Belt high angle conveyors are employed at the various transfer terminals en route to market and at the storage and transfer facilities of the coal market; the power plants, steel mills and chemical plants. These units have included a trailing high angle elevating conveyor to a ship loader in England in order to reduce the length of the costly dock structure and a complete high angle mobile ship loader at the Port of Adelaide in Australia.

Most interesting is the latest commercial system as it dramatizes the high angle advantage from a practical, environmental and cost standpoint.

The major expansion of a lower Mississippi River transfer terminal resulted in a short conveying path from the down river CBU (continuous barge unloader) to a new transfer complex over parallel yard conveyors that can alternately direct the coal flow to the storage yard or to ship loading. This most direct path subtends an incline angle that far exceeds the capability of any conventional open troughed belt conveyor. The reflexive solution was to use two conventional conveyor flights in a switch-back arrangement. A specified 9° maximum incline limitation and the location over water made this a costly proposition. The large (environmental) footprint was also a negative.

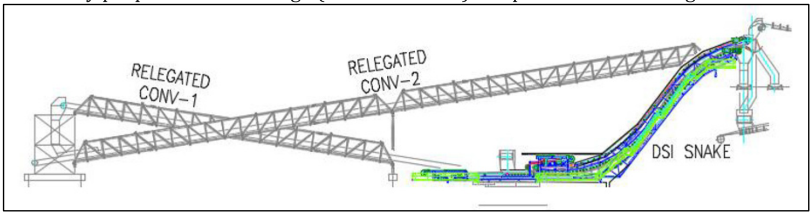


Fig. 4: Comparison: DSI Snake direct path vs conventional conveyor switch-back path

Because of the long relationship with Dos Santos International, terminal personnel knew there was a better solution, a DSI Snake Sandwich High Angle Conveyor. At 3629 t/h of coal this is the highest volumetric rate to date for a Dos Santos Sandwich Belt high angle conveyor. The Terminal management, in their due-diligence, sent key professionals to visit the operation of Dos Santos Sandwich units handling coal at high volumetric rates. These visits, and discussions with operating and maintenance personnel, confirmed that the DSI Snake was the best solution.

The DSI Snake was ordered in April of 2014. Presently in construction, the high angle conveyor will begin operation in the third quarter of 2016. The DSI Snake profile is depicted in Figure 4 along with the alternate conventional conveyor solution for contrast. A summary of the design features is found on Table 1, DS 105 (last entry). At 2438 mm of belt width this is the widest Dos Santos Sandwich Belt unit.

## Conclusion

Dos Santos Sandwich Belt high angle conveyors have found wide use in the marketplace. They have proven to be versatile with widely varying applications. Furthermore, they have proven their suitability for applications from underground mines to prep plants and beyond. The possibilities with Dos Santos Sandwich Belt high angle conveyors are far from being fully exploited. This continues to make for a bright and exciting future.

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# The Role of Fine Coal Classification on Fine Coal Cleaning Circuit Performance

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## ABSTRACT

Size classification and coal-ash separation are interdependent unit operations in fine coal cleaning circuits. An improvement in size classification performance could potentially increase the overall fine clean coal quality and quantity recovered from the fine coal cleaning circuit. It is quite surprising to note that few articles have been published that discuss the role of fine coal classification technologies on fine coal cleaning circuit performance.

This paper reviews the existing fine coal classification technologies, including classifying cyclone, sieve bend, and Stack Sizer™, and simulates their performance in fine coal cleaning circuits using USimPac mineral processing software. Detailed ash cleaning performance achieved by different fine coal cleaning circuits is discussed and compared. It is concluded that fine coal cleaning performance can be significantly improved by more efficient size classification technology.

**Key Words: Classification; Hydrocyclone; Sieve Bend; Stack Sizer; Fine Coal Circuit; Spiral Concentrator; Flotation**

## 1. INTRODUCTION

In modern coal processing plants, fine coal (typically in the size range of 1mm x 0) is normally cleaned by the combination of gravity concentrator and flotation. With the objective to improve the overall fine coal cleaning performance, there have been significant developments in gravity separators and flotation technologies for cleaning fine coal, and the circuits in which they are incorporated in the past two decades (Honaker and Forrest, 2003; Kohmuench et al., 2010; Zhang et al., 2011). Typically, a primary classifying cyclone is utilized to produce a thickened underflow stream with suitable PSD for the gravity separators while simultaneously to generate an overflow stream with PSD that optimizes the flotation recovery. However, the classifying cyclone has two fundamental limitations: 1. fine particles in the underflow due to hydraulic entrainment (fine light coal particles) and density effect (fine high density tailings particles); 2. coarse light fine particle in the overflow due to density effect. The misplaced fine particles in the cyclone underflow are often reported to the fine clean coal stream, ending up as contaminants. Therefore, a sieve bend is typically used to remove the misplaced fine particles and to dewater the fine clean coal product. However, the misplaced coarse fine particles in the cyclone overflow cannot be effectively recovered by flotation due to the high probability of detachment and becoming lost in the tailings. Therefore, high-efficiency size classification, capable of preparing optimal feed particles for a particular cleaning unit operation, is desirable.

State of the art classification technologies generally can be divided into three groups: centrifugal classification, hydraulic classification and screening classification. Hydrocyclones are the most widely



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used centrifugal classification devices. They have been the principal unit of operation for fine coal classification for several decades due to their high mass and volumetric throughput capacity, small floor space requirement, and relatively effective classification. Advancements in cyclone structural design, as well as circuit design, have greatly improved the size classification performance for this duty (Rong and Naper-Munn, 2003; Mohanty et al., 2002). However, their fundamental limitations, as discussed above, are yet to be overcome. Hydraulic classifiers are also used to achieve size classification for fine particles. The principle of operation of the hydraulic classifier is based on the concept of differential terminal settling velocity of solid particles of different size or mass. Due to its low classification efficiency, as well as the large floor space requirement, the hydraulic classifier has become increasingly unpopular in the coal processing industry. Screens are another method of achieving fine particle classification. Static sieve bend screens are widely used in coal processing plants to separate heavy medium from coarse coal in the heavy medium circuit and also to partially dewater the fine coal spiral product. Sieve bend screens tend to be low efficiency, low capacity, high maintenance, and high in operating cost (Firth, 2012). A high frequency linear motion screen, for example the Stack Sizer™ developed by Derrick Corporation (Figure 1), is capable of achieving high tonnage, high efficiency screening at fine size ranges. Zhang et al (2011) successfully evaluated the Stack Sizer screen technology at a plant site using both 75 micron and 100 micron urethane mesh panels. High efficiency size separation and ash reduction were achieved using both 100 micron and 75 micron Polyweb™ urethane panels on the Stack Sizers.

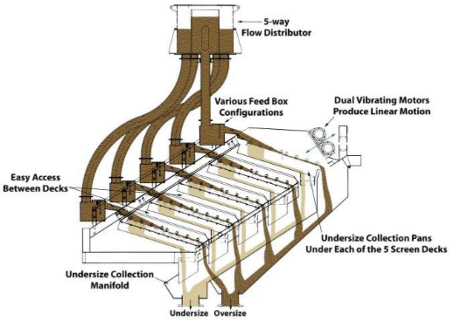

Figure 1. Schematic of full-scale Stack Sizer with five parallel screen decks

## 2. LITERATURE REVIEW

Fine coal classification is an integral part of fine coal cleaning circuit. It is rather surprising that few articles have been published to discuss the role of fine coal classification technologies on the performance of fine coal cleaning circuits. Firth et al (1997) published an article on this topic and simulated fine coal cleaning circuits consisting of different desliming technologies (single stage cyclone, two cyclones in series, hydrocyclone followed by sieve bend, three stage counter current cyclone circuit) followed by a spiral concentrator. He concluded that the size classification unit operation could be responsible for significant clean coal loss and clean coal high ash slime contamination. Further performance improvement of the fine coal cleaning circuit requires increased efficiency in the size classification technology. Zhang et al (2014) conducted a simulation study to evaluate the effect of different classification technologies on the performance of the fine coal cleaning circuits. Four fine coal cleaning circuits—Hydrocyclone-Spiral-Sieve Bend (CYC-SPL-SB) circuit, Hydrocyclone-Spiral-Stack Sizer (CYC-SPL-STK) circuit, Stack Sizer-Spiral-Sieve Bend (STK-SPL-SB) circuit, and Stack Sizer-Spiral-Stack Sizer (STK-SPL-STK) circuit—were simulated with a typical fine coal feed. The simulation results showed that CYC-SPL-STK circuit performed better than a conventional CYC-SPL-SB circuit. Plant production data in the US and China confirmed that improvement in post-classification can help to improve fine coal cleaning performance, as indicated by an additional 2 to 4 percent and 4 to 10 percent ash reduction of the fine clean coal. The STK-SPL-SB circuit performed better than the conventional CYC-SPL-SB circuit. This was due to improvement of the pre-classification performance that resulted in lower ash content of the clean coal product for the STK-SPL-SB circuit. The STK-SPL-STK is predicted to be the best-performing circuit in terms of producing cleaner product and also requiring less water to be clarified by the thickener. Weber et al (2014) published their simulation study on the interdependence of size classification and flotation performance. With a series of mathematical simulations, they evaluated the hypothetical response of a typical fine coal cleaning circuit to changes in classification cut size. The simulation results indicated that a raw coal classifying cyclone with D95 cut point around 0.23mm would provide the best opportunity for increasing clean coal yield.

This paper simulated four fine coal cleaning circuits that were fed with the same fine coal. To simplify the problem, the feed particle size distribution (PSD) and washability data were hypothetically specified. To evaluate the importance of size classification performance, the spiral and flotation performance were set to provide the same result. Although improved circuit performance could be achieved by fine tuning the spiral and flotation unit operations, it is, however, outside the scope of this discussion. The size classification equipment simulated in this study included hydrocyclone, Stack Sizer, and sieve bend. Different partition curves were used to differentiate their classification performance.

## 3. SIMULATION METHODOLOGY

Fine Coal Cleaning Circuits

Fine coal in the range of 1mm x 0 micron size fraction is typically cleaned using the flowsheet shown in Figure 2 (CYC-SB), where the fine coal feed is classified (pre-classification) using classifying cyclones to achieve 150 micron size classification. The 1mm x 150 micron size fraction will be cleaned using a gravity separator such as a spiral separator. The spiral product is sized (post-classification) at 150 microns to remove the misplaced high ash ultrafine particles. The classifying cyclone overflow and sieve bend undersize are combined and sent to flotation for further cleaning. The flotation product and sieve bend oversize product are then combined and dewatered using a screen bowl centrifuge or filter press to produce final clean fine coal. Figure 3 (CYC-STK) shows a modified flowsheet where a Stack Sizer is used to replace the sieve bend. Many US and China coal processing plants are now using this modified flowsheet for fine coal cleaning. Two additional flowsheets for fine coal cleaning are also described: Stack Sizer-Sieve Bend (STK-SB) circuit and Stack Sizer-Stack Sizer (STK-STK) circuit. The first of these circuits, shown in Figure 4, replaces the pre-classification hydrocyclone with a Stack Sizer-Sieve Bend (STK-SB) circuit. The other circuit, shown in Figure 5, replaces both pre-classification hydrocyclones and post-classification sieve bend with Stack Sizer-Stack Sizer (STK-STK) circuits.

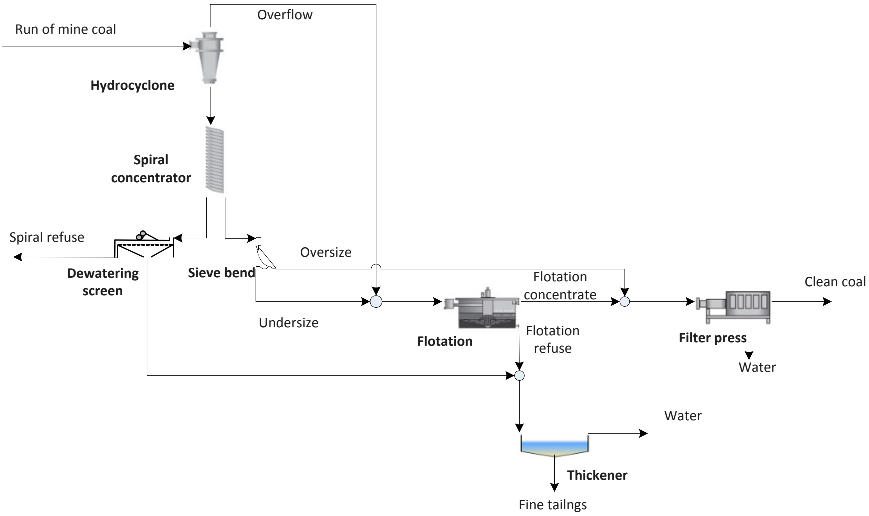


Figure 2. Typical fine coal cleaning circuit consisting of Hydrocyclone-Sieve Bend (CYC-SB) as classification equipment

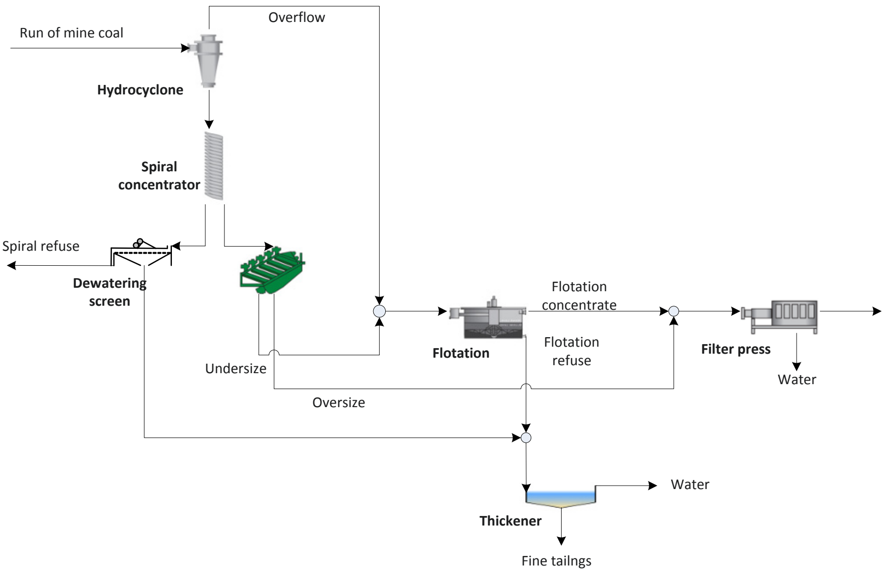


Figure 3. Modified fine coal cleaning circuit consisting of Hydrocyclone-Stack Sizer (CYC-STK) as classification equipment

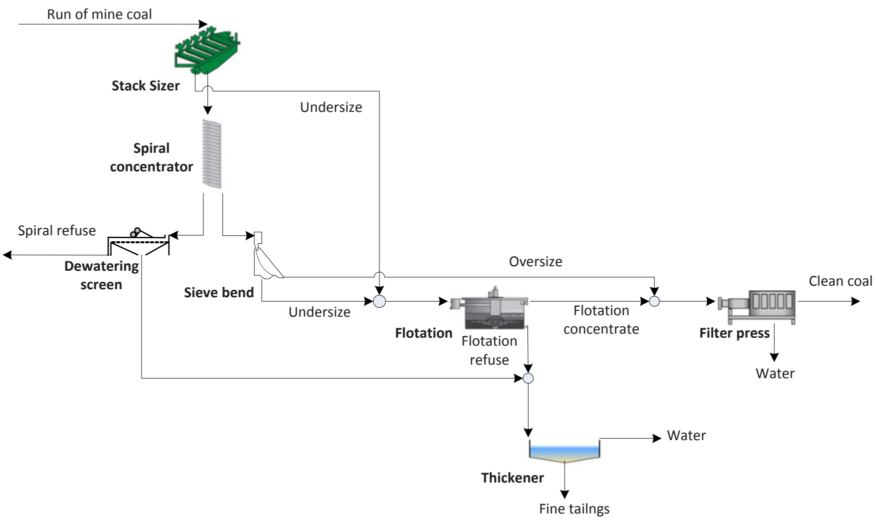


Figure 4. Modified fine coal cleaning circuit consisting of Stack Sizer-Sieve Bend (STK-SB) as classification equipment

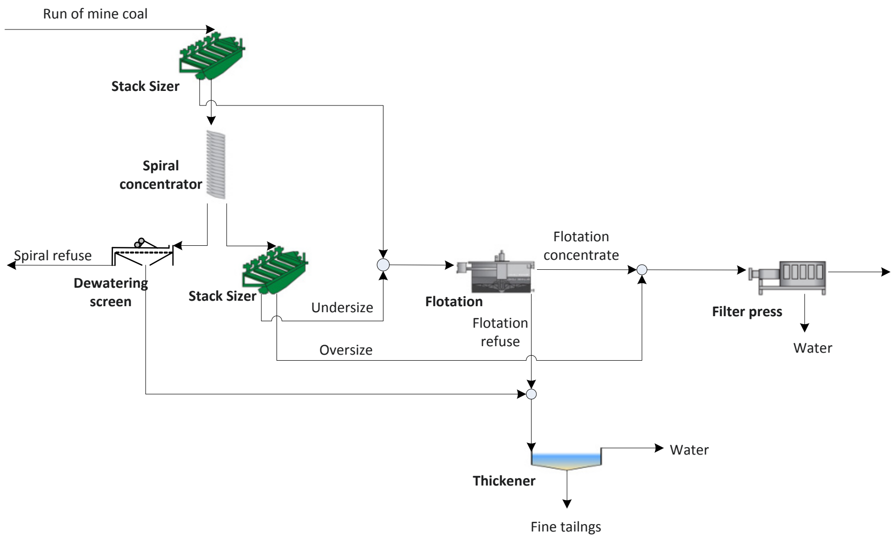


Figure 5. Modified fine coal cleaning circuit consisting of Stack Sizer-Stack Sizer (STK-STK) as classification equipment

Feed PSD and Washability

The four fine coal cleaning circuits described above were studied using USimPac simulation software to generate a hypothetical fine coal feed. The particle size distribution and washability analysis of this hypothetical fine coal feed are given in Table 1. The overall washability curves are shown in Figure 6.

Table 1. Fine coal feed particle size distribution and washability analysis

| Size Fraction (Micron) | Weight (%) | Ash (%) | F1.3 | S1.3-F1.4 | S1.4-F1.5 | S1.5-F1.6 | S1.6-F1.8 | S1.8-F2.0 | S2.0-F2.2 | S2.2 | Specific Gravity Fraction | Weight (%) | Ash (%) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| -2000+1000 | 3.0 | 22.24 | 15 | 25 | 25 | 15 | 10 | 3 | 2 | 5 | F1.3 | 13.11 | 4.0 |
| -1000+500 | 20.0 | 27.13 | 15 | 22 | 20 | 15 | 12 | 5 | 1 | 10 | S1.3-F1.4 | 21.01 | 8.0 |
| -500+250 | 18.0 | 26.56 | 15 | 22 | 20 | 15 | 12 | 5 | 4 | 7 | S1.4-F1.5 | 19.27 | 12.0 |
| -250+150 | 16.0 | 28.1 | 12 | 22 | 22 | 15 | 12 | 5 | 1 | 11 | S1.5-F1.6 | 15 | 30.0 |
| -150+100 | 12.0 | 30.24 | 12 | 22 | 18 | 15 | 12 | 8 | 1 | 12 | S1.6-F1.8 | 11.94 | 48.0 |
| -100+75 | 8.0 | 32.3 | 12 | 20 | 18 | 15 | 12 | 6 | 1 | 16 | S1.8-F2.0 | 5.38 | 59.0 |
| -75+45 | 6.0 | 35.72 | 10 | 18 | 18 | 15 | 12 | 5 | 1 | 21 | S2.0-F2.2 | 1.57 | 66.0 |
| -45 | 17.0 | 35.56 | 12 | 18 | 16 | 15 | 12 | 5 | 1 | 21 | S2.2 | 12.72 | 85.0 |
| Sum | 100.0 | 29.77 | 13.11 | 21.01 | 19.27 | 15 | 11.94 | 5.38 | 1.57 | 12.72 | Sum | 100.0 | 29.77 |

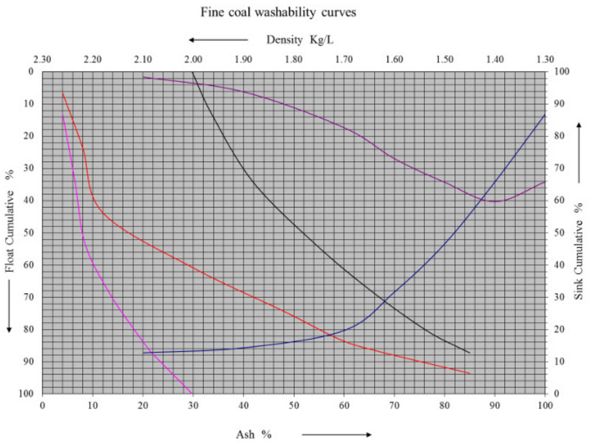


Figure 6. Overall feed washability analysis

Model Parameters

The performance of the spiral separator was simulated using a partition curve model developed by Fourmol (1974). The partition curve is the integral of the normal law:

$$Y(\rho) = C_1 + (C_2 - C_1)\frac{0.6744}{E_p\sqrt{\pi}}\int_{-\infty}^{\rho} e^{-0.4548\left(\frac{\rho-\bar{\rho}}{E_p}\right)} d\rho \qquad (1)$$

Where:

$Y(\rho)$ = partition to the product

$E_p$ = probable error of separation

$\bar{\rho}$ = specific gravity of cut point; $C_1$ minimum value of the density partition curve

$C_2$ = maximum value of the density partition curve

The simulation was conducted using the parameters shown in Table 2.

Table 2. Parameters for spiral separation performance simulation

| Parameters | Values | Unit |
|---|---|---|
| Cut point specific gravity (SG50) | 1.68 | $10^3$kg/m$^3$ |
| Probable error of separation (Ep) | 0.20 | $10^3$kg/m$^3$ |
| Minimum partition coefficient | 0.02 | |
| Maximum partition coefficient | 0.98 | |

Rosin-Rammler's partition curve model was used to determine the proportion of each particle size class in the feed that reports to the coarse product of the classifier. Rosin-Rammler's partition curve model is expressed as follows:

$$Y(d) = H + (100 - H)\left(1 - e^{-0.693\left(\frac{d}{d_{50}}\right)^{M}}\right) \qquad (2)$$

$$M \approx \frac{0.77}{I_m} \qquad (3)$$

$$I_m = \frac{(d_{75} - d_{25})}{2 * d_{50}} \qquad (3)$$

Where:

Y(d) = partition to the coarse product
H = short circuiting fraction
d = mean particle size in each size fraction; d50 corrected particle size of classification, the particle size at which has 50% of probability of reporting to the oversize
M = sharpness of separation
Im = imperfection of the corrected partition curve
$d_{75}$ = the particle size at which has 75% of probability of reporting to the oversize
$d_{25}$ = the particle size at which has 25% of probability of reporting to the oversize.

The simulation was conducted using the parameters shown in Table 3.

Table 3. The parameters for classification performance simulation

| Parameters | Classifying Cyclone | Sieve Bend | Stack Sizer |
|---|---|---|---|
| Short circuit of fines (%) | 0.30 | 0.45 | 0.15 |
| Corrected cut size: d50c (mm) | 0.15 | 0.15 | 0.15 |
| Corrected imperfection | 0.30 | 0.45 | 0.15 |

Several researchers (Fan and Tao, 2014) have investigated the flotation recovery for each density fraction under different operating parameters. Their research found that recovery of the low density fraction is quite high. It is not uncommon for the 1.3 Float particles to achieve near 100 percent recovery; however, recovery decreases correspondingly with an increase in particle density. The recovery for 2.2S can approach 0 percent. Following the Fan and Tao conclusions, the simulation was conducted using the parameters shown in Table 4.

Table 4. Parameters for flotation performance simulation

| Specific Gravity Fraction | Recovery in the froth (%) |
|---|---|
| F1.3 | 98 |
| S1.3-F1.4 | 92 |
| S1.4-F1.5 | 75 |
| S1.5-F1.6 | 60 |
| S1.6-F1.8 | 40 |
| S1.8-F2.0 | 20 |
| S2.0-F2.2 | 5 |
| S2.2 | 0 |

## 4. SIMULATION RESULTS AND DISCUSSION

Mineral processing software, USimPac, was used to conduct the simulation study with the feed data and model parameters provided above. The simulation results for different fine coal cleaning circuits were summarized in the Table 5. Generally speaking, the STK-STK circuit Stack Sizer produced the lowest oversize product ash content, while the CYC-SB Sieve Bend oversize product ash content produced the highest ash content. By providing more efficient pre-classification, the STK-STK circuit helps the gravity circuit to achieve lower product ash. The flotation product ash contents are 15.27 percent, 15.11 percent, 15.59 percent, and 15.55 percent for CYC-SB, CYC-STK, STK-SB and STK-STK circuits, respectively. Recovery for each density fraction per size fraction might require further definition to produce more accurate simulation. Overall, the conventional CYC-SB circuit, modified CYC-STK circuit, modified STK-SB circuit, and modified STK-STK were able to produce clean coal product with ash content of 16.12 percent, 15.94 percent, 16.13 percent, and 16.07 percent, respectively. The clean coal product yield for the four circuits described above were 60.25 percent, 59.61 percent, 60.75 percent and 60.57 percent, respectively. The combustible recoveries were 71.96 percent, 71.34 percent, 72.55 percent, and 72.38 percent, respectively.

Organic efficiency is an indicator of coal cleaning performance and is calculated by dividing actual clean coal yield by the theoretical maximum yield attainable at the same ash content according to washability analysis. The calculated organic efficiency for each circuit is shown in Table 6. It is clear that organic efficiency for the conventional fine coal cleaning circuit (CYC-SB) is less than optimal. Significant improvement in organic efficiency can be achieved by modifying the classification system . As shown in Table 6, the STK-SB circuit organic efficiency is 0.64 percent higher than the conventional fine coal cleaning circuit (CYC-SB).

To quantify the economic benefits of improving fine coal classification, a preliminary economic analysis was conducted for a hypothetical coal preparation plant. The plant was assumed to have capacity of 500 MTPH and 20 percent of feed was fine coal required to be washed with the fine coal cleaning circuit. Assuming the plant was operated 7000 hours annually, the STK-SB circuit could produce an additional 4480 tons of clean coal (equivalent to 0.64 percent higher yield) than the conventional fine coal cleaning circuit (CYC-SB). Assuming coal price at $60US/ton, additional revenue of $268,800.00US could be generated. Improved fine coal cleaning performance can allow the plant to achieve higher heavy medium circuit recovery, lower operating cost, and more flexible circuit operations.

Table 5. Simulation results for different fine coal cleaning circuits

| Unit operation\Circuit | | | CYC-SB | CYC-STK | STK-SB | STK-STK |
|---|---|---|---|---|---|---|
| Run of mine coal feed | Flowrate | MTPH | 100 | 100 | 100 | 100 |
| | Ash content | % | 29.77 | 29.77 | 29.77 | 29.77 |
| | Combustible recovery | % | 100 | 100 | 100 | 100 |
| hydrocyclone underflow/Stack Sizer Oversize | Flowrate | MTPH | 73.48 | 73.48 | 65.02 | 65.02 |
| | Ash content | % | 28.49 | 28.49 | 27.75 | 27.75 |
| | Combustible recovery | % | 74.81 | 74.81 | 66.89 | 66.89 |
| Spiral product | Flowrate | MTPH | 46.54 | 46.54 | 41.69 | 41.69 |
| | Ash content | % | 16.68 | 16.68 | 16.54 | 16.54 |
| | Combustible recovery | % | 55.21 | 55.21 | 49.54 | 49.54 |
| Sieve bend / Stack Sizer product | Flowrate | MTPH | 39.94 | 37.29 | 37.43 | 36.70 |
| | Ash content | % | 16.55 | 16.44 | 16.46 | 16.40 |
| | Combustible recovery | % | 47.46 | 44.37 | 44.52 | 43.69 |
| Flotation feed | Flowrate | MTPH | 33.12 | 35.77 | 39.24 | 39.96 |
| | Ash content | % | 30.15 | 29.26 | 31.75 | 31.53 |
| | Combustible recovery | % | 32.94 | 36.03 | 38.13 | 38.96 |
| Flotation product | Flowrate | MTPH | 20.31 | 22.32 | 23.32 | 23.86 |
| | Ash content | % | 15.27 | 15.11 | 15.59 | 15.55 |
| | Combustible recovery | % | 24.51 | 26.98 | 28.03 | 28.69 |
| Final product | Flowrate | MTPH | 60.25 | 59.61 | 60.75 | 60.57 |
| | Ash content | % | 16.12 | 15.94 | 16.13 | 16.07 |
| | Combustible recovery | % | 71.96 | 71.34 | 72.55 | 72.38 |

Table 6. The organic efficiency for each circuit

| Circuit | Theotical yield (%) | Product ash (%) | Product yield (%) | Organic efficiency (%) |
|---|---|---|---|---|
| CYC-SB | 75.16 | 16.12 | 60.25 | 80.16 |
| CYC-STK | 74.75 | 15.94 | 59.61 | 79.75 |
| STK-SB | 75.18 | 16.13 | 60.75 | 80.80 |
| STK-STK | 75.05 | 16.07 | 60.57 | 80.71 |

## 4. CONCULSION

The fine coal classification technology, especially fine screening technology, has advanced significantly in the past few decades. Those advancements have provided coal processing operations worldwide with more practical options for dealing with this challenge. It is, therefore, worthwhile to examine the effects of these advancements on fine coal cleaning circuit performance. In response to this need, the USimPac-based simulation program was developed to simulate and compare the performance of four different fine coal cleaning circuits. The conclusions obtained from this simulation study are summarized as follows:

1. The STK-STK circuit Stack Sizer oversize product ash content is lowest, while the CYC-SB Sieve Bend oversize product ash content is highest. The flotation product ash content are 15.27 percent, 15.11 percent, 15.59 percent and 15.55 percent for CYC-SB, CYC-STK, STK-SB, and STK-STK circuits, respectively.
2. The STK-SB circuit organic efficiency is 0.64 percent higher than that of the conventional fine coal cleaning circuit (CYC-SB).
3. Preliminary economic analysis showed that additional revenue of $268,800.00US might be achieved using STK-SB circuit instead of using the conventional CYC-SB circuit.

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# TECHNOLOGIES IN USE FOR THE PROCESSING OF FINE COAL

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**ABSTRACT**

Coal is the most abundant fossil fuel and about 42% of the world electricity is generated through coal fired power stations. However, mechanized mining has led to an increase in the quantity of fine coal production. The value of the fine coal necessitated the selection of the correct equipment for the gravity beneficiation of this coal. On the other hand, fine coal might be unwanted; it all depends on the coal, the market and the decision makers. Each technology in use for the processing of fine coal therefore has its own place in the process flow diagram. Technologies have improved, DMS cyclones can treat the particle range spirals would normally do, while efficiency still remains good. Spirals have low capital and operating costs, the teeter bed separator offers a low cut point and water only cyclones are cheaper to operate than froth flotation. The paper addresses the different technologies available for the beneficiation of fine coal and their space and place in the coal processing industry.

**KEYWORDS**

Teetered bed separator; Reflux classifier; Spiral concentrator; Water only cyclone; Dense Medium; Flotation; Coal; Beneficiation

## INTRODUCTION

The increase in the amount and value of fine coal led to a need in the industry for a reassessment of the principles for the selection of the correct equipment for the gravity beneficiation of fine coal particles. With the increase in fine coal production caused by mechanised mining and the need to utilise as much of the mined coal as possible, increased attention is being given to the beneficiation of this material. Up to 15% of run - of - mine coal can be in the minus 0.5mm fraction and economically this material should not be discarded or sold at a low price as an inferior product.

On the other hand, coal in South Africa is difficult to separate. In some instances, fines might be unwanted. For Medupi power station the -4mm fines are not washed. Eskom does not want the fines and have put a limitation on the % fines which can be present in the feed. It is not always necessary to beneficiate all the coal. The technologies in use for the processing of fine coal include teetered bed separators; reflux separators; spiral concentrators; water only cyclones and fine dense media cyclones. The application of these technologies depends on the market, it depends on the requirement to beneficiate the fine coal or not.

## TEETERED BED SEPARATORS

The teetered bed separator (TBS) is also known as a hindered settling classifier. The TBS uses a continuous upward current of water to suspend particles of a predetermined size (-3+.25mm) or gravity (density). The water introduced at the bottom of the column has the greatest velocity. When the falling feed particles achieve the same velocity as the upward current they will not fall any longer. In this teetered state, coal particles will classify themselves so that the coarse particles report to the bottom of the

 
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column. The finer particles disperse to the higher levels of the column where they stay suspended. An overflow allows for the discharge of the finer or lighter material from the classification. A pressure sensitive device is inserted into the teetering pulp to give an indication of its specific gravity, which is used to control the separation taking place in the device. The pressure sensitive device is used to provide a variable signal to operate a valve that controls the discharge of the coarse/heavy material that has accumulated at the bottom of the column. The fundamental operation of the TBS is simple but the TBS is sensitive to particle size. The teetered bed, however, is able to obtain a low cut-point.

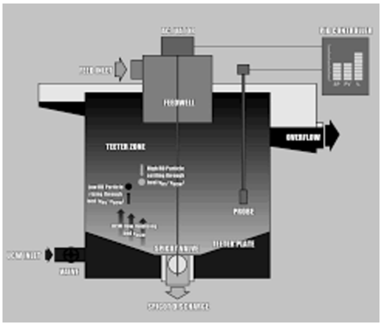


Figure 1: Teetered bed separator

**REFLUX CLASSIFIER**

A reflux classifier separates small particles based on a difference in density or particle size. The feed distributor delivers the slurry into the reflux classifier then undergoes a sorting process due to the force of gravity and the up flow of fluidisation water.

At the bottom of the mixing chamber, a higher density bed of settling solids is formed. The high density bed is kept in suspension by sets of incoming fluidisation water at the base of the chamber. The coarse and fine low density particles that may be trapped in the dense fluidised bed tend to be sorted upward due to the turbulent motion created by the fluidisation water injected from below. The high density particles sink to the bottom of the fluidised bed due to gravity and migrate to the central underflow valve.

The density of the middle section in the mixing chamber is measured and used to determine when to allow high density solids to be discharged from the bottom of the reflux classifier fluidisation chamber via the central underflow valve. The low density particles are held in the slurry suspension and migrate to the lamella section for the reflux classifier. The lamella channels enhance the settling rate of any misplaced fine high density solids, which slide down the plates and slowly re-circulate back into the bed zone of the mixing chamber. The low density and small particles tend to overflow from the reflux. The overflow from the reflux classifier contains most of the process water, the low density solids and any misplaced slimes in the feed.

**SPIRAL CONCENTRATORS**

Spirals are a flowing film concentrator. The capital cost is relative low with virtually no operating cost. Spirals are robust and simple to operate, however they do not always get the attention they should. Spirals need to be washed regularly, one needs to check for blockages and they should not be overfed and not be used as ladders. Spirals are higher cut-point devices but this improves the yield. Spirals are less sensitive to particle size than a TBS.

Figure 2: Spiral Assembly

Table 1 below shows the typical operating conditions for coal spirals.

Table 1: The typical operating conditions for coal

| | | |
|---|---|---|
| Feed size | -1.0 + 0.1 | mm |
| Dry feed | 3.5 | STPH |
| Slurry volume /start | 35 | GPM |
| Feed % solids | 30-35 | w/w |

Deviation from optimum operating conditions influences separation performance:

- Increase in density of separation;
- Lower sharpness of separation

Spiral performance is size dependant. Slimes (usually high ash content) will not be separated efficiently on a spiral concentrator. De - sliming is essential.

**WATER ONLY CYCLONES**

Water only cyclones are a cheaper option (capital and operating costs) for upgrading fine coal than froth flotation or DMS cyclones. Froth flotation is commonly used to process coal fines below 0.5mm due to environmental hazards and the use of costly chemical reagents, it is imperative to identify an alternative technique to process coal fines. Water only cyclones treat coal with a fine feed size, typically 0.5mm. Water only cyclones differ in design from conventional hydrocyclones and heavy media cyclones primarily in their longer vortex finder and wide angled conical bottom. Figure 4 shows the water only cyclone.

Figure 3: Water only cyclone

The water only cyclone receives its feed (-0.5mm) from the de-sliming screen. The underflow is retreated by spirals and the overflow is sized by classification cyclones. The underflow from the classification cyclones reports to the final product and the overflow reports to flotation. Water only cyclones are used to “clean” (or “wash”) raw coal. The truncated cone bottom allows a refuse bed to form which rejects lighter coal particles, while a relatively long vortex finder “vacuums up” the light coal particles. A characteristic of the water only cyclone design is that some coal losses takes place through the spigot because larger coal particles are classified. Pressure and feed density affect the separating gravity. The separating density can be adjusted by the spigot size.

**FINE DENSE MEDIUM CYCLONES**

Fine coal DMS cyclones typically treat a feed top size of 10mm and a bottom cut off of 0.5mm. However they can handle a wide PSD range from 100µm – 1mm.

Figure 4: DMS cyclone

Typical throughput rates are in the order of 50 – 60 tph. Recent advances in magnetic separators to better recover the medium, have prompted renewed interest in using the dense medium cyclone for fine coal processing. The efficiency of separation obtainable with dense medium cyclones is much better than that of water – only units such as spirals and teeter beds. The change that enabled the successful use of larger diameter cyclones and coarser media for fine particle dense medium separation was the multistage treatment of the fine coal which resulted in an overall efficient process. Dense medium cyclones, by virtue of their high separation efficiency, are the method of choice for processing difficult – to – process raw coals.

Dense medium processing is usually more expensive than water only processes since magnetite is consumed. Often, the cost of magnetite is more than compensated for by the improved recovery efficiency of dense medium processes.

## FINE COAL FLOTATION

Flotation is a process that exploits differences in the surface wettability of particles. Various sparging devices produce a bubble swarm that promotes contacting and attachment of the particles. Coal particles strike a bubble and become attached.

Flotation is efficient to about 0.25mm (250µm). There is a larger drop – off in recovery above this size range. It is difficult to maintain a constant and acceptable recovery above this size. The recovery of coal particles in a flotation cell is dependent on the rate; retention time and mixing. Air bubbles must be produced at a small size and on a continuous basis. Residence time indicates how long particles stay in the cell pulp. Allow sufficient time in the system for recovery to occur. Cells must be perfectly mixed. There should be an equal concentration of any component at any location in the system.

## CONLUSIONS

It is well known that coal is the most abundant fossil fuel. Coal supplies about 42% of electricity for the world. The known coal reserves in the world are enough for more than 215 years of consumption. It is predicted that coal will still play the most important role in the energy supply until 2050. Each of the technologies has its own place in the processing plant, for example DMS cyclones are easy to control while they can handle a wide PSD range, but still maintain a good efficiency. Spirals have low capital and operating costs. The teeter bed separator offers a low cut point, while the particle size should remain fairly constant. Water only cyclones offer a cheaper option to beneficiate fine coal than froth flotation or DMS cyclones. Flotation is the process to use to recover ultra-fine coal (10 - 250µm). The first important decision is to beneficiate or not, it all depends on the market, the properties of the coal and the people making the decisions. The next step is to use the appropriate beneficiation technique as each of them has its own unique features to achieve a common goal.

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# Emergency braking of fast running coal handling gantry hoists

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**Abstract**

Worldwide bulk handling processes are realized by massive logistic chains. In ports ship unloaders are part of these logistic chains. Often they are realized as gantry cranes with cable grabs, equipped with increasing speeds for trolley drive and hoist drive. To ensure safety by restricted braking distances, hoists have to be equipped with brakes of higher performance as well. For stop category zero these higher braking torques lead to higher loads on the drive train, maybe causing initial damages in zones of Hertzian contacts. Those damages are liable to provoke serious damages caused by fatigue. Appropriate cases related to roller bearings in hoist gearings were observed in practice. A new hoist design is presented. The design of the hoist and especially its braking system is suitable to prevent high loads on the drive train caused by braking with high torques. Initial damages as a source of further weakening are assumed to be reduced significantly. Functionality of the design is shown on basis of rigid body calculations in connection with an assessment according to DIN EN 13001 and elastic body simulations.

**Key Words**

Mineral handling, Ship unloader, Hoist, Grab, Braking system, Loads, Lifetime, Balanced braking

## 1. Structural development

Worldwide bulk handling processes are realized by massive logistic chains. Part of these logistic chains is long distance transport via ships and trains. With regard to efficiency ships were growing during the last decades. In addition it was and is a permanent target, to reduce port time by fast loading and unloading procedures. This correspondingly applies for train transport.

These aspects are of interest for Russian coal industry, which is to grow 30% till 2030 from its current level of about 347 Mio. t p.a. in 2013. This will be due to increased production of electric energy, start of production of liquid fuels and increase in export. As a result the capacity of Russian coal ports is to grow from 69 Mio. t p.a. up to 230 Mio. t p.a. in 2030. Train capacities are developed as well, both especially in direction of the Far East (Umann, 2014, v. Hartlieb-Wallthor, 2014).

Equipment for such logistic chains, especially for unloading processes, are grab ship unloaders. As an energy efficient system (Tilke, 2010) they are most times provided with the auxiliary trolley system (figure 1). In comparison to other design principles for movement of trolley and grab this leads to a simple mechanical realization of a horizontal load path while travelling (Qi, 2011).

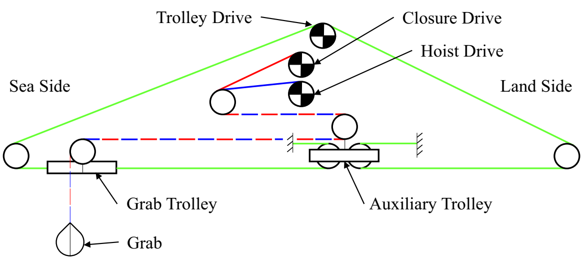


Figure 1: Drive structure for ship unloader with auxiliary trolley



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## 2. Hoist structure

Generally hoists consist of a drive train, to the ends of which loads are applied: At one end the motor and the brake are located, at the other end the load is attached. Relative transparent relations concerning the dynamic behavior of the hoist in different service conditions result out of this. At safety oriented hoists as partly in ship unloaders this looks a little bit different. To cover a rupture of the drive train an additional safety brake is located on the board disc of the rope drum. Thus a load can be applied in the middle of the drive train. In comparison to the general hoist structure a modified dynamic behavior of the hoist in general and the elastic drive train especially is the consequence.

Especially for the case of emergency-off of safety oriented hoists with a safety brake on the rope drum disc high dynamic internal forces occur in the drivetrain. Thus exists the risk of component failure, which can be observed for crane equipment in practical applications (Schmeink, 2014). The failure of a component, especially the hoist gearing, results in consequences relating safety and availability.

The actions of motor and brakes during a braking procedure are not permanent. In fact a sequence of omitting and adding loads on the drive train occurs. Especially the case of emergency-off is considered here with following chronological scenario: After activation of emergency-off and a dead time $\Delta t_M$ the motor torque is dropped out. Parallel the brakes get into action. Takes the brake application more time than the drop out of the motor torque, exist the dead times for the service brake $\Delta t_{BB}$ and the safety brake $\Delta t_{SB}$.

## 3. Reference system

For a closer look on the behavior a loss-free, partly redundant hoist with safety brakes is considered (Vöth, 2015). For the hoist represented as a rigid body model the behavior of load speed over time can be calculated. As a result for example the speed over time for different mechanical braking scenario out of hoisting/lowering the dead load respectively the full load are gained (figure 2). This assumes no variation of frictional behavior, practically given (Römer, 2012).

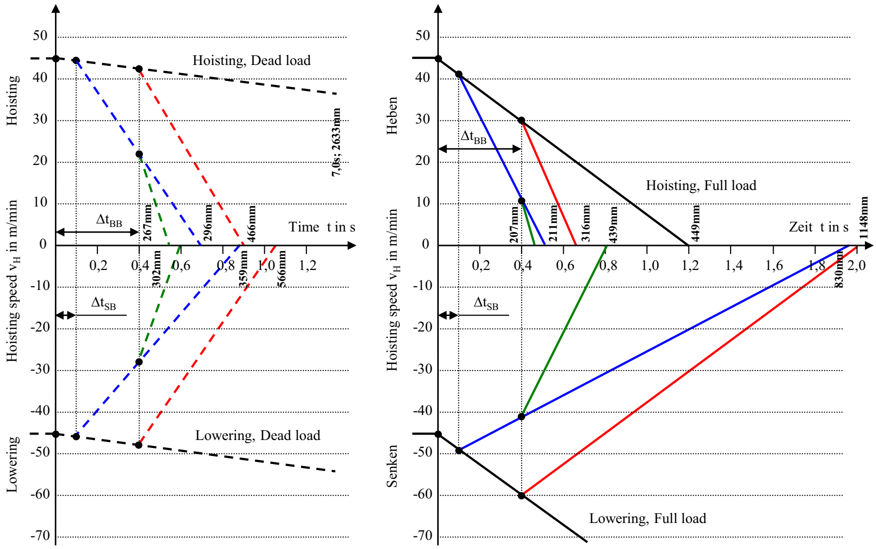


Figure 2: Speed characteristics for braking the dead load out of hoisting/lowering

## 4. Braking during hoisting

Examinations show the special relevance of the load case emergency-off out of hoisting the maximum load. As observed the deceleration of the hoist occurs quickly. This is caused mainly due to the braking by the high load. Thus the deceleration process is finished as fast that the holding brake with greater dead time in general is not or fairly not coming into action anymore. Firstly the torque for braking the motor mass is supplied by the load and the safety brake and transferred to the motor mass via the drive train.

Assumed

- simultaneous action of the safety brake and switch-off of the motor,
- braking torque built up according to the character of a jump function,
- a delayed action of the service brake (holding brake) in comparison to the safety brake and
- a loss free drivetrain,

the maximum relative gearing input torque for braking with the safety brake out of hoisting reaches the level shown in figure 3, as far as they are supported by the static load, braking torque and inertia torque. Here the torque jump resulting out of the change of service condition according to the rigid body model is assessed with the dynamic factor for drives $\phi_5$ corresponding DIN EN 13001-2 (DIN, 2014). Details regarding this are to be determined by an elasto-kinetic analysis. It is obvious for the considered load case that internal forces resulting in the drive train are a multiple of the static holding torque.

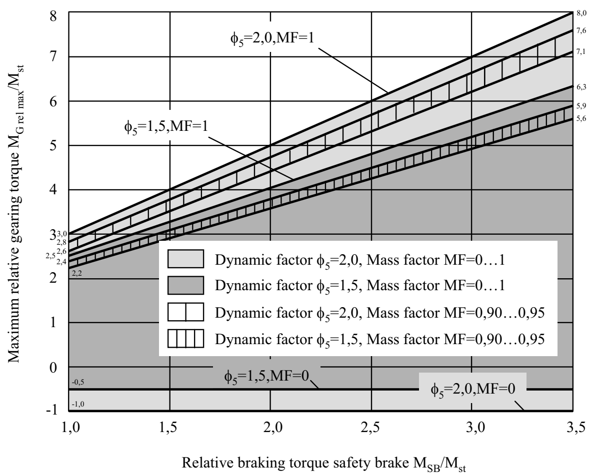


Figure 3: Maximum relative gearing torque

For a braking factor of the safety brake of $BF_{SB}$=1.7 results a relative peak gearing torque assessed according to DIN EN 13001-2 of $M_{G\ rel\ max}$=2.5…4.2. For $BF_{SB}$=3.3 this value reaches a level of $M_{G\ rel\ max}$=3.9…7.2. That means the gearing torque is about factor 7.2 higher than the maximum static internal force in the drive train.

## 5. Braking during lowering

Analyses show that especially the load case of emergency-off out of lowering the dead load is of interest. Emergency-off immediately initiates switch-off of the motor and activation of the safety brake.

For the mentioned data a maximum relative gearing input torque of $M_{G\ rel\ max}$=8.9 is calculated (figure 4). That means the gearing torque is factor 8.9 higher than the maximum static loading of the drive train.

During lowering the hoist is driven by the load, which is hold in steady state condition by the motor. When the safety brake gets into action, the stoppage is executed very fast for this case as well. On one hand this is caused by the low load level, dead load. Assuming clearances in the drive train (in gearing and/or couplings) it is expected furthermore, that a flank change will occur. During this the motor side masses and the braked load side are uncoupled. Respectively the motor side masses need not to be decelerated.

At an appropriate constellation the load side will stand still before running through the clearance is finished. In this case after running through the clearance a shock will occur. The motor side pitches on the standing load side. Toothed wheels and bearings in the gearing are loaded significantly by this shock. A special shock load may occur to the bearings of helical gearings. In this case the shock is led in axial direction of the shafts and with it on the roller bearing acting as fixed bearing.

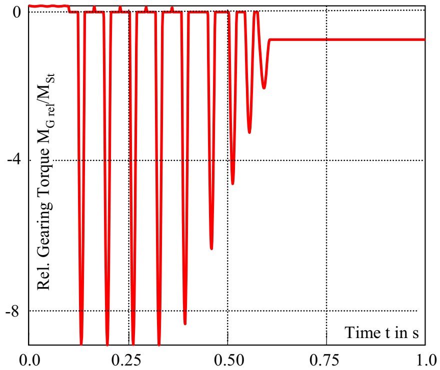


Figure 4: Relative gearing torque according to elasto-kinetic hoist model

## 6. Gearing loading

From the calculations maximum internal gearing torques much higher than according to static or rigid body approaches can be derived. Especially in the gearing such shock-like internal loads appear after running through clearances in relation with Hertzian contacts (toothings, roller bearings). To ensure safety such maximum internal loads must be covered statically. To ensure durability such maximum internal loads should not impose pre-damages, which would lead to fatigue under further service loading.

## 7. „Intelligent“ braking

In zones of maximal exploitation of the strength of material pre-damages must be prevented. Shall the gearing not be dimensioned too large, the appearing internal forces are to be restricted. For the given hoist structure approaches to reduce the maximum values and amplitudes of the internal forces are demanded. As measure to reduce peak values and amplitudes of the internal loads is considered: Synchronous and balanced action of all brakes participating in the braking process, here the holding brake and the safety brake.

Main reason for high internal forces in the drive train during a safety braking process is following situation: The maximum of the kinetic energy to be reduced is concentrated in the masses on the axis of the fast running motor shaft: masses of motor, coupling and braking drum/braking disc. Braked will be at first at the board disc of the rope drum and by the load. So the brake torque is not induced at the location of demand. A significant part of the brake torque must be led from the location of induction to the fast rotating masses. To prevent this torque put through the gearing it makes sense, to bring the service brake into action synchronous to the safety brake.

This leads to a direct participation of the service brake in the braking process. This ideally results in a switching scenario with dead times of the service brake and the safety brake of $\Delta t_{BB}=\Delta t_{SB}=0s$. Requirement is a holding of the motor torque until both brakes get into action.

Remains the question with which amount of torque the safety brake and the service brake should act. Favourable would be braking in a way that the quasi static internal torque before braking is still present during braking. Hereby at the beginning of braking a jump in the internal torque during transition from “hoisting/lowering” to “decelerated hoisting/lowering” is prevented. Likewise at the end of braking a jump in the internal torque during transition from “Decelerated hoisting/lowering” to “holding” is prevented. Assuming these requirements given for the structure of the reference hoist following braking factors for the safety brake and the service brake for the braking out of hoisting or lowering are calculated as shown in figure 5.

For braking out of hoisting the safety brake has to be applied to a little account only ($BF_{SB}$=5%-12%). The service brake has to deliver a significant torque under partial load. With increasing loads up to full load the braking torque of the service brake is decreasing continuously ($BF_{BB}$=92%-8%).

For braking out of lowering the safety brake has to deliver only a small torque ($BF_{SB}$=5%-12%). The service brake has to deliver a significant torque under partial load. With increasing loads up to full load the braking torque of the service brake is increasing continuously ($BF_{BB}$=124%-208%).

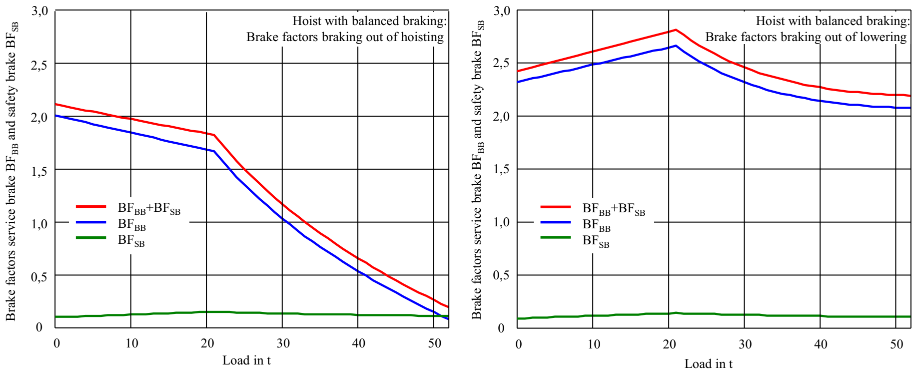


Figure 5: Braking factors for braking out of hoisting / Braking factors for braking out of lowering

## 8. Conclusions

For the higher internal loads occurring especially due to emergency-off the hoist may be not dimensioned reasonably and efficiently. Assuming a corresponding hoist concept this also applies for emergency-stop, a load case occurring more often. Accordingly measures have to be considered in order to reduce internal loads induced to the drive train. The internal loads in the drivetrain are reduced especially by “intelligent braking”. Ideally the braking process is designed in a way, that during braking in the drivetrain between motor and safety brake the torque during static hoisting is pas well as the amplitudes of the internal forces are reduced significantly. Suitable measures to be applied are:

- Reduction of clearances and increase of system elasticity: As result shocks can be reduced and absorbed, as well as internal loads are reduced in connection with system damping.

- Minimisation of dynamic effects: Following DIN EN 13001-2 this may be realized by little clearance and a gradual implementation of the braking torque.
- Reduction of mass factor MF: By a small share of the motor mass in relation to the total mass of the drivetrain the torque put through the gearing is reduced.
- Minimisation of brake factor $BF_{SB}$: A small braking torque of the safety brake generally leads to less braking action and reduced internal forces.
- Braking action synchronous to motor switch-off: Is he motor moment decreasing before braking action takes place, the drivetrain is relieved slightly. The resulting internal forces can be prevented by synchronity of the events.
- Synchronized application of safety brake and service brake: In order to prevent torques put through the drive train a synchronized application of brakes is inevitable. As a result the collision of the non-braked massive drive side mass and the braked load side mass is prevented. Corresponding shocks in assemblies with clearances as gearing and rope drum coupling are reduced. For typical hoists the dead time of the safety brake is significantly lower than that of the service brake. An expandation of dead time of the safety brake in most cases cannot be accepted. Accordingly a suitable approach is to shorten the dead time of the service brake (Römer, 2013).
- Balanced braking torque of safety brake and service brake: For adjusting the torques in the drive train defined braking torques at safety brake and service brake are required. Advisable is the balancing of both braking torques according to the energies to be dissipated at the locations of brakes. These braking torques depend on the service condition and the suspended load. Brakes with controllable torques are applied ideally. For cranes they are not state of the art today. Instead of the stepless adjustment of torques a stepped adjustment of braking torques may be considered. This is realized by a parallel arrangement of several smaller brakes at one braking location.

Described effects are the result of action of safety brakes on the rope drum. Best way to prevent these effects would be the omittance of safety brakes at all, if justified by the risk analysis.

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# HiBar Steam Pressure Filtration of Coal Ultrafines - New Developments and Results

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**Abstract**

HiBar Filtration is the most modern technology of continuous pressure filtration also known as hyperbaric filtration. HiBar Filtration enables low moisture contents, high specific solids performance and efficient filter cake wash if fine grained, difficult to filter products have to be processed. Lowest cake moisture contents are achieved with HiBar Steam Pressure Filtration. With this hybrid separation process the filter cake is treated with steam immediately after emerging from the slurry. In a specially designed and patented steam hood the filter cake is only partially exposed to steam, which accelerates and intensifies the dewatering process. For many products the dryness of the filter cakes significantly improves quality, handling and transport of the product. At filtration of bulk materials such as coal ultrafines these are decisive criteria. HiBar steam pressure filtration is capable to produce extremely dry ultrafines below 10% w/w free moisture which now offers new options in coal ultrafines treatment. In July 2014 the BOKELA HiBar pilot plant was operated at the coal washery Auguste Victoria (RAG), Germany, for Steam Pressure Filtration of coal ultrafines in front of experts from the international coal industry. It was a world premiere and for the first time filter cakes of dry coal ultrafines below 9% w/w free moisture were produced in a semi-industrial scale.

**Keywords**

Continuous Pressure Filtration, Hyperbaric Filtration, Coal Ultrafines, Flotation Coal, Rotary Pressure Filter, Steam Pressure Filtration

## 1. Introduction

Depending on the coal deposit the amount of ultrafines can sum up to some 10% - 40% w/w. If the free moisture content of the coarse and fine coal fraction is below 10% w/w, then filtration of ultrafines with modern rotary vacuum disc filters is a profitable way to produce a dewatered product with 20 – 30% w/w free moisture, allowing a mixture of a considerable amount of ultrafines in to the end product. Results of filtration and dewatering of coal ultrafines (Australian flotation concentrates, South African coking coal flotation cell overflow and underflow) with vacuum disc filters have been reported by Hahn et al (2011 and 2013). If the coarse and fine coal fractions have a moisture content close to 10% w/w, then HiBar steam pressure filtration is capable to produce extremely dry ultrafines below 10% w/w free moisture which now offers new options in coal ultrafines treatment. Ultrafines dewatered with steam pressure filtration can be either marketed as own product or admixed to the coarse and fine fraction.

## 2. Ultrafines Dewatering with continuous HiBar Steam Pressure Filters

### 2.1 Plant and process design of HiBar Filtration technology

HiBar Filtration technology uses a rotary filter - for coal dewatering a disc filter is used - which is installed inside a pressure vessel filled with compressed air (7 bar(a)). Accordingly, the filter runs with a differential pressure of up to $\Delta p$=6 bar. The filtrate pipes are connected to the environment and the suspension is pumped by an appropriate pump into a pressurized vessel. The filter cake is removed from the filter cloth by compressed air blowback and discharged from the pressurized zone through a sluice system. The vacuum pumps used with vacuum filters are replaced by a compressor that supplies the necessary compressed air to the vessel and for compressed air blowback. The compressed air from cake blowback also serves as process air to maintain the overpressure in the vessel for the filtration process.

 
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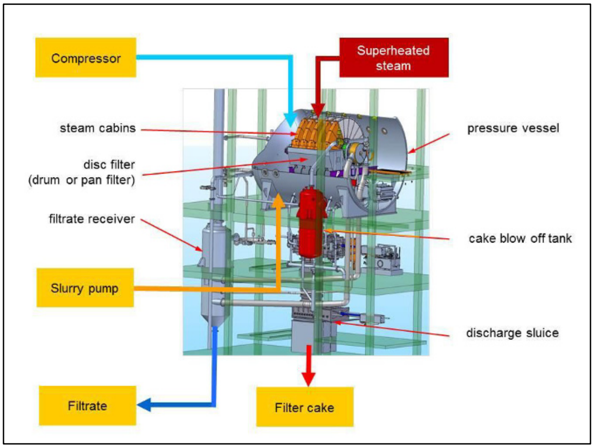


**Figure 1: Plant design of HiBar Filtration**

**Figure 2: HiBar disc filters (70 m² each) with steam cabins for steam pressure filtration in pressure vessel (left); filter building with two HiBar disc filters 70 m² each (right)**

### 2.2 HiBar Steam Pressure Filtration

For steam pressure filtration the filter discs are equipped with steam cabins and with feed pipes for steam supply. The use of steam at continuous HiBar steam pressure filtration leads to a combined thermal/mechanical phenomenon during filter cake dewatering followed up by a subsequent convective drying with pressurized gas (Gerl and Stahl, 1996).

At HiBar steam pressure filtration the filter cake is only partially exposed to steam, which accelerates and intensifies the dewatering. This process can be shortly explained as follows:

A filter cake which is formed at the low temperature of the feed slurry enters a specially designed steam cabin immediately after emerging from the slurry in the filter trough. Here, a superheated steam atmosphere exists and the following phenomena take place which can be described by the model of the "condensate front":

- The steam condenses on the cold cake surface and a homogeneous condensate layer is formed moves through the cake in a piston like flow ("condensate front").
- The moving "condensate front" replaces nearly 100% of the 'mother liquor'.
- When the "condensate front" reaches the filter cloth, the filter cake is 'heated up' completely to the steam temperature. At this point the cake leaves the steam cabin.
- Then compressed air passes the pre-dewatered and hot filter cake causing a very effective thermal drying which leads to extremely low cake moisture contents.

These thermal/mechanical processes inside the cake lead to a nearly homogeneous and highly intensive cake demoisturing without pressure and energy loss by a "fingering".

### 2.3 Results of ultrafines dewatering with HiBar Steam Pressure Filtration

Steam pressure filtration achieves extremely dry ultrafines in a new dimension that eliminates former limits in ultrafines treatment. Typical values for solids throughput and moisture content are below 10% w/w as shown in figure 3 and table 1.

In figure 3 moisture content for pressure filtration (black line) and steam pressure filtration (blue line) for a flotation coal ($x_{50}$ < 45 µm) is shown versus dewatering ratio $\alpha_2/\alpha_1$ ($\alpha_2$ = dewatering angle, $\alpha_1$ = cake formation angle) respectively $t_2/t_1$ ($t_2$ = cake dewatering time, $t_1$= cake formation time). Results shown have been attained from lab tests with a pressure difference of 5.5 bar, feed solids of 350 g/l and flocculent dosage of 13 g per 1000 kg. The red line shows the corresponding specific solids throughput. Typical dewatering ratios on rotary filters range to 1 - 3 as indicated in figure 3. Within this range pressure filtration achieves free moistures from 21 to about 17.5% w/w while steam pressure filtration achieves free moisture contents from 11 to 9% w/w depending on dewatering ratio and amount of steam. As a rule of thumb 10 kg steam/t (d) are necessary to reduce moisture by 1%-point compared to pressure filtration.

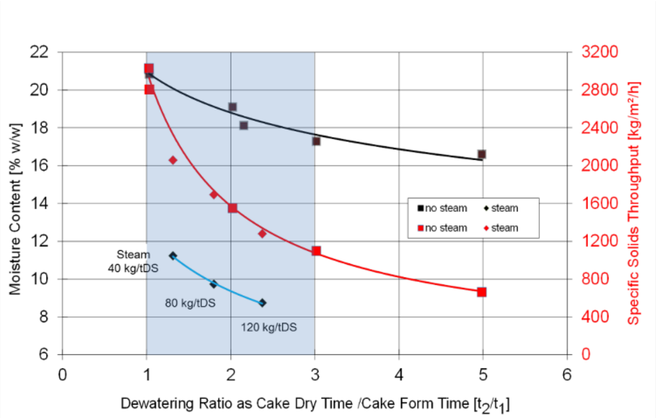


**Figure 3 Free moisture content and specific solids throughput for flotation coal with pressure filtration and steam pressure filtration vs dewatering ratio (dewatering ratios on rotary filters range to 1 – 3)**

**Table 1: Typical moisture contents of coal ultrafines with vacuum filtration, pressure filtration and HiBar Steam Pressure Filtration for -250 µm**

| Method | % w/w |
|---|---|
| Vacuum | 23 – 28 |
| Continuous Pressure Filtration | 16 – 19 |
| **HiBar Steam Pressure Filtration** | **8 – 12** |

Since steam pressure filtration accelerates and intensifies the dewatering not only lower moisture contents are achieved compared to pressure filtration but also higher specific solids throughputs are attained since the filter can be operated with a bigger cake formation angle and higher filter speed.

### 2.4 HiBar pilot plant operation for steam pressure filtration of coal ultrafines

In July 2014 the BOKELA HiBar pilot plant was operated at the coal washery Auguste Victoria (RAG), Germany, for Steam Pressure Filtration of coal ultrafines (figure 5). Experts from the international coal industry travelled to Ruhrcoal to witness this event. It was a world premiere and for the first time filter cakes of dry coal ultrafines below 9% w/w moisture were produced in a semi-industrial scale. Figure 5 shows steam consumption versus moisture content for HiBar steam pressure filtration of coal ultrafines as achieved during these test trials. Pressure difference for cake dewatering was $\Delta p$=3 bar (vessel pressure = 4 bar). As can be seen cake moistures significantly below 10% w/w were achieved with steam consumptions of some 120 kg/t (d). As mentioned above it can be stated as a rule of thumb that with coal ultrafines 10 kg steam per kg (d) effect a moisture reduction of 1% w/w which is valid in the range of 20 to 8% w/w moisture. Slight deviations from this correlation in the graph of figure 4 result from edge effects which falsify steam consumption to somewhat higher values due to the small filer area of the pilot filter of only 1 m².

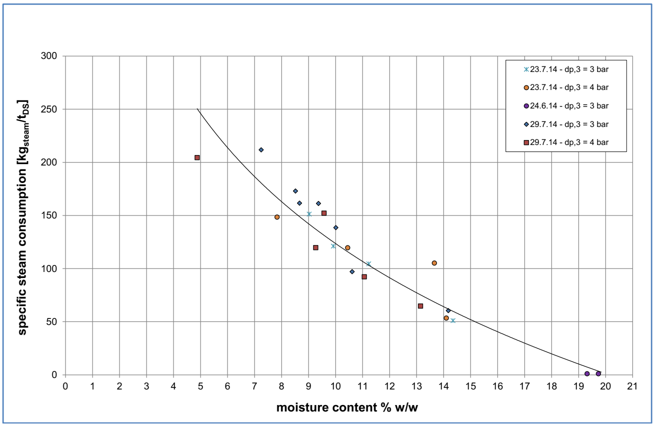


**Figure 4 Steam consumption vs moisture content for HiBar steam pressure filtration of coal ultrafines; results of HiBar pilot plant operation (1m² disc filter) at the Coal Mine Auguste Victorica/Germany**

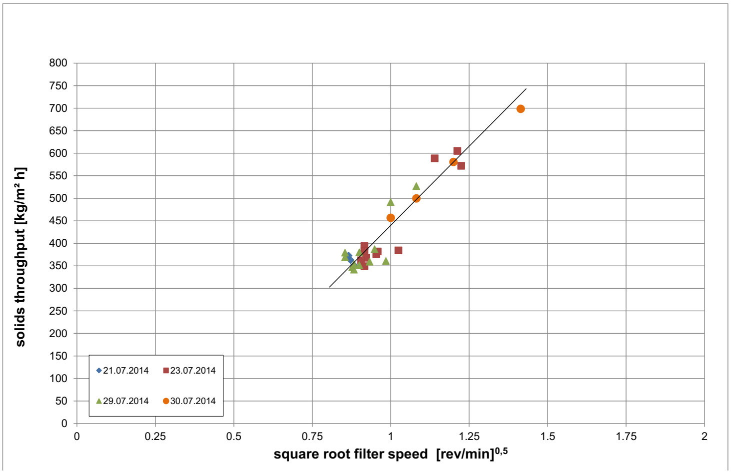


**Figure 5 Results of HiBar pilot plant operation (1m² disc filter) at the Coal Mine Auguste Victorica/Germany; solids throughput vs square root of filter speed for HiBar steam pressure filtration of coal ultrafines**

Figure 5 shows solids throughput vs square root of filter speed for values of filter speed from 0.7 to 2 rev/min. According to filtration theory (Ehrfeld et al 2008) the solids throughput increases linear with square root of filter speed from about 350 kg/m²h to 750 kg/m²h. While pressure difference for cake dewatering was $\Delta p=3$ bar the pressure difference for cake formation was adjusted via throttle valve to only $\Delta p=1$ bar. This limitation in pressure difference for cake formation was necessitated by the discharge sluice of the pilot plant which was not able to handle all solids which would have been produced with full pressure difference $\Delta p=3$ bar in cake formation zone. With higher pressure difference and with use of flocculent solids throughput for these coal ultrafines would increase – with cake moisture unchanged - by a factor of 3 to 4 to values comparable to that of figure 3.

**Figure 6 HiBar pilot plant operation (1m² disc filter) at the coal washery Auguste Victoria (RAG), free moisture content of discharged filter cake: 8% w/w**

### 2.5 Waste to product - advantages of dry ultrafines

The continuous HiBar Steam Pressure Filter is able to operate at extraordinarily high throughputs in continuous filtration and also to achieve dry filter cakes below 10% w/w moisture, which eliminates former limits and offers new options in the treatment of coal ultrafines, such as

- Allowing mixtures of coarse and fine fractions in any given amount;
- marketing as an own product;
- reduced transport costs through reduced water content;
- bulky flow behaviour for discharge of railway wagons;
- improved transport in cold regions with long frost periods (prevention of freezing solids); and
- lower or even no energy cost for thermal drying.

To turn coal ultrafines from a waste into product the cake moisture must not exceed 9-10 % w/w. BOKELA HiBar Steam Pressure Filtration produces ultrafines at this target moisture. For this some 10 kg steam/t (d) are necessary to reduce moisture by 1 percentage. Regarding a filter cake of 17% w/w free moisture achieved with pressure filtration steam pressure filtration requires 80 kg steam/t (d) to reduce free moisture to 9% w/w (□mc=8% w/w). Total operational cost for this range to some 5 US dollar per 1000 kg solids. Considering current market prices, this represents a profit of about 50 US dollar per ton solids. As a result, the return of invest of a large turnkey HiBar Steam Pressure Filtration plant with a performance of 300,000 t/a is achieved in about one year.

### Conclusion

HiBar Filtration technology for continuous pressure and steam pressure filtration enables profitable solutions and new options in filtration of coal ultrafines and iron ore concentrates. The continuous HiBar steam pressure filtration is capable to produce extremely dry coal ultrafines below 10% w/w free moisture which now offers new options in coal ultrafines treatment and to turn waste into a saleable product. Ultrafines dewatered with steam pressure filtration can be either marketed as own product or admixed to the coarse and fine fraction in any wanted amount. Iron ore concentrates can be dewatered to lowest moisture contents of 3% w/w. Comparable to coal ultrafines this improves transport in cold regions with long frost periods or makes transport possible at all.

The next pilot operation of the HiBar pilot plant with ACARP at Tahmoor coal mine in NSW will take place most likely in autumn 2016.

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# Coal Preparation Plant Design Requirements for 2050

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## ABSTRACT

Coal preparation technology has come a long way in the past 35 years. Larger unit operations with enhanced efficiency, increased availability and new technologies in fines processing have been developed and implemented on a global scale in this timeframe. But where to for the next 35 years? This paper proposes the concept of developing an operator-free preparation plant as the vision for future coal preparation development. This will result in much more compact plant designs unencumbered with having to provide the necessary floor space and headroom for operators and maintenance staff to safely and freely access all parts of the plant. This will allow the cost effective application of small scale distributed plants across large multi-pit mine sites to optimise the overall coal chain by minimising raw coal and waste haulage. A large number of technical, operating, maintenance and management challenges will need to be addressed if this vision is to be implemented.

## KEYWORDS

Coal Processing, Plant Design, Advanced Coal Processing, Plant Cost Analysis, Operator-free plants

## INTRODUCTION AND BACKGROUND

By 1980, a large number of coal preparation plants were being designed and constructed in Australia and elsewhere to exploit the growth in the export trade in both coking and thermal coal. Most of these plants were based on the latest CPP design in the USA at the time utilising dense medium cyclones to process the 50 mm topsize coal after desliming using 0.5 mm ww screens. With flotation processing the fines, this resulted in relatively simple two circuit plants. Alternative coarse circuit designs incorporated dense medium baths to process the coarsest sizes, often at a larger plant feed topsize. Alternative gravity processes for fines such as water washing cyclones were introduced into western Canada and elsewhere. Use of 2.4 m wide low-head screens supplemented with sieve-bends prior to the screen was commonplace, which effectively limited the capacity of a single module to approximately 200 t/h when installed with two parallel 700 mm dense medium cyclones. For large capacity plants, this resulted in multiple parallel modules (e.g. Peak Downs with a 4 x 2 module configuration for a nominal 1,500 t/h plant feed rate).

Over the past 35 years, however, significant advances in coal preparation design have resulted in improvements in overall plant efficiency and significant reductions in plant size, Capex and Opex due to use of much larger unit operations such as large dense medium cyclones, multi-slope screens, flotation cells, and alternative fine coal gravity separators such as spirals and more recently reflux classifiers. Availability has improved due to the simpler layout and significant reduction in the number of process components such that operating hours of well designed plants, subject to effective preventative maintenance programs and well trained and motivated operators is of the order of 7,500 h/a or more.

## ECONOMIC DRIVERS FOR CHPP

A high level study of the key economic drivers of CHPP design was undertaken to provide context for any future changes and improvements (Clarkson & Hillard, 2016). This study focussed on three main factors – yield, Capex (Capital Expenditure) and Opex (Operating Expenditure).



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**Yield**

Table 1 shows a notional table of organic efficiency (ratio of practical yield to theoretical yield) for a given washability for a range of process sizes in a typical contemporary coking coal plant using large diameter dense medium cyclones for the coarse fraction, reflux classifiers for the mid-size and column flotation for the finest size fraction. Such a circuit is representative of current ‘best practice’ plant design.

Table 1: Comparison of organic efficiency for different size fractions

| Unit Process | Size Range | *Theoretical Yield% (d/d) | **Practical Yield% (d/d) | Organic Efficiency% |
|---|---|---|---|---|
| Dense Medium Cyclone | -50 + 1.4 mm | 55.1 | 54.9 | 99.6 |
| Reflux Classifier | -1.4 + 0.25 mm | 54.6 | 49.8 | 91.2 |
| Flotation | -0.25 mm | 58 | 42 | 72.4 |

*Theoretical yield at 1.50 cutpoint

**Actual Yield at same product ash

Table 1 shows that the dense medium cyclone yield is very close to the theoretical yield, despite 25% near gravity material (+/-0.1 RD) due to its low Ep (0.023 for -50+1.4 mm) and constant cutpoint by size above 4 mm. Hence there appears to be little potential for meaningful gains in yield for the coarse circuit, other than maintaining a constant offset to avoid the cutpoint varying over time, despite a constant medium density (Crowden et al, 2013). The reflux classifier has relatively low Ep’s (for the size range in question) of the order of 0.08 (Mitchell et al, 2014), but together with continuous changes in cutpoint with size still result in an appreciable loss of efficiency. Hence the need for the further development of fine coal processing units and sensors so that the operator can measure and control the true cutpoint on-line (Firth et al, 2014).

Despite flotation being accepted as the most effective process for treating the ultrafine feed in coking coal and thermal coals which are amenable to flotation, there are still significant losses in practice. Hence the area of ultrafine processing is perhaps one of the most important areas requiring further development to maximise total yield. Alternative processes that promise increased recovery of fine coal and improved selectivity such as oil agglomeration (van Netten et al, 2015) or ultrafine coal reflux classifiers (Vaughan et al, 2016) require further development and optimisation to provide a cost effective means of efficiently implementing such technology. Supplementary processing areas such as improved fine coal classification and dewatering are of importance.

**Capex**

When “greenfield” projects are analysed over a life of mine basis, the influence of Capex is usually less important than Opex, predicted product pricing and exchange rates. However, in a practical sense development of new projects is often constrained by the reluctance of investors to expend large amounts of capital on coal projects. Hence a focus on minimising Capex is almost always pertinent to plant design. The Capex for a specific plant designed using contemporary design concepts largely depends on the complexity of the circuit and plant throughput. Up to 600 t/h, economies of scale available with larger coarse circuit processing units can be offset to a degree by using simpler layouts such as combining multiple sumps into a single vessel with internal dividers for each stream, and installing pumps and plumbing external to the plant as all maintenance can be achieved using external mobile cranes. Depending on location, the plant may have to be clad due to environmental considerations or in colder climates, often has to be enclosed in weather-proof, heated buildings.

Similarly coal handling costs change with capacity. At low tonnages up to say 600 t/h, above ground ROM feeders can be used which require 100% rehandle, but at tonnages >800 t/h it is no longer practical to manage 100% recycle using a front end loader for tramming coal on the ROM, and it becomes more cost effective to install a more expensive elevated dump hopper into which coal is direct dumped from the mining operation whenever possible.

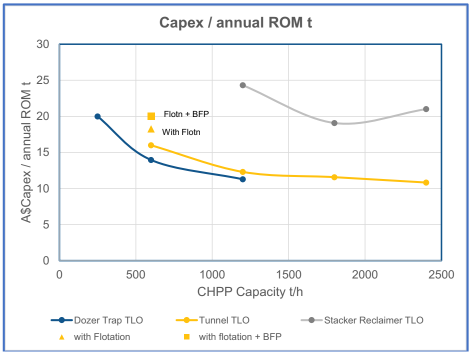


Figure 1: Capex cost curve versus capacity (7,200 operating h/a) for various reclaim and train load out (TLO) configurations

Figure 1 shows a plot of estimated CHPP Capex per annualised ROM tonne versus capacity for a single stage DMC/RC CPP with two different styles of coal handling for product. Low Capex but higher Opex options using radial stackers and dozer reclaim from either above ground dozer traps at low tonnages or underground tunnel reclaim with coal valves for product handling for higher tonnages. High

Capex options using linear stackers and full face mechanical reclaimers for product handling, which provide superior blending capability and control of product quality, are included for throughput options > 1,000 t/h. Two additional CPP options for the 600 t/h case are shown including flotation and belt filter presses (BFP) for dry disposal of all rejects.

The Capex estimate reflects full project costs in 2015 Australian dollars, including engineering, procurement, fabrication, construction, commissioning and commercials such as contingency and constructor's margin. Owner's costs such as land acquisition are excluded.

**Opex**

A high level analysis has been made of the key Opex components for a modern plant with basic coal handling components of ROM, sizing station, surge bin, CPP, product stockpiles and train load out.

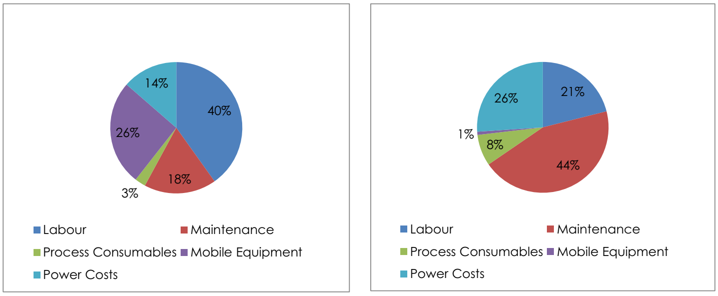


Figure 2 Breakdown of Comparative Opex for 250 t/h and 2,400 t/h CHPP

The Opex has been broken down into five basic components of labour, power, consumables, maintenance and mobile equipment. Figure 2 shows conceptual pie charts for both a low capacity 250 t/h modular style plant operation and a 2,400 t/h large capacity dual module fixed plant with mechanical stacker reclaimers. In each case, a continuous 24/7 four shift operation is assumed with 7,200 annual operating hours. As expected there is a significant reduction in Opex up to a plant capacity of 600 to 1,000 t/h due to reduced usage of mobile equipment and economies of scale in labour and to a lesser degree maintenance where similar work is required to maintain much larger capacity process items. However, above 1,000 t/h, there are only modest reductions as the scalable components of Opex such as power, consumables and maintenance begin to dominate. Figure 3 shows the combined Opex and Capex over a 20 year period in constant 2015 dollars. There is a significant reduction in total cost / ROM t as plant capacity increases to approximately 1,000 t/h, above which the curves tends to flatten out. In addition, the relative cost differential / ROM t of more expensive capital options using mechanical stacker / reclaimers is considerably reduced because the Capex is amortised over a 20 year period.

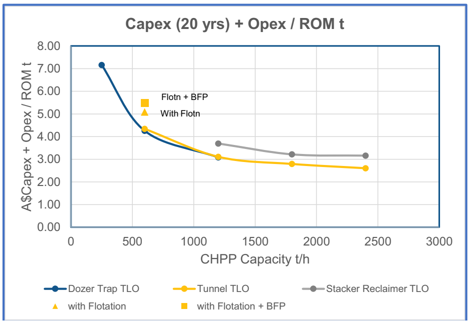


Figure 3 Combined Capex (over 20 years in 2015 dollars) and Opex

**Conclusions from Capex / Opex Study**

From the above study of the combined interaction of plant capacity with both Capex and Opex, it can be seen that economies of scale favour CHPP facilities of 1,000 t/h or greater, which has led to the conventional wisdom of having large, central coal processing facilities. However, for high capacity mine sites of 10 million t/a or greater, the raw coal may be sourced from multiple open cut pits and various underground mines with the ramps and/or portals being located over a wide area. Average yields for a number of established coal mining regions are steadily decreasing as the higher yielding resources are becoming worked out, and new "greenfield" resources that are currently being developed such as the Mozambique coking coal mines are also often lower yielding. This has a substantial impact on the cost of the total coal chain from the mining face to the TLO as increased quantities of raw coal have to be transported from a variety of mining faces to a central area, and increasing quantities of rejects including fine tailings have to be transported back to a suitable emplacement location. Significant savings can be made in hauling coal and rejects if a large central processing plant is replaced by multiple, smaller plants located closer to the original source of raw coal, allowing rejects to be emplaced closer to its original location in each instance.

So what are the factors to be addressed that increase relative costs of small plants? A comparison of the breakdown of Opex costs for small plants compared with large plants (Figure 2) indicates that labour consumes an increasing proportion of the Opex budget for smaller plants, resulting ultimately in a higher cost per ROM tonne on a combined Capex and Opex basis. This labour cost increase is due to the economies of scale with the same number of operators (say three) capable of running a much larger

~1,000 t/h CPP as a small modular plant. Other major Opex components such as power and consumables, and to a lesser extent, maintenance, are much more scalable to the tonnage.

If the combined Capex and Opex cost/ROM t is to be reduced for small scale plants to allow for the increased cost-effectiveness of distributed, smaller CPP's, continuous labour demand at the plant site must be reduced so that operational and maintenance functions can largely be carried out remote from the plant. This will require the plants to become more reliable, requiring only periodic manual intervention when something goes wrong and requires repairing. Innovative coal handling solutions will also be required to minimise raw coal and reject handling at the remote CPP site, but product coal blending and storage prior to railing can still be centralised in a single stockyard.

## VISION OF CPP DESIGN 2050

With the above factors in mind, the key concepts of a plant that will be designed in 2050 could be as follows.

Each unit process within the CPP will be scaled down to the smallest physical size required for processing of the required throughput in the most efficient manner. Each unit, together with the supporting plumbing, electrics and controls will be assembled as a compact, removable module or "cassette" designed to standard dimensions, which may be readily removed and replaced when necessary for major maintenance or replacement. Each module will be fabricated off-site and the modules for different process units will be stacked together into a much smaller overall package than current plants of similar throughput. A key to success will be standardising the interface between each module in terms of plumbing, electrics and controls.

The concept is simply an extension of the current style of fabricating and constructing small modular plants, with all process equipment, plumbing and ancillary equipment being fabricated off-site prior to being installed in its final location as discrete modules. This will also reduce construction costs by transferring as much fabrication and assembly as possible off-site to a lower cost and more productive environment.

The key difference compared with current design practices is that the modules will be much smaller and more compact as they will be designed such that operators and maintenance crew will not have access to directly inspect, clean and maintain each individual component of the module in its operating position. Allowing for safe and efficient access to all plant gear in situ results in a greatly expanded plant volume, with downstream impacts on the scale of initial earthworks, structural costs, painting requirements, increased energy requirements due to increased pumping heights, and increased electrical costs due to longer cable runs. If a failure occurs that can't be "reset" in situ, the module is removed to a nearby location for repair in a clean, well-lit and dust free environment, or replaced, so that it can be sent off-site for repair or replacement depending on the severity of the failure.

### Benefits of the Proposed Design Concepts

Key benefits of such a design will be a significant reduction in overall scale of plant, with resultant savings in earthworks, civils, structural steel, bulk fabrication and electrics, so reduced Capex; reduced construction time on site; the modules will be constructed in appropriate fabrication shops with skilled staff in an ergonomically effective and clean environment; and reduced operating labour, power and maintenance, so reduced Opex; but it will require increased engineering for successful implementation.

Note these are all common elements driving current coal preparation plant design, and the concepts presented in this paper simply represent an evolution of the current CHPP design trend. However to truly get the best out of this cassette approach there will need to be significant development on the interfacing and connections, and the development of appropriate wireless controls and remote drives will prove to be of immense value in this regard.

### Constraints to Implementing Proposed Design Concepts

When the current state of the art is considered, there are clearly a number of constraints in reducing the scale of process units, especially to standardised dimensions to allow for ready assembly using common lifting equipment. A key factor is that most of the space in a current CPP is to provide access to all parts of the process equipment for inspection, cleaning and maintenance. This increases the distance between adjacent units as well as above and below the unit, substantially increasing the total plant volume. If safe access by personnel wasn't required, process components, ancillary plumbing, electrics, controls and structural elements could all be located in close proximity, reducing both the footprint and the height of the overall plant.

Some process units are much more space efficient than others. For example, large dense medium cyclones processing coarse coal can process hundreds of tonnes per hour within a unit less than two metres in diameter and several metres long, which is a major improvement on the baths and jigs previously used for processing coarse coal, but ancillary processes including screens and magnetic separators still require substantial space. Fine processes are inherently less space efficient due to the need for finer particles to be processed in a relatively dilute environment to avoid entrainment of different particle types which reduces process efficiency. Hence focus should be on fine coal processes that retain high efficiencies with reduced processing volume that can operate efficiently at higher % feed solids such as agglomeration techniques (Galvin, 2015). Screens in particular take up a lot of CPP space, so use of DMC technology to process the full size range in a DMC without prior desliming can greatly reduce screening area (Yu and Zhao, 2013).

Of most importance is the proposal for reducing the module size without personnel access for inspection, cleaning and maintenance, but to still operate efficiently and continuously on a 24/7 basis for long periods. This requires reliable control of feed solids and slurry feedrates with no oversize, suitable wear and corrosion resistant materials in all contact areas with the solids, reliable sensors to be developed and installed to allow for constant monitoring and control of all stream flows (Firth et al, 2014), and smart sensors for condition monitoring of motors, etc.

It is not anticipated that the proposed vision will be accomplished in a single step, but will be accomplished as a result of a series of incremental, evolutionary steps over the coming decades. Each step will result in benefits in improved yields and reduction in costs. The purpose of this paper is to provide a focus on the aspects of coal processing design that require further development if as an industry we are to continue improving plant design along that same path.

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# Establishment and Operation of the Ravensworth North CHPP

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**Abstract**

The Ravensworth North Coal Handling and Preparation Plant Project was completed in late 2013 and it was designed to wash a wide range of coals. This paper describes the design and establishment of the facilities, in a difficult brownfields environment, and some learnings from subsequent operations.

Stage 1 of the project included modifications and debottlenecking of the existing coal preparation facilities and the addition of a new raw coal system including crushing and raw coal stacking/reclaiming which linked the existing Ravensworth Open Cut ROM to the Ravensworth Coal Terminal (RCT). In Stage 2 of the project an 1800 t/h, two module extension to the existing CPP was constructed, along with an additional product stockpile and tunnel reclaim system. The processes used at Ravensworth include dense medium cyclones for the -50+1.4 mm (ww) coal and spirals for the -1.4+ 0.15 mm coal. The remaining slimes are treated in refurbished high rate thickeners.

The plant has now been operating for two years and there have been a number of process and operational improvements.

Keywords: coal, coal preparation, Ravensworth North, CHPP design, gravity separation, dense medium cyclones, spirals, coal handling

## Introduction

Glencore has a large portfolio of mining operations around the world, and is the world's largest exporter of thermal coal, as well as a leading exporter of hard coking and semi-soft coking coal, producing over 130 Mt/y of saleable product. At the time of writing, Glencore operates 13 coal mine complexes in New South Wales and Queensland which include 12 open cut and eight underground mines.

The Ravensworth North mine is located 30 kilometres north-west of Singleton, NSW and has seen a number of owners, and uses, over its 40 year history. The original coal processing plant located at the current RCT site was constructed by the state government in 1983 to wash coal from the nearby Hebden and Ravensworth open cut mines.

The original CPP, which had been on care and maintenance since the 1980's, was refitted as a single module 1250 t/h plant by Resource Pacific to wash coal from a new longwall mine. The operation was known as Newpac, and the Newpac infrastructure upgrade project is outlined in Sorensen and Atkinson (2008). The Newpac operation was acquired by Xstrata Coal (now Glencore) in 2008. The close proximity of the site to Glencore's undeveloped Ravensworth North open cut resource provided the basis of the coal preparation and rail loading infrastructure that the open cut project required. The project proceeded in 2010 and was named the Ravensworth North Project (RNP). The CHPP upgrade was originally outlined by Walsh et al (2014) and this paper has been extended to include post commissioning experiences.

The operations at Ravensworth North can produce a range of products, both domestic and export including semi-soft coking (9% ash ad), domestic thermal and export thermal coal with ash values from 12 to 22% (ad).



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Xstrata elected to execute the project under an alliance agreement with AECOM and QCC Resources. The scope of work for QCC Resources involved the delivery of a nominal 1800 t/h CHPP including the upgrades to the raw coal system and sizing stations, two new CPP modules, a new product handling facility and upgrades to the reject and train load-out (TLO) infrastructure. An overall view of the CPP addition and product stockpile area is shown in Figure 1.

**Figure 1 Ravensworth preparation plant and product stockpiles**

**Plant Feed Envelope**

The coals washed are typical of the Ravensworth-Camberwell region, and have moderate to high levels of volatiles, generally low sulphur and phosphorous content, moderate to low ash values and reasonably low moisture content. The seams processed come from the Wittingham coal measures with Lemington, Pikes Gully, Arties and Liddell seams making up over 83% of the projected feed. Broonie, Bayswater and Barrett seams account for the remaining portion of the feed.

***Size distribution***

The plant feed size distributions were estimated from a series of 63 mm and large diameter (LD) bore cores. The bore core samples were subjected to the conventional drop-shatter, size reduction and wet tumbling tests to simulate the degradation that happens during real life processing. Figure 2 shows the resulting size distributions.

The feed coals at Ravensworth were expected to be many and varied, and this diversity required considerable flexibility in the plant design, and excess capacity was allowed for, particularly in the desliming screens, product drain and rinse screens and in the fines circuit.

***Washability***

The Ravensworth yield-ash design envelope is shown in Figure 3. As expected for a multi seam Hunter Valley operation, it is rather broad. Overall, the yield-ash curves showed there to be minimal near gravity material, so the greatest design issue became the sheer variations in feed quality. For some seams, expected yield was as little as 50% (ad) even if washing to a product ash value greater than 20% (ad).

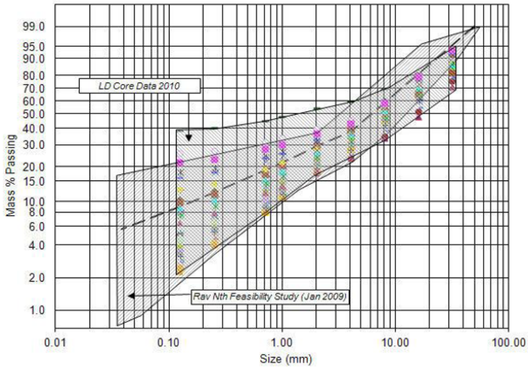


**Figure 2 Rosin-Rammler curves for the feasibility study and additional LD core data**

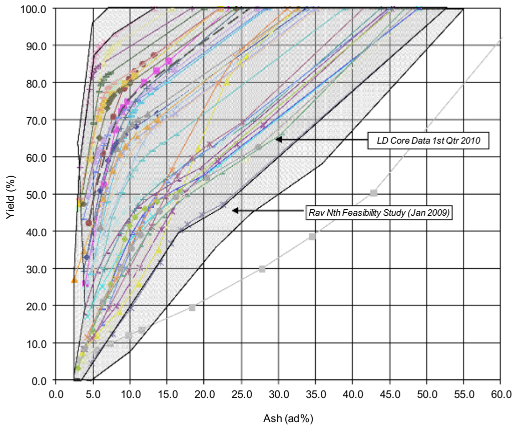


**Figure 3 Washability curve for the feasibility study and additional LD core data**

## Process Design for Ravensworth CPP

### *Process Selection and Simulation*

The process flow sheet design was developed using a Limn® simulator, and a simplified process block diagram of the CPP is shown in Figure 4. The new modules 2 and 3 each consist of a single stage dense medium cyclone (DMC) to process the +1.4 ww mm coarse coal fraction, single stage spiral processing of the -1.4 ww + 0.08 mm fine coal fraction (with the option of recycling the spirals middlings fraction), and the - 0.08 mm material being discarded to tailings.

The nameplate capacity of the two new modules is 1,800 t/h (900 t/h per module). The coal quality data indicate that up to 65 percent of the Ravensworth coal could be processed to produce a single product semi-soft coking or thermal product, with the remaining 35 percent processed as dual product.

Due to the high variability in mining operation and coal quality, the CPP was designed to operate in the following processing scenarios:

- Single stage processing with DMC operating cut-point approximately 1.70 RD to produce an export thermal coal product of greater than 9% ash (ad).
- Primary stage of a dual stage processing option with DMC operating cut-point approximately 1.35 RD to produce a semi-soft coking coal product of less than 9% ash (ad). In this instance the DMC reject from modules 2 and 3 is diverted to module 1 to extract middlings coal generally as a domestic thermal coal product.

The equipment list for the new equipment is given in Table 1.

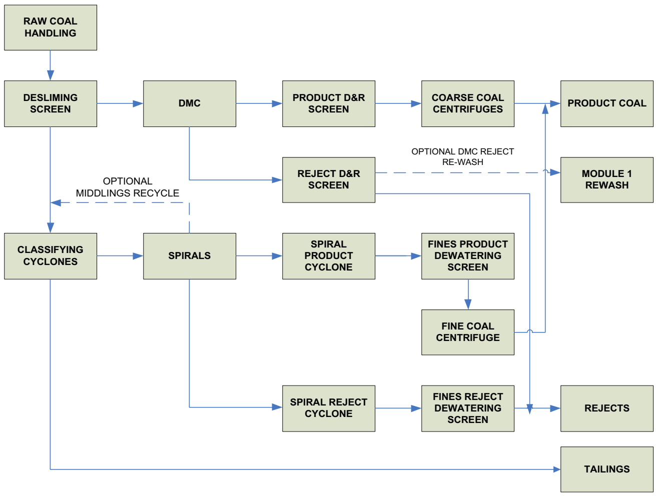


**Figure 4 Simplified CPP process diagram**

### *ROM Receival and Handling*

Raw coal from the Ravensworth open cut undergoes primary and secondary crushing using sizers to reduce the raw coal to nominal -200 mm, and is then conveyed and stacked onto a 200,000 tonne linear stockpile. Stockpiled coal is recovered by feeders and reclaim tunnel and directed to a roller screen / tertiary sizer combination for size reduction to 50 mm, then conveyed at up to 3,500 t/h to the raw coal bins situated at the CPP site.

### *Coal Preparation Plant*

The new CPP capacity consisted of two nearly identical modules with the process defined in Figure 4 and the equipment outlined in Table 1.

### *Tailings Circuit*

The original plant used three 27 m diameter thickeners and these were upgraded to handle all the CPP tailings streams. All three thickener underflows are pumped to a common tailings disposal sump. Three disposal lines then deliver the tailings to an existing tailings void via three 2 stage pump sets.

**Table 1 New equipment list for modules 2 and 3 Ravensworth North CHPP**

| Equipment | Quantity | Fabricator | Model/Size |
|---|---|---|---|
| **Raw coal handling** | | | |
| Secondary Sizers | 2 | Abon | 7/250HSCTD |
| Roller Screen | 1 | Abon | RS10R300-50 |
| Tertiary Sizer | 1 | Abon | 7/420HSSTD |
| Stacker | 1 | Downer | 1.4 m × 47 m boom |
| Tunnel Reclaim Feeders | 5 | Metso | VMO 40/27 |
| **Coarse coal** | | | |
| Dense Medium Cyclones | 4 | Minco | 1150 mm dia |
| Coarse Coal Centrifuges | 4 | Ludowici | VM1650 |
| DMC Feed Pumps | 4 | Weir | 12/10 GGH |
| Desliming Screens | 2 | Metso | 4.2 m × 7.3 m |
| DMC Product Screens | 2 | Metso | 4.2 m × 7.3 m |
| DMC Reject Screens | 2 | Metso | 2.4 m × 6.1 m |
| Magnetic Separators | 2 Parallel Drum | Steinert Sturton Gill | 1.2 m dia × 3.0 m |
| **Fine coal** | | | |
| Desliming Cyclones | 4 Clusters | Minco | 7 × 450 mm dia |
| Spiral Product Thickening Cyclones | 2 Clusters | Minco | 6 × 450 mm dia |
| Fine Coal Centrifuge | 6 | Ludowici | HFC1300 |
| Desliming Cyclone Feed Pumps | 4 | Weir | 300 FFL |
| Spiral Product Thickening Cyclone Pumps | 2 | Weir | 300 FFL |
| Spiral Reject Cyclone Feed Pump | 1 | Weir | 300 FFL |
| Spiral Reject Transfer Pump | 1 | Weir | 8/6 EAH |
| Spiral Reject Thickening Cyclones | 1 Cluster | Minco | 6 × 450 mm dia |
| Fines Area Floor Sump Pump | 1 | Weir | 100RVSP |
| Fine Product Dewatering Screens | 6 | Metso | 1.8 m × 3.0 m |
| Fine Reject Dewatering Screens | 2 | Metso | 1.8 m × 3.0 m |
| Spirals | 8 banks | Mineral Technologies | 7 × triple LD7 |
| **Reject and tailings handling** | | | |
| Reject Bin | 1 | Downer | 700 t |
| Tailings Thickener Mechanisms and Feedwells | 3 | Outotec | 27 m dia |
| Tailings Pumps | 3 x 2 stage | Weir | 8/6 SYFC-AH |
| **Product Handling** | | | |
| Reclaim Coal Valves | 12 | Halley & Mellowes | Carmen 8 GHD |
| Product Stacker | 1 | Downer | 1.4 m × 47 m boom |
| Sampling Systems | 6 | QHS | Various |

***Product Handling***

The product handling conveyor system was upgraded as part of stage 2 of the project. This involved extension of the existing module 1 product stacker rails and reclaim length to increase capacity of the existing stockpile to 400,000 tonnes, and construction of a duplicate product handling system with additional stockpile capacity of 600,000 tonnes. The existing TLO bin and reclaim systems were upgraded to a capacity of 4,500 t/h.

***Reject Handling***

The plant's original coarse reject system consisted of a conventional conveyor and bin arrangement followed by trucking to a void area some 7 km from the bin. The conveying system was extended and a new reject bin installed adjacent to the open cut raw coal dump station.

### Innovations of the Ravensworth North CHPP

Some of the challenges and innovations of the project included:

- designing the CPP expansion with tight constraints on three sides
- re-use, where feasible, of existing infrastructure, such as conveyors, gantries, tanks
- construction of module 2 and 3 around an operational reject conveyor
- design of a unique jack-up and rail roll-out system to provide sufficient maintenance access to module 1 DMC pumps
- a common (existing) feed conveyor for modules 2 and 3 located in the existing module 1 building with ability to feed module 2, module 3 or modules 2 and 3 simultaneously and evenly (The project elected to use a splitter and sluice pipe to each module.)
- two stage washing of feed utilising module 1 for re-wash
- a personnel elevator to enable emergency removal of a person on a stretcher
- a relatively complex tailings disposal system (three disposal lines with two stage pumping per line coming from three thickeners to ensure there was enough flexibility to handle all feed scenarios.

### Commissioning

The materials handling systems were progressively commissioned, with the CPP commissioning team commencing work in early August, 2013 and maintaining a presence on site until the end of November, 2013, when the guaranteed throughput and efficiency had been demonstrated. Subsequently the individual unit efficiencies were tested to confirm that process guarantees had been achieved.

### Plant Operations

The Ravensworth Underground Mine was placed on Care and Maintenance in Q3 2014 resulting in the original plant module (module 1) being placed in care and maintenance.

A crew of seven (six technicians and one supervisor) currently operate the CHPP on a 24 h/d, 7 d/w basis, with the two new modules operating consistently at the nominal feed rate of 1800 t/h. Feed rate is reduced when the raw coal bin 2002 is being used (throughput is limited to 1500 t/h with the original vibrating feeder). Plant maintenance is performed by contractors operating weekdays, with one major plant shutdown scheduled fortnightly. For the year to date period up to October 2015, the plant availability (2 modules) was 89.7%

Following commissioning and initial operations, a number of plant improvements have been identified and implemented, including the below.

#### *Raw Coal Handling:*

- Installation of fingers between rolls on the primary crushers and roller screen for treatment of high clay content coals to alleviate blockages between the discs and maintain capacity.
- Replacement of many conveyor transfer ultrasonic blocked chute devices with tilt switch and pressure plate probes for more reliable blocked chute detection.
- Installation of additional conveyor transfer blocked chute probes for more reliable blocked chute detection.
- Recommissioned ROM bin 2002 (at reduced plant feed rates) due to regular blockages in raw coal feed bin 2001 and clay build up on the vibrating feeder tray.

- Installation of combs on the tertiary sizer to remove clay build-up, eliminate “kebabbing” of the sizer rolls and to maintain throughput .
- Water addition spargers installed to ROM plant feed bins and rejects bin to assist with removing blockages.
- Water addition spargers installed to stockpile reclaim vibrating feeders to assist with removal of bridges/blocked feeders.
- Increasing the size of conveyor transfer chute access doors to improve the safety and efficiency of unblocking/cleaning chutes.
- Installation of additional mist sprays on the raw coal stacker for dust mitigation.
- Installation of mesh guards inside conveyor transfer chute access doors for compliance with AS1755 and AS4024.
- Modification of conveyor guards to facilitate easier cleaning while still complying with Australian Standards.

***Processing:***

- Redirected the fines product thickening cyclone overflow to the fines sump to reduce the recirculating clays in the fines plant and thus reduce product moisture.
- Overcoming feed mal-distribution to both product D&R screens in modules 2/3 through the use of tapered orifice plates inserted into the DMC overflow launders.
- Strengthening of the DMC pipe supports to reduce flexibility.
- Redirection of approximately 20% of the raw coal feed sluice water to the entry of the plant feed conveyor discharge chute to eliminate clay build up and blockages.
- Modification of the medium entry points at the discharge of the desliming screens to reduce desliming screen discharge chute blockages.
- Installation of oversize protection and blocked chute probes on the fines sumps’ trash screen overflow pipes.
- D&R screen spray modifications to reduce magnetite consumption.
- Removal of the polycarbonate wall sheeting to increase air flow and reduce the humidity inside the plant building.

***Product Handling:***

- Clean coal transfer tower modification to allow coal from the new modules to be stacked on the original product pad (formerly for underground product only), as outlined below.

***Increased product coal types:***

There has been a number of product quality differences identified (compared to the original plan) as different seams have been processed by the plant, and this has resulted in the requirement to process and separately stockpile additional product coal types. There can be up to seven coal types stacked on the product stockpile at any one time, and there are nine coal valves with 40 m between them, meaning that some coal types are limited to one valve. In this case, the maximum stacker height is restricted to maintain coal type segregation. This segregation has resulted in a significantly reduced stockpile capacity.

To increase the total product coal stockpile capacity, modifications have been performed to the clean coal transfer tower to allow product from the new plant to be stacked on the original underground coal stockpile (Pad 1). The capacity of Pad 1 infrastructure is limited to 1000 t/h and therefore, to maintain plant feed rate, only seams with a yield of 55% or less are transferred to this pad.

**Acknowledgements**

The project was delivered under a three way alliance contract model with the project team formed from personnel drawn from Glencore, Downer EDI Engineering/QCC and AECOM. The project involved considerable brownfields integration of legacy installations from several past projects with engineering histories spanning thirty years. Assimilating generations of engineering information, standards and expectations into a cohesive and successful project is a testament to the many participants over the project's recent history.

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# Coal Beneficiation: Initiatives Taken By Mahanadi Coalfields Limited (a subsidiary of Coal India Limited)

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**Abstract**

Power or electricity is one of the most critical components of infrastructure affecting economic growth and well-being of nations. The existence and development of adequate infrastructure is essential for sustained growth of the Indian economy. The Indian power sector is one of the most diversified in the world. Coal accounts for about 61% of India's total installed capacity and 72 % of the electricity generated is from coal based power plants. Demand of coal is increasing year after year on account of increasing demand from power sector. Coal India Limited (CIL), single largest coal producer in the world accounts for about 80% of India's Coal production. Coal India is planning to ramp up coal production to one billion tonne by 2019-20, in which share of Mahanadi Coalfields Limited (MCL) is 250 million tonnes ie 25% of CIL's targeted coal production.Augmentation of coal production by CIL also requires matching coal washing capacity for improving the quality of coal to meet the requirement of user industries as well to comply the environment protection Rule 2014 of Government of India.

This paper illustrates the initiatives taken by CIL in general & MCL in particular in setting up of Non-Coking Coal washeries realising the need of Coal Beneficiation and also the steps towards Good Governance adopted by MCL for bringing transparency in tendering process for awarding contracts for setting up of washery which is very complex in nature, through e-bidding route with a system of auto evaluation of Bids for making the entire process simpler, transparent and less time consuming.

**Key words**: Coal India Limited, MCL, BOM, Beneficiation, e-bidding, auto evaluation, auto refund

## 1.0 Introduction

Achieving energy security is of fundamental importance for India's economic growth. Since thermal power is still has occupied a cardinal place in the power sector contributing about 69 % of the total installed capacity of power generation and coal being the most important source of thermal energy, the quality of coal particularly in Indian context where majority of coal reserve is with high ash content, is of great concern.

MCL being the second largest coal company of India is more concerned for beneficiation of coal due to a number of reasons starting from statutory provisions related to Environment, quality of coal, the profitability in the business and for a sustainable development. Recently, MCL has taken a number of initiatives for expeditious execution of Coal Beneficiation Plants which shall be exemplary in nature and shall provide a roadmap to other coal companies with similar problem.

## 2.0 Indian Power Scenario

The Indian power sector is one of the most diversified sector in the world. Sources for power generation range from conventional ones such as coal, lignite, natural gas, oil, hydro and nuclear power to other viable non-conventional sources such as wind, solar and agricultural and domestic waste. Out of the total installed capacity of 272.687 GW,165.236 GW is of Coal based power plants, which is 61% of installed capacity (as of end of April 2015).

 
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India's energy needs are fast expanding with increased industrialisation and capacity addition in Power generation. This is where 'Coal' steps in. India now ranks third (after China & USA) amongst the coal producing countries in the world. The development of core infrastructure sectors like power, steel, and cement are dependent on coal. About 80% of the coal in the country is consumed in the power sector and as the demand for electricity is expected to rise dramatically over the next decade, Coal will continue to be the dominant energy source in the foreseeable future.

## 3.0 Indian Coal: Characteristics, Reserves & Coal Production.

### 3.1 Indian Coal characteristics

Gondwana seams in India are proof of drift origin. The Gondwana coals are largely confined to the river valleys like the Damodar, Mahanadi, Godavari etc. Indian coal is characterized by its high ash content (~40 to 45%) and low calorific value.

### 3.2 Coal Reserves of India

The Coal reserves of India up to the depth of 1200 meters have been estimated by the Geological Survey of India (GSI) at 301.56 Billion tonnes as on 01.04.2014. Coal is located mainly in the central and eastern part of the country.

### 3.3 Coal Production Scenario in India

CIL, the country's largest coal company & the world's single largest coal producing company accounts for over 80 percent of India's coal sector.

**Table-A**: **Coal Production in India (Million Tonnes)**

| Production of Coal | | | | |
|---|---|---|---|---|
| Company | 2013-14 | | 2014-15 | |
| | Target | Actual | Target | Actual |
| CIL | 482.00 | 462.41 | 507.00 | 494.23 |
| Others | 122.55 | 103.35 | 123.25 | 120.81 |
| Total Coal production | 604.55 | 565.76 | 630.25 | 615.04 |

Non coking coal constitutes about 90 % of total coal produced by Coal India.

## 4.0 Coal Beneficiation: A necessity for Indian Coal

India's Ministry of Environment & Forests(MoEF) has made it mandatory through a gazette notification of 1997 with subsequent amendments to use beneficiated coals whose ash content has been reduced to 34% (or lower) in power plants in urban, ecologically sensitive and other critically polluted areas.

In view of India's significant rely on coal for electricity generation & due to the high ash content in Indian Non-coking coal (i.e. power coal) and MoEF Gazette Notification, Coal beneficiation of Power coal in particular is inevitable.

### 4.1 Coal Beneficiation: Initiatives taken by Coal India Limited (CIL)

Realising the importance of Non-coking coal washing, Coal India Limited has planned to set up of number of Non-coking coal Washeries in its different subsidiaries to supply the required quality of Coal opening big opportunities for both Indian and foreign players. India depends primarily on Open Cast mining (90%) to meet the growing demand. In response to the regulation promulgated by MoEF in 1997, Coal India Limited had begun a massive plan to set up number of washeries on Build-Operate-Maintain (BOM) concept, where the selected bidder has to install the washery and operate & maintain the same for initial period of 10 years with a provision of further extension of 5 years subsequently. As a long term strategy, CIL is planning to supply washed coal to consumers located away from pit heads and alongside also develop big opencast mines of 2.5 MTPA capacity and above with integrated Washeries.

A Model Bid Document for installation of washery under BOM concept was formulated and subsequently approved by Coal India in 2008 with a purpose that the subsidiaries of CIL would customize the model document as per the requirement to suit the ground conditions in each case for Global bidding.

**5.0 Coal washing : Mahanadi Coalfields Limited - Current & Future Scenario**

To comply the MoEF Notification, as on date, MCL has identified to set up 8 numbers of Washeries of 130.0 MTPA in different phases in its command area either on BOM concept or on Turnkey concept, details of which are tabulated as under.

**Table-B: Washery of MCL Under Different Phases**

| Sl.No | Name of Washery | Capacity (MTPA) | Phase & Concept | Remark |
|---|---|---|---|---|
| i | Hingula Washery | 10.0 | 1st Phase & Under BOM Concept | Advanced stage of finalisation of tenders. |
| ii | Basundhara Washery | 10.0 | | |
| iii | Jagannath Washery | 10.0 | | |
| iv | Ib-valley Washery (Lakhanpur) | 10.0 | | |
| v | Lakhanpur Washery | 20.0 | 2nd Phase & Under Turnkey Concept | Scheme is under preparation |
| vi | Siarmal Washery | 40.0 | Beyond 2nd phase & Under Turnkey Concept | In planning phase |
| vii | Garjanbahal Washery | 10.0 | | |
| viii | Kaniha Washery | 20.0 | | |
| | **TOTAL CAPACITY** | **130.0** | | |

Of these, tender for Hingula Washery and Basundhara Washery were floated in conventional (manual) mode.To avoid the prolonged time consumed in the manual bidding, bidding of remaining two Washeries (Jagannath Washery & Ib-valley Washery) are done in e-tender mode, wherein Lowest Bidder for both the Washeries are identified within 3 months right from the floating of bid.

**6.0 Expediting Execution of Washery Projects in MCL**

Coal beneficiation requires a number of washeries to be constructed in a planned way within a definite timeframe. The gestation period for such projects is normally high due to following main reasons:

a. Non availability of readily available suitable land for washery projects
b. Delays in Environmental clearances of washery projects
c. Inbuilt complexities in Public Procurement Process for washery projects

The first two issues are being taken care by MCL by earmarking land for washeries in advance and expediting the EC with the help of MOC. At present since the technology is decided only after selection of winning bidder in the public auction (under BOM Concept) and the process of EC is initiated thereafter, which makes these two activities in series and time consuming. In future MCL is planning to fix the technology first, so that the process of EC can run parallel to the bidding process.

**6.1 Issues in Contract Finalisation**

For installation of Washeries, MCL has to engage outsourced agencies and sometimes outsourcing is also resorted for running & maintenance of washeries. Since MCL is a Public Sector, hence to award such procurement contracts, the system of public auction through open bidding is resorted. Different contractual models like Turnkey Contracts Build-Own-Operate (BOO) model, Build-Operate-Maintain

(BOM) model and different bidding systems like a single stage two part system , two stage (RFQ and RFP with two parts) have been experimented. However, in each mode of bidding and each model of contract, a number of issues related to transparency, equal opportunity to the bidders and more prominently the delays in the procurement cycle are being faced leading to ultimately delay in the actual process of coal beneficiation to start.

### 6.2 Adoption of E-Tendering mode by MCL for Expeditious Execution of Coal Beneficiation Projects

In order to take care of such issues, MCL has adopted a method of information based auto evaluation online system of e-Procurement complying guiding principles of public procurement. Since e-Procurement throughout the world is in developing stage (including developed countries) and degree of automation of different processes of procurement are not very high, adoption of a system which eliminates the subjectivity due to human interventions is an innovative concept and it has given a very good encouraging outcome in MCL. This information based evaluation system was customised for the washery construction tenders under BOM model, wherein the bid evaluation process was very complex.

Various factors, such as set-up cost, washing charge, cost of water & power consumption & impact of yield percentage on the quoted price, quotations in different units & currencies for a long period of 11.5 years(1.5 Years for Construction period and 10 years for initial maintenance and Operation) by applying Long Range Marginal Cost (LRMC) & Discounted Cash Flow (DCF) method for arriving the notional cash outflow to decide the financial impacts over a period of time to identify the most competitive bid etc. are to be considered during bid evaluation. In conventional manual system of bidding, this was a very cumbersome post tender exercise consuming too much time & other recourses and sometimes attracting different questions from different stakeholders regarding the subjectivity of the evaluation process creating disputes.

To overcome such complex issues, a system of online information based auto evaluation system enriched with pre-defined business logic, proper controls over input type and with alert messages was developed by MCL in association with CMPDI (a sister concern of MCL under CIL for in-house consultancy) and the bidders were facilitated during bid submission to submit all the relevant information in pre-defined format. The business logic of evaluation of bids was in-built in the portal software to decide the Lowest Bidder immediately after opening Price Bid, thus making the entire process simpler, faster and transparent.

The model was developed in-house by customizing the Government e-Procurement System of National Informatics Centre, Govt of India. The salient features of the system are as follows:

#### 6.2.1 Technical/Financial Qualification Requirement

i. A customized Technical Parameter Sheet (TPS) with in-built business logic for seeking the information from the bidders for Technical Capability w.r.t. Set up Qualification Requirement and Operating Qualification Requirement and Financial Capability w.r.t. turnover & availability of working capital of the bidder in desired format is captured and evaluated by the system
ii. All the bidders have to comply the TPS for technical qualification for opening of Price Bid.
iii. The details of the Profile of the bidder are captured and the salient requirements regarding eligibility are evaluated online with giving alert messages to the bidders.

#### 6.2.2 Financial Evaluation of Price Bids

**i.** The financial evaluation of bids are done by portal by evaluating submitted Price bid which contains all the salient information in a structured manner with all details required for deciding the total financial impact over the period of construction of the washery and thereafter during operation and maintenance for next ten years.

**ii.** Price bids of all the bidders, qualified in Part-1 evaluation, financially evaluated and Lowest bidder is identified immediately after opening Price Bid online based on the lowest Impact Value i.e the Notional Cash Outflow.

**iii.** After identification of Lowest Bidder, Confirmatory documents are to be uploaded online by Lowest Bidder within stipulated time i.e. 30 days as per Bid Document.

**iv.** Examination/Scrutiny of the uploaded confirmatory documents of bidder for confirmation of all the information/declarations furnished by the bidder online.

**v.** The bidder, after confirmation of all the information/declarations furnished online, will be eligible for issue of 'Letter of Intimation (LoI)' in line with the provision of Bid Document.

### 6.2.3 The Flow Chart of the System

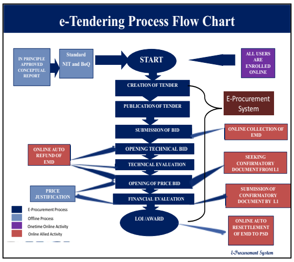


Fig1: Generic Flow Fig 2: Auto Evaluation Fig 3: Auto Refund of EMD

**6.2.4 Salient Features of the system:** In the above model the following features are in built:

**i.** The evaluation of the Bids is done by the portal software equipped with logic required for evaluation based on the data provided by the Bidder with least human intervention

**ii.** Bidders are not required to submit of any document off-line for evaluation of their Bid except in case of Bank Guarantees.

**iii.** The Bidders can online view the bid opening events live remotely, thus saving the time and other resources.

**iv.** Only Lowest Bidder has to upload scanned copy of Original documents in support of the information furnished by them (Bidder) online.

**v.** In order to automate the system of receipt of EMD (Earnest Money Deposit or Bid Security), a system of online receipt of same had been started through Electronic Fund Transfer/NEFT/RTGS mode for all the Bids floated in MCL. The online EMD management consist of three distinct processes:

a. Online Collection of EMD in a pool account
b. Online Auto Refund of EMD to unsuccessful bidders
c. Online Resettlement of EMD into Performance Security for successful bidder

All these processes are automated with a business logic eliminating the human intervention and facilitating the bidders to pay online and to receive the refunds automatically at an early date in T+1 date basis.

**vi.** Bidders are not required to visit MCL premises before start of Work.

**6.2.5 The outcomes of the system:** Some of the outcomes are as follows which are very encouraging**:**

**i.** Enhanced Competition due to ease in participation in the bidding process in online mode on 24X7 basis globally from anywhere.

**ii.** Reduction in cost of bidding process for the department and bidder both due to less stationary cost, reduced travel expenses and publication cost.

**iii.** Reduction in Cycle time of Procurement (at present it is 89 days from the end date of Bid Submission. In case of washery projects it has reduced from 1095 days to 100 days).

**iv.** Transparency in the bidding process leading to reduction in complaints from all the stakeholders and increased level of trust on the evaluation process which has created a good image of the company in the society.

**v.** Ease in bid creation and elimination of clerical errors and uniformity of the bidding process.

**vi.** Increased Confidence level of executives in decision making, thus motivating them to work, work more and work better.

## 7.0 Conclusion

Coal beneficiation in India is being planned in a big way due to more than one reason. Out of these reasons sustainability and commitment to the society for a clean fuel and a green environment is of prime concern for MCL being the second largest coal company in India. It has initiated the process of enhancing washing capacities by addressing all the salient issues like use of rejects, availability of land, environmental clearances and most prominently to use a faster and more efficient mode of electronic procurement aided with automated information based evaluation process which has been conceived, developed and implemented by MCL. Leveraging technology for bringing efficiency is an age old practice; however, inventing innovative methods for combating challenges of the changing environs is one of the strategies to remain fittest for growth and survival. MCL is trying to expedite the process of coal beneficiation with such strategy in a big way.

***The views expressed in this technical paper are of the authors, not necessarily of the organization she/he is working.***

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# Dense Medium Baths and Drum Separators
# A Re-Evaluation of Their Role in Modern Coal Preparation Plants

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**Abstract**:

In recent years Coal Preparation Plant designers globally are increasingly considering large diameter Dense Medium Cyclones (DMC) for treating the full size range of feed, in preference to a combination of open Bath or Drum Dense Medium Separators (DMB) and DMC's. This has also coincided with the general trend of curtailing the DMC process at a bottom size of 1 or 2mm, and using water based gravity processes, such as Spirals or Upward Current Separators, to process the minus 1 or 2mm fraction on which the performance of the very large diameter DMC's starts to decline.

The main perceived advantage of this preference, for DMC's only, is plant design simplicity, by removing a sizing stage between the two Dense Medium processes and reducing the number of moving parts.

However, there are several reasons why use of a circuit, which contains both a Large Coal open bath and DMC's, is more efficient with reduced operating costs when compared with an "all DMC" circuit.

**Key Words**: Dense Medium Bath, Dense Medium Cyclone, Medium Circulation, Magnetite costs, Power Costs, WEMCO Drum, Wing Tank system.

## 1.0 Coal Preparation and Dense Medium Baths - A Brief History

Mechanised Coal preparation began in the 19th Century with the introduction of simple Coal Jigs, where a box containing a bed of Coal on a perforated steel bedplate was moved mechanically upwards and downwards. Jig technology was revolutionised in 1892 by the introduction of the Baum Jig, which used a static box with a perforated bedplate, and the jigging motion was created by the intermittent introduction of compressed air in a U–tube arrangement (1).

Dense Medium Separation, using a suspension of fine dense solids in water to simulate a heavy liquid, had been patented in 1858 by Sir Henry Bessemer, but was only practically introduced in the early 20th Century. The Chance Cone (2), an American invention using sand as a medium solid, was patented in 1917, and was initially deployed in Pennsylvania on Anthracite.

Gradually Dense Medium Bath technology developed to large capacity units, and different heavy minerals were used for the medium solids such as Calcium Chloride (3) and Barytes. The Barvoys Bath, widely used in Europe the 1940's, originally used Barytes as a medium solid, was the basis of the first Coal Preparation Plants built in both Australia and South Africa in the early 1950's.

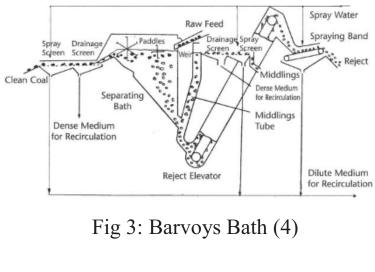


Fig 3: Barvoys Bath (4)

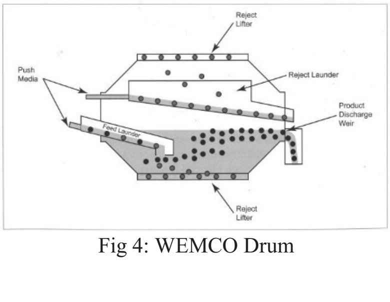


Fig 4: WEMCO Drum

 
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Magnetite was first used as a medium in 1937, in a vessel designed by the distinguished Dutch Coal Preparation engineer K F Tromp, for his original deep bath.

Between 1940 and 1960 Magnetite became the preferred Dense Medium solid and many types of shallower Bath were developed to avoid medium segregation. These were units such as the Daniels Bath in the USA (5), the Tromp Vertical Wheel Bath in Holland, the Teska Vessel in Germany (Humboldt Wedag), the Drewboy Bath in France (Venot PIC) and the Norwalt Vessel in South Africa (Nortons Tividale). In all these designs of Bath, and many other variations, the sinks product is removed by a chain conveyor, or an inclined or vertical wheel.

In 1950 WEMCO from Sacramento in California introduced the WEMCO Drum Separator, a unique design of shallow open Bath, in which the Clean Coal and Discard were separated in a revolving Drum. The Drum is unique because it is the only vessel where there are no mechanical moving parts immersed in the Dense Medium, thus reducing wear.

## 2.0 Dense Medium Bath (DMB) and Large Diameter Dense Medium Cyclones (DMC). A Comparison of Capital Costs, Performance and Operating Costs of the Various Combinations

### 2.1 Feed Size Range and Capacity

The Dense Medium Bath or Drum (DMB) can handle large sized Raw Coal up to 300mm in diameter, although it is more normal that the feed top size is reduced to 150mm. The bottom size, which is usually fed to the DMB, is 6mm. More commonly 13mm or 25mm is the bottom size when combined with a DMC plant.

With a conventional DMC, the maximum particle that can be handled is 1/3 x the diameter of the feed inlet. This is generally considered to be a top size of 50mm.

Cylindrical centrifugal separators, such as the Larcodems or GT Two–product separators, where the feed is delivered separately from the medium at atmospheric pressure into the air core, can handle particles up to 150mm diameter (6).

The advantage of being able to handle a larger feed size means that the Raw Coal crushing requirements from a typical ROM feed of 1200mm top size for a plant with a DMB, can be reduced to a maximum of two stages, compared to 3 for plants using DMC's only.

If the DM Drum Clean Coal product is required to be ultimately crushed to meet a product requirement (~50mm), then this will take less power, as only the lighter and generally softer Clean Coal will need to be crushed.

One other significant factor with a plant containing a Dense Medium Bath, is that the oversize protection for Cyclone feed pumps is not required and tramp metal can more easily be dealt with.

### 2.2 Separation Efficiency

Separation efficiency of the DMB in the +50mm size range is highly accurate, approaching lab Float/Sink test accuracy if properly designed. With the Separation efficiency in the size range 50mm to 12.5mm, where both DMB and DMC can overlap, performance is comparable. For raw feed minus 12.5mm, Bath/Drum pool area required greatly increases, making the DMC the best practical choice.

For a plant with both DMB and DMC, the DMC's can be of a smaller diameter than in an all DMC plant, as the top size feed to DMC is much smaller. Consequently, the efficiency in the DMC's for the 2 mm x 1 mm fraction will be improved, when compared with very large diameter DMC's.

### 2.3 **System Flexibility**

The flexibility of a DMB/DMC plant is greater because of the ability to tailor optimum cut point, in large and small Coal sections, because of different washability characteristics in coarse and small particle size ranges.

### 2.4 **Volumetric Medium Circulation Comparison**

There are significant differences in Medium circulation requirements for a DMB/DMC combination design, with a DMC only design.

In order to compare Medium circulation requirements, we must first look at the different systems for the feed to DMC's, which have a significant effect on the amount of medium which needs to be pumped. Figures 5 and 6 illustrate the main types of pump feed arrangement to DMC's.

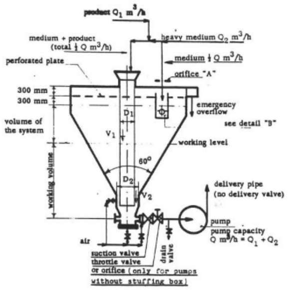


Fig 5: "Centre Tube" DMC Pump Feed System (DSM)

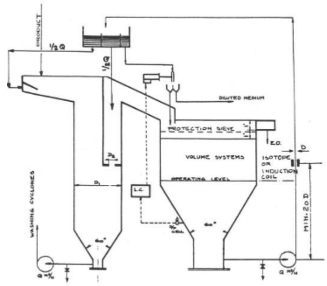


Fig 6: Twin Tank DMC Pump Feed System (DSM)

Pump feeding Cyclones is now almost universally adopted, since the improvement in the efficiency of modern Centrifugal Slurry Pumps. The original Dutch State Mines design criteria was a Solids Feed/Dense Medium volume ratio of ~1:2.8.

In recent years DMC technology has been researched extensively, the geometry of the DMC has changed, with the original Tangential inlet being replaced by the Involute feed inlet, and enlargement of the feed inlet and overflow vortex finder. This in turn allows a greater volumetric throughput of the DMC. The reason for the increased flow is that it has been recognised that a higher Solids to Medium volumetric ratio is often desirable to optimise separation performance, particularly in the case where there is a large amount of near gravity material in the feed.

For DMC's, raw Coal Solids to Dense Medium volumetric ratios are now usually between 1:4 and 1:6. This increase has obvious implications for the medium circulation and DMC pump feed system.

These systems have changed little since introduction by DSM in 1945 (these figures are actually reprinted here from the original DSM design manual) (7).

In the first system the raw feed is directed to a feed tube located inside the Cyclone Feed Sump. The idea of the tube is to direct the raw feed solids directly to the pump, and avoid the escape of feed solids into the main body of the medium feed tank.

There are some disadvantages of this design. Firstly the gap between the centre feed tube and the wall of the feed tank is a figure calculated to avoid feed solids escaping into the main body of the tank. This is subject to a very high wear rate. Secondly, the measurement of medium density by a nucleonic density gauge, for control purposes, cannot be undertaken accurately in the rising pump feed line, because of the inclusion of the raw coal. Consequently, the measurement has to be undertaken after collection of drained Dense Medium on its re-circulating path back to the DMC Feed Tank.

Given the perceived problems with the Centre Tube system, a more common design now employed in many plants is shown in Figure 6. This "Twin Pump" system, which was originally designed by DSM for other minerals, uses two pumps rather than one. The feed is delivered to a "Wing Tank" and mixed with the dense medium, before pumping to the DMC. The required medium is distributed to the feed chute to flush in the raw feed, and the remaining medium is added through an orifice tube within the wing tank. This ensures that only medium, rather than medium plus solids, overflows to the correct medium stock tank, when the raw feed displaces dense medium as it enters the wing tank.

The second pump is used to collect recirculating medium and pump it to a constant head tank, from where it is distributed as required. The nucleonic density gauge can be mounted on the rising pump line free of raw feed, and the density can therefore be accurately measured and controlled.

The Twin Pump system has proved to be the best system for large modern plants, particularly as the top size of feed solids being pumped has increased from 13mm up to 50mm.

If we accept that the Twin Pump system is desirable for a DMC-only flowsheet, and the newer large diameter DMC's are working at a Raw Feed Solids to Medium volumetric ratio of 1:4 upwards, the Twin pump feed system is effectively increasing the ratio to a minimum of 1:8, and in some cases to 1:12.

Table 1 compares the total circulating medium requirements of a 1,000 TPH, two product Coal Preparation Plant, with a fines (-1mm) capacity of 100 TPH, requiring a 900 TPH Dense Medium Section. Figures 7 and 8 show the simplified flowsheets of the two systems being compared.

Option a) is a DMB/ DMC plant treating 150mm x 13mm in a bath/drum and minus 13mm x 1.0mm in the DMC section.

Option b) is an all DMC circuit treating Coal crushed to 50mm top size down to 1mm.

For the DMB/DMC case it is assumed 450tph is +13mm, and 450tph 13mm x 1.0mm.

A feed mean SG of 1.6 is assumed, making the volume of feed solids 900/1.6 = 562.5 $m^3$/hr.

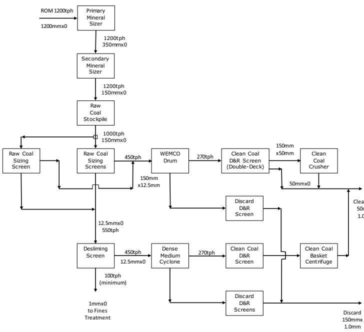


Figure 7: a) Drum/DMC Flow Diagram

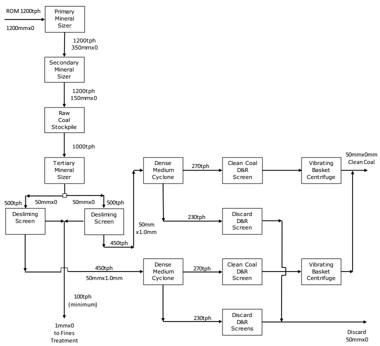


Figure 8: b) DMC – only Flow Diagram

**Table 1**
**Comparison of Medium Circulation Requirements**

| **Option a) - DMB/DMC** | | **Option b) - DMC Only** | |
|---|---|---|---|
| DMB Separating Medium Required ($450/1.6 \times 4 = 1{,}125 m^3/hr$) | $1{,}125 m^3/hr$ | DMC Separating Medium ($900/1.6 = 562.5 m^3/hr \times 4 = 2{,}250 m^3/hr$) | $2{,}250 m^3/hr$ |
| DMB Raw Coal/Medium V-ratio* | 1:4 | Raw Coal/Medium V-ratio^ | 1:4 |
| Total Drum Circulating Pump Capacity | $1{,}125 m^3/hr$ | DMC Feed Pump Capacity (4x $562.5 M^3/Hr$ + 562.5M3/Hr solids) | $2{,}812.5 m^3/hr$ |
| DMC separating Medium (450/1.6 = 281.25 x 4 =1,125 M3/Hr) | $1{,}125 m^3/hr$ | DMC Circulating Medium Pump | $2{,}812.5 m^3/hr$ |
| DMC Raw Coal/Medium Ratio^ | 1:4 | | |
| DMC Feed Pump Capacity (4 281.25 + 281.25 solids | $1{,}406.25 m^3/hr$ | | |
| DMC Circulating Medium pump Capacity | $1{,}406.25 m^3/hr$ | | |
| **Total Pump Capacity DMB/DMC** | **$3{,}972.5 m^3/hr$** | **Total Pump Capacity DMC's** | **$5{,}625 m^3/hr$** |

Raw Coal mean S.G. is assumed to be 1.6 for this comparison.
If a Three product Separation is required the difference is increased further to:-
Total pump Capacity DMB/DMC: **$5{,}723 m^3/hr$** Total pump Capacity DMC: **$8{,}272 m^3/hr$**
* Highest Large Coal system medium ratio. Actual figures range from 1:2 for a Drewboy Bath (8) to 1:4 for a Daniels Bath or WEMCO Drum.

If a higher Raw Solids feed to Medium ratio is used for the DMC's (>1:4), the increase in medium circulation and power consumption required for the two options will escalate to approximately 2 x for the DMC-only option.

## 2.5 Capital Costs of DMB/DMC Plant versus DMC-Only Plant

The Capital Cost for the Process equipment for Option a) DMB/DMC is **$4,346,968** (9)
The Capital Cost for the Process equipment for Option b) DMC's only is **$5,052, 930** (9)

## 2.6 Operating Costs

Using the above flowsheets and equipment lists, the operating cost difference is basically the difference in Power consumption and Magnetite consumption. Plant labour is considered to be equal for both schemes.

### 2.6.1 Magnetite Consumption

Magnetite consumption is a function of surface area of the coal products to be drained and rinsed, and the efficiency of the Magnetic Separators.

If we assume the efficiency of the Magnetic Separators is the same, the loss for the DMC-only plant will be slightly higher, due to the increase in surface area of the feed as it is crushed to a top size of 50mm, suitable for pumping to the DMC's.

For the DMC-only plant treating 50mm x 0, the magnetite consumption is estimated to be 0.75kg/t of Raw Coal received into the Dense Medium Section. This is a good average figure of efficiently run plants of this type globally. Using an annual operating time of 6,500 hrs, the magnetite consumption will be 6,500 x 900 x 0.75 = 4,387 tonnes per annum (TPA).

For the Drum/Cyclone plant treating 450 TPH of 150 mm x 12.5 mm feed in the Drum, the magnetite consumption of the very coarse Coal 150mm x 50mm will be in the order of 0.25kg/t as the surface area of this fraction is very low. If we then consider the example above where 150 TPH of the raw feed is 150mm x 50mm, the loss of magnetite for this portion of the feed is:- 6,500 x 150 x 0.25 = 243.75 TPA

The 300 TPH feed to the Drum plus the 450 TPH feed to the DMC in option a) will have a consumption similar to the all DMC plant, ie 0.75kg/t received into the plant.

- 6,500 x 750 x 0.75 = 3,656.25 TPA
- Total predicted Magnetite consumption Option a) Drum/DMC plant **= 3,900 TPA**
- Total predicted Magnetite Consumption Option b) DMC-only plant = **4,387 TPA**
- The cost of the difference is 487x$275/t (USA price 2015)=$133,925 USD/annum

### 2.6.2 Electrical Power Consumption

The total installed Power in kW of the Dense Medium sections of the two systems being compared is:-
Option A) Drum/DMC plant: **1,922.75.kW** Option B) DMC–only plant: **2,763.5 kW**

Assuming an absorbed power factor of 75% the DMC–only plant will consume **630.6 kW** more per hour of operation.
Assuming 6,500 hrs operation a year and a Cost of Electrical Power of $0.10/kW (10), the DMC–only plant will cost an extra $409,865 in electrical power per annum. <u>This difference would be escalated further if the chosen DMC coal to medium ratio is higher than 1.4, or a 3 product plant was considered.</u>

## 3.0 Conclusions

As can be seen from the presented data, the combination of a Large Coal Separator treating the coarser sizes, combined with a Dense Medium Cyclone, is both economically and performance wise, superior to a plant employing DMC's only. **For every Raw Feed Tonne which can be processed in a Dense Medium Bath, which instead is processed in a Dense Medium Cyclone, between 2 and 3 x the circulating Medium is required.**
In certain instances the crushing of Raw Coal can be advantageous for liberation of saleable values, and in these situations the case for using DMC-only circuits is more logical.
Despite its long history, the large Coal separating Bath or Drum still has a vital role in Coal Preparation plants of the future.

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# Development Trend of China's Coal Preparation Industry In Critical Process and Equipment

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**Abstract:** This paper gives a brief introduction about the current status of China's coal preparation industry in the context of new environmental requirements. By analyzing the current development status of the development in coal preparation process and equipment, the paper points out the key technologies and critical equipment needs to be developed for the industry, and the breakthrough in developing the large-scale and reliable critical equipment, automation technology with independent intellectual property, perfecting coal preparation process and environment-friendly coal preparation technologies, breaking through the key technical bottleneck, enhancing core competitiveness and solving the critical problems of the industry. In addition, the paper forecasts that the future trends of coal preparation industry in China of upsizing scale, higher efficiency, intelligentization, environmental protection and cleanness.

The paper also analyzes the major problems and contradictions existing in China's coal preparation industry, and gives directions to optimize the utilization of coal resources and increase the clean and adaptation degree of coal product to steadily attain sustainable development and step into an age of "green coal preparation industry".

**Key words:** Coal preparation; key technology; critical equipment; large-scaling; development trend; clean and adaptation degree of coal; green coal preparation industry

**Foreword**

China is the largest coal producer and consumer in the world, accounting for 67.5% of China's primary energy consumption. As per the relative statistics, the global total coal production was 7.9 billion ton, nearly half of which, 3.87 billion ton, was from China. With increasing economic growth, China will increase its coal production. As per the forecasts of several national coal science institutes, the consumption peak of coal in China will increase to between 4.5 and 4.8 billion ton.

In the past, the purpose of China's coal industry was to meet the energy consumption of China. As such, the production and utility of coal was too extensive with problems in composition, environment and safety etc. Mr. Liu Jiongtian, a national academician put forward the concept of "clean and adaptation degree" to evaluate the cleanness of coal by determining the ash content, moisture and sulphur content in the coal, and the degree to meet the requirement of the coal-consuming equipment (rational allocation degree). Currently, the average clean and adaptation degree of China's coal product is 25%, compared to that of the USA which is 60%（Wang Jieqiong,2013）. As the pollution index of Beijing-Tianjin-Hebei region is increasingly worsening, the issue of optimizing the utilization of the coal resource, increasing the clean and adaptation degree of coal and cleanness of coal utilization has become a priority.

As per the requirement of National Energy Administration, the preparation rate of coal will be over 80% in 2020, 15% higher than that in 2015. In the light of National Energy Strategy and Action Plan, the overall coal consumption in China will be confined to 4.2 billion ton, of which 3.36 billion ton will be prepared, 800 million ton higher than that in 2015.

The increase in coal preparation volume brings about new opportunities and challenges to China's coal industry. As such, it is increasingly important to further develop coal preparation technologies and critical equipment of higher performance, higher reliability, higher "intelligentialization" and larger scale (over 6.0 $Mt \cdot a^{-1}$ for coking coal and over 10.0$Mt \cdot a^{-1}$ for steam coal), promote the development of "green" coal preparation technology and attain "green utility of coal".

 
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## 1 Current Status of China's Coal Preparation Industry

### 1.1 Current Status of Coal Preparation Plant

Up to 2014, the number of coal handling & preparation plants (CHPP) with capacities of more than 0.3 Mt·a$^{-1}$ was more than 2000.With a coal preparation input of 2.42 billion ton and coal preparation ratio 62.53%, China is ranked first in the world. Currently China has over 50 CHPP with treatment capacity of over 10.0Mt·a$^{-1}$ with a total capacity of 650.0Mt·a$^{-1}$, accounting for over 25% of the total coal preparation capacity in China. Among them, the largest coking coal preparation plant has a capacity of 30.0Mt·a$^{-1}$ and the largest stem coal preparation plant 35.0Mt·a$^{-1}$(Guo Jianbin,2014).

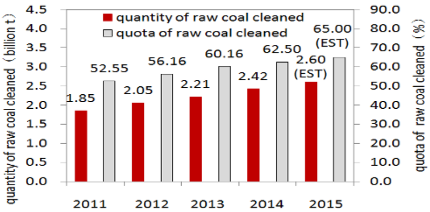


Fig.1 Coal Preparation Status over the past 5 years

The distribution of CHPP's is not balanced to that of the coal resources. West China, the dominant coal producing region, has inadequate CHPPs. On the other hand, coal preparation capacity in the middle and eastern region is in excess of the requirements as the local coal resources are depleted（Li Minghui, Li Zhiyong, Xu Sheng，2014）. As such, it's a priority to construct large-scale modern coal preparation plants, optimize the distribution of coal preparation capacity and promote the preparation rate of raw coal.

### 1.2 Current Status of China's Coal Preparation Technology

Currently in China, the heavy medium coal preparation technology accounts for 60% of the total. Generally, the coking coal preparation process comprises of gravity-feed three-product HM cyclone plus TBS separator plus flotation (see Fig.2). In addition, the steam coal preparation process comprises of raw coal classification plus HM bath separation for lump coal (see Fig.3). Furthermore, dry separation technologies have been developed (see Fig.4). All the slurry water generated during the preparation process can be recycled in a closed circuit.

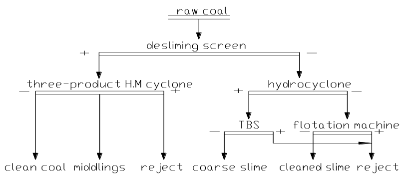


Fig.2 Process for Coking Coal Preparation

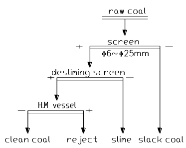


Fig.3Process forSteam CoalSeparation

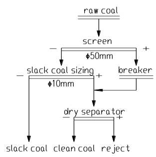


Fig.4 Process for Dry coal Separation

### 1.3 Current Status of Coal Preparation Equipment

Diversification and large capacity equipments the main development trend of Coal preparation equipment. China's gravity-fed three-product HM cyclone (largest diameter: 1500mm, treatment capacity: 600 t·h$^{-1}$) ，XJM flotation cell (unit tank volume: 90m³with slurry treatment capacity 3000 m$^{3}$ ·h$^{-1}$,see Fig.5)，FGX dry separation machine (treatment area: 48m$^{2}$, treatment capacity: 480 t·h$^{-1}$) have been widely applied.

Other equipment such as the classifying crusher, hyperbaric filter press and magnetite separator have attained reliable quality and are promoted in the global market. Vibrating dewatering screens and centrifuges have been extending the market share and competing with imported equipment.

Fig.5 XJM-KS90 Flotation Cell

Fig.6 Modular HM Fluidized Bed Air CHPP

## 2 The Key Technology and Development Trends of China's Coal Preparation Industry

### 2.1 Air Separation Process in the Water- deficient Area and Its Development Trend

80% of China's coal resources, most of which is steam coal, are located in the water-deficient areas of west China. As such, the CHPP's with wet separation process there have been under multiple pressures of the vulnerable ecological environment, deficiency of water resources and freezing problems in the winter(Xing Yumei ，2005).Therefore, dry separation processes is a necessity in these regions.

#### 2.1.1 Compound Air Separation Process

The compound air separation process, with simplified structure, applies air and autogenous slime as medium for separation of steam coal and has been widely applied. However, its separation efficiency for coal feed of difficult washability needs to be further improved.

#### 2.1.2 HM Fluidized Bed Air Separation Process

The world's first dry process of HM separation fluidized bed(see Fig.6) has been developed in China and has been put into use by Sinkiang Energy LLC of Shenhua Group (treatment capacity of $50t\cdot h^{-1}$). The feed coal, with ash content of 18.69%, reports to the system for the production of cleaned coal of less than 3.5% in ash content with an EP value of 0.05 to 0.08(Zhao Yuemin,Li Gongmin,Luo Zhenfu 2014). This process is a breakthrough for the air separation technology and is the first practical application in the world. In the future, the large scaling of modularized air HM fluidized bed CHPP will receive more attention and should prove to be a reliable solution for the dry separation along with the compound dry separation process.

### 2.2 Coal Slime Treatment and Utility and Its Development Trend

With the development of coal mining mechanization, the content of ultra-fine particles increases, and causes a significant increase of coal slime in the separation system. As such, the separation of coarse slime and the flotation of coal slurry with poor floatability must be solved.

#### 2.2.1 TBS

Presently, TBS is the main solution for the separation of coarse slime in China. The TBS separator's advantage lies in the ability to regulate density and it's flexibility to varied coal properties. However, TBS is limited to a narrow size range and is less-efficient when processing coals of difficult washability. Thus in the recent years, the application of the 3 product TBS separator and process of TBS plus high-frequency screen (small aperture)significantly pushed forward the development of the separation of coarse slime in China(Lian Jianhua,LiuJiongtian,Bai Suling,2011).

#### 2.2.2 Flotation

Chinese Scientists have conducted much research on the flotation of coal slurry. The development of new reagents, pulp preparation facility, application of micro bubble flotation column and optimizing of new processes improved the flotation of slurry with poor floatability to some extent. The XJM-KS90 flotation cell has been successfully applied by CCTEG Beijing Huayu Engineering Co., Ltd for Pingshuo CHPP for the flotation of long-flame coal slurry as a breakthrough in the separation of low-rank coal. It can lower the ash content of coal slurry from 26.6% to 15.24%. However, the processing of low-rank coal slurry is still a headache in China. As such, the development of coal pulp preparation technology, surface modification of low-rank coal and multi-stage flotation technology will significantly improve the problem of flotation of low-rank coal.

2.3 Coal Storage and Blending Technology and Its Development Trend.

The traditional coal blending process has disadvantages in inadequate process, low detection and automation level, unstable quality of coal blending, and negatively influences the effectiveness of separation of the distributed coal. Thus the application of the technologies was restricted.

Selection and integration of CHPP of low efficiency, the proportion of group-mines' CHPP and centralized CHPP will be increased and separation of distributed coal will be more popularized. Coal distributing facilitates the stability of coal feed to the CHPP and its product to achieve a rational matching of various coal products and raw coal to meet the requirement of industrial boilers, power plant and coal chemical industry on the clean and adaptation degree of the coal product, and avoid excessive separation of coal to raise the profitability.

In recent years, China has attained great progress in coal blending technology and automation. The Coal blending system developed by CCTEG Beijing Huayu Engineering Co., Ltd has attained industrial application. The application of an Ultimate analyzer and speed-regulating coal feeder significantly promotes the accuracy of coal blending and automation level. In the future, China's coal blending industry will attain unattended operation. Optimization of the coal blending process, perfection of detection technology and localization of control elements manufacturing will be critical challenges.

2.4 Upgrading of Lignite and Its Development Trend

The proven reserves of lignite in China is 130 billion ton, accounting for 13 -15% of China's total coal reserves（Du Jingwen, Yin Jianan，2015）. China experienced high energy demand and an energy shortage in the early century, and as a result developed a large number of lignite mines and developed technology to upgrade lignite. On one hand, lignite has low CV and is not easy to store and transport. On the other hand, it is abundant in reserves and low in price, and applicable as fuel and material for the chemical industry. Thus the development of upgrading technology of lignite is important（Huang Jian,Lei Wenjie，2013）.

As lignite easily degrades to slime, the wet process is not applicable and the dry separation process is mainly applied. The compound dry separation technology effective in rejected contamination to improve the quality of lignite, but the disadvantage of low separation efficiency needs to be addressed.

After drying and shaping, lignite is available for transporting and storage. Presently roller drying and mix-flow drying are applicable. For upgrading of lignite, the belt boiler technology is applicable. For the foreseeable future, optimizing of the dry separation process is still the main stream, but wet processing shall attain more attention. Dry upgrading and pyrolysis upgrading will be hot subjects in this field.

2.5 Underground Reject Removing and its Development Trend

The popularization of underground reject removing technology can help to solve the environmental problems caused by reject stacking and reduce the power consumption for material lifting. The combination of underground reject removal and reject refilling is significant in improving the problems of ground settlement of goaf area and mining under buildings（He Qi，2014）. Currently, this technology is applied in Tangshan Coal mine and Jiyang Mine in China. Due to inadequate space and poor environmental conditions underground, the layout of the technology is very difficult. And high requirement of flexibility, safety and reliability of equipment are the key elements to restrict the application of the technology.

HM bath separator and moving-bed jig technologies have advantages in simplified layout and less land occupation and is recommended in the early stage of underground reject removing application. Besides, to deal with the various underground environments, the development of specialized facilities of high safety and reliability will be the tendency.

## 3 Key Equipment of China's CHPP and Its Development Trend

3.1 Key Separation Equipment and Development Trend

The development of separation equipment in China has been a success. HM cyclones, jigs, HM baths, flotation cells, dry separators, TBS of various specifications have been successfully applied. The locally-manufactured equipment can basically meet the demand of CHPP operation. Currently, the main problem is the poor performance of TBS and dry separator in separating of coal with difficult washability and need to be further optimized. The large separation equipment, such as flotation cells, has a problem of low power efficiency.

The research of separation equipment shall focus on the promotion of reliability and improved power efficiency of the mature separation equipment and develop new equipment of highefficiency for separation of coal slime and low-rank coal and dry separation.

3.2 Reliability of Large-scale screening machine and Its Development Trend

In recent years, the quality of local-manufactured screens has been significantly promoted. Vibrating screens of various models have been manufactured locally. Small screens can take the place of the imported ones, though large screens can still not be replaced with local ones. The main problems for local vibrating screens are poor structure design, dead weight body, high power consumption and travel load, low accuracy in processing and high failure rate of screen reel, beam and exciter（Gao Yongchun, Xu Wenbin，2012）. Particularly, there is an evidence gap in the development and manufacturing level of flip-flow screen and other deep separation equipment of fine coal.

To improve the reliability and large scaling of the screening equipment, the local vibrating screen manufacturers should get rid of the idea of low-price competition and focus on the promotion of key technologies and R&D capability. High quality material need to be applied to promote the reliability and competitiveness of the product and lead China’s screen manufacturing industry to the direction of large scaling, high efficiency, high reliability, energy saving and standardization.， And the breakthrough of the technology in flip-flow screen development shall be emphasized.

3.3 Detection and Control Equipment and Component

In recent years, the manufacturing of level indicators, flow indicators, pressure indicators, belt scales and belt conveyor detectors has been mature, but high precision instruments, such as on-line ash & moisture analyzer and ultimate analyzer still have to be imported.

In the future, the unattended operation of CHPP will be the focus of automation development. The higher requirement for detection and control facilities will be important. Therefore, to speed up the localization of electrical component, the development of high-precision detection facility and new control system will be important.

**4 The Development Trend of China’s Coal Preparation Industry.**

With the increasing occurrence of resources and environment problems, the development mode for China’s coal industry has been turned from the pursuit of speed and quantity to the pursuit of efficiency and quality. The importance of coal preparation has been learned further. Meanwhile, the clean and efficient utility of coal resources will have great influence on the idea of coal preparation and technical advancement. For the future 10 years, it is crucial for the development of China’s coal preparation industry. It is predicted that the following developments will take place:

- Large scale CHPP’s. The small CHPP’s of low efficiency will become obsolete and the average scale of CHPP’s will be raised significantly.
- High efficiency. Complete flow of high accuracy separation is a trend. The separation of lump coal, fine coal, slime coal and slurry will be made with high-accuracy equipment. The efficiency of dry separation equipment will be promoted.
- Intelligentization. With the application of Chinese Manufacturing 2025, the reliability of China’s coal preparation equipment, especially the large-scaling equipment, will be applied much more. The automation and intelligentization level will be promoted. Unattended operation and remote control of CHPP will be attained.
- The promotion of management of CHPP will be improved. Entrusted operation of high efficiency and other operation mode will be promoted.
- “Green utility”. The utilization of by-products will be become a focus to reduce the negative effect on the environment. New processes and technology will be developed to attain sustainable development.

- Clean and adaptation degree. The separation of coal will focus more on the improvement of clean and adaptation degree. Besides, typical separation, the removal of hazardous substance and improvement of coal property will become more important. The degree of purification of coal will be improved. In addition, coal products will be able to be more customized and the added value and utilization efficiency will be significantly promoted.

## 5 Conclusion

China is seeing an excess production of coal, increasing pressure of environment protection and transforming of energy structure. Consequently the coal preparation industry is undergoing a hard times. Therefore, to accurately analyze the existing problems, manage the contradictions, develop “greener” coal processing technologies and utility applications have special importance in the future development of the coal preparation industry in China.

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# Part III
# Use of Quality Control, Automation and Computer Technologies in Coal Processing

# Models and control algorithms of coal washing processes

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## Annotation

The process of automation systems design for coal enrichment processes control is described, following tasks of control are posed: dynamic optimization of technological modes, recycling object control, forming of coal concentrate multicomponent mixture.

The foundations of technological modes optimization based on full-scale/ model approach are stated. The examples of recycling objects in coal enrichment field, complex of dense-media separation units and hydrocyclones are shown. The scheme of object with different recycle types is represented.

General and detailed algorithms schemes of coordinate-parametric recycling object control and results of comparative assessment of control systems with traditional and prospective algorithms are given. The peculiarities of control object for coal concentrate furnace burdening provided coal concentrate dosing from several bunkers on the assembly line are demonstrated.

The structure of control algorithm for one frequency drive issuing commands to several feed tracks is shown. Efficiency estimates of furnace burdening are represented.

## Key words

Coal enrichment processes, recycle, coordinate and parametric control, optimization of technological modes.

The general trend of automation systems design for industrial projects control is characterized by using dependable and full-function computer aids and user-friendly service software.

At the same time specialized system algorithms have not changed significantly recently because of insufficient development of control theory approaches and applied methods. Control automation of coal enrichment plant has the actual state of affair.

The vast majority of these systems perform the functions of control and accounting industrial processes, of planning and emergency start/stop of process equipment, of information displaying to staff. Unfortunately, dynamic optimization problems of enrichment modes and control of enrichment stages, coal concentrate furnace burdening and appropriate complexes of technological units practically haven't been solved yet. Complicated algorithms and underestimating economic effect of implementation are the main reasons why exploratory work of addressing mentioned above problems are not widely used.

*Dynamic optimization of technological modes* using in the preparation for coal enrichment, coal concentrate storing and shipment is successfully carried out under two conditions.

1. Adequate models of all technological complexes should exist.
2. Availability of reliable data about main technological factors.

Design and implementation of complete mathematical models for all coal enrichment processes are costly and sometimes impossible, particularly model construction of uncontrolled disturbances.

Positive results are achieved by using full-scale/model approach that means closely integration of technological unit Input/Output information and counting math models around technological mode [1]. In other words

$$Y^M(t) = Y(t) + \varphi_{Y,U}(t) \cdot [U(t-\theta) - U^M(t-\theta)], \tag{1}$$

where $Y^M(t)$ and $Y(t)$ – model and full-scale output actions of the object (process) in t-point time;

$U^M(t-\theta)$, $U(t-\theta)$ – model and full-scale control actions;

$\varphi_{Y,U}(t)$ – math operator of counting $[U(t-\theta) - U^M(t-\theta)]$ changes according to output action changes;

 
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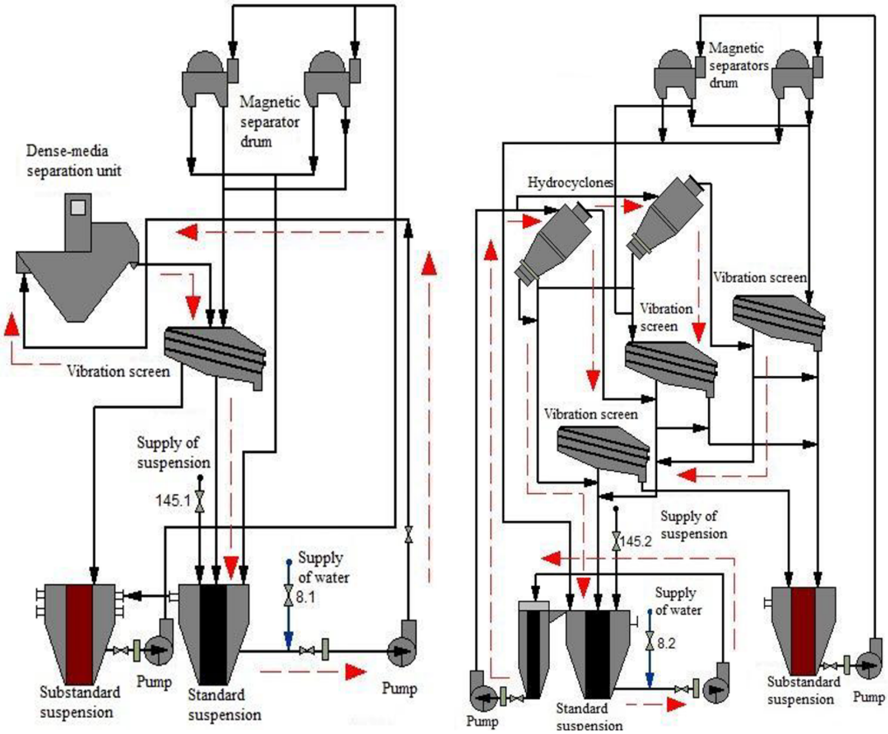
$\theta$ – object persistence.

Changing the $U^M$ you can obtain the whole range of model modes $\{U^M, Y^M\}$, of which the best is selected according to technico-economic criteria.

Identification of the operator $\varphi[\cdot]$ is carried out according to the data of operational stage. That is why, to obtain informative data, special testing actions on control modes are necessary [2, 3].

*Control of coal enrichment processes* as recycling objects. Existence of internal positive feedback (recycles) in control objects makes application of theoretical methods of control algorithm's synthesis difficult [4, 5, 6]. In coal washing synthesis task is complicated by different recycling types (concentration, mass and parameter recycling) and their various combinations [7, 8]. Besides, number of control actions is more than number of controlled variables that also limits the usage of theoretical methods.

The examples of these objects are dense-media separator and heavy medium cyclone (figure 1).




SDM - sump of dilute media; SCS - sump of conditioning suspension; dashed arrowhead line – circulation circuit

Figure 1. Technological scheme of two coal enrichment stages: on dense-media separator complex (a), on heavy medium cyclone (b)

Adjustment of fresh magnetite suspension density on each enrichment complex is necessary to maintain the required quality of concentrate. Regulation of work suspension that is being fed to separator and to hydrocyclone supply sump is carried out by diluting conditioning suspension with make-up water feeding into inlet of suspension pump. For water consumption adjustment pneumatic governor valves (pos. 8.1 and 8.2 on fig.1) are used, for fresh conditioning suspension adjustment pneumatic governor valves are also used p (pos. 145.1 and 145.2 on fig.1). Average consumption of suspension entering the circulation circuit from flow dividers and separation screens is about 70% of work suspension consumption.

Обобщенная схема объектов с различными типами рециклов представлена на рисунке 2.

Generalized scheme of objects with different recycling types is shown on figure 2.

Percentage decreasing of end product that is returned to control object on recycling chain leads to reducing of the processed material amount in the reaction zone of the control object.

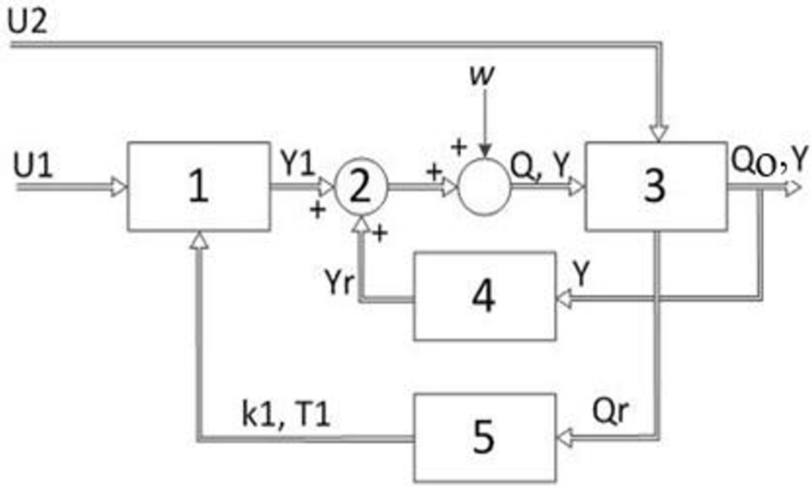



Figure 2. Generalized scheme of recycling control object

1 – control channel of coordinate control action, 2 – blending device, 3 – flow divider, 4 – conversion channel of product quality changes in recycle chain, 5 – conversion channel of product consumption changes in recycle chain, $Q_o$ – consumption of output product being fed to control object output, $Q_r$ – consumption of output product that is returned to recycle chain, $U_1$ – coordinate control action, $U_1$ – parametric control action, $Y_r$ – recycle chain output action.

For those objects coordinate-parametric algorithm with calculation of two control action types is designed [9]:

- coordinate control action which changes influence object controlled output action. For example, consumption or material quality characteristics which are processed in control object;

- parametric control action which changes influence dynamic characteristics of changes transformation channel of coordinate control action into changes of controlled object output action. For example, the ratio between the quantity or end product consumption entering to control object output and entering in control object recycling chain.

Generalized algorithms structure is shown on figure 3 [10], it detailed scheme is given on figure 4.

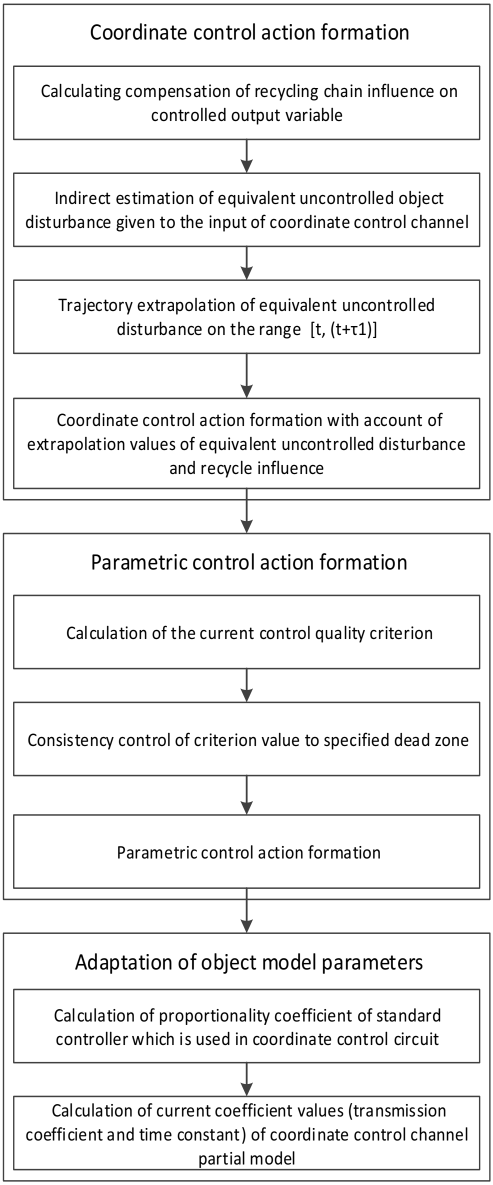



Figure 3. Generalized coordinate-parametric algorithms structure

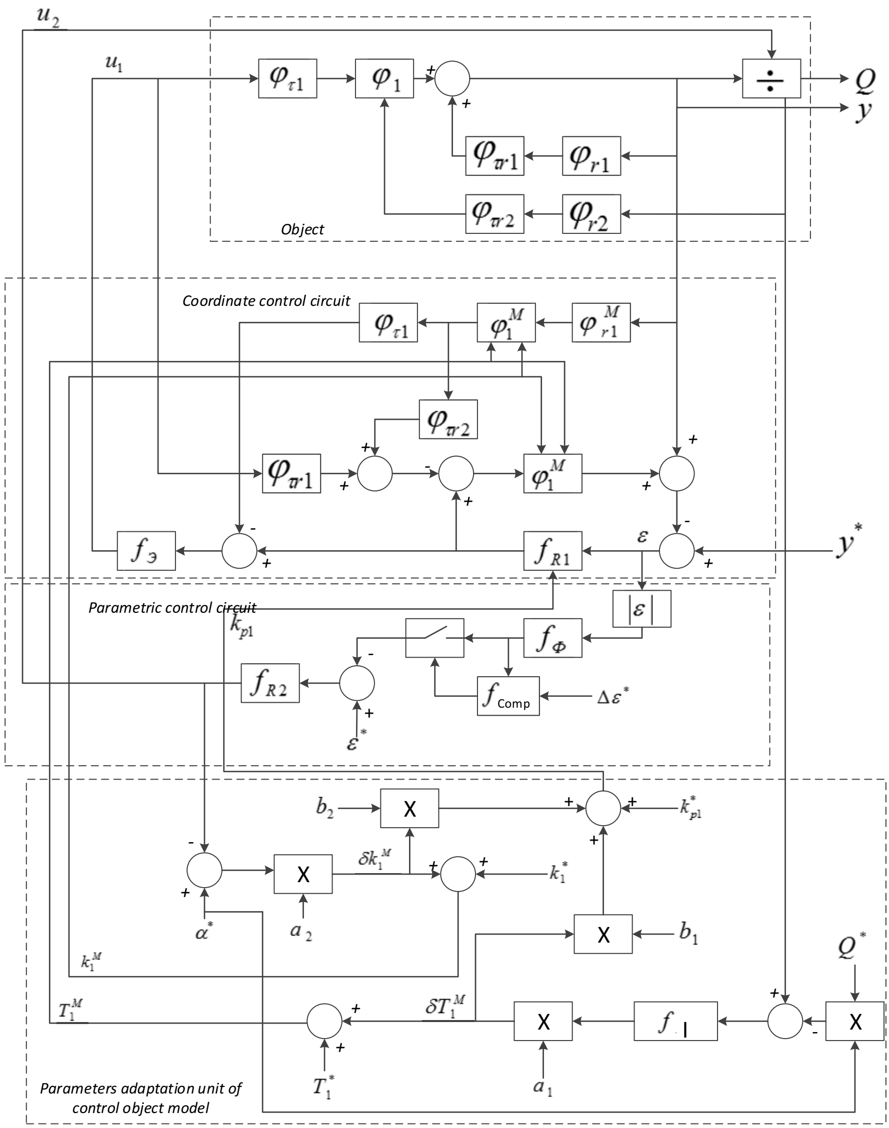



Figure 4.Scheme of coordinate-parametric control system

$f_{R1}, f_{R2}$ – the first and the second control unit; $f_e$ – extrapolation unit; $f_f$ – filter unit; $f_i$ – integrator; $f_{comp}$ – comparator; |ε| - unit of control error module calculation; ÷ - flow divider; X – multiplier unit;

$\alpha^*, k_1^*, T_1^*, k_{p1}^*, Q^*$ - basic system parameters; $a_1, a_{2,} b_1, b_2$ –constants calculated from the parameters of a specific technological object.

With offered algorithms mean-module error of adjustment magnetite suspension density 3-4 times less than if we use traditional algorithms.

To find the control problem solution of multicomponent furnace burdening shipping from commercial output stock you often have to deal with distributed control objects when the current changes of one or another furnace component are determined at the collecting conveyor by control action changes through several channels.

For example, the brand tract shipment of coal (T, TS, K, KS, OS) from concentrator shelter warehouse (Figure 5) shows that material control from each stack is carried out through several channels and each channel includes the execution unit (inverter) and feeders group arranged along collecting conveyor and parallel connected to the transformer output.

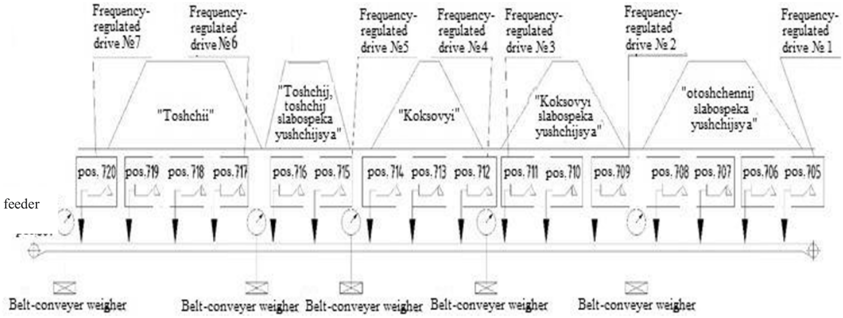


Figure 5. Control object scheme

Cost saving on the amount of frequency converters in comparison with the equipment of each individual feeder frequency converter is obvious. However, the control channel characteristics are significantly deteriorate.

Under these conditions, it is impossible to achieve the desired quality of multicomponent furnace burdening using controllers with typical control law. More effective control algorithm presented, as part of the control system, on figure 6.

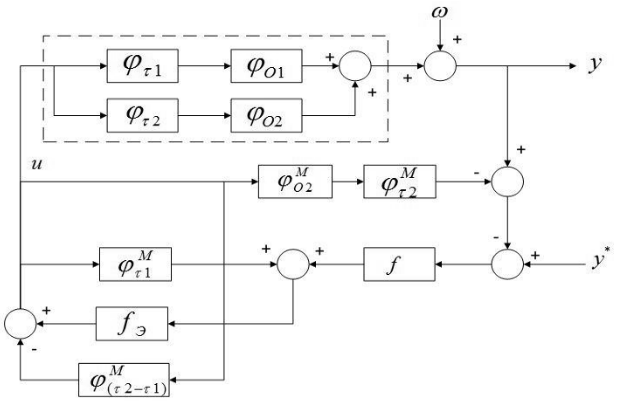


Figure 6. Control system with more effective control algorithm

$\varphi_{\tau 1}, \varphi_{O1}$ – delay operator and model operator of the first feeder; $\varphi_{\tau 2}, \varphi_{O2}$ – delay operator and model operator of the second feeder, $f_e$ – extrapolation unit, $f$ - controller unit, index «M» means «model».

In particular, the transient process graphs (figure 7) shows that the control quality indicators in automation control system with proposed $Y_2$ control law are much better than for automation control system with typical control law.

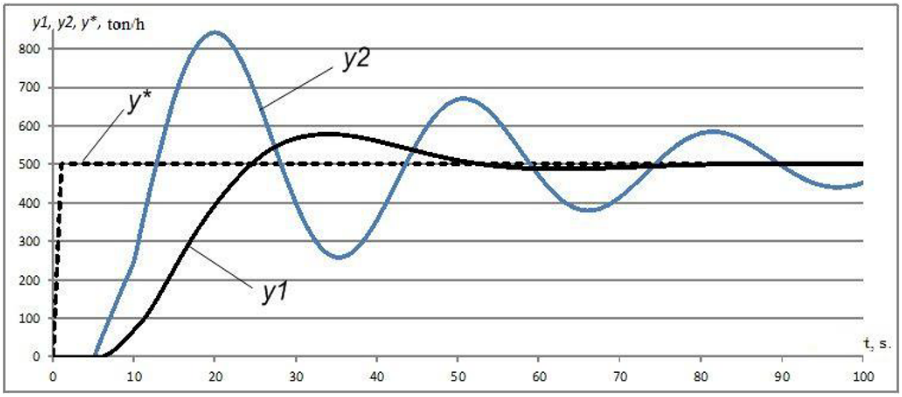



Figure 7. Transient processes for control input where $Y_1$, $Y_2$ – automation control system output action with proposed control law and with typical control law; $Y^*$- control input

Design and implementation of control algorithms for coal enrichment objects in accordance with their dynamic properties will allow to significantly improve technico-economic criteria and to develop Automation control system of the entire industrial process.

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# ANALYTICAL REPRESENTATION OF WASHABILITY CURVES WITH APPLICATION IN THE SIMULATION OF GRAVITY CONCENTRATION METHODS

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**Abstract**

The present level of economy sets more and more stringent requirements for product quality, equipment and technologies used in the coal preparation industry. In this regard, methods of process optimization and efficiency improvement, based on the information technologies, become of current interest at existing facilities as well as in designing of new ones. Mathematical models of preparation processes are of particular relevance in the design and operation of concentrators, because they allow exploring various options for process flowsheets and determining optimal process parameters. The choice of process parameters is based on analytic representation of washability curves using numerical analysis.

The results of float-and-sink testing, presented in graphical form as washability curves, allow making a theoretical balance of coal processing products and establishing an optimal separation density to reach the maximum production of clean coal. Usually, their plotting is conducted manually on graph paper, which is a very time-consuming and laborious procedure. This issue is arising, from time to time, in publications of authors all around the world, but there is still no commonly accepted procedure designed.

In the present article authors made an attempt to examine different existing methods for analytical representation of coal washability curves suitable for the application in computer simulation of gravity concentration processes. During the work on this topic, a software complex has been developed. Description of the developed software and an example of its use for the purpose of flowsheet selection for the coal preparation unit is provided.

**Keywords:** coal, washability curves, float and sink analysis, gravity concentration, interpolation, flowsheet, simulation.

## 1. Introduction

Selection of the optimal flowsheet for treatment of raw materials is, ultimately, the main objective of mineral processing engineers. Although the method of physical modeling stays the most common in mineral processing, mathematical models help to save research resources, reduce the time needed for data analysis and investigate various options to obtain marketable products. In this regard, the use of modern information technologies is becoming of particular relevance for the improvement of equipment efficiency and optimization of existing facilities as well as at designing of new ones.

Coal washability studies are usually based on the method of float and sink analysis described in international standards [1-3]. These results are useful when designing and redesigning a plant, as well as in predicting, controlling and assessing its performance [4]. The results of float-and-sink testing, presented in tabular form, are the basis for the plotting of washability curves, which indicate the ease or difficulty of the separation.

In addition, washability curves allow making a theoretical balance of coal processing products and establishing an optimal separation density to reach the maximum production of clean coal. The conditions for obtaining maximum yield of total concentrate were first described in [5] and known as Reinhardt's theorem: "If several size classes of coal are processed separately, the maximum yield of the total concentrate with a given ash content will be obtained at the same average ash content of elementary layers of separation". For a practical use it is necessary to plot washability curves for all size classes, which are



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subjected to gravity separation, specify the desired ash content of the total concentrate and then determine the separation density for each size class.

As a rule, the family of washability curves should include at least 4 plots:

1. characteristic ash curve $\lambda$ showing the highest ash content of the floating particles at a given density, plotted as cumulative weight percent of floats against the ash content of this fraction;
2. cumulative floats curve $\beta$ which indicates the yield of the concentrate at a set ash content, obtained by plotting the cumulative yield of floating fraction at each relative density against the cumulative ash content at the same point;
3. cumulative sinks curve $\theta$ plotted as cumulative mass percent of sinks at each relative density against cumulative ash content at the same point, indicating the ash content of discards that will be obtained for a given yield of concentrate;
4. densimetric curve $\rho$ showing the maximum output of clean coal at a selected density of medium in case of perfect separation.

In some cases, other curves, such as near density curve, total sulfur content or calorific value, may also be plotted.

Russian standard [3] implies construction of these curves manually on graph paper. In order to improve the accuracy and reduce the complexity of analysis, the use of computer methods for the design and analysis of washability curves should be taken into consideration. The development of the computer software that will use methods of numerical analysis to determine the functions approximating washability curves enables the successful implementation of the Reinhardt's theorem.

## 2. Analytical representation of the washability curves

Some authors [6-8] apply the least squares approach to obtain an approximation function for washability curves. The essence of the method is as follows.

Suppose there are $n$ values of the argument $x_i$ and the corresponding values of $y_i$. It is necessary to find a function $F(x)$, for which the sum $Q$ of the squares of its deviations from the values $y_i$, is minimum:

$$Q = \min \sum_i \left(F(x_i) - y_i\right)^2 \tag{1}$$

Unknown function $F(x)$ can be written as

$$F(x) = C_0 \cdot \phi_0(x) + C_1 \cdot \phi_0(x) + \ldots + C_m \cdot \phi_m(x), \tag{2}$$

where: $\varphi_o, \ldots, \varphi_m$ – basis functions; $C_o, \ldots, C_m$ – unknown coefficients.

A variety of functions can be used as the basis, such as integer powers of the argument, logarithmic functions etc. To find the unknown coefficients, the following system of equations has to be solved:

$$\begin{cases} \sum_{i=0}^{n}\left(C_0 \cdot \phi_0(x_i) + C_1 \cdot \phi_1(x_i) + \ldots + C_m \cdot \phi_m(x_i) - y_i\right) \cdot \phi_0(x_i) = 0 \\ \sum_{i=0}^{n}\left(C_0 \cdot \phi_0(x_i) + C_1 \cdot \phi_1(x_i) + \ldots + C_m \cdot \phi_m(x_i) - y_i\right) \cdot \phi_1(x_i) = 0 \\ \ldots \\ \sum_{i=0}^{n}\left(C_0 \cdot \phi_0(x_i) + C_1 \cdot \phi_1(x_i) + \ldots + C_m \cdot \phi_m(x_i) - y_i\right) \cdot \phi_m(x_i) = 0 \end{cases} \tag{3}$$

Since the number of equations in the system is small, its solution can be found with the use of direct methods, for example by Gaussian elimination.

The approximation of the washability curves with the least squares method allows setting a relationship between yield and ash content and makes it possible to evaluate the accuracy of the plotting by means of sum of residuals (see Formula 1). On the other hand, resulting function does not pass through all the points of the dataset, representing just an approximation, aspect that does not meet the requirements of the standard technique.

This limitation can be overcome by constructing curves using the interpolation of float and sink testing results. There are various methods of interpolation that have been developed, such as Lagrange interpolation polynomial, cubic splines interpolation, etc.

The application of the Lagrange interpolation polynomial to obtain analytical equation of washability curves will result in the following equation:

$$\gamma(A^d)=\sum_{i=1}^{n}\gamma_i\cdot\prod_{i\neq j}\frac{A^d-A_j^d}{A_i^d-A_j^d}, \tag{4}$$

where: $\gamma_i$, $A^d_i$ – experimental values of yield and ash content in density fraction *i*.

This method is less time-consuming in terms of calculations among other interpolation methods, but its reliability is low. Problems of using Lagrange interpolation polynomial with the increase of number of data points and ways of solving them are well described in literature [9].

The possibility of using the spline interpolation method for constructing coal washability curves was described in several works [10-12]. This method provides the smoothest curves that will pass through the experimental points; it is also possible to obtain an analytical representation of washability curves in the form of a piecewise third degree polynomial.

Cubic spline is the function *s(x),* satisfying the following conditions:

1. on each interval $[x_{i-1}; x_i]$ ($i$ = 1, 2,…, $n$) between the nodes of interpolation, *s(x)* is a polynomial of the third degree;
2. its values at the interpolation nodes are equal to experimental values $y_i$;
3. the function, its first and second derivatives are continuous on the entire range of interpolation.

On each interval $[x_{i-1}; x_i]$ ($i$ = 1, 2,…, $n$) cubic spline is expressed as follows:

$$s_i(x)=a_i+b_i\left(x-x_i\right)+\frac{c_i}{2}\left(x-x_i\right)^2+\frac{d_i}{6}\left(x-x_i\right)^3, \tag{5}$$

where: $a_i$, $b_i$, $c_i$, $d_i$ – the unknown coefficients to be determined.

The coefficients $c_i$ are defined as a solution of the tridiagonal system of linear equations, which can be solved very efficiently by applying Thomas algorithm [13]:

$$h_i c_{i-1}+2\left(h_i+h_{i+1}\right)c_i+h_{i+1}c_{i+1}=6\left(\frac{y_{i+1}-y_i}{h_{i+1}}-\frac{y_i-y_{i-1}}{h_i}\right),\quad c_0=c_n=0, \tag{6}$$

The remaining coefficients ($a_i$, $b_i$ and $d_i$) are calculated by the formula:

$$d_i=\frac{c_i-c_{i-1}}{h_i},\quad b_i=\frac{h_i}{2}c_i-\frac{h_i^2}{6}d_i+\frac{y_i-y_{i-1}}{h_i},\quad a_i=y_i,\quad i=1,2,\ldots,n \tag{7}$$

**3. Software realization of washability curves approximation**

The analysis of the methods used for the construction of washability curves allows the following conclusions to be drawn:

- in most cases, to plot the curves a big set of initial data is needed, but the implementation of computer-based methods for data analysis should reduce the complexity and laboriousness of coal washability studies;
- only a few numerical methods can produce results that will be rather accurate for routine process control and which will comply with the standard procedure;
- possibilities to reduce the time spent on data processing and to obtain the precision required for engineering calculations should be investigated.

In this regard, the most promising way of building washability curves seems to be by combining multiple numerical methods. This approach was implemented by the authors in a software complex. The washability curves are plotted based on fractional composition of the sample calculated from the results of float-and-sink analysis.

The characteristic ash curve $\lambda$ is approximated by the least squares method, as this approach gives an analytical expression of the relationship between yield and ash content of the elementary density fractions. This relationship is of great importance to the modeling of coal preparation processes; moreover, using the formulas described in [6], functions for the remaining curves can be defined as it follows:

$$\int_0^{\gamma_\beta} A_\lambda^d d\gamma = A_\beta^d \cdot \gamma_\beta, \qquad \int_0^{100-\gamma_\beta} A_\lambda^d d\gamma = A_\theta^d \cdot (100-\gamma_\beta), \tag{8}$$

where $\gamma_\beta$ – yield of the floats, $A^d_{\lambda,}$ $A^d_{\beta,}$ $A^d_\theta$ – the ash content of the elementary fraction, floats and sinks respectively.

To plot the cumulative floats $\beta$ and sinks $\theta$ curves, as well as the densimetric curve $\rho$, the cubic splines interpolation is applied. In this case, the output value can be easily calculated for any ash content needed to produce, for example, a theoretical balance of gravity concentration products. The general view of the washability curves obtained with the developed software is shown on Figure 1.

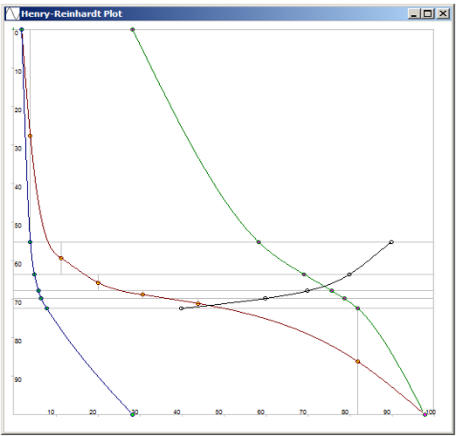


Figure 1. Example of washability curve plot

**4. Application of washability curves for gravity methods simulation**

The developed mathematical models implemented as a computer program can be used to predict the technical and economic performance of the plant, carry out process control, and assess quality of products depending on the particle size and the fractional composition of raw coal.

The results of this research found practical realization in the pre-feasibility studies for the reinstatement of the coal preparation unit at one of the mines operating in Kuznetsk coal basin. The refurbishment objective was to optimize the process flowsheet of the coal preparation unit in order to reach a unit capacity of 3.0 Mtpa, producing a concentrate with a total ash content less or equal to 10%.

To select the optimal flowsheet for obtaining the maximum yield of clean coal with a given ash content, a number of options were compared. The recommended flowsheet (Figure 2) which allows getting a coal concentrate with the required quality characteristics includes the following operations:

- removal of large rocks in the trommel;
- preliminary screening followed by crushing to a particle size of 100 mm;
- dry sieving to separate the 0-6 mm size class as a final product;
- wet screening on a sieve with an opening size of 13 mm;
- heavy medium separation of the 13-100 mm size class in a drum separator;
- separation of the 6-13 mm size class in a dense media cyclone;
- dewatering of final products.

This option provides the highest yield of total concentrate, allows the use of existing steel structures and transportation routes, reduces pumping of slurries, decreases energy consumption and maintenance costs.

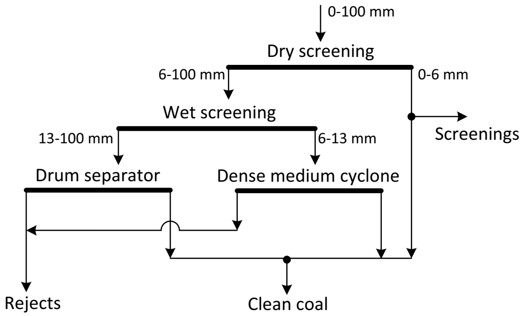


Figure 2. Proposed flowsheet for coal preparation unit

To determine the right separation densities, at which the highest yield of concentrate with the required quality is obtained, a simple mathematical model of the flowsheet was constructed:

$$\gamma_{total} = \max_{\left|A_t^d - A_r^d\right| \le \varepsilon} \sum_i \gamma_i(\rho_s), \tag{9}$$

where $\gamma_i$ –yield of the concentrate for the size class $i$; $A^d_t$ – planned ash content of total concentrate; $A^d_r$ – calculated ash content; $\rho_s$ – separation density, $\varepsilon$ – precision of calculations.

According to this model, the best flowsheet will be the one in which the sum of concentrate yields of all size classes is the highest while the difference between calculated and planned ash content in the total concentrate is lower than the calculations precision.

Using the software developed by the authors, the optimal parameters of the coal preparation process have been determined. To do this, washability curves were drawn and with their use, the yield and ash content were obtained at different separation densities. The maximum yield of total concentrate is expected at a specific gravity of 1.51 g/cm$^3$ in drum separator and 1.58 g/cm$^3$ in dense-media cyclone. The material balance of the flowsheet as well as quality characteristics of the products are shown in Table 1.

Table 1. Calculated material balance of the flowsheet.

| **Product** | **γ, %** | **Q, t/h** | **$A^d$, %** | **$W^r$, %** |
|---|---|---|---|---|
| Concentrate 13-100 mm | 31.5 | 157.5 | 7.7 | 8.0 |
| Concentrate 6-13 mm | 11.8 | 59.0 | 5.8 | 8.0 |
| Screenings 0-6 mm | 8.8 | 44.0 | 24.0 | 8.0 |
| **Total concentrate** | **52.1** | **260.4** | **10.0** | **8.0** |
| Rejects >100 mm | 4.0 | 20.0 | 82.7 | 8.0 |
| Rejects 13-100 mm | 8.0 | 40.0 | 73.5 | 7.0 |
| Rejects 6-13 mm | 3.7 | 18.6 | 83.2 | 12.0 |
| **Total discard** | **15.7** | **78.6** | **78.1** | **8.5** |
| Screenings 0-6 mm | 28.7 | 143.7 | 24.0 | 8.5 |
| Dewatered slime | 2.9 | 14.7 | 22.0 | 30.0 |
| **Total fines** | **31.7** | **158.4** | **23.8** | **11.1** |
| Circulating water | 0.5 | 2.5 | 38.0 | - |
| **FEED** | **100.0** | **500.0** | **25.2** | **-** |

**5. Conclusions**

Coal washability curves have been used in processing plant design and operations for a long time. With advances in computers and software, this tool has become a lot more accessible to practitioners of this field. For this, the washability curves should be first digitize with the aid of numerical analysis methods.

Approximation using the least squares method allows the adjustment of the function fitting the results of float and sink analysis, making it possible to achieve the required accuracy of washability curves construction. Lagrange interpolation polynomial provides a relationship between yield and ash content of the density fractions in analytical form, but this method does not provide accurate results acceptable for technological calculations. The spline interpolation method brings the highest amount of complications to programming. However, the shape of the interpolated curves is very close to the one manually plotted, and calculations of the yield or ash content of a fraction of given specific gravity can be simply performed.

Analytical representation of washability curves was applied to calculations of expected performance of the coal preparation unit. Based on the developed mathematical model of the proposed flowsheet and well-known laws of mineral separation, quantitative and qualitative characteristics of products were estimated.

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# Elemental management system (ElMan) based on TNC + NIS Technology for On-line Coal Analysis and Process control

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**Abstract.**

The key tasks of each industry are to reduce costs and improve quality of the product, which can be obtained through the increased use of automation systems. On-line quality measurement is the front tool to optimize process efficiencies, to maximize the use of resources and to provide the improvement of the quality. Relating to coal, the major quality parameters are the content of moisture, flight substances, ash content, carbon content, sulfur content and others.

Neutron activation analyzing (NAA) technologies are the basis of many on-line analysis systems that are now widely used in different industries [1,6,10] and are included in different process automation systems. The root principle is using neutrons interacting with atoms of the target material and producing gamma-rays with energies characteristic of the emitting nuclei which are specific for each element.

In this article the most precise and accurate method of NAA online elemental analysis – TNC+NIS is described and compared with precede technology of PGNAA (TNC) and results of online measurement of coal quality by TNC+NIS analyzer are presented. Calculations of calorific capacity, based on data from TNC+NIS analysis is compared with real values from burnt coal.

**Key words:** online elemental analysis, NIS, TNC, coal calorific value, ElMan, neutron activation analysis

## 1. Introduction. Existing technologies of measurement of coal quality parameters. Their advantages and disadvantages.

There are two basic techniques for bulk elemental analysis that utilize neutron sources and promptly emitted gamma-rays. These are thermal neutron capture (TNC) often referred to as prompt gamma-ray neutron activation analysis (PGNAA) and neutron inelastic scatter (NIS). Both TNC and NIS have the advantages of using highly penetrating radiation so that measurements are averaged over a large volume of material.

TNC involves bombarding a sample with neutrons from a radioisotope source, usually $^{252}$Cf. Thermal (low energy) neutrons are captured by the nuclei of elements present in the sample. The capture process in most cases is accompanied by the immediate release of energetic gamma-rays that are characteristic of the element. The complex spectrum of energies produced can be interpreted to provide analytical information on the proportion of the various elements present in the sample. The response of different elements to TNC differs widely although the technique is particularly useful for measuring elements such as H, S, Si, Fe, Ca and Cl.

 
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NIS uses high-energy neutrons, usually from $^{241}$Am–Be neutron sources or 14 MeV neutron generators. In contrast to TNC, the fast neutrons are not captured but scatter inelastically from the nuclei of elements in the sample. During this process, prompt gamma-rays are produced which are characteristic of the elements present. NIS is well suited to the analysis of elements such as C, O, Al, Si and Fe. Compared to TNC, NIS has the advantages of being independent of the complex slowing down and capture processes inherent in TNC and of having simpler gamma-ray spectra

TNC+NIS detects both NIS and TNC gammas. This means that a wider range of elements can be detected. Analyzers based on PGNAA-TNC are optimized for TNC and, therefore, in situations where the only elements of interest are those best detected using TNC, PGNAA-TNC will often be preferred. For other situations TNC+NIS will be superior.

## 2. TNC+NIS technology: what is it?

The TNC+NIS technology uses $^{241}$Am-Be neutron sources. The high energy neutrons from these sources interact in the sample resulting in a combination of NIS and TNC reactions [2]. Slow TNC neutrons are created by inelastic scattering of fast neutrons from hydrogen atoms. The relative proportion of NIS to TNC is primarily determined by sample composition and analyzer design.

The major differences between TNC+NIS and PGNAA-TNC analyzers are as follows:

- NAA (TNC+NIS) analyzers use BGO detectors whereas most commercial PGNAA-TNC analyzers use NaI detectors. BGO detectors are more expensive but, unlike NaI detectors, do not suffer from thermal or mechanical shock and are relatively insensitive to neutrons. Furthermore, whilst BGO has a lower resolution at room temperature than NaI, it has much higher photopeak efficiency, particularly for high-energy gamma-rays.
- The source used in most commercial PGNAA-TNC analyzers is $^{252}$Cf, with a half-life of 2.65 years. TNC+NIS uses a much higher energy source $^{241}$Am-Be, with a half-life of 432 years. Consequently, PGNAA-TNC analyzers will require "topping up" or renewal of the source every few years, whereas NAA(TNC+NIS) analyzers will be limited only by the local health authority's requirements (10-15 years).

To address this problem of spatial sensitivity, TNC+NIS configurations have been developed (2) that comprise fast neutron source(s) and gamma-ray detectors disposed in transmission and/or backscatter geometries as shown in Fig. 1. For a transmission configuration in which the neutron source and gamma-ray detectors are placed on opposite sides of the sample, the spatial response depends on the relative attenuation of neutrons and gamma-rays in the sample. The spatial response will be the most intensive towards either the source or detector sides of the sample. To detect gamma-rays from both spatial zones, another transmission or backscatter configuration is located adjacent to the first transmission configuration such that its spatial response is biased to compensate for the spatial bias of the first mentioned transmission configuration [1] (Fig. 1).

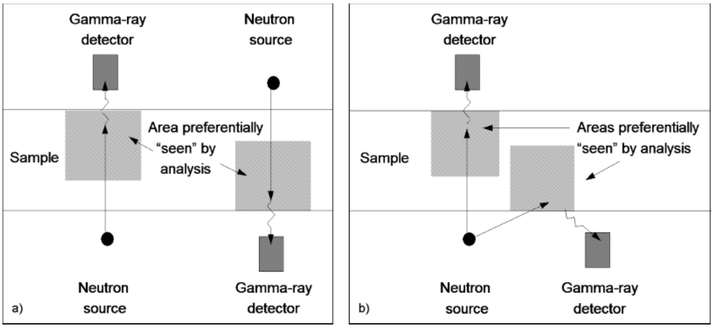


Fig. 1. Schematic drawings of two neutron/gamma transmission gauges (a) and combined transmission and backscatter gauge (b), showing how each combination can be used to provide a more uniform spatial response

## 3. Elemental management system based on TNC+NIS Analyzer. General technical specification

The accurate on-belt determination of coal quality is required in a wide range of applications in the coal industry, including mine grade control, coal preparation plant control, product blending, stockpile management, power station feed, and monitoring at coal shipping ports. The key parameters that determine coal quality are ash content and calorific value [3].

Both parameters are defined in terms of the behavior of coal when it is burnt, for example, ash content is defined as the amount of residue after burning under prescribed conditions. Non-destructive and direct measurement of either parameter is not possible. Instead, they are inferred from measurements of mineral matter (primarily Si, Al, Ca and Fe) and/or organic matter (C, H and O).

Because of their sensitivity to changes in segregation and profile/loading, PGNAA gauges primarily operate on sample by-lines and are, therefore, expensive and subject to sampling error. The potential advantages of using NITA in this situation are that unlike PGNAA, TNC+NIS can measure C directly, and Al independently of Fe; satisfactory compensation can be made to allow for segregation in the sample. In the case of coal, it is common for "lumps" and "fines", which are usually vertically segregated, to have different ash and specific energy values. Difference in ash content between lumps and fines is typically ~6 wt%.

The industrial testing of the elemental management system based on TNC+NIS analyzer of the geometry shown in Fig. 1b for the direct on-belt analysis of ash and coal calorific capacity. Measurements have been carried out on Kendai power plant of ESCOM electric company, SAR. Analyzer was installed on the conveyor feed from coal stockyard. Measurements by NIS+TNC analyzer where then compared with laboratory sample measurements. The comparison of two different measurement techniques are presented in Table 1[1].

| 2 hourly | Lab ASH | TNC+ NIS ASH | ASH Differe | Lab CV (MJ/k | TNC+ NIS CV | CV differen ce | Lab Vols | TNC+ NIS Prog | Vols differ |
|---|---|---|---|---|---|---|---|---|---|

[1] The results were obtained with NITA II (NIS+TNC Analyser).

| **tons** | **(%)** | **(%)** | **nce** | **g)** | **(MJ/kg )** | **(MJ/kg)** | **(%)** | **Vols** | **ence** |
|---|---|---|---|---|---|---|---|---|---|
| 6262 | 39.50 | 40.37 | -0.87 | 14.96 | 14.61 | 0.35 | 18.30 | 18.54 | -0.24 |
| 5701 | 41.26 | 41.52 | -0.25 | 14.96 | 14.17 | 0.79 | 18.01 | 18.56 | -0.54 |
| 6283 | 41.28 | 41.48 | -0.21 | 14.80 | 14.04 | 0.76 | 18.15 | 18.10 | 0.05 |
| 5301 | 41.26 | 41.00 | 0.26 | 14.71 | 14.24 | 0.47 | 18.27 | 18.05 | 0.22 |
| 3000 | 41.40 | 43.59 | -2.19 | 14.62 | 14.27 | 0.35 | 18.27 | 18.05 | 0.22 |
| 1971 | 41.55 | 43.75 | -2.20 | 14.55 | 13.63 | 0.92 | 18.25 | 18.14 | 0.11 |
| 2885 | 41.62 | 43.19 | -1.57 | 14.53 | 14.20 | 0.33 | 18.30 | 17.85 | 0.45 |
| 2838 | 41.62 | 42.93 | -1.31 | 14.61 | 14.27 | 0.35 | 18.30 | 19.97 | -1.67 |
| 4198 | 41.62 | 42.37 | -0.75 | 14.72 | 14.39 | 0.33 | 18.30 | 18.21 | 0.09 |
| 4872 | 41.62 | 41.33 | 0.29 | 14.89 | 14.82 | 0.07 | 18.30 | 18.52 | -0.23 |
| 2595 | 41.62 | 41.15 | 0.47 | 14.93 | 14.83 | 0.10 | 18.30 | 18.58 | -0.28 |
| 5384 | 41.62 | 40.75 | 0.87 | 14.98 | 14.94 | 0.04 | 18.30 | 18.87 | -0.57 |
| **51290** | **41.33** | **41.95** | **1.03** | **14.77** | **14.37** | **0.29** | **18.25** | **18.45** | **0.56** |

Table 1. Tabulated data of the one-day dynamic measurement on a 2 hourly basis

There were realized three types of observations - ash content, calorific value and volatiles content of coal were measured in lab and by TNS+NIS analyzer. Feed value of conveyor is 2137 t/h. In Table 1, the results of both techniques are compared. The average difference between two techniques is 1,03% for ash content, 0,29 MJ/kg for calorific value and 0,56% for volatiles content. Taking in consideration, that laboratory test has its own error, which appears from sampling inaccuracy and measurement error, it can be concluded that TNC+NIS analyzer gives sustainable data to operate the furnaces and predict the energy, recuperated from the burned fuel.

## 4. Current TNC+NIS installations: ElMan system for power plants coal quality parameters control

System of elemental management ElMan for solid coals can increase the performance efficiency of coal-fired thermal power plants.

Most coal-fired thermal power plants of the CIS use the results of laboratory analysis to determine the coal quality and operational manage the combustion process.

The introduction of automated system of coal flow management ElMan helps to overcome the existing limitations of the lab control methods and to achieve a new quality level of plant management that will lower the cost of power generation [7].

TNC+NIS analyzer is designed for on-line measurement of bulk materials on conveyor belt. The analyzer has proved its worth at elemental analyzing of the solid fuels and is able to measure chemical elements as well as calculate calorific capacity, ash and uncombined carbon content [8].

Integration of readings from TNC+NIS analyzer, belt scales, moisture gauge and bulk density measuring system allows to carry out reliable measurements of qualitative characteristics of the solid fuel. Measurable parameters of the analytical system are:

- Complete elemental composition of the coal (C, O, H, S, N, Si, Al, Ca, Mg, Fe, Mn, P, Na, K, Ti);
- Ash content and elemental composition, calculation in oxide forms;
- calorific capacity, volatile hydrocarbons and uncombined carbon content;
- mass and volumetric flow of the coal conveyors;
- Moisture and bulk weight.

During the measurement of bulk materials all the elements are being determined individually. Concentration can be calculated in oxide forms or be displayed as simple molecular species.

Calorific capacity and volatile hydrocarbons content calculation is based on complete elemental analysis and on-line measurement of the free moisture. Using the information about free and total moisture, the system calculates the concentration of $H_{org}$ and thereupon calorific capacity, volatile hydrocarbons and uncombined carbon content [].

Fig. 2 shows the correlation between ElMan (TNC+NIS) analyzer measurement of calorific value and data from the furnaces about the recuperated energy [4]. White curve is calorific value predicted by TNC+NIS analyzer, blue line is real calorific value from the incinerated coal. Blue and white curves shows a very similar behavior, in contrast to a black line, representing laboratory calorific value analysis.

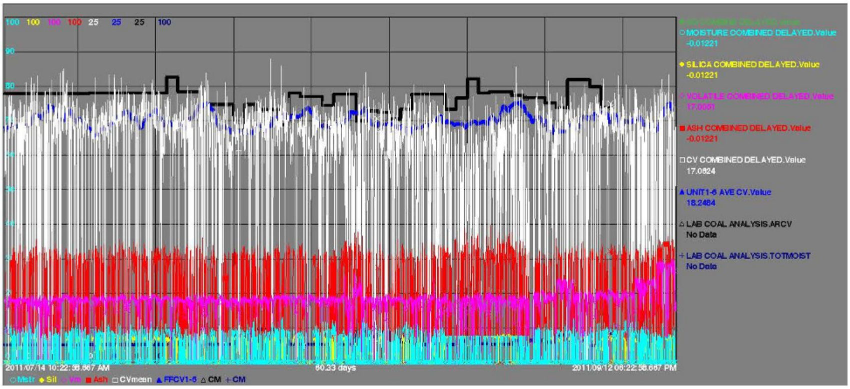


Fig.2 Comparison of laboratory and TNC+NIS calorific value measurements with real data from incinerated calo

Basing on measurement of elemental composition of incoming coal, automotive control systems can produce various adjustments of technological process to increase the efficiency of thermal power plants.

Our company have realized system of elemental management (ElMan) for automotive control and quality management of coal feed for thermal power plants/

The ElMan combustion control unit allows increase the efficiency management of thermal power plants. ElMan system analyses the elemental composition of fuel feed and can realize technical coordination and support in the following directions:

- ElMan provides technical staff and management with on-line coal quality data before combustion, thereby providing a forecast of energy production for a few days/hours in advance;
- ElMan guides the performance of pulverized coal injection system with predetermined values of power generation and the actual calorific value of the fuel mixture;
- The ACS controls the air flow for combustion to create the optimal combustion conditions for coal of predetermined composition;

## 5. Conclusion

- TNC+NIS gamma technology allows to register and analyze specters from neutron reactions of two different natures: thermal neutron capture (TNC) also known as (PGNAA) and neutron inelastic scatter (NIS) and provides more complete analysis than technologies, based only at PGNAA-TNC;
- TNC+NIS technology is more accurate than PGNAA-TNC, due to utilization of two BGO detectors, registration gamma-radiation from both sample sides more effectively, so TNC+NIS detection system can be adjusted for fluctuations of loading of conveyors and expected segregation of the coal stream;
- Experiment measurements of calorific value by TNC+NIS analyzer at Kendal power plant of ESCOM electric company, SAR, shows a very high correspondence of measured data to real fuel characteristics [4];
- System of elemental management based on TNC+NIS analyzer can provide operational plant with forecast of energy production for a few days/hours in advance.

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# Managing Efficiency from a Coal Mine/Preparation Plant Complex


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**Abstract**

Profitability at any coal mine is a function of the efficiency of the operation, maximizing yield will always maximize profits. There is a lot of focus on the efficient operation of a preparation plant and determining its optimum operational point, maximum profit point. This involves two steps, plant testing to measure and set up of the plant to run efficiently, then based on individual cleaning unit performance, the raw coal washability and the sales specifications, defining at what gravities each individual circuit should operate at to get the maximum yield for a given quality specification.

This paper will address each of these issues and then discuss the type of automated control concepts that could be developed and implemented at a mine and plant to insure gains made are maintained over time to keep the complex running at its optimum profit point

This paper will mainly deal with the issues of test plant.



**Keywords**

Coal wash plant, test plant, washability, capacity, yield, plant flow sheet, plant samling, plant availability, flow sheet efficiency, equipment efficiency.





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### Yield and Mine Profitability

In mining and coal processing most of the costs incurred are related to the raw coal tonnages produced. The two exceptions are the costs to handle the clean coal product and its shipment from the mine and the cost of handling the reject material. The coal sold is the clean coal produced by the plant, and the clean coal cost for the mine mine's total costs divided by the clean coal tons shipped.

If it costs $25/raw ton to mine and deliver the raw coal to the plant, $3.50/raw ton to wash the coal, $1.50 to ship the clean coal and $2.50 to store the rejects and the plant had a 65% yield, then the cost per clean ton in the wagon would be:

| | | | Costs |
|---|---|---|---|
| Raw Tons per year | 5,000,000 | tons | $142,500,000.00 |
| Plant Yield | 65% | | |
| Clean coal tons shipped | 3,250,000 | clean coal tons | $4,875,000.00 |
| Reject tons | 1,750,000 | reject tons | $4,375,000.00 |
| | | Total Costs | $151,750,000.00 |
| | | Cost/CC ton | $46.69 |

On a clean coal selling price of $50 per ton, there is a profit of $3.31 per ton sold.

If the plant yield could be improved by 2% from 65% yield to 67% yield the clean coal cost at the mine would improve to:

| Assumptions | Base Case | | | | With Yield Increase | | | |
|---|---|---|---|---|---|---|---|---|
| Mining Costs | $25.00 | per raw ton | | | $25.00 | per raw ton | | |
| Coal Preparation Costs | $3.50 | per raw ton | | | $3.50 | per raw ton | | |
| Clean coal loading Costs | $1.50 | per clean ton | | | $1.50 | per clean ton | | |
| Reject handling Costs | $2.50 | per reject ton | | | $2.50 | per reject ton | | |
| | | | Costs | | | | Costs | |
| Raw Tons per year | 5,000,000 | tons | $142,500,000.00 | | 5,000,000 | tons | $142,500,000.00 | |
| Plant Yield | 65% | | | | 67% | | | |
| Clean coal tons shipped | 3,250,000 | clean coal tons | $4,875,000.00 | | 3,350,000 | clean coal tons | $5,025,000.00 | |
| Reject tons | 1,750,000 | reject tons | $4,375,000.00 | | 1,650,000 | reject tons | $4,125,000.00 | |
| | | Total Costs | $151,750,000.00 | | | Total Costs | $151,650,000.00 | |
| | | Cost/CC ton | $46.69 | | | Cost/CC ton | $45.27 | |
| | | | | | | **Cost/ cc ton savings** | **$1.42** | |
| Selling price | $50.00 | per clean ton | | | $50.00 | per clean ton | | |
| Mine profit | $10,750,000.00 | | | | $15,850,000.00 | | | |
| | | | **Additional Mine Profit** | | **$5,100,000.00** | | | |

This represents a clean coal cost reduction of $1.42 per ton. For a 5 million raw ton per year mine this would also result in 100,000 additional tons sold which results in a total increase in mine profits from $10,750,000 per year to $15,850,000 per year an increase in profit of $5,100,000 per year for a 2% yield gain at the plant.

Managing the yield at a mine is critical for managing profitability. While the example shows the increase in profits possible from improving a plants yield, losing yield because the mine does not have control of its mining operation will also severely impact.

While this paper concentrates on measuring and improving the yield of a preparation plant, never forget that the mine controls the possible plant yield by how well out-of-seam dilution is controlled in the mining operation. Out-of Seam dilution reduces the plant yield. If the out-of-seam dilution increases or is more than absolutely necessary is there will be fewer clean coal tons to sell. (Normally) A wash plant has a fix raw ton capacity per year; therefore as out-of-seam dilution increases the plant runs the same number of raw tons per year and produces fewer clean coal tons at an increased clean coal cost. Often clean coal costs can be improved if more attention is paid to minimizing out-of-seam dilution in the mining process.

**Testing a Coal Wash Plant**

For Russian (preparation) coal preparation plants some main problems are:

Working with unstable feed. Traditionally in Russia, coal wash preparation plants were desinged a coals with specific washability characteristics. But the coal preparation plants actually have to operate washing different coals and the plant feed day to day can be variable. For some coals the preparation plants can be ineffective when washing some kinds of coal.

Working at increased capacity. Preparation plants are under a lot of pressure to run with high availability to process all the raw tons the mine produces. As mine capacity will increase with mining equipment upgrades, the plant capacity normally remains unchanged. This will lead to the plant operating over its rated capacity, or over its most efficient feed rate, to keep up with the mine. This will reduce a plant’s efficiency.

Using inefficient equipment. Some preparation coal plants are using old inefficient equipment. The plant therefore operates less efficiently than it could with modern equipment. But without plant test it's impossible to find and develop the justification for updating plant equipment.

Low degree of plant availability. Plant availability is also important therefore the plant operational personnel are focused on maintenance issues. The focus is to increase availability and reduce maintenance costs while pushing tons to keep up with the mine. Both of these factors can result in actions being taken that will reduce the efficiency and hence the yield of the plant. Not because these actions are being taken to intentionally reduce yield but because tons run, plant availability and plant supply costs are easily measured but their affect on plant yield and hence clean coal costs cannot be easily seen.

For example miss place material is one of the sources of yield loss that is tied to maintenance of screens and centrifuges. There is always pressure on a plant to save material costs. One way to do this is to not change out screen panels as often or let centrifuge baskets operate longer. Both show up a savings in material expenditures quickly, but the loss of coal from coarse material getting through the screens or the percent solids in the centrifuge effluent increasing with basket wear are not as easily seen.

It is important that the economics of screen and centrifuge basket wear are understood. If the amount of coal lost because the screen panel or centrifuge basket was not replaced is greater in value than the cost of the screen or centrifuge panel then the perceived saving in replacement cost is more than offset by coal the coal losses that result.

Therefore plant’s efficiency is depends the flow sheet, the efficient capacity of the equipment, equipment's efficiency and degree of plant availability.

The only way to determine a plant’s efficiency is to test the plant.

Therefore it is important that coal wash plants are tested on a routine basis and that performance monitoring is routinely performed to assist in maintaining the plant’s efficiency.

Traditionally, test plant has seven steps.

– Step 1. Design of the plant test.

- Step 2 Check the test plan.
- Step 3 Finalize the Test Plant.
- Step 4 The Plant Sampling.
- Step 5 Laboratory work.
- Step 6 Test Data Analysis.

**Design of the plant test**

It is important to take time to properly design a plant test to insure all the information needed to evaluate the plant is collected. The test is expensive therefore the time to properly design the test is very important.

The test design also defines what will be learned from the test and it is important that the testing team and the plant client agree and have a common understanding of what results the test will produce.

A properly designed plant test will not only measure the cleaning efficiency of the coal washing units in the plant, it will also measure are they being fed correctly, screen efficiencies, reagent consumption, instrument accuracy, how the load is distributed in the plant, dewatering performance and will even provide the data needed to accurately balance the flow rates in each circuit. The results will not only be used to define the efficiency of operation but will also provide the data to identify how to correct problems or otherwise improve performance and/or operating costs.

Typical results from a well designed plant test:

- The efficiency of the cleaning equipment
  - Unit performance by size fraction
    - Additionally the process issues that determine cleaning efficiency
      - Feed rate (loading) and distribution
      - Media to coal ratios
      - Actual feed sizes
      - Moisture or percent solids in feed – optimum for unit?
- The screen efficiency
  - Where and how much material is being misplaced
- Product moistures
  - From screens, centrifuges, and filters.
- Magnetite and other reagent consumption
  - Losses in the Magnetic separators
  - Losses on coal and reject products
  - Froth cell chemical dosage and optimization
  - Flocculent dosage and optimization
- Coal flow rates in each circuit/unit (t/h) allow actual balance of flow sheet.
  - Allows unit capacity issues to be examined.
- Instrument performance

At this step it's important to have a balanced flow sheet for the plant. If a balance flow sheet is not available one will have to be developed. It's necessary for preliminary evaluation of the effectiveness of plant.

A balanced flow sheet for the plant is used to:

- To select sampling points to meet the objectives of the test
- To estimate the amount of sample that will be collected at each sample point.
  - To insure that enough sample containers are brought to the test so all the sample can be collected.

  - In each sample is there enough material mass collected to allow the laboratory work to be done.
  - Based on what will be measured in the field, what equipment is needed to support the sampling?
- Highlight the additional data that should be collected during the test to get maximum value.
  - Depths of material on screen decks
  - Actual media densities
  - Feed pressures
  - Flow rates

Also at this step it is important choose coal which will feed of plant during test. Traditionally, It is coal which is main part of feed of plant and will be the same in the near future (base coal). But if plant works with five and more coals this is not enough. In this situation it is recommended to make a comparison base coal and other coals on the main indicators using results of washability tests. Example is in table 3. Table 3 shows that coal #3 has Mean Difference more 25%. It is recommended in this situation to do special plant test for this coal.

Table 2 shows the detail that must be developed for each sample point to insure that each sample collected has sufficient material for the testing required and that the plan includes sufficient sample storage for each sample.

Table 3 shows the man power determined to manage the plant sampling. The sampling equipment and necessary sample storage containers for each sample point are also show.

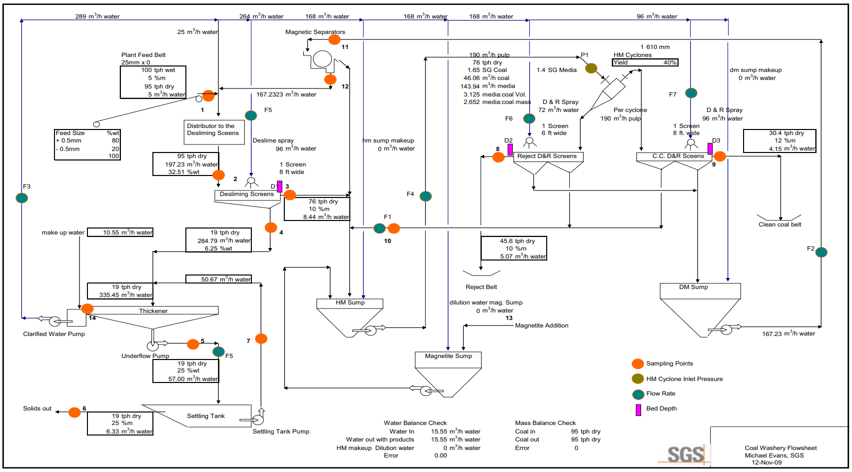


Figure 1 Example of a wash plant flow sheet with sampling points identified.

Table 1 Example of the Estimated sample mass that will be collected from each sampling location.

| Size | Index | Process | Base coal | Coal #1 | | Coal #2 | | Coal #3 | |
|---|---|---|---|---|---|---|---|---|---|
| | | | Value of index | Value of index | Differnce | Value of index | Differnce | Value of index | Differnce |
| | Ash Raw Coal | | 20.1 | 15.0 | 25.6 | 21.0 | 3.9 | 8.7 | 56.8 |
| .+13mm | Yield | Jig D50=1.6/1.8 | 39.4 | 43.9 | 10.2 | 22.1 | 43.9 | 25.4 | 35.5 |
| | Ash | | 22.2 | 16.2 | 27.2 | 22.7 | 2.1 | 8.0 | 64.0 |
| | Yield +1.6 SG | | 31.2 | 38.6 | 19.2 | 16.6 | 46.8 | 23.7 | 24.0 |
| | Ash +1.6SG | | 9.1 | 12.0 | 24.3 | 5.1 | 43.7 | 4.2 | 53.7 |
| | Yield 1.6-1.8 SG | | 1.2 | 3.5 | 64.3 | 0.3 | 73.1 | 0.5 | 61.2 |
| | Ash 1.6-1.8 SG | | 30.7 | 33.2 | 7.4 | 38.4 | 19.9 | 31.3 | 2.0 |
| | Yield-1.8 SG | | 6.9 | 1.8 | 74.2 | 5.1 | 25.9 | 1.2 | 82.6 |
| | Ash -1.8 SG | | 79.8 | 72.3 | 9.4 | 78.6 | 1.5 | 73.1 | 8.4 |
| .1-13mm | Yield | Jig D50=1.6/1.8 | 35.6 | 25.1 | 29.5 | 42.3 | 16.0 | 32.5 | 8.7 |
| | Ash | | 14.9 | 14.3 | 4.1 | 19.5 | 23.3 | 8.3 | 44.4 |
| | Yield +1.6 SG | | 31.3 | 22.2 | 29.0 | 33.9 | 7.6 | 30.3 | 3.4 |
| | Ash +1.6SG | | 8.0 | 8.6 | 6.9 | 6.0 | 25.1 | 3.7 | 53.5 |
| | Yield 1.6-1.8 SG | | 1.1 | 0.9 | 17.5 | 1.0 | 8.6 | 0.3 | 73.5 |
| | Ash 1.6-1.8 SG | | 36.0 | 39.6 | 9.2 | 43.4 | 17.2 | 31.9 | 11.3 |
| | Yield-1.8 SG | | 3.2 | 1.9 | 39.2 | 7.4 | 57.6 | 1.9 | 38.8 |
| | Ash -1.8 SG | | 76.4 | 68.7 | 10.1 | 77.6 | 1.5 | 76.6 | 0.3 |
| 0.5-1mm | Yield | Spirals D50=1.8 | 8.2 | 7.4 | 10.0 | 11.5 | 28.8 | 10.7 | 23.1 |
| | Ash | | 17.2 | 15.2 | 11.7 | 19.9 | 13.7 | 8.2 | 52.1 |
| | Yield +1.8 SG | | 7.2 | 6.7 | 6.8 | 9.4 | 23.5 | 10.0 | 28.3 |
| | Ash +1.8 SG | | 9.0 | 9.8 | 8.7 | 7.1 | 21.0 | 4.0 | 55.4 |
| | Yield-1.8 SG | | 1.0 | 0.7 | 32.9 | 2.1 | 52.3 | 0.6 | 37.9 |
| | Ash -1.8 SG | | 76.4 | 68.7 | 10.1 | 77.6 | 1.5 | 76.6 | 0.3 |
| 0-0.5mm | Yield | Froth Flotation | 16.8 | 23.6 | 28.8 | 24.1 | 30.1 | 31.4 | 46.5 |
| | Ash | | 27.7 | 13.4 | 51.6 | 22.5 | 18.7 | 9.8 | 64.4 |
| | Yield Froth | | 9.0 | 21.1 | 57.6 | 17.7 | 49.3 | 19.3 | 53.6 |
| | Ash Froth | | 6.9 | 8.2 | 15.9 | 10.4 | 33.7 | 5.9 | 14.5 |
| | Yield Tailings | | 7.9 | 2.5 | 68.0 | 6.4 | 18.7 | 12.1 | 35.2 |
| | Ash Tailings | | 51.3 | 57.0 | 10.0 | 55.8 | 8.1 | 16.1 | 68.6 |
| Total Differnce | | | | | 719.3 | | 717.1 | | 1101.7 |
| Mean Differnce | | | | | 24.8 | | 24.7 | | 38.0 |

Table 2 Example of the Estimated sample mass that will be collected from each sampling location

| | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Example of Sample Design Spread Sheet | | | | | | | | | | | | |
| Metric | | | | | | | | | | | | |
| Stop Belt Samples | | | | | | | | | | | Final | Final |
| | tph | Belt Speed | Cutter Opening | | | Cut Mass | | Number of cuts | Number of | Density | Sample mass | Sample Volume |
| | Metric ton | m/s | mm | | | kg | | Per increment | Increments | $kg/m^3$ | kg | m3 |
| Plant Feed Belt | 1000 | 3 | 300 | | | 27.78 | | 1 | 16 | 800 | 444.44 | 0.556 |
| Reject Belt | 425 | 2.3 | 300 | | | 15.40 | | 1 | 16 | 800 | 246.38 | 0.308 |
| Falling Stream Samples (cut across entire stream) | | | | | | | | | | | | |
| | tph | Cutter speed | Cutter Opening | | | | | | | | | |
| | Metric ton | m/s | mm | | | | | | | | | |
| Prewet screen | 225 | 0.6 | 150 | | | 15.63 | | 1 | 16 | 800 | 250.00 | 0.313 |
| CC D&R Screen | 150 | 0.6 | 150 | | | 10.42 | | 1 | 16 | 800 | 166.67 | 0.208 |
| Special case falling stream random collection within a flowing stream | | | | | | | | | | | | |
| | tph | Cutter length | Cutter Opening | Area of chute | Time in flow | | | | | | | |
| | Metric ton | mm | mm | $m^2$ | seconds | | | | | | | |
| Centrifuge product | 40 | 300 | 40 | 0.78 | 5 | 0.85 | | 1 | 16 | 600 | 13.68 | 0.023 |
| Falling Stream Slurry Samples (can sample entire stream) | | | | | | | | | | | | |
| | | | | | | Increment | | | | | Solids | Final |
| | Flow | Cutter speed | Cutter Opening | | | Volume | | | | | Sample mass | Sample Volume |
| | $m^3/hr$ | m/s | mm | % Solids | SG of Solids | liters | SG of Slurry | | | | kg | m3 |
| Froth Product | 196 | 0.4 | 50 | 22 | 1.4 | 6.806 | 1.067 | 1 | 16 | | 25.562 | 0.109 |
| Centrifuge Effluent | 10 | 0.2 | 50 | 13 | 1.4 | 0.694 | 1.039 | 1 | 16 | | 1.500 | 0.011 |
| | | | | | | Increment | | | | | Solids | Final |
| | | | | | | Volume | | | | | Sample mass | Sample Volume |
| Fixed volume slurry sample | | | | % Solids | SG of Solids | liters | SG of Slurry | | | | kg | m3 |
| Froth Cell Feed | | | | 8.2 | 1.45 | 1 | 1.026 | 1 | 16 | | 1.346 | 0.016 |
| Froth Cell Tails | | | | 3 | 1.8 | 1 | 1.014 | 1 | 16 | | 0.486 | 0.016 |

Table 3 Example of Manpower determination with sampling device and sample storage requirements for each sample location

| Sample Name | | | | |
|---|---|---|---|---|
| | # People | | Sampling Device | Sample Storage |
| Plant feed from automatic sampler | 1 | | none collected with plant mechanical sampler | Heavy plastic bags |
| Thickener Underflow - This will be a sample that is very wet! | | | Small bucket with a top with a rectangular opening | Plastic buckets with sealed tops |
| | | | | |
| Prim. HM Cyclone Quad Group 1 Four Desliming Screens overflows | 1 | | Standard screen overflow sampler | Heavy plastic bags |
| Coarse Coal Centrifuge 1 Feed two desliming screen overflows | | | | Heavy plastic bags |
| Coarse Coal Centrifuge 2 Feed two desliming screen overflows | | | | Heavy plastic bags |
| | | | | |
| Prim. HM Cyclone Quad Group 2 Four Desliming Screens overflows | 1 | | Standard screen overflow sampler | Heavy plastic bags |
| Coarse Coal Centrifuge 3 Feed two desliming screen overflows | | | | Heavy plastic bags |
| Coarse Coal Centrifuge 4 Feed 2 desliming screen overflows | | | | Heavy plastic bags |
| | | | | |
| Prim. HM Cyclone Quad Group 1 CC D&R Screens overflow | 1 | | Standard screen overflow sampler | Heavy plastic bags |
| Prim. HM Cyclone Quad Group 2 CC D&R Screens overflow | | | | Heavy plastic bags |
| Mid. HM Cyclone Dual Group Middlings D&R Screen overflow | | | | Heavy plastic bags |
| Mid. HM Cyclone Dual Group Mid. Rejects D&R Screen overflow | | | | Heavy plastic bags |
| | | | | |
| Prim. HM Cyclone Quad Group 1 Reject Sieve Bend overflow | 1 | | Standard screen overflow sampler | Heavy plastic bags |
| Prim. HM Cyclone Quad Group 2 Reject Sieve Bend overflow | | | | Heavy plastic bags |
| Froth Cell Feed to Conditioning Tank 1 | | | Thief | Plastic buckets with sealed tops |
| Froth Cell Feed to Conditioning Tank 2 | | | | Plastic buckets with sealed tops |
| | | | | |
| Coarse Coal Centrifuge 1 Product | 1 | | Standard Screen overflow sampler | Heavy plastic bags |
| Coarse Coal Centrifuge 2 Product | | | | Heavy plastic bags |
| Coarse Coal Centrifuge 3 Product | | | | Heavy plastic bags |
| Coarse Coal Centrifuge 4 Product | | | | Heavy plastic bags |
| | | | | |
| Coarse Coal Centrifuge 1 Effluent and measure and record flow rate | 1 | | Small bucket capable of capturing all the flow and | Plastic buckets with sealed tops |
| Coarse Coal Centrifuge 2 Effluent and measure and record flow rate | | | a one liter container for measuring flow rate | Plastic buckets with sealed tops |
| Coarse Coal Centrifuge 3 Effluent and measure and record flow rate | | | | Plastic buckets with sealed tops |
| Coarse Coal Centrifuge 4 Effluent and measure and record flow rate | | | | Plastic buckets with sealed tops |
| | | | | |
| Froth Cell 1 product sample [and screen bowl feed] | 2 | | Froth sampler | Plastic buckets with sealed tops |
| Froth Cell 2 product sample [and screen bowl feed] | | | | Plastic buckets with sealed tops |
| Froth Cell 3 product sample [and screen bowl feed] | | | | Plastic buckets with sealed tops |
| Froth Cell 4 product sample NOT EXPECTED TO BE RUN | | | | Plastic buckets with sealed tops |
| Froth Cell 1 Tailings sample | | | Thief with a long handle | Plastic buckets with sealed tops |
| Froth Cell 2 Tailings sample | | | | Plastic buckets with sealed tops |
| Froth Cell 3 Tailings sample | | | | Plastic buckets with sealed tops |
| Froth Cell 4 Tailings sample NOT EXPECTED TO BE RUN | | | | Plastic buckets with sealed tops |

### Check the test plan

It is critical that with a desired plant testing plan developed that the plant is inspected and each sampling point examined to determine can it be sampled safely, what equipment is needed to do the sampling, are plant modifications required to allow sampling to occur and are some samples impossible (test objectives may have to be modified). This plant inspection is required before the test plan is finalized.

Part of the plant inspection will very likely include collection of some test samples to validate the sample mass that will be collected. Part of this validation process is to insure there is enough sample mass for the NTS of the sample for sizing, washability and analysis work that is required.

### Finalize the Test Plan

With the test plan finalized after the plant inspection, it is critical that the testing agency and the wash plant owner meet to reach as common understanding of what the final results will include. And what work each is responsible for completing prior to the sampling date to allow the test to happen. This might affect the actual sampling date selected as there maybe work the plant has to perform to make it possible to sample some locations.

With the test design finalized the testing company can finalize the proposal to supply the defined testing service. With test locations determined the number of technicians required to sample the plant can be determined. The types of sampling equipment needed determined. The mass of the sample from each sample can be finalized. And finally the lab with the sample masses and the testing and analysis required for each sample can determine the laboratory costs and the time required to process all of the samples.

**The Plant Sampling**

Conducting the plant sampling is no small matter. There are often a large number of samplers. They must be safety trained, trained to take proper samples at their sample locations, trained to properly label the samples and seal them. There is a lot of sampling equipment and sample containers required. It is not uncommon for a plant test to generate 10 tons or more of sample that must be transported from the plant to the lab.

Normally sampling done to measure plant efficiency is done in one day. But it requires two days of site time. Day one for site specific safety training of the sampling team, sample set up (distribution of all the sampling equipment and sample containers) and finally training each sample on to properly take the samples they are responsible for and what other data they are to collect or measure during each sampling interval.

On day two the plant sampling is done. SGS recommends a minimum of 16 individual samples be collected and combined from each sampling point. Therefore, if the sampling is done on a 15 minute schedule, it would take four hours of continuous plant operation to collect the 16 increments.

The plant should be operated as it normally operated same feed rate, density settings, etc. The plant should be in steady state; this normally requires 45 minutes of continuous operation. This means any time the plant goes down during sampling, sampling must be suspended and not restarted until the plant has been operating 45 minutes. This can greatly affect the actual time it takes to collect the samples.

With sampling completed, the samples have to be checked that they are all correctly labeled and sealed. They have to be inventoried by sample location before they are moved. Then each sample must be removed from the plant and placed on transport to the lab.

Typically the efficiency sampling requires a long day and a lot of physical labor.

There are often samples that will be collected that cannot be done while the plant efficiency testing is underway. Raw Coal Plant feed samples normally require stop belt sampling to obtain good samples. In addition there may be a need to sample multiple coal sources to allow evaluation of their washing characteristsics based on the test work. As the plant should not be stopped while the efficiency testing is being done, it is normal for these samples to be collected on the day after the plant is sampled. This work typically requires a much smaller site sampling crew.

**Laboratory work**

Traditionally,for all samples the following is required:

- Determination of Moisture or % Solids for each sample;
- Drain and Rinse screen overflow samples, rinsed and measure magnetite mass;
- Samples sized in size fractions relevant to plant process circuits;
- Float and sink the size fractions plus 0.15 mm for all washing equipment;
- The minus 0.15 mm sized and tested in froth cells if relevant to the circuit or future plans;
- On all size and gravity fractions determine at minimum the dry ash;
    - On plant feed samples also perform sulfur and GCV analysis on all gravity fractions.

At this stage for plants which have unstable feed SGS recommends to work out special laboratory test. It's necessary because testing plant during preparation each coal it is very expensive. Special laboratory test includes laboratory wet tumbling test, filtration and froth flotation tests and uses correlation between actual results and laboratory results. This test can help to calculate a balanced flow sheet for all coals using only results of laboratory tests.

**Test Data Analysis**

Results of test contains:

- calculation of balanced flow sheet using results of plant test;
- analysis of flow sheet efficiency;
- analysis of equipment efficiency;

The updated process flow sheet plant will shows the performance of all equipment and show problem points.

The process flow sheet will show the following indicators

- theoretical yield for feed plant (Yt)
- calculation yield for real flow sheet if equipment operating at normal efficiency (Yc)
- actual yield according to plant test results (Ya)

Usually for plant with good efficiency will have Yc/Yt greater than 0.95 and for plant which work in the normal mode attitude Ya/Yc is more 0.95, too.

Table 4 Example calculation of flow sheet efficiency.

| | | Theoretical balance | | Calculation balance | | | Actual balance | |
|---|---|---|---|---|---|---|---|---|
| Size | Product | Yield | Ash | Process | Yield | Ash | Yield | Ash |
| .+13mm | Clean Coal | 20.2 | 9.7 | HM Vessel D50=1.5/1.8 | 19.8 | 9.3 | | |
| | Middlings | 0.4 | 30.2 | | 0.8 | 29.0 | | |
| | Rejects | 5.8 | 80.7 | | 5.8 | 80.7 | | |
| .1-3mm | Clean Coal | 49.8 | 8.2 | Jig D50=1.5/1.8 | 47.5 | 7.2 | | |
| | Middlings | 1.8 | 35.4 | | 4.1 | 31.3 | | |
| | Rejects | 5.2 | 76.5 | | 5.2 | 76.5 | | |
| .-0.5mm | Clean Coal | 11.8 | 8.0 | Froth Flotation | 9.0 | 6.9 | | |
| | Middlings | 0.4 | 36.0 | | | | | |
| | Rejects | 4.7 | 76.4 | | 7.9 | 51.3 | | |
| | Total Clean Coal | 81.7 | 8.5 | | 76.2 | 7.7 | 67.7 | 8.0 |
| | Total Middlings | 2.6 | 34.8 | | 4.9 | 30.9 | 10.3 | 20.7 |
| | Total Rejects | 15.7 | 78.1 | | 18.9 | 67.3 | 22.0 | 57.3 |
| | Total Raw Coal | 100.0 | 20.1 | | 100.0 | 20.1 | 100.0 | 20.1 |
| | **Yc/Yt** | **0.93** | | | | | | |
| | **Ya/Yc** | **0.89** | | | | | | |

It is important to determine the best D50 point for each circuit. One rule of thumb is to run all the circuits at the same gravity. And often this is the right solution. But it is coal dependent and is controlled by the coal washability in each size fraction washed and the percentage of the feed in each size fraction.

The following example was published in [5].

A washability simulation on this data targeting a 7.00 % dry clean coal ash, shows the best yield point for the plant is obtained using a D50 of 1.55 for both the HM Bath and the HM Cyclone, see Table 6.

But if the washability stayed the same and the percentage of the feed in each of these units changed the best operating point results in a different answer.

If the HM Bath washes 350 t/h of the plant feed and the HM Cyclone circuit 200 t/h, the maximum yield at 7% dry ash is obtained washing the HM Bath at 1.35 and the HM Cyclone at 1.80 because the changes in incremental ash as the separation gravity increases are smaller in the HM Cyclone circuit.

With good washability data on the coal feed by size fraction and the actual distribution curves for the different circuits obtained in a plant test, it is possible to develop the best separation gravities to produce the maximum yield at any required plant product quality.

SGS has also used the following methodology to describe the optimum D50 for a HM Bath and HM Cyclone plant. It shows that there are many combinations that work. This work was done over a narrow gravity range based on an initial determination and is shown in Tables 8,9. Note the analysis requires an acceptable ± range to be established around the target product ash.

Table 5 Ooptimum yield raw coal washability data

| | | 4" x 1/4" | % of Feed | **150.00** | tph | 1/4" x 1 mm | % of Feed | **350.00** | tph |
|---|---|---|---|---|---|---|---|---|---|
| | | Heavy Media Bath | | Cum | Cum | Heavy Media Cyclone | | Cum | Cum |
| Sink | Float | Mass (%) | Ash (%) | Mass (%) | Ash (%) | Mass (%) | Ash (%) | Mass (%) | Ash (%) |
| Float | 1.30 | 11.4 | 3.9 | 11.4 | 3.9 | 59.2 | 3.5 | 59.2 | 3.5 |
| 1.30 | 1.35 | 27.5 | 8.3 | 38.9 | 7.0 | 6.1 | 7.6 | 65.3 | 3.9 |
| 1.35 | 1.40 | 16.3 | 13.7 | 55.2 | 9.0 | 3.7 | 13.1 | 69.0 | 4.4 |
| 1.40 | 1.45 | 3.8 | 20.1 | 59.0 | 9.7 | 2.3 | 18.5 | 71.3 | 4.8 |
| 1.45 | 1.50 | 2.5 | 25.8 | 61.5 | 10.4 | 1.9 | 23.8 | 73.2 | 5.3 |
| 1.50 | 1.55 | 1.3 | 29.7 | 62.8 | 10.8 | 1.0 | 29.0 | 74.2 | 5.6 |
| 1.55 | 1.60 | 0.8 | 33.9 | 63.6 | 11.0 | 0.9 | 33.2 | 75.1 | 6.0 |
| 1.60 | 1.65 | 0.3 | 37.1 | 63.9 | 11.2 | 0.5 | 37.2 | 75.6 | 6.2 |
| 1.65 | 1.70 | 0.3 | 40.1 | 64.2 | 11.3 | 0.6 | 40.4 | 76.2 | 6.5 |
| 1.70 | 1.80 | 0.4 | 45.0 | 64.6 | 11.5 | 1.1 | 44.5 | 77.3 | 7.0 |
| 1.80 | 1.90 | 0.6 | 52.7 | 65.2 | 11.9 | 1.0 | 51.6 | 78.3 | 7.6 |
| 1.90 | 2.00 | 0.5 | 62.1 | 65.7 | 12.3 | 1.0 | 60.8 | 79.3 | 8.2 |
| 2.00 | Sink | 34.3 | 88.5 | 100.0 | 38.4 | 20.7 | 87.2 | 100.0 | 24.6 |
| | | | | | | | | | |
| Totals | | 100.0 | 38.4 | | | 100.0 | 24.6 | | |

Table 6 Results of all possible wash cases with 150 t/h in HM Bath feed and 350 t/h in the HM Cyclone feed.

| **All Cases** | | | | | Yield at | |
|---|---|---|---|---|---|---|
| HM Bath | HM Cylone | % Ash | Yield | Ash OK | right ash | Best |
| 1.35 | 1.80 | 7.00 | 65.78 | Yes | 65.78 | |
| 1.40 | 1.70 | 7.05 | 69.90 | | | |
| 1.40 | 1.65 | 6.85 | 69.48 | | | |
| 1.40 | 1.60 | 6.70 | 69.13 | | | |
| 1.40 | 1.55 | 6.45 | 68.50 | | | |
| 1.40 | 1.50 | 6.22 | 67.80 | | | |
| 1.40 | 1.45 | 5.87 | 66.47 | | | |
| 1.40 | 1.40 | 5.55 | 64.86 | | | |
| 1.40 | 1.35 | 5.24 | 62.27 | | | |
| 1.40 | 1.30 | 5.07 | 58.00 | | | |
| 1.45 | 1.70 | 7.26 | 71.04 | | | |
| 1.45 | 1.65 | 7.06 | 70.62 | | | |
| 1.45 | 1.60 | 6.91 | 70.27 | | | |
| 1.45 | 1.55 | 6.68 | 69.64 | | | |
| 1.45 | 1.50 | 6.45 | 68.94 | | | |
| 1.45 | 1.45 | 6.11 | 67.61 | | | |
| 1.45 | 1.40 | 5.81 | 66.00 | | | |
| 1.45 | 1.35 | 5.51 | 63.41 | | | |
| 1.45 | 1.30 | 5.36 | 59.14 | | | |
| 1.50 | 1.70 | 7.45 | 71.79 | | | |
| 1.50 | 1.65 | 7.26 | 71.37 | | | |
| 1.50 | 1.60 | 7.11 | 71.02 | | | |
| 1.50 | 1.55 | 6.88 | 70.39 | | | |
| 1.50 | 1.50 | 6.66 | 69.69 | | | |
| 1.50 | 1.45 | 6.32 | 68.36 | | | |
| 1.50 | 1.40 | 6.03 | 66.75 | | | |
| 1.50 | 1.35 | 5.74 | 64.16 | | | |
| 1.50 | 1.30 | 5.61 | 59.89 | | | |
| 1.55 | 1.70 | 7.57 | 72.18 | | | |
| 1.55 | 1.65 | 7.38 | 71.76 | | | |
| 1.55 | 1.60 | 7.24 | 71.41 | | | |
| 1.55 | 1.55 | 7.00 | 70.78 | Yes | 70.78 | Best SG Mix Max Yield |
| 1.55 | 1.50 | 6.79 | 70.08 | | | |
| 1.55 | 1.45 | 6.46 | 68.75 | | | |
| 1.55 | 1.40 | 6.17 | 67.14 | | | |
| 1.55 | 1.35 | 5.89 | 64.55 | | | |
| 1.55 | 1.30 | 5.77 | 60.28 | | | |

Table 7 Results of all possible washing combinations with 350 t/h feeding the HM bath and 200 t/h feeding the HM Cyclone.

| **All Cases** | | | | | Yield at | |
|---|---|---|---|---|---|---|
| HM Bath | HM Cylone | % Ash | Yield | Ash OK | right ash | Best |
| 1.35 | 1.80 | 7.00 | 52.86 | Yes | 52.86 | Best SG Mix Max Yield |
| 1.40 | 1.70 | 7.87 | 62.84 | | | |
| 1.40 | 1.65 | 7.75 | 62.62 | | | |
| 1.40 | 1.60 | 7.67 | 62.44 | | | |
| 1.40 | 1.55 | 7.53 | 62.11 | | | |
| 1.40 | 1.50 | 7.41 | 61.75 | | | |
| 1.40 | 1.45 | 7.22 | 61.05 | | | |
| 1.40 | 1.40 | 7.07 | 60.22 | | | |
| 1.40 | 1.35 | 6.93 | 58.87 | | | |
| 1.40 | 1.30 | 6.90 | 56.65 | | | |
| 1.45 | 1.70 | 8.32 | 65.25 | | | |
| 1.45 | 1.65 | 8.21 | 65.04 | | | |
| 1.45 | 1.60 | 8.13 | 64.85 | | | |
| 1.45 | 1.55 | 8.01 | 64.53 | | | |
| 1.45 | 1.50 | 7.89 | 64.16 | | | |
| 1.45 | 1.45 | 7.71 | 63.47 | | | |
| 1.45 | 1.40 | 7.57 | 62.64 | | | |
| 1.45 | 1.35 | 7.45 | 61.29 | | | |
| 1.45 | 1.30 | 7.44 | 59.07 | | | |
| 1.50 | 1.70 | 8.74 | 66.85 | | | |
| 1.50 | 1.65 | 8.63 | 66.63 | | | |
| 1.50 | 1.60 | 8.56 | 66.45 | | | |
| 1.50 | 1.55 | 8.43 | 66.12 | | | |
| 1.50 | 1.50 | 8.32 | 65.75 | | | |
| 1.50 | 1.45 | 8.16 | 65.06 | | | |
| 1.50 | 1.40 | 8.02 | 64.23 | | | |
| 1.50 | 1.35 | 7.91 | 62.88 | | | |
| 1.50 | 1.30 | 7.92 | 60.66 | | | |
| 1.55 | 1.70 | 8.99 | 67.67 | | | |
| 1.55 | 1.65 | 8.89 | 67.45 | | | |
| 1.55 | 1.60 | 8.82 | 67.27 | | | |
| 1.55 | 1.55 | 8.70 | 66.95 | | | |
| 1.55 | 1.50 | 8.59 | 66.58 | | | |
| 1.55 | 1.45 | 8.43 | 65.89 | | | |
| 1.55 | 1.40 | 8.30 | 65.05 | | | |
| 1.55 | 1.35 | 8.19 | 63.71 | | | |
| 1.55 | 1.30 | 8.22 | 61.49 | | | |

Table 8 D50 combinations that generate a 7% dry ash product

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | **Target Ash** | **7.00** | **%** | | | | | | | | |
| | Range allowed | 0.02 | % plus or minus | | | | | | | | |
| **Plant Product Ash** | | | | | | | | | | | |
| | | | | | HM Cyclone D50 | | | | | | |
| HM Bath | D50 --> | 1.53 | 1.54 | 1.55 | 1.56 | 1.57 | 1.58 | 1.59 | 1.60 | 1.61 | 1.62 |
| D50 | Ash | 5.48 | 5.55 | 5.61 | 5.68 | 5.74 | 5.80 | 5.86 | 5.92 | 5.97 | 6.03 |
| 1.48 | 10.06 | 6.67 | 6.71 | 6.76 | 6.81 | 6.85 | 6.89 | 6.94 | 6.98 | 7.02 | 7.06 |
| 1.49 | 10.17 | 6.70 | 6.75 | 6.80 | 6.84 | 6.89 | 6.93 | 6.97 | 7.01 | 7.05 | 7.09 |
| 1.50 | 10.28 | 6.74 | 6.79 | 6.83 | 6.88 | 6.92 | 6.96 | 7.01 | 7.05 | 7.09 | 7.12 |
| 1.51 | 10.38 | 6.77 | 6.82 | 6.86 | 6.91 | 6.95 | 6.99 | 7.04 | 7.08 | 7.12 | 7.16 |
| 1.52 | 10.47 | 6.80 | 6.85 | 6.89 | 6.94 | 6.98 | 7.02 | 7.07 | 7.11 | 7.15 | 7.18 |
| 1.53 | 10.55 | 6.83 | 6.87 | 6.92 | 6.96 | 7.01 | 7.05 | 7.09 | 7.13 | 7.17 | 7.21 |
| 1.54 | 10.63 | 6.85 | 6.90 | 6.94 | 6.99 | 7.03 | 7.08 | 7.12 | 7.16 | 7.20 | 7.23 |
| 1.55 | 10.70 | 6.88 | 6.92 | 6.97 | 7.01 | 7.06 | 7.10 | 7.14 | 7.18 | 7.22 | 7.26 |
| 1.56 | 10.77 | 6.90 | 6.94 | 6.99 | 7.03 | 7.08 | 7.12 | 7.16 | 7.20 | 7.24 | 7.28 |
| 1.57 | 10.83 | 6.92 | 6.96 | 7.01 | 7.05 | 7.10 | 7.14 | 7.18 | 7.22 | 7.26 | 7.30 |

Table 9 The Yield at each D50 combination

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Plant Yield Relationship** | | | | | | | | | | | |
| | | | | | HM Cyclone D50 | | | | | | |
| HM Bath | D50 --> | 1.53 | 1.54 | 1.55 | 1.56 | 1.57 | 1.58 | 1.59 | 1.60 | 1.61 | 1.62 |
| D50 | Yield | 73.37 | 73.62 | 73.85 | 74.07 | 74.27 | 74.46 | 74.63 | 74.80 | 74.96 | 75.11 |
| 1.48 | 59.93 | 69.34 | 69.51 | 69.67 | 69.82 | 69.97 | 70.10 | 70.22 | 70.34 | 70.45 | 70.55 |
| 1.49 | 60.45 | 69.50 | 69.67 | 69.83 | 69.98 | 70.12 | 70.25 | 70.38 | 70.49 | 70.60 | 70.71 |
| 1.50 | 60.90 | 69.63 | 69.81 | 69.97 | 70.12 | 70.26 | 70.39 | 70.51 | 70.63 | 70.74 | 70.84 |
| 1.51 | 61.30 | 69.75 | 69.93 | 70.09 | 70.24 | 70.38 | 70.51 | 70.63 | 70.75 | 70.86 | 70.97 |
| 1.52 | 61.66 | 69.86 | 70.03 | 70.19 | 70.34 | 70.49 | 70.62 | 70.74 | 70.86 | 70.97 | 71.07 |
| 1.53 | 61.97 | 69.95 | 70.13 | 70.29 | 70.44 | 70.58 | 70.71 | 70.83 | 70.95 | 71.06 | 71.17 |
| 1.54 | 62.25 | 70.04 | 70.21 | 70.37 | 70.52 | 70.66 | 70.79 | 70.92 | 71.03 | 71.14 | 71.25 |
| 1.55 | 62.50 | 70.11 | 70.28 | 70.45 | 70.60 | 70.74 | 70.87 | 70.99 | 71.11 | 71.22 | 71.32 |
| 1.56 | 62.72 | 70.18 | 70.35 | 70.51 | 70.66 | 70.80 | 70.93 | 71.06 | 71.17 | 71.28 | 71.39 |
| 1.57 | 62.91 | 70.24 | 70.41 | 70.57 | 70.72 | 70.86 | 70.99 | 71.12 | 71.23 | 71.34 | 71.45 |

For analysis clean coal equipment SGS using Distribution Curves (Tromp Curves).

SGS provides experience curves the Green Excellent spline curve and the Blue Good Spline Curve to graphically show how the equipment tested compares to international performance of the same equipment type. The HM Bath example shown above shows for the most part the unit is operating a little better than good, but the reject tail shows more rejects reporting to product (like due to reject particles rafting on the clean coal) than should be expected. This is a yield opportunity as correcting this problem would lower the units product ash at the same 1.673 cut point (D50). Therefore the plant could increase the D50 to gain yield to increase the product ash back to the level it was in this test. This would be a yield increase.

The distribution curves give SGS coal preparation engineers very good insight to how the unit is operating and if problems are seen, the likely causes of the problems will be identified to the plant operators so they can be corrected. The report will graphically show unit performance and provide a technical assessment of any issues identified by the curve.

Misplacing coarse material into a fine circuit is large issue that causes yield loss. Screens and Cyclones are used to classify the feed into the size fractions bested suited for cleaning in each of the different cleaning circuits in the plant. Screen wear or cyclone classification problems can quickly misplace coarser particles into circuits designed to clean finer size fractions. The misplaced coarse coal in these circuits look like refuse and are rejected causing loss of yield in the plant. Testing for the size fractions feeding each circuit is critical to identify if losses are occurring due to screen panels not being replaced in time.

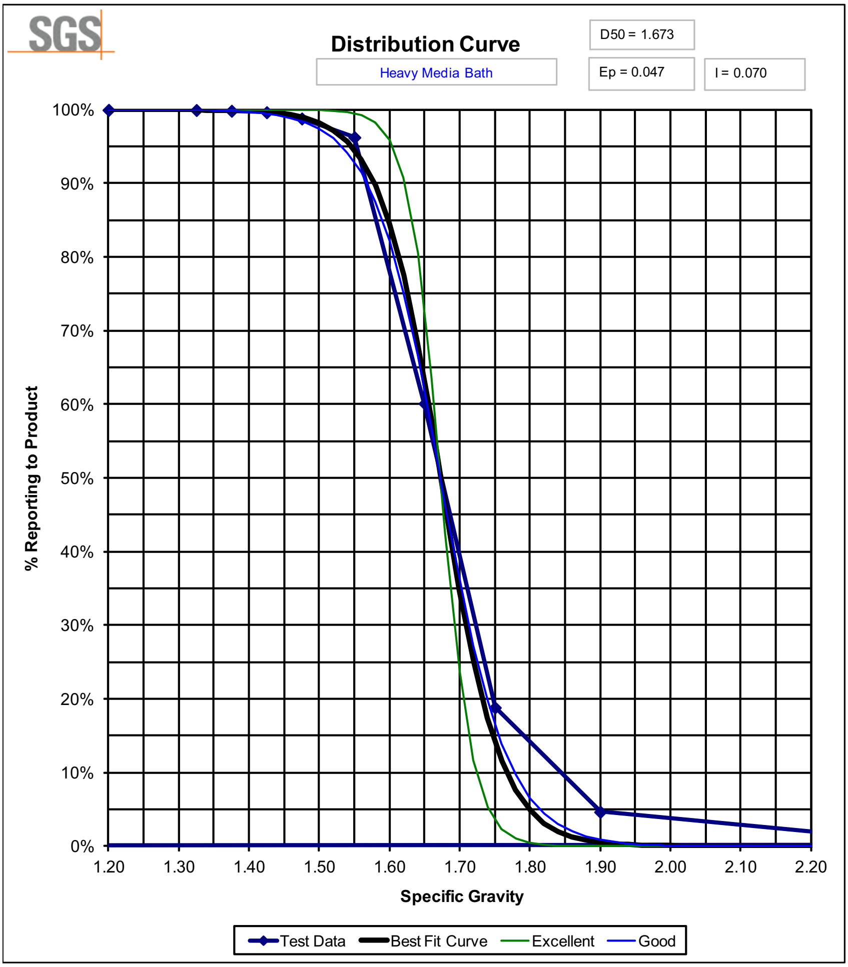


Figure 2, Heavy Media Bath Distribution Curve.

Table 10 shows, the issue can also be the misplacement of fines in the overflow of a screen. In this test 4.52% of the minus 0.5 mm material reported to the screen overflow. This can cause problems like increased magnetite consumption.

Dewatering equipment is also tasted and analyzed, centrifuges and filters. If a centrifuge is tested correctly the feed rate to the unit can be determined, excellent for development as test balanced flow sheet, as well as recovery efficiency and generation of fines within the unit.

In addition testing of all auxiliary plant equipment, Magnetic Separators, Instrumentation, and Thickeners can be analyzed and issues about magnetite loss, over or under use of chemical reagents and effectiveness of the plant instrumentation are all evaluated.

Table 10 Example of Desliming Screen Efficiency Results

| **Desliming screen Efficiency** | | |
|---|---|---|
| f = | 19.29 | %wt. passing 0.5 mm in Feed |
| o = | 4.52 | %wt. passing 0.5 mm in Screen Overflow |
| u = | 93.34 | %wt. passing 0.5 mm in Screen Underflow |
| | | |
| Yu = | 16.63 | Percent to Undersize |
| Yo = | 83.37 | Percent to Oversize |
| | | |
| Eu = | 80.46 | Undersize Efficiency |
| Eo = | 98.63 | Oversize Efficiency |

In summary SGS design the plant tests to identify opportunity to improve yield and costs.

- Are there opportunities to improve plant yield?
  - Are the cleaning units operating as efficiently as possible?
  - Is coal being lost because it is over size for the circuit it is feeding?
    - Saving money on screen panels but throwing away coal?
  - Are there middlings that with crushing would add yield?
  - Is the plant operating at a feed rate that is inefficient?
- Are there instrumentation problems?
  - Calibration problems
  - Instrument failures
  - Opportunity for better operation with new instruments?
- Are there opportunities to reduce reagent costs?
  - Is magnetite being lost that should be captured?
  - Are reagents being used in excess (cost savings) or is additional dosage required to maximize yield.
- Are there circuit issues that if corrected will improve performance?

And if there are opportunities identified to improve a wash plant's profitability SGS can assist in defining the economic justification and the actual modifications required to achieve the highest possible profitability at a coal wash plant.

Table 11 Centrifuge test data, measured data in blue and results calculated.

| | | |
|---|---|---|
| | | Lud |
| | | Pant Leg |
| **Feed** | | |
| Sample # | | 15 |
| % Moisture | | 24.18 |
| Inherent Moisture ASTM D1412 | | 6.23 |
| % Dry Ash +0.5mm | | 30.20 |
| % Dry Ash -0.5mm | | 31.25 |
| % plus 0.5mm | | 79.95 |
| % minus 0.5mm | | 20.05 |
| | | |
| **Product** | | |
| Sample # | | 19 |
| % Moisture | | 4.85 |
| Inherent Moisture ASTM D1412 | | 6.23 |
| % Dry Ash +0.5mm | | 30.00 |
| % Dry Ash -0.5mm | | 39.20 |
| % plus 0.5mm | | 77.64 |
| % minus 0.5mm | | 22.36 |
| | | |
| **Effluent** | | |
| Sample # | | 23 |
| % solids | | 34.19 |
| % Dry Ash +0.5mm | | 31.00 |
| % Dry Ash -0.5mm | | 33.20 |
| % plus 0.5mm | | 48.71 |
| % minus 0.5mm | | 51.29 |
| | | |
| **Results** | | |
| Recovery | | 85.70% |
| Estimated Centrifuge Feed tph | | 48.28 |
| Estimated Product tph | | 41.38 |
| Estimated tph in Effluent Stream | | 6.90 |
| TPH -0.5mm in Feed | | 9.68 |
| TPH -0.5mm in Product | | 9.25 |
| TPH -0.5mm in Effluent | | 3.54 |
| TPH -0.5mm Generated in Centrifuge | | 3.11 |
| Feed Ash | | 30.41 |
| Product Ash | | 32.06 |
| Effluent Ash | | 32.13 |

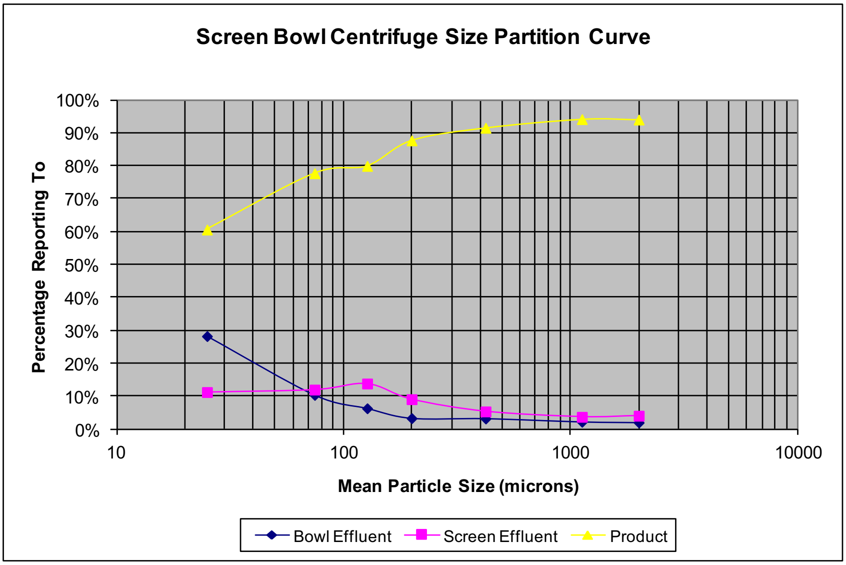


Figure 3 Distribution of material by size to the three product streams of a Screen Bowl Centrifuge.

## On Going Plant Monitoring

With a plant tested and the operational improvements made, it is important that a monitoring plan is put in place to insure the efficiency obtained is maintained.

If plant has unstable feed Test Feed this is very important. Testing Feed of plant requires special attention and special development. It needs to meet the following requirements:

- to require the least time;
- to be adapted to flow sheet of plant;
- to have internal control system.

It is important to have automatic system preparation of results of Test Feed of plant.

Ideally this system should calculate to flow sheet balance of plant and give the following information:

- the best capacities of plant during process of preparation this coal;
- the best D50 point for each circuit;
- the expected performance of coal preparation (yield and quality).

Test Plant can be the first main step to create such a system. But never forget that this system should be based on statistical information from monitoring system of plant. This is the only way for it good work.

Monitoring system of plant should be based on system of monitoring of equipment, on-line analysis of plant products and system of control of capacity of each circuit.

It is important to set up a control sampling program that looks at screen panel wear and centrifuge basket replacement based on percentages of misplaced material that results. When the monitoring shows the levels of misplaced material exceed an economic threshold then the panels or baskets shall be replaced. The program balances material costs with yield losses to get the most economic solution for the plant. In a preparation plant this is a daily ongoing process.

Secondly the performance of the clean tools should be check on a regular basis, say once a month. It is not necessary to do a full 10 or 12 gravity washability test. Washing the coal at 4 or 5 selected gravities would allow a quick picture of where 4 or 5 points are on an expected distribution curve. In this way the unit performance can be assessed. Note percent float or sink in the reject of product sample at a single gravity is another good performance monitoring method. But if the coal washability quality changes the changes seen could be due to coal as well as unit performance, hence the monthly multi gravity check is recommended and could also been done when the single point monitor turns up a question.

Expert systems have been successful used in coal wash plants to monitor the load in the various plant circuits. The software advises the plant operator on the efficient plant feed rate to prevent overloading of a given circuit that would affect the efficiency of the cleaning equipment in that circuit. Thus assisting the operator so the plant is operated efficiency. But secondly these systems have also been used to increase the average capacity of a plant. Operators know to run a plant so it will not plug up, because a plugged chute will shut down the entire plant hence reducing the tons run on a shift. With software that monitors loads in all circuits of the plant, experience has shown the operators with this feedback obtain higher average though put rates than they do without the software. It can be a very inexpensive way to increase plant yearly through put.

Finally a quality instrument maintenance and calibration program is important to give the plant operator reliable accurate data on the plant’s operation.

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# Probing the Preparation of Low-Rank Lignite and Non-Caking Coal

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**Abstract**: The establishment of large scale thermal coal preparation plants provides society with high-quality environmentally friendly coal product. It also allows China to reliably and efficiently utilize coal resources. The situation and characteristics of thermal coal resources in China were briefly introduced. Major issues existing in the processing of thermal coal, Due to the characteristics of thermal coals such as non-caking coal, long flame coal and lignite, it is necessary to explore and optimize the beneficiation of thermal coal to overcome disadvantages arising from the preparation of all size fractions. The authors believe that optimization surrounding pre-washing screening could provide a new idea and beneficial reference to the selection of flowsheet for the preparation of thermal coal. typically lignite and non-caking coal, were extensively analyzed. The optimization of the ratio of process investment was proposed. A screening classification was recommended prior to the primary preparation of thermal coal. The feasibility of this optimization was investigated. It is concluded that such optimization provides an excellent reference to the selection of beneficiation flowsheet for thermal coals.

**Keywords**: Thermal coal,washing and processing,flowsheet investigation, screening, optimized flowsheet , environmentally friendly , efficient utilization, comprehensive benefits

## 1 Situation and Characteristics of Thermal Coal Resources in China

Coal provides primary energy in China. It constantly accounts for approximately 70% of energy production and consumption. Coal constitutes 95% of China’s fossil fuel reserves. Coal combustion contributes toaround 82% of electricity generation.In order to meet energy demand for the rapid economic development, coal will consistently play a dominant role in the pattern of energy consumption in China(Ma, 2011). Cleaner, more environmentally friendly and more efficient utilization of coal according to coal type remains an open topic for exploration in the field of coal preparation.

Conservative coal reserve in China is approximately 1006.2 billion metric tons, of which coking coal and thermal coal are 264.5 and 741.7 billion tons, respectively.They accounting for 26.29 and 73.71%, respectively(Ma, 2011). Thermal coal is the major type of China’s coal resources. They are extensively distributed in Western China including Inner Mongolia Shaanxi, Shanxi, Xinjiang, Ningxia and Yunnan(Xie, 2001). Table 1 summarizes the type and portion of coal resources in China. The details of the reserves of thermal coal in China are tabulated in Table 2.

Table 1 Thetype and proportion of coal resources in China(Ma, 2011).

| Type | Percentage/% | Usage |
|---|---|---|
| Lignite | 13.08 | Thermal coal |
| Long flame coal | 14.81 | Thermal coal |
| Non-caking coal | 16.31 | Thermal coal |
| Weak-caking coal | 1.48 | Thermal coal |
| Gas coal | 12.00 | Coking coal |
| Fat coal | 3.36 | Coking coal |
| Coking coal | 6.20 | Coking |

 
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| | | |
|---|---|---|
| | | coal |
| Lean coal | 4.17 | Coking coal |
| Meagre coal | 5.73 | Coking coal |
| Anthracite | 11.18 | Coking coal |
| Nonclassfied coking coal | 0.51 | Coking coal |
| Nonclassfied non-coking coal | 9.69 | Thermal coal |
| Unknown | 1.31 | Thermal coal |

Table 2 The classification and reserves of thermal coal resources in China(Ma, 2011).

| Type | Reserves/Billion ton | Percentage/% |
|---|---|---|
| Anthracite | 113.1 | 15.24 |
| Lean coal | 59.0 | 7.95 |
| Non-caking coal | 161.9 | 21.83 |
| Weak-caking coal | 16.1 | 2.18 |
| Long flame coal | 148.8 | 20.07 |
| Lignite | 131.2 | 17.69 |
| Natrual Char | 1.6 | 0.21 |
| Unclassified | 110.0 | 14.83 |
| Total | 741.7 | 100 |

According to the utilization requirement of coal resources, the scarce coking coal has been extensively processed at all size fraction class. However, only 30% of thermal coal is washed in China. It can be seen from Tables 1 and 2 that thermal coal contributes primarily to China's coal resources. Among the thermal coal resources, non-caking coal, long flame coal and lignite possess highest reserves. They account for 59.59% of total coal reserves. The low-rank lignite and non-caking coal are characterized by high inherent moisture, low heating value, ease of weathering, and ease of mudding upon contact with water (Xie, 2001).

The combustion of unwashed thermal coal not only causes environmental pollution, but also results in inefficient utilization of coal. In addition, the quality of unprocessed thermal coal significantly fluctuates, hardly meeting end users' requirements on product quality. Therefore, the preparation of thermal coal is critical to the improvement of coal quality and efficient use of coal resources.

## 2 Existing Problems in the Processing of Thermal Coal in China

Currently most mine-affiliated thermal coal preparation plants use sieve-moving jigging machine and/or dense medium shallow vessel to reject gangues from lump coal. The sieve-moving jig is applicable to the rejection of gangues from the run of the mine coal. Its lower separation limit is >25 mm. It has relatively high requirements on feed size and stability(Dai, 2010). Moreover, the processing capacity of a single unit is low (Yang, 2002). The dense medium shallow vessel has several advantages including high capacity, precise separation, low separation limit (as low as < 13 mm). It is the most commonly used device for the preparation of thermal coal (Qi, 2006). However, China's thermal coal resources mainly consist of non-caking coal and lignite that contain high inherent moisture. The existing size classification prior to primary processing is inefficient. In order to ensure appropriate operation, the projected lower limit cannot always be reached in real practice (Kuang, 2006). A common measure taken is to manipulate the lower cut-off size to >25 mm. The size fraction lower than 25 mm will not be processed. Figure 1 shows a representative flowsheet of this kind.

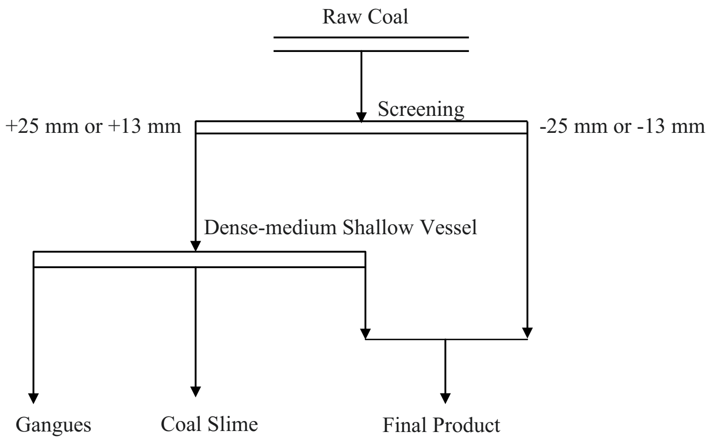


Figure 1 Flowsheet for the production of regular thermal coal

The two flowsheets for the beneficiation of thermal coal mentioned above focus on ash reduction via the rejection of gangues from lump coal. The preparation proportion is highly restricted to approximately 30% by coalbed mining methods. More than 60% of unprocessed slack coal is directly mixed with washed lump coal, and sold as final product. The quality of final product fluctuates greatly, and entirely depends on the coal quality of underground coalbed. Therefore, the thermal coal products generated by such washing processes cannot meet the increasingly improved requirements of customers on the quality of coal commodity.

In response to this issue, several plants in China have been attempting to process thermal coal in the manner for the preparation of coking coal. All size fractions of thermal coal are washed. In some mining areas, thermal coal preparation plants employ dense medium shallow vessel and dense medium cyclone for lump coal and slack coal, respectively. This combined processing system is shown in Figure 2. The objective of such flowsheet is to process all size fraction of thermal coal, to improve product quality, and to enhance economic benefits. However, those plants fail to fulfil their objectives after commissioning. Even worse, those plants cannot maintain regular operation. The major reasons are as follows.

(1) The most abundant thermal coal resources in China are non-caking coal and lignite constitute. The inherent moisture of those coals can be as high as 20% or so. Furthermore, those types of coal are simply mudded upon contact with water (Deng, 2011). Therefore, the preparation of all size fractions results in a great mount of secondary coal slime (Zhu and Xu, 2012).

(2) The change in size distribution stemming from the beneficiation of the run of the mine coal is large. A large portion of > 0.5 mm size fraction of coalturns into slime with size of < 0.5 mm. Accordingly, subsequent thickener for the recovery of coal slime and circulating water needs to be upgraded, which

greatly increases operating cost.

(3) The product pattern varies substantially before and after preparation. The percentage of < 0.5 mm coal slime increases significantly. The dominant product before preparation is electricity-generation coal. It becomes coal slime after washing, leading to a dramatic decrease in selling price. Although the preparation of all size fractions ensures the quality control of the > 0.5 mm product to meet the end users' requirements, it generates numerous coal slime products. Those inferior commodities are overstocked in the plants. Therefore, the washing of all size fractions fails to achieve the original objectives, which is to improve product quality and economic benefits, and to produce popular coal products(Xu, 2011). Moreover, it requires a huge number of sites for the drying of coal slime. The regular operations are discontinued once all sites are inundated with the coal slime products.

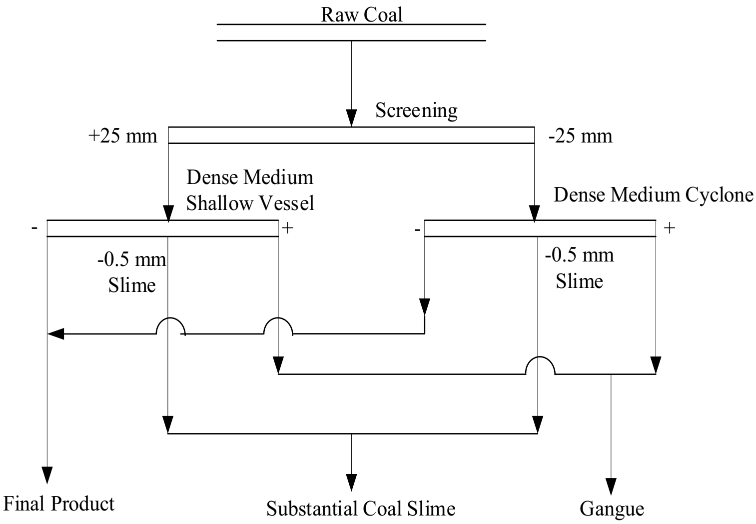


Figure 2 The flowsheet for the preparation of all size fractions of coal

## 3 Probing the Flowsheet for the Separation of Low-Rank Coals

Although the beneficiation of all size fractions of thermal coal can ensure the stability and control of product quality and effectively improve the cleanness of coal, the operational cost of its system for the recycling of coal slime is huge. In addition, it is highly bottlenecked by the overstocking of substantial amount of coal slime. The preparation of all size fractions of thermal coal does not fundamentally solve the contradictions between quality control of the beneficiation of thermal coal and the maximization of economic benefits. Therefore, the flowsheet to process thermal coal should be optimized according to actual conditions. The following points need to be seriously considered in the optimization. Firstly, the preparation proportion of raw coal should be sufficiently high, and the separation lower size cut-off should be sufficiently low. The flowsheet should be able to greatly reduce ash and improve quality. It should ensure stability of coal product to meet the quality requirements of customers. Secondly, the optimal flowsheet should ensure that the amount of coal slime directing to thickening system is low. It should attempt to relieve the burden of thickening section as much as possible, and reduce the generation of coal slime by-product.

The authors argue that a significant change be taken in the design of thermal coal preparation plants. If the comprehensive investment capital remains unchanged, the investment proportion should be optimized.

It is preferred to focus on the classification before washing instead of heavy investment into subsequent coal slurry system. Highly efficient pre-washing classification is the core link of the selection of suitable flowsheet. The investment into the coal slurry system of the preparation for all size fractions can be transferred to the pre-washing classification. The details of this capital transferring are summarized as follows.

(1) Drastically improve the design capacity of raw coal screening prior to preparation, and increaseabundance coefficient;

(2) Select efficient and advanced screening devices for fine raw coal;

(3) Scientifically determine the lower classification cut-off of screening (adjustable between 6 – 3 mm) according to the quality of mine coalbed and market demands;

(4) Never wash < 3 mm fine coal particles.

The design mode mentioned above firstly enables the lower classification cut-off to reliably shift downwards, therefore, ensuring the performance of screening. It will largely enhance the preparation proportion of the run of the mine coal, and in turn improve the quality of coal product. Secondly, the proposed design mode will considerably reduce fine particles that enter the primary washing system. Hence, the burden of coal slurry system will be relieved, decreasing the overstocking of coal slime product and cutting operational cost. As a result, the stability of production is secured and the overall benefits is maximized.

Figure 3 shows an optimized flowsheet for the preparation of thermal coal based on the proposed design mode. First, the run of mine coal is sufficiently classified and de-slimed. Gangues are then rejected from lump coal by dense medium shallow vessel. The slack coal is separated by dense medium cyclone. The coal slurry is thickened and dewatered by press filtration. The core of this process is to optimize and adjust flowsheet of thermal coal production through highly efficient pre-washing screening. On one hand, the optimized process increases preparation proportion and meets customers' quality requirements. On the other hand, it effectively overcomes the disadvantages in the beneficiation of all size fractions, including overwhelming amount of coal slime entering subsequent thickening section, high operational cost, and overstocking of coal slime product (Sun,2011). This processing scheme leads to optimum economic benefits.

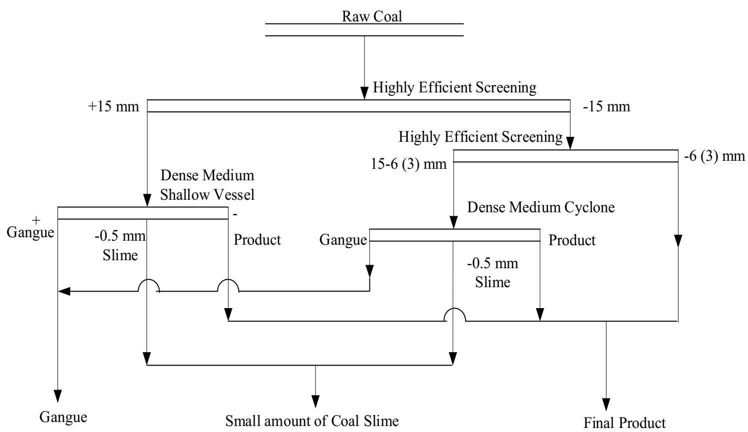


Figure 3The optimizedflowsheet for the preparation of thermal coal

## 4 Conclusions

The establishment of large scale thermal coal preparation plants provides society with high-quality environmentally friendly coal product. It also allows China to reliably and efficiently utilize coal resources. Due to the characteristics of thermal coals such as non-caking coal, long flame coal and lignite, it is necessary to explore and optimize the beneficiation of thermal coal to overcome disadvantages arising from the preparation of all size fractions. The authors believe that optimization surrounding pre-washing screening could provide a new idea and beneficial reference to the selection of flowsheet for the preparation of thermal coal. It is expected to bring more ideas into the perfection of thermal coal processing in China.

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# The Economic Impact Analysis of Coal Quality Management in China

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Abstract: China is the largest consumer of coal in the world and has been a net importer since 2008. Coal plays a significant role in China's economy, especially in power supply. The Ministry of Environment Protection, Ministry of Commerce and National Development and Reform Commission in China, along with 3 other authorities, issued the Interim Measures for the Quality Management of Commercial Coal on Sep 15th, 2014. The measures were effective from Jan 1st, 2015. The Measures apply to activities such as the production, processing, storage, transport, sale, import, and utilization of commercial coal within the territory of the People's Republic of China. This article gives an introduction of background and main contents, analyzes the economic impact of the Measures especially on coal preparation in China and abroad. Another important policy on coal quality management is the issue of the Technical Guide on Commercial Coal Quality Access and Control. The article compare the technical guide with interim measures, and propose next step policy China government should formulate to further strengthen coal quality management.

Key words: Coal Quality Management; Coal Preparation; Economic Impact; environmental protection; commercial coal; technical requirement; calorific value

## 1. Introduction

Coal is the main energy sources in China, and accounts for more than 65% in China's primary energy consumption structure for a long time. The environmental problem becomes increasingly serious due to extensive use of coal while coal ensures the repaid development of China's economy. Especially since the winter of 2012, severe haze in the North China and mid-east region caused widespread concern in whole society. The study showed that the fine particulate matter (PM2.5) is a major cause of haze, and the combustion of coal and other fossil fuels is the main source of PM2.5 and other atmospheric pollutants.

The pollution caused by coal utilization lies in the backward mode of use. At present, the coal quality instability is main cause of low efficiency and serious pollution. About 70% of coal is used for power generation and industrial boiler fuel, which are major pollutant emissions source. The power industry with most coal consumption has made remarkable energy-saving and low emission in recent years by measures such as upgrading and technological process. But the decline and volatility of coal quality is affecting the efficiency of the power generation system, safe and stable operation and pollutant emissions. Unstable coal quality can not meet with requirements coal-fired industrial boilers, and result in uneven burning, low thermal efficiency, high dust emissions, high fixed carbon content, larger original dust concentration, etc. One important reason of environmental pollution results in low quality and large combustion of civil coal. In addition, the low quality coal import has a certain impact on the environment in China.

Coal clean utilization has become one of important and urgent task for Chinese government. The clear

 
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requirements for promotion of coal clean utilization were made by *the 6$^{th}$ meeting of Central Finance and Economics Leadership* and *State Council Announce of 2014-2020 Energy Development Strategy Plan.* In September of 2013, Air Pollution Prevention and Control Action Plan issued by State Council put forward to forbid the import of coal with high ash and sulfur and low quality, and study and propose measures on coal quality management

## 2. Coal quality management measures and technical guide

### 2.1 Interim Measures for Commercial Coal Quality Management

In order to improve coal quality and promote the efficient and clean utilization of coal, and promote the improvement of air quality, the Interim Measures for Commercial Coal Quality Management(hereinafter referred to as "Coal Quality Measures") are formulated and issued by the National Development and Reform Commission, the Ministry of Environmental Protection, the Ministry of Commerce, the General Administration of Customs, State Administration for Industry and Commerce, and the State Administration for Quality Supervision and Inspection and Quarantine. And it is put into effect on January 1, 2015.

"Coal Quality Measures" is divided into five chapters, twenty-four articles. The first chapter clears the formulation basis, the scope and management authority. The second chapter is quality requirement, stipulate responsible party for coal quality, specify coal quality requirement based on different situation, and related requirement for production, processing, storage, transport, sale, import, and utilization of commercial coal. The chapter three, four and five stipulate the responsible party of coal quality supervision and inspection, and management method and punishment of coal supervision department. The second chapter is the core content of "Coal Quality Measures".

"Coal Quality Measures" sets out the basic quality requirements of commercial coal reached, it means that the coal products must meet this basic quality requirement before entering the circulation including ash content, sulfur content and trace elements indicators. All commercial coal parameters except lignite should meet: ash content≤40%, sulfur content≤３%; mercury≤０．６μｇ／ｇ, arsenic≤８０μｇ／ｇ, phosphorus≤０.１５０%, chlorine≤０．３００%, fluorine≤２００μｇ／ｇ。The lignite not only meet the above trace elements but also should meet: ash content≤30%, sulfur content ≤1.5%.

"Coal Quality Measures" sets out quality requirement for long-distance transportation (over 600KM) of commercial coal and add the indicator of calorific value. The lignite must meet: calorific value≥ 16.5MJ/kg，ash content≤20%，sulfur content≤1%。Other coal must meet: calorific value≥18MJ/kg, ash content≤30%, sulfur content≤2%. In regions of Beijing-Tianjin-Hebei, Yangtze River Delta, Pearl River Delta, the bulk coal with ash content≥16% and sulfur content≥1% is forbidden to sell and use.

### 2.2 Technical Guide on Commercial Coal Quality Access and Control

The Technical Guide on Commercial Coal Quality Access and Control (hereinafter referred to as "Technical Guide") was drew up by China National Coal Association and issued by State Administration for Quality Supervision and Inspection and Quarantine and China National Standard Committee. This standard specify the content including access indicator, control indicator and control value of commercial coal, applying to the production, processing, storage, transport, sale, import, and utilization of commercial coal, but not applying to coal used at coal mine and low calorific value coal for coal-fired plant. The standard set a transitional period, carried out since July 01, 2015.

**2.3 Relationship between "Coal Quality Measures" and "Technical Guide"**

a) They have the same purpose to strengthen the process quality management, improve the quality of end consuming coal. "Coal Quality Measures" specify same control indicators as "Technical Guide", but some indicators are stricter such as indicators of dust content, ash content, sulfur content, and the content of arsenic, phosphorus, chlorine and fluorine in coal.

b) "Coal Quality Measures" is mandatory rule, but "Technical Guide" is a recommendable standard. "Coal Quality Measures" focus on supervision and "Technical Guide" focuses on technical standard and specify measures to implement "Coal Quality Measures".

c)"Technical Guide" gives definition to terminology including "commercial coal" and "transportation distance". "Commercial coal" is coal products for sale after process and handle of draw coal. In accordance with current coal classification and different ways of coal processing and utilization, the commercial coal is divided into steam coal, metallurgical coal, and chemical coal. This standard also set different quality access indicator, control indicator, and control value according to different use of commercial coal. It is believed that control value of commercial coal in "Technical Guide" is the basic requirement for commercial coal should meet in trading circulation.

**3. Importance of strengthening commercial coal quality management**

In terms of environmental protection and energy efficiency, reducing impact of low quality coal import on Chinese coal market, improving commercial coal quality of enterprises in production, processing, circulation, and utilization, measures and guide of commercial coal quality access and quality control must be formulated.

a. The harmful elements control indicators for commercial coal are proposed from the source. It can effectively control the production, processing, distribution and utilization of low quality coal, thereby reducing the impact of coal emissions on the environment.

b. Promoting coal industry actively adjust the industrial structure, eliminate some poor quality and low-quality coal production capacity, further advance high efficient and clean use of coal resources, fulfill the real transformation and upgrading of coal enterprises. That is also one of important work to improve the financial situation of coal enterprises.

c. Effectively reducing import of poor quality coal with high content of harmful elements, and reducing low-quality coal trade and utilization in China. In 2014, China import 290 million tons coal from other countries including Australia, Indonesia, Vietnam and Mongolia, etc. "Coal Quality Measures" will impact these countries in different ways.

Australia is one of biggest coal producer and exporter in the world. It is main supplier of coking coal to China, about two-thirds of the average annual coking coal imports from Australia. The coal from Australia has better quality, high calorific value, and low sulfur, nitrogen and ash content. “Coal Quality Measures” has limited impact on Australian coal.

Compared with Australian coal, the coal from Indonesia and Vietnam has lower quality. According to official statistics, Indonesia has 36.5 billion tons coal resources, 5.22 billion tons proved recoverable reserves. Most of coal from Indonesia was exported to China, Japan, and India, etc. The calorific value of Indonesia coal is usually below 4500Kcal. And the coal with calorific value over 4500Kcal always has sulfur content over 1%. For anthracite from Vietnam, the ash content is over 25%.

The China’s imports of low-quality lignite increased rapidly in recent years. In past 3 years, China lignite imports increased by more than 9 times. After the implementation of “Coal Quality Measures”, the import merchants will concern the coal quality, which will improve structure of import coal in China.

d. Meeting quality control requirements of commercial coal in processing at local, provide the basis for local to select appropriate coal products, then achieve the objective of reducing urban pollution and energy saving.

**4. Next step for China to strengthen coal quality management**

It is necessary to put forward to a sound scientific standards, specifying requirements to commercial coal for power generation, coking, industrial boilers, civil and others, just like standards of commercial coal quality for power generation, commercial coal quality for civil. For different uses of commercial coal , these standards will put forward to meet the relevant technical requirements under the premise, select most representative quality indicators (such as calorific value, ash content, sulfur content) of each coal product as a measure to evaluate product quality evaluation. According to the utilization, market, resource distribution, environmental requirements and future technological development and quality improvement potential, the various types of commercial coal products will be divided into different quality levels, and thus establish commercial coal brand identity, respectively guiding its application.

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# OPTIMAL UTILIZATION OF DENSE MEDIUM CYCLONES

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## Abstract

In the current world mining climate the focus is on reducing cost and increase productivity. In many instances this productivity is associated with increased tonnages, "sweating the assets", which in many instances result in equipment being pushed to above their design limits. Simultaneously cost are being reduced, costs like spares holding, consumable costs and cheaper equipment alternatives with the end objective to reduce operating cost. The question is with these initiatives taking place on the mines, what can be expected from the performance of the DMS cyclone? The paper will look at some factors that could be influenced by these initiatives, namely: the effect of near dense material, the effect of cyclone spigot wear and the effect of overloading the process by focussing on the medium to coal ratio and desliming screen capacity. The efficient utilization of energy is also touched on.

**Key Words:** DMC efficiency, Energy, Efficiency, Optimization, Washability, Low Near Dense Material, High Near Dense Material, Revenue.

## 1. INTRODUCTION

In a survey by Ernst & Young Global Limited, 2014, the top risk in many commodity groups including the coal industry, has been identified as productivity improvement. With specific focus on the coal processing plant some of these productivity improvements and cost reduction factors may include increase in solids throughput, reduced maintenance cycles, reduced consumption of consumables e.g. magnetite and reduced energy usage. While these factors are necessary for survival, it is sometimes necessary to stand back and evaluate the effects on the overall process and specifically for this paper, on the effect it could have on dense medium separation cyclones. *The objective of this evaluation is to try and put into perspective the financial impact operational factors have on the process by relating it to the separation efficiency or EP value of the DMC and the effect of changing cut densities. Further down in this paper the relationship of separation efficiency and energy utilization will be explored.*

## 2. THE DENSE MEDIUM CYCLONE CIRCUIT

Although the Dense Medium Cyclone (hereafter referred to the DMC) is just another piece of equipment in the overall process, this unit is responsible for separating the incoming ore into valuable and non-valuable fractions as efficiently as possible and within a limited time frame. To ensure optimal efficiency from a dense medium process in order to increase productivity and maximise revenue, optimal separation efficiency is therefore required from the DMC as revenue can be generated or destroyed in seconds. The efficiency of separation in a DMC is normally expressed via a partition curve generated from sample analyses or tracer tests. The steepness of the slope of the curve determines the separation efficiency with a steeper slope indicating higher separation efficiency. The DMC efficiency is however not only influenced by the steepness of the partition curve but also the cut density and these factors are seldom considered together when optimising the performance of the DMC.

 
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## 3. DESCRIPTION OF THE PROCESS PLANT USED IN THE EVALUATION

### 3.1. Plant flow diagram and Financial factors used

In order to evaluate the effect of the productivity improvement and cost reduction factors mentioned earlier, a plant model was used consisting of a basic DMC circuit and with the fines being discarded as seen in Figure 1.

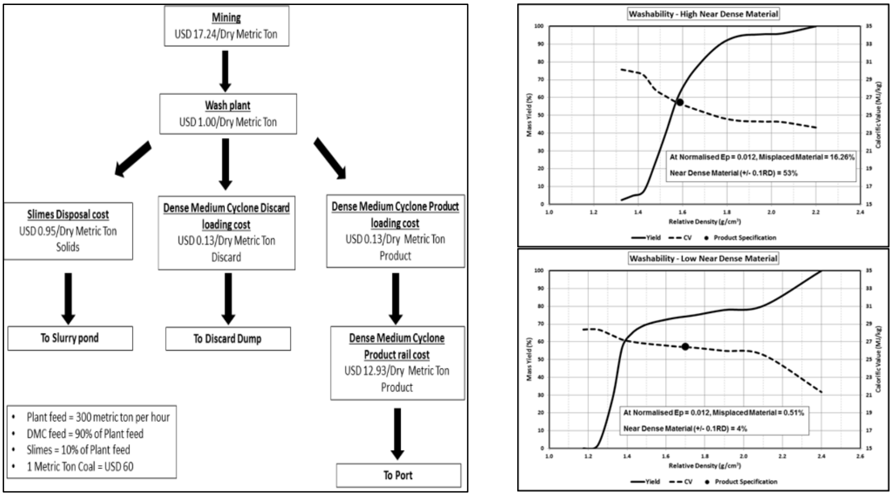


Figure 1: Description of Plant Model and washabilities used in the simulation

In order to understand the financial implications, operational cost was allocated to the main operations. These costs are indicative only and were generated solely for the purpose of the simulations. Where possible, values were checked against the industry norm to try and provide realistic answers. To allow for a better understanding of what impact the productivity improvement and cost reduction factors can have on the overall process, two different washabilities were used. The difference is the amount of near density material (+/- 0.1RD) present around the cut densities used in the simulation can be seen in Figure 1. In all scenarios a product quality of 26.48MJ/kg Gross Calorific Value was used (Net Calorific Value of 6000kcal/kg) which is represented as a black dot on the graphs. The revenue calculated in this evaluation is based only on the information supplied in Figure 1 and has been used for comparative purposes only. No attempt has been made to calculate the profit in these examples due to lack of information.

## 4. IMPACT OF PRODUCTIVITY IMPROVEMENT AND COST REDUCTION FACTORS ON THE DMC PERFORMANCE AND REVENUE

### 4.1. The effect of separation efficiency with different ore washabilities

The importance of low EP values on dense medium separation equipment is well documented and understood. The current economic climate has also assisted in operations paying renewed attention to maximising equipment efficiency. Based on the plant parameters provided in Figure 1 the effect of EP on the revenue being generated by this process is shown in Figure 2 where the two different washabilities have been evaluated.

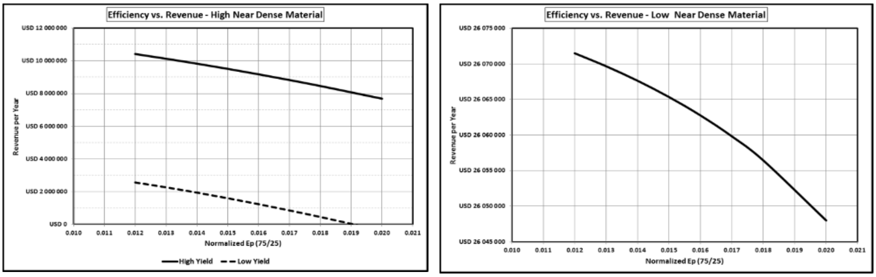


Figure 2: Effect of separation efficiency on revenue for a high and low amount of near dense material

From Figure 2 it can be seen that the impact of lower separation efficiencies on operations processing high near dense material ore is significant. With operations operating at lower yields, lower separation efficiencies can easily result in operations starting to operate at a loss as separation efficiency deteriorates. Ensuring optimal equipment performance is therefore of utmost importance for operations processing high near dense ore. On the contrary Figure 2 also indicates that operations processing ore containing ore with low amounts of near dense material could actually tolerate lower separation efficiencies without resulting in a dramatic reduction in revenue. The objective is obviously to maximise the revenue stream in any process but having a basic understanding of how efficiency impacts on the financial position of an operation can help operational personnel understand why certain cost saving measures might not always provide the desired results.

### 4.2. Increased plant throughput

In order to improve productivity and reduce operational cost one option is to increase the plant feed rate which sometimes exceeds design. These levels are then maintained unless frequent blockages are being experienced which would be an indication that the limitations of the equipment or process has been reached. By operating at the higher feed rates and assuming no operational problems are being experienced, some other factors can start to influence the efficiency of separation in the DMC process (Bekker, 2014). For the purpose of this discussion two factors will be focussed on namely the medium to coal ratio to the DMC and the effect the desliming process can have on the DMC performance.

#### 4.2.1. Medium to Coal ratios

In the design of a DMC the volumetric flow rate to the unit is dictated by the solids feed rate and the volume of medium required for an efficient separation process. Different medium to ore standards exits for the different industries in which dense medium separation is used in and it mainly depends on the washability characteristics of the ore, the type of ore and the value of the commodity. In the coal washing industry the medium to coal values typically range from 3.5:1 to 4.5:1. In designing the dense medium process the design houses typically ensures that the minimum medium to coal ratios are still being adhered to under maximum feed conditions. However, once the plant design feed rate is being exceeded, these medium to coal ratios can drop to values which start to impact on the DMC separation efficiency. Similar trends have also been reported by others (Swanson & Atkinson, 2007). When the feed rate to the process is increased, the medium to coal ratio generally decrease since adjustments are seldom made to the medium flow rate since either pumps have to be changed or the drainage capability of the drain and rinse screens are becoming limiting factors. The effect of the lower medium to coal ratio is therefore seldom further investigated and the feed rate is increased until a visual problem is identified for example an overloaded DMC spigot when the discharge capacity has been exceeded. Increasing the feed rate does increase the amount of revenue being generated but as the efficiency of separation starts to decline losses start to appear. It is important to note the effect the washability of the material has on the financial performance of the process. The easier the separation becomes the more tolerant the process performance becomes against equipment inefficiencies although financial losses will still be incurred.

#### 4.2.2. Desliming screen performance

With the increase in feed rate to the dense medium process the desliming screen performance can deteriorate due to the screen's capacity being exceeded. This can allow for unwanted fines entering the DMC circuit resulting in lower DMC efficiencies and also possible medium contamination (Bekker, 2014). There is however another problem and that is the effect that the increased water carry-over, due to surface moisture, will have on the actual cut density inside the DMC unit. The medium density is measured normally on the correct medium pipe line before it mixes with the ore and thus any water entering the medium circuit after the densitometer location is not accounted for. This is usually corrected by adjusting the medium set point if the product quality specification is not being met anymore. The increased dilution that occurs from the surface water being introduced has a more pronounced effect on the high near dense material ore due to the sensitivity of shifting cut densities on the yield achieved. Note that these changes occur without the adjustment to the medium set point. Operational personnel may therefore still believe that the DMC is operating at the correct cut density.

#### 4.2.3. The combined effect of medium to coal and dilution

With increased feed rates the combined effect of reduced medium to coal ratios, which reduces the DMC efficiency, and the dilution effect that shifts the actual cut density inside the DMC cyclone is shown in Figure 3. The argument can well be that this situation can be compensated for by adjusting the cyclone feed density to correct for the deviation from the product specification and this is true provided this change can be noticed. Many operations process coal with a high degree of variability which results in a scatter of results in their quality data. It would therefore be a very difficult if not impossible task to correct for changing product qualities with increasing feed rates because of the spread in results that occurs on a day-to-day basis masks the deviation in product quality. It is therefore important to understand the effect that increased feed rates have on the plant performance and the financial implications associated with these actions to then try and mitigate these effects via proactive changes like measuring desliming screen moisture contents and adjusting densities accordingly even though the actual sampling data may not show any deviation.

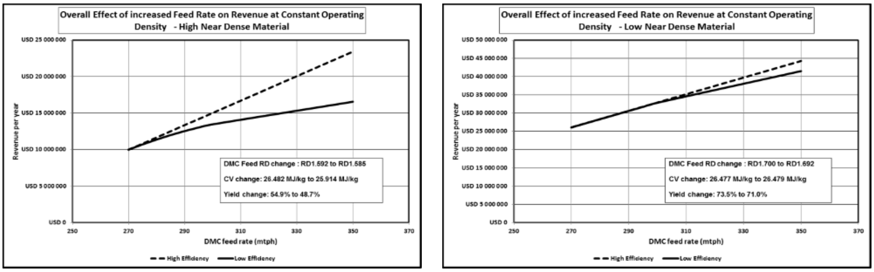


Figure 3: Combined effect of reduce medium to coal ratios and increased dilution due to increased feed rates for high and low near dense material ore

### 4.3. The DMC spigot

The DMC spigot selection is based on the required sinks discharge rate and forms an integral part of the performance of the DMC. Since this portion of the DMC is housed within the underflow discharge collection system frequent inspections are seldom carried out on some of the mines, mainly due to shortage of staff or inaccessibility to this section. Replacement of the spigot only happens then when the complete cyclone is being replaced. In order for the DMC to operate effectively it is recommended that the spigot diameter do not exceed 85% of the vortex finder diameter. If the spigot is only being replaced when a new DMC unit is installed, there is a good chance that this spigot to vortex finder ratio had been exceeded and that separation efficiency has deteriorated. A less common known phenomenon is that the cyclone cut density also starts to decrease as the cyclone spigot diameter starts to increase (Bekker, 2014). The combination of both the lower efficiency and change in cut density is shown in Figure 4 for high and low near dense material scenarios. This change in cut density manifests itself within the DMC without any outside influence from the operator.

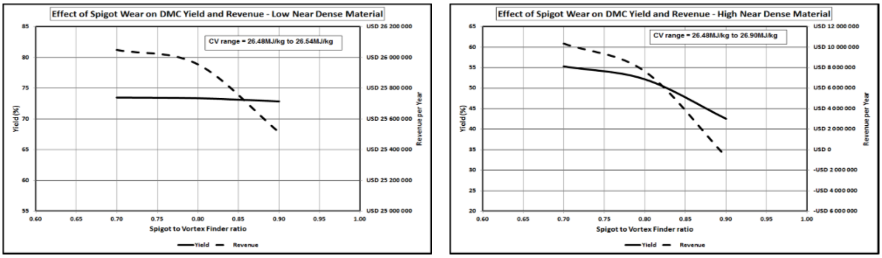


Figure 4: The expected operational impact a worn DMC spigot can have on the process for both high and low near dense material

In the event of using the spigot over an extended period, medium differentials or % misplaced material tests can be used to determine if the unit is still performing adequately.

## 5. THE IMPACT OF THE DMC PERFORMANCE AND ENERGY UTILISATION

### 5.1. The impact of DMC efficiency on energy recovery

The purpose of mining and the beneficiation of coal are to provide the correct quality product to the various industries that are being served. One of those industries is the supply of coal to power stations, both locally and international. Energy is therefore being recovered from within the earth to be supplied to the population. The aim of any coal mine operation is to do this as efficiently as possible and as profitable as possible. The impact that the DMC efficiency has on revenue has been assessed on a high level, see Figure 2 in the preceding section, but how does the DMC efficiency impact on the efficiency of energy recovery? As the DMC efficiency deteriorates the rate at which product is misplaced into the discard stream increases and thus also recoverable energy. This recoverable energy represents energy lost that could have been used to generate electricity. The product lost contains a certain amount of energy available and based on an efficiency conversion factor of 33%, the electricity amount available from this lost energy source has been calculated. This therefore represents energy wasted that could have been used to generate electricity. This type of situation can also lead to environmental problems like spontaneous combustion on discard dumps and it also strengthens the negative view that environmentalists have of coal mining. In addition to this energy loss there is another impact that lower DMC efficiency has on energy and that is the energy required to replace the product that has been lost to waste. The amount of energy required for mining and processing coal has been estimated to range between 18 - 55 kWh/metric ton depending on the mining method (Bleiwas, 2011). The combined effect of energy loss from both misplaced product and the additional energy required to replace the product misplaced to discard is shown in Figure 5. This energy loss has been put into perspective by relating it to the number of people that could have been supplied with electricity based on an average usage rate.

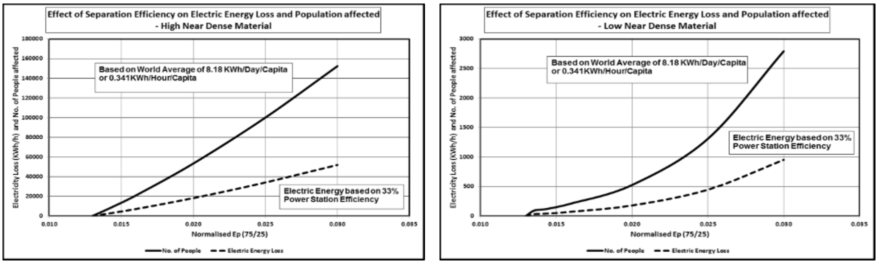


Figure 5: The amount of electric energy lost and number of people affected as a function of DMC efficiency for a 300 mtph coal operation

Poor DMC efficiency therefore not only affects the financial position of an operation but can also affect the social environment by way of energy wastage.

## 5.2. The impact of the DMC inlet design

The original Dutch State Mines used a tangential inlet design on the DMC and over the years the DMC inlet design has undergone various changes with the latest design that of a scrolled evolute design. Over the years some people has perceived this inlet change to be irrelevant but the increased capacity and improved separation efficiency has convinced many others over the years. In addition, the configuration flexibility that can be incorporated into the Multotec design led to increased volumetric capacities and associated energy savings, as seen in Table 1 based on information supplied by others (ACARP, 2013).

Table 1: Comparison of DMC capacity and energy saving with different inlet and configuration designs

| | Volume (m^3/hr) | Cyclone diameter (m) | DMC Head (D) | Total delivery head (m) | Absorbed power - Slurry (Kw) | Operating hours per year | Energy usage (kWh/year) | Cost per year @US 6.9 cent/kWh | Saving per year |
|---|---|---|---|---|---|---|---|---|---|
| Dutch State Mines DMC design | 420 | 0.80 | 12.3 | 29.8 | 224 | 6800 | 1526328 | USD 105 317 | USD 0 |
| Multotec DMC (Standard Capacity Design) | 420 | 0.80 | 9.0 | 27.2 | 199 | 6800 | 1353540 | USD 93 394 | USD 11 922 |
| Multotec DMC (High Capacity Design) | 420 | 0.80 | 6.1 | 24.9 | 179 | 6800 | 1216792 | USD 83 959 | USD 21 358 |

## 6. CONCLUSION

In an on-going effort to try and reduce operating cost and maximising efficiency, understanding the factors that influence the DMC separation efficiency and how it impacts on the overall process is important. Focussing on a single piece of equipment is not enough, a holistic approach is required. A thorough understanding of how the equipment is impacted on by the process is therefore a prerequisite for maximising the performance of the assets while maintaining optimal separation efficiency.

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# The Importance Of Coal Particle Characterization In Setting Up A Dem Model For The Flow Of Fine Coal Through A Hopper

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Processed coal has to adhere to an additional requirement of being able to pass through chutes, transfer points, hoppers and other handling or discharge systems with little or no difficulty to ensure efficient transportation thereof from a mine, through subsequent processing until delivery. Various handling difficulties such as irregular flow and decreased flow capacities occur as a result of the formation of dead zones, ratholing and arching. With the new worldwide focus on processing finer coal fractions it has also increased the need for understanding fines handling. The true behaviour of a material during handling can only be understood by means of experience or by evaluating the handling performance, which requires extensive financial resources as well as time. However, if a Discrete Element Model (DEM) is readily available with which assesses the handling performance easily and quickly, it would result in improved plant operations as well as aid in hopper and chute design. One of the most crucial aspects to consider when creating a validated DEM model of the coal flow is the coal particle characterization. With this particle characterization it is important to look at differences in shape, size and density as this will influence the coefficients of restitution, static and rolling friction. The DEM software, used in this study, makes use of spherical particles with default values assigned for the coefficients of restitution, static and rolling friction. It was found that the difference between the DEM default parameters and those measured is significantly different, by as much as 97%, which illustrates the importance of experimentally determining these parameters. This study will aim to show what the influence of shape, size and density have on the various coefficients. The study will also show that with proper coal characterization it is possible to create a validated DEM model of the coal flow through a hopper with 95% accuracy.

Keywords: Fine coal handling, Restitution, static and rolling friction coefficients, DEM modelling

## 1. INTRODUCTION

All mining operations are essentially concerned with processing: the handling and processing of particulate solids in a series of operations from extraction, removal, transport and size reduction, through to mineral extraction in the most economical and efficient way (Wolfgang et al, 2008). For a smooth, reliable and efficient bulk materials handling and processing, a good understanding of granular flows is of great importance (Alsphaugh et al, 2002) because the success of any bulk materials conveyor system is highly dependent upon the proper transfer of the conveyed material from conveyor-to-conveyor and into storage bins and onto stockpiles (www.Flexo.com)

According to Moore et al, 2012 previously many common bulk materials handling devices such as transfer chutes and bins had been designed with empirical knowledge by draftspersons that only had the constraints of predetermined conveyor locations and the fundamental physical properties of the materials they carry with no consideration of the material flow characteristics (Morrison and Wu, 2007).

This approach resulted in innumerable transfers that were inefficient because many of these devices failed to perform to their specification and had problems such as blockage and spillage. Aside from the economic losses due to plant interruptions, these problems also created potential environmental problems and safety hazards (Morrison and Wu, 2007).

The fundamental problem with bulk material handling is that there is no universally accepted standard to asses handling, by which companies had to rely on experience or to use a method of trial and error to establish what worked best for their situation. The difficulties that these two approaches yielded were that they were very much time consuming and expensive. This then magnified the need to development some relatively cheaper and reliable alternative to asses bulk materials handling. (Morrison and Wu, 2007).



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Such a development is the Discrete Element Modelling (DEM) which uses software such as EDEM, for example, and when used correctly the software can enable the engineer to obtain key design information on bulk solid material flow behaviour as shown in fig 1. Both quantitative (e.g. velocities) aspects and qualitative (e.g. visual representation) aspects of bulk materials flow through transfer chutes, hoppers, bins and other equipment systems can be obtained (EDEM software platform).

The way in which the model works is that the physical properties of the bulk solids are incorporated into the parameters of the software, which will then be able to model the flow of a real bulk solid accurately (Morrison and Wu, 2007). These bulk solid's physical properties include; shape, size, shear Modulus, Poisson ratio, Young's modulus and the coefficients of rolling friction, static friction and restitution (EDEM software platform).

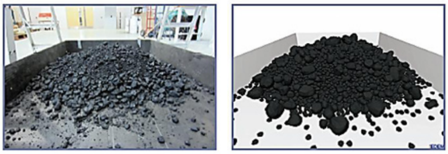


Figure 1 A comparison of a DEM software simulation (right) and actual flow (left) of coal (Moore et al, 2012)

The aim of the research work is to determine the effect of size, shape and density on the coefficient of restitution, static friction and rolling friction of a typical South African coal. These values will be used in creating a DEM model of the flow through transfer chutes and bins. The variables to be investigated included coal ash content (density), particle shape, particle size and also the impact surface of the coal (i.e. steel or coal). All of these variables will be experimentally determined and the effect of these measured coefficients will be illustrated with an EDEM model.

## 2. DISCRETE ELEMENT MODEL (EDEM SOFTWARE PACKAGE)

In order to represent the actual flow of a material in DEM, various parameters need to be measured experimentally. The parameters include the coefficient of static friction, coefficient of rolling friction, and the coefficient of restitution (Grima and Wypych, 2010). All of these coefficients also need to be measured on various impact surfaces, which are coal on steel and coal on coal.

### 2.1 Coefficient of Restitution

The Coefficient of restitution can be defined (equation 1) as the ratio of the differences in velocities before and after the collision, that is (Meriam and Kraige, 2003);

$$e = \frac{v_2' - v_1'}{v_1 - v_2} \tag{1}$$

where $v_2'$ and $v_1'$ are the relative velocities of the colliding bodies after impact, and v1 and v2 the relative velocities of the colliding bodies before impact.

A perfectly elastic collision has a coefficient of restitution of 1 whilst a perfectly inelastic collision has a coefficient of restitution of 0. This depends on the material types, particle geometries and the relative velocities of the particles at impact.

The coefficient of restitution is an important parameter in DEM simulations since it is incorporated in the DEM contact models, and to calculate (equation 2) it for individual particles as required by DEM the potential energies before and after collision are related, that is;

$$e = \sqrt{\frac{h_r}{h_i}} \tag{2}$$

where $h_\gamma$ is the rebound height and $h_i$ is the impact height.

### 2.2 Coefficient of Static Friction

The Coefficient of static friction (μs) is defined (equation 3) as the ratio of the static friction force to the normal force of an element on a surface (Halliday et al, 2005);

$$\mu_s = \frac{f_s}{F_N} \tag{3}$$

It is used to calculate the amount of force required for putting an object into motion that is initially at rest on another object (Brummer, 2010), and it is largely dependent on the material interaction and the surface finishes at the point of contact. It is an important parameter within DEM and needs to be accurately specified as it greatly affects particle flow behaviour.

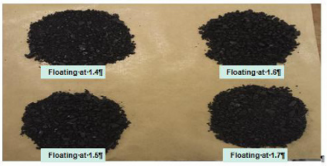
### 2.3 Coefficient of Rolling Friction

The Coefficient of rolling friction is used to determine the amount of force required to put an object into motion that is initially at rest on a flat surface (Brummer, 2010). Empirically it is defined as:

$$\mu_r = \frac{f_r}{F_N} \tag{4}$$

## 3. SAMPLE PREPARATION AND METHODOLOGY

Four samples with different ash content were received from a coal producer in South Africa (fig 2). The samples were firstly dried (according to ISO 1953) and a particle size distribution (PSD) was determined to document the "as-received" condition of the samples (fig 3).



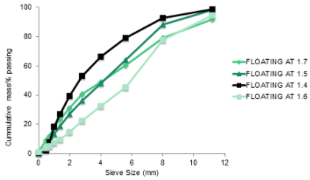




Figure 2: Different density coal samples received    Figure 3: PSD for the as received samples

Fig 3 shows that the "floating at 1.6" had the highest fraction of fines compared to the others, while the "floating at 1.4" had a higher fraction of coarse particles than the other samples. This phenomenon could be attributed to the characteristic of each of the samples.

The sample was then divided into three size fractions; 11.2 to 8 mm, 8 to 5.6 mm and 5.6 to 4 mm and then subsequently within those size fractions the particles were divided into three different shapes, as shown in fig 4.

A B C

Figure 4: A: Elongated particles, B: Angular particles and C: Modular particles

The particles were deemed elongated if they had a length of at least 1.5 times the width, particles that were deemed angular were those with sharp angles and the modular particles were those that were more rounded in shape in comparison to the others.

All the various coefficients were statistically determined by using in-house practical measurement techniques.

## 4. RESULTS AND DISCUSSION

The static friction and restitution coefficients were determined for the coal on steel interaction as well as for the coal on coal interaction. Only the coal on steel interaction for the rolling coefficient was determined.

### 4.1 Coefficient of Static Friction

Fig 5 represents the coal on steal interaction coefficients for the size fraction 4 to 5.6 mm for the various shapes at the various densities. Fig 6 represents the coal on coal interaction coefficients for the same size fraction, shapes and densities.

From figs 5 and 6 it can be observed that there is a substantial difference in the default coefficient value and the measured values. There seems to be a nearly invert relationship between the coal on steel

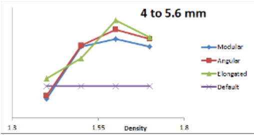



Figure 5: Coal on steel coefficients of static friction

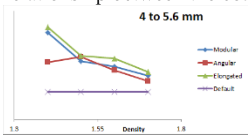
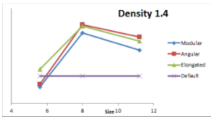



Figure 6: Coal on coal coefficients of static friction

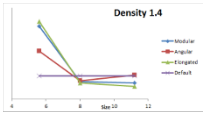
and the coal on coal coefficients. It is also clear that shape and density have an influence on the coefficients.

From figs 5 and 6 it can be observed that there is a substantial difference in the default coefficient value and the measured values. There seems to be a nearly invert relationship between the coal on steel and the coal on coal coefficients. It is also clear that shape and density have an influence on the coefficients.

Fig 7 indicates the results for a density of 1.4 for the coal on steel (A) and for the coal on coal (B) interaction. It is clear that particle size also have an influence on the static friction coefficient. The "inverse" effect of coal on steel and coal on coal is again observed.



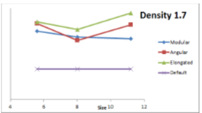

A



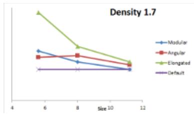

B

Figure 7: A: Coal on steel at a density of 1.4. B: Coal on coal at a density of 1.4.

Fig 8 indicates the results for a density of 1.7 for the coal on steel (A) and the coal on coal (B) interaction. From the results it can be concluded that the coefficient of restitution is strongly dependent on particle size.




A




B

Figure 8: A: Coal on steel at a density of 1.7 B: Coal on coal at a density of 1.7.

### 4.2 Coefficient of Restitution

Fig 9 shows the results for the coal on steal interaction coefficients for the size fraction 5.6 to 8 mm for the various shapes at the various densities. Fig 10 represents the coal on coal interaction coefficients for the same size fraction, shapes and densities.

Figs 9 and 10 indicate that there are significant differences between the default restitution coefficient and the measured values. The results also indicated that the coefficients of restitution are influenced by particle shape and density.

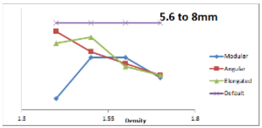



Figure 9: Coal on steel interaction

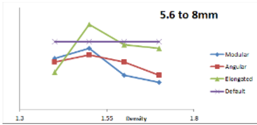
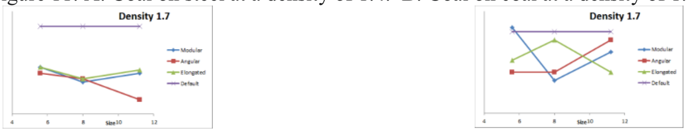



Figure 10: Coal on coal interaction

The coefficients of restitution for the different size fractions at constant density are given in figs 11 and 12 at densities of 1.4 and 1.7 respectively.


A B

Figure 11: A: Coal on steel at a density of 1.4. B: Coal on coal at a density of 1.4




A B

Figure 12: A: Coal on steel at a density of 1.7. B: Coal on coal at a density of 1.7

The results in figs 11 and 12 indicate that the coefficient of restitution is strongly dependent on particle shape and size.

## 4.3 Coefficient of Rolling Friction

Significant differences (fig 13) are observed between the measured values for the coefficient of rolling friction and the default value. There is not a big difference between the coefficient of rolling friction for the modular and angular shapes. The results also indicated that the coefficients of restitution are influenced by particle shape and density.

Nearly similar values (fig 14) and trends are observed for the modular and angular shapes at constant densities of 1.4 and 1.7.

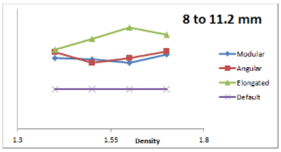
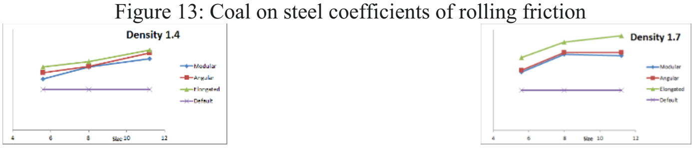



Figure 13: Coal on steel coefficients of rolling friction




A B

Figure 14: A: Coal on steel at a density of 1.4. B: Coal on steel at a density of 1.7.

## 5. APPLICATION OF RESULTS

Blignaut, 2014 showed the importance of material characterization and the correct measurement of the various coefficient in setting up a validated EDEM model. Figure 15 shows the huge errors that can be encountered if default software values are used rather than the actual experimentally determined values through materials characterization. The actual angles that results are given in purple, the simulated angles are given in red and the errors between the actual angles and the simulated angles are given in orange.

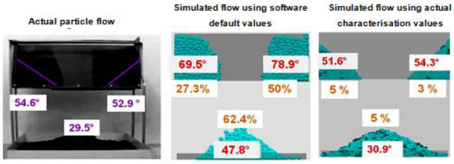



Figure 15: The importance of using the correct coefficients (Blignaut, 2014)

This work conclusively shows that if the material characterization is done successfully and the correct coefficients are determined, DEM models with a 95% accuracy can be created.

## 6. CONCLUSIONS

From the results discussed, it can be concluded that proper material characterization is of the utmost importance when considering creating a calibrated DEM model. It was pointed out that the coefficients of static friction, restitution and rolling friction is strongly dependent, among other, on the particle size, shape and density. There is a significant difference in the values for the coefficients for the coal on steel and the coal on coal interaction. This work also indicated what huge effect these parameters have on the model created and by accurately measuring these parameters a 95% model accuracy can be achieved.

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# Simulation of washability and liberation information from photographs

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**Abstract**

Imagine that a sample of coal could be crushed unlimited times to the same or different top size; a different washability would be obtained for each crushing experiment. Depending on the size and distribution of the raw coal components (say vitrinite, inertinite, and mudstone), better yields for a low ash product are obtained as a top size is approached that is the same or smaller than the components that need to be liberated. As the top size approaches the size of the components that need to be liberated (or smaller), near density material will start decreasing and the density differences between the components that need to be separated will become larger, because mostly “clean” single component particles are produced, that have widely different densities. As the virtual particle size decreases, the concentration or yield of the low ash product increases to a maximum, as the macerals are completely liberated from mineral matter

In this paper a method is proposed to predict washability characteristics by simulating how coal will behave when crushed to different top sizes and with different virtual particle size distributions, from photographs.

**Key Words**

Coal, crushing, washability, raw coal components, near density material, simulation, macerals.

## Introduction

There are two types of factors that influence coal washability: 1) the inherent characteristics of the coal that cannot be controlled, and 2) the factors that can be controlled.

The following inherent coal characteristics influence washability (McMillan et al., 2015):

 
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- Composition
- Component size
- Inherent fracture patterns

The following characteristics are induced, and could be controlled (McMillan et al., 2015):

- Blasting
- Type of crusher used
- Design of the crusher
- Top size of the crush
- Impact force of the crush

Even though some control may be exerted over how particles break, particle breakage essentially remains random. It is not possible to know exactly what type of particles will be produced when coal is crushed, and therefore the type of washability that may result. However, by simulating washabilties that may develop at different top sizes and different particle size distributions, the washability behaviour of coal may become more predictable.

## Methodology

The following steps were performed to simulate washabilities from photographs (Dorland et al., 2015):

- Photograph the core or ROM coal at close range
- Map out the components of the coal to scale on the photographs
- Create a base grid over the map of the coal at a grid cell size of 0.5mm × 0.5mm. The grid cell size must be small enough that most of the grid cells only consist of a single component.
- Estimate the areas of components in the base grid cells
- Assign estimated densities and ash to the different mapped components
- Calculate density and ash for each drawn block
- Import grid, density and ash data into geological resource estimation software
- Create a model based on the imported data in the geological resource estimation software
- Estimate density and ash for different block sizes using geological resource estimation software
- Calculate the washability tables per block size from the different block sizes

## Case Study

A sample of coal from the Limpopo Coalfield, South Africa (Figure 1), was analysed by the above methodology to determine the effect of different particle size distribution on yield (Dorland et al., 2015). This Medium Rank C bituminous coal is known to be rich in vitrinite, with mineral inclusions, in agreement with the visual inspection. The mudstone is finely disseminated between the vitrinite (shiny component). The size of the vitrinite and mudstone areas vary between approximately 1 mm and 3mm.

This sample of coal was visually mapped (Figure 2). An area was chosen from the mapped out area and a grid was placed over it. The selected smallest block size was 0.5 mm by 0.5mm in the example. At this size, most of the blocks are either clean vitrinite or clean mudstone, with 22400 blocks in the selected grid area.

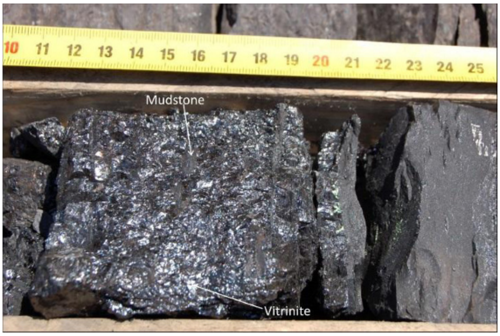


**Figure 1.** Limpopo Coalfield coal sample used in the case study.

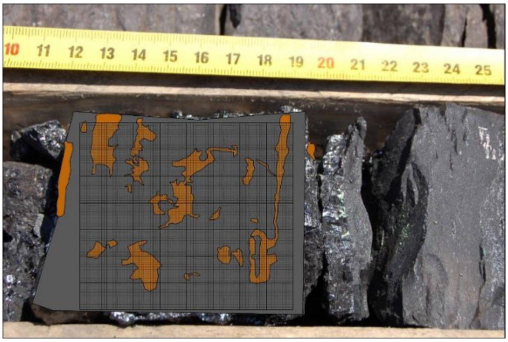

**Figure 2.** Map of the coal used for the case study. Grey areas are vitrinite and brown areas are mudstone. The large grid blocks are 10 mm by 10 mm and the small blocks are 0.5 mm by 0.5 mm.

All the 0.5mm by 0.5mm blocks were named and then the compositions of all of the block areas were visually estimated by enlarging the blocks in a graphics software package. The results of the block area estimation exercise were imported into a geological resource estimation software package to create a geological model using the estimated volumes.

It was assumed that mudstone and vitrinite volumes remained the same through the sample as at the sample surface. Using this assumption, volumes for vitrinite and mudstone were determined for the following block sizes using the geological software model:

- 0.5mm by 0.5mm by 0.5mm
- 1mm by 1mm by 1mm
- 3mm by 3mm by 3mm
- 5mm by 5mm by 5mm
- 10mm by 10mm by 10mm
- 25mm by 25mm by 25mm

The volumes of vitrinite and mudstone in the blocks for the different block sizes were then imported into Excel. In Excel, calculations were then performed to estimate relative density and ash % for each block. Densities of approximately RD1.35 for vitrinite and RD1.45 for inertinite were determined for South African medium rank bituminous coals by gradient centrifugation experiments (Crelling, unknown). The most abundant minerals in South African coals are quartz (RD2.65), kaolinite (RD ± 2.16-2.68) and dolomite (±2.84-2.86) (Matjie and Van Alphen, 2008; Van Dyk et al., 2009; Bunt et al., 2011, Everson et al., 2013). Hence, the following assumptions were used for the base case relative density (RD) and ash scenario for the blocks in question (Table 1):

**Table 1.** Assumptions used in base case washability simulations

| Component | Relative density (g/cm$^3$) | Ash (wt %) |
|---|---|---|
| Vitrinite | 1.35 | 4 |
| Mudstone | 2.54 | 98 |

Abbreviation :wt % -weight percent, g/cm$^3$ – grams per cubic centimetre

After the relative densities and ash for all the blocks were calculated, the blocks in each size fraction were grouped into the following relative densities:

[0-1.35); [1.35-1.40); [1.40-1.45); [1.45-1.50); [1.50-1.55); [1.55-1.60); [1.60-1.70); [1.70-1.80); [1.80-1.90); [1.90-2.10); [2.10-2.20); [2.20-2.30); [2.30-2.40); [2.40-2.50); [2.50-2.60].

Although density classes were determined as above, density information is available for all the estimated virtual particles. Therefore, using this process, more density classes may be used to gain a better understanding of the washability characteristics of a simulation. Weight averaging was then performed to calculate yield and ash for each density class for all the block sizes.

## Results

The obtained simulated washabilities for different sizes are presented in Figure 3.

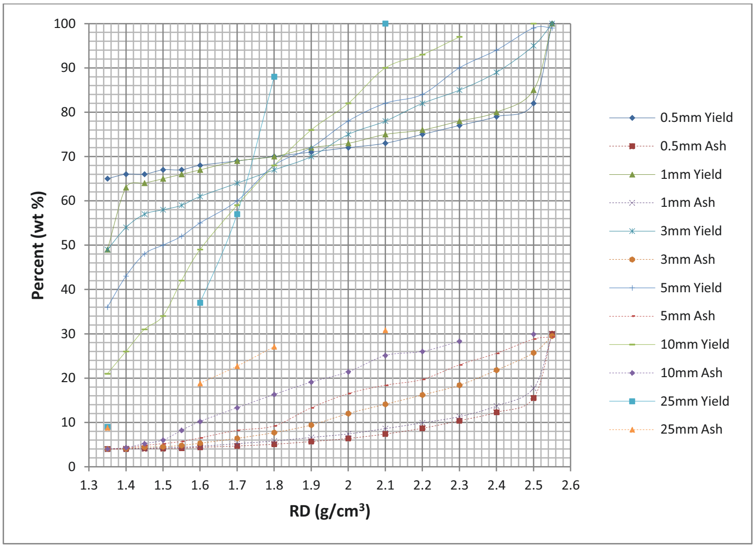


**Figure 3.** Simulated washability curves for the virtual particle sizes analysed.

From the results it can be seen that as the virtual particle size increases, the 10% ash product yield decreases. As the virtual particle size increases, the slope of the simulated washability curves increases, showing that there is an increase in near density material with an increase in virtual particle size.

The yield for a 10% ash product may be plotted against virtual size (Figure 4). From Figure 4, it can be seen that with an increase in virtual particle size, there is a decrease in 10% ash product yield.

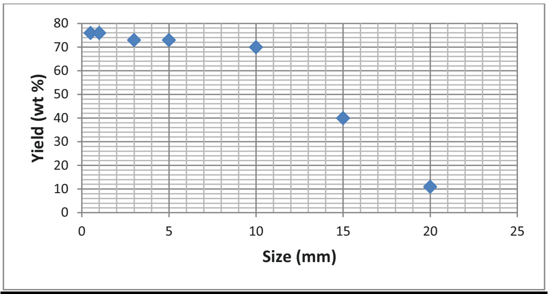


**Figure 4.** 10% ash product yield in percentage plotted against virtual particle size in millimetre.

This result clearly shows the risk to 10% ash product yield with an increase in particle size in the coal sample analysed.

**Automation**

Visually logging coal core is a fundamental part of coal resource estimation (Dorland, 2013). An attempt has been made to automatically map out the raw coal components from photographs using image recognition (Van Vuuren et al., 2015). This has been successful where there are clear visual differences between components; however, more work is needed to handle the less clear boundaries. Currently this option is under development.

Also, where there are not clear visual differences between components, x-ray scanning may be used to take an image of the coal sample. The coal sample may then be automatically mapped out using software that recognizes the different phases in the coal.

By using software that can estimate volume or area of a block, the component volume or area percentage can be calculated and washability and product liberation determined. Several simulations may then be rapidly obtained that simulate different virtual particle size distributions and their washabilities.

**Application**

Traditional physical washability tests performed in a laboratory are costly and time consuming. When automated, this proposed process would be able to provide washability information at a fraction of the cost and time that it takes by traditional laboratory analysis. This process may then be used where traditional washability information cannot be obtained due to cost and time reasons. It is important to note that calibration with real laboratory tests will add accuracy to the simulations. A comparison between the simulation and a laboratory test will be undertaken.

Places where simulated washabilities may be utilised are the following:

- Exploration boreholes to inform coal resource estimation, laboratory test work and plant design
- Investigate mined material before it enters the plant to inform the plant of what to expect.

**Conclusion**

The method described in this paper is non-destructive and several simulations using different specifications for the top size of particles and particle size distributions can be created. Although this method is not proven yet to be 100% accurate, it rapidly and inexpensively provides relatively accurate washability data. Currently experiments are being performed to validate this method.

This method will compete in the areas where traditional laboratory information is too time consuming and costly to obtain, especially in the control of coal beneficiation plants.

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# Online washability: Comparison of dual parameter and triple parameter analysis

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**Keywords:** washability, gamma transmission, X-ray transmission, dual energy, optical particle size analysis, density distribution, correlation ash/density

## Abstract

Growing demand on coal quality leads to the need of enhanced analysis tools for coal parameters. Therefore fast analysis of washability properties becomes a must in coal preparation.

The washability monitor OREGON which was introduced to the coal market in 2012 uses the combination of optical and radiometric methods for the fast determination of the washability curve. The concept of the analyzer is based on the measurement of a sample which will be processed in about 20 to 30 minutes.

The sample is fed through the analyzer on a conveyor belt in a single layer. Each particle is characterized by size and density. The washability characteristics of the full sample are then generated by Statistical evaluation. The method does not utilize any chemicals and the analysis can be performed even by untrained personell.

Basic techniques which are used for analysis were first presented at ICPC 2010. It is obvious that basic combination of radiometric dual energy measurement leads to inaccuracies if ash composition undergoes variations. This effect is well known from dual energies ash gauges.

Therefore the analyzer utilized a combination of dual energy and optical measurements.

The paper discusses the improvements which are achieved by the triple parameter measurement concept in comparison to alorithms based on only two parameters.

## Physical Background

Basically the OREGON washability monitor for coal utilizes principles which were derived from the well known dual energy ash analysis. There coal is transmitted by gamma radiation of low and high energy. While the absorption of the low energy radiation is dependent on the layer thickness (or more precisely the mass per area) and the composition of the material the absorption of the high energy radiation is only dependent on the mass per area. Therefore the signal derived with the high energy measuring path can be used to correct the signal of the low energy radiation path in order to produce a signal which is only dependent on the composition of the coal, i.e. the ash content.

Obviously this technique can be also applied on single particles to determine the particle´s ash content. The approximate size respectively mass can be calculated from the signals also and therefore a distribution curve can be generated for washability analysis.

 
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A problem is caused by the fact that the particles to be measured are considerably small. While coal layer on the conveyor belt have typical heights of 150 to 300 mm the particles to be examined for washability have a mesh size of 1 to 100 mm. Consequently the attenuation of the radiation will be much smaller if the radiation energies of the dual energy devices are used. Figure 1 illustrates the low (60 keV, $^{241}$Am) and high energy (662 keV, $^{137}$Cs) signals which will be measured for an average two different coals with an ash content of approximately 20 % respectively 30 %. Both calculations are done with a bulk density of 0,85 g/cm³ for better comparison.F*igure 1: Transmission intensity vs. area weight for gamma radiation of 60 keV and 662 keV*

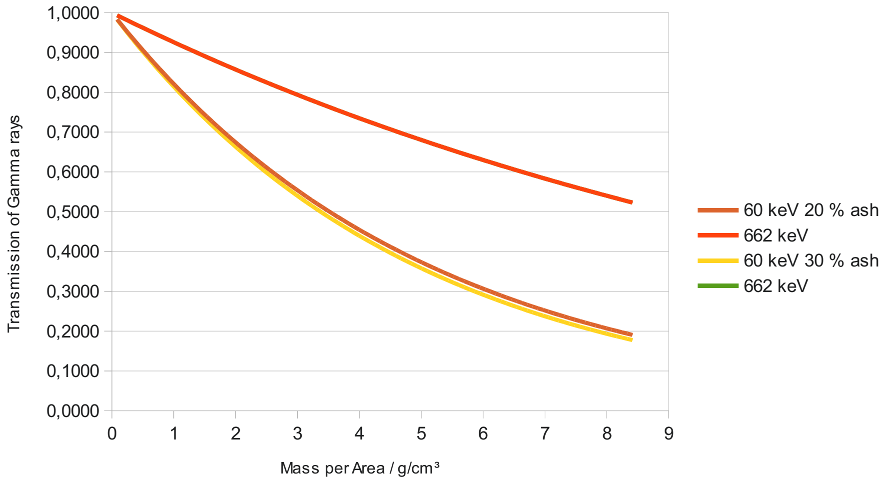


*Figure 1: Transmission intensity vs. area weight for gamma radiation of 60 keV and 662 keV*

It can be easily seen that the low energy signals can be differentiated with high masses per area but the curves become statistically insignificant by small masses per area. Furthermore, the high energy signal is completely independent of the ash content.

This problem can be overcome by exchanging the energy. While dual energy ash meters are limited to gamma sources which are stable and commercial available the washability monitor utilizes X-rays produced by X-ray tubes. The physics are the same for X-rays and gamma rays but the use of an X-ray tube offers the advantage that the energy of the produced radiation can be defined by the user.

Figure 2 shows the corresponding signals for two X-ray beams of 25 and 40 keV. It is obvious that the derived signals are of better significance and are therefore suitable for washability. This applies especially for the lower mass per area region. However, in the higher mass per area section the 25 keV signals become very weak and cannot be used. This means that the energy of the X-rays needs to be optimized to the particle size if the washability measurement is done in size classes.

At this point it should be mentioned that some non linear effect has to be taken into account; discussion of the physical and mathematical background would exceed the scope of this paper.

Based on this curves it can be stated that the use of X-ray tubes instead of gamma sources is a must for washability analysis. At this point there is no significant difference between the dual energy method for ash analysis and the washability analysis.

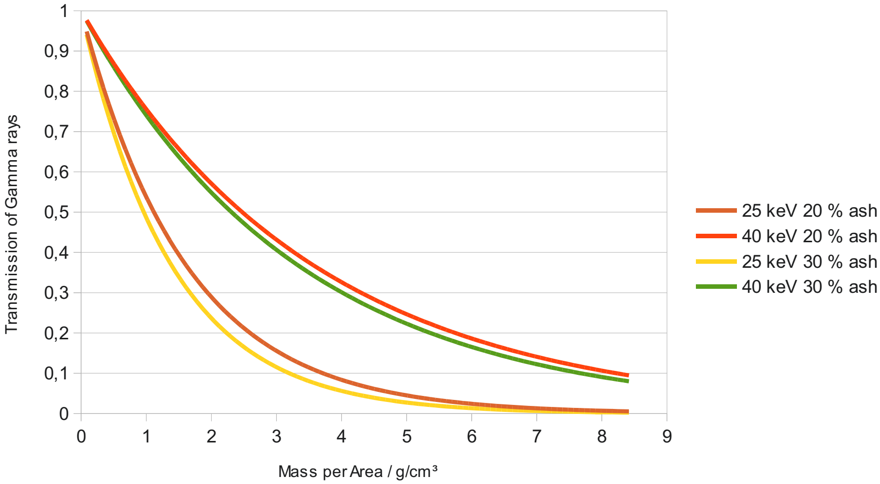


*Figure 2: Transmission intensity vs. area weight for X-ray radiation of 25 keV and 40 keV*Unfortunately the dual energy method has a strong drawback which leads to the fact that in modern process control this method does not longer meet the requirements. It works well as long as the ash composition does not vary. However, if ash composition is not constant then these variations will cause a significant error.

Unfortunately the dual energy method has a strong drawback which leads to the fact that in modern process control this method does not longer meet the requirements. It works well as long as the ash composition does not vary. However, if ash composition is not constant then these variations will cause a significant error.

Investigations performed by Bachmann [1] show that the relative error caused by a change of 1 % in iron content in the ash is about 6,3 % in ash indication.  Figure 2 illustrates the effect for german hard coal. The effect is not as strong if the calcium content changes but even here a change of 1 % Ca causes an error of 2-3 % in ash reading.

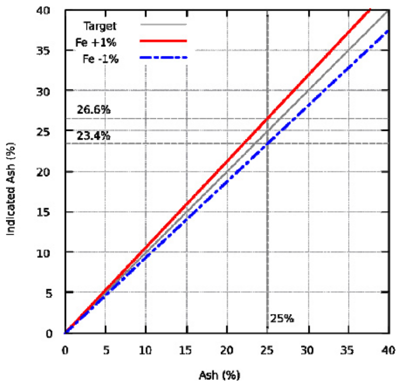


*Figure 3: Effect of change in ash composition on indicated ash*

This leads to the conclusion that application of the dual energy method is limited to applications where the ash composition does not change. If coal of different origins have to be measured then the accuracy of these devices does not fulfill the requirements. Similar results and attempts to solve these issues were published elsewere [6, 7, 8, 9, 10].

As the physics are the same these drawbacks apply also for low energy X-rays. It even becomes worse. First of all both signals in the low energy range are dependent on the material composition. Secondly, since the absorption coefficients of heavy elements like Iron and Calcium increase strongly with decreasing energy. The correlation between density and ash content depends on the ash composition. This indicates that a washability monitor which is based just on the dual energy principle needs intense recalibration which takes varying ash compositions into account.

## Washability Analysis of Coal

Washability analysis requires three steps:

- screening

different size fractions show different density distributions

- sink float analysis

this analysis delivers the mass percentage of coal for each density class

- correlation between density and ash

if this correlation is known then the results can be used to determine the yield.

To sum it up, three basic properties have to be determined for the coal particles: size, density and correlation ash/density. The last point distinguishes the washability analysis of coal from the washability analysis of iron ore. While the correlation between ash and density changes with the genesis of the coal the correlation between density of iron ore and iron content is fairly constant. Therefore the washability analyzer for iron ore utilizes only one X-ray measuring path [5].

The OREGON washability monitor is equipped with an additional optical measurement path which measures exactly each particle´s dimensions. The principle of the optical measurement was already presented in detail at ICPC 2013 [2]. Instead of requiring the high energy X-ray path as a measure of the layer thickness the thickness is therefore known exactly with a resolution of about 100 µm.. After correcting the X-ray signals with the known layer thickness both X-ray signals deliver information about the composition of the material.

The advantage of the use of optical particle characterization is obvious:

- the particle can be scanned in detail;
- the system determines length, width and height of the particle;
- in addition a complete particle´s outline is determined;
- no coal specific calibration is required.

Therefore one term in the washability determination, the size, is measured directly. So the radiometric part is only required for determination of an ash composition independent density; any size related effects can be compensated.

Having two radiometric measuring pathes which are both depending on density and composition pose a system which can be resolved distinctly. Therefore even variations in ash composition are taken into account which allows to use the analyzer without frequent coal specific recalibration. This is an important advantage if coal of different origins need to be characterized.

## Conclusions

In contrary to washability determination on iron ore where a stable correlation between density and iron content is established the correlation between coal density and ash content varies with the coal origin. Not only an online analyzer but also laboratory washability analysis is faced with this problem.

The use of an additional X-ray measuring pathes in the originally for iron ore developed washability analyzer OREGON allows to compensate for this effect.

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# Real-world model based validation of blending bed simulation techniques

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**Abstract**

Existing research claims highly detailed simulation of stockpile buildup and reclaiming processes in blending beds based on sophisticated physics simulation. Optimization mechanisms integrating these simulations allow significant improvement over standard homogenization techniques using existing machines by adding real-time quality measurement and software-based stacker control.

The effect of the optimization system on the homogenization however is evaluated within the proposed simulation system itself, which, although based on real-world physics and simulated bulk material, has not been validated with measurements on actual physical matter yet.

In this paper the simplifications are discussed which need to be made to physics simulation in order to be applicable in optimization systems and their implications on simulation quality and degree of realism. In order to validate the actual degree of realism in simulation systems the young engineer team of J&C Bachmann built a functional real-world model of a stacking and reclaiming system using inexpensive materials and their creativity, crafting and implementation capabilities. For the first time this flexible and highly detailed real-world validation system for bulk material blending systems is available and utilized to evaluate existing simulation approaches.

**Key words**

bulk material homogenization, longitudinal blending, real-world model, simulation validation

## 1 Introduction

The distribution of natural resources like coal and other bulk materials and their quality properties (Alderman, 2012) usually require blending of material from different sources in order to economically get a processable product. The natural variation in material properties and the variation introduced by the blending process is then reduced further by homogenization systems like longitudinal blending systems utilizing techniques like the well-known Chevron stacking (Bond, 2000). These homogenization systems mostly still operate unoptimized in a static way without reacting to the quality distribution and thus performing inefficiently leading to uncontrolled process parameters which have to be corrected later in the process and thus leading to increased cost and risk.

Optimized homogenization techniques based on real-time quality parameter surveillance were suggested in previous publications (Cipold, 2013) (Shukla, 2014). The methods used in the referenced paper utilize fast and detailed simulation techniques for longitudinal blending systems combined with evolutionary optimization approaches in order to modify the stacking order for bulk material. The material is stockpiled in a way to get minimal quality fluctuation and thus maximized homogenization for the best possible reclaimed product given the system's restrictions. The simulation techniques used are based on assumptions about real-world physics. Especially the fast simulation method introduces strong abstractions which are only validated against the detailed simulation. None of the simulations is validated against a real-world system.

 
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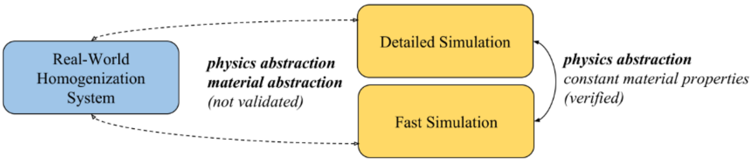
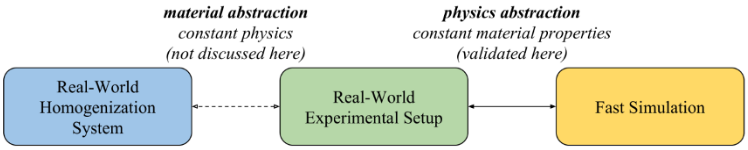



Validation of different simulations in present research papers

This paper bridges the gap between real-world and simulation by introducing a real-world experimental setup designed to the degree of detail of the fast simulator but with real-world physics. This allows comparison of the real-world blending system to the experimental setup in the physical world independent of other abstractions (not discussed in this paper) but more importantly validation of the fast physics simulation (mainly used in the optimization) to the real-world experimental setup and thus calculation of the error introduced by the abstractions of the fast physics simulation itself.




Validation of simulation against real-world with experimental setup presented in this paper

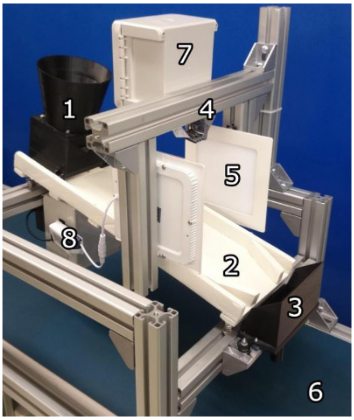
The approach allows a clear distinction between the abstractions made from the real-world model to the simulation. The experimental setup uses real-world physics as in the modelled homogenization system, abstracts machinery in a way which models the modelled system closely and only abstracts the material properties by reducing the degree of detail drastically to the level of the fast simulation. In the simulation strong abstractions are made to the physics simulation described in the experimental setup section of this paper. The degree of detail and material properties are reflected as closely as possible. Using this approach the validation of the simulation can be performed in two steps: physics and material properties (degree of detail). The results section discusses the validation of the real-world experimental setup against the fast simulation.

## 2 Experimental setup

This real-world model of a longitudinal homogenization system represents a full system including a stacking and reclaiming mechanism. For stacking and reclaiming several abstractions had to be made to the machinery keeping the physics involved as close to reality as possible. In the following all parts of the experimental setup are described.

### 2.1 Blending bed

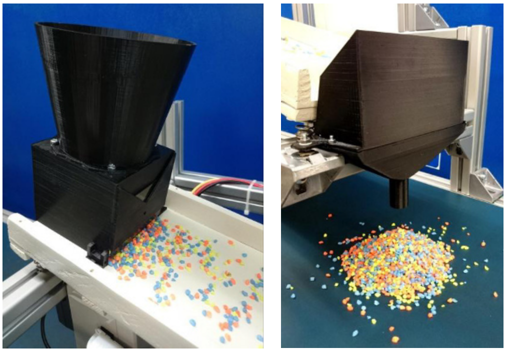
Allowing the buildup of a linear stockpile without moving the stacking mechanism itself the blending bed is formed by a flat conveyor belt with a top surface of 3550mm in length and 400mm in width. In the stockpile buildup area (700mm x 400mm) used for the experiments the belt is strengthened additionally with supports directly below the belt surface to prevent any bending of the belt and thus variation of the stockpile shape when moving the belt. The belt motion is controlled by an embedded computer capable of repositioning the belt in sub-millimeter precision.




1 bunker
2 vibratory feeder
3 stacking funnel
4 camera
5 lighting
6 conveyor belt / blending bed
7 embedded computer
8 electrical box

**2.2 Mechanical stacking**

A custom-designed, 3D-printed bunker feeding mechanism easily top-filled by a human operator with adjustable output rate provided by a gate mechanism feeds material to a narrow board-structure. The bunker is designed to provide constant particle feed without blocking and over-feeding particles to the board-structure. Below the board a voice coil-based subwoofer driver is mounted. It can be finely adjusted in frequency, amplitude and signal form. Together with the other components this forms a vibratory feeding mechanism and allows a detailed control over the particle motion on the feeder surface. The board has a narrow but long white surface which allows a clear top view of the particles for real-time quality and flow analysis (described later) as they are fed. A feedback loop with the flow measurement (described later) provides additional control over the flow rate.



Bunker (left) feeding material on vibrating board leading to funnel stacker (right)

From the vibratory feeder particles are dropped into a custom-made funnel merging the particles to a narrow stream representing the actual material stacking mechanism. From the funnel the particles are dropped onto the blending bed. This stacking mechanism does not move itself but the stacking position on the blending is moved relatively by moving the blending bed (conveyor belt) precisely.

An embedded computer controls the exact location where material is dropped off on the blending bed following instructions predefined in stockpile definition files (e.g. Chevron stacking with fixed amount of layers or more complex stacking structures).

### 2.3 Mechanical reclaiming

To achieve a reclaiming mechanism closely comparable to a bridge reclaiming system the stockpile stacked on the blending bed, which is located on a conveyor belt, is simply dropped slowly over the edge of the belt. This results in diagonal cross sections falling of the belt in a similar angle to a bridge reclaiming system without scraping falling naturally as the ground is removed. The particles are caught in an empty bunker structure and fed to the reclaimed material analysis without any buffering or further mixing. The flow rate and quality distribution is measured again using the same technique as described in the section about the stacking mechanism.

Reclaiming of material at the end of the conveyor belt with diagonal cross sections falling off

### 2.4 Material

The main abstraction between the real-world homogenization system and the experimental setup is made for the material properties. For the model material with distinguishable physical (color) properties was chosen. The material is classified by color and attributed with virtual quality parameters based on this classification.

The material chosen is colored (red, blue, yellow) gravel with a size of 0.5-3mm and volume of 1-15mm³ per particle. The bulk density of the material is 1.43g/cm³ with an angle of repose of 40°.

### 2.5 Quality and flow determination

For every particle stacked and reclaimed the quality parameters need to be determined. Combined with flow measurement precise knowledge about the particle and quality distribution can be derived and used for evaluation.

The distinguishably colored particles can be classified using an optical evaluation system. In this setup a camera is mounted above the feeding mechanism and recording images of the particles moving over the feeder board with 16 frames per second. A custom image processing software evaluates these frames in real-time determining the amount of material in every image together with precise classification in background, red, blue or yellow particle or unknown object for every pixel (resolution: 2592x1944 pixels). Strong surrounding light sources create constant lighting conditions for consistent results regardless of the ambient lighting situation.

### 2.6 Output

The data recorded by the system for the stacking and reclaiming situation contains information about the precise belt (stacker / reclaimer) position, material flow per color and time per camera image frame (16 times per second). This is all information necessary to recreate the stockpile in the simulation framework and also the information allowing the comparison of the reclaimed material properties from the experimental setup and the simulation.

### 2.7 Simulation

The simulator used to model the experimental setup was already presented in previous publications (Shukla, 2014). It creates a virtual blending system dropping particles along a central line of a longitudinal blending system according to the stacker positions and amount of material provided. Quality information is stored for every particle. The dropped particles arrange in a 3D space falling with simplified physics simulation forming a stockpile with an angle of repose of 45 degrees.

After virtual end-to-end reclaiming with a reclaiming angle of 45 degrees a quality curve is derived for the reclaimed, homogenized product. All particles in this simulation are exactly the same size. Sliding and rolling of particles besides the 45 degree angle are not simulated.

In order to allow comparison of the results determined by the simulation to the results achieved with the experimental setup the simulation settings needed to be adapted. These adaptations include the replication of dimensions, degree of detail and reclaiming angle. Furthermore only particles simulating three different material classes (here: red, blue and yellow) are used as input for the simulation. The output of the simulator was matched to the experimental setup output as well. The simulation performs the same reclaiming process as the model and calculates the amounts of material per color for every reclaimed slice of the stockpile.

## 3 Results

### 3.1 Goal and error assessment

The goal of the experiments is to show the effect of the physics abstractions. If everything else was identical in the two systems the effects could for example directly be measured by the difference of the reclaimed material output curves of the experimental setup and the simulation. The effect could be evaluated to a comparable number by calculating the standard deviation over the differences.

Additionally to the error which is measured for the physics abstractions other components add up to a systematic error:

- imperfect camera classification and flow during stacking and reclaiming
- errors due to slightly different material properties in experimental setup and simulation

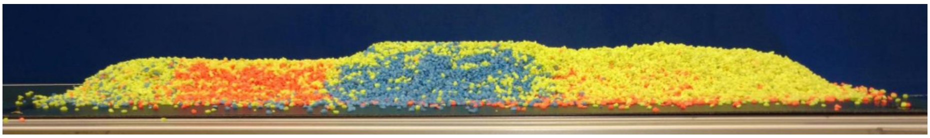
Separate repeatability experiments have shown significantly lower values for these errors as introduced by the physics abstraction. For simplicity these errors are combined together with the physics abstraction error to form the total error described by the standard deviation of the differences in reclaimed quality output curves for every experiment directly measurable by the systems.

### 3.2 Simulation settings

The simulation chosen for comparison included a particle size of 1mm³ with a maximum stacker traverse path distance of 700mm on a virtual blending bed of 1100mm x 400mm resulting in virtual stockpiles consisting of up to $2 \cdot 10^6$ particles. According to results determined by the experimental setup the reclaiming angle of the simulation was adapted to the model to be 60° to match the model closely.

### 3.3 Experiments

During the evaluation stockpiles of up to $2 \cdot 10^6$ mm³ were stacked using a varying combination of material classes, flow rates, stacker traverse paths and thus resulting stockpile shapes.



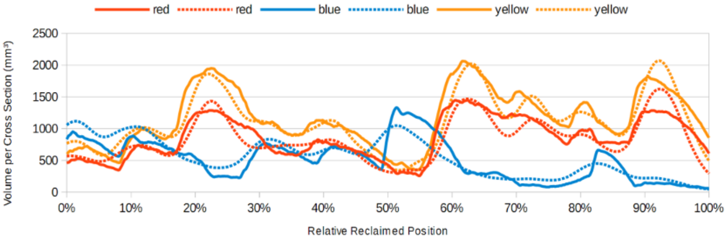
Side profile of a stockpile with intentionally varying height and material classes

### 3.4 Exemplary results

The following graph shows an exemplary result comparison between reclaimer output quality curves of simulation and experimental setup for a stockpile with varying height and material classes. The correlation between the same-colored solid (simulation) and dotted lines (experimental setup) is clearly visible. The relative standard deviations for the three color classes are:

**red**: 7,17% **blue**: 7,46% **yellow**: 7,82%

Other measurements show similar results with relative standard deviations between 5% and 10%.




Results showing high correlation between simulation (solid lines) and experimental setup (dotted lines) for reclaimed material analysis

## 4 Conclusions

This paper presented a real-world experimental setup and compared the results with a previously established simulator adapted to the situation. The results show a high correlation and thus prove the low influence of the high level of abstraction in the physics simulation. For the first time this confirms the suitability of the algorithms based on the simulation using a real-world counterpart.

The experimental setup presented is an ongoing development used for real-world verification of stockpile simulation techniques and real-time optimization. It will be used to support further development regarding the optimization of homogenization systems.

Software source code and 3D models are available open source at https://github.com/jcbachmann.

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# Rapid Coal Analysis with the Online X-ray Elemental Analyzer OXEA®

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## ABSTRACT

To control the mineral processing, e.g. coal preparation, an online coal analyzer system should be installed on a bypass belt. Compared to main-belt installations an improved accuracy is obtained at a bypass stream, because the material stream can be crushed down to a constant particle size distribution and shaped to a constant cross section profile. To minimize sample preparation, the material stream running across a bypass should be small, but the thin material layers require a re-design of the existing coal analyzers. The bypass was optimized for the OXEA® Online X-ray Elemental Analyzer regarding a constant material layer with a smooth surface. Consequently only minor changes were necessary concerning the OXEA®. However, the state-of-the-art microwave transmission moisture meters are designed for thicker material layers. Therefore a novel microwave transmission moisture meter was developed. As in the past, it is based on the measurement of the microwave attenuation and phase-shift, but the new device also makes it possible to measure thin material layers all the way down to 3 cm thickness with a much higher accuracy.

**Key Words:** Online Measurement, Moisture, Ash Content, Calorific Value, Elemental Analysis, Sulfur, XRF, Microwave Transmission

## INTRODUCTION

In a previous paper (Klein, 2013) a concept for an online analysis system for minerals has been described. Now this concept has been realized within a European Research Project called DRAGON, which was finalized in September 2015. The task of this project is to re-use the excavated material from the tunnel. The material is analyzed directly in the tunnel and sorted according to its quality and potential use. For this purpose the bypass system was designed to be installed directly on a Tunnel Boring Machine (TBM). The DRAGON analyzer system has to measure the physical and chemical properties of the excavated material. The bypass including the Online X-ray Elemental Analyzer OXEA® and the PMD 2500 moisture meter is part of the DRAGON analyzer system. All tests were made with the prototype, as shown in Figure 1, installed at the Herrenknecht works in Schwanau (Germany). The width of the bypass-belt is 40 cm, the speed is about 20 cm/s. The material layer is about 25 mm thick. The bypass-belt has been optimized for the OXEA®. Therefore only small modifications of the OXEA® were required. But in order to achieve the requested accuracy on thin material layers (25 mm), a complete redevelopment of the moisture meter's microwave unit became necessary.

## ONLINE X-RAY ELEMENTAL ANALYZER OXEA®

The improvements regarding the accuracy and the detection limit of the OXEA® are described in (Klein, 2013). For the installation at the bypass of the DRAGON Prototype, the OXEA® was modified. The bypass belt has side walls to guide the material. The OXEA® sensor box (OSB) is located between these sidewalls. For a very simple and convenient maintenance, a pneumatic lift has now been integrated in the fixing frame as a compact unit, as shown in Figure 1. To optimize the material flow, the material is

 
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scraped to a constant layer thickness and smoothed with a roller. With these arrangements the bypass is designed to be ideal for the OXEA®. To control the sorting process, the type recognition feature of the OXEA® was used. This method is based on the comparison of spectra. A quantitative analysis and a calibration of the OXEA® is not necessary for this purpose. The automatic sorting of different excavated tunnel material worked perfectly.

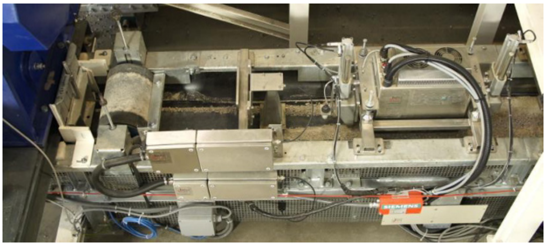

Figure 1: Bypass with OXEA® elemental analyzer and the PMD 2500 moisture meter

**ONLINE MOISTURE MEASUREMENT**

In the same step within the DRAGON project, the novel microwave transmission moisture meter PMD 2500 was developed, which is specially designed to measure thin material layers on bypass belts with very high accuracy of about 0.1 %.

The technique is based on the measurement of the attenuation and phase shift of a microwave beam transmitted though the material layer. State of the art is to install the instrument on a main belt, where the material layers are typically 100 to 200 mm thick. However, below 3 cm layer thickness the measuring effect is rather small and the accuracy is reduced. The greater material layer thickness at main belts is ideal for microwave transmission measurements in the frequency range of 2 - 4 GHz. In order to increase the measuring effect for thin layers, using higher frequencies is beneficial: Both, attenuation and phase shift become higher. But also the influence of particle size becomes bigger with increasing frequency (Klein, 1981). Measurements of Australian coal widely confirm these results. (Cutmore et al., 1989 and 1991). Therefore we decided to develop a new microwave moisture meter. We used a wider frequency range, which also includes the old frequency range, to improve the stability of the moisture meter (Klein, 2013). The newest data of the prototype is given in Table 1.

Table 1: Advantages of the PMD 2500 compared to PMD 2450

| | **PMD 2500** | **PMD 2450** | **Improved** |
|---|---|---|---|
| **Frequency range** | 1.2 – 4.5 GHz | 2.4 – 3.0 GHz | Factor 5 |
| **Long-term accuracy** | Phase shift: 0.5 °/GHz<br>Attenuation: 0.1 dB | 2 °/GHz<br>0.3 dB | Factor 4<br>Factor 3 |
| **Noise** | -120 dBm | -90 dBm | Factor 1000 |

For the design of the PMD 2500 it is taken into account, that the accuracy is reduced by multiple reflections at the surfaces of the material, if the attenuation of the transmitted material layer is low (Klein, 1981). Additionally the antennas generate multiple reflections, because they are not ideally matched over the whole bandwidth. To compensate this effect, the interferences caused by the multiple reflections must be averaged out by measuring over a wide frequency range. This is very important, especially for thin material layers, which are typical for bypass-belt applications.

The use of the wideband technology makes it necessary to transmit very low microwave power levels, in order to meet the new ETSI UWB regulations for Europe (ETSI, 2010) or the corresponding FCC (FCC, 2010) and RSS (RSS, 2014) regulations for USA and Canada. The allowed transmitted power level is about –60 dBm for a transmitting antenna with 15 dB gain. The insertion loss of an antenna pair with a typical distance for mounting at a conveyor belt is about 15 dB, i.e. the received power level for an empty measuring path is then –75 dBm. To calculate the dynamic range we must determine the noise. The noise floor was determined by increasing the attenuation with a calibrated step attenuator (HP8496 A/110 dB). In this test the transmitted power was set to – 40 dBm. Figure 2 shows, that the noise floor of the PMD 2500 is at 80 dB below the transmitted power, i.e. the noise floor of the PMD 2500 is at –120 dBm.

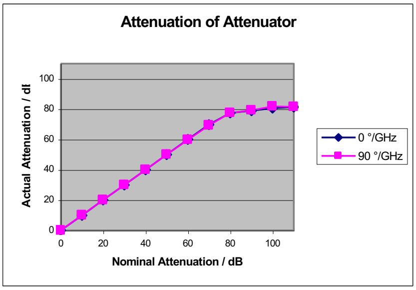


Figure 2: The attenuation reading of the PMD 2500 versus the setting of a calibrated step attenuator

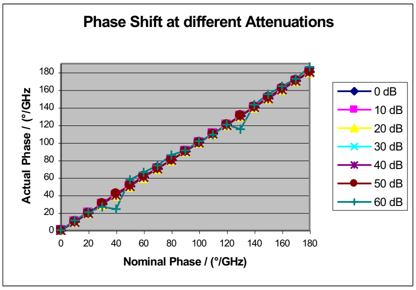


Figure 3: Phase shift reading of the PMD 2500 versus the setting of the phase shifter with different setting of the step attenuator

For the second measurement additionally a calibrated microwave phase-shifter (Narda 3752) was inserted in the measuring path, to check the accuracy of the phase measurement. Figure 3 shows the phase measurements versus the settings of the phase shifter from 0 to 180 °/GHz with different attenuations from 0 to 60 dB.

At 60 dB attenuation, i.e. at –100 dBm and a S/N ratio of 20 dB, we see the first significant deviations, as expected. With an allowed max. receiving level of –75 dB in an open system with the pair of antennas according the new ETSI UWB or FCC regulation, the insertion loss of the measured material is limited to <25 dB, i.e. the new PMD 2500 fulfills all requirements, even for thick or rather moist material layers.

The noise floor of the previous microwave transmission moisture meter is typically –90 dBm only. This means, that the noise floor is only 15 dB below the allowed receiving power level of the empty measuring path. These instruments consequently have a noticeably reduced accuracy, because the S/N ratio is too low.

The ETSI UWB and the FCC regulations are very restrictive, because the wideband technique uses bands, which are reserved for the original users of these frequencies. Therefore the allowed power level for UWB applications must be low enough, so that the original users are not disturbed. Vice versa, the PMD 2500 must be robust enough against electromagnetic fields generated by the original users. This is widely realized by several provisions.

Especially for thin material layers the long-term stability of the novel moisture meter is eminently important, because for all transmission measurements the measuring effect is proportional to the thickness of the measured layer. The typical measuring effect for coal is 1 °/GHz per % moisture and cm material layer. A long-term drift is created by a drift in the electronics and by temperature effects of the microwave cables. With a new method (patent pending) the drift of the electronics and the temperature effect of the cables are compensated. With these features the long-term accuracy given in Table 1 is achieved. For a coal layer of 3 cm thickness, this results in an error of 0.16 % moisture.

As mentioned above, the disturbance from interferences caused by multiple reflections at the material surfaces is reduced by the wide frequency range technique. Tests were carried out by filling a vertical positioned S-Band waveguide with crushed sandstone with a moisture content of 4.48 %. The S-band waveguide, which is used as sample-holder, has a length of 22 cm. The cross section of the S-band waveguide has the dimensions 24 x 72 mm. The material was filled in 50 g steps.

The first test was made with the novel PMD 2500 with 10 frequencies between 2.41 and 2.94 GHz, as used for the PMD 2450. Figure 4 shows the plot phase shift versus the mass of the filled in material. The linearity is excellent with a correlation coefficient of 99.99 %. The phase shift is known to be much more robust regarding multiple reflections compared to attenuation measurements (Klein, 1981).

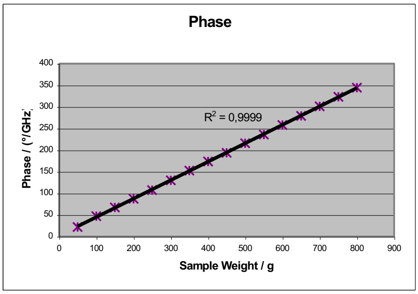


Figure 4: Phase shift of the sandstone measured in a waveguide as function of sample weight

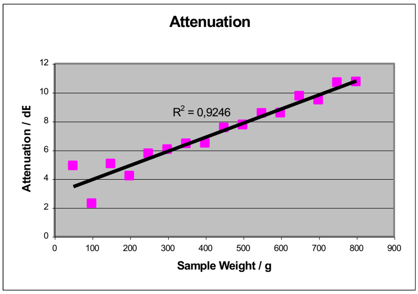


Figure 5: Attenuation of the sandstone as function of sample weight measured in the frequency range from 2.41 – 2.94 GHz

Figure 5 shows the corresponding plot of the attenuation. Here we see remarkable deviations at thin layers, which are caused by the multiple reflections.

The second measurement was done with 41 frequencies in the range from 2.4 to 4.4 GHz according to ETSI. The improvement in the low load range is obvious: The correlation coefficient is increased from 0.9246 to 0.9948, as shown in Figure 6.

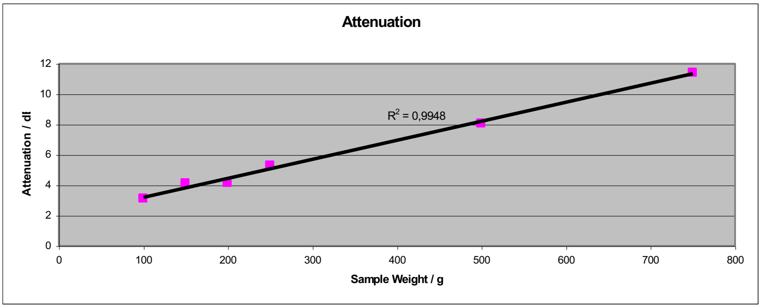


Figure 6: Attenuation of the sandstone measured in a waveguide as function of sample weight measured in the frequency range from 2.41 – 4.4 GHz

## ONLINE MEASUREMENTS

Parallel to the development of the PMD 2500 a bypass analyzer system was installed at the NRG Conemaugh Powerstation in PA, USA. Here an existing XRF analyzer and a capacitive moisture meter were removed by the plant, because these devices did not meet the performance and did not generate reliable results, neither for BTU and sulfur nor for moisture. It was very difficult to convince the customer to replace the insufficient instrument with another instrument, which is based on the same technology. After first tests in the laboratory scale we could install the OXEA® analyzer together with the PMD 2450 as a long-term test with 5 milestones. The required accuracy (RMSD) is 0.2 % sulfur and 180 BTU to control the process. Here we installed an OXEA® 3000 analyzer, which was modified with the improvements, developed within the DRAGON project (Klein, 2013). As moisture meter the old PMD 2450 was used. Within this project we also developed a new pneumatic lifting system, which has dramatically improved the design of the prototype developed within the Dragon project. The installation was in summer 2013. The first test was scheduled for September 2013. 33 samples were taken with a secondary automatic sampling system, which is installed at the bypass belt after the analyzers. Each sample was taken with 20 cuts over a period of 30 minutes. The readings of the analyzers were collected and saved using the sampling software package of the OXEA® and the PMD 2450 software. The data was evaluated by an independent consultant. The results are summarized in Table 2.

Table 2: Results of the first Conemaugh Performance test

| | **RMSD** | **arranged limit** | **comment** |
|---|---|---|---|
| **BTU** | 165.958 | 180 | passed |
| **Sulfur %** | 0.108 | 0.2 | passed |
| **Ash %** | 0.585 | not specified | |
| **Moisture %** | 0.781 | not specified | |

The first performance test was passed successfully. In the meantime all 5 performance tests have been successfully passed.

In 2015 the beta series of the PMD 2500 was completed. The Frequency range is from 2.4 to 3.5 GHz according to the FCC regulations. In May 2015 the PMD 2450 was replaced with the novel PMD 2500. Within these tests it was found, that in the hopper of the bypass-belt the material gets classified: On one side of the belt coarse material with lower moisture accumulates and on the other side the fines with higher moisture. This means, that the microwave beam is partly running through the less moist material and partly through more moist coal. This is a typical two path propagation, by which the accuracy of the moisture meter is significantly reduced. Different changes were done to solve this problem. Further improvements will be done soon. A preliminary calibration of the PMD 2500 was done with 8 samples only, which gave an RMSD of 0.3 %. This shows, that we are on the right way. A performance test with a sufficient number of samples will be done, as soon as the mechanical issue is sorted out.

Furthermore, the Conemaugh installation allows the comparison of OXEA® with a PGNAA analyzer. Some miles away from Conemaugh is the Keystone Power station, which uses the same coal. Keystone also has a bypass system for the blended coal. Here a PGNAA analyzer is installed. From both installations the results of the analyzers are saved and updated daily as a one week tracking plot (s. Figure 7 and Figure 8). The analyzer readings for BTU, Sulfur and moisture are shown in green for boiler unit 1 and in yellow for boiler unit 2. For comparison the sulfur is also measured in the flue gas. BTU, moisture and ash are determined over the heat balance (blue: unit one, magenta: unit 2) The red line is a digital signal, which indicates, if the online analyzer is measuring (high) or not (low). In times when the analyzers are not measuring, the analyzer readings are substituted with default readings by the Accutrack® software (Santucci, M. et al., 2008), (Garaventa et al., 2014). The delay between the different measuring points is compensated by a bunker model. The calculation of moisture content with the heat balance is naturally poor, because the heat balance cannot distinguish, if a reduction of the BTU is caused by a higher ash or moisture content in the fuel. However, the results for BTU and sulfur are reliable.

A comparison of the results from the online analyzers shown in Figure 7 and Figure 8 gives good evidence about the superior accuracy and availability of the Conemaugh installation with OXEA® compared to the Keystone installation with the PGNAA analyzer.

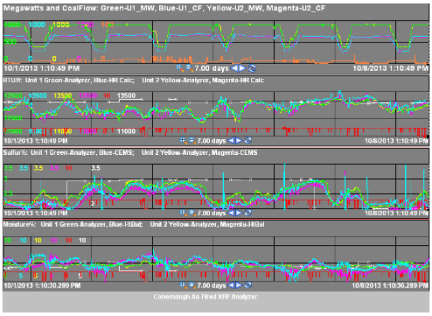

Figure 7: Trend Plot of BTU, Sulfur and Moisture results from the OXEA® Analyzer

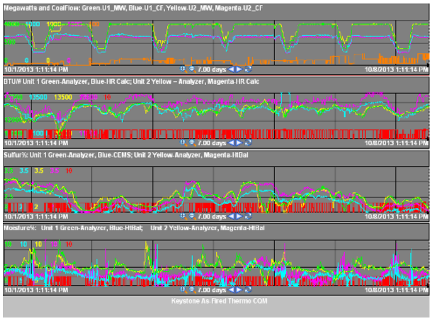

Figure 8: Trend Plot of BTU, Sulfur and Moisture results from a PGNAA Analyzer

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# TECHNOLOGICAL MANAGEMENT IN COAL PREPARATION ON THE BASIS OF INFORMATION TECHNOLOGIES

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Technological management at coal preparation is carried out with the purpose of a technology choice, a combination of constructive and regime parameters of concentrating devices for getting of set quality coal concentrates at their maximal output for the given structure of initial products. The decision of this problem is offered for carrying out with the help of information technologies which allow calculating variants of technological modes for acceptance of administrative technological decisions. For this purpose the mathematical model of coal preparation process as complex system is developed. It includes a coal composition, their compatibilities picked up by a principle at joint processing. Distribution of coal components is offered to describe by a square-law spline, and actual granulometric and fractional structure calculation to make for current ash level on the basis of the information available in a data base about earlier acting coal from the given mine or coal - analogue. The choice of separation density for each machine class or coal at their separate dressing is carried out on the basis of theorem Reinhardt analytically with use of fraction ash-density dependences for each of coal and their composition. The developed mathematical model of the technological coal composition dressing circuit is based on balance of each narrow size and density fraction in view of probability of its transition in this or that product, determined by the separation characteristic of the concentrating device. This model was a basis for drawing up of algorithm and development of computer calculation programs of qualitative and quantity indicator for rational mode variants.

Key words: coal, structure, composition, information technologies, a concentrate, the maximal output

Technological coal preparation management is necessary for a substantiation of rational structure coal composition, acting in dressing process, a choice of constructive and regime parameters of processes and the concentrating factory devices providing the most possible coal concentrate output at its set quality.

Problem of technological management is regime parameter definition for getting of a coal concentrate of set quality at its most possible output for the given initial product and a technical level of concentrating factory.

The operative establishment of a coal preparation rational mode for various variants of composition is a basis for decision-making on technology management.

Design procedures of qualitative - quantity coal preparation indicators started their formation in the thirtieth years of the last century [5, 16, 17, 18, 19]. However they have not received wide application in coal preparation practice because of their complexity and laborious method, caused by the information volume and use grapho-analytical method for application of Anry dressability curves. Therefore, and also because of not operative entry of the necessary volume of the initial product information, they are not applicable for technological management.

Actually at concentrating factory the situational coal composition structure is formed which dynamics is predetermined by transport logistic and stochastic of acting coal properties, defined their dressability. Thus coal is processed in the separate streams formed by so-called machine size classes, adapted for each used concentrating device.

For machine classes the separation density (border) should be determined, i.e. density of indefinitely narrow fractions equiprobably allocated between separation products which are specific to each concrete technological event and provide producing of the maximal concentrate output.

The decision of this problem is carried out with the help of Reinhardt theorem (a rule of the maximal output) [1] according to which the maximal output of a coal concentrate at separate dressing of its com-



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ponents will be in that case when the separation density for each component corresponds to the same ash elementary fraction.

Traditionally for these purposes dressability curves were used. However grapho-analytical method application does not provide necessary calculation accuracy of dressing parameters and complicates computer application for these purposes.

Exception of curves dressability calculation technology is provided by the analytical description of sufficient accuracy of coal distribution and its components according to density. The concentrated material structure is determining for achievement of the set qualitative - quantity indicators. The fullest information about coal structure is given by the suggested distribution function according to a dividing attribute $x$ (for bituminous coal it is density) of all concentrated material $\varphi(x)$ and its separate components $\varphi_\beta(x)$ [6]. Thus $\varphi_\beta(x)=\varphi(x)\beta(x)$.

Thus in this material the output of any fraction is equal:

$$\gamma_i = \int_{x_i}^{x_{i+1}} \varphi(x)dx. \tag{1}$$

And the content of any component in it:

$$\beta_i = (1/\gamma_i)\int_{x_i}^{x_{i+1}} \varphi_\beta(x)dx\,. \tag{2}$$

In the real separation processes which are carried out in industrial conditions, there is a mutual contamination of fractions which is determined by the separation characteristic. For bituminous coal it is dependence of probability of fraction extraction $E(x)$ on their density.

The concentrate output in real concentrating process is:

$$\gamma_c = \int_{x_{\min}}^{x_{\max}} \varphi(x)E(x)dx, \tag{3}$$

And the useful component contents in it are:

$$\beta_c = (1/\gamma_c)\int_{x_{\min}}^{x_{\max}} \varphi_\beta(x)E(x)dx \tag{4}$$

If the concentrated material structure is set by the fraction content $\gamma_i$ and the useful component content in them $\beta_i$ then calculation formulas for output and concentrate quality look like:

$$\gamma_c = \sum_{i=1}^{n} \gamma_i E_i\,, \tag{5}$$

$$\beta_c = (1/\gamma_c)\sum_{i=1}^{n} \gamma_i \beta_i E_i\,, \tag{6}$$

where $E_i$ - average values of fraction extraction in a concentrate.

Values $E_i$ are defined with the help of Gauss probability integral at values of argument:

- for dense-media separation $x = 0{,}675\left(\delta_{50} - \bar{\delta}\right)/E_{pm}$ (7)

- for jigging $x = \ln\dfrac{\delta_{50} - \Delta}{\bar{\delta}_i - \Delta}\cdot\dfrac{0{,}675}{\ln\left(I + \sqrt{I^2+1}\right)}$, (8)

where $\delta_{50}$ - separation density; $\Delta$- density of the liquid separation medium; $\bar{\delta}_i$ - the average density of an initial product fraction, which extraction is determined; $I$ - imperfection which is defined by the average probable deviation related to a difference of separation density and separation medium density, i.e $I = E_{pm}/\left(\delta_{50} - \Delta\right)$.

By the experimental data analysis it is established [7,15], that for processes of coal gravitational separation an average probable deviation is in inverse proportion to a root square of a separated material size: $E_{pm} = (\delta_{50} - a)/(b\sqrt{x})$.

From experience of dense-media separation of coal $a$ =-1300, $b$=13 [7, 15] and $E_{pm} = (\delta_{50} + 1300)/(13\sqrt{x})$, where $x$ - concentrated material size, mm.

For jigging $a$=1000; $b$=2,4, that gives $I = E_{pm}/(\delta_{50} - \Delta) = 1/(2{,}4\sqrt{x})$, i.e. the imperfection for jigging does not depend on separation density.

The maintenance problem of quality and efficiency of coal industry production use is substantially determined as an environment (geological conditions, layer capacity, their parent ash level, the sulfur contents), so by used extraction technologies.

Mineral coal is considered as uniform system of organic substances and inorganic components (minerals), the structure and quantity of a mineral part are its major characteristics. Presence of mineral impurity in coal causes not only ash level, but also the density, dependence ash level on which has linear character.

Each of these factors, except for natural, can be attributed to controlled or in partly controlled, and unguided, including commercial one. The controlled factors are those on which effects are more or less probable with the purpose of coal quality or its stability change concerning to a desirable level. Unguided factors do not submit to any managing influences. The group of natural and economic factors belongs to them [2].

In developed coal preparation practice the qualitative - quantitative characteristic of coal and its separation products in a sufficient measure is characterized by granulometric and fractional structures according to the density, specific to each coal. Thus each of these structures is represented as consisting of unguided and partly controlled components.

The decision of operative technological management problem is possible only for known ganulometric and fractional structures which definition is rather difficult technical and metrological task. Full enough information about coal structure can be received at their general approbation. But the databases generated on coal structure by information technologies use would allow synthesizing size and fractional structures of delivering on processing coal its identification (belonging to a concrete coal layer) and its actual ash level after extraction.

We offered to calculate fractional structures of coal size classes at its new ash level according to the fact that a ratio between amount of coal and intermediate fractions in a layer remains constant at amount change of the coal rock getting in coal at its extraction. The actual ratio of coal and rocky fractions is determined on the component balance basis of the given size class of coal.

Thus, on consideration that ash level fraction density in size classes and sulfur contents in them remain constant, we receive fractional structures at new ash level ordinary coal, using the data of earlier tested coal of a similar origin and having another ash level.

For operative fractional structure calculation of new delivered coal and forecasting of dressing parameters in conditions of its entry changes and quality, it is enough to define its general ash level. The results are more exact if in addition the granulometric coal structure with ash level definition of each size class is measured.

The component distribution function description of size and density fraction is carried out with the help of square-law splines [6] which factors are calculated from tables of granulometric and fractional structures.

These values are the initial information for getting of distribution functions $\varphi(x)$ and $\varphi_A(x)$. I.e. for any interval of density:

$$\varphi_i(x) = a_i(x - x_{i-1})^2 + b_i(x - x_{i-1}) + c_i \tag{9}$$

For finding of equation factors (9) it is necessary to satisfy three conditions, connecting these factors. Two of them are caused by concurrence of spline joining points on borders of transition from the previous density fraction to the next one. The third condition is the substance balance for each fraction, resulting to the equation:

$$\overline{\varphi}_i\left(x_i - x_{i-1}\right) = \int_{x_{i-1}}^{x_i} \varphi_i(x)dx. \qquad (10)$$

The coal components balance of separation, grating and phase separations is put into a basis of technology coal preparation model. Each concentrating operation is characterized by the separation characteristic, and their certain sequence representing the technological dressing circuit, also has its own integrated separation characteristic. Separation characteristics are defined for each device experimentally and described by Gauss probability integral. The sludge formations from each coal size class and its fractions density are taken into account, information of which is stored in computer database. At slurry decantation the sludge delay in dressing products is taken into account.

Modern technologies of coal preparation provide separate coal dressing preliminary separated on size fractions (machine classes). The range of size change gets out so that each used concentrating device functions with the greatest efficiency for the given conditions.

For all that, coal streams, generated in such a way, are processed separately. The total concentrate should have established ash level, and the chosen separation density for each machine class should provide its maximal output.

The decision search on calculation of the maximal set quality concentrate output and its necessary technological modes is carried out with the help of Reinhardt theorem (a rule of the maximal output) [1].

Basically, it is possible to proceed to an analytical method of a rational dressing mode definition by the table data interpolation of the coal fractional analysis for each machine class, submitted in a format necessary for plotting dressability curves. However specific curve character of elementary fractions $\lambda = \gamma^{л}\left(A^e\right)$ does not guarantee a correctness of linear interpolation application.

It is established [15], that there is a linear dependence between ash level and coal fraction density. It is defined that each coal fraction density consists of organic mass and the mineral impurity in certain proportions. If the density of organic mass is equal to $\delta^{org}$, and mineral impurity is $\delta^{\min}$ the density of coal fraction is:

$$\delta = \varphi^{org}\delta^{org} + \varphi^{\min}\delta^{\min}, \qquad (11)$$

where $\varphi^{org}, \varphi^{\min}$ - are a share of organic mass and mineral impurity in coal fraction, accordingly.
Ash level of this coal fraction is equal:

$$A = A^{org}\varphi^{org} + A^{\min}\varphi^{\min}, \qquad (12)$$

where $A^{org}$, $A^{\min}$- are ash level organic mass and mineral impurity, accordingly.

The joint decision of these equations gives the following dependence of ash coal fractions on their density:

$$A = \frac{A^{\min} - A^{org}}{\delta^{\min} - \delta^{org}}\left(\delta - \delta^{org}\right) + A^{org}. \qquad (13)$$

The received equation confirms linear dependence of ash fractions on their density. At the same time, coal fractions of various sizes have specific ash organic mass and mineral impurity that is proved by earlier researches [15].

If the total coal machine class consists of $n$ machine classes with individual shares $\alpha_i$ then its ash level is:

$$\overline{A} = \sum_{i=1}^{n}\alpha_i A_i = \delta\sum_{i=1}^{n}k_i\alpha_i - \sum_{i=1}^{n}k_i\alpha_i\delta_i^{org} + \sum_{i=1}^{n}\alpha_i A_i^{org}. \qquad (14)$$

The received equation is also linear; its decision according to density $\overline{\delta}$ allows defining corresponding ash level value.

If the certain determined separation density $\bar{\delta}$ for a fraction composition is known, then knowing dependences of ash level change on density $A^e(\delta)$ for each machine class, on the basis of Reinhardt theorem it is possible to define the separation density, providing getting of the maximal total concentrate output.

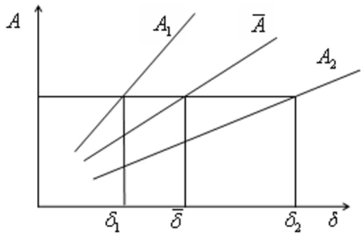


Fig.1 - To definition of separation density for machine coal classes

Effective achievement of the final technological purpose is considered as joint interaction of natural coal properties, its extraction and dressing technology. For this purpose there were developed the mathematical model of this system [4, 8, 9, 10, 11, 12, 13, 14], a design procedure of technological dressing parameters [3] and its software for the computer.

Thus, the set purpose of technological management is solved with the help of computer programs. The developed programs allow choosing the composition rational structure, to expect qualitative - quantity indicators of dressing products, to define boundary density for operations of gravitational separation and to do this operatively, in process of coal delivery to concentrating factory. In such program service the comparison of predicted coal preparation parameters and the technology constants achieved for automatic updating is constantly made, such parameters as separation characteristics, factors of sludge formation and their delays in products, updating of granulometric and fractional structures database by results of last coal approbations. Such programs developed at National mining university, are successfully used by some coal preparation factories of Ukraine. Acceptance of managing technological decisions allows rising at some enterprises a concentrate output approximately on 1 % at preservation concentrate ash at a constant level.

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# SYSTEM OPTIMAL CONTROL OF THE BALL MILL FOR THE PREPARATION OF THE PULVERIZED COAL MIXTURE FOR FORCED DRAFT

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**Abstract**

Pulverization is one of the most important stages of the electric power station, is often determined by the working capacity of the boiler and the entire plant as a whole. Also in the process of pulverization in a ball mill (grinding and transportation of raw materials) for burning in boilers of fossil fuel burning power station spent about a third of all electricity for its own needs power. Existing methods for controlling the operation of the ball mill due to the instability of the parameters of the feedstock and the lack of control and accounting procedures mentioned parameters and other external disturbances cannot provide the optimum mode of operation. Therefore, to prevent collapse or emasculation drum mill at production tend to work on a much understated performance of the ball mill from its maximum possible. In this regard, the decision of questions ensure optimal performance of the ball mill, and the quality of grinding products, as well as measures to reduce the specific consumption of electric energy and the cost of raw material grinding is an extremely urgent task.

**Key words**

ball mill, correction device, pulverized coal mixture, mathematical model, control algorithm, system optimal control, raw coal.

**Introduction**

The main task of system optimum control of the ball mill is to maintain maximum performance while keeping a predetermined fineness with minimal cost of electricity. One of the technical characteristics of the ball mill is the amount of the crushed material, which can accommodate the drum with associated performance determination with a ball mill in the steady state. However, loading of the ball mill may vary due to a number of external factors affecting the grinding of raw materials, such as: changing the mechanical properties of the crushed raw material supply, its humidity, the uneven distribution the materials in the drum during the loading process. Most of the enterprises control of the raw coal feeding using method of power consumption drum drive. Control by this method does not allow to achieve optimal loading of the drum, since the above-mentioned instability of raw coal parameters and external disturbances cannot manually hold the drum filling in premaximal zone «A» characteristic shown in Figure 1. It is in this area of the mill runs steadily, without blockages and devastation.

Maintaining the level filing of optimum performance is a complex problem [1,2]. In the state, $V=V_1<V_{max}$, $G=G_1$ have stable operation mode of the ball mill, when $V=V_2>V_{max}$, $G=G_2$ have an unstable operation mode of the ball mill, resulting in impaction, and, consequently, to a loss of productivity. From the above it follows that within $0 \leq V \leq V_{max}$ and $0 \leq G \leq G_{max}$ the load of the drum is controlled in a stable position.



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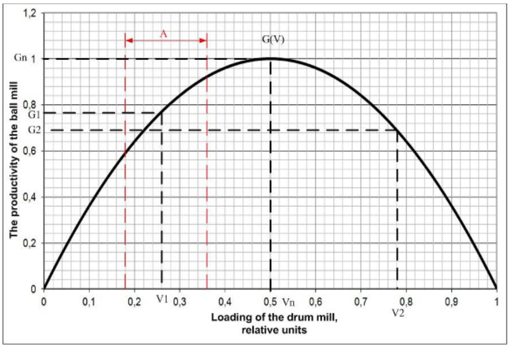


Fig. 1. The dependence of the productivity of the ball mill of pulverized coal from loading of the drum.

However, the grinding process is subject to constant external disturbances which affect the position of the optimum point $G_{max}$, shifting it in different directions depending shown in Figure 1. This may bring the loading of the drum to the point V2, which corresponds to the unstable operation of the ball mill. Consequently, the state of loading of the drum mill with respect to the extreme must be monitored at each moment of time to provide the control system with the necessary information for further steps in the change of load to avoid blockage and loss of productivity of the ball mill.

**Problem situation**

The aim of this work is to improve the productivity of the ball mill by creating a system optimal control for the supply of pulverized coal mixture.

**Problem solution**

The ball mill is a complicated object with an extreme dependence on the productivity of the volume filling of the drum, the form of which is largely dependent on the physical - chemical characteristics of raw coal (Figure 1) and is expressed by a transfer function [3,4]:

$$W_0(p) = \frac{1}{T_0 p + A} \tag{1}$$

Where $A = 1 - \left(\frac{a - V}{a}\right)^2$, V - volumetric filling of the drum; a - volumetric filling of the mill corresponding to maximum performance; $T_0$ - time constant of the control object $T_0 \approx 15\text{min}$; $\frac{1}{p}$ - the object of control.

The optimization algorithm must operate stably at both the uplink and downlink branch according to that shown in Figure 1. Stability is provided a correction device Z(p) with a transfer function [2]:

$$W_k(p) = \frac{W_o(p)}{W_o(p) \cdot Z(p) + 1} = \frac{1}{T_k p + 1} \tag{2}$$

From the expressions (1) and (2) follows:

$$Z(p) = (T_k - T_0)p + 1 - \left(\frac{a - V}{a}\right)^2 \tag{3}$$

where T - time constant of the correction device; $T_o$ - the time constant of the control object based on the correction level. Accordingly, a block diagram of the control object has the form (see Figure 2)[5]:

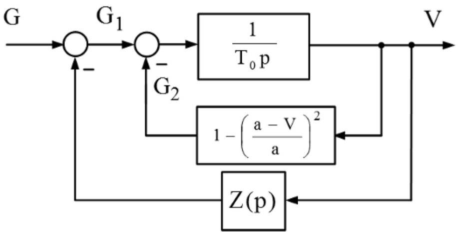


Figure 2. Block diagram of the corrected control object.

Where G - volumetric flow rate of raw coal, $G_1$ - volumetric flow rate of the dry part of the supply of raw coal, $G_2$ - productivity of the mill of pulverized coal.

Consumption of raw coal supplied to the ball mill is determined by the product of the rotation speed of the electric feeder and weight of raw coal, located on the belt feeder.

$$G = k \cdot F \cdot n, \tag{4}$$

where n - rotational speed of the drive, r/min; F - the weight of raw coal on the belt feeder, tons; k - a constructive factor of the belt feeder.

To control the supply of raw coal using a standard PID controller. Using (4) we obtain the functional diagram of raw coal feed system (Figure 3)[6].

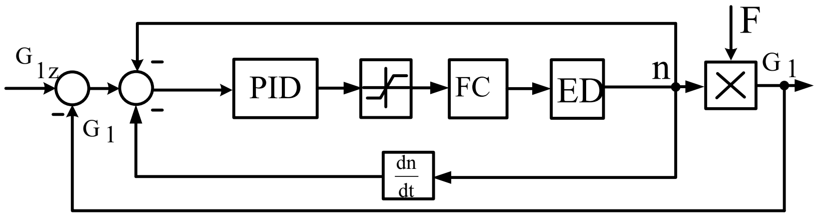


Figure 3. Control system of feeding of the raw coal

Where FC- frequency converter; ED - electric drive.

The optimal control algorithm is constructed as follows: when the optimal control unit turn on periodically by the time $T_{BO} = 2 \cdot T_k$ to change task to control volumetric filling $V_z$ of the ball mill by the value $\Delta V_z$. This value searches for optimum productivity. The optimal control algorithm is shown in Figure 4[7,8].

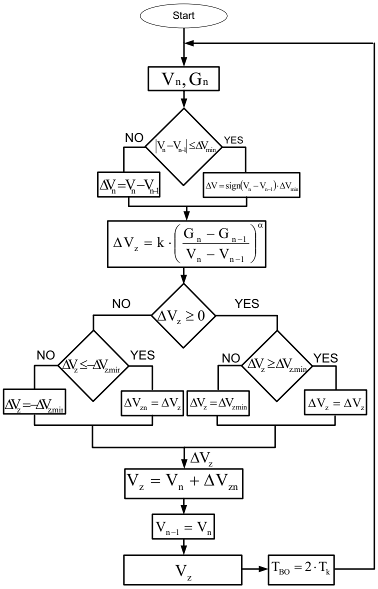


Figure 4. The optimal control algorithm volumetric filling drum mill.

Where $T_{BO}$- time constant optimization block, T - time constant of the control object.

The basic functional unit, deciding to find the optimal value of productivity of the ball mill is a functional unit for determining the value $\Delta V_z$ of the formula $\Delta V_z = k \cdot \left(\frac{G_n - G_{n-1}}{V_n - V_{n-1}}\right)^{\alpha}$. If the value $\Delta V_z > 0$, search of the extremum continues by increasing $V_n$. When passing through an extremum, i.e., at $\Delta V_z < 0$, search extremum will occur by reducing the value $V_n$. When this change step value $V_n$ it depends on the magnitude of the calculated value $\Delta V_z$ and decreases as it approaches the value of G to its optimal value. Dynamic performance of the optimal control algorithm based on the constructed mathematical model was evaluated by simulation software environment EXEL[9,10,11].

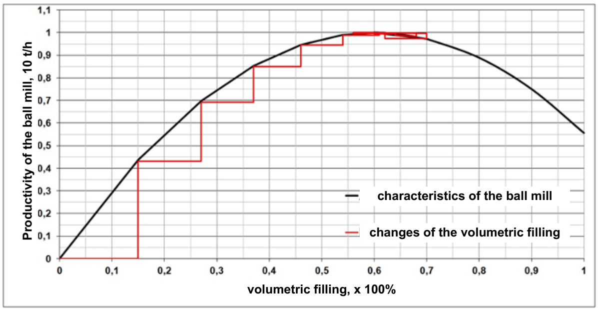


Figure 5. Illustration of the algorithm optimal control of volumetric filling of the ball mill

Specified Figure (see. Figure 5) demonstrates that the impact on the value of a step tasks on the volume filling of raw coal when approaching the maximum productivity decreases, reaching its minimum in the vicinity of the point of extreme. In general, the algorithm provides for the regulation of the control system to any external influence, deflecting factor productivity of the ball mill from its optimal value.

**Conclusions:**

1. Designed and tested by mathematical modeling of the algorithm optimal control productivity of the ball mill provides its optimal value at varying properties of raw coal feeding, as well as greatly reducing the specific energy consumption for grinding of coal.
2. Application correcting device provides stable operation of the control loop loading ball mill as a rising and descending branches, which ensures stable operation of the mill.
3. During the work of the electromechanical system of optimal control of filling volume on four ball mills demonstrated a high quality process control of grinding raw coal throughout the range of volume filling of the drum with a considerable saving of electrical energy per ton of ground coal

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# RADIOMETRIC CONTROL SYSTEMS FOR REFUSE DISCHARGE IN A JIG

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**Abstract:** The paper presents the results of industrial investigations on radiometric control systems for refuse discharge from a jig. A new monitoring system based on the monitoring of the natural radiation emitted by the material in the separation zone of a jig compartment has been developed and tested in parallel with the radiometric density meter and the conventional float. The correlation between the separation density monitored by the radiation density meter and the intensity of the natural radiation has been investigated. The correlation coefficient was ca. r = 0.973 and the standard deviation of the measurement in regression with the density meter indications was sc = 0.034 g/cm3. This shows that after a float as a measuring unit in the control system is replaced by a natural radiation monitor, good stabilisation of the separation density can be expected. The paper discusses advantages of such systems over the float based systems. This system can stabilise the desired quality of products without the intervention of an operator.

Key words: mineral processing, coal preparation, jig, remote control, discharge control, radiometric density meter, natural radiation.

## 1.INTRODUCTION

The beneficiation process of fine coal in jigs consists of two phases: stratification of coal grains in the bed according to their density and then splitting the stratified material into the product and the discharged refuse. At first, during subsequent water pulsations induced by opening and closing of air valves, the stratification of coal grains takes place due to their different velocity of movement upwards and downwards. Grains of low density migrate to upper material layers and grains of high density migrate to lower layers. The material is transported on the screen along the jig compartment by the additional horizontal flow of water. The stratification of grains due to their density is not perfect because their velocity of movement upwards and downwards depends in part on the grains diameter, the shape and variations in the degree of material loosening during a pulsation cycle. The distribution of coal density fractions in the bed, characterized by the imperfection factor *I*, has been investigated by many researchers, for instance: Jonkers et.all (1998), King (2001) and Cierpisz (2012). The maximum mass of the product of the desired quality (ash content) can be achieved for the ideal process when the imperfection $I = 0$. The stratified bed is then, in the end part of the jig, split to the product which overflows the end wall of the compartment and refuse (or middlings) discharged through the bottom gate. The separation density (cut point) is established by the tonnage of the discharged bottom product (opening of the discharge gate). The separation density depends also on the tonnage of raw coal feeding the jig, and its washability characteristics. The fluctuations in the separation density decrease the product tonnage so the proper operation of a refuse discharge system in a jig plays an important role in final economic results of a coal separation process determined by the tonnage and quality of the product..

## 2. REFUSE DISCHARGE SYSTEM WITH A FLOAT

Those refuse discharge control systems in jigs which are most widely used in the industry are based on a metal float monitoring the thickness of the heavy fraction in a jig compartment. Simplified schemes of such systems are presented in Fig.1. The main task of the refuse discharge system is to stabilize the separation density in a jig. Although the float is the main sensor in this system, its operation as a

 
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measuring unit has not been adequately studied. Its shape (cuboid, cone, "fish"), dimensions and density are selected according to the capacity of a jig, type of washed raw coal and the compartment of a jig (discharge of waste or middlings).

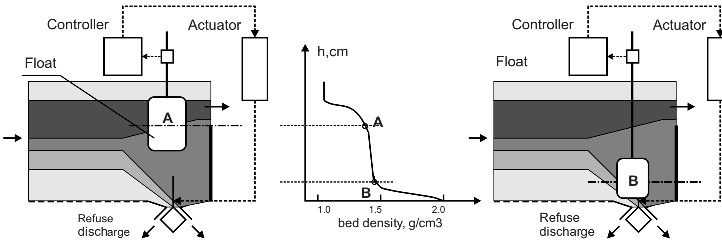


Fig.1. Two types of refuse discharge systems with floats

Two types of refuse discharge systems applied in Polish jigs (Komag) have been described by Będkowski (2002) and Bartoniek (2006). These authors apply two types of cuboid floats shown in Fig 1: type (A) which is relatively big (height $H_f = 30 – 35$ cm) and type (B),which is smaller (height $H_f =$ ca. 20 cm). The proper operation of float systems highly depends on changes in the tonnage of the raw coal feed. Significant changes in the feed tonnage change the position of the float with respect to the material layer of the chosen density. This results in the fluctuations in the separation density. Controllers applied in these systems are responsible for additional errors as they are usually based on proportional ("P") algorithms of control. Fluctuations of the separation density in jigs with float discharge systems were investigated by Cierpisz (2012). In field tests the separation density was measured by a radiometric density meter (described in p. 5) installed at A-A level (Fig.1). Fluctuations of the separation density over a longer period of time have been presented in Fig.2. Variations in the separation density are small ($\Delta\rho_s \approx \pm$ 0,03 g/cm$^3$) when the feed tonnage is fairly stable but they increase during significant changes of the feed ($\Delta\rho_s \approx \pm$ 0,12 g/cm$^3$). It is those changes that explain the frequent need for manual interventions from the jig operator who changes "the weighers" on the float to adjust its density to new circumstances. In search for better instruments to measure separation densities radiometric monitors were tested.

The first application of a radiometric density meter in a jig to replace a float was reported by Bartelt (1962). Extensive studies on this subject have been performed in Australia (Lyman & Jonkers, 1992 and Jonkers & Cameron, 2000). These works resulted in designing the JigScan - an advanced control system for a jig. Recent investigations on radiometric monitors for a jig in an experimental monitoring and control system installed in the "Murcki" mine, have been conducted by ITI EMAG and reported by Cierpisz (2013).

## 3. THE EXPERIMENTAL MONITORING AND CONTROL SYSTEM

Three monitoring and control systems shown in Fig.3 were tested in the coal preparation plant "Murcki" over the period between 2013-2015. The systems were installed in the second compartment of the OM20-type jig and the aim of control was to stabilize the separation density at desired values. The operation of the systems was monitored by the radiometric density meter (RDM) indicating the separation density evolution with time. The RDM was installed close to the upper edge of the product overflow wall to measure the density of the material separation layer reporting in half to the product and in half to the refuse.

A conventional float (FL), indicating the position of the heavy fraction in the bed, was used in the first system (A). In the second system (B) the RDM replaced the float as a main sensor in the closed loop control. In the third system (C) a new monitor, based on the measurement of the natural radiation emitted by the material (NRM) accumulating below the product overflow wall, was used. This concept of monitoring of a coal separation process in a jig is not available in the open literature.

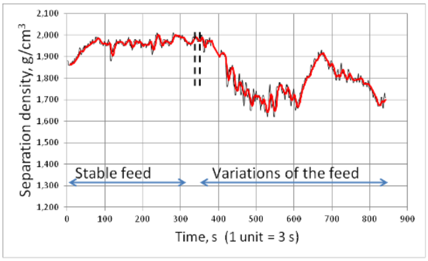


Fig.2. Fluctuations of the separation density due to the changes in the tonnage of the raw coal feed

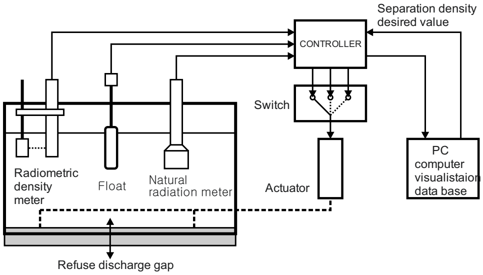


Fig.3. The experimental monitoring and control system the “Murcki” mine

Three closed loop control circuits were configured (A, B, C) remotely from the PC. A proportional (P) algorithm of control was used for the FL loop while the RDM loop was controlled by a proportional-integrating (PI) algorithm.

### 3.1. Radiometric density meter

The measuring method applied in the experimental monitoring and control system is shown in Fig.4. The measuring head consists of a $^{137}Cs$ radiation source and a detector in the form of a scintillation counter. The collimated radiation beam is absorbed by the material layer situated ca. 5 cm below the product overflow wall (thickness of the layer: 5 cm, length: 30 cm). The signal from the detector (Fig.4) is measured over a period of 0,15 s at the end of each cycle of pulsations (1,2 s) when the material is compressed. The standard deviation in the density measurement for 10 cycles of pulsations (effective time of measurement: 1,5 s) is ca. $\sigma_d = 0{,}005$ g/cm$^3$ (for the 1,5 g/cm$^3$ density).
The density of the dry material $\rho_m$ (the separation density) was calculated from the density of the medium $\rho_o$ measured by the RDM for a given size composition of the medium (Eq.1).

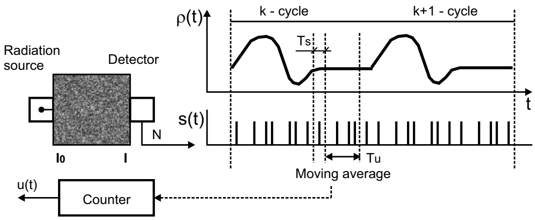


Fig.4. Radiometric density meter type OSC

$$\rho_m = \rho_o + \frac{(\rho_o - 1)V_{H2O}}{V_z} \qquad (1)$$

where:
$V_{H2O}$ – volume of water in a unit volume of the medium,
$V_z$ – volume of the dry material.
The ratio $V_{H20}/V_z$ measured during tests was ca. 0,65.

Detailed description of the radiometric density mater (RDM) was presented by Cierpisz (2013).

### 3.2. Natural radiation monitor

In gravitational coal preparation processes in heavy media or in jigs, the raw coal separation into the product and the refuse is performed at a given separation density (cut point), i.e. the density of the raw coal fraction which reports in 50% to the product and in 50% to the refuse (or middlings). Gravitational processes are based on a good correlation between ash content in coal density fractions and their density. The empirical relation between ash content and the density for a coal was presented by Newell & Grisafe (2004) and shown in Fig.5. The separation density then can be replaced by the separation ash if this parameter can be measured in the process. In the case of a jig the separation ash is the ash content of the coal layer below the product overflow weir (the end wall of the compartment). To measure this ash content we can measure natural radiation emitted by this part of the bed. Application of the natural gamma radiation measurement for determination of ash content in coals was reported by Taylor *et.al.* (2013) and Sikora & Smyla (2013). Natural radiation of coal is emitted by trace amounts of radioactive elements in the mineral matter such as $^{40}$K, $^{238}$U and $^{232}$Th. The content of radioactive potassium $^{40}$K as a radiation source in coal density fractions was investigated by Rog (2005) and shown in Fig.6.

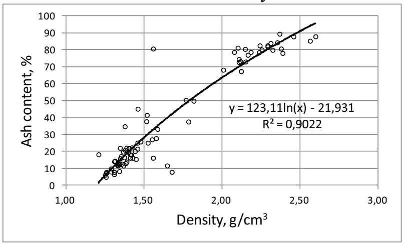


Fig.5. Empirical relation between ash content and density of coal

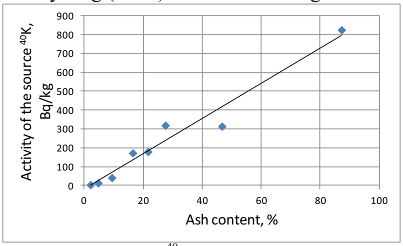


Fig.6. Activity of the $^{40}$K gamma source as the function of the coal density

To measure the separation density (separation ash) in the material below the overflow weir, the scintillation detector (100 x 70 mm) was used. The scintillation detector was properly screened from the radiation emitted by the oncoming bottom (discharged) material. The intensity of pulses at the output of the scintillation counter is relatively small - ca. 250 1/s for the density of material 1,8 g/cm$^3$ and 70 1/s for the density 1,4 g/cm$^3$). The time of measurement was set to 20s which gives standard deviation of the measurement ca. $\sigma$ = 0,008 g/cm$^3$. The view of the NRM monitor and its calibration characteristics are shown in Fig.7 and Fig.8.

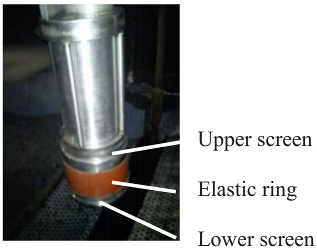


Fig.7. Natural radiation monitor (NRM)

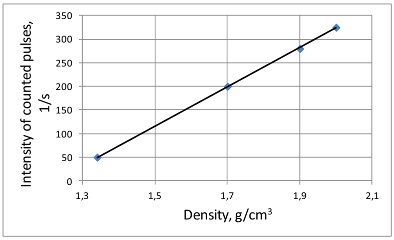


Fig.8. Calibration characteristics of the NRM

## 4. FIELD TESTS OF THE CONTROL SYSTEM

The radiometric density meter (RDM) and the natural radiation meter (NRM) were installed in the second compartment of the OM-24 type (Komag) jig, washing 0 ÷ 20 mm steam raw coal (Fig.3). The basic control system of the refuse discharge was the float-based BOSS 2000 system described by Bedkowski (2002). During the tests the third compartment of the jig was used practically as a product transporting unit. The time evolution of signals from the monitors was registered for the system operating with the float as a basic sensor and during significant changes in the feed tonnage (Fig.9). Both meters were calibrated in density units of the dry material. The NRM and the RDM signals for the correlation analysis were taken for longer periods (1-2 min) of their steady states. Both signals had different dynamics because they were averaged for different volumes of the material and different times of measurements (10 s and 20 s). The variations in the RDM signal are faster and greater as the RDM gathers information from the ca. 1,5 $dcm^3$ volume of the bed. In the case of the NRM the registered natural radiation is emitted by ca. 20 ÷ 25 $dcm^3$ volume of the bed.

Fig.9. Time evolution of signals from radiometric monitors: the solid line (NRM), the dotted line (RDM)

The standard deviation of the separation density indicated by the RDM is $s$ = 0,16 $g/cm^3$ at the mean value of 1,90 $g/cm^3$. These considerable variations in the separation density for the system with the float are caused by significant changes in the feed tonnage and its washability characteristics. The measurements were performed in the control system shown in Fig.10. The correlation coefficient r = 0,971 and the standard deviation of the NRM indications in relation to the RDM measurements was $s$ = 0,038 $g/cm^3$. This result indicates that the radiometric density meter RDM can be replaced effectively by the NRM meter especially in the case of control systems in which the separation density is stabilized at desired values.

Operation of the control system presented in Fig.10 for different values of the desired separation density has been shown in Table 1.

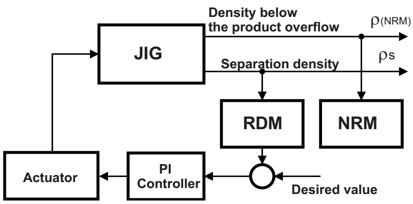


Fig.10. Control system with RDM in closed loop and NRM as the monitor

The separation density is kept fairly close to the desired values. The standard deviations of the separation density fluctuations around its desired value range from 0,023 $g/cm^3$ to 0,04 $g/cm^3$. These fluctuations have been significantly reduced in comparison to the control with the float only ($s$ = 0,16 $g/cm^3$).

Table 1. Results of the field tests of the control system

| Parameter | | Period of measurement | | | |
|---|---|---|---|---|---|
| | | **I** | **II** | **III** | **IV** |
| Desired value | g/cm$^3$ | 1,85 | 1,75 | 1,8 | 1,85 |
| Radiometric density meter RDM in the closed - loop control | | | | | |
| Mean value | g/cm$^3$ | 1,850 | 1,750 | 1,801 | 1,853 |
| Standard deviation | g/cm$^3$ | 0,023 | 0,040 | 0,035 | 0,026 |
| Natural radiation meter NRM (parallel indications) | | | | | |
| Mean value | g/cm$^3$ | 1,82 | 1,76 | 1,81 | 1,88 |
| Standard deviation | g/cm$^3$ | 0,035 | 0,041 | 0,043 | 0,056 |

The further reduction of fluctuations can be expected for the control system with optimized parameters. The indications of the NRM were registered in parallel with RDM measurements. The mean separation density indicated by the NRM differed from the RDM indications by ± 0,03 g/cm$^3$ and the standard deviation ranged from 0,035 g/cm$^3$ to 0,056 g/cm$^3$. Better correlation of both meter indications can be expected for the optimized geometry of the NRM measurement especially as regards screening of background radiation and reducing the undesirable radiation from the coal bed.

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# Optical analysis of the coal on belt distribution and its use in qualitative and quantitative measurements

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Abstract:
The authors presented a short description of two optical devices developed for the analysis of the cross-section of the coal transported by a belt conveyor. Devices applied a laser with a vision camera and infrared distance sensors. The results presented in the paper were collected during field trials at the coal preparation plant. Tested devices were installed near the ash analyzer RODOS-W which uses natural gamma radiation for the ash determination and microwaves for the moisture measurement. The influence on ash and moisture measurements of the changing profile of the coal cross-section on the belt was investigated. Authors discussed advantages and disadvantages of both methods basing on results of field tests and analysed their usefulness for other purposes. The belt conveyor scale installed nearby the ash analyzer provides information about the amount of coal in the measurement zone. The comparison of cross-section area with corresponding scale results was used to check the possibility to develop an optical scale without a strain gauge.

Key words:
Laser, infrared, coal distribution, cross-section, ash meter, image analysis, on-line measurement

## 1. PROBLEM DESCRIPTION

The coal processing technology is strongly related to belt conveyor transportation. Monitoring of each of the coal beneficiation processes is performed by a variety of devices, like on-line ash and moisture analyzers, belt scales and other specialized sensors. Control devices have their specific requirements which must be fulfilled to obtain reliable results. Very popular and widely used dual energy ash meters have a very narrow measurement zone which should be set in a particular place of the coal on the belt cross-section (Kryca,2013). This place is not always the centre of the belt. The measurements based on natural gamma radiation (NGR) are also sensitive to coal on belt distribution. Devices utilising X-rays fluorescence or X-ray transmission are out of scope of this paper because they are not so popular yet. X-rays transmission together with infrared and visible light optical analysis is used in separators in sorting systems rather than in ash monitors.

The authors presented a short description of two devices developed for the analysis of the cross-section of the coal transported by a belt conveyor. The results were collected during the field trials at the coal preparation plant. Tested devices were installed near the ash analyzer RODOS-W which uses natural gamma radiation for the ash determination and microwaves for the moisture measurement. The belt conveyor scale installed near ash analyzer provides information about the amount of coal in the measurement zone. Variety of methods for ash content determination in coal are results of different technological requirements concerned with places of installations, and desired accuracy. The measurement results are obtained in strictly defined periods of time which depend on the type of ash analyzer. The measurement times for different devices vary from 10 seconds for gamma back-scattering up to 100 seconds for NGR technology. Long time of measurement means that the sample taken for a laboratory test becomes larger to avoid the increase of sampling error. The ratio of the analyzed coal mass to the total transported coal mass and amount of coal being measured in a single measurement for



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different technologies were described in literature (Kryca, 2010). Another important information about coal quality is moisture content. Applied moisture meters together with ash meters provide information about calorific value determining the final price of the coal product. Microwave transmission method requires information about thickness of the coal layer because the microwave attenuation depends on the moisture content and amount of analyzed material as well. Belt conveyors scales used for production control do not solve all metrological problems. Typical scale measures mean values of the coal load in the measuring belt zone. The conveyor vibrations influences the strain gauge signal which must be averaged. The NGR ash meter with the relatively long time of the measurement calculates amount of coal being analyzed subtracting indications of the precise mass counter at the beginning and at the end of the measurement cycle. The critical point of that algorithm is to detect a period of disappearance of coal in measurement zone to discard the result (Sikora, 2012). The accuracy of measurement strongly depends on coal distribution across the belt. The calibration procedures assume that coal is distributed not symmetrically but without frequent changes of the profile. Experiences gathered on working systems show that material profile changes as a result of belt slippage or chute clinging. In such cases the distance between measured coal and the centre of the measurement zone changes and the result of the ash measurement decreases.

The dual gamma energy ash meters have very narrow measurement zone and changes in material distribution have different impact on results. The minimal thickness of the coal layer to be measured is about 10 cm. Assuming that the highest point of the cross-section is in the center of the belt where measurement is preformed we can expect that almost all the time the measurement can be done, but in the case when these two points are shifted one from another the measurement often will not be performed due to lack of coal. Installing skids at the chutes outputs is technologically sticky. Solutions based on moving isotopes and detector across the belt are effective – significant part of the time is spent on scanning empty belt to find position of highest point in cross-section. In such cases an Information from optical device would be desirable.

The experiments conducted in the laboratory and at the coal mine were focused on solving the problem of NGR ash meter accuracy shown in figure 1. The goal was to detect changes in the coal distribution in the cross-section and compensate its influence on the measurement results.

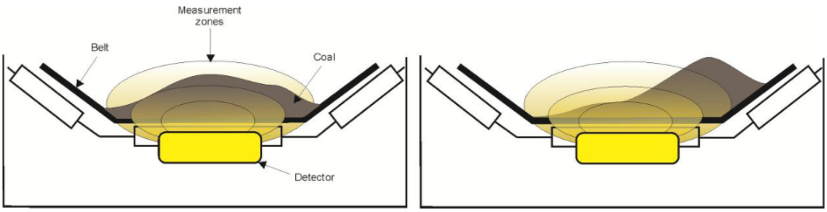


Figure 1. Different coal distribution on the belt

## 2. PROPOSED SOLUTIONS

The experimental works were conducted in the “Wujek” preparation plant where the NGR ash meter RODOS-W with the belt scale and the microwave moisture meter were installed. Two devices were tested, devices with different principles of operation, optical resolution and prices.

### 2.1 DISTANCE MEASURING SENSOR UNITS

Tested device was made with module sensors dedicated for distance measurement in desired range. The distance measuring sensor unit is composed of an integrated combination of PSD (position sensitive detector), IR-LED (infrared emitting diode) and signal processing circuit. Typical problems with infrared sensors are well known (Benet,2002, NIOSH, 2012). The differences in the reflectivity of the object and the environmental temperature do not disturb easily the distance detection because of adopting the triangulation method. This device outputs the voltage corresponding to the detection distance. The sensor with the MODBUS communication controller is installed in the PCV case with the transparent cover to avoid dust penetration into the optical path. Modules are grouped to control the desired width of the belt. The experimental installation was made of ten units. All sensor measurements are not performed simultaneously because the infrared beam of one unit can disturb results of another. The applied algorithm starts measurements sequentially in two steps – odd and even numbered devices. The acquisition time of single sensor is about 17 ms so taking into account time needed for simply averaging and transmission the effective scanning time is about 100 ms. The result of the distance measurement must be recalculated to obtain information about the coal layer depth. This calculation is performed in three steps. Firstly all sensors must be calibrated to converse results of analogue-to-digit conversion to millimetres. The example of such calibration is shown in figure 2.

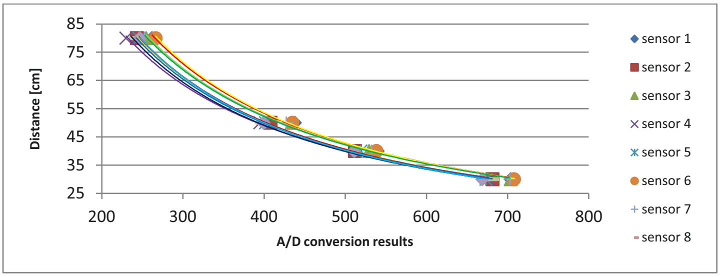


Fig. 2. The calibration of distance measurement units

The next step is to measure the distance from the empty belt. This measurement is an equivalent of compensating a belt weight for a typical belt scale. The result of this measurement is used to compensate the curvature of the belt being the result of pulleys arrangement. The final result is ~~a~~ the difference between the actual distance and stored in the memory the empty belt distance. The graphical representation of the results is a group of points which are connected by lines and the area of the polygon represents amount of material in the measurement zone. The desired information about symmetry of the coal distribution is calculated by dividing the area limited by polygons created on the left and right part of the belt. The perfect symmetrical distribution gives the result of the symmetry coefficient equal to 1.

## 2.2 THE LASER LINE

The second tested solution was the device with the laser line camera for the image analysis. The advantage of this device is a continuous measurement of a cross-section, not limited to several sensors. Unfortunately costs of application is significantly higher because of much more complicated software and expensive industrial components. The principle of operation of the developed device is based on the projection of laser light line on coal being transported on the belt and simultaneous~~ly~~ observation of light dispersion points. There are two possible measurement geometries shown in figure 3.

In case a) the laser is mounted at an angle of α to the axis of ~~a~~ the belt conveyor and the camera directly over the area of expected light dispersion. The camera fixing height is selected accordingly to the lens focal length covering the measurement area for all possible coal layer depths. The real coal depth could be calculated using formula:

$$h = b * tg(\alpha) \quad (1)$$

The changes in values from h1 to h2 will move position of the laser light from b1 to b2. The base point for calculations is the point where the laser line can be seen on the empty belt.

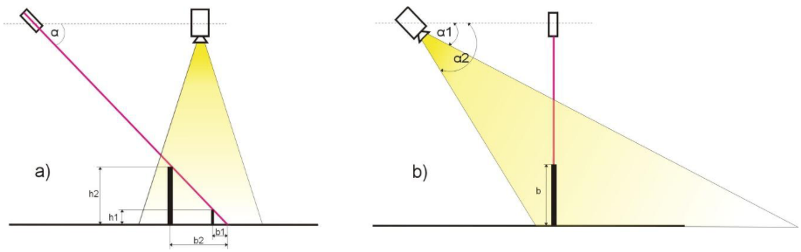


Fig. 3. Geometries for determining coal on belt distribution using laser technology

The sensitivity of this method can be increased by decreasing the angle α and increasing the distance between the camera and the laser. Assuming the angle value α=45° the measured value b is equal to the real coal layer depth. The figure 3 b) describes the case when the laser is installed directly over measured coal but the camera is moved aside to analyze the position of the laser line. In that case the position of the laser line also depends on the coal layer thickness but a real value is difficult to calculate due to changes in the angle α1 to α2. This error can be minimized by moving the camera further away from the laser end thus reducing angle changes. There is possibility of combining this two methods to increase method sensitivity but the required space for the installation becomes larger which sometimes is not acceptable.

## 3. FIELD TESTS

The experimental devices were installed near the ash analyzer RODOS-W which uses natural gamma radiation for ash determination and microwaves for the moisture measurement. The influence of the changing symmetry of the coal cross-section distribution on the belt on ash and moisture measurements was observed and measured. The belt width was 1000 mm and amount of coal being transported was near low acceptable limit for that kind of the ash monitor. In such cases coal crosswise distribution is important. The chutes located near ash meter changed periodically symmetry of the material distribution and finally influenced the measurement accuracy.

### 3.1 DISTANCE MEASUREMENT

The mechanical construction was designed for ten sensors and the small HMI (human-machine interface) module for calibration, performing necessary calculations and results observation as well. Output values were transmitted to the ash meter. The test site with installed optical sensors is presented in figure 4.


Fig.4 . The test site at a preparation plant (1- optical sensor, 2 – lead shield of the RODOS-W ash analyzer)

Results were presented on the LCD display and in the case of long-term disturbances the samples were taken to be checked in the laboratory. The comparison of lab and ash analyzer and moisture meter results was used to modify calculations performed by devices. The graph of changes in symmetry and the total area of the cross-section is shown in figure 5.

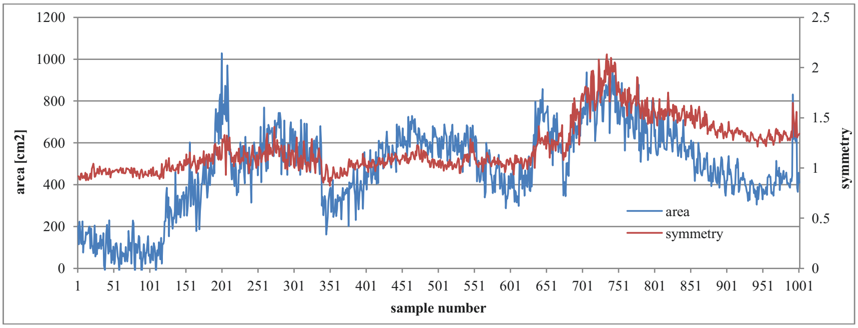


Fig.5. The changes in symmetry and total area of coal cross-section

The example above shows that changes in the cross-section (amount of coal being transported) not always are connected with significant changes in symmetry which can be observed near the sample 200. The lab results confirmed that coal movement aside the axis of conveyor increases the ash measurement error by 2% $A^r$ and the moisture error by 1% $W^r$. The trial approach of correlating the cross-section area with belt scale results was performed. The high correlation coefficient would confirm the possibility of replacing the scale with strain gauges with the optical one which would be desirable because of susceptibility for mechanical damages. The device with ten sensors has too low resolution for such application. Additionally changes in correlation were observed because of the specific gravity and the grain size fluctuations. The tests at the preparation plant revealed that boundary sensors measuring distance to skew parts of the belt have greater errors. Periodically observed excessive water makes thin film that reflects infrared light in the false direction. The sensors report the belt_slippage so the additional functionality of the device cannot be used. Monitoring the position of the lower belt eliminate that phenomena (Smyła, 2015).The belt slippage is the often occurring when water, clays or any other factor influences the friction between the belt and its drive rollers (Hadrygóra, 2011). The conveyor belt fires have been caused by belt slippage (MSHA, 2002). To prevent it, a belt slippage detection system should be provided to stop the conveyor drive automatically when belt slippage occurs.

### 3.2 THE LASER LINE

The application of the laser, camera and image analysis is able to precise calculate the cross-section area. The fluctuations of the specific gravity and the grain size of the measured material limit application of this method as the optical belt scale. Tests with two geometries gave similar results but another limitations were found:

- the low level of ambient light increases the time of exposition and due to the material movement line in analysed pictures are blurred (fig. 6a);
- the laser line discontinuity in picture(fig. 6b);
- the unsymmetrical ambient light makes difficult to set the proper exposition time (fig 6c).

a) b) c)

Fig. 6. The application of laser line for coal distribution analysis

## 4. RESULTS DISCUSSION

Both tested devices utilize the optical phenomena to determine the coal on belt distribution and both devices are exposed to dust. Half year lasting field tests was not disturbed by the optical path pollution although coal dust settled on all parts of the construction. The vertical down orientation of optical devices significantly reduced the signal degradation. The basic goal of performed tests was achieved - optical compensation methods of changes in coal on the belt distribution are applicable and this information reduce measurement errors of ash and moisture meters. The required changes in the ash analyser software are simple and easy to be applied. The optical belt scale can be used only when the transported material has stable grain size composition, stable density and fast camera is applied. That limitations are common for commercially available systems (Hense, Keytrade, 2015). Another task was to simulate measurement results of a dual energy ash meter where the data from a high energy isotope were replaced by the data from an optical device. The laboratory tests of compensation the attenuation of $Am^{241}$ radiation caused by the coal layer thickness confirmed that the distance measurement cannot replace the $Cs^{137}$ isotope.

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# OPTIMISATION AND CONTROL OF DENSE MEDIUM CYCLONE CIRCUITS

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## Abstract

The Commonwealth Scientific and Industrial Research Organisation (CSIRO) and the University of Queensland's Julius Kruttschnitt Mineral Research Centre (JKRMC) have conducted a number of Research projects focussed on the Dense Medium Cyclone (DMC) circuit in coal preparation plants. Research covered areas such as screen optimisation, drain rates of multi-sloped screens, medium quality and the effect of medium quality on the DMC operation. These projects have led to the development of instruments and techniques to measure the aperture and open area of screen decks, stroke and screen motion and the DMC overflow and underflow medium densities. This has provided researchers with a unique opportunity to investigate the factors that control the efficiency of DMC operation, and to determine control strategies to optimise the DMC process with particular emphasis on the underflow and overflow medium densities as an indicator of DMC and correct medium "health".

**Keywords:** *Dense Medium Cyclone, Dense Medium, Density Control, Dense Medium Circuit Optimisation, Dense Medium Circuit Control, Correct Medium, Medium Quality.*

## Introduction

Dense medium cyclone research in CSIRO and in the later years jointly with the JKMRC has been the result of funding provided by ACARP together with support from a number of coal preparation plants in both Queensland and New South Wales. The outcomes from this research has led to a better understanding of the operation of dense medium cyclone circuits in the coal industry and, as recent research has shown, has the potential to save the coal producers from loss of saleable coal due to cyclone instability as a result of low non-magnetic material in the correct medium. Research has also shown the effects of screen apertures on the loss of coal (O'Brien et al., 2007), identified required instrumentation (Firth, 2008) and developed instruments, new methods models and techniques to monitor the DMC operation (Firth et al., 2010).

Current practices in controlling the Dense Medium Cyclone circuits in coal preparation plants use a measurement of the coal ash value directly by sampling and analysis, the density set point will only be changed should the ash value from the laboratory prove to be outside of specification and this can result in the plant producing a coal product with a higher or lower ash value than specified for 2 to 4 hours



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while awaiting the laboratory result. Multiple coal cleaning circuits are normally optimised using the incremental ash. (Luttrell, 2013).

## Background

Australian coal preparation plants primarily employ two approaches to the control of the relative density (RD) of the correct medium used to control the separation density of the coal (Crowden et al., 2013).

***Dutch State Mines (DSM) Density Control System***

The DSM Control System has the medium from the product and reject drain of the drain and rinse screens returned to the correct medium sump and the dilute media from the rinse sections of the screens fed to a magnetite thickener for concentrating prior to magnetic separation (most modern plants are operated without a magnetite thickener with the dilute going to a dilute medium sump and then to the magnetic separators).The overdense media from the magnetic separator is sent to an overdense sump and is used for density control. The correct medium sump is usually of the draft tube arrangement.

***Rising Density System***

Medium from the product and reject drain sections of the drain and rinse screen is returned to the correct medium sump, the dilute is fed to a dilute medium sump and then to the magnetic separators. The overdense (RD 1.6 to 2.1) from the magnetic separators is returned directly to the correct medium sump which results in the correct medium RD increasing as more water is removed via the magnetic separators than that added to the system from carry over from the desliming screens. A wing tank sump and pump is added to the circuit to feed the DMC and some of the correct medium is allowed to overflow back to the correct medium sump. Water addition to the correct medium as it exits the correct medium sump is used in this instance for density control.

## The Medium

Finely ground magnetite is the medium of choice for the Australian coal preparation industry. While the magnetite being supplied to the plant has a nominal size distribution, the actual characteristics of the steady state medium found in the plant will be a balance between a number of activities occurring within the circuit:

- Some of the material in the feed magnetite has poor magnetic susceptibility and is quickly lost by the magnetic separators.
- The operation of the magnetic separators is such that complete recovery of magnetite is not possible.
- Some of the magnetite is not recovered from the drain and rinse screens and there is a steady loss to product or reject.
- The desliming screens remove most but not all of the particles less than the nominal bottom size set for the DMC circuit. This is one source of non-magnetic material in the medium.
- The clay bands and shale tend to disintegrate in the water as they pass through the DMC circuit and, since the majority of the medium reports to the 'correct' medium stream from the drain and rinse screens, there is accumulation of this material in the correct medium. This is the second source of non-magnetic material in the medium.

To control the level of build-up of material from the last two actions, some of the correct medium is also sent to the magnetic separators (the bleed) to reduce the level of non-magnetic material. The presence of non-magnetics in the medium will increase its stability, and the level of build-up of these non-magnetics is not normally closely controlled. In some cases a build-up of non-magnetics to increase suspension stability is desirable and clays (e.g. kaolinite) are sometimes added deliberately for this purpose. (Crowden et al., 2013).

The effective control of the DMC circuit medium is focused on three factors:

1. The relative density of the feed medium
2. Stability of the medium
3. Quantity of the medium available

To counter rising sump levels operators will often increase the bleed which can result in the removal of non-magnetics from the system and at low density result in stability issues.

***Medium Stability***
The separation density for a DMC decreases with increasing medium stability, for a given medium density. This dependence could affect a circuit's performance particularly at low separation densities if the amount of non-magnetic material varies significantly.

The ability to add some additional non- magnetic material to the correct medium in a plant should be considered, particularly for plants which will be operating at low feed medium densities. The redirection of a small amount of thickener underflow would be one source of this material. Even while the small coal (> 200 um) is in the medium, and would add to the apparent density of the medium, it does not contribute to the effective medium density.(Firth et al., 2011) and (O'Brien et al., 2013).

**The Intelligent Plant**
A project titled "Intelligent Plant" (Firth, 2008) was initiated in 2007. For the DMC circuit the project identified the variables that should be monitored and measurement accuracy required to keep the variables within "healthy" limits, in this case the change in the measurement required to change the separation density of the DMC by 0.01 RD (Table 1).

*Table 1 Changes required in variables to produce a RD 0.01 change in the DMC separation density*

| | | Lower | Base Case | Upper | % change in value |
|---|---|---|---|---|---|
| Separation Density | pRD50 | 1.465 | 1.475 | 1.485 | |
| Spigot Diam. | **Ds mm** | 390 | 360 | 335 | 8 |
| Vol. Flow | **Q $m^3$/hr** | 895 | 987 | 1100 | 11 |
| Pressure | **P kPa** | 126 | 140 | 154 | 10 |
| Feed Density | **$\rho$f** | 1.333 | 1.350 | 1.366 | 1.3 |
| Feed Solids | **FS tphr** | 275 | 250 | 225 | 10 |
| Med./Coal ratio | **M:C** | 4.75 | 4.9 | 5.1 | 4 |
| Vortex Finder Solids Loading | **DVFL** | 0.59 | 0.63 | 0.67 | 6 |
| Density of underflow medium | **$\rho$u** | 1.54 | 1.57 | 1.59 | 1.5 |

**Electrical Impedance Spectroscopy**
Measurement of the DMC product and reject medium is an important factor identified in the Intelligent Plant project. ACARP project C9045 (Hu and Firth, 2005) showed the use of Electrical Impedance Spectroscopy (EIS) to measure density of the underflow and overflow mediums, the medium to coal ratio and the correct medium density to within the change criteria shown in Table 1. The technique uses the measurement of electrical impedance between two electrodes over a frequency range from 100 Hz to 1MHz. The slope of the linear section of the spectrum decreases with decreasing RD.

**Monitoring the DMC circuit in a plant**
The second module of a plant (Firth et al., 2010) situated in Queensland was chosen for the installation of the monitoring instrumentation. The DMC circuit was of the wing tank design and the RD of the correct medium was measured after the addition of the water using a nucleonic gauge.

EIS sensors were placed in the drain sections of the product screens and in the reject drain section. A Hall Effect instrument constructed by the JKMRC was placed in the overflow pipe of the DMC and a magnetite monitor also designed and constructed by the JKMRC placed in the reject drain section prior to the EIS sensor. Another EIS instrument for the early part of this work was placed in the overflow of correct medium from the wing tank to the correct medium sump. Mass rate instruments based on the screen motion analyser were used to monitor the mass on the screens and to calculate the on line yield.

A number of the results recorded showed that there were unexplained factors affecting the operation of the DMC in particular it was noted that significant changes in the underflow medium density were evident. These issues were investigated in ACARP project "Influencing Factors for Dense Medium Cyclones" (Firth et al., 2011). It was identified that two areas needed quantitative investigation:

1. Quality of the medium in terms of viscosity and stability.
2. Volumetric flow of coarse solids through the cyclone.

This project addressed the first issue using a well-controlled pilot plant circuit with a 150 mm diameter dense medium cyclone in a closed circuit. Anecdotally it was known that non-magnetics could be an important factor in stabilizing the medium and the experiments were targeted to look at the effect of non-magnetics on the efficiency of the cyclone. Viscosity of the mediums were examined in ACARP Project C15053 (O'Brien and Firth, 2008) where it was found that at the concentrations of magnetite and non-magnetics in the medium used in most Australian coal preparation plants, viscosity is not considered to be an influencing factor. Thomas, (1965) published a graph showing viscosity versus volume fraction of particles which also shows that for medium slurries used in coal preparation the viscosities are low.

It was clear from experiments that the amount of non-magnetics in the medium has an important role in stabilizing the medium. This effect which increases the separation size and the partition Ep is one of the significant causes of uncontrolled DMC operation and if it is not causing surging operators would not have any indication that an issue has occurred. In ACARP Project C20051 (O'Brien et al., 2013) the authors showed that even after a short feed stoppage there is noted an immediate decrease in the non-magnetic concentration. Raw coal feed is the major source of the non-magnetics in the medium and this will vary with the amount and type of coal being fed to the plant.

The instruments at the Queensland plant and plant operating data are available via a VPN connection from CSIRO to the plants LAN network. Figure 1 shows the data from some of the plants instruments together with the CSIRO instruments installed at the Queensland plant. In this instance the density is changed at about 10:30 from an RD of 1.34 to an RD of 1.41 to do this the plant has stopped the water injection and as the correct medium sump level is high they increased the bleed from 30% to 80% then back to 60% it remains at 60% for a few hours before its dropped to 40%. Unfortunately this method of changing the density by increasing the bleed has resulted in the overflow density increasing over the course of the day indicating unstable operation, a state that the operators would be unaware of until the next ash value was received. The high bleed rates removes not only the water but significant amounts of non-magnetics from the medium with the result that the medium at these low density levels becomes unstable.

The cost of the instability due to low non-magnetics was investigated in ACARP C23046 (O'Brien and Firth, 2015). As an example low non-magnetic concentrations in the correct medium (7.5% w/w) when targeting an ash value of 7% can result in the loss of between US$125k to US$150k for every 100k tonnes of ROM coal processed.

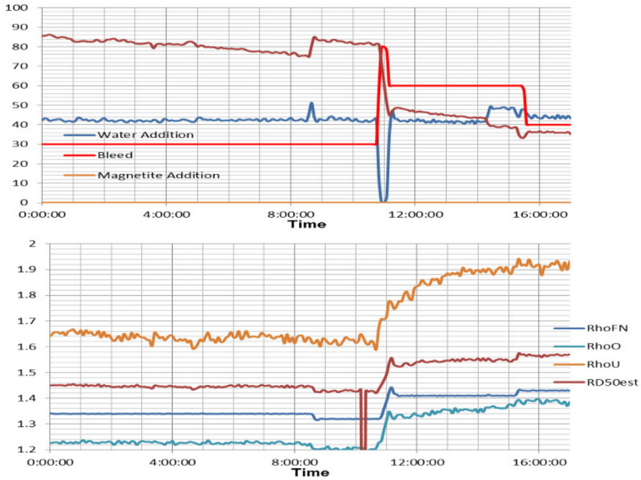


*Figure 1 Example of data examined from the plant instruments over a period of 1 day.*

## Conclusions

Conclusion from the latest research monitoring the DMC circuit has shown that:

- The overflow and underflow densities need to be monitored.
    - Options for the controlled addition of non-magnetics.
- Approaches to control the underflow density need to be devised
- There are good methods to change the medium density and bad methods which can cause instability of the medium for changing the medium density particularly at low medium densities.
- The amount of non-magnetics in the circuit appears to be sensitive to some changes in water injection, bleed level and magnetite addition.
- Medium quality is not controlled or monitored in plant operation and unstable operations can go unnoticed by the operators unless obvious signs such as surging occur.


## Acknowledgements

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# Handling Data in Coal Quality Monitoring

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**Abstract**
Bretby Gammatech systems have been in the field providing customers with information since 1994 and seven years prior to this under British Coal. The method of on-line detection has not changed much over two decades. However, the way we handle data most certainly has. This paper will describe the current suite of Bretby Gammatech equipment for detecting the ash in coal using the Natural Gamma method and what we can now do with this enormous amount of data being collected in terms of getting it to the people who need it in the most suitable format, wherever they may be in the world. Any reporting software will have to be compatible with industry standard operating systems and a client's own management systems, which gives our engineers an exciting challenge. We will look at recent installations around the world and how our Matrix Data Delivery is giving customers control of the Coal processing system, when they need it and where they need it.

**Key Words**

Natural Gamma. Coal Quality Monitoring. Matrix Data Delivery. EyeGraffix. Real Time. PassKey.Ash Detection

## Introduction

Bretby Gammatech have been producing coal quality monitoring systems since 1994 when the company was first formed. The founding directors had all previously worked for British Coal, a state owned mining company and had been involved in the first attempts to measure coal quality using natural gamma radiation seven years previously. Bretby Gammatech now provide systems around the globe to customers with widely differing infrastructures.

Bretby Gammatech provide products including both portable monitoring systems, such as the Ash Probe, which can be used with piles of coal at different locations; and online systems, such as the Ash-Eye and Heat-Eye, which are mounted on conveyor belts, providing quality information in real time. The conveyor mounted systems can generate a large amount of real time data which must be processed and passed to the appropriate destinations and it is this transfer of data that this paper is primarily concerned with.

In our experience, the nature of some of the information technology infrastructure at sites around the world, can be quite outdated and it can never be assumed that there is any supporting IT system at a



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customer's site and therefore, the monitoring systems must essentially include all the mechanisms required to handle and move the data without requiring complex servers or communications systems that require constant maintenance and can only be configured by trained professionals.

The solution has been to create a new method of data transport that can be handled entirely by the monitoring computers connected to the Bretby systems, requiring only a network connection to deliver data and no support from a customer's own IT staff, who historically don't often get involved in external projects.

The main principle of the new data handling systems, termed the Matrix Data Delivery method, is that any computer running the new Bretby EyeGraffix suite of software can physically connect to up to 25 natural gamma monitoring systems via a direct connection such as an RS485 signaling cable but can then mirror the data for any one, or all, of those systems to up to ten other EyeGraffix terminals simultaneously. These other terminals will see the data as though they themselves are physically connected to the monitored online system(s). They in turn can pass this data onto other EyeGraffix terminals over a local or wide area network connection to additional EyeGraffix terminals, linking together to form a matrix of terminals, or nodes, where data can be passed from any node to any other in order to reach its ultimate destination, which with an internet connection, can be anywhere in the world.

This is currently being used to allow a number of systems throughout the globe to be monitored by Bretby Gammatech engineers in the UK and can be used to make data available to any of a customer's own offices or key stakeholders around the world.

**Natural Gamma detection**

Traditionally, coal quality was measured in terms of it ash content which would normally be done by taking coal samples from the conveyor belt and analyzing them in a laboratory. The results would be available some considerable time later, perhaps several days if an external laboratory was used. To alleviate the delays, techniques were developed by several vendors to provide a means of determining the ash content in real time as the coal passed by sensors on the conveyor belt. The most common of these methods was to pass gamma radiation beams through the conveyed coal and then measuring the attenuation of the signals ((IAEA, 1986). This had the distinct disadvantage of requiring radiation sources and therefore all the necessary associated safety procedures, individual worker dose monitoring and problems with disposing of the radiation generation equipment once the systems were no longer required.

In the 1980s, the possibility of measuring the gamma radiation naturally emitted by the main coal impurities was being investigated (Wykes et al, 1989) and the founders of Bretby Gammatech were significantly involved in this.

The theory is that principal contributing isotopes in the ash are potassium K40 and members of the uranium and thorium radioactive series. The level of natural gamma radiation emitted from a given weight of mined material increases monotonically with ash content and a system calibrated to particular coal sources by means of comparing a range of samples measured with the on belt system against laboratory results for the same samples, allows the system to determine the ash content with a relatively high degree of accuracy (around ± 1.5% for run of mine coal) (Wykes et al, 1989).

Whilst the instantaneous measurement of gamma radiation is not consistent due to the very nature of the radiation source, averaging the measurements over periods of time produces a more representative measurement. This was eventually put into practice with the development of the Natural Gamma Coal Quality Monitor (NGCQM) system by Bretby Gammatech (Taylor, 2000)

**Ash-Eye/Heat-Eye and EyeGraffix**

Over the years the original NGCQM system has been developed into the Ash-Eye on belt monitoring system, which utilises the same principles as the original designs, adding enhanced data processing and functionality. The Ash-Eye system, as the name implies, is designed principally to determine the ash content of conveyed coal in near real-time.

The Heat-Eye system is an extension of the Ash-Eye, which uses the addition of a microwave over-belt moisture analyser, to allow calculation of the Calorific Value of the conveyed coal.

Whilst the Ash-Eye and Heat-Eye systems are stand-alone and can present their data by the local display on the main processor cabinet, most customers prefer to present the data on a monitoring computer connected via an appropriate communications link, typically RS485 running a proprietary protocol. The monitoring computer collates all the data and calculates average ash content over periods of time ranging from a few seconds to an entire shift, allowing the customer to get the data they need in the required resolution.

The latest EyeGraffix software suite takes this further by providing a much more modern user interface, the ability to generate a number of user configured reports automatically and email these to multiple recipients. Each terminal can monitor up to 25 Ash-Eye or Heat-Eye systems and larger systems can be accommodated by linking together EyeGraffix terminals.

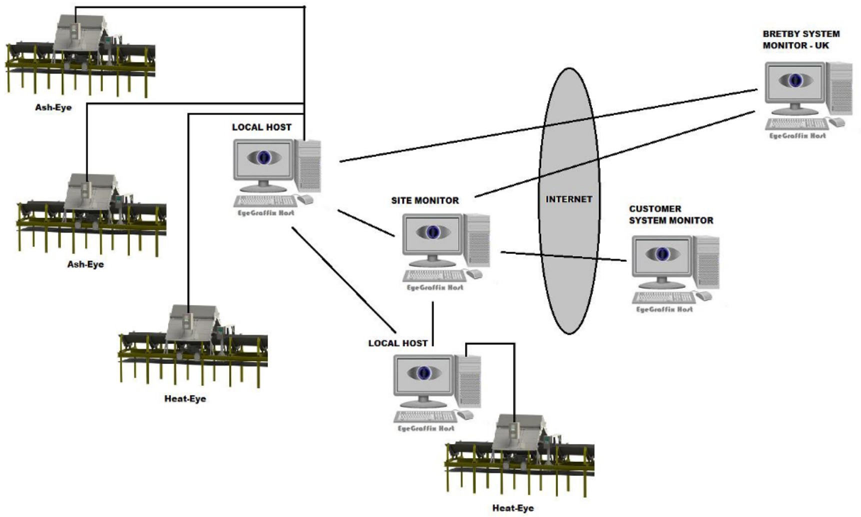

Figure 1: EyeGraffix system information display screen

**Matrix Data Delivery (MDD)**

Most customers want the data to be analysed and then reported in terms of average figures for each work shift or the averages on an hour by hour basis (or some other appropriate resolution) , which can be provided as a spreadsheet, sent automatically or on demand via email from any of the directly connected EyeGraffix host terminals. However, there are occasions when a customer wants the ability to drill down into the data from a remote location and in order to achieve this, the large amounts of raw data needs to be passed to their local EyeGraffix terminal. Similarly, Bretby Gammatech can also offer to receive the data from a customer's on site equipment at their UK base in order to track system performance and make adjustments remotely. In such cases, the large volumes of raw data generated by the system each second need to be passed in their entirety to all remote EyeGraffix terminals rather than just the processed data. To achieve this, the Matrix Data Delivery (MDD) system was developed. This is essentially a method of providing large volumes of data across, and via, a number of terminals/nodes using the most basic of infrastructure and therefore minimising the amount of support required from the customer. A network connection is all that's required in most cases.

Each host terminal can connect to up to 25 online monitoring systems by direct physical connection (i.e. an RS485 serial interface). Any of these sites can then optionally be made available to the MDD system, such that all data from the individual systems is passed out to any of the other terminals connected by MDD. Therefore remote host terminals can connect to any such site via MDD, either through a Local Area Network connection, or through an internet connection if available. In turn, any sites monitored in such a way can also be made available to other host terminals using the MDD system.

Figure 2: A typical EyeGraffix based system




For a customer's satellite office, there could be a single EyeGraffix terminal which links over the internet to other local host terminals connected to actual Ash-Eye/Heat-Eye systems at different sites but any other interested parties within the office can then link to the single terminal so only one internet connected terminal is required to establish communications, minimising the effect on bandwidth of connecting lots of terminals over the slower internet connection rather than a high speed office local area network.

Should the route be lost through one particular terminal or link, then the MDD network can be quickly reconfigured so that the data takes a different route, using as many terminal nodes it takes to get to the source data.

Each EyeGraffix terminal is usually configured in such a way as to make available any received site data to any other EyeGraffix terminal that connects to it. This means that complex reconfiguration is not required in the event of a loss of link. Usually the terminal affected needs only one simple parameter change to then connect to another node through which it can get its data without needing to know how the data is getting to this new node. A further development under consideration, is automating this process so the entire MDD network becomes self-configuring under fault conditions.

The MDD system was first installed at a number of locations in Kazakhstan operated by ArcelorMittal. There are currently five on-belt systems at two separate locations being monitored by three local EyeGraffix terminals on site, with data available to ArcelorMittal management at locations outside of Kazakhstan as well as directly to Bretby Gammatech in the UK, who are able to diagnose any reported issues immediately.

**Why not build a web site?**

Perhaps the most obvious way of making data available to multiple users over large areas is to establish a web site, a solution which has the added advantages of allowing any user with the appropriate access level to get the data from any location using any browser enabled device. Whilst this method has many clear advantages, the principle disadvantage is the need for a web server to be setup and maintained at the customers premises or externally on behalf of the customer. Once again, the problem we have encountered with this solution is the lack of an interface between a customer's IT department and the engineers who control and use the system. Whilst we can arrange to get network and internet access as this is a common task within the plant, setting up the necessary database and website servers and linking them to our installed systems proves to be a step too far. Therefore our solution has had to have minimal involvement from other departments and as such, the MDD method restricts the configuration and maintenance of the system to the individual computers installed, which more often than not, are supplied by Bretby Gammatech, meaning the link can be set up quickly and efficiently.

**Security Concerns**

With direct connections to computers over the internet, there will always be a level of concern of the security of the system, not just in protecting a customer's potentially commercially sensitive data but also in the risk of providing some form of *back-door* onto the computer and the local network to which it is attached.

There are a number of ways in which this concern has been addressed. First is the use of a unique encrypted PassKey which both ends of a connection must be provided with in order to communicate but secondly, the protocol used for the connection is not a standard IT protocol but a proprietary one designed specifically to meet the needs of passing the relevant data packets around but without providing any other facilities, such as connecting to elements of the operating system on the connected computers.

Such a protocol leaves no possibility of any other system utilising the link to gain access to any part of the computer or network.

## Future Developments

Currently, the next generation of Ash-Eye/Heat-Eye systems are under development utilising a Programmable Logic Controller (PLC) as the processing heart of the system. Apart from improvements in scalability, standardisation and production, this also allows far easier integration into a customer's other control and monitoring systems, by using standard protocols such as Modbus. It can also give us the possibility of linking the actual systems directly into the data matrix. Whilst using an industry standard protocol has advantages, the protocol is converted to the Bretby Gammatech proprietary one when connect to the MDD matrix to maintain the security of our system.

A further development, concerns the use of Bretby host website and web applications. Whilst the idea of using a web site or web application to access the data worldwide has many difficulties as described previously, we are currently investigating hosting a secure dedicated web server that each customer's systems can optionally link to, to provide a web based user interface – but without requiring customer involvement in the installation and maintenance of the system. There would, of course, be sufficient partitioning to ensure that each customer's data will be distinct and securely separated from any other customer's data. The data would be held on a Microsoft SQL Server database back-end with links to the on-site equipment through the MDD system, or in the case of the new PLC based equipment, by secure Modbus connection directly to the processor (when the facilities provided by MDD are not required and a secure enough link to the site can be established).

Such a system will, in turn, allow us to produce mobile apps on Android and Apple iOS as well as Microsoft platforms on all devices to make the data available to our customers through a much faster user interface when on the move.

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# Part IV
# Analysis, Processing and Preparation of Coal Slimes and By-Products of Coal Preparation, Coal-Mining and Combustion

# TECHNOGENIC COAL DEPOSITS: ACTUAL STATUS AND PROCESSING PROSPECTS

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**Abstract:** Russia is one of the world leaders in coal production and processing. The coal industry has accumulated about 15 billion t of man-made solid wastes. The dumps of coal-fired thermal power plants contain about 1.7 billion t of ash and slag waste. Rational solving of the problem of efficient use of industrial waste depends on a number of factors, namely, material composition of waste, its aggregative state, quantity, technological characteristics, etc. The problem of non-waste technologies has ecological, resource saving, technological and technical, economical and organizational aspects. There have been determined the areas of efficient use, technical requirements and innovative trends in the processing of technogenic coal deposits, main economical and technological conditions of processed wastes use as energy materials, in the production of construction materials, for recovery of alumina, rare and precious metals, production of ferroalloys, in the agriculture and others. The economical and technological conditions of cost-effective industrial implementation of processes of using slimes as fuel, e.g. slimes processing into water-coal fuel and fuel-slime briquettes have been substantiated. A problem of both ecological and economical value of coal recovery from coal-bearing overburden at surface coal mines has been solved. The prospective viability of ash and slag waste separation into organic and mineral components by flotation with a preliminary separation of aluminosilicate microspheres has been proved experimentally. A classified list of areas for the utilization of ash and slag waste of thermal power plants has been suggested.
**Key words:** Coal waste, classification, storage, material composition, processing technology, fields of application, efficiency of utilization

Russia is one of the world leaders in coal production. In-place coal reserves of Russia account for one third of total world coal reserves and one fifth of world explored reserves. For the production of 1 t of coal some 4-5 t of overburden rock is stripped in surface mining and up to 0.2-0.3 t of mine rock is produced in underground coal mines. In addition, in 2015 coal preparation plants generated more than 30 mln t of solid waste and coal-fired thermal power plants (TPPs) produced 25 mln t of ash and slag waste (ASW).

Today the coal industry has accumulated approximately 15 billion t of solid man-made waste, of which 70% in Kuznetsk Basin, and there are 1.7 billion t of ash and slag waste in the ash-disposal areas of coal-fired thermal power plants [1-5].

Solid wastes of coal mining, preparation and burning are discharged to dumps which occupy large areas of valuable ploughlands, worsen landscape of the territory, are the sources of environmental pollution, and their storage involves relatively high capital and operational costs. Rational solving of the problem of efficient use of industrial waste depends on a number of factors, such as material composition of waste, its aggregative state, quantity, technological characteristics, etc.

The objective of this work is to determine the areas of efficient use, technical requirements, technical solutions and innovative trends in the processing of technogenic coal deposits based on the analysis of basic regularities of their physical and chemical properties development, aggregative state, quantity and natural heterogeneity.

Storage of coal waste is associated with significant capital expenditures and operating expenses, removal of land from use and harmful environmental effect. At the same time, so-called wastes by their composition and properties in most cases represent economically valuable mineral and organo-mineral raw materials for different industries and their processing reduces consumption of traditional mineral raw materials and fuel.

In a number of instances production of some types of marketable products from technogenic raw material (wastes) is appreciably easier and cheaper than using primary raw material. The problem of non-waste technology has ecological, resource-saving, technological and technical, economical and organizational aspects.

The main directions in the processing and use of wastes of coal mining, preparation and burning are the following: use as energy commodity for burning in the fluidized bed process or for gasification; use as water-coal

 
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suspension; production of briquettes; production of construction materials (bricks, porous aggregate for cell concrete, ceramic tiles, tiling, etc.); recovery of alumina, rare and scattered elements and precious metals, production of ferroalloys; production of fertilizers for agriculture; fills construction, stowing of underground workings, land reclamation, construction of roads, dams, foundations and other artificial earthworks (Figure 1).

First of all, consider processing of finely dispersed coal slimes. At present there exist different methods of solving this problem: 1 – conversion of slimes into water-coal fuel (WCF); 2 – production of briquettes; 3 – gasification; 4 – burning in fluidized bed furnaces (FB). The conditions of economic industrial implementation of any particular method and fuel production require economical and technical justification.

Efficiency of burning of high-ash wastes in fluidized bed furnaces has been proved by practical implementation of this method in China, USA, Belgium and Germany.

One of the promising methods of carbon-bearing products processing is gasification. Product gas has calorific values of 3.3-5.2 MJ/m$^3$. The process is efficient when ash content of coal waste is less than 60 %.

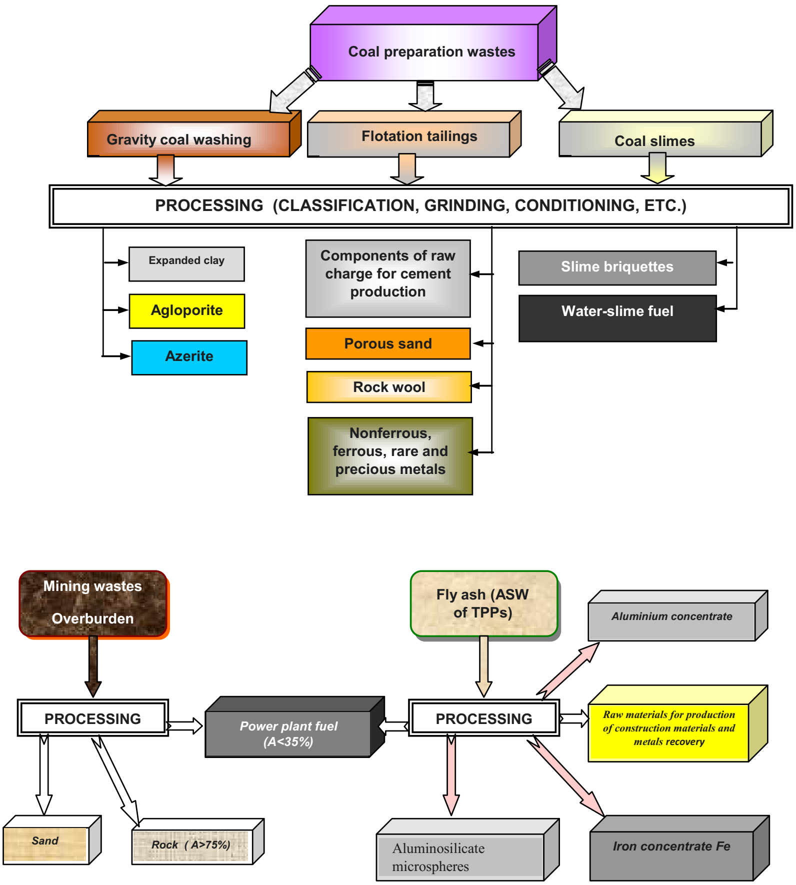


Figure 1 – Separation and processing of solid wastes of coal mining, preparation and burning

Coal slimes are very cheap, but efficiency of fuel produced from them is questionable because of their high ash and water content. Cost-effectiveness of use of slimes as water-coal fuel or briquettes depends on the value of thermal energy which can be produced by burning this fuel reduced by the energy consumed for its preparing. Production of briquettes from slimes seems more promising for commercial implementation.

It is obvious that the recovery of coal from coal-bearing overburden rocks at surface mining operations is a vital task. Coal losses with coal-bearing overburden rocks with ash content A=50-75 % at coal opencasts are as high as 10-15% of total volume of coal.

The analysis of physico-mechanical properties of coal-bearing rocks shows that their processing by traditional methods (heavy medium separation, jigging, etc) is not economically feasible. In the IOTT Institute there has been developed a steeply inclined separator. It incorporates the principle of countercurrent specific gravity separation in vortex tubes with a big number of recleaning cycles of products being separated. At present coal preparation plants at 16 surface mining operations in Kuzbass are fitted with steeply inclined separators.

It makes sense to evaluate the efficiency of all branches of industry on the basis of the balance between the main product output and the volume of technogenic wastes produced. The most marginalized in this respect are the production units of Fuel and Energy Complex (TPPs).

The share of expenses for the recovery, stockpiling and storage of ash and slag waste is about one third of the cost of electrical energy produced at coal-fired TPPs.

In Russia every year some 30 million t of ash and slag wastes are produced and the level of their processing and use is very low (5-10 %). The ash dumps of 172 Russian coal-fired TPPs have accumulated more than 1.7 billion t of such wastes; they occupy a territory of 28 thousand ha. About 70 % of these resources are situated near the largest cities in the European part of the country.

Mineral part of ash and slag wastes comprises iron oxides, aluminium, silicium, rare, non-ferrous and precious metals. It is a valuable raw material for many branches of industry and construction activities. By the complexity and multi-component nature of material composition the wastes of TPPs are technogenic deposits, which can be processed using well-known processing and hydrometallurical methods with a recovery of valuable components.

At present rare metal potential of coal is practically not used. However, while the issues of efficient recovery of non-ferrous, rare and precious metals require major studies, environmentally friendly and non-waste technologies of recovery from fly ash of unburnt coal, aluminosilicate microspheres, hematite and raw material for the production of a wide range of construction materials have for the most part been developed and are used (Figure 2).

The IOTT Institute has experience in the development of non –waste technology of fly ash processing of a number of Russian thermal power plants. Based on this technology the main characteristics of corresponding flowsheets and regimes of flotation and separated products dewatering have been determined for different kinds of fly ash.

This technology makes it possible

- to recover large part of wastes of power plants by conversion into a wide range of construction materials, such as additives to cement, filling material (coarse and fine) for light concrete, sand and clay alternative for the production of bricks, components for lime and porous concrete, raw material for building ceramic, rock wool, slag-silicate and others;
- to return the coal recovered as flotation concentrate into the power generating cycle;
- to use it as a complex sorbent for sewage cleaning from organic and inorganic impurities;
- to separate aluminosilicate microspheres
- to increase the content of microelements in the mineral part to the values sufficient for their recovery from this product.

Integrated processing of ash and slag waste makes it possible to not only recover unburned coal, but also to recover aluminium oxides, magnetite and precious metals [6]. Recovery of aluminium oxides is possible by flotation and the methods of two-stage acid extraction of aluminium with orthophosphoric and sulphuric acids [7-9]. From 1 t of ash and slag waste it is possible to produce 100-120 kg of waste coal, 40-80 kg of iron-ore concentrate, 200-600 mg of gold and 600-800 kg of construction material (inert mass).

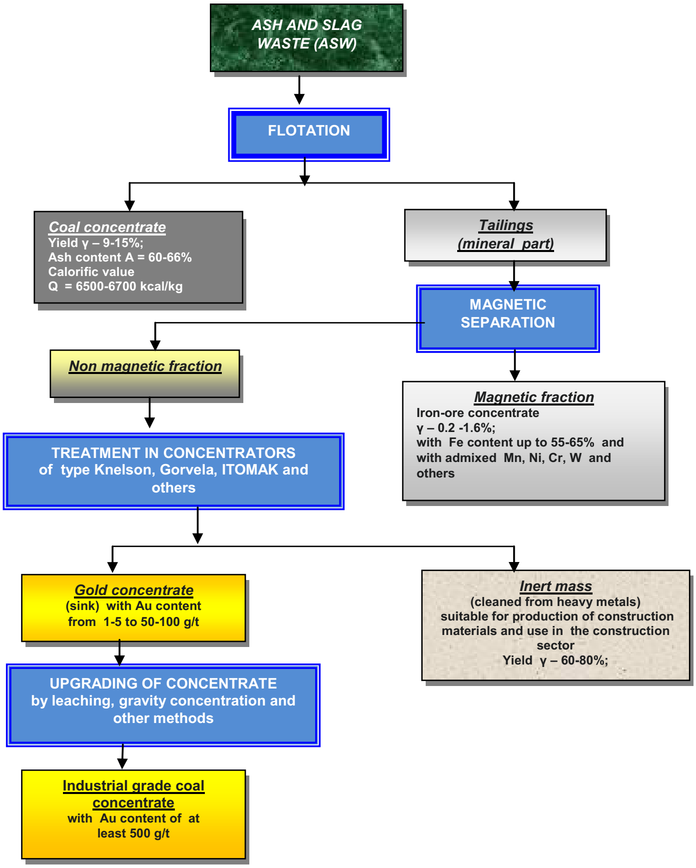


Figure 2 – Process flow diagram of ash and slag waste processing

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# Physical-chemical grounds of producing valuable trace elements concentrates and preventing unfavorable toxic trace elements actions on the environment while coal processing

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**Abstract**

An element which concentrations in coals are ≤ 0.1% is called a trace element (TE). Such elements are divided into two groups: potentially valuable (PVE - Ge, Ga, U, Se, Te, Co, Ni, V, Re, Hg, Mo, W, Zn, Pb, Cr, Bi, Zr, Nb, Ta), TRE (rare earth metals including Y and Sc), PM (precious metals – Ag, Au, Pt and other platinoids); and potentially toxic (PTE - As, Be, Cl, Co, Cr, F, Hg, Mn, Ni, V, Pb, Sr, Sb, Se, Te, Tl, V, Zn, U, Th, Ra, Rn, $^{40}$K). According to our calculations, for the majority of cases related to production of TE marketable compounds or industrial concentrates it is economically feasible to use by-products (called the primary concentrates, or PRTE, containing PVE) of coal processing, i.e. with maximally possible utilization of coal energochemical potential. There are proposed quantitative equations allowing selection of these products on the calculations basis (by the programs of chemical thermodynamics) or with help of experimental studies on laboratory pilot scale or industrial installations. Results of these studies can also be used to develop recommendations for the implementation of measures to reduce to acceptable levels the adverse environmental impact of PTE compounds formed during the processing of coals and other fossil fuels.

According to the studies, for almost all the PTE, the PRTE may be obtained as captured fly-ashes or their mixtures with slags with the minimum expenditures while coal combustion. aimed at production of heat and electricity energy. The selection of combustion devices and modes of coal burning to produce primary concentrates with optimal composition for various PVE, and also development of proposals to reduce the adverse environmental impact of PTE compounds depend on their distribution between the combustion products. Summing up the study's results, PVE and PTE may be divided into three groups. 1$^{st}$ group contains TE forming gaseous compounds in zones of combustion devices with the highest temperature values. Such compounds are being removed from these zones by gas flow along with the other combustion products and are then after cooling condensed under temperature intervals of 800-120°C (Ge, Ga, Mo, W, Ag, Au, As Pb, Sb, Tl, Zn, partially Cl, F, Hg, Se, Te and Re). 2$^{nd}$ group contains mostly such compounds that are formed in solid or liquid phases in the high temperature zones and are being distributed between slag and fly ashes. Fly ashes are removed with gaseous combustion products to purification equipments (fiber or electrostatic filters), caught there with other solid particles before throwing into the atmosphere (Be, Co, Cr, Mn, Ni, V, Pb, Sr, Cr, Co,Ni, Zr, TRE, Th, U, Be). The 3$^{rd}$ group contains TE forming gaseous compounds in the high temperature zones of furnaces. Such compounds are removed there along with other combustion products and are not condensed after cooling of the latter (partially Hg, and, depending on coals mineral matter composition, Se, Cl, F, Te and Re).

Fly ashes with PVE of the 1st group, fly ashes and slags with the PVE of the 2nd group could be considered as PRTE. Maximal extraction of PVE of 1$^{st}$ and 2$^{nd}$ groups from gas flow could be reached at purification of gas flow from solid particles at degrees of ≥ 98% with the fiber or electrostatic filters applied to gaseous compounds of combustion products evolved into the atmosphere. Under such conditions also a minimal hazardous effect of PTE of 1$^{st}$ and 2$^{nd}$ groups is reached. In order to achieve it for PTE of the 3$^{rd}$ group it is mandatory to utilize the special methods of gaseous compounds purification before releasing them into the atmosphere.

 

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**Key words**
Potentially toxic and potentially valuable trace elements, distributions while preparation and thermal processing, combustion, recovery conditions, primary concentrates, marketable compounds, fly ashes, slags

Over 70 elements of Mendeleev's Periodic system are found in coals and other solid fossil fuels. Most of them are characterized by concentrations of ≤ 0.1% and are therefore called trace elements (TE). Many of TE are divided by two groups: potentially valuable (PVE: Ge, Ga, U, Re, Se, Te, Co, Ni, V, Hg, Mo, W, Zn, Pb, Cr, Bi, Zr, Nb, Ta, TRE (rare-earth elements including Y and Sc), PM (precious metals – Ag, Au, Pt and other platinoids) and potentially toxic (PTE: As, Be, Cl, Co, Cr, F, Hg, Mn, Ni, V, Pb, Sr, Sb, Se, Te, Tl, V, Zn, U, Th, Ra, Rn, $^{40}$K). For the majority of TE the average concentrations (clarkes in coals) are evaluated on the basis of coal within all the basins of the Earth [1-5, 7-10]. Such values should be considered as tentative due to the fact that their estimation methods do not take into account many important factors: such as coal mass corresponding to the analyzed sample, highly inhomogeneous distribution of TE along the area and depth of coal layers, different precision and sensitivity of analyses, ets. Concentrations of the same TE differs from sample to sample of coal from the same field or their layers. Some coal deposits demonstrate concentrations of TE, such as Ge, Au, Ag, TRE, ets., of ten or more times higher than their clarkes in sedimentary and igneous rocks, and also clarkes of coals and other solid fossil fuels. Such coals are called metal-bearing and are considered as feed stocks for obtaining the marketable compounds and concentrates of PVE, including Ge, TRE, PM and other (uranium-bearing coals of some deposits are commercially utilized during many years, but producing of uranium from them is outside of this report).

According to technological and economical calculations, the raw material for production of PVE marketable compounds or concentrates is not mined coals but by-products or solid wastes from processing of coals and other solid fossil fuels used for the realization of their energochemical potentials. Such by products (wastes) are called the primary concentrates of PVE (PRTE) [1].

Quantitative relations are proposed [1,6] for determination of enrichment degrees of PVE or its relative concentration ($Y_{ig}$) in *g*-product of a process along with the extraction degree of *i*PVE in this *g*-product ($U_{ig}$).

$$Y_{ig} = C_{ig} / C_{oi} \tag{1}$$
$$U_{ig} = Y_{ig} \cdot \gamma_g \tag{2}$$

where $C_{ig}$, $C_{oi}$ are concentrations of *i*PVE in a considered product and initial coal or other solid fossil fuel; $\gamma_g$ – is a product yield (% mass or relative unit).

The product is called a PVE concentrator if $Y_{ig} \geq 1$ and bearer of PVE if $U_{ig} \geq 0.5$ (50%). It is found that the same product is sometimes not concentrator and bearer of PVE simultaneously.

Results of the processes of high-ash coals preparation show that for some PVE and PTE bearers are usually the low-ash concentrates and the so-called middlings with ash contents close to the one of initial coals, but the values of $Y_{ig}$ are usually found to be varying between 0.6 - 1.2. It was found for coals of some deposits that the concentrators of PTE (Co, Cr, Hg, Mn, Ni, V, Pb, Sb, Se, Tl, Te, Zn, ets.) are high ash fractions, but the corresponding $U_{ig}$ values are usually lower than 25 – 30%. Therefore, coals beneficiation does not allow to substantially decrease PTE amjunts in combustion products in comparison with combustion of initial coals.

As for the thermal coals processing, values of $Y_{ig}$ and $U_{ig}$ can be found using chemical thermodynamic software, but in such case there may be found a series of errors connected with some factors: industrial processes may not achieve an equilibrium state, software database may have imperfections and other. Therefore, it is necessary to provide experimental studies.

Summing up the thermo dynamical calculations results, along with investigations at laboratory installation, pilot and industrial plants installations it could be concluded that the most perspective means of PRTE obtaining is using fly ash or fly ash and slag of coals combustion [1,6]. According to the PVE behavior in such processes, PVE may be divided into 3 groups:

- 1st group include TE which form in high temperature zones of a furnace gaseous compounds removed from them by a gas flow with other combustion products and condensed after cooling of the latter in temperature intervals of 800 – 120 $^0$ C (Ge, Ga, Mo, W, Ag, Au, As, Pb, Sb, Tl, Zn, partially Cl, F, Hg, Se and Re). For TE of this group, the primary concentrates are fly ash particles removed at $a_i \geq 98$ -99% from gaseous combustion products with dry purification installations (fiber or electrostatic filterers). Under such conditions, using furnaces for coals combustion with slag removal coefficients > 0.6 utilization leads to obtain fly ashes with $Y_i \geq 50$ (in comparison with PVE concentrations of fuels) and $U_i \geq 75 – 90\%$) at $a_i \geq 98$ -99%. It is observed at $a_i \geq 98$ -99% that that the decreasing of PTE release into the atmosphere not less than 94 – 96%.

- 2nd group – TE formed compounds in solid and liquid phase in high temperature zones, i.e. distributed between slag and fly ash, removed from them with gaseous products from them with gas flow and captured while furnaces cleaning and gaseous products purification from solid particles (Be, Co, Cr, Mn, Ni, V, Pb, Sr, Cr, Co, Ni, Zr, TRE, Th, U, Be, partially Cl, F, Hg, Se, Te and Re). The primary concentrates are fly ashes or their mixtures with slags. Combustion temperatures and type of furnaces are chosen depending on PRTE processing technology. Decrease of PTE release to atmosphere could reach 97 – 98% at $a_i \geq 98$ -99%.

- 3rd group contains TE forming gaseous compounds in zones of highest temperatures of the furnaces. Such TE are removed from high temperature zones along with other combustion products with gas flow but are not condensed after cooling of the latter till 120 $^0$ C (Hg, and, depending on coals mineral matter composition, Se, Cl, F, Te and Re). Marketable compounds of PVE or decrease of PTE release into the atmosphere is reached by utilization of wet scrubbers and with simultaneous usage of dry purification methods for gaseous products and for Hg [11] adding of active coals.

There are the largest practical production experience for obtaining of primary Ge concentrates at coals combustion. Methodology of calculations is developed for choosing technological schemes and equipment of energy-aimed combustion (heat performance ≥ 70 – 90%) for coals with ash contents ≤ 30% and Ge contents in fly ashes bearing 50-200 times higher Ge concentration in comparison with raw coals and $U_{ig} \geq 70 – 92$ % [6]. Such methodology could be used also for obtaining PVE concentrates in fly ash referred to 1st group.

Also, a methodology for calculations, choosing of installments and technological schemes for coal combustion with 90 – 95% decreasing of PTE release for 1st and 2nd groups are proposed [1].

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# Zeolite Synthesis as Potential Application of Coal Fly Ash

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**Abstract**

The fly ash, a by-product of coal combustion in thermal power plants, is one of the most complex and abundant of anthropogenic materials, which large accumulations represent a serious environmental threat. To reduce the environmental burden and improve the economic benefits of energy production the science and industry focus on the transformation of coal combustion by-products into new functional materials. The fly ash of power plant (Pechora coal basin, Russia) and reaction products were studied by modern analytical methods. As a result of the hydrothermal reaction several types of zeolites were synthesized from the fly ash: analcime, faujasite (zeolite X) and gismondine (zeolite P). It was shown that the experimental conditions (temperature, reaction time and alkali concentration) have a significant influence on the type of zeolite and its content in the reaction products. The series of experiments resulted in more detailed schematic diagram of zeolites and other phases. New data on the sorption-structural characteristics and cation-exchange properties for $NH_4^+$, $Ba^{2+}$ и $Sr^{2+}$ for the fly ash from Pechora basin coals were obtained.

**Keywords:** zeolites, coal fly ash, hydrothermal synthesis, sorbents, ion exchange, barium, ammonium, strontium.

**Introduction**

The fly ash is a bulk industrial waste of coal combustion in thermal power plants, steel mills, etc. Therefore, the problem of utilisation of this technogenic waste, occupying large areas and causing damage to the environment, is very important. Many papers were published on the properties of fly ash and possibilities of its use [1-4], but mass utilization lacks. One of solutions may be represented by the synthesis of zeolites from fly fly – important type of sorbents for mining-chemical industry. Conversion products of the fly ash in zeolites has many applications, including ion exchange, molecular sieves and adsorbents [5-8]. Identification of new applications has a real commercial interest: the list of marketable products is expanding and energy costs are reduced, environmental risks are reduced and the efficiency of sustainable development in the region is increased. Technologies of zeolite synthesis from fly ash are being constantly improved in both the experimental (variations of temperature, pressure, co-reagent and other methods of exposure) sphere, and the material composition of the initial raw [9-12]; the quality, application area and cost of final product depend on it.

The aim of this work is to study the conditions of hydrothermal reaction (temperature, reaction time, alkali concentration) to the received products of reaction, and also to study sorption-structural parameters of synthesized zeolites and their cation-exchange properties for $NH_4^+$, $Ba^{2+}$ и $Sr^{2+}$.

**Objects and Methods**

For the experiments, we used the fly ash from thermal power plants of Pechora coal basin.

The synthesis methods of zeolites were based on [12]. Firstly, using a magnetic separator we removed ferriferous phases, which do not participate in the synthesis of zeolites. Dry fly ash is mixed with the solution of sodium hydroxide (NaOH) in a certain ratio, mixed thoroughly, and the suspension was placed in an autoclave. The resulting products of hydrothermal reaction was washed with distilled water and

 
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dried. This resulted in powders consisting of the mixture of zeolite and unreacted residue in various proportions.

The chemical composition of the fly ash and synthesized products was determined with the help of X-ray fluorescence analysis (energy dispersive spectrometer MESA500W, Horiba). To study the morphology and chemical composition we used a scanning electron microscope TESCAN VEGA 3 LMH with energy dispersive Oxford Instruments X-Max. The phase composition studies were performed on powder diffractometer (Shimadzu XRD 6000, radiation $CuK_{\alpha}$, Ni filter) within range 2 - 65° 2θ angle.

The specific surface area, total pore volume, volume of micropores were measured by the analyzer of the surface area and pore size Quantachrome NOVA 1200e.

To study the kinetics of extraction of ammonium and strontium by the synthetic zeolites and to evaluate the corresponding sorption capacity the experiments were performed on cation exchange of the specified cations from aqueous solutions of $NH_4Cl$ и $SrCO_3$ within 5-1440 minutes using a shaking device. The initial concentration of $NH_4^+$ and $Sr^{2+}$ were 100 and 25 mg/l respectively; o S:L ratio = 1:500. Ammonium concentration in the solution was determined by the photometric method with Nessler reagent according to Guiding document 52.24.486.2009 [14] using a photoelectric colorimeter KFK-2 at a wavelength of 440 nm; the concentration of strontium was determined by ICP-AES spectrometer Vista MPX Rad. The cation-exchange capacity of barium was determined by Bobko-Askinazi-Aleshin method according to GOST 17.4.4.01-84 [13].

Exchange capacity and removal efficiency of metal ion on the adsorbents were calculated according to the equation (1) and (2), respectively:

$$q_e = \frac{(C_0 - C_e)}{m} \cdot V, \qquad (1)$$

$$R = \frac{(C_0 - C_e)}{C_0} \cdot 100\%, \qquad (2)$$

where $q_e$ is the amount of exchanged ion at equilibrium (mg/g); $C_0$ and $C_e$ are the initial and equilibrium metal ion concentrations in solution (mg/l), respectively; V is the solution volume (l); m is the adsorbent weight (g); R is removal efficiency of metal ion from solution (%);

**Results and discussion**

**Initial fly ash.** X-ray diffraction (Fig. 1) showed quartz, mullite, magnetite and hematite in the fly ash. The broad "hump" (area of increased background) on the diffraction pattern in the area 15–35 °2θ indicates the presence of amorphous phase (probably silicate or aluminosilicate glass).

The main components of the chemical composition are $SiO_2$ (57.78 %) and $Al_2O_3$ (18.25 %), iron oxide content is about 9.0 %, oxides of other elements - 7.42 %, loss on ignition - 7.90 % [15].

The fly ash is represented under the electron microscope by globules, which are divided by the chemical composition to aluminosilicate and iron containing. The aluminosilicate globules composition is predominated by $SiO_2$ (from 41.82 to 61.27 %) and $Al_2O_3$ (from 17.03 to 22.8 %); FeO and $Fe_2O_3$ (up to 8.31 %), MgO (up to 4.83 %), $K_2O$ (up to 3.05 %), $TiO_2$ (up to 1.04 %) and $Na_2O$ (up to 0.93 %) are also present. Globule size varies from the first to about hundred micrometers; on the surface bubbles and elongated structures are observed.

On the surface of iron containing globules both flat areas and skeletal forms are observed, which are significantly different from each other by their chemical composition. The skeletal forms have a high content of iron oxides (68.14-74.66 %) and low $SiO_2$ (1.06-6.22 %), $Al_2O_3$ (1.33-4.17%) and CaO (0.48-3.59%) contents. On the flat areas iron oxides content is greatly reduced (19.29-31.81 %), $SiO_2$ and $Al_2O_3$ content increases (27.12-37.86 and 2.06-6.22 %, respectively); CaO presents in amounts of 10.45-25.3 %. Globule size ranges from several to tens micrometers. Globules, which contain smaller globules within, are often observed.

**Hydrothermal synthesis**. Two sets of experiments were carried out. In the first set the effect of temperature of hydrothermal reaction on zeolite synthesis was studied (reaction temperature 80, 95, 140 and 180 °C, reaction time 12 hours, the ratio of NaOH: fly ash = 1:1, NaOH concentration 3.0 mol/dm$^3$).

The second set of experiments studied the influence of reaction time and concentration of alkali on synthesis process (reaction temperature 140 °C, reaction duration 2, 4, 6 and 8 hours, ratio of NaOH: fly ash = 1:1, NaOH concentration 1.5, 3.0 and 4.5 mol/dm$^3$). Earlier we showed the scheme of transformation of fly ash into zeolites in [15].

According to [10, 11], this process consists of three stages: dissolution, condensation and crystallization. When fly ash interacts with sodium hydroxide it is dissolved, and Si and Al are released to the solution. Then, the condensation of silicon and aluminum ions occurs, followed by gelling and nucleation (forming nuclei or crystallization centers) and crystallization of zeolites.

The synthesis results in powders consisting of the mixture of zeolite and unreacted residue in different proportions, in which output was 70-80 % of the weight of the initial fly ash.

**Effect of reaction temperature on the synthesis of zeolites.** In the result of the reaction at 80 °C, the intense reflections of quartz were diagnosed; no newly formed phases were detected (Fig. 1). Electron microscopic studies revealed numerous globules destroyed by alkaline solution.

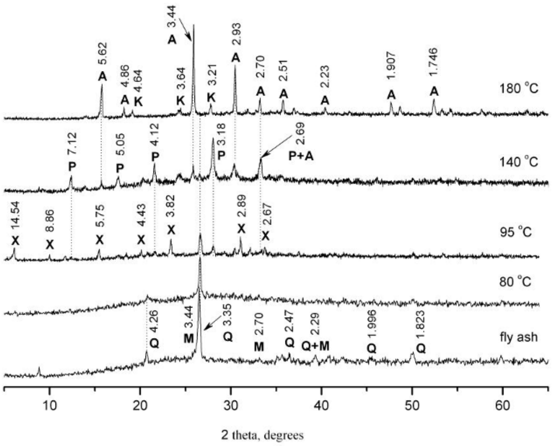


Fig. 1. Diffraction patterns of products synthesized at 80, 95, 140 and 180 °C for 12 hours (Q - quartz, X – zeolite X, P – zeolite P, A - analcime, K - cancrinite). The interplanar distances are given in Å.

By increasing the reaction temperature to 95 °C silica the intensity of quartz reflections decreased, i.e. it was dissolved in alkaline solution. Alongside with quartz reflections the intense reflections were determined, which are characteristic for faujasite zeolite (zeolite X), and weak reflections characteristic for gismondine zeolites (zeolite P). By Si/Al ratio, the zeolites are low silica: silica-aluminum module of zeolite X varies from 1.51 to 1.57, zeolite P varies from 1.65 to 1.69. SEM images present numerous crystals of zeolite X with octahedral shape with the size of 1-3 μm. Zeolite P crystals have a rounded shape, their size is about 5 μm [15].

The diffraction patterns of the reaction products obtained at 140 °C showed zeolite P and analcime, weak quartz reflections were also present. Zeolite P is more high-silica compared to the phase obtained at 95 °C: Si/Al ratio varies slightly from 1.93 to 1.94. Silica-aluminum module of analcime varies from 2.12 to 2.21. SEM images showed that zeolite P formed skeletal crystals with size of 10-15 μm. Analcime crystals with size of 15-20 μm were observed.

The reaction at 180 °C resulted in the formation of analcime and cancrinite; no quartz reflections were diagnosed. Si/Al ratio of analcime varies from 2.00 to 2.15. The analcime crystals are formed by tetragonal faces, their size ranges from 15 to 25 μm. Cancrinite columnar crystals with length of up to 2 μm and about 200-300 nm in diameter are often observed on the surface of analcime, indicating later crystallization of cancrinite.

These results indicate that the reaction temperature influences the type of synthesized zeolite, which differs by the efficient diameter of entrance windows, and according to classification [16], they are divided into narrow, medium and wide porous types. It is determined that increasing reaction temperature results in the formation of narrow porous zeolites; at 95 °C zeolites X formed that are related to wide porous type, at 140 °C – zeolite P, related to medium porous type, and at 180 °C – analcime related to narrow porous type. Pore size of zeolites were as follows: analcime, 0.26 and 0.42×0.16 nm; zeolite P, 0.31×0.45 and 0.28×0.48 nm; and zeolite X, 0.74 nm.

**Effect of reaction time and alkali concentration on zeolite type.** The set of experiments resulted in the approximate crystallization field of zeolites and other phases (hydrosodalite) at 140 °C, the reaction time from 2 to 8 hours, NaOH concentration 1.5, 2.9 and 4.5. Additional experiments allowed specifying the schematic diagram of zeolite crystallization (Fig. 2) compared to the diagram represented in [19]. In the area P* on diffraction pattern only weak peaks are observed, possibly belonging to zeolite P, and indicate a low crystallinity of the phase.

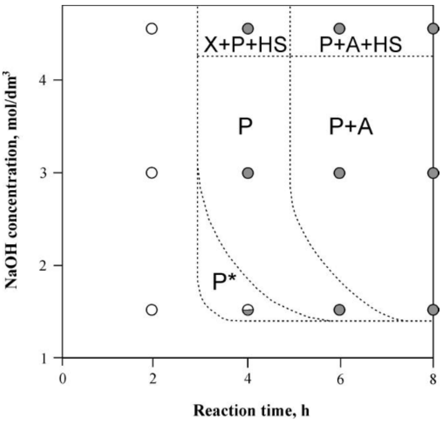


Fig. 2. Schematic diagram of zeolites crystallization specified after the set of additional experiments compared to [19] (X – zeolite X, P – zeolite P, A - analcime, HS – hydrosodalite); P*: XRD weak peaks, probably of zeolite P, show low crystallization phases on plot

As seen in Fig. 2 the wide porous zeolites X are formed after 4 hours of reaction at a high concentration of alkaline solution (4.5 mol/dm$^3$). Longer reaction leads to the disappearance of the metastable phases of zeolite X and the occurrence of more thermodynamically stable - zeolite P and then analcime.

Zeolite P is crystallized under a wide range of reaction conditions. At the same time, the fields of crystallization of analcime and zeolite P are significantly overlapped, that is, at the same conditions of the hydrothermal reaction, the mixture of zeolites in various quantitative relations is formed. Higher concentrations of alkali results in the increase of the content of narrow porous phases (analcime) compared to zeolite P, and contributes to the formation of non-zeolitic phase - hydrosodalite.

**Sorption-structural and ion exchange properties**

Fig. 3a represents a specific surface area (SSA) of the initial fly ash and hydrothermal reaction products obtained at different alkali concentrations and temperature 140 °C, depending on the duration of the reaction.

When comparing Fig. 3a with the schematic diagram (Fig. 2), dependence of the specific surface area of the reaction products on the type and zeolite content in the mixture is observe. In this case it is necessary to take into account the fact, that it is impossible to measure the SSA of narrow porous zeolites, such as analcime, by the BETmethod (low-temperature nitrogen sorption), since the pore size of analcime (0.26 and 0.16 × 0.42 nm) is smaller than the diameter of nitrogen molecule (0.32-0.35 nm). The measured SSA of pure analcime was only 1.5 $m^2/g$.

As a result of two-hour reaction, the SSA is increased by 4-6 times compared to the SSA of the initial fly ash, equal to 5.6 $m^2/g$, although the zeolite phases were not found in these products. Smooth increase of the SSA of the reaction products obtained at the concentration of 1.5 M NaOH, with increasing reaction time, is connected with the increase of zeolite in the mixture. In the series of experiments with concentration 3.0 M NaOH the SSA reaches its maximum after 4 hours of reaction, and remained at approximately the same level at increasing of the duration. The reaction products, obtained at 4 hours duration and concentration 4.5 M NaOH, are characterized by a high SSA equal to 269 $m^2/g$, connected with the formation of wide porous zeolite X that is related to wide porous zeolites. Sharp decrease of the SSA at increasing of reaction time is a result of the disappearance of zeolite X and appearance of more thermodynamically stable zeolite P and analcime.

The total volume of pores and the volume of micropores are directly correlated to the specific area of surface and determined by the type of synthesized zeolite and its content in the mixture.

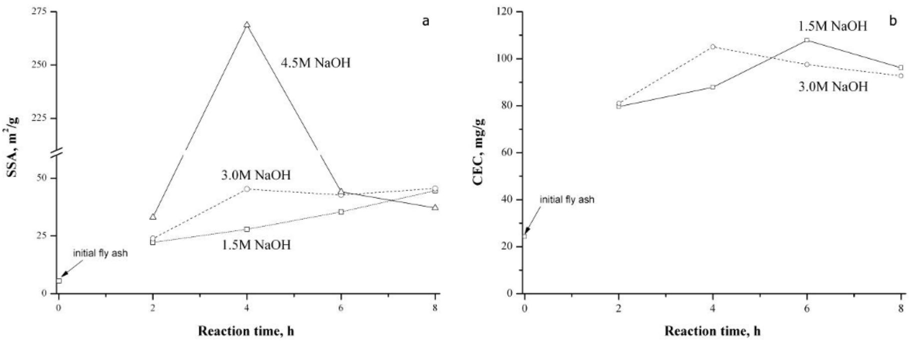


Fig. 3. Specific surface area (a) and cation-exchange capacity for $Ba^{2+}$ (b) of fly ash and hydrothermal reaction product obtained at 140 °C, alkali concentrations of 1.5, 3.0 and 4.5 mol/$dm^3$ for 2, 4, 6 and 8 h

Fig. 3b shows a cation-exchange capacity (CEC) for barium of the initial fly ash and hydrothermal reaction products obtained at 1.5 and 3.0 M NaOH, depending on the duration of the reaction. The cation-exchange capacity of the products, obtained by the two-hour reaction, is increased by more than 3 times to 79.6-81.0 mg/g compared to the initial fly ash CEC, although zeolite phases were not identified. Increasing of the CEC to 105.1-107.8 mg/g in the reaction products with duration 4-6 hours is connected with the formation of zeolite P. Reduction of the CEC is a result of reducing content of zeolite P and analcime formation, which has a low exchange capacity with respect to barium (14.4 mg/g).

Experiments on cation exchange for ammonium and strontium were carried out on a sample obtained at 140 °C, alkali concentration 1.5 mol/$dm^3$ and duration of reaction 8 hours, which main mineral phase is zeolite P with analcime impurity. Both exchange reactions have a high rate, so that after 5 min the maximum ammonium concentration 31.9 mg/g and strontium 12.3 mg/g are reached respectively, and further increase in the exposure time does not result in the change of capacity. Removal efficiency of $NH_4^+$ и $Sr^{2+}$ for the given conditions of the ion exchange is 63.9 and 95.4 %, respectively.

**Conclusions**

Efficient sorbents have been synthesized on the basis of technogenic materials resulted from the burning of coal of thermal power plant (Pechora coal basin, Russia). Final product is represented by powders - mixture of zeolite and unreacted residue in various proportions, which yield was 70-80 % to the weight of the initial fly ash. The influence of hydrothermal reaction (temperature, alkali concentration, reaction time) on the type of zeolite was presented. It was shown that the sorption-structural characteristics of the synthesized products depend on the type and content of zeolite in the mixture. Additional experiments specified the schematic diagram of crystallization compared to the diagram presented in [15]. The synthesized zeolites exhibit a relatively high activity in relation to $Ba^{2+}$ and less to $NH_4^+$, и $Sr^{2+}$ cations.

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# Coal Utilization in Thermal Power Plants. Influence of Coal Quality on Technical-economical and Ecological Indices

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Abstract

A revive of state of the art of Russian coal firing thermal power plants (CTPP) and coal consumption were done. The equipments of these plants are very old. The efficiency of electricity production is low, especially for combined heat and electricity (CHP) in summer regimes. It was shown that only 3 units 225 MW and unit 300 Mwt Kashirskay CTPP have been start in operation for last 5 years. The 300 MW unit for AC combustion in circulating fluidized bed (CFB) have constructed on Novocherkasskay CTPP just now. The leader electricity productions companies aspire to diversification of coal consumptions. The influences of coal quality on CTPP operation indexes, including ecological indexes and choosing of optimal technical decisions were considered. It was shown that combustion of enriched coals gives a small rising of technical indexes and a good rising of ecological indexes. These advantages are realized on utility size units with high supercritical parameters and long coal delivery. CFB boilers could be implemented for wastes of coals enrichment. Results of the development of a new generation of coal-fired CHP with high efficiency and low pollutions were done.

Key words

Coal-fired thermal power plants, coals consumption, coal quality, enriched coals, combustion technologies, circulating fluidized bed, ecological indexes.

**Introduction. A brief revue of state of the art of Russian CTPP**

About 110 condensing TPP and CHP are coal generation plants in Russia [1]. Some experts says that were are no to intensive rising of coal generation in Russia just now. CTTP will construct in Siberia and Far East regions. The reason is relative close places of electricity generation and coal mining. A grate development of CTTP in other regions (for example in the European part of Russia) is unlikely as CTPP have a low efficiency instead of natural gas turbine electricity generation [1]. The most importance aspects of the development of coal generation around the world and in Russia were considered on II International Scientific and Technical Conference "Implementation of solid fuels for efficient and environmentally friendly production of electricity and heat", October 28-29, 2014. In the report of Dr. V. F. Veselov [2] was shown, that a share of coal in fuel balance of last year's was very stable and relative high in Siberia and Far East regions. But a share of natural gas in fuel balance of European part of Russia is raised from 80% to 98%. CTPP remain a largest part of internal demands on coal, however internal demands rising is limited by low electricity demands and the competition with natural gas and nuclear power stations. The motivation to replacement of existing power units should be created by new ecological demands, efficiency demands, equipments age demands and so on. However, CTPP project cost recovery demands special economic rules [2]. In the report of RPA [3] was shown, that structure of coal consumption is characterized by variety of ranks and mining fields. The main coals for Russian CTPP are kuznetsky hard coals, kansko-achinskie brown coals and import ekibastuzskie coals. Low grade coals are about 90 % of all CTPP consumption coals. There is no consumption of enriched coals on Russian CTPP.

A state of the art in PSC "INTER RAO UES" may be considered as an example of coals consumption for Russian CTPP. This Company has second place of summary installed units capacity after JSC 'SCEC. The coal share in fuel consumption of PSC "INTER RAO UES" is 25 %, installed units



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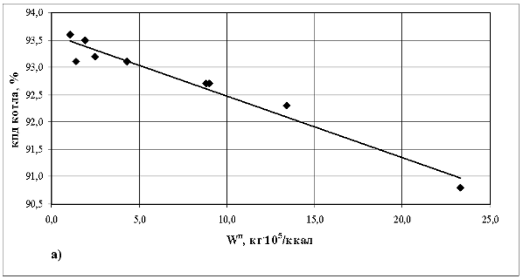
capacity is 5419 MW. This share is not essential change in the nearest time. The ekibastuzskiy coals have 37 % and brown coals 2B and 3B (13 % and 22 %) in coals consumption. Brown coals are typical for Siberian CTPP and ekibastuzskiy coals – for Uralian CTPP and CTPP of West Siberia

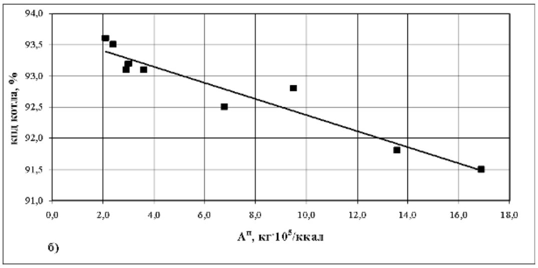
The high share of transport expenses in final coal price is typical for CTPP of European part of Russia. (50-60 %). High coals prices and very old equipment leads to low call for electricity of CTPP. The main principles of coal strategy of PSC "INTER RAO UES" are diversification of coal consumption, vertical integration, long time contract and unification of coals marks. It is possible to implement enriched coals for CTPP of European region in case of economically proved prices of enriched coals.

Russian CTPP have different power – about 30 % units are 100 – 200 MW [4] and near 50 % - 200 – 300 MW (150 MW – 28%, 200 MW – 35%, 300 MW – 29% [5]). Most of these units were in operation 35 – 50 years; reliability and efficiency of units equipment's are low. The net efficiency are 30 – 36 % only, there are no equipped new process control systems. High NOx, $SO_2$ and dust pollutions are typical for some CTPP. The replacement of old coal CHP is most important problem of Russian co-generation [6]. Some main results are presented in the paper. Only 3 units 225 MW (Horonorskay and Cherepetskay) and unit 300 MW Kashirskay CTPP have been start in operation for last 5 years. The 300 MW unit for AC combustion in CFB have constructed on Novocherkasskay CTPP just now.

**The impact of fuel quality on the performance of the power plant and the choice of optimal technical solutions**

The impact of fuel quality is considered an example of calculation of CFB boilers for combustion of various fuels from low-calorific, and of high-wet brown coal - to high-calorific, low-ash hard coal. Calculations for the whole range of fuels were carried out in relation to the power of 225 MW turbine K-225-12,8-3R (steam temperature 565/565 $^0$C). Similar calculations were made for once-through supercritical boiler (330 MW) [7]. Figure 1 shows the effect given to the lower calorific value of moisture, ash and sulfur content in the boiler efficiency. These data are typical for coal-fired boilers, but the CFB boiler efficiency decrease with increasing ash content and sulfur content is more significant. The important thing is that the operating and capital costs can be reduced, including the costs of environmental protection measures and ash dumps. The most favorable conditions are created for a large shoulder delivery, tightening requirements for harmful emissions in relation to large units at a higher supercritical steam parameters.





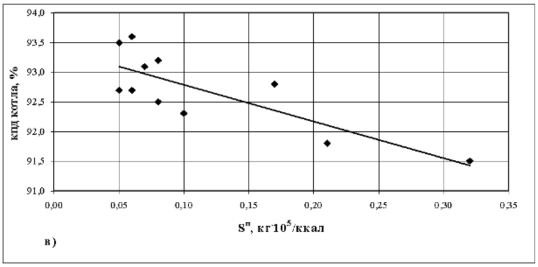




Fig. 1 - Effect of lower calorific value, moisture, ash and sulfur content on boiler efficiency.

Pursuant to the Order of the Government of the Russian Federation in PSC "INTERr RAO UES", it was decided to hold some investigation on possibilities of burning enriched coals in their power units. Foundation of supporting the scientific, technical and innovative activity "Energy without borders" signed a contract with JSC "VTI" on work "Study and selection of alternatives to coal for combustion at thermal power plants of JSC" Inter RAO UES ", taking into account characteristics of the boiler equipment." Technical possibility and economic feasibility of burning the enriched coals (rank T) in boilers of 300 MW units (TPP-110 and P-50) and in boilers in Kashirskaya and Cherepetskaya TPP were analyzed. Available in Russian enrichment plants, only "Kuzbassrazrezugol" can supply the required quantities of coal. And optimal for burning in boilers of Cherepetskaya and Kashirskaya TPP is enriched Krasnobrodsky T brand coal. With respect to the base fuel calorific value of coal Krasnobrodsky enriched above 10% while half the ash. Analysis of the properties of coal and thermal calculation results shows that no reconstruction of equipment is required with the transition to burning enriched coals. There are no changes in auxiliary power consumption. Significantly reduces emissions of dust and sulfur oxides and nitrogen oxides are approximately equal. It is shown that the transition to burning enriched coals will add the costs amount about to 1.17 billion rubles per year for the Kashirskay TPP and 0.305 billion rubles for Cherepetskaya TPP. Additional costs due to the substantial increase in fuel prices. Thus, the introduction of the practice of burning enriched coal to the power station in Russia is possible in the case of reducing their cost in terms of tightening standards on emissions (emission charges).

CFB boilers are best suited to burning coals of different quality. In accordance with the data [8], it was found that the coal with the high calorific value gives the temperature in furnace equal to 887 $^{0}$C, combustion of biomass leads to lower temperature in the furnace to 838 ° C. The flue gases from the combustion of biomass increased by 12 $^{0}$C with respect to the coals. The results of VTI calculations close to the data [8]. Using the multi-fuel CFB boilers in Europe is determined by two factors: the requirement to limit $CO_2$ emissions and diversify fuel supply using local fuels and wastes, including municipal solid wastes. In [9] was observed the issues of limiting of greenhouse gas emissions in a co-firing of coal and biomass in the case of the Netherlands. Co-firing technology is especially developed in the Nordic countries. In [10] indicated that the energy sector has undergone significant changes over the past 5 - 7 years under the influence of requirements to reduce emissions and the market price of coal has not changed much. In this regard, the benefits of CFB technology in terms of features of burning a wide range of fuels provide additional impulse for its implementation. In China, where there is a rapid development of coal units with CFB boilers have reached the capacity of 600 MW. [11]

Taking into consideration the current trends in the development of coal-fired power generation it can be concluded that the use of enriched coal increases the technical and environmental performance of power plants. These benefits are realized in large power units with higher steam parameters, a long distance delivery, tightening emissions standards (emission charges) at economically reasonable cost of enriched coals. For the incineration of enrichment wastes is advisable to use CFB boilers.

**The results of the development of a new generation of coal-fired thermal power station**

JSC "VTI" together with JSC "EMA", CJSC "Teploenergoservis", JSC "Institute Teploelekroproekt" and NIU MPEI under the agreement with the fund "Energy without frontiers" performed work on the topic "Development of coal-fired CHP units of the new generation capacity of 100-120 MW with high technical and economic parameters for future replacement of existing equipment or new construction". The main features of the new generation of thermal power station implemented the following technical solutions:

- The new steam turbine with increased efficiency and economically viable steam parameters. The unit design, which allows to use throttling steam distribution and ensure operation of the unit on sliding pressure at low loads. Using turbine pivoting arm sleeve allows increasing maneuverability and switching off the low-pressure turbine for the entire heating season and avoiding losses.
- Boilers with increased steam parameters to guarantee the efficiency of more than 92%. CFB boilers that provide advanced standards on emissions of nitrogen oxides and sulfur, and the diversification of fuel supply. Combined or bag filters provide high collection efficiency of ash, nitrogen and desulfurization systems, providing values for emissions of 200 mg/m3. Systems of dry ash and slag removal, allows the use of ash and slag without ash dumps.

Cases with CFB boilers and PC boilers equipped with means of nitrogen and desulfurization systems were considered. Figure 2 shows a sketch of a CFB boiler.

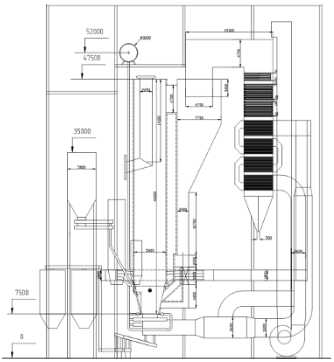



Fig. 2 – Sketch of a CFB boiler

Performed the calculations of CFB boilers for various steam parameters, the temperature of the exhaust gases are in a wide range of fuels. For heat recovery of flue gas heat exchanger and design of the heat utilization system was proposed. PC boiler has tower-type arrangement. It is made from single-hull turret arrangement of the heating surfaces and suspended on its own structures. Basic layout solutions in use cases of non-catalytic NOx reduction (SNCR) and catalytic reduction of NOx (SCR) was developed. As an example, Figure 3 shows the arrangement of the boiler equipped with a tubular air heater and the SCR system.

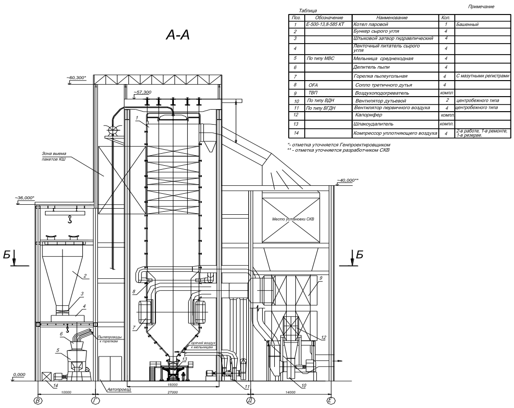



Fig. 3 – The arrangement of the PC boiler equipped with the SCR system

Multiple calculations of thermal coal-fired CHP schemes of the new generation based on developed technical solutions for main equipment were carried out. The calculation results are presented in [6]. A comparison of indicators of the new generation with existing CHP in Russia and CHP «Elho» in Poland are done in Tables 1 and 2.

Table 1 – Performance comparison of existing coal-fired with T-110 turbines and a new generation of CHP units

| № | Indicators | Existing CHP | | New CHP | Comparison |
|---|---|---|---|---|---|
| | | Calc. | In fact | | |
| 1 | Unit efficiency in condensing mode, % | 34,9 | 25 - 34 | 38,5 – 39,0 | on 4,5% (abs.) higher and above |
| 2 | Specific heat consumption for electricity generation, kJ/kW*h | 9537 | to 12000 | 8253 | More than 10 % lower |
| 3 | Specific fuel consumption for supplied electricity for heat consumption, g /kW*h | 197 | 200- 270 | 185,4 | More than 6 % lower |
| 4 | Boiler efficiency, % | 90 - 91 | 89,5 – 90,7 | 92,2 – 93,2 | on 2% (abs.) higher and above |
| 5 | Dust emissions, mg/nм$^3$ | | 150 and higher | < 50 | 3 times lower |
| 6 | NOx, mg/nм$^3$ | | 400 - 800 | 200 | 2-4 times lower |
| 7 | $SO_2$, mg/nм$^3$ | | 600 - 2100 | 200 | 3 -10 times lower |

Table 2 - Comparison of CHP «Elho» and new generation CHP

| № п.п. | Indicators | CHP «Elho» | | New CHP | Comparison |
|---|---|---|---|---|---|
| | | Calc. | In fact | | |

| | | | | | |
|---|---|---|---|---|---|
| 1 | Unit efficiency in condensing mode, % | No data | | 38,5 – 39,0 | higher |
| 2 | Specific heat consumption for electricity generation, kJ/kW*h | 10720 | 10178 | 8253 - 8600 | More than 10 % lower |
| 3 | Specific fuel consumption for supplied electricity for heat consumption, g /kW*h | No data | | 185,4 | equal |
| 4 | Boiler efficiency, % | 90, 17 | 92,03 - 94,17 | 92,2 – 93,2 | equal |
| 5 | Dust emissions, mg/nм$^3$ | 50 | 31 - 9 | < 50 | equal |
| 6 | NOx, mg/nм$^3$ | 246 | 124 | 200 | A bit higher |
| 7 | $SO_2$, mg/nм$^3$ | 383 | 356 - 329 | 200 | Lower |
| 8 | Unit capital cost, $/kW | 1570 (in the prices of 2010) | | 1888( in the prices of 2013) | Lower or equal |

Considered project of a new generation CHP is technically feasible. Its implementation will improve the efficiency and security of electricity and heat supply; will provide the basis for the replacement of worn-out and obsolete equipment of existing coal-fired thermal power station in the need to reduce the burden on the environment in the cities of their application. An urgent need to improve the efficiency of existing CHP and tightening standards on emissions require implementation of the proposed technical solutions even in large commercial risks and long payback period.

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# Technology of Fly Ash Recycling at Coal-Fired Power Plants

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## ABSTRACT

A technology for dry beneficiation of fly ash generated at coal-fired electric heat and power plants has been developed. Several samples of fly ash from different coal-fired heat and power plants were studied. The studies of ash samples material composition showed, that unburned carbon content in the samples varied from 5 to 20%. The main processing technique, used for ash beneficiation in the performed studies, combined magnetic and electrical separation of ash in vibrofluidized state, permitting to significantly increase the separation process throughput rate and selectivity. The studies of dry ash material composition and physical properties showed, that coal particles most commonly form aggregations with spheroidized magnetite-hematite, and are characterized by larger size in comparison with other particles of main mineral mass. In addition to that, fly ash is characterized by an extremely low bulk density and a unique conditional coefficient of internal friction, which is close to unity. On the basis of the above, a combination technology for processing heat and power plants fly ash has been developed, that includes magnetic and electrical separation in vibrofluidized state, providing for a stable production of marketable product with carbonaceous particles content within 3 %, that may be used as general-purpose binding material in construction industry.

**Keywords**: Fly ash, Recycling, Magnetic separation, Electrical separation, Vibrofluidization, dry beneficiation, construction industry.

## INTRODUCTION

The main worldwide trend in power generation with coal firing is flame combustion of finely ground mineral coal. However, changing over from lump to pulverized coal flame combustion results in decrease of coal-ash slag and increase of fly ash share [1, 2].

Main mineral phase of heat and power plants fly ash is a promising binding material for construction industry, yet inclusions of unburnt coal particles, for the most part, account for 3 to 20% of ash, constituting an obstacle for its efficient recycling. In order to provide for a stable use of fly ash as a general-purpose binding material in construction industry, unburnt coal particles content in it must not exceed 3 % by the Russian standards [3].

Application of wet technologies for separation of unburnt coal particles from heat and power plants fly ash leads to an obvious hydration effect on ash main inorganic (mineral) part, resulting in loss of binding properties, that may be recovered only through expensive roasting. Therefore, this work attempts at development of a new flow sheet for heat and power plants fly ash beneficiation through application of dry beneficiation methods. These methods are electrical and magnetic separation, based on essential differences in electric conduction and magnetic susceptibility of coal and associated spheroidized magnetite-hematite, in contrast to ash inorganic mineral components. A special feature of the proposed technology consists in intensification of electrical and magnetic beneficiation process based on the vibrational pseudofluidization effect (vibrofluidized) [4-6].

In state of rest, bulk material particles are in constant contact with one another, with their relative position not changing. Vibration actions of medium intensity may create forces in material layer, overcoming static friction between particles, and leading to continuous mutual shifting between these particles. Further increase in vibration intensity may lead to overcoming the Earth gravitational force, and

 
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bulk material particles will be “swarming” over vibrating surface. In other words, under vibration actions of various intensity, bulk materials may pass into state of quasi-aggregation as “granular liquid” and “granular gaseous” [7-9].
A “granular gaseous” state is of a prime practical importance regarding bulk materials separation processes, since it permits to break links between particles, prevent cohesion and act selectively on recovered particles [10].

Application of vibrofluidization permits to pass from separation in thin layer to separation in bulk volume, consequently, providing for process selectivity and throughput rate to be increased more than by the factor of two.

## EXPERIMENTAL

Ash recycling technology final products are a binding material for construction industry and a power-generating fraction, which is returned to re-combustion.

Four samples of fly ash from coal-fired heat and power plants, located in different regions of the Russian Federation, were studied.

By means of microscopic studies, ash particles were identified by morphological and physico-optical characteristics, and the following types of crystalline and amorphous ash particles were revealed: unburnt coal particles (UC), white spheroidized (WS), red spheroidized (RS), spheroidized magnetite-hematite (SMH), agglomerated fine particles of glassed silica (GS), agglomerated particles of complex composition (AC).

The results of the material composition studies for all samples of fly ash are presented in Table 1.

Table 1 – The ashes composition quantitative characteristics.

| Particles type | Content, % | | | |
|---|---|---|---|---|
| | Ash 1 | Ash 2 | Ash 3 | Ash 4 |
| UC | 15-20 | 5-8 | 10 | 10-15 |
| WS | 40-45 | 80 | 60 | 10 |
| RS | 15 | 8 | 5-7 | 10-15 |
| SMH | 5-8 | 2-5 | 4-6 | 8-12 |
| GS | 10 | 5 | 8-10 | 15 |
| AC | 15 | 2 | 10 | 40 |
| Coarse vitrified products | 0 | 0 | 5 | 10 |

The study of fly ash samples revealed three essential mineralogical and processing specialties:

- coal particles most commonly form aggregations with spheroidized magnetite-hematite;
- coal particles are characterized by larger size in comparison with other particles of main mineral mass;
- main silicate mass is represented by vitrified and non-vitrified aluminosilicates.

Taking into consideration these specialties, solution of a problem of coal particles recovery from dry fly ash may be accomplished by means of a combination technology that includes dry size classification with subsequent magnetic and electrical separation in vibrofluidized state. The diagrams of the magnetic and electric separators used is given by Fig.1 and Fig.2 [4, 5].

Based on the above, a combination technology for processing dry fly ash from combined heat and power plants has been developed, that includes the following main processing stages: screening with removal of slag and coarse accidental inclusions; dry size classification of undersize in pseudofluidized state in order to separate dust fraction that does not contain unburnt coal particles; magnetic separation of +40 µm size fraction in vibrofluidized state with recovery of coal particles' aggregations with spheroidized magnetite-hematite and other magnetic inclusions; and electrical separation in vibrofluidized state with recovery of remaining coal particles into conducting product. Nonconducting product is joined with -40 µm dust fraction, forming a marketable product with carbonaceous particles content within 3 %.

The process flow diagram for dry processing of heat and power plants fly ash in vibrofluidized state is shown in Fig. 3.

By way of illustration, Fig. 4 shows the micrographs of ash separation products, their size varying from 0.005 mm to 0.1 mm. With respect to qualitative composition, quartz globules and aggregations predominate in overall mass, accounting for approximately 55 %. Mixed quartz-mica ("red") and silicate-carbonaceous globules are represented in smaller quantities, but approximately equal proportions (15-20 %), besides, aggregations of silicate-carbonaceous composition are also found (approximately 20 %), with average size varying from 0.01 mm to 0.1 mm.

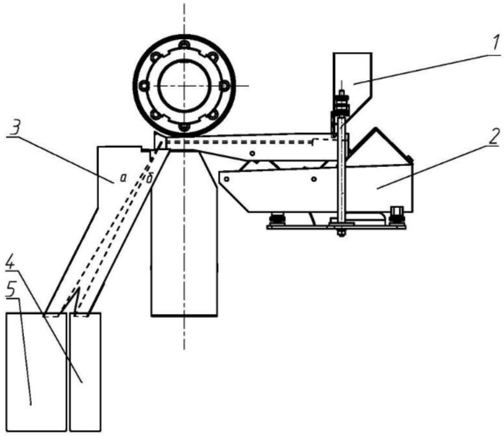



Figure 1- The unit for electromagnetic separation in the vibration fluidized state:
1 – bunker, 2- vibrating feeder, 3- duct, 4.5 – receivers.

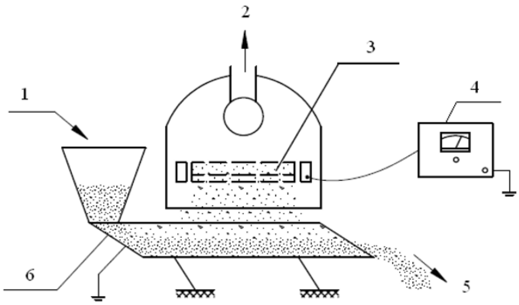



Figure 2 - The unit for electrical separation in the vibrofluidized state:
1 – feeding; 2 – conducting product; 3 – high voltage mesh electrode; 4 – high voltage power source; 5 – non-conducting product; 6 – vibratory feeder.

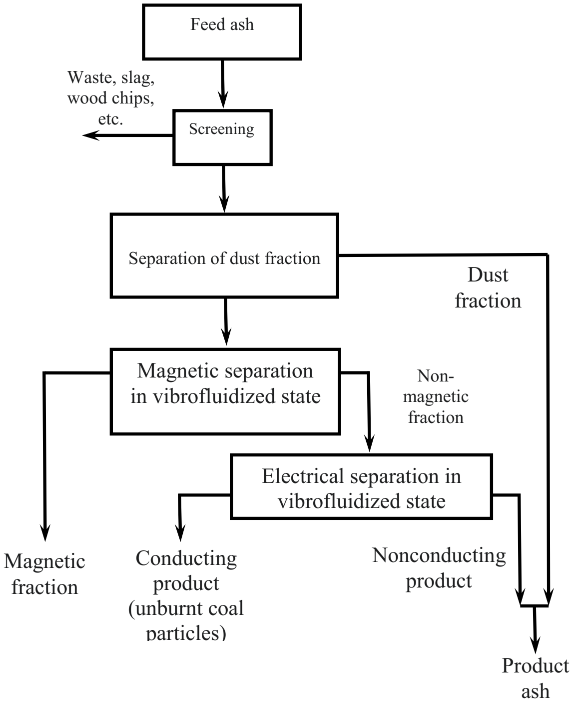
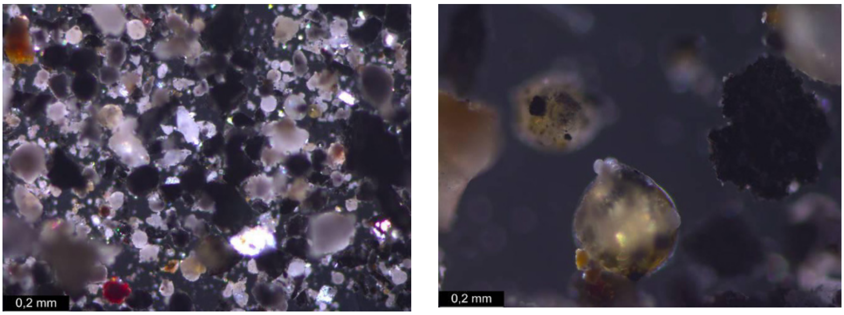



Figure 3 − The process flow diagram for dry processing of fly ash in vibrofluidized state.




a b

Figure 4 – Ash beneficiation products. Silicate fraction (a). Silicate-carbonaceous aggregations and silicate-carbonaceous globules (b).

All tested ash samples were treated by the process flow diagram shown in Fig. 3, obtaining final product as mineral fraction with carbon content of 0.5-2.5 %. With that, 3-8% of feed ash mineral constituent passes into carbon product.

## CONCLUSIONS

The studies of dry ash material composition and physical properties showed, that coal particles most commonly form aggregations with spheroidized magnetite-hematite, and are characterized by larger size in comparison with other particles of main mineral mass. In addition to that, fly ash is characterized by an extremely low bulk density and a unique conditional coefficient of internal friction, which is close to unity. On the basis of the above, a combination technology for processing heat and power plants fly ash has been developed, that includes magnetic and electrical separation in vibrofluidized state, providing for a stable production of marketable product with carbonaceous particles content within 3 %, that may be used as general-purpose binding material in construction industry.

## ACKNOWLEDGEMENTS

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# The Development Of A Technological Complex For Utilization Of Fine Waste Coal PP "Tugnuyskaya"

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## Annotation

The technology of coal preparation plants in modern Russia is characterized by using of closed water-sludge circuit to produce coal concentrate with the required parameters for moisture content without thermal drying of fine coal classes. However, as part of shipped rock to the dump fine waste of coal preparation class 0-0.5 mm appeared, usually represented by sludge filter presses - filter-cake.

The outcome of filter-cake is up to 7% by the weight of the processed coal [1]. The shipping of this product with the rock to the dump caused by the fact that its use is complicated by the high ash content of up to 50% and the moisture content to 45%. A similar situation exists in the LTD "Tugnuy PP" of JSC "SUEK". The annual yield of the cake here is 800 thousand tones.

For a solution of this problem, the reducing of the release of enrichment waste and increasing of the marketable products on Tugnuy coal preparation plant it was suggested to use the technology of production and combustion of coal-water slurry fuel (CWF) on semi-industrial installation. As a result 2 variants of filter-cake usage were suggested:

- transfer on the CWF combustion boilers the existing boiler factory;
- construction of mini CHP plants running on CWF, to provide electric and thermal energy needs of the Tugnuysky mine.

**Key words**

Utilization of coal prepared waste, filter-cake, preparing, combustion of suspension coal-water fuel, structural and rheological characteristics, mini CHP.

### Introduction

For the experimental studies a representative sample of the filter-cake PP "Tugnuysky" weighing 5 tons was selected. The research of the possibility of preparing CWF with determination of structural-rheological characteristics of the slurry fuels was conducted on a shaker by well-known methods [1,2].

Table 1 presents the characteristics of the original filter-cake and suspension coal-water fuel, obtained on its basis.

As a result of the performed work it was shown the optimal composition and consumption of the complex plasticizer – reagent-plasticizer and developed process procedure for preparing a coal-water fuel and the technological scheme of its preparation (Figure 1) [3-10].



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Table 1. Characteristics of the original filter-cake and the received fuel

| Name index | The numeric value for samples | |
|---|---|---|
| | filter-cake | CWF |
| Granulometric composition: мм | | |
| +1,0 | 4,0 | - |
| 0,35-1,0 | 28,9 | 6,2 |
| 0,25-0,35 | 9,7 | 9,2 |
| 0,07-0,25 | 24,5 | 32,7 |
| 0,00-0,07 | 32,9 | 51,9 |
| The solid phase mass fraction | 65,0 | 60,5 |
| Ash content | 34,5 | 34,5 |
| Effective viscosity at a shear rate 81 $c^{-1}$ и t=20°C | - | 270 |
| The lowest combustion heat | 2705 | 2475 |

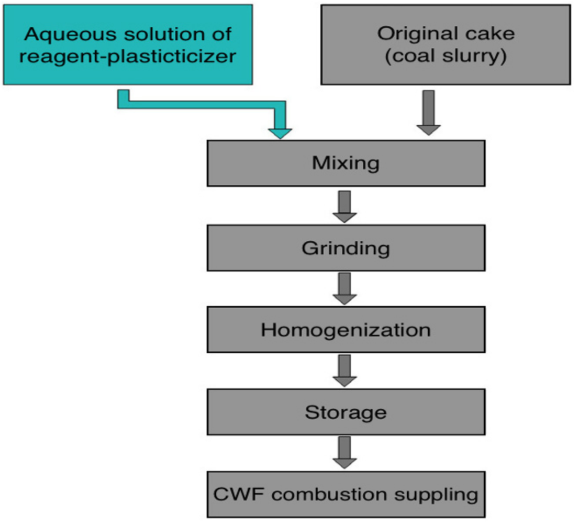


Figure 1 – the Technological scheme of CWF preparation

As t he results of rheological characteristics measurements on a rotational viscometer show, the received fuel is a non-Newtonian fluid, the flow curve of which can be described as an exponential equation:

$$\tau = k \cdot \dot{\gamma}^n , \qquad (1)$$

where $\tau$ - shear stress, Pa;

$\dot{\gamma}$ - shear rate, $s^{-1}$;

$k$ - suspension consistency coefficient, Pa·$s^n$;

$n$ - the stream index;

also as the equation of Bingham fluid:

$$\tau = \tau_0 + \mu_0 \cdot \dot{\gamma} , \qquad (2)$$

where $\tau_0$ - initial voltage, Pa ;

$\mu_0$ - structural viscosity, Па·с.

Figure 2 shows the graphs of the rheological flow curves of the received fuel (for certain values of the coefficients: $k = 0{,}59$; $n = 0{,}79$; $\tau_0 = 7{,}56$; $\mu_0 = 0{,}15$).

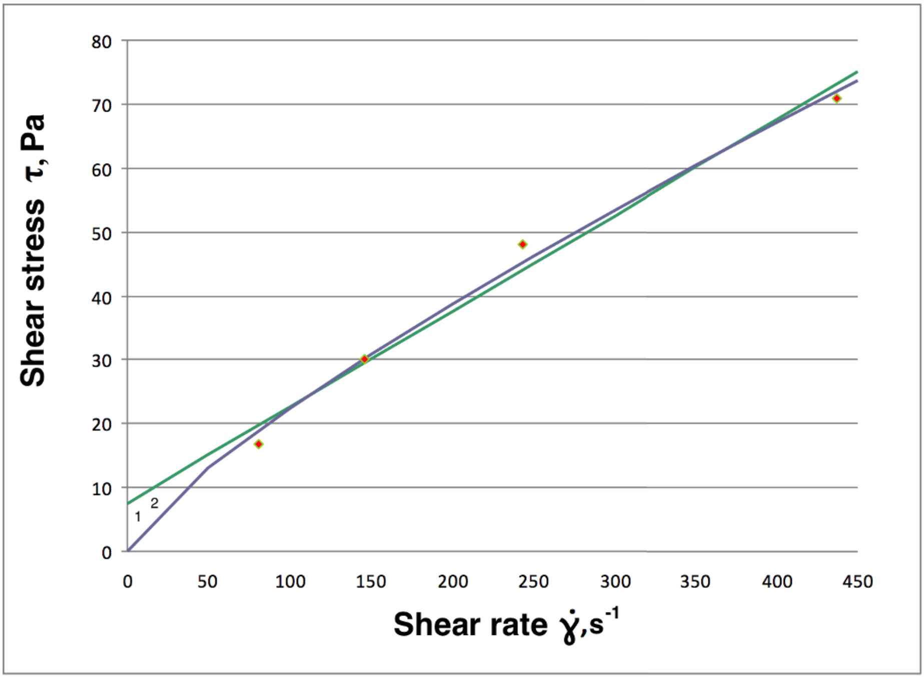


Figure 2 Rheological curves of water-coal fuel.

- experimental values;

1 - the graph of power-liquid fluid (equation (1);

2 – the graph of the Bingham fluid (equation (2)

The assessment of the accuracy description degree by equations (1) and (2) of the experimental values obtained on the viscometer, showed that both models accurately simulate the current CWF (accordingly: 1 - σ1=0,93; 1 - σ2=0,90), where σ is the degree of experimental points deviation from the corresponding analytical values.

The obtained data allow calculating the pipeline system for CWF hydrotransportation.

On the basis of the obtained results a pilot batch of suspension coal-water fuel was prepared on the semi-industrial plant in the Cherepanovo city, Novosibirsk region (Figure 3).

Figure 3 – General view of semi-industrial installation for CWF preparation and burning

Technology of CWF preparation was as follows. Initial filter-cake was loaded into a mixer and with simultaneously dispensing of reagent-plasticizer solution. After stirring the received mixture went through a strainer into the sump, from which it was pumped in a vibrating mill. The product from the grinding mill fed into the container from which the special pump activator pumped it into storage tanks.

The received fuel combustion was carried out in the boiler plant, equipped with a boiler with an adiabatic vortex combustion chamber. It should be mentioned that such previously performed similar projects using the technology of low-temperature vortex combustion of fuels, deballasting moisture and mineral components, in adiabatic or similar conditions showed high efficiency to burn organic mass with a minimum outlet of harmful substances in exhaust gases. The pump through the nozzle carried out dosed CWF to the combustion chamber. To prevent clogging of the injector nozzle on the CWF supply line self-cleaning fine filter is installed. The CWF spraying operation was carried out with compressed air. The results of an experimental CWF batch combustion are presented in Table 2.

Table 2. The results of CWF burning

| Name index | Unit of measurement | The numeric value for samples |
|---|---|---|
| WCF consumption | l/ hour (kg / hour) | 200 (240) |
| Fuel pressure | MPa | 0,24 |
| The temperature in the furnace | $^{0}$C | 1072 |
| The flue gas temperature | $^{0}$C | 195 |
| The thermal efficiency of the boiler | MW (Gcal / h) | 0,56 (0,48) |
| Boiler efficiency | % | 80,8 |

The test results were prepared by the following choices of investment projects:
- transition of boilers in the existing boiler house of the mine "Tugnuysky", on the CWF combustion obtained on the basis of the filter-cake;
- construction of mini CHP with 20MW electrical and 69.6 MW (Gcal/h) thermal capacity.

The main technical and economic indicators of investment projects are presented in Table 3.

Table 3. The main technical and economic indicators of the investment project

| Name index | Unit of measurement | Numeric value | |
|---|---|---|---|
| | | Variant 1 | Variant 2 |
| Annual consumption of filter-cake | tons | 94000 | 442500 |
| Cost value of CWF preparation | rub./ ton | 250 | 250 |
| Fuel costs in the production of 1 MW·h (1 Gkal): | rub | | |
| Coal | | 158,6 (184) | 158,6(184) |
| CWS | | 107,7 (125,0) | 102,6(119) |
| Payback period | years | 2 | 4,6 |

**Conclusion**

The conducted experimental studies and semi industrial testing of the technology for utilization of fine coal waste PP "Tugnuysky" through the preparation of a suspension of coal-water fuel and its combustion in the boiler with a vortex furnace have shown high efficiency of the proposed method. The technology of WCF preparation can be described as simple and low operating costs. Implemented vortex system of CWF combustion ensures energy efficiency of the boiler at least 80%.

There are two variants of investment projects for the implementation of the developed technologies for the utilization of coal waste in terms PP "Tugnuysky".

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# The Usage Of Boilers With A Vortex Furnace For Burning Enrichment Products And Deballasting Coal

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**Annotation**

The article shows the possibility of efficient combustion of the products of enrichment and deballasting coal using low-temperature vortex technology of burning in boilers equipped with combustion chamber "Tornado", which is implemented with joint flare-vortex grate and the burning fuel. This technology is applicable to virtually any kind of solid fuels and wastes. Due to the active aerodynamic structure of gas and fuel flows established in the furnace space, the vortex combustion technology can effectively reduce the sulfur oxides in the flue gas while burning high sulfur fuel. In this case, into the combustion space of the boiler is carried out simultaneous supply of fuel and sulfur-absorbing reagent (SAR). The article presents the research results of the possibility to suppress the release of sulfur oxides due to the suspension coal-water fuel (CWF) preparation with the simultaneous entry in the CWF along with the reagent-plasticizer of sulfer-absorbing reagent.

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**Key words**

Boilers with vortex furnaces, furnace "Tornado", products of coal enrichment, filter cake, deballasting fuel, coal-water slurry fuel, sulfur-absorbing reagent.

## Introduction

At present, almost all Russian processing plants have their own boiler plants for heating and hot water supply. Boilers installed in the boiler, as a rule, are equipped with furnaces with layer combustion of coal. The efficiency of operation of such boilers is from 50 to 70% due to high mechanical and chemical incomplete combustion especially during coking coal combustion. In addition boilers with grate furnaces are not designed for burning of such enrichment products as middlings, coking coal and fine coal slurries, such as filter-cake.

Currently, in Russia the production of boilers with different capacities using the effective vortex combustion system as a furnace "Tornado" is arranged [1].

A design feature of the furnace "Tornado" is the presence of a baffle with a window and exhaust systems of tangential flow acute after-burning blow. In the lower side of the chamber there is installed grate, usually made in the form of the furnace with strap. As a result in the furnace "Tornado" joint flare-vortex grate and combustion is implemented, which is applicable to virtually any kind of solid fuels and wastes. The biggest classes of fuel are burned in a layer on mechanized grate, and smaller ones – with flare-vortex



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method due to the fact that aerodynamically held in the furnace and fill the vortex radiant flux of burning particles.

Accordingly, in the furnace “Tornado” there is a smoothing of the irregularity of heat with the suppression of maximum temperature of the flame, there is a significant fraction of the convective component of heat transfer, increases the degree of blackness and the emissivity of furnace volume. Heat perception of furnace walls is increased markedly, but the maximum of heat perception decreases; the heat load of screens is increased, but distributed uniformly, which increases the reliability of their work. Thanks to mechanization and the low weight of the fuel in the layer, furnaces do not require much manual labor, and easily automated.

Thus, boilers equipped with furnaces "Tornado" are different with:

- The uniformity and the reduced temperature in the furnace volume;
- Increased convective component relative to the emitting;
- The decline in the mechanical and chemical incomplete burning due to the retention of small particles of fuel in the vortex flow and the constant burning of large particles on a lattice with stoke strap;
- The reliability of the furnace tubes;
- The reduction of emissions of harmful substances into the atmosphere;
- The possibility of using various waste fuels and deballasting fuels as a fuel.

The (PP) "Matyushinskaya", mines "Anzherskaya-Yuzhnaya", "Bolshevik", "Krasnogorsk” are equipped with such boilers in Kemerovo region.

In addition, vortex furnaces, thanks to increased forces of combustion processes, are more compact. As a rule, they can be adjusted in residual amounts of existing boilers through minor renovation. Today there is a lot of experience installing furnaces "Tornado" not only in solid-fuel boilers, but also in over-heat furnaces gas-oil-fired steam boilers, with their transition into coal, CWF or combustible waste.

In the combination with the use of gas-tight screens using furnaces "Tornado" allows to create more compact boilers with lower metal content and increased capacity in transportable units with factory readiness. This significantly reduces the volumes of construction and installation works and the timing of starting the boiler.

These advantages were confirmed by the results of experimental-industrial and industrial tests of the combustion of various coal conversion products of coal preparation plants and various coal companies. In the process of experimental research the possibility of the using the vortex combustion system to reduce sulfur oxides in the flue gas when fed to the furnace high-sulfur coals with sulfur-absorbing reagent was shown. In this case the normative values of sulfur oxides in the flue gases were provided. The SAR input into the furnace was carried out as a part of a pre-prepared suspension of water-coal fuel. At the preparation stage of CWF on the basis of the high-sulfur Bulgarian coals simultaneously with the reagent-plasticizer sulfur-absorbing reagent was injected, the quantity of which was determined from the molar ratio of sulfur in raw coal and the alkaline-earth metals in sulfur-absorbing reagent. Thus due to the formation of solid sulfate compounds provided the content of sulfur oxides in the flue gases below the normative values.

**Original raw materials and equipment**

For evaluating the possibility of using boilers with vortex furnace "Tornado" for the combustion products of coal, especially fine coal sludge (filter cake), coking coal middlings, screening coal, brown coal of B2-and high-sulfur coal were conducted experimental investigations and industrial tests in the adiabatic vortex combustors, in the reconstructed boiler DKVR-6,5/13, the serial and boiler KV-1,2-105 ShVF.

As starting materials were used: the middling brand of the SS (PP "Mezhdurechenskaya"), elimination of grade-D coal (mine "Belovskya"), brown coal of B2 (coal cut "the Chulym Coal"), the fine coal slurry PP "Shestaki", PP CJSC "Sibantratsit", PP "the Barzasskoe Association" and high-sulfur Bulgarian coal (2 samples). In tables 1 and 2 presents the characteristics of raw materials used as fuel.

Thin coal slimes and Bulgarian coal is processed in coal-water slurry fuel technology SPE "Sibecotechnika" [2-10].

**Table 1** – characteristics of the original fuel (cabalistical coal)

| The name of indicator | The value of indicators | | |
|---|---|---|---|
| Boiler type | DKVR-6,5-13 Novoaltaisk, Altay region | | KV-1,2-105 ШпВТ Barnaul city |
| Fuel type | Crushed middlings mark SS | Elimination of grade-D coal | Brown coal |
| The particle size of the fuel, mm | 0-50 | 0-13 | 0-40 |
| Moisture, % | 3,0 | 12,1 | 43,3 |
| Ash, dry basis, % | 19,0 | 19,6 | 11,3 |
| Sulfur content (on dry basis), % | 0,3 | 0,39 | 1,0 |
| Low combustion temperature, MJ/kg (Kcal/kg) | 24,3 (5800) | 23,03 (5500) | 11,81 (2820) |

**Table 2** - characteristics of the original fuel (CWF)

| The name of indicator | The value of indicators | | | | |
|---|---|---|---|---|---|
| Boiler type (furnance) | KV-0,8 with adiabatic vortex combustion chamber, Cherepanovo City, Novosibirsk region | | | Experimental adiabatic vortex furnace | |
| | CWF on the basis: | | | | |
| | Filter-cake | | | Bulgarian coal | |
| | PP «Shestaki», Kuzbasss | «the Barzasskoe Association», Kuzbass | PP CJSC «Sibantratsit», Novosibirsk region | sample №1 | sample №2 |
| The particle size of the fuel, mm | 0-0,250 | 0-0,500 | 0-0,250 | 0-0,250 | 0-0,250 |
| Moisture, % | 39,0 | 38,0 | 48,0 | 34,5 | 55,4 |
| Ash, dry basis, % | 19,0 | 24,0 | 29,0 | 41,4 | 31,0 |
| Sulfur content (on dry basis), % | 0,30 | 0,30 | 0,25 | 2,41 | 3,8 |
| Low combustion temperature, MJ/kg (Kcal/kg) | 14,13 (3375) | 13,06 (3120) | 11,30 (2700) | 10,47 (2500) | 7,33 (1750) |

**The results of experiments and tests and its discussion**

Table 3 presents the results of the work used boilers and adiabatic vortex furnaces for combustion products of coal enrichment and deballasting fuels.

**Table 3** – Tests results of boilers with vortex furnace "Tornado" and adiabatic furnaces.

| The name of indicator | The value of indicators | | | |
|---|---|---|---|---|
| Boiler type | DKVR-6,5-13 Novoaltaisk, Altay region | КВ-1,2-105 ШпВТ | KV -0,8 with adiabatic vortex combustion chamber Cherepanovo City, Novosibirsk region | Experimental adiabatic vortex furnace |

| | | | Barnaul | CWF on the basis: | | | | |
|---|---|---|---|---|---|---|---|---|
| | | | | Filter-cake | | | Bulgarian coal | |
| Fuel type | Crushed middlings mark SS | Elimination of grade-D coal | Brown coal | PP «Shestaki», Kuzbass | «the Barzasskoe Association», Kuzbass | PP CJSC «Sibantratsit», Novosibirsk region | Sample №1 | Sample №2 |
| Heating capacity, MW(Gkal/h) | 14,65 (3,5) | | 1,2 (1,0) | 0,8 (0,69) | | | 0,35 (0,3) | |
| The Temperature in the furnance, $^0$C | 1100 | 1070 | 1050 | 1100 | 1072 | 1160 | 1020 | 1000 |
| The temperature of the exhaust gases, $^0$C | 150 | 150 | 64 | 195 | 175 | 205 | 420 | 370 |
| The composition of flue gases, mg/m3: | | | | | | | | |
| CO | 250 | 310 | 185 | - | - | - | - | - |
| NO | 210 | 97 | 110 | - | - | - | 110 | 70 |
| $SO_2$ | | 15 | 320 | No data | No data | No data | 800* | 1100* |
| Boiler efficiency | 0,82 | | 0,83 | 0,85 | 0,85 | 0,85 | 0,80 | 0,80 |

* - data by burning CWF with SAR

Figures 1, 2 show the trend in the content of sulfur oxides in the exhaust gases while burning high-sulfur Bulgarian coals

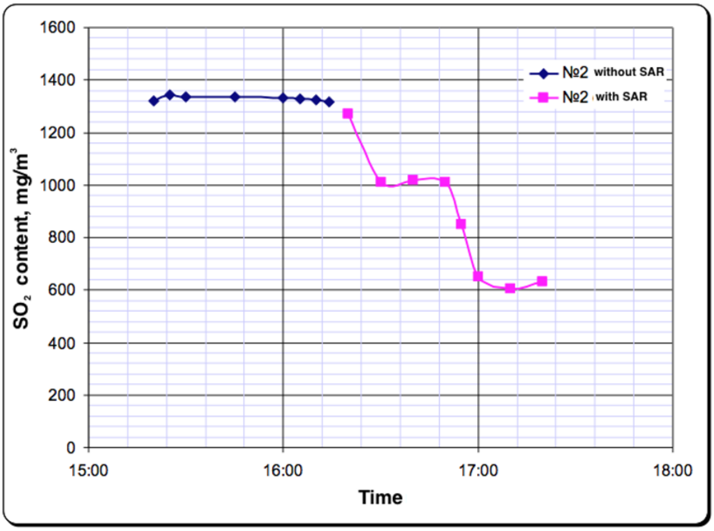


Figure 1 - the Effect of SAR on $SO_2$ reduction in the flue gases at combustion of CWF (sample №1).

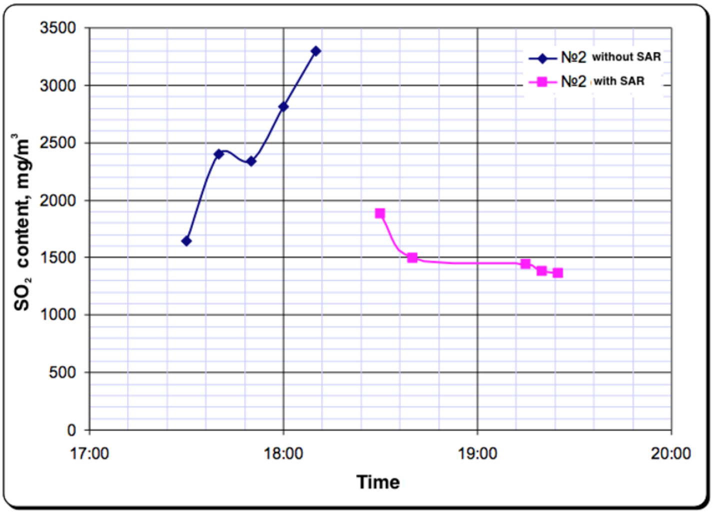


Figure 2 – the Effect of SAR on $SO_2$ reduction in the flue gases at combustion of CWF (sample №2).

As the results of industrial tests show, the use of boilers with vortex combustion system provides high efficiency using low-processed products of coal and lignite deballasting. Thus the efficiency of operation of the boilers exceeds 80% in harmful emissions, significantly lower acceptable values.

The results of the SAR application showed a high efficiency of the method of reducing sulfur oxides by preparation and combustion in swirl adiabatic combustion chamber CWF made from high-sulfur coals.

**Conclusion**

The use of the vortex combustion system of the furnace with the use of "Tornado" in the series, and reconstructed boilers provide an effectively burning of different enrichment products and deballasting coals. In this case, fine coal slurries are useful utilized in the suspension hydrocarbon fuel.

The combustion of suspension coal fuel based on high-sulfur coals, together with SAR allows reducing the content of sulfur oxides by more than 2 times until target level of emissions.

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# Complex processing of ash and slag wastes from coal-fired CHP plants

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**Abstract**

The paper focuses on both Russian and international practices concerning the solution of ash and slag wastes processing and prospective areas of their application. The results of lab and pilot plant tests on concentration ability of ash and slag wastes which were obtained during coal combustion at coal-fired CHP plants of Irkutskenergo JSC (Irkutsk, Russia) are presented. Ash and slag wastes, themselves a product generated by high-temperature thermal processing of the mineral matter of the coal during combustion in boilers, are a gray grainy mass. Dozens of ash and slag wastes millions of tons which generate annually during coal combustion at coal-fired power plants of the Irkutsk region can be fully processed with the generation of saleable products: iron concentrate, aluminum silicate product as well as products containing rare-earth, sparse and precious elements. In this case, ash and slag wastes processing can be disposable. Pilot plant tests confirmed lab tests results in terms of the quality of the resulting iron-bearing concentrate and aluminum silicate product. The mass yield of iron-bearing concentrate and aluminum silicate product in terms of ash and slag volume was 3-9% and 40-60%, respectively. Lab and pilot plant tests conducted demonstrated viability of the ash and slag wastes efficient processing. This resulted in their complete recycling and reduction in the environmental impact combined with profit generation from selling saleable products. Economic calculations showed that the payback period for the construction of an ash and slag wastes concentration plant with the throughput of 1.0 MTPA is 2 years. The production line is equipped with environmentally friendly equipment produced in Russia.

**Key words**

Ash and slag wastes, processing technology, spiral separation, shaking tabling, iron-bearing product, aluminum silicate product, pilot plant for processing

 
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In Russia, there are over 170 current coal-fired CHP plants. Ash and slag wastes are produced by coal combustion and are disposed of in ash dumps. Annual generation of ash and slag wastes accounts for approximately 30 mln*tons. The dumps have accumulated over 1.5 billion tons, and they have an area of 30,000 ha. A number of ash dumps are overfilled. Additionally, the construction costs for a new ash dump will be, up to RUB 4 billion. The operating costs for ash and slag wastes annually increase and represent around 7% of power and heat production cost at a coal-fired plant. Therefore, a solution to the problem ash and slag wastes processing is of high importance in Russian Federation [1,2].

Currently, the level of ash and slag wastes processing and utilization is still insufficient. Slightly over 11% (3.5-4.0 MTPA) of ash and slag wastes are used. At the same time, ash and slag wastes are fully utilized in other countries. For example, both Germany and Denmark use up to 100% of ash wastes annual yield to produce construction materials.

More than 300 processing and utilization methods of ash and slag wastes are known, with the major use occurring in building industry and for construction materials production. However, the recovery of valuables as saleable concentrates has not yet been extensively studied [4,5].

It was found that they contain numerous valuables, such as ferrous, base, rare and precious metals. This is shown in Figure 1. Burning coal is a natural adsorbent. This results in the increase of valuables, up to 5-6 times, in the ash during its combustion. The use of the wastes as secondary raw materials is a major issue and, first of all, requires process design. Technologies which can quickly and efficiently produce a competitive and saleable product from ash and slag wastes are needed. Metals in the coal can be found as components of various minerals and metallorganic compounds. Their major portion converts to fly ash during coal combustion. In order to recover aluminum oxides from ash materials as a source for manufacture of metallic aluminum, the technologies were developed.

Ferrosilicon is an alloy of iron and silicon the formation of which is observed due to the lack of air and the presence of unburned coal particles in a molten slag. Germanium and uranium may be recovered from the fly ash. Recovery of gallium, lead, zinc, molybdenum, selenium, gold, silver, rhenium and rare-earth elements is considered to be prospective.

Aluminum silicate hollow spheres are arguably the most original and valuable components of fly ash. They are hollow, almost perfectly shaped silicate beads with a smooth surface. They have diameters ranging from >10 up to hundreds μm. Their average diameter is approximately 100 μm. The wall thickness is 2-10 μm [3]. Complex processing of ash-slag of Ekibastuz coal was studied by Kazakh National Research University (Almaty) in order to obtain aluminum oxide and iron concentrate [6].

Therefore, ash dumps are man-manned deposits of solid mineral resources and should be processed from a long-term developmental perspective.

The development of valuable components recovery technologies from ash and slag wastes to saleable products has been studied for a long time. Various methods are used, particularly: flotation and chemical methods, such as the recovery of gallium from coal fly ash using acid and chemical extraction solvents. The use of these methods to process ash and slag wastes poses a problem due to the fact that ash dumps are situated near and even within large human settlements. Also, toxic chemical reagents are used for flotation and chemical processes. In this case, environmental issues are of paramount importance and environmentally friendly ash and slag wastes concentration processes are required.

In order to solve the above-mentioned problem, manufacturing company Spirit LLC in collaboration with Institute of the Earth's Crust of SB RAS has been developing ash and slag wastes concentration technologies for 10 years. Test work using samples from CHP plant-9, Novoirkutskaya CHP plant of Irkutskenergo JSC and Gusinoozerskaya Region Power Station was carried out. Gravity concentration methods – jigging, spiral separation, centrifugal concentration, tabling and wet magnetic separation at different intensity of magnetic field – were used for the test work. The best results were obtained using magnetic separation, spiral separation using slimes devices and shaking tables.

Table 1 –The results of semiquantitative spectral analysis of ash and slag wastes from CHP plant-9

| Element | Symbol | Percentage abundance*, g/t | Grade in the sample, g/t | Coefficient ratio** |
|---|---|---|---|---|
| Barium | Ba | 670 | 600 | - - |
| Beryllium | Be | 6 | 6 | - - |
| Zirconium | Zr | 240 | 300 | - - |
| Manganese | Mn | 530 | 600 | - - |
| Titanium | Ti | 3100 | 3000 | - - |
| Vanadium | V | 53 | 300 | 5.7- 5.7 |
| Chromium | Cr | 35 | 100 | 2.8- 5.7** |
| Nickel | Ni | 19 | 100 | 5 -10 |
| Cobalt | Co | 12 | 50 | 4 -6.7 |
| Lead | Pb | 17 | 30 | 1.8 – 1.8 |
| Copper | Cu | 14.3 | 300 | 20.9 – 20.9 |
| Zinc | Zn | 52 | 200 | 3.8 – 5.8 |
| Tin | Sn | 3 | 5 | 1.6 – 1.6 |
| Molybdenum | Mo | 1.4 | 6 | 4.3 – 4.3 |
| Lithium | Li | 22 | 30 | - - |
| Rare-earth elements | | | | |
| Yttrium | Y | 20.7 | 60 | 2.9 – 2.9 |
| Ytterbium | Yb | 1.5 | 6 | 4 - 4 |
| Lanthanum | La | 32.3 | 40 | - - |
| Scandium | Sc | 7 | 40 | 5.5 -4.2 |
| Cerium | Ce | 65.7 | 300 | 4.6-<4.6 |
| | | | | |
| Niobium | Nb | 26 | 10 | - - |
| Strontium | Sr | 316 | 300 | - - |
| Gallium | Ga | 14 | 20 | 1.4 – 1.4 |
| Silver | Ag | 0.055 | 0.15 | 2.7 – 1.8 |
| Arsenic | As | 2.0 | <100 | |
| Antimony | Sb | 0.31 | <20 | |
| Bismuth | Bi | 0.123 | <2 | |
| Tungsten | W | 1.4 | 10 | 7.6 - |
| Tantalum | Ta | 1.5 | <60 | |
| Germanium | Ge | 1.4 | 1 | - 2 |
| Thallium | Tl | 1.5 | <5 | |
| Phosphorus | P | 665 | 300 | - - |
| Boron | B | 17 | 100 | 5.9- 11.8 |
| Iron | Fe | 30890 | 40000 | 1.3 - 3.2 |
| Magnesium | Mg | 13510 | 1000 | - - |
| Calcium | Ca | 29450 | 30000 | - - |
| Sodium | Na | 25670 | 6000 | - - |
| Potassium | K | 28650 | <3000 | - - |
| Silicium (chemical analyses) | Si | 294500 | 264000 | - - |
| Aluminum (chemical analysis) | Al | 77440 | 144000 | 1.8 – 1.6 |

| | | | | |
|---|---|---|---|---|
| Mercury | Hg | 0.083 | <1000 | |
| Gold (fire assay-AA) | Au | 0.005 | 0.013 | 2.6 – 2. |
| Cadmium | Cd | 0.102 | <30 | |
| Tellurium | Te | 0.001 | <30 | |
| Uranium | U | 2.5 | <300 | |
| Thorium | Th | 10.3 | <30 | |

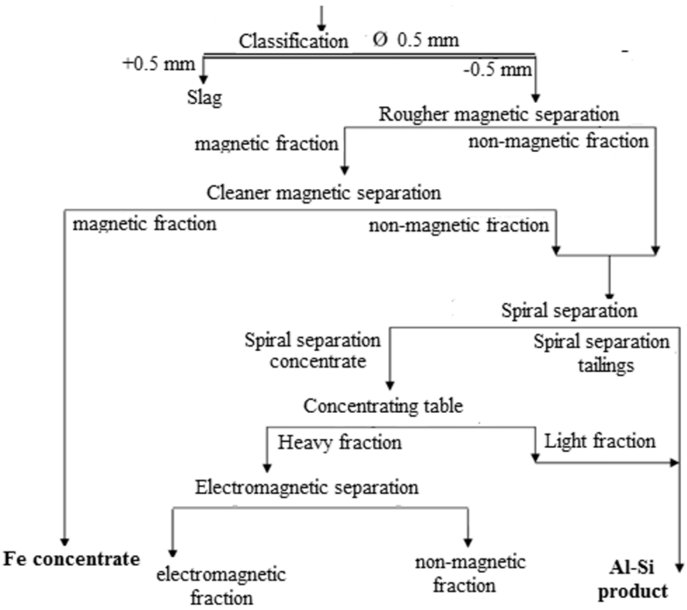


Figure 1 – Ash and slag wastes test work flow sheet

Table 2 –The results of ash and slag wastes concentration test work

| Products | Yield, % | Fe total | | $Al_2O_3$ | | $SiO_2$ | |
|---|---|---|---|---|---|---|---|
| | | Mass per-cent, % | Recovery, % | Mass per-cent, % | Recovery, % | Mass per-cent, % | Recovery, % |
| Fe concentrate | 3.58 | 60.83 | 37.94 | 5.3 | 0.73 | 12.11 | 0.77 |
| Spiral concentration tail-ings | 73.99 | 3.85 | 49.63 | 28.07 | 79.83 | 59.33 | 77.86 |
| Shaking table tailings-Al-Si product | 9.86 | 2.95 | 5.07 | 26.02 | 9.86 | 60.55 | 10.59 |
| ∑ Al-Si product | 83.85 | 3.74 | 54.69 | 27.83 | 89.69 | 59.47 | 88.45 |
| Electromagnetic fraction | 0.56 | 9.58 | 0.93 | 9.36 | 0.20 | 17.8 | 0.18 |
| Non-magnetic fraction | 0.94 | 2.65 | 0.43 | 18.3 | 0.66 | 55.28 | 0.92 |
| Slag (+0.5mm) | 11.07 | 3.11 | 6.00 | 20.5 | 8.72 | 49.3 | 9.68 |
| Fly ash initial sample CHP plant-9 | 100.00 | 5.74 | 100.00 | 26.01 | 100.00 | 55.72 | 100.00 |

Iron-bearing concentrates with 52-60% Fe content were generated during the test work. The content of detrimental impurities (sulfur, zinc, phosphorus and arsenic) was within the permitted limits in order to use the concentrate as a source for steel smelting. After heavy minerals were removed using magnetic and spiral separation, so called aluminum silicate product remained. X-ray diffraction analyses showed that aluminum oxide can be found in the form of sillimanite ($Al_2SiO_5$). It was found that aluminum silicate product which was obtained during ash and slag wastes concentration can be used as raw materials to obtain aluminum oxide. Nepheline concentrate which is used to produce aluminum oxide has a similar composition ($Al_2O_3$ – 29.4 %, $SiO_2$- 42.80).

Based on the test work results, pilot plant tests on processing current ash and slag wastes from CHP plant-9 of Irkutskenergo JSC are being carried out by company Spirit. A plant with throughput of 2t/h was installed at an ash-pump house of the CHP plant. The tests on fly ash and slag concentration were carried out separately.

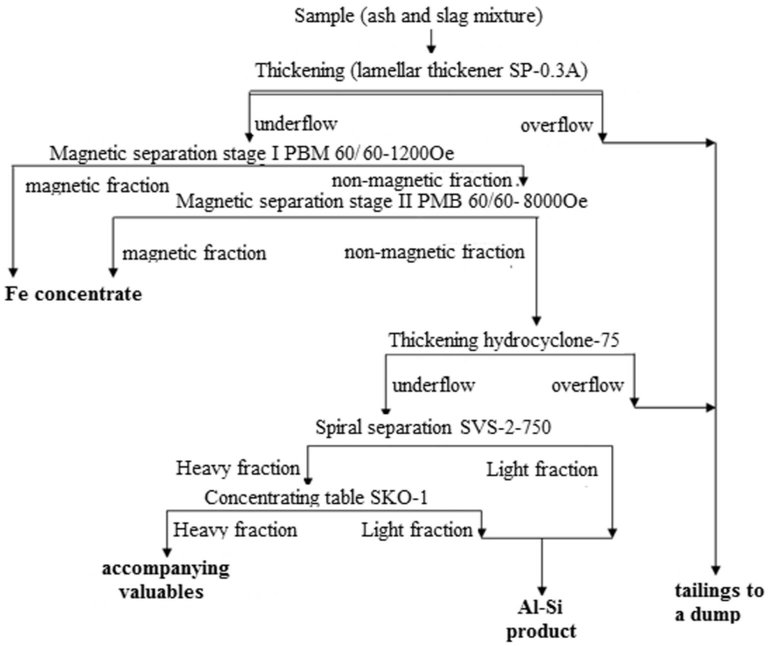


Figure 2. Pilot plant flow sheet for fly ash processing

Table 3 –Chemical composition of the resulting products generated during fly ash of CHP plant-9 concentration

| Material name | Chemical composition of the material, % | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | $Fe_{total}$ | $S_{total}$ | As | P | Cu | Pb | $Al_2O_3$ | $SiO_2$ | ZnO | CaO | MnO |
| Iron concentrate | 58.3-63.0 | 0.11 | 0.02 | 0.018 | 0.006 | 0.0006 | 5.45 | 13.00 | 0.015 | 2.23 | 0.30 |
| Silicate product | 5.24 | - | - | - | - | - | 24.3 – 28.2 | 58.10 | - | - | - |

Pilot plant tests confirmed the test work results in terms of iron-bearing concentrate quality (the content of iron is, up to 63%), the content of detrimental impurities (sulfur is less than 0.2%, zinc is less than 0.12%) and aluminum silicate product ($Al_2O_3$ 24.3 – 28.2 %). The mass yield of iron-bearing concentrate and aluminum silicate product from the ash and slag wastes volume was 3-9% and 40-60%, respectively.

Test work and pilot plant tests conducted demonstrated a technical feasibility to process ash and slag wastes efficiently. This resulted in their almost complete recycling and reduction in environmental impact combined with profit generation from selling saleable products.

Economic calculations showed that the payback period for the construction of ash dumps concentration plant with the throughput of 1.0 MTPA is 2 years.

The production line is equipped with environmentally-friendly equipment produced in Russia.

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# Implementation of Fine Screening Technology in Fine Coal Tailing Circuits

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## ABSTRACT

The coal industry has been moving toward elimination or size reduction of tailings ponds, as they pose major environmental and potential safety concerns. Thickening and dewatering of the dilute slurry to produce a high solids discharge has become the industry standard practice for processing coal tailings. Large filter presses are normally used for dewatering the thickener underflow. However, filter presses are well known for their high cost, non-continuous operation, high maintenance, and large floor space requirements. Alternatively, high frequency fine dewatering screens have been applied to dewater thickener underflow, working in conjunction with filter presses. This process not only produces a high solids discharge, eliminating the need for a large number of tailings ponds, but it also reduces drastically the high cost of large filter presses, ultimately reducing the overall dewatering costs.

This paper describes two applications in which coal processors in China and India have benefitted from fine screening technology.

**Key Words:** Fine Screening Technology, Coal Tailings, Classification, Dewatering, Fine Coal Recovery

## INTRODUCTION

The generation of fine coal tailings is inevitable with modern coal production technology and coal beneficiation technology. Therefore, a large number of huge tailings ponds are already present or being built to store those fine coal tailings. Recent estimated by US coal industry suggests that at least 2 billion tons of fine coal have been discarded in more than 300 ponds, and additionally, the U.S. coal producers discard approximately 30 to 40 million tons of fresh fine coal to ponds (Yoon and Basilio, 1997; Condie et al., 1996 and 1997, Couch, 1991). Those tailings ponds occupy massive good lands, posing major environmental and potential safety concerns to the general public. The coal industry has been moving toward elimination or reducing the size of the tailings ponds. A great number of research work has been conducted regarding this issue (Singh, 1997; Lockhart and Veal, 1996; Yoon, 1999; Basim and Yoon, 1998). Thickening and dewatering of the dilute slurry to produce a high solids discharge has become the industry standard practice for processing coal tailings. Large filter presses are normally used for dewatering the thickener underflow. However, filter presses are well known for their high cost, non-continuous operation, high maintenance, and large floor space requirements. There are other technologies to handle this problem. One way is to dewater the tailings with alternative technology and another is to reduce the amount of fine tailings generated. The paper discusses two case studies that are examples of those two approaches.

The handling of fine solids, being generated in the coal washing plants, is by far the largest industry concern. The Derrick Hi-G Dryer (Figure 1) provides the most cost-effective solution for handling these fines. This proven technology consists of a Derrick Hi-G force linear motion vibratory screen and a radial design cluster of Derrick 4” unibody hydrocyclones. The hydrocyclones are fed slurry under a consistent pressure of 35 to 40PSI. The centrifugal separation extracts the slimes, discharging them out the underflow at 55 to 60 percent solids, which are then fed to a high G-force dewatering screen. The Derrick Hi-G dryer dewatering screen is outfitted with two 2.5HP electromechanical vibrators operating at 1500RPM with a stroke length of 0.19”. The combination of stroke length and rotational speed create 7.3 G’s of acceleration on the screen surface, resulting in efficient separation of water from the fine solids. The dewatered fine solids discharged from the screening machine contain 70 to 75 percent solids.

 

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**Figure 1. Derrick Hi-G Dryer**

## CASE STUDIES

### Dewatering of Coal Washery Tailings and Elimination of Tailings Ponds

At a coal washery in India, fine tailings (typically –0.5mm) from the flotation circuit were thickened in a thickener and then pumped to a tailings pond. After settlement, clear water was pumped back to the washery, and solids were allowed to dry; the solids were then removed and sold. This process required a large land area for the tailings pond, and considerable of water was being lost to evaporation and seepage.

Derrick Hi-G Dryers were recommended to assist in dewatering the thickened solids. The units are now operating in combination with filter presses to provide on-line solids dewatering. Clear water is recirculated to the washery, and dewatered solids are taken to a storage yard for disposal and/or sale. In the plant, which was designed with assistance from Derrick Corporation, tailings from washery 2 and washery 3 are pumped to this plant where hydrocyclones are used for separating solids at 0.15mm and thickening to 50 percent solids. Thickened solids are dewatered in six Hi-G Dryers to contain 25 to 30 percent moisture in the discharge as shown in Table 1, and the –0.15mm fraction is sent for dewatering in filter presses. Nearly 75 percent of the total 240 t/h solids are recovered by Derrick screens, and 25 percent are dewatered in the filter presses. Clear water containing less than 100ppm solids is pumped back to the respective washeries, while solids are stored for disposal.

This environmentally friendly plant has helped to eliminate several tailings ponds. Other coal companies are now planning to implement this technology in their coal washeries. The flow sheet (Figure 2) and photograph (Figure 3) show the Derrick equipment in operation.

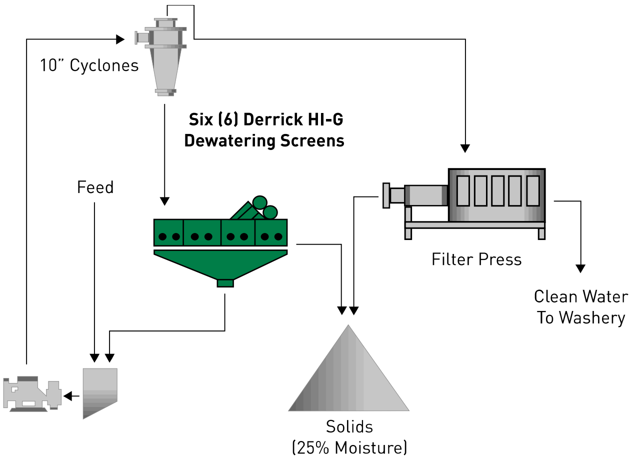


**Figure 2. New flow sheet incorporating Derrick Hi-G® Dryers**

**Figure. 3 HI-G® Dryer dewatering thickened solids**

**Table 1. Percent Solids of Various Streams**

| Feed | Cyclone Overflow | Cyclone Underflow | Screen Oversize |
|---|---|---|---|
| **Solid%** | **Solid%** | **Solid%** | **Moisture%** |
| 11.65 | 4.85 | 61.1 | 27.03 |
| 11.65 | 5.96 | 33.33 | 29.42 |
| 11.65 | 6.2 | 63.48 | 26.88 |
| 11.65 | 4.91 | 36.04 | 27.43 |
| 11.65 | 5.02 | 39.68 | 26.45 |
| 11.65 | 4.51 | 61.27 | 27.19 |

## Thermal Coal Processing Plant in Inner Mongolia - Tailings Recovery Application

Fine coal below 0.5mm is relatively difficult and expensive to beneficiate with current gravity- or flotation-based technologies (Zhang et al., 2011; Mohanty et al., 2002). It is, therefore, common for coal processing plants, especially thermal coal producers, to send the fines stream to the tailings thickener without any further processing. Even for those plants with fine coal cleaning circuits, frequently a significant amount of fine coal is lost to the tailings thickener due to inefficient equipment performance or circuit arrangements. Fine screening as fine as 45 microns has become an important technology to reduce the fine coal ash content and recover fine clean coal from the thickener feed.

A major thermal coal producer in Inner Mongolia had initiated a testing program targeting recovery of fine clean coal from thickener feed.. No flotation circuit was installed to recover the –0.5mm size fraction coal and, therefore, fine clean coal contained in this size range was being lost to the coal thickener. The current flowsheet for this stream (Figure 4) consists of a fine coal stream (also known as tailings stream for the plant) fed to the thickener and then the thickener underflow, which reports to the filter press.The alternative circuit designed to recover fine clean coal from the tailings stream is shown in Figure 5. In this circuit, a bypass stream from the thickener feed is directed to a Derrick high frequency screen fitted with non-blinding polyurethane panels. The oversize product is dewatered using the host plant's existing coal centrifuge. Dewatered fine clean coal product is recovered, and the screen undersize product is sent back to the thickener.

After the screen installation, samples from the feed, oversize, and undersize streams were collected and analyzed. A total of 30 samples from 10 different test runs were included in the sampling. The odd-numbered test runs were conducted without the use of repulp wash water, and the even test runs were conducted with repulp wash water. Size and ash analyses were conducted with 0.50mm, 0.25mm, 0.125mm, and 0.075mm screens for test samples 2, 5, and 10. For the remaining seven test samples, the size and ash analyses were conducted using only a 0.075mm screen.

Solids concentration in the constantly fluctuating feed conditions at the host plant was about 30 to 50g/L, and the flowrate was controlled to about 50 to 70$m^3$/h. Results from the 10 test runs are listed in Tables 4 and 5. The feed ash content was in the range of 19.23 to 31.13 percent, averaging 26.35 percent. The feed contained 18.78 to 53.44 percent +0.075mm fraction, which had a low ash content. The average +0.075mm fraction weight percentage and ash content were 30.68 and 14.89 percent, respectively. This fraction was being lost to the coal thickener and eventually discarded as tailings.

In the new circuit (Figure 5), the Derrick fine screen fitted with 0.075mm panel recovered 27.01 percent of +0.075mm oversize product (very close to 30.68 percent theoretical yield value) with ash content of 14.51 percent (very close to 14.89

percent theoretical yield value). The oversize product contained 92.26 percent +0.075mm size fraction. In other words, the high ash ultrafine clays (–0.075mm size fraction) were removed successfully. With the addition of repulp wash water, the content of +0.075mm in oversize product was improved from 88.65 to 95.87 percent, which clearly showed the improved performance resulting by adding repulp wash water. The average undersize ash content was 30.73 percent. A certain amount of +0.075mm size fraction would pass through the panel due to rectangular panel openings. The average amount of bypassed material was 8.29 percent, and the largest bypass particle was about 0.125mm. Figure 6 shows the +0.075mm percentage and ash content changes in the feed, oversize, and undersize streams. Significant changes in ash content and size distribution are evident in the streams, clearly demonstrating the high efficiency of the Derrick fine screen.

Other than the size and ash comparisons, the undersize separation efficiency (amount of –0.075mm in undersize stream versus amount of –0.075mm in feed stream), the undersize ash rejection (amount of ash in undersize stream versus amount of ash in feed stream) and ash reduction (oversize ash versus feed ash) were also calculated and listed in Table 2. Undersize separation efficiency ranged from 81.86 to 99.66 percent, averaging 96.57 percent. Ash rejection ranged from 60.97 to 94.55 percent, averaging 85.13 percent. Ash reduction achieved ranged from 42.08 to 70.99 percent, averaging 55.07 percent. The addition of repulp wash water helped to improve undersize separation efficiency, ash rejection, and ash reduction by 5.39, 6.33, and –1.66 percent, respectively. Figure 8 shows the relationship between undersize separation efficiency and ash rejection, as well as the correlation between the feed +0.075mm percentage and oversize mass yield, with the two empirical models listed.

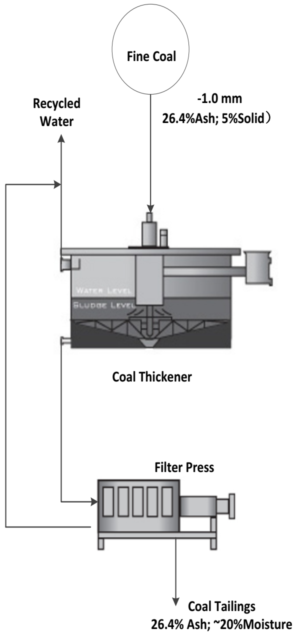


**Figure 4. Existing fine coal cleaning circuit**

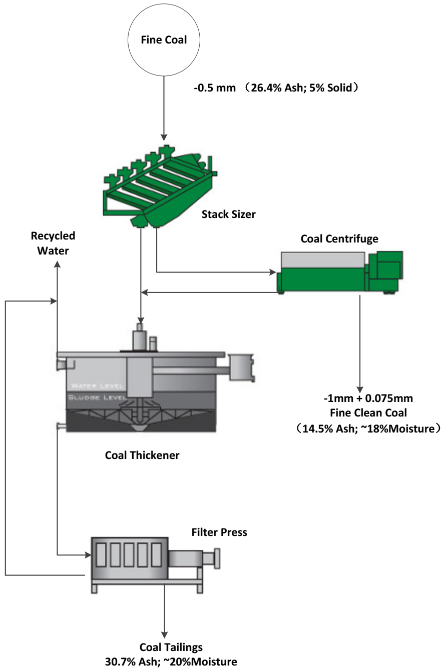


**Figure 5. Alternative fine coal cleaning circuit with fine screen**

**Table 2. Detailed size and ash analysis for test 2, 5 and 10**

| Test ID | Size | Feed | | Oversize | | Undersize | |
|---|---|---|---|---|---|---|---|
| | | Weight | Ash | Weight | Ash | Weight | Ash |
| | mm | % | % | % | % | % | % |
| #2 | >0.5 | | | 6.16 | 9.05 | | |
| | 0.5-0.25 | 1.38 | 8.55 | 17.91 | 9.61 | | |
| | 0.25-0.125 | 6.65 | 10.80 | 53.44 | 10.05 | | |
| | 0.125-0.075 | 10.75 | 17.10 | 20.28 | 12.06 | 7.02 | 19.80 |
| | -0.075 | 81.22 | 27.82 | 2.21 | 24.42 | 92.98 | 28.03 |
| | **Total** | 100.00 | 25.27 | 100.00 | 10.64 | 100.00 | 27.45 |
| #5 | >0.5 | | | 3.69 | 7.62 | | |
| | 0.5-0.25 | 5.82 | 8.63 | 17.20 | 9.85 | | |
| | 0.25-0.125 | 14.04 | 11.36 | 47.87 | 12.25 | | |
| | 0.125-0.075 | 11.72 | 18.09 | 18.74 | 14.42 | 5.97 | 25.03 |
| | -0.075 | 68.41 | 32.82 | 12.50 | 34.41 | 94.03 | 32.55 |
| | **Total** | 100.00 | 26.68 | 100.00 | 14.84 | 100.00 | 32.10 |
| #10 | >0.5 | | | 6.29 | 9.69 | | |
| | 0.5-0.25 | 5.89 | 9.94 | 16.98 | 11.96 | | |
| | 0.25-0.125 | 11.63 | 10.07 | 46.06 | 12.73 | | |
| | 0.125-0.075 | 9.05 | 11.87 | 22.62 | 15.06 | 6.43 | 24.84 |
| | -0.075 | 73.44 | 33.06 | 8.06 | 35.20 | 93.57 | 31.34 |
| | **Total** | 100.00 | 27.11 | 100.00 | 14.75 | 100.00 | 30.92 |

**Table 3. Summary of size and ash analysis at 0.075mm for 10 test runs**

| Test ID | Size | Feed | | Oversize | | Undersize | |
|---|---|---|---|---|---|---|---|
| | | Weight | Ash | Weight | Ash | Weight | Ash |
| | mm | % | % | % | % | % | % |
| #1 | **0.075** | 53.44 | 10.06 | 85.81 | 9.96 | 5.80 | 19.44 |
| | **-0.075** | 46.56 | 29.75 | 14.19 | 28.58 | 94.20 | 29.56 |
| | **Total** | 100.00 | 19.23 | 100.00 | 12.60 | 100.00 | 28.97 |
| #2 | **0.075** | 18.78 | 14.24 | 97.79 | 10.3244 | 7.02 | 19.80 |
| | **-0.075** | 81.22 | 27.82 | 2.21 | 24.42 | 92.98 | 28.03 |
| | **Total** | 100.00 | 25.27 | 100.00 | 10.64 | 100.00 | 27.45 |
| #3 | **0.075** | 31.52 | 12.61 | 93.38 | 9.74 | 8.05 | 11.70 |
| | **-0.075** | 68.48 | 29.13 | 6.62 | 29.76 | 91.95 | 30.29 |
| | **Total** | 100.00 | 23.92 | 100.00 | 11.07 | 100.00 | 28.80 |
| #4 | **+0.075** | 29.78 | 16.13 | 98.85 | 12.28 | 11.85 | 18.39 |
| | **-0.075** | 70.22 | 30.13 | 1.15 | 44.98 | 88.15 | 30.89 |
| | **Total** | 100.00 | 25.96 | 100.00 | 12.65 | 100.00 | 29.41 |
| #5 | **+0.075** | 31.59 | 13.36 | 87.50 | 12.05 | 5.97 | 25.03 |
| | **-0.075** | 68.41 | 32.82 | 12.50 | 34.41 | 94.03 | 32.55 |
| | **Total** | 100.00 | 26.68 | 100.00 | 14.84 | 100.00 | 32.10 |
| #6 | **+0.075** | 33.93 | 15.63 | 96.75 | 12.94 | 10.72 | 24.77 |
| | **-0.075** | 66.07 | 31.73 | 3.25 | 33.53 | 89.28 | 31.69 |
| | **Total** | 100.00 | 26.27 | 100.00 | 13.61 | 100.00 | 30.94 |
| #7 | **+0.075** | 26.65 | 23.49 | 87.63 | 16.19 | 12.57 | 30.29 |
| | **-0.075** | 73.35 | 33.96 | 12.37 | 33.68 | 87.43 | 34.68 |
| | **Total** | 100.00 | 31.17 | 100.00 | 18.35 | 100.00 | 34.12 |
| #8 | **+0.075** | 26.76 | 19.75 | 94.02 | 20.90 | 6.30 | 25.98 |
| | **-0.075** | 73.24 | 34.55 | 5.98 | 34.58 | 93.70 | 33.78 |
| | **Total** | 100.00 | 30.59 | 100.00 | 21.71 | 100.00 | 33.29 |
| #9 | **+0.075** | 27.77 | 12.95 | 88.91 | 12.78 | 8.16 | 19.87 |
| | **-0.075** | 72.23 | 32.78 | 11.09 | 31.59 | 91.84 | 32.26 |
| | **Total** | 100.00 | 27.27 | 100.00 | 14.87 | 100.00 | 31.25 |
| #10 | **+0.075** | 26.56 | 10.65 | 91.94 | 12.95 | 6.43 | 24.84 |
| | **-0.075** | 73.44 | 33.06 | 8.06 | 35.20 | 93.57 | 31.34 |
| | **Total** | 100.00 | 27.11 | 100.00 | 14.75 | 100.00 | 30.92 |

**Table 4. Evaluation of Derrick fine screen performance**

| Test ID | Feed | Oversize | Undersize | Feed | Oversize | Undersize | Oversize Yield | Classification Efficiency | Ash Rejection | Ash Reduction |
|---|---|---|---|---|---|---|---|---|---|---|
| | +0.075mm | +0.075mm | +0.075mm | Ash% | Ash% | Ash% | % | % | % | % |
| #1 | 53.44 | 85.81 | 5.80 | 19.23 | 12.60 | 28.97 | 59.54 | 81.86 | 60.97 | 65.56 |
| #2 | 18.78 | 97.79 | 7.02 | 25.27 | 10.64 | 27.45 | 12.96 | 99.65 | 94.55 | 42.08 |
| #3 | 31.52 | 93.38 | 8.05 | 23.92 | 11.07 | 28.80 | 27.51 | 97.34 | 87.27 | 46.26 |
| #4 | 29.78 | 98.85 | 11.85 | 25.96 | 12.65 | 29.41 | 20.61 | 99.66 | 89.96 | 48.74 |
| #5 | 31.59 | 87.50 | 5.97 | 26.68 | 14.84 | 32.10 | 31.42 | 94.26 | 82.52 | 55.64 |
| #6 | 33.93 | 96.75 | 10.72 | 26.27 | 13.61 | 30.94 | 26.98 | 98.67 | 86.02 | 51.81 |
| #7 | 26.65 | 87.63 | 12.57 | 31.17 | 18.35 | 34.12 | 18.76 | 96.84 | 88.96 | 58.88 |
| #8 | 26.76 | 94.02 | 6.30 | 30.59 | 21.71 | 33.29 | 23.33 | 98.10 | 83.44 | 70.99 |
| #9 | 27.77 | 88.91 | 8.16 | 27.27 | 14.87 | 31.25 | 24.28 | 96.27 | 86.77 | 54.51 |
| #10 | 26.56 | 91.94 | 6.43 | 27.11 | 14.75 | 30.92 | 23.54 | 97.42 | 87.20 | 54.39 |
| Odd Average | 34.19 | 88.65 | 8.11 | 25.65 | 14.35 | 31.05 | 32.30 | 93.31 | 81.30 | 55.92 |
| Even Average | 27.16 | 95.87 | 8.46 | 27.04 | 14.67 | 30.40 | 21.48 | 98.70 | 88.23 | 54.26 |
| Average | 30.68 | 92.26 | 8.29 | 26.35 | 14.51 | 30.73 | 27.01 | 96.57 | 85.13 | 55.07 |

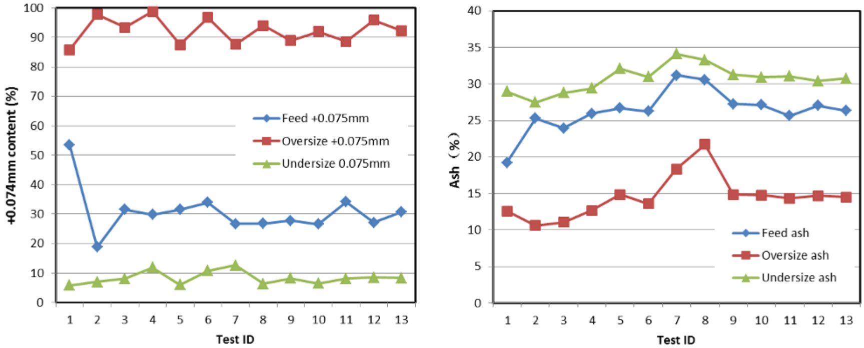


**Figure 6. Changes in +0.075mm content and ash content within feed, oversize and undersize**

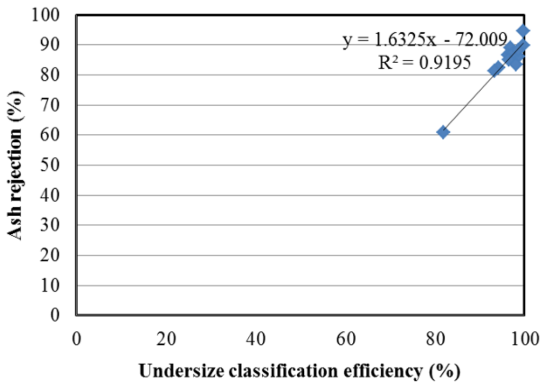


**Figure 7. Undersize classification efficiency versus ash rejection**

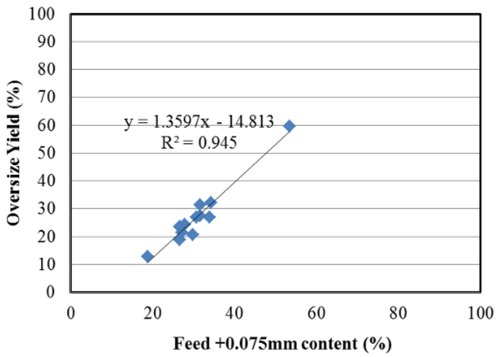


**Figure 8. The relationship between feed +0.075mm content and oversize yield**

## CONCLUSION

The two case studies discussed above, as well as results from numerous installations worldwide, have demonstrated the superiority of Derrick high frequency Hi-G fine screens in providing highly efficient, high-capacity fine screening or dewatering solutions. The efficient removal of high ash fine coal significantly reduces oversize ash content, thereby improving the overall clean coal yield and ash content. Derrick's patented fine urethane screen surface technology has been proven to be a long wearing, anti-blinding screen surface for coal applications that require size classification/dewatering as fine as 25 microns.

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# Problem Analysis and Optimization Test for Separation Process of Coal Slime Classifying Flotation

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**Abstract**：This paper tried to solve problems caused by industrial application of "2+2" mold coal slime classifying flotation system. The inefficiency of equipment and ambiguity of particle size will cause insufficient in-material of first flotation and overload in-material of second flotation. Technology of classifying hydrocyclone or sieve bend series has been applied in the process of classification, and flotation column in second flotation. A series of tests combined with practical application were conducted to verify the technology mentioned above. The result shows that ambiguity of particle size are acceptable if coarse and fine particle are separated by subsequent equipments, and application of classifying equipment such as sieve bend and classifying hydrocyclone, which is more flexible in operation, are also employed in the process of classifying flotation; If we focus on recovery rather than separation after separating coarse slime, classification particle size should be accurate, thus efficient classifying equipments like sieve bend series should be applied. At the same time, in order to eliminate the effect of fine slime, desliming of coarse slime should also be considered. With high yield, low ash content and improved perfection index, flotation column as equipment of second flotation shows great advantages comparing with regular flotation machine.

**Keywords**: coal slime; classifying flotation；classifying equipment; classifying particle size; classifying hydrocyclone ; sieve bend；flotation column

With the improvement of mechanization in coal mining，the raw coal quality declines gradually, and the proportion of fine and ultrafine slime in the raw coal are increasing year by year. Therefore, in recent years, the advantages of classified flotation technologies have become more and more obvious comparing with bulk flotation (Xie Guangyuan, Wu Ling, Ou Ze-shen,et al,2004). Classification equipment can separate coal slime into two parts: coarse slime and fine slime. Based on different properties such as floatability and filtration of coarse and fine slime, corresponding flotation and dewatering equipments, and reagent system could be used to take full advantage of slime treatment equipment' separating and dewatering ability in different particle size range, helping coal production enterprises gain good economic benefits(Xie Guangyuan, Wu Ling, Ou Ze-shen,et al,2005). At present, there are various of classified flotation technologies, among which classification is the premise and the key. Classification equipment, classification technique, and particle size are hot research areas now. According to classification process, classification efficiency of common classification equipment is evaluated by small screening tests and practical application. Which particle size is optimum to take advantage of classified flotation and which classification equipment is adopted to achieve the optimum classification efficiency were researched and discussed, providing beneficial reference to designing and producing of classifying flotation technique.

## 1. Introduction of typical slime classifying flotation technique

Fig.1 shows five procedures of technique process:①Cutting coarse of flotation feed. Sieve bend is adopted to cut ＞0.25mm particles that exceed the upper limit of the flotation feed and with certain separation and low ash in gravity separation.②Subsequent flotation. Cutting coarse flotation feed can discharge high-ash fine slime in tailings by separating of subsequent flotation. ③Recovery of coarse

 
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slime. Cut coarse slime mixed with foam products of subsequent flotation are fed into sedimentation-filtration dewatering centrifuge for recovery.④Secondary flotation. Centrifugal liquid and filtrate(＜0.045mm) of sedimentation-filtration dewatering centrifuge are fed to secondary flotation.( Gao Ｒong, Long Zhanyuan, Zhang Yebing, et, al,2010).⑤Recovery of fine slime. Foam products in secondary flotation can be recovered by filter press that with complete solid-liquid separating. (Pang Liang,2011).

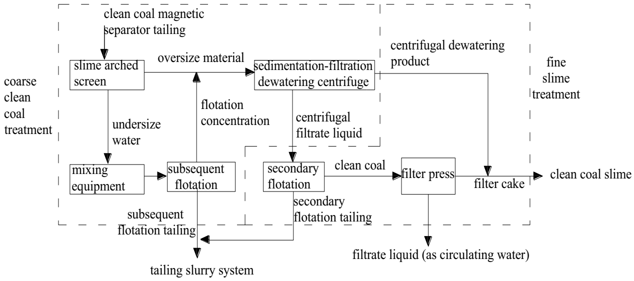


Fig.1 Typical case 1 of slime classifying flotation technique

Fig2 shows that classification hydro cyclone group is adopted as classification equipment in this technical process, 0.5～0.125mm coarse slime is separated and recovered by traditional flotation machine and pressure filter; 0.125～0mm fine slime is treated by flotation column and clean coal pressure filter.(Li Zhen-tao, Zhang Yue-qiu, Xie Guang-yuan, et, al,2007)

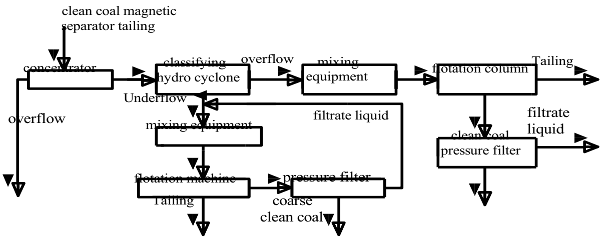


Fig.2 Typical case 2 of slime classifying flotation technique

Two technical processes mentioned above are relatively typical slime classifying flotation technique at present, respectively adopted sieve bend and classifying hydro cyclone as classification equipment. As classification particle boundaries are 0.25mm and 0.125mm, they separate slime into two parts, coarse slime and fine slime, to conduct further procedure (Sha Jie, Xie Guang-yuan, Wu Ling, 2009). Although following separation and equipment are not the same, their purposes are similar.

**2. Application of classification equipment**

For classification of fine slime, particle size is not a relative conception but an absolute one. This paper focuses on the influence that caused by distributing particle size of different slime separation equipment in practical application. (Ji Yu-hua, Zhao Tian-bo, Hao Tian-feng, et, al,2012)

Tianchen coal preparation plant in Zaozhuang Mining Group adopted slime classifying flotation technique shown in Fig 1 before July, 2012. With the increasing of slime amount and screen feed, swapping material happened in screen, that made subsequent equipments cannot have enough feed in subsequent flotation but excessive feed in secondary flotation. Thus transformation was speeded up in cutting coarse procedure of slime classifying flotation, that is, classifying hydro cyclone replaced sieve bend in cutting coarse procedure.

**2.1 Classification effect of fine slime arched screen before transformation**

To comprehensively compare effects of different classification equipment in different classifying flotation technique, industrial tests were done on classification equipment of Tianchen coal preparation plant before and after transformation.

We can see from table1 that ash content had an obvious growth as slime particle decrease. The ash content of 0.25～0.125mm particle size in feed is 7.08%,which meet the requirement of clean coal ash, indicating recovery should be adopted. The ash content of particle size＜0.030mm increase rapidly, showing that high-ash clay minerals gathering after degradation in water. Particle size ＜0.125mm take 48.49 percentage of oversize product, in which particle size＜0.030mm with 26.59%, indicating that screen surface operate abnormally, such as water running out and slime swapping.

Table 1 data of feed particle size composition and distribution rate in arched screen

| Particle size/mm | feed | | oversize | | undersize | | composition | | Distribution rate % |
|---|---|---|---|---|---|---|---|---|---|
| | Yield % | Ash content % | Yield % | Ash content % | Yield % | Ash content % | Yield % | Ash content % | |
| >0.5 | 5.31 | 5.79 | 7.88 | 5.58 | 0.17 | 6.54 | 5.25 | 5.59 | 98.90 |
| 0.5-0.25 | 10.70 | 6.06 | 16.26 | 6.15 | 0.24 | 6.02 | 10.80 | 6.15 | 99.24 |
| 0.25-0.125 | 20.62 | 7.08 | 27.37 | 6.45 | 6.17 | 7.58 | 20.15 | 6.57 | 89.57 |
| 0.125-0.075 | 8.56 | 11.64 | 9.70 | 10.87 | 6.25 | 13.15 | 8.52 | 11.44 | 75.02 |
| 0.075-0.045 | 13.41 | 15.26 | 8.10 | 14.76 | 23.59 | 16.64 | 13.38 | 15.89 | 39.92 |
| 0.045-0.030 | 4.76 | 17.43 | 4.10 | 16.98 | 4.17 | 17.66 | 4.12 | 17.21 | 65.55 |
| <0.030 | 36.64 | 33.09 | 26.59 | 31.78 | 59.42 | 33.05 | 37.78 | 32.46 | 46.41 |
| Sub-total | 100.00 | 18.41 | 100.00 | 14.60 | 100.00 | 25.61 | 100.00 | 18.35 | |

From distribution curve in Fig 3, we can obtain distribution particle size is 0.014mm, representing the probability of this very particle size entering oversize and below size products are half and half. Which has a large different with required 0.25mm by technique and equipment.

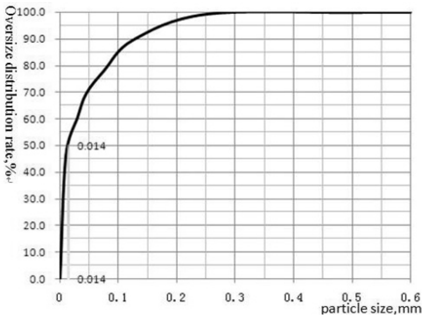


Fig.3 Distribution rate of arched screen

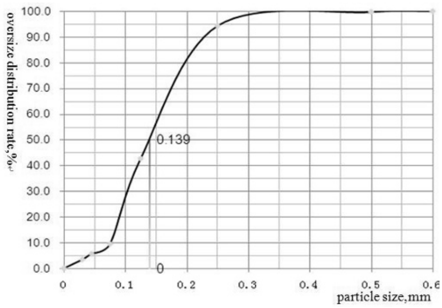


Fig.4 Distribution rate of hydro cyclone

**2.2 Effect of post transformation classifying hydro cyclone**

We can see from table 2 that particle size ＞0.25mm in hydro cyclone overflow is 1.24%, taking low percentage of the production rate at the corresponding level, that is; particle size <0.045mm is a dominance, which is 62.07%,and ash content is 32.03%. While particle size <0.125mm in hydro cyclone underflow take low percentage at the corresponding level, that is 9.72%. Empirical data that mentioned above showed that hydro cyclone has an extraordinary classifying effect.

Table 2 data of feed particle size composition and distribution rate in classifying hydro cyclone

| Particle size / mm | Actual feed | | Overflow | | | Underflow | | | Composition | | Distribution rate |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | $\gamma$ | $A_d$ | $\gamma$ 占本级 | $\gamma$ 占入料 | $A_d$ | $\gamma$ 占本级 | $\gamma$ 占入料 | $A_d$ | $\gamma$ | $A_d$ | |
| >0.5 | 8.41 | 6.53 | 0.04 | 0.03 | 3.79 | 26.89 | 8.38 | 3.43 | 8.41 | 3.43 | 99.67 |
| 0.50-0.250 | 17.56 | 6.61 | 1.20 | 0.83 | 6.79 | 43.89 | 13.68 | 7.11 | 14.50 | 6.92 | 94.30 |
| 0.250-0.125 | 14.88 | 6.64 | 11.83 | 8.14 | 7.78 | 19.50 | 6.08 | 9.08 | 14.22 | 8.34 | 42.73 |
| 0.125-0.075 | 10.31 | 8.78 | 15.24 | 10.49 | 6.10 | 3.53 | 1.10 | 21.39 | 11.59 | 7.55 | 9.49 |
| 0.075-0.045 | 6.96 | 12.95 | 9.62 | 6.62 | 10.94 | 1.27 | 0.40 | 43.48 | 7.02 | 12.77 | 5.64 |
| <0.045 | 41.88 | 32.00 | 62.07 | 42.73 | 32.03 | 4.92 | 1.53 | 43.49 | 44.26 | 32.43 | 3.46 |
| total | 100.00 | 17.91 | 100.00 | 68.84 | 22.83 | 100.00 | 31.16 | 9.26 | 100.00 | 18.60 | |

From distribution curve shown in Fig 4, we can obtain the distribution particle size of classifying hydro cyclone is 0.14mm, indicating that the probability of this very particle size entering overflow and underflow is half and half, also showing that particle size >0.14mm incline to underflow, which undertaken by screen bowl centrifuge, about 85% of particle size >0.045mm can be recovered. We can see from table 2 that ash content of particle size >0.125mm in hydro cyclone feed is below 10%, which meet clean coal requirement. This part of feed should be recovered rather than separated. Therefore, in this process, classifying hydro cyclone divide coarse and fine slime in 0.14mm, accomplishing the goal of classifying flotation.

**2.3 Comparation of industrial effect of different classification equipment**

By comparing two classification equipment data that listed in table 3, we can know that solid yield in under screen water of fine slime arched screen is 34.06%, while the overflow solid yield of classifying hydro cyclone is 68.84%. This replacement solve the problem that subsequent equipment cannot have enough feed in subsequent flotation; furthermore, underflow density of hydro cyclone reach at 709.2g/L, avoiding water in sieve bend. Particle size >0.25mm in hydro cyclone overflow take only 1.24 percentage at the corresponding level, means that classification hydro cyclone can prevent coarse slime from subsequent flotation. Therefore, Tianchen coal preparation plant replaced sieve bend by classifying hydro cyclone as classification equipment in classifying flotation process can meet the requirement of technique well.

Table 3 Quality balance sheet

| Technical index | Clean coal arched screen | | | Classifying hydro cyclone | | |
|---|---|---|---|---|---|---|
| | feed | Undersize water | Oversize slime | feed | Over flow | Under flow |
| flow/( $m^3 \cdot h^{-1}$) | 376.84 | 174.51 | 202.33 | 396.84 | 369.89 | 26.95 |
| Dry slime / ($t \cdot h^{-1}$) | 56.39 | 19.21 | 37.18 | 51.49 | 34.92 | 19.11 |
| yield/% | 100.00 | 34.06 | 65.94 | 100.00 | 68.84 | 31.16 |
| concentration/% | 14.33 | 10.66 | 17.43 | 12.51 | 9.24 | 57.36 |
| concentration /( $g \cdot L^{-1}$) | 150 | 110 | 184 | 130 | 94 | 709 |
| Ash content/% | 18.41 | 25.61 | 14.69 | 17.91 | 22.83 | 9.26 |

**3 discussion of classification equipment's application**

As classification equipment, classifying hydro cyclone and sieve bend have merits and defects respectively. The former can separate and concentrate slurry in a centrifugal force field, thus has merits such as high efficiency of classification, no moving parts, low energy cost, small occupied area, and operating readily, etc; the defect is that centrifugal force of material can be influenced by not only particle size but also density of the material (Pang Liang ,2011). As to sieve bend, slurry is fed to screen surface along tangential direction with a certain velocity through feed box. Screen bars play an important role in cutting slurry, thus can dewater and deslime in large amount, but also running out coarse slime and water readily is also a defect in practical application (Ji Yu-hua, Zhao Tian-bo, Hao Tian-feng, et, al,2012). The replacement in Tianchen coal preparation plant can solve the problems that restricting classifying flotation technique.

Shengyu coal preparation Ltd in Panxian county, Guizhou province adopt slime classifying flotation technique shown in Fig 1. The particle size ＞0.25mm in magnetic separator's tailing yield 24.49%, average ash content is below 10.22%, meeting the requirement of slime ash content, thus should be recovered. While particle size ＜0.25mm should be separated in flotation machine because its high ash content. According to this situation, the optimum particle size is 0.25mm. If adopted classifying hydro cyclone in Tianchen coal preparation plant, high-ash slime will mix with clean coal, and ash content of product can't meet the demand. But if sieve bend's surface is under controlling properly and slime doesn't swapping, classification target will be easily accomplished and management will be convenient. Therefore, in practical, to increase assurance of classification process, sieve bend series can be adopted to cut coarse slime and avoid swapping slime and water.

**4. Selection and upgrade of secondary flotation equipment**

Both small flotation machine and small column separation tests should be given to secondary flotation feed with equal reagent amount. Results in table 4 shows that, with equal reagent amount, small flotation column has advantage in clean coal yield, ash content, and flotation perfection index, indicating that flotation columns have merits when comparing with traditional flotation machine.

Particle <0.045mm is a dominance in secondary flotation feed. Flotation columns have advantage such as stable flotation equipment, no mechanical stirring device, heavy foam layer, and strong secondary concentration. Water injection at the top forced high ash slime entrained among foams discharging, which highlight the precise of classification (An Lijun,2011 ).

Table 4 Test result comparation of small flotation machine and flotation column

| equipment | Kerosene consumption g/t | 2-octanol consumption g/t | Clean coal Yield % | Clean coal Ash content % | tailing Yield % | tailing Ash content % | Flotation Perfection % |
|---|---|---|---|---|---|---|---|
| small flotation machine | 1000 | 100 | 29.67 | 12.45 | 70.33 | 42.01 | 27.80 |
| | 1250 | 125 | 39.50 | 12.14 | 60.50 | 48.19 | 38.42 |
| | 1500 | 150 | 45.45 | 12.17 | 54.55 | 51.22 | 43.48 |
| | 1750 | 175 | 54.66 | 12.68 | 45.34 | 56.90 | 49.77 |
| flotation column | 1000 | 100 | 31.52 | 12.12 | 68.48 | 43.37 | 30.27 |
| | 1250 | 125 | 42.45 | 12.01 | 57.55 | 49.56 | 41.11 |
| | 1500 | 150 | 47.58 | 12.05 | 52.42 | 52.32 | 45.32 |
| | 1750 | 175 | 56.12 | 12.64 | 43.88 | 59.32 | 51.89 |

**5. Conclusion**

Classification procedure is an important restrictive factor in classifying flotation technique. How to classify the coarse and fine slime is related to whether classifying flotation can play the biggest advantage of technology or not. Practical application proofs that:

(1) The grading particle size of sieve bend is 0.25mm or 0.20mm. In normal condition, separating according to particle size is better. But if feed is fluctuating, swapping slime or water is readily take place in screen surface, thus technical effect will be influenced strongly. Sieve bend series can solve the problem that mentioned above, and increase the assurance of classification process.

(2) The grading particle size of classifying hydro cyclone is related to feed pressure and underflow amount, and can be adjust at a certain range. The grading effect can be influenced by not only particle size but also density of the material. Classifying hydro cyclone also has a fuzzy limit when comparing with sieve bend.

(3) The setting of classification procedure boned closely with subsequent techniques. Both classification equipments have its own merits, so it is important to choose equipment properly based on feed and technique in practical.

(4) As secondary flotation equipment, flotation column has more advantages than conventional flotation machine. Under the condition of the same reagent consumption, the cleaned coal yield and flotation index increase, while the ash content reduces.

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# The Progress and Review of Fine Coal Pulp-mixing Technology

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**Abstract:** Pulp-mixing operation is the premise and safeguard to achieve effective mineralization and separation. According to the present difficulties in slime flotation, the paper provided that pretreatment of pulp-mixing could be effective. With added force, the particle dispersion and surface modification could be achieved. Under good conditioning, the problems of high ash content and low coal recovery rate would be solved. From the high efficient slime water slurry blending, the paper introduced the study progress of the mixing technology and theory for the slime water slurry and introduced the application and existing problems of the available slime water slurry blending equipment. Through the discussion of present pulp-mixing technology, research status and difficulties, high efficient slime water slurry blending equipment should be developed in order to meet the requirement of the fine and difficult preparation slime flotation technique. Based on the above discussion, it pointed out that high-efficient pulp-mixing technique is a new research direction of mineral processing technology in future.

**Key words:** slimes; fines; pulp-mixing; dosage; equipment; pretreatment; flotation;

## 1.Introduction

With the improvement of mechanical utilization in coal mining, quality of raw coal become increasingly worse. In recent years, dense medium cyclone are widely used in coal preparation plant, which leads to dramatic increase of the high ash content of fine clay in flotation[1]. Flotation has good processing effect especially for -0.5mm fine coal. However, flotation is still inefficient for separation of -20um fine coal, resulting in low recovery of flotation concentrate [2]. Therefore, pulp-mixing technology should provide good initial conditions for the entire flotation process.

Pulp-mixing technology is an important part of flotation process, which not only plays role in the material mixing, but also facilitating collecting agent dispersion and adsorption on mineral particle surface. Under such conditions, collision probability and adsorption probability are improved to activate coal particle surface and enhance the coal particle surface hydrophobicity. Coal surface modification is achieved, which is helpful for flotation followed.

## 2.Research status of pulp-mixing equipment

Pulp-mixing equipments widely utilized at present are based on mechanical agitation, pretreatment of pulp-mixing is completed with the help of high speed impeller rotation which results in high dispersion of mineral particles and agent in turbulent flow field. Common facilities include mixing bucket, pulp preprocessor, pulp preparation etc. Mixing bucket was widely used in 1960s, however, there exist poor shearing intensity, adverse effect of dispersing agent, low collision probability among coal particles.During 1980s, slurry preparation device and pre-processor emerged. Although slurry preparation device can make agent dispersed, poor agitation and dreadful



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conditioning effects still exist. Currently, slurry preprocessor was widely utilized in fine coal conditioning. Due to high intensity, mineralization could be fulfilled positively accompanying with slurry, agent and air mixed that were absorbed from upper blade and lower blade separately.
Superficial quality modifier developed by Mitsui Engineering & Shipbuilding Co.Ltd. possessed perfect application in Chinese coal processing plant. Slurry and agent were inputted to superficial quality modifier as it worked, excellent dispersion and emulsion could be done because of high shear.New surface were exposed in hydrophobic coal particles, at the same time, high ash slime adhered to the coal particle surface fell out, which promoted adsorption between reagents and coal. But in industrialized application, shortcomings still exist, such as huge volume, inconvenient maintenance, high energy consumption. Accordingly, the equipment need to be further improved.

All the time, agitation mechanism of conditioning equipment is mainly based on single-stage layer.Recently, multi-stage conditioning equipment has gradually become a hot research topic, and promising improvement has been achieved, but the shortage of this kind of equipment is the high energy consumption. Multi-chamber surface reforming machine[3] developed by MA Li-qiang was applied in coal plant and excellent results were acquired[4]. Each chambers were separated by circula mixing plate fixed on the mixing barrel wall and a shear disc fixed on the stirring shaft. Two-stage compulsory stirred pulp-mixing device[5] developed by GUI Xia-hui can make slurry forced restructuring with mechanical fluid force through the two layers of leaves fixed on different axial positions. HUANG Gen[6] proposed a new stirring structure which is consist of double shafts and impellers. Through this way, the fluctuating velocity and frequency in the tank can be further enhanced. Due to the flow from the lower layer moving towards upper, and at the same time the flow from the upper impeller moving towards lower, consequently, the collision of two flows took place, so that it can promote the collision between the particles and the ore, as well as the ore particle and pharmaceutical. LI Zhen[7] developed a multi segment forced mixing plant, whose main elements included a multi-stage impeller, static guide vane device and drainage collection mechanism. Pulp homogenized, material dispersion and activation can be achieved by circulation and building a strong shear unit in order to boost the recovery of fine or micro-fine minerals.

## 3.Research status of conditioning theory

### 3.1 Solid-solid collision effect

A particle collision model was established by German two phase flow scholar named Sommerfeld[8]. Particle collision probability is positively related to particle concentration, particle size and relative velocity according to the formula of particle collision probability. Total impact probability will increases after particles dispersion. That is to say, in the process of mixing slurry, increasing shear stress and improving the collecting agent dispersion degree is conducive to trapping agent and slime particle binding, especially when catching received agent oil droplet size is large, dispersion can increase the collision probability.

Pulp-mixing is a process constant dispersion of collector accompanying with adsorption and desorption of slime particles. The effective adsorption is not desorbed before flotation phase. Effective adsorption probability $p_e$ in the actual flow field is as following.

$$p_e = p_o - p_r - p_d$$

In the formula, $p_o$ is the particle collision probability; $p_r$ is the particle flow probability; $p_d$ is the particle collision probability.

Effective adsorption between the slime particles and collector depends on the relationship among the theoretical collision probability, flow probability and desorption probability. Therefore, the key to ensure the quality and recovery of flotation concentrate is enhancing the pulp strength in the process of pulp-mixing. But due to the influence of flow and desorption probability, excessive mixing is unfavorable to improve the yield of clean coal.

**3.2 Solid-liquid suspension theory**

The purpose of mixing is to make collector agent adsorbed on the coal particle surface, and the adsorption effect depends on the suspension characteristics of solid-liquid. LI Zhen[9] simulated the practical pulp-mixing process by solid-liquid two-phase experiments. In the experiments, the parameters and concepts of cumulative concentration variance, interval concentration variance, just-suspended capacity and effective range were summarized to characterize the suspension characteristics of materiel, with which systematic analysis on the effect of typical axial flow field and radial flow field's role in two-phase medium was made. It is pointed out that material suspension is a process decided by just-suspended capacity and effect range. Under the same operation condition, suspension capacity of stirred system is decided by just-suspended capacity when the suspension level is low, while effect range gradually becomes dominant when good just-suspended status has been achieved.

**3.3 Liquid-liquid dispersion theory**

Under the action of the slurry conditioning, the collector can be dispersed into the turbulent liquid-liquid system. Experiments concerning turbulent liquid-liquid system was conducted by YU Shui-bo[10]. In turbulent liquid-liquid system, under the influence of the turbulent force, density, viscosity of the continuous phase and the dispersed phase, energy input of the turbulence, the size of the dispersed phase droplets will be different. The larger the energy input of the system, the smaller the particle size of the dispersed phase will be formed.

**4. Parameters of slurry conditioning**

**4.1 Conditioning time**

Time length has great influence on conditioning effect which can determine the stability of foam layer. Increasing the mixing time can improve the concentrate grade to a certain extent, but it is not obvious to increase the recovery rate. However, increasing the time of the mixing time can decrease the stability of the foam, resulting in the decrease of the solid content in the concentrate. The best time length was affected by pulp concentration, particle size, natural mineral properties, which should be adjusted depending on experimental regulation.

**4.2 Conditioning intensity**

The conditioning effect is largely determined by the mixing speed. Increasing the stirring speed could improve the shear rate, which would make cluster particle mass broken and increase collision probability. At the same time, the agent will be spread in the new surface[11]. Under the condition of weak intensity, with the increase of the mixing intensity, affairs will improve including collector agent dispersion, collision probability between oil droplets and coal particles, concentrate recovery.

The shear strength of the flow field is determined by the input energy. In recent years, multi-axis and multi-axial impeller are the new conditioning equipment. Compared with traditional type of single-axis and single-shaft, more energy need to consume. But the excessive increase of energy input can bring about collector agent desorption that have been absorbed in particles surface under strong shear. Moreover, effective adsorption probability decreased, resulting in the concentrate yield declining and deterioration of flotation effect.

**4.3 Impeller and addition of common accessories**

The impeller is the main device that lead the characteristics of the flow field in the conditioning process. According to the flow direction, there are two kinds of impeller, axial flow and radial flow. Basing on pulp-mixing effect, axial flow impeller are widely applied in the slime slurry[12], such as folding blade type, hinge open type and push type.

Addition of common accessories are generally added in conditioning equipment, which can improve the flow field. The baffle can make the fluid flow, change the direction of the fluid, and lead to suspension. The draft tube can guide the flow, improve the swirl velocity and make pulp cycle, enhance dispersion and mixing process.

**5.Prospects and suggestions for the conditioning research**

Compared with the new and large equipment developed in coal preparation plant in recent years, the development of coal slurry pretreatment device is extremely backward in China. The existing conditioning equipment has been far from meeting the needs of production. As for the research in the future, suggestions and recommendations are as following.

(1)Simulate the flow field characteristics of the impeller, optimize the structure of the impeller to reduce energy consumption.

(2)Improve the various parts of conditioning process, including agent dispersion, particle collision, effective adsorption and mechanism of surface modification. Perfect the evaluation system of pulp-mixing effect.

(3) Solve the problem of "effective contact" of coal particle and agent in the condition of strong shear flow.

(4)Research and develop new type of industrial conditioning equipment, demonstrate and test the existing equipment configuration, Transform theoretical ideas into practical productive forces.

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# Three-Product Teetered Bed Separator——

# A New Design For Coal Slime Separation

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**Abstract:** Three-product teetered bed separator, labeling as **TPS,** as an upgraded device, is emerging to get rid of an awkward facts thoroughly that traditional TBS could only produce two products, overflow and underflow. This paper introduces the structure characteristics of TPS with type I and type II, and its four possible applications. On a basis of actual separation performance, it's demonstrated that two-stage separation is necessary when treating coal slime. The float-sink analysis shows that TPS has the similar separation performance as heavy medium cyclone when treating easy-to-clean coal with -1.0mm size. Combining the washability of the feed, when clean coal with ash 8.56% is produced, theoretical separation density is about 1.95 g/cm$^3$ and theoretical recovery is 85% from washability curve while actual recovery is 82.54% and separation efficiency goes to 97.11%. The disadvantage of traditional TBS could be exaggerated when treating difficult-to-clean coal due to only producing two products. The birth of TPS provides a better way of solving the problem.

**Keywords:** Teetered bed separator, Coal slime separation, TBS, Three products, TPS, Float-sink, Sizing analysis, Coal washery

## 1 Application status of traditional TBS

Traditional teetered bed separator (TBS) is one major device of dealing with coarse slime nowadays, with a great appreciation of only circulating water used as power source without other medium, low investment, low operating cost, perfect automation control and easy layout[4].

Coal slime could be only separated into two products at a time using traditional TBS; when to-be-treated material contains more than 20% middle density fraction, the separation performance gets lower with high clean coal loss in underflow. These mentioned disadvantages limited the application of TBS[5]. Meanwhile, due to effect of equal settling particles, when the feed has wide size fraction, mineral particle with high density and small size goes into overflow, which leads to stringent requirements of the following dewatering; sometimes it will determines the total separation performance of coarse slime separation[1].

Even so, TBS is still irreplaceable nowadays due to its advantages and has widely used to tackle raw coal slime[6], magnetic separation tailings, slime of jig clean coal and so on. The main advantages could be concluded into the following two aspects,

1) With the use of TBS, the flowsheet of "first classifying by size and then separating by density" is achieved to reduce the effect of size on the density-based separation to maximum extent, thus improving the total separation efficiency of whole size material. The flowsheet that the whole size raw coal without classification is separated using jig or heavy medium separation technique accounts for most in coal preparation plant in China before 2008[7]. Except some plants with high quality coal, under normal circumstance, slime of jig clean coal or magnetic separation tailings of clean coal has more ash content than total clean coal, average 2%, in worse case as high as 10%[2], which is called "carrying ash" phenomena. Reconstruction of these preparation plants with original technique is usually carried out with the use of TBS, or slime of jig clean coal and magnetic separation tailings is separated again using TBS, removing +1.6 kg/l fraction and eliminating "carrying ash" phenomena, thus improving clean coal quality

 
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and total benefit. In another case, pre-desliming of raw coal before separation is performed to achieve jig separation or heavy medium separation without coal slime. No matter which form is adapted, coarse slime separation is carried out by itself[8].

2) The flowsheet of desliming of raw coal + TBS separation of raw coarse slime+ heavy medium separation has been widely used in a lot of new-built separation plants[9]. Comparing with original flowsheet, the new one has a great appreciation due to high magnetic material content and low viscosity in qualified medium, low separation size and obvious low medium loss with the same equipment of recycling medium.

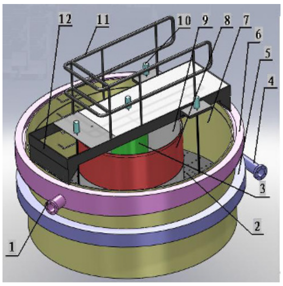
**2 Introduction of three-product TBS**

The parameters affecting separation performance of TBS could be summarized into four aspects, that's, washability of coal, size range of feed, dewatering and desliming performance of the following procedures and performance of the device[10]. Due to its intrinsic property of producing two products, when treating +1mm material with all the same four parameters above, TBS has lower separation efficiency than heavy medium cyclone, with the least probable error value (Ep) 0.071. Therefore, it's important to develop coarse slime separator which utilizes all kinds of advantages and produces three or more products, leading to the perfection of the "irreplaceable technology at present" to maximum extent. Under the circumstance, three-product teetered bed separator comes into being.

Three-product or more-product teetered bed separator, developed on a basis of traditional TBS technology, is authorized to be manufactured by national invention patent with complete intellectual property, and is also called three products separator, labeling as **TPS**.

Two kinds of TPS were invented, type I and type II.

TPS with type I is comprised of two or more barrels with the same central axis [3]. Taking two barrels as an example, it's named inner barrel used for first stage of separation and outer barrel used for second stage. Feed goes into inner barrel via feed well, and the first upwards water is moving from the bottom of inner barrel, thus producing the first teetered bed layer with material from feed well. High density particle settles through the first teetered bed layer and discharged from the outlet at the bottom of inner barrel as product (1), and the light particle goes upwards into the outer barrel. Meanwhile, the second upwards water is going up from the bottom of outer barrel and mixing with the particles in overflow of inner barrel and forming the second teetered bed layer. The heavy particle sinks and discharges from the outlet of outer barrel as product (2), and the light particle goes up and concentrates as product (3). After separation of first stage and second stage, three products are obtained, that's, underflow of first stage, underflow of second stage and overflow of second stage, which is illustrated as Figure 1.




Notes: 1. Overflow tube; 2. Water spray nozzle; 3. Feed well; 4. Outer water feeding tube; 5. Outer water feeding circular belt; 6. Overflow box; 7. Outer barrel; 8. Guide barrel; 9. Inner barrel; 10. Discharge actuator; 11. Handrail; 12 Platform;

Figure 1 Structure diagram of TPS with type I

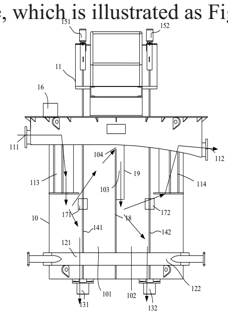



Figure 2 Structure diagram of TPS with type II

TPS with type II is comprised of two or more chambers from left to right. Taking two chambers as an example, it's named left chamber used for first stage of separation and right chamber used for second stage. The overflow of first stage is used as the feed of second stage, forming three-product teetered bed separator with two chambers in one barrel. Separation barrel could be circular barrel or square barrel. The first chamber totally separates from second chamber, and every chamber contains one or more discharge outlets at the bottom. The to-be-treated material enters first stage and is separated into two products, that's, overflow of first stage and underflow of first stage. Underflow of first stage discharges from outlet at the bottom of first stage and concentrates as product (1), and overflow goes into second stage to be treated as feed. With the interaction of upwards current and gravity of particle, the feed can be separated into two products. Underflow of second stage could be final product alone, and also could mix with the underflow of first stage as one product while overflow of second stage becomes final product. After two-stage separation, three products are obtained, that's, underflow of first stage, underflow of second stage and overflow of second stage, which is shown as Figure 2.

**3 Possible applications of TPS**

TPS is upgraded from traditional TBS and could be adjusted flexibly according to raw characteristics and product quality requirements. In summary, there are four possible applications as followed.

1) Technique of combining underflows of two stages

This technique is characterized of overflow being clean coal and underflows of first and second stage being middlings or gangue. In other words, two-stage separation is achieved using traditional TBS to improve separation efficiency, which is suitable for easy-to-separate coal or low requirement of thermal value of middlings, shown as Figure 3.

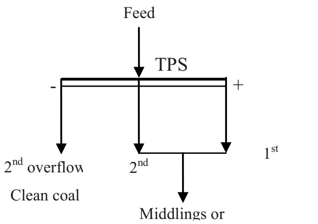



Figure 3 Technique of combining two

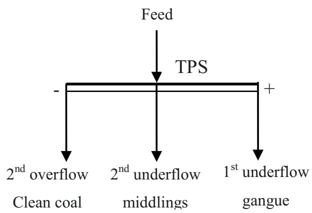



Figure 4 Technique of three products

2) Technique of three products

This technique is characterized of 1st underflow being gangue, 2nd underflow being middlings and 2nd overflow being clean coal. For coarse slime of difficult-to-clean coal, this produces three products, avoiding loss of middlings in gangue or contamination of gangue in middlings when using traditional TBS, suitable for easy-to-clean coal, difficult-to-clean coal or high requirement of thermal value of middlings, seen as Figure 4.

3) Technique of desliming of 2nd overflow

This technique is characterized of 1st underflow being gangue, 2nd underflow being clean coal and 2nd overflow going to flotation or concentrator. For easy-to-clean coal but with high content of -0.074mm fine slime, this technique is adapted to reduce dewatering pressure of clean coal slime extensively and to reduce slime content in dewatered TPS clean coal, thus eliminating "carrying ash" phenomena, which is shown as Figure 5.

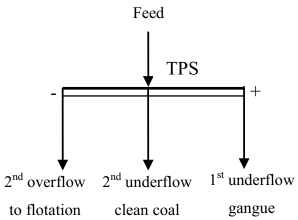



Figure 5 Technique of desliming of 2nd overflow

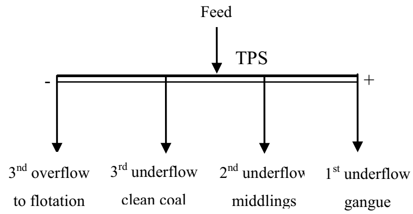



Figure 6 Technique of desliming of 3rd overflow

4) Technique of desliming of 3rd overflow

This technique is characterized of three stages and four products. It is achieved that 1st underflow is used as gangue, and 2nd underflow is middlings and 3rd underflow is clean coal while 3rd overflow is going to floatation or concentrator. This contains advantages of above three techniques and is suitable for difficult-to-clean coal, extreme difficult-to-clean coal or case of high fine slime content like Figure 6.

## 4 Application Effects of TPS

### 4.1 Size-based analysis

One coal preparation in Shanxi province, adapting the flowsheet of "raw coal desliming + three-product heavy medium cyclone separation + TPS + flotation", and one three-product teetered bed separator, TPS-3.0, is chosen to treat the underflow of classifying cyclone which is used for dealing with raw coal slime. Samples of Clean coal comes from centrifuge while middlings and gangue are collected from vibrating screen with high efficiency. The screening results of feed, clean coal, middlings and gangue are shown as Table 1-2.

Table 1 Sizing results of Feed and Clean coal of TPS

| Size, mm | Feed of TPS | | | | Clean coal of TPS | | | |
|---|---|---|---|---|---|---|---|---|
| | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% |
| +1.00 | 1.5 | 23.62 | 1.5 | 23.62 | 1.3 | 5.86 | 1.3 | 5.86 |
| 1.0-0.5 | 49.19 | 15.22 | 50.7 | 15.47 | 60.79 | 6.49 | 62.09 | 6.48 |
| 0.5-0.25 | 33.23 | 24.45 | 83.92 | 19.02 | 34.5 | 13.44 | 96.59 | 8.96 |
| 0.25-0.125 | 11.06 | 43.78 | 94.99 | 21.91 | 3.02 | 34.94 | 99.61 | 9.75 |
| 0.125-0.075 | 2.49 | 41.87 | 97.47 | 22.42 | 0.2 | 25.81 | 99.81 | 9.78 |
| -0.075 | 2.53 | 28.06 | 100 | 22.56 | 0.19 | 13.76 | 100 | 9.79 |
| Sum. | 100 | | | | 100 | | | |

Table 2 Sizing results of middlings and gangue of TPS

| Size, mm | Middlings of TPS | | | | Gangue of TPS | | | |
|---|---|---|---|---|---|---|---|---|
| | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% |
| +1.00 | 1.95 | 63.06 | 1.95 | 63.06 | 4.27 | 78.62 | 4.27 | 78.62 |
| 1.0-0.5 | 36.07 | 75.81 | 38.02 | 75.16 | 66.34 | 84.06 | 70.6 | 83.73 |
| 0.5-0.25 | 52.39 | 83.74 | 90.41 | 80.13 | 28.63 | 86.72 | 99.23 | 84.59 |
| 0.25-0.125 | 9.25 | 87.52 | 99.66 | 80.82 | 0.67 | 75.8 | 99.9 | 84.53 |
| 0.125-0.075 | 0.24 | 76.92 | 99.89 | 80.81 | 0.04 | 80.95 | 99.94 | 84.53 |
| -0.075 | 0.11 | 77.2 | 100 | 80.8 | 0.06 | 80.49 | 100 | 84.53 |
| Sum. | 100 | | | | 100 | | | |

It’s demonstrated from Table 1 that the feed of TPS contains seldom coal slime with -0.25mm material, 16%, indicating good control ability of feed size using classifying cyclone. As shown from Table 1, 1-0.25mm accounts for most of clean coal with yield 96.59% while the yield of -0.25mm fraction is 3.41%, demonstrating good performance of desliming.

It’s concluded from the screening results in this case,

1) The dewatered materials are chosen to be used for analyzing rather than overflow and underflow of TPS, reduced the influence of fine slime, thus achieving representative results.

2) Obvious improvement of upward water velocity and set separation density are achieved, more suitable for coal sample in the case. There is not obvious difference in ash of underflows of 1st stage and 2nd stage, and TPS could be used for technique of removing gangue by two-stage separation.

3) proper condition provided for teetered bed separation.

**4.2 Density-based analysis**

The feed, clean coal, middlings and gangues are the same materials as in 4.1, and the float-sink results of these materials are described as Table 3-4.

Table 3 Float-sink results of feed and clean coal of TPS

| | Feed of TPS | | | | Clean coal of TPS | | | |
|---|---|---|---|---|---|---|---|---|
| Density, g/cm3 | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% |
| -1.3 | 28.98 | 1.96 | 28.98 | 1.96 | 29.45 | 1.54 | 29.45 | 1.54 |
| 1.30-1.40 | 33.58 | 5.62 | 62.55 | 3.92 | 50.38 | 5.18 | 79.83 | 3.84 |
| 1.40-1.50 | 7.93 | 15.26 | 70.48 | 5.2 | 9.71 | 15.05 | 89.54 | 5.05 |
| 1.50-1.60 | 4.85 | 26.3 | 75.33 | 6.56 | 3.19 | 27.24 | 92.74 | 5.82 |
| 1.60-1.70 | 3.12 | 36.18 | 78.45 | 7.74 | 7.26 | 58.32 | 100 | 9.63 |
| 1.70-1.80 | 1.82 | 44.1 | 80.27 | 8.56 | 0 | 0 | 0 | 0 |
| +1.8 | 19.73 | 78.05 | 100 | 22.27 | 0 | 0 | 0 | 0 |
| Sum. | 100 | 22.27 | | | 100 | 9.63 | | |

It’s stated clearly that the ash of -1.80 g/cm$^3$ fraction only accounts for 8.56% in the feed; separation density will be set as 1.80 when clean coal with ash 8.56% is required. Therefore, the washability grade of coal is classified as extreme easy-to-clean, and basically no middlings could be produced.

Table 4 Float-sink results of middlings and gangue of TPS

| | Middlings of TPS | | | | Gangue of TPS | | | |
|---|---|---|---|---|---|---|---|---|
| Density, g/cm3 | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% | Yield,% | Ash,% | Cum. Y,% | Cum. Ash,% |
| -1.3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| 1.30-1.40 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| 1.40-1.50 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| 1.50-1.60 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| 1.60-1.70 | 0.8 | 36.47 | 0.8 | 36.47 | 0 | 0 | 0 | 0 |
| 1.70-1.80 | 1.83 | 45.62 | 2.63 | 42.85 | 0.14 | 46.84 | 0.14 | 46.84 |
| +1.8 | 97.37 | 81.6 | 100 | 80.58 | 99.86 | 84.44 | 100 | 84.39 |
| Sum. | 100 | 80.58 | | | 100 | 84.39 | | |

From the density composition of clean coal shown as in Table 3, as +1.7g/cm$^3$ fraction are all zero, the actual separation density will be not greater than 1.7g/cm$^3$. On a basis of float-sink results of feed, when the ash of clean coal is 8.56%, the separation density will be greater than 1.8 g/cm$^3$. The error is caused by two possible reasons, 1) experiment error, especially sampling error; 2) polluting clean coal by fine slime with high ash which is seldom removed.

As seen from table 4, there are no -1.6g/cm$^3$ fractions in middlings and gangue, indicating that TPS has the similar separation performance as heavy medium cyclone when treating easy-to-clean coal with -1.0mm. Combining the washability of the feed, when clean coal with ash 8.56% is produced, theoretical separation density is about 1.95 g/cm3 and theoretical recovery is 85% from washability curve while actual recovery is 82.54% and organic efficiency is 97.11%.

As discussed above, TPS has good separation performance in this case and the possible reasons could be summarized into two aspects: 1) qualified clean coal could be produced through removing gangue easily for easy-to-clean coal; 2) low slime content and good desliming performance prevent clean coal from polluting by fine slime with high ash, which is a key parameter of good operation of TPS also.

## 5 Conclusions and Suggestions

The disadvantage of traditional TBS could be exaggerated when treating difficult-to-clean coal due to only producing two products. The birth of three-product teetered bed separator (TPS) provides a better way of solving the problem. The following conclusions could be attained from the application of TPS in coal preparation plant, and some suggestions are proposed.

1) There are obvious difference in 1$^{st}$ overflow ash and 2$^{nd}$ overflow ash of TPS, indicating the necessity of two-stage separation which is important for improving the separation efficiency.

2) TPS, treating easy-to-clean coal, has the similar separation performance as heavy medium separator treating coarse particles, with small loss of coal in gangue and/or middlings.

3) Small size range of feed for TPS is required; superior limit of the feed should be lower than 1mm when treating raw coal slime.

4) More attention should be paid to four factors affecting the teetered bed separation.

5) Without actual application results of extreme difficult-to-clean coal, more researches about them need focusing on and performing.

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# Technical and economic aspects of granulation of coal

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**Abstract**

Mining and processing as well as handling of coal generate ultra-fine fractions of coal worldwide. The transport of this coal fraction is difficult, in particular the tendency to dust explosion attracts major technical and environmental challenges. Agglomeration of coal dust is a potential way to facilitate safe transportation and economic use of the material.

The main objective of agglomeration is to convert fine-grained materials into coarser products, so called agglomerates, pellets or granules. These products exhibit considerably improved properties with regard to transport, processing and use (Pietsch, 2005).

One possibility of agglomeration is the use of disc granulators. The ultra-fine coal will be energy efficiently formed into agglomerates by the addition of liquid binders, usually water, in an inclined rotating process chamber. Due to the special movement of the material in the disc granulator a narrow grain size distribution of the product can be reached. This special movement of the material is caused by a set of selected parameters, such as inclination angle, rotational speed, dimension of the disc, moisture content and the material behavior itself.

This paper presents experimental results of coal agglomeration in disc granulators. Exemplary flow-sheets are presented and the required investment and operating costs will be discussed.

**Key words**

Agglomeration, Pelletization, Granulation, Coal, Coal dust, Coal emissions; Powder, Efficiency

 
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**Background**

Mining, processing and handling processes of coal often lead to undesired fine grained particles (Höffl, 1986). The mitigation of risks such as dust explosions and environmental pollution, health issues for humans and animals require to fight these dusts (HdA-Transfer, 1983). Dust control requires individual action to minimize or eliminate the emissions. It is assumed that the restrictions on the emissions continue to rise in the future.

Figure 1: Generation of coal dust while stacking coal in an export terminal (Lieberwirth & Lampke, 2013).

An economically sensible use of coal dust is therefore the focus of this investigation. Current methods of dust control include the precipitation with water, occasionally including chemical additives, and the dedusting by filter systems. The moisture content of coal increases due to precipitation with water. The mass of water can therefore significantly reduce the specific calorific value of the transported coal. Other disadvantages of wet coal are pointed out by Taulbee et al (Taulbee, Hodgen, & Patil, 2013).

When using dedusting by filter systems the extracted dust fraction is commonly recirculated to the lumpy coal fraction. The problem of dust is thus solved only temporarily and passed on to subsequent process steps. A reasonable alternative or supplement is the agglomeration of the coal dust. Its main objective of is to transform fine-grained material into agglomerates (granules or pellets). These exhibit considerably improved properties for transport, processing and use (Table 1) compared to the dust. In particular, the use of a disc granulator can lead to an energy-efficient agglomeration of the coal dust. The agglomerates produced exhibit a significantly improved flow and transport behavior. Recirculating of these agglomerates to the lumpy coal fraction will reduce dust emissions in the following process steps. Since the similar sized pellets exhibit excellent permeability features and, respectively, excellent burning characteristics, they may even be sold as a high quality fuel. Finally, the market value of the coal formerly contained in the dust is increased by agglomeration.

**General information to Granulation**

Crucial for the formation of agglomerates are forces of attraction between the particles. During agglomeration in the disc granulator, attraction is based mainly on capillary bonding forces caused by particle collisions. The necessary relative movements in the material bed are generated in a tilted, rotating, flat cylindrical vessel (Fig. 2). Depending on the agglomeration requirements, the fine-grained feed material is fed in an exposed position into the interior of the process chamber. Owing to the rotation of the vessel, the fines are swept along as a function of the coefficient of entrainment, the speed and the pitch to the highest position of the disc and then roll down onto the material bed (Schubert, 2002).

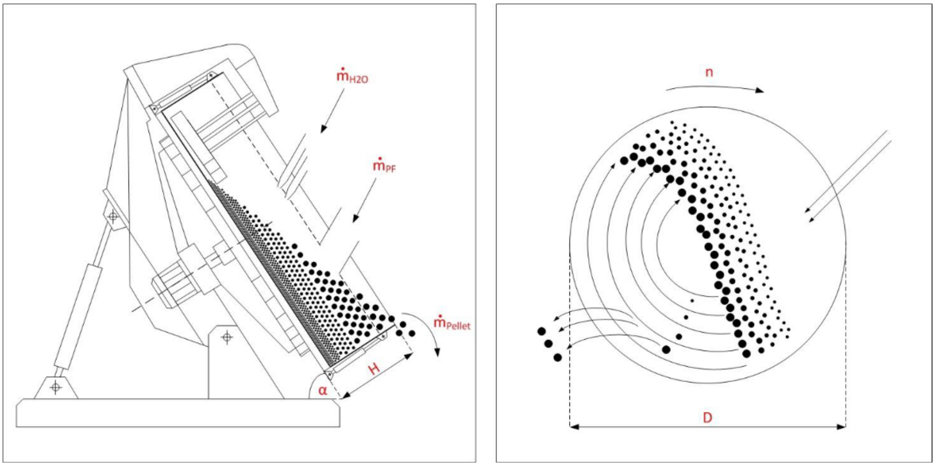


Figure 2: Disc granulator with segregation effect (Lampke, New Possibilities to Influence Pellet Properties by Photo-optical Particle Analyses, 2013).

The material is sprayed with binder, usually water, to produce the adhesive conditions. In this wet material bed, liquid bridges are formed as a result of irregular, random particle contacts, leading to the aggregation of individual particles. These seeds are transported upwards on the fines in the material bed (Metzger, Remy, & Glasser, 2011), and other particles and granule fragments attach themselves to the seeds. On account of the random particle movement, separating forces such as abrasion, breakage or disintegration of the granules also take effect. For this reason, feed material, seeds and granules as well as their fragments and abraded material are all contained in the material bed of the granulating bed at the same time.

If the granules formed have the required size in line with the selected settings, they are discharged over the edge of the disc as a result of the segregation effect. Owing to this segregation effect, in contrast to the granulating drum, a very narrow granule size distribution is obtained in the discharge. On account of the continuous rolling process, mainly spherical agglomerates with a homogeneous green strength are formed (Lampke, Messerschmidt, Folgner, & Lieberwirth, 2015).

Table 1: Arguments for agglomeration and typical applications (Lampke, Weyrauch, & Silge, Firm and compact, 2011)

| Arguments for agglomeration | Typical applications |
|---|---|
| • Reduced specific surface area | • Hazardous substances |
| • Defined shape value | • Pharmaceuticals |
| • Improved properties for transport, processing and use | • Coal |
| • Good flowability and spreadability | • Fertilizers and soil conditioners |
| • Warranty of zero dust | • Detergents, fertilizers and coal |
| • Improved permeability | • Charge for pig iron production |
| • Avoidance of segregation | • Feedstuffs |
| • Production of narrow particle size distributions | • Iron ore concentrate |
| • Production of set particle sizes and shapes | • Pharmaceuticals |
| • Sustained-release action | • Fertilizers, crop protection agents and medicines |
| • Improvement of dispersion and/or dissolution behavior | • Inertization of foods, beverages, and tobacco |

Most binders evolve their performance while drying and / or curing of the granules. Therefore, a thermal treatment of the green granules is recommendable.

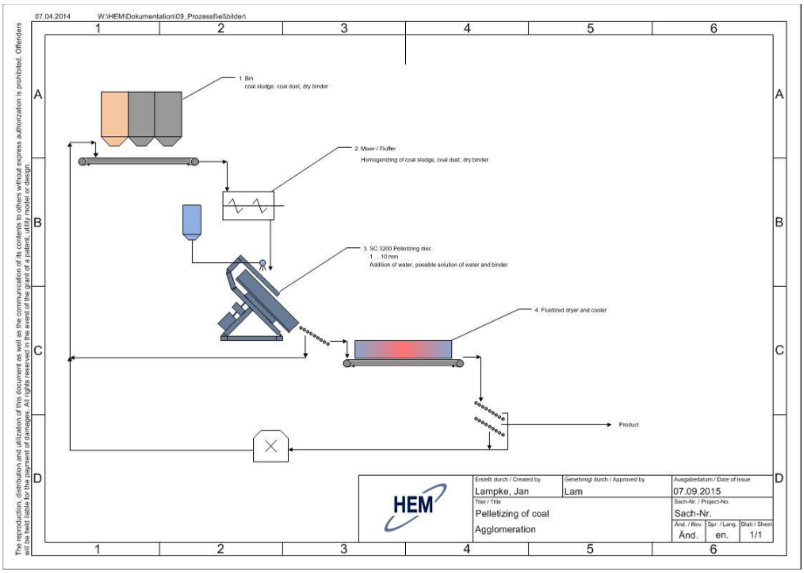

Figure 3: Process flow sheet for granulation of coal.

Fig. 3 shows a potential flow sheet of the agglomeration plant. The main components of the system are the mixer, the granulating disc and the drying/cooling system, incl. exhaust air filtration.

Subsequent trials of coal fines granulation show the potential of this technology.

**Granulation of coal – lab trials**

Homogenizing of the materials and the binders is done by use of an intensive lab mixer. Granulation of the homogenized mixture is done using a lab disc granulator with a diameter of 400 mm. The rim height of the disc is fixed at 100 mm, the inclination and the rotation speed, however, are variable. The mixed material is fed to the disc discontinuously. If necessary, the material is wetted by a spray during feeding. The granules are discharged in a narrow grain size distribution over the rim of the disc. Moisture, green strength, drop number, bulk density and size distribution of a sample of the granules is measured regarding. Another sample is dried in a drying cabinet at 60 °C.

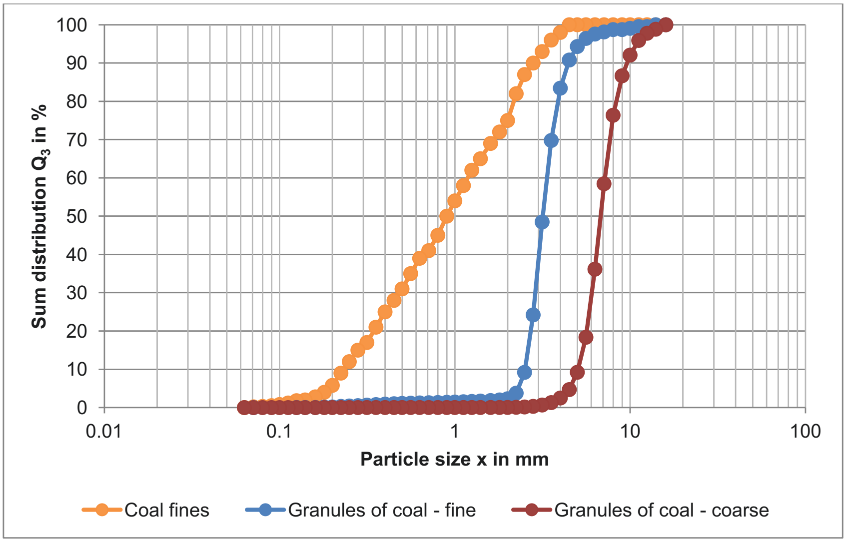


Figure 4: Particle size distribution of coal fines and granules.

Fig. 4 shows the cumulative particle size distribution curve of an exemplary coal and two granule size distributions of exemplary products. While common understanding is, that only materials much finer than the filtered coal dust are suitable for this type of agglomeration, it is conspicuous that even relatively coarse feed materials can be agglomerated. Depending on the binder, cured granule strength of more than 50 N/Granules, in some cases up to 100 N/Granule could be proven.

The produced coal granules exhibit a regular shape (Fig. 5 and 6) which is good to handle in down stream processes or to even to sell as high quality fuel.

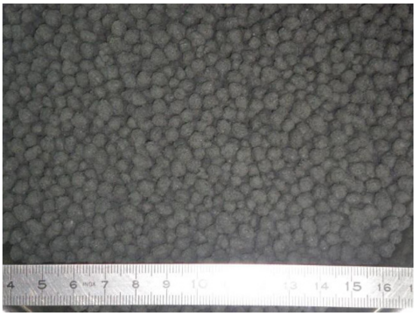

Figure 5: Granules of coal – fine.

Figure 6: Granules of coal - coarse

**Conclusion**

Granulation of coal fines help to minimize the dust emissions while handling coal. A potentially hazardous waste can be transformed into a valuable product. Energy consumption for handling the same coal dust at every consecutive transfer point can be reduced.

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# PERSPECTIVES OF USE OF TECHNOGENIC RAW MATERIALS OF THE METALLURGICAL ENTERPRISES AS WEIGHTING COMPOUNDS OF MINERAL SUSPENSIONS

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**Abstract**
Industrial wastes resulted from production activity of mining, coal and metallurgical enterprises in the Ukraine make considerable amounts. Waste disposal issues became the economic and environmental problems. Demand for solution of environmental problems combined with the urgent need to find additional sources of cheap dense solids for heavy-density media used in coal preparation gives rise to interest into the recycling of technogenic materials from metallurgical enterprises. Metallurgical production wastes were used as research subject. Electric furnace gas treatment slurries at the Krivoy Rog Central Mining Equipment Repair Plant were demonstrated to be polygenic formation of very complex mineral and petrography composition, which has majority of particles consisting of multiple crystalline or amorphous phases of various chemical composition and physical properties. It is found that dry magnetic separation of electric furnace gas treatment slurries from the Krivoy Rog Central Mining Equipment Repair Plant allows separation of high-magnetic iron-bearing product with density of over 5.2 t/m$^3$. Iron recovery to the magnetic product made 73-75%. Conducted researches resulted in solution of the problem of dense solids production from the technogenic material of metallurgical enterprises, which meet the following requirements: water insolubility; mechanical abrasion resistance; chemical non-reactivity with water and products of separation; sufficiently fine size; relatively low cost.
**Keywords**
SLURRY, DENSE SOLID, MAGNETIC PRODUCT, CYCLONE, TECHNOGENIC MATERIAL

**Background**
Industrial wastes resulted from production activity of considerable number of mining, coal metallurgical, chemical, machine-building and other industry enterprises in the Ukraine make considerable amounts enterprises. In some country's regions and districts, industrial wastes accumulation reaches such a huge amount, which makes disposal issues the economic and environmental problems. It is primarily true for the Donbass, Krivbass, Dnieper region and Carpathian region, where technogenic burden level achieves 2-10 million tons and over per 1 sq. km of area in some localities. For example, in the Krivbass, annual economic loss due to environmental pollution is estimated at 300 million USD based

 
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on the incomplete data. Potential derivative impacts are even more hardly predictable. During the period of commercial development of the Krivbass iron ore deposits, more than 25 thousand ha of lands have been withdrawn from agricultural enterprises for mining and metallurgical production sites. Dumps in the Krivbass contain more than one billion $m^3$ of rock material, and tailing pounds contain 3 billion tons of cleaning rejects [1,2].

Demand for solution of environmental problems combined with the urgent need to find additional sources of cheap dense solids for heavy-density media used in coal preparation gives rise to interest into the recycling of technogenic materials from metallurgical enterprises both in the Ukraine and worldwide.

Up to the present time, coal industry used magnetite concentrates from South Mining and Processing Integrated Plant (Krivoy Rog Basin) for preparation of heavy-density media. These magnetites were chosen as early as in 1960s at the beginning of the heavy-medium separation development in our country based on the best world practice and in consultation with professionals of the "Veno-pick" Company, France. Principal requirement to dense solid was sufficiently high magnetic properties providing its regeneration by means of magnetic separators. Magnetite concentrate density makes 4400— 4600kg/$m^3$. However, quality of the magnetite concentrates supplied to the coal industry considerably changed over the past years due to improvement of grinding and separation process layouts at the GOK. Substantial increase in the fine sizes content in magnetite concentrate and respective reduction of average grain size resulted in the magnetite loss increase during the magnetic regeneration of heavy-density medium and with the separation products. Therefore, study of potential for enhancement of efficiency and expansion of heavy-medium separation application area due to the finding of dense solids not used previously and possessing specific physical and mechanical properties is the problem of important applied significance. In particular, one approach to this problem solution is production of concentrates for dense solids preparation from metallurgical wastes.

We will consider waste generation conditions at the metallurgical treatment of iron-bearing material.

**Waste generation conditions at the metallurgical treatment of iron-bearing material.**

Over 80 million tons of iron and steel industry slag is dumped annually, of which blast-furnace slag makes about 45-50 million tons. Blast furnace slag is formed by the gangue minerals contained in iron-bearing burden material, fluxes and coke ash. Average specific yield of slag makes approximately 320 – 800 kg per ton of cast-iron, and cast-iron loss with slag makes 1–4% of slag weight. Approximate chemical composition of blast-furnace slag is as follows (%): $SiO_2$ – 30–40; CaO – 30–50; $A1_2O_3$ – 4–20; MnO – 0.5–2; FeO – 0.1–2; $SO_3$ – 0.4–2.5, etc.

Steelmaking processes differ in the process conditions, methods of working the heat and melting furnace types. Presently, primary processes are main Martin scrap and scrap-ore processing (about 40% of total production), basic oxygen process (≈40%) and steel melting in the arc and induction furnaces. Average yield of steelmaking slag makes 150–160kg per ton of steel. At the average annual steel production of 170 million tons, approximate yield of steelmaking slag would make about 25.5–27 million tons including approximately 10 – 12 million tons of Martin slag and the same amount of basic-oxygen slag. Basic steelmaking slag types have the following chemical composition (%): melting of low-carbon steel grades — $\sum$(CaO + MnO + MgO) – 60% and $\sum[SiO_2 + P_2O_5 + Fe\ (total)]$ – 30–32%, and melting of carbon steel grades – approximately 65 and 35%, respectively. Average iron content in Martin slag – 8–20%, in basic-oxygen slag – 2–13%, electric-furnace slag – 8–17%.

The ferroalloy smelting process also generates large amount of slag (more than 5 million tons annually), chemical composition of which also relies on composition of initial raw material. Maximum amount of slag (>90% of total ferroalloy smelting slag yield) is generated at smelting of chromium and manganese ferroalloys. At the same time, basic losses of e.g. manganese in ferroalloy production are due to its incomplete reduction during melting and losses at the tapping and pouring, and metallic beads entrapped by slag account for 30% of total losses.

Dust is another source of metallurgical wastes. Approximate weight of the dusts collected in blast-furnace and steel-melting processes makes 2.5 and 1.5 tons per 100 tons of produced iron and steel, respectively. Dust is collected in the gas treatment systems by means of various dust-collecting units (electrostatic separators, wet scrubbers, filters, etc.) Considerable quantities of secondary iron-bearing material are recovered specifically from dust (dry gas cleaning) and slurries (wet gas cleaning). Specific yield of gas treatment wastes ranges within 1–3%. For example, annual steel production of about 125 million tons results in generation of about 14 million tons of iron-bearing dust. When melting low-carbon and stainless steels in electric furnaces, dust collected in filters makes 14 and 20kg per ton of steel, respectively. Over the period of converter refining, collected dust weight ranges from 8.5kg in case of gas cleaning in wet scrubbers to 20.5kg per ton of smelted low-carbon steel in case of gas cleaning in the electrostatic separators. It should be noted that dust recovered from the blast-furnace gas cleaning systems is suitable for use in sintering process without the preliminary treatment as a substitute for about 67kg of ore and 40kg of coke (100kg of dust). All metallurgical dusts vary markedly in the physical and chemical properties and grain size distribution. Table 1 shows indicative characteristics of dusts [3].

Table 1. Metallurgical dust characteristics

| Chemical composition | % | Density, $t/m^3$ | |
|---|---|---|---|
| Fe | 36.6 – 69 | Real density of fine fractions | 4.2 – 5.1 |
| Mn | 0.34 – 5 | Real density of larger fractions (>15μm) | ≈6.3 |
| CaO | 0.86 – 6.3 | Apparent density of fine fractions | 1.1 –2.2 |
| Mg | 0.08 -1.8 | Apparent density of larger fractions | 3.5 |
| S | 0.02 – 0.73 | Grain size distribution, %: | |
| P | 0.29 – 0.82 | Fractions > 15μm | 1 – 8 |
| Zn | 0.4 –18.3 | Fractions > 2μm | 42 – 80 |
| Pb | 0.06 – 2.9 | Fractions > 1μm | 74 – 95 |
| Si | 0.19 – 2.03 | | |
| C | 0.23 – 1.7 | | |

**Subject of research.**

Subject of research was electric-furnace gas treatment slurry from the Krivoy Rog Central Mining Equipment Repair Plant. Sample was collected from the bins of the gas treatment system units.

Particles are generally of oval and grainy shape. Fraction of over 15μm consists of large porous black particles, and fraction below 2μm consists of yellow, greenish-brown and yellow-brown particles of fiber and platelet shape. Table 2 shows distribution of dust particles after the rotary separation in laboratory with the use of multistage impactor. Grain size analysis showed that minus 0.025 to plus 0.0016mm fraction makes 72 % of total sample.

Table 2. Distribution of dust particles

| Particle size, mm | 0.0016 | 0.0025 | 0.004 | 0.0063 | 0.01 | 0.016 | 0.025 |
|---|---|---|---|---|---|---|---|
| yield of fraction, % | 15 | 17 | 14 | 12 | 6 | 5 | 3 |
| Suspension velocity of dust particles in air, $\times 10^{-2}$, m/s | 0.03 | 0.07 | 0.18 | 0.44 | 1.12 | 2.88 | 7.0 |

Dust mechanical properties: dust particles material density – 5980kg/m$^3$; bulk density – 762kg/m$^3$; maximum tapped density – 950kg/m$^3$; static angle of natural slip (spatula angle) – 51°; rupture strength of dust layer – 330Pa; based on the magnetic dust collection and magnetic coagulation principles [4,5], this demonstrates that high-magnetic dust particles can be separated from the dust-gas stream by providing their agglomeration conditions and recovery from the stream. Table 3 shows dust chemical composition.

Table 3. Dust chemical composition (water extract pH 7.5):

| Components | other | $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | CaO | MgO | $SO_3$ | MnO |
|---|---|---|---|---|---|---|---|---|
| Contents, % (by weight) | 37.5 | 6.2 | 1.65 | 37.8 | 8.1 | 1.2 | 0.5 | 0.15 |

In terms of mineral composition, dust material could be described as a multi-component system containing several dozens of natural and technogenic mineral phases. Technogenic phases are characterized by variety due to formation under different thermodynamics conditions of metallurgical process [6-8]. Small (from fractions of micrometer to 1mm and over in size) spherical particles are found in the sample consisting of metallic iron, wustite, magnetite, martite and other iron-bearing minerals, and spheroidal particles of silicate and iron-silicate glass, individuals of hematite and zincite. It should be noted that small metallic iron and magnetite particles flocculate actively due to high magnetic susceptibility with formation of small crumbly, chain and other segregations (fig. 1).

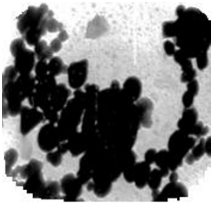

Fig.1 Magnetite nodules flocculated into isometric and chain aggregates. Electronic microscope; magnification 22000.

Floccules could persist in aged slurries for many years, which can be observed under microscope in oriented thin sections prepared from the aged slurries of dewatered sludge ponds.

Relatively coarse-grained slurry materials (particle size over 0.01 mm) are dominated by sinters of iron hydroxide (goethite, lepidocrocite) and fragments of spinel crystals and aggregates.

Material of every grain-size fraction contains graphite. Graphite crystals and aggregates are larger then nodules of other minerals. Graphite exists in two morphological types. The graphite of first type occurs as well-facetted plate crystals, hexagonal plates and tablets, and as fine-flaky foliated aggregates. Well-formed crystals prevail. The graphite of second morphological type occurs in slurries as radially-fibrous spherulite-like aggregates. It forms reniform build-ups on the surface of larger plate crystals of graphite and already existing individuals and aggregates of metallic iron, wustite, magnetite, acicular crystals of silicates and other minerals.

Quartz is the main source of silica contained in all metallurgical treatment products – it is one of components of iron ore raw materials (sinter ore, concentrate, etc.) It takes active part in all metallurgical processes during melting. It should be noted that sample material contains multiple quartz modifications at the same time. These are natural quartz, tridymite and cristobalite. They occur in dust as well-facetted or partly fused single crystals and less frequently as xenomorphic grains and fragments of crystals. Some individuals achieve 1 mm and over in size. These minerals are always accompanied by well-facetted magnetite in form of octahedral crystals and dendrites. Magnetite is formed after the cristobalite, cristobalite nodules often act as seeds at formation of magnetite individuals and aggregates, and their occurrence inside is quite common. Different thermodynamics conditions of formation processes result in variety of silica modifications. All silica minerals are of near density and have similar mechanical and magnetic properties. However, complex processes of technogenic mineral formation are responsible for measurable differences in processing characteristics of natural and “slurry” quartz. These include tendency of the latter to overgrinding due to increased fracturing, and increased values of specific magnetic susceptibility and density resulting from the presence of magnetite inclusions and crusts.

**Research findings.**

In order to produce factions suitable for the use as dense solids of heavy-density media from such raw material, authors designed layout of equipment circuit, manufactured and assembled installation with capacity of 2t/h at the dry-cleaning process laboratory of the State higher educational establishment «Krivoy Rog National University»; installation circuit diagram is shown in the Fig. 2. Its technical characteristics are as follows: air flow rate - 4000m$^3$/h; number of cyclone separators - 1 pcs.; diameter - 04m; length – 1.5m; number of magnetic cyclones - 1 pcs.; diameter – 0.4m; height -2m; rotational speed of the magnetic systems of separators and magnetic cyclones - 5-100rpm; installation dimensions – 2.5×6×5m.

Entire installation operation principle is simultaneous exposure of air dispersion medium to magnetic, aerodynamic and gravitation forces. The installation consists of feeder with hopper 1 for feeding of material subject to separation to the horizontal air classifier 2; material can also be fed directly from dust-exhaust system hoods in air stream induced by fan 11. In classifier 2, particles carried by air stream are subjected to two forces: gravity and resistance. Heavier and near-spherical particles settle down in the classifier hoppers. In addition, this product can be separated into the necessary number of fractions. Air stream carries scaly, light and small particles to the preliminary magnetic component separators 3.

Based on the magnetic dust deposition theory, magnetic cyclone was designed implementing processes of continuous settling surface regeneration and collected dust separation into the magnetic and non-magnetic products [9]. The proposed separator design allows staged separation of high-magnetic particles and then low-magnetic ones from the aerosol stream with non-magnetic final product at the end of process.

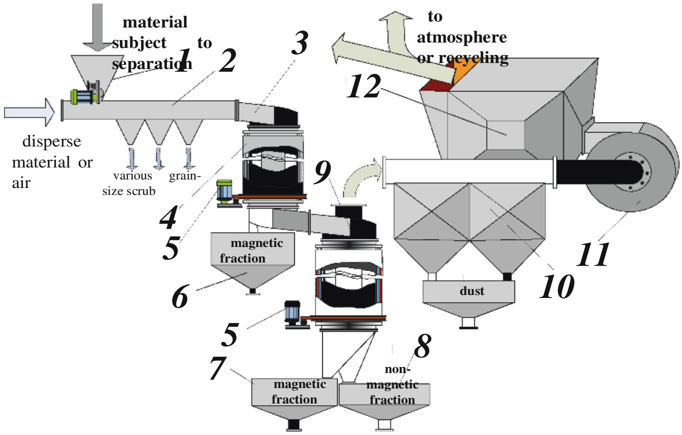


Fig. 2. System for magnetic fraction recovery from the dust material:
1 - feeder with hopper; 2 - horizontal air classifier; 3 - preliminary magnetic component separators; 4 – magnetic system; 5 – electric motor drive; 6, 7 – magnetic fraction collecting hopper; 8 – non-magnetic fraction collecting hopper; 9 - magnetic cyclone; 10 - inertial electrostatic precipitator (chamber); 11 – fan; 12 – electric filter for cleaning of the gas portion released to atmosphere

Magnetic product from these separators is collected in the hopper 6. Non-magnetic particles and particles detached for any reason from poles of the magnetic systems 4 rotated by the drive 5 are transferred to series connected magnetic cyclone 9 for the final separation of magnetic component and collection of non-magnetic one. Process taking place in this separator is similar to the process in the magnetic separator 3 with the only difference being that non-magnetic product is also collected and settled in the hopper 8, while magnetic product is settled in the hopper 7. Then dust-air stream enters inertial electrostatic precipitator 10 (for description see [10]) where dust particles are removed. In case of open-circuited separation process, final cleaning of air stream takes place in electric filter 12, otherwise major portion of air stream is fed to the inlet of air classifier 2, and excess air resulting from suction is released to atmosphere through the similar electric filter, of which operation is described in [11].

The result of conducted experiments is that iron-bearing product having total iron mass fraction of 61.3% and consisting of fine magnetite and metallic globules was produced from the sample of electric-furnace gas treatment slurry from the Krivoy Rog Central Mining Equipment Repair Plant with iron oxide mass fraction of 37.8% or 26.46% of total iron. Product density made 5.2t/m$^3$, which allows us to recommend this product as dense solids for heavy-density media. Iron recovery to the magnetic product made 73-75%.

**Conclusions**

1. Conducted researches resulted in solution of the problem of dense solids production from the technogenic material of metallurgical enterprises, which meet the following requirements: water insolubility; mechanical abrasion resistance; chemical non-reactivity with water and products of separation; sufficiently fine size; relatively low cost.

2. Electric furnace gas treatment slurries at the Krivoy Rog Central Mining Equipment Repair Plant were demonstrated to be polygenic formation of very complex mineral and petrography composition, which has majority of particles consisting of multiple crystalline or amorphous phases of various chemical composition and physical properties.

3. Magnetic dust collection and magnetic coagulation theories imply that high-magnetic dust particles can be separated from the dust-gas stream by providing their agglomeration conditions and recovery from the stream, which allowed development, manufacture and testing of pilot installation with capacity of 2t/h with the use of ironworks dust.

4. It is found that dry magnetic separation of electric-furnace gas treatment slurries of the Krivoy Rog Central Mining Equipment Repair Plant provides separation of high-magnetic iron-bearing product with density of over 5.2t/m$^3$. Iron recovery to the magnetic product made 73 - 75%.

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# PROSPECTS FOR RAISING THE CALORIFIC VALUE OF COMPOSITE WATER-COAL FUEL

A.I. Yehurnov, A.S. Makarov., D.P. Savitskiy, A.Y. Lobanov

**Abstract**
The fundamentals of coal-water fuel composition containing a liquid organic waste: fusel oil, glycerol, oil sludge. The rheological properties of coal-water fuel composition based on various stages of coal metamorphism are developed. Determined the calorific value of coal-water fuel composition. It is proved that the use of dense organic liquids such as fusel oil as a dispersion phase fuel disperse systems based on coal calorific value accompanied by a significant increase in fuel abilities whole. Usage of high-viscosity oil and glycerol sludge cannot completely replace the aqueous phase in the composition of the composite fuel based on coal while maintaining a low viscosity of fuel, which requires a search of other less viscous organic liquids which would be assigned to the class large mass waste. Completeness of burning fuel study of disperse systems is 98-99%, the calorific value of the original lump of coal when burned is 80-85%.

**Key words:** coal-water fuel, rheological properties, coal metamorphism, calorific value, composition, disperse systems, viscosity of fuel

## Introduction

The efficiency of extraction and processing of natural coal is determined by selecting the most appropriate technology. Modern concepts in the use of coal fuel is mainly focused on the development of technologies to reduce emissions and improve the completeness of the organic mass burn of solid fuel. A promising area, which develops in many countries, is the development of technologies for the production of fuel disperse systems based on natural coals. The most common of these is highly concentrated water-coal fuel (WCF) [1-4]. The main environmental benefits WCF include: the possibility of waste coal preparation, waste water, increasing the burn-up of the original coal, and reduce harmful emissions. Feature combustion WCF is that water, acting as an intermediate oxidant, activates the surface of the particles of the solid phase, thereby reducing the temperature of ignition. Accordingly burning drops WCF starts not from ignition of volatile vapors from a heterogeneous reaction on the surface of the coal particles with the water and the steam. Nevertheless, most of the research in this area indicates the necessity of increasing the calorific value of the fuel and to allow storage at low temperatures. Implementation of the task can be performed by using as a dispersion phase liquid organic waste polymer, pharmaceutical, coke-chemical,



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alcohol, oil refineries. Liquid organic wastes which can be used as a dispersion phase fuel disperse systems are characterized by viscosity, density, and the nature of intermolecular interactions. Certainly when filled environments such fine particles of coal is a problem of even distribution over the volume of the dispersion phase. When aqueous alcoholic phase and interaction between the coal particles and the dispersion medium is realized mainly by hydrogen bonds, with the use of environment that contain hydrocarbons and aromatic compounds - are realized hydrophobic interactions.

Since most processes for producing fuel disperse systems from coal depends on the rheological properties required reviewing main aspects of the regulation of these properties. It should be noted that the fuel dispersions based on natural coal are highly concentrated. If dilute disperse systems concentration uniform distribution of the dispersed phase is set arbitrarily, as a result of their participation in the thermal Brownian motion, the highly dispersed systems required to achieve the uniformity of the active movement of the particles under the influence of Brownian motion is eliminated as a result of a strong fixation of the spatial structural grid [5]. The determining factor in the destruction of the structure becomes totally combination of values of the kinetic energy of the particles and the energy of the repulsive forces. The main contribution to the change in the strength and energy of interaction between the particles can be accomplished by mechanical influence and change the properties of the interface: the dispersed phase-dispersion medium. The main methods of influence on the properties of the disperse phase, dispersion environment, and accordingly the phase boundary are physical and chemical methods. Application of chemical methods is the use of chemical reagents surfactants, polymers, electrolytes, the use of physical methods using electric and magnetic fields, ultrasound, high-frequency radiation, changes in temperature and pressure [6]. The fuel manufacturing processes disperse systems lend themselves to different types of mechanical effects: mixing, homogenization, pumping, spraying. The intensity of mechanical action determines the rheological properties of the system. For example shear rate at the pump for conveying coal-water slurries when they come into effect is 500-600 $s^{-1}$. During the passage of the suspension through a nozzle shear rate reaches 5000-7000 $s^{-1}$.

**Results of experiment**

Technology fuel disperse systems from coal consists of three main steps: a) grinding raw coal with a particle size of 10-20 mm, b) dispersion (grinding) of coal to a desired particle size distribution, and c) homogenization. Considering each stage separately, it should be noted that the choice of concept for producing fuel disperse systems will depend on the structural and mechanical properties of the brand of coal, and the nature of the dispersion medium. For the grinding of coal in the first stage, standard crusher: hammer, roll, jaw. For dispersion, which is accompanied by processes of mechanical activation and mechanical activation in the presence of chemical reagents used ball, rod, vibration, colloid mills, disintegrators, cavitators. When grinding can be used as dry coal, and supplemented with the dispersion phase and chemicals (so-called "wet" grinding). The latter option is more advantageous in terms of energy consumption and rheology systems obtained under the implementation of the "Rebinder effect." The homogenization step is used to implement the process of mixing the microparticles with the dispersion phase carbon and chemical reagents or for mixing ready-made systems. At this stage apply mixing vane, peristaltic pumps-homogenizers with pressure and temperature.

The main technological properties of fuel disperse systems based on coal and liquid natural organic phase include: 1) the rheology, 2) heating values, 3) environmental. Fuel dispersions based on coal liquid organic phase and water, may be various kinds of composition [7,8]. Let's review the physicochemical characteristics of some of them. To study the rheological properties of the fuel slurries used coal grade "B" (Dnipro basin, MCC "Aleksandriyaugol" Protopopovskiy section), mark "DW" and "T" (Donets basin, MCC "Luganskugol" mine Proletarian and Artema mine), grade "A" (Donetsk Basin, MCC "Sverdlovantratsit" Sverdlov mine). Technical coal and elemental analysis are shown in (Table. 1). In order to eliminate the influence factors on the dispersity of the rheological behavior of dispersed systems, all samples have the same coal particle size distribution: 250-160 micron - 40% 160-100 microns - 20%, 100-63 micron - 5%, 63-40 micron - 32 % <40 microns - 3%. As the dispersion phase used fusel oil (SE "Stadnitskii Alcohol Plant", Kyiv region., Water content 3%, the viscosity - $\eta$ = 1,49 Pa•s, density - $\rho$ = 1,263g/sm$^3$, net calorific value Q = 16 6 MJ /kg) (Table. 2), glycerol (Zaporozhskiy biofuel plant, $\eta$ = 5,8 mPa•s, $\rho$ = 0.84 g / cm$^3$, Q = 32 MJ / kg) (Table. 3), oil sludges (Naftoprom, Ivano-Frankivska reg., $\eta$ = 1,9 Pa • s, $\rho$ = 0,92 g / cm$^3$, Q = 21,5 MJ / kg) (Table. 4).

**Conclusion**

The results show that the use of dense organic liquids such as fusel oil as a dispersion phase fuel disperse systems based on coal calorific value accompanied by a significant increase in fuel abilities whole. Use of high-viscosity oil and glycerol sludge cannot completely replace the aqueous phase in the composition of the composite fuel based on coal while maintaining a low viscosity, which requires a search for other less viscous organic liquids which would be assigned to the class large-capacity waste. Weight burn up fuel studies disperse systems is 98-99% as opposed to raw coal.

Table. 1. Technical and elemental analysis of coal.

| Coal type | Technical analysis, mass.% | | | Elemental analysis, % on daf | | | | |
|---|---|---|---|---|---|---|---|---|
| | $W^a$ | $A^d$ | $V^{daf}$ | C | H | N | O | S |
| B | 51 | 20 | 48,5 | 70,1 | 5,0 | 1,2 | 19,7 | 4,0 |
| DG | 9,3 | 22,3 | 43,8 | 76,2 | 4,9 | 1,1 | 13,7 | 4,1 |
| T | 5,1 | 25,0 | 14,9 | 88,5 | 3,8 | 0,67 | 5,33 | 1,7 |
| A | 3,2 | 10,5 | 3,8 | 95,7 | 2.,3 | 0,4 | 1,1 | 0,5 |

Table 2. Technological characteristics of the fuel dispersed systems based on coal and fusels $C_t = 40\%$.

| Coal type | $\eta_{ef}$, Pa·s | $Q$, MJ/kg | $F$, % |
|---|---|---|---|
| B | 1,13 | 24,2 | 99,2 |
| DG | 0,96 | 28,5 | 99,5 |
| T | 0,86 | 30,0 | 99,4 |
| A | 0,77 | 31,5 | 99,4 |

Table 3: Process performance of the fuel dispersed systems based on coal, glycerol and water.

| Coal type | Coal | Water | Glycerin | $\eta_{ef}$, Pa·c | $Q$, MJ/kg | $F$, % |
|---|---|---|---|---|---|---|
| | $C$, % | | | | | |
| B | 50 | 32 | 18 | 1,4 | 13,1 | 99,2 |
| DG | 55 | 27 | 18 | 1,35 | 16,2 | 99,5 |
| T | 58 | 22 | 20 | 1,38 | 19,1 | 99,4 |
| A | 60 | 20 | 20 | 1,42 | 21,85 | 99,4 |

Table 4. Processing characteristics of the fuel dispersed systems based on coal and oil sludge

| Coal | Coal | Water | Oil sledges | $\eta_{ef}$, Pa·s | $Q$, MJ/кg | $F$, % |
|---|---|---|---|---|---|---|
| | $C$, % | | | | | |
| B | 50 | 35 | 15 | 1,42 | 13,2 | 99,5 |
| DG | 55 | 27 | 18 | 1,45 | 17,3 | 99,8 |
| T | 58 | 24 | 18 | 1,4 | 19,7 | 99,8 |
| A | 60 | 20 | 20 | 1,43 | 21,9 | 99,8 |

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# A STUDY OF THE BENEFICIATION OF COAL SLURRY DEPOSITS TO OBTAIN QUALITY PROPERTIES WHICH WOULD MAKE THEM USABLE COMMERCIAL ENERGY PRODUCTS

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**ABSTRACT**
The paper presents the results of the studies conducted during the implementation of the project involving the identification of the quantity and quality of coal slurries deposited. Altogether 20 impoundments were analysed and samples of slurries were taken to determine their quality properties such content of ash, sulphur, volatile matter and calorific value (Table 1). These properties do not always permit use of the slurries as fuel for energy generation. For this reason an attempt was made to beneficiate these slurries, using such methods as froth flotation, separation with the use of Reichert spirals, hydrocyclones & centrifugal classifiers (Table 2). The best results were obtained using flotation process. Coal slurry deposits represent some energy potential defined by the heat content. The energy potential was calculated for each analysed slurry impoundment (Table 3). The energy potential of the obtained beneficiation products was also calculated. The results are shown in Table 4 along with the loss of energy potential in the process of slurry beneficiation. The studies and analyses made it possible to identify the impoundment that can be used as source of feedstock for the production of fuel for energy generation.

**KEY WORDS:** coal slurries, slurry impoundments, beneficiation, separation, Reichert spirals, flotation, hydrocyclone, energy potential.

## 1. INTRODUCTION

The Centre of Waste and Environmental Management from the Katowice Branch of the Institute of Mechanised Construction and Rock Mining together with the Faculty of Mining and Geology, the Institute of Mining and Preparation Technologies and Waste Management at the Silesian University of Technology have carried out a research and development project entitled "Identification of the energy potential of coal slurries in the national energy balance and the strategy of the technological development in the field of their utilization" (Baic 2015). The project's aim was to resolve the issue of the accumulation of 120 million tons of coal slurries generated by beneficiation of hard coal (Baic 2015, Blaschke el al. 2012). Over more than a century of hard coal mining in Poland, the criteria for usable coal have been changing constantly. Until flotation was mastered on an industrial scale and pulverised coal-fired boilers were introduced to the power industry, fine particles of raw coal (usually less than 1 mm) were considered as a waste and deposited in impoundments. In the post-war period, when water-slurry circuits coal preparation plants were not closed, coal slurries were also collected in impoundments, thus increasing the amount of this material. These impoundments contain slurry of waste material from steam coal mines and tailings from coking coal flotation process. Coal slurries deposited in the impoundments were treated as refuse. They have some energy potential that can be effectively used. One of the ways to recover this energy is the use of beneficiation processes. Also discussed in this paper are our beneficiation tests and the energy potential of both unprocessed coal slurries and slurries beneficiated using various methods (Baic 2015, Blaschke el al. 2011, Lutynski el al. 2012, 2013, 2014).

 
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## 2. QUALITY PROPERTIES OF COAL SLURRIES DEPOSITED IN THE IMPOUNDMENTS

As part of the project, altogether fifty nine slurry impoundments were identified, of which twenty were selected for detailed study. After drilling boreholes, dozens of samples were taken from each impoundment. The samples were averaged and analysed for chemical composition to determine the content of selected elements; the water extract was also analysed. The results of this study were presented in a paper, which is printed in the proceedings of the 17$^{th}$ ICPC (Blaschke el al. 2011). The results of studies on quality properties are summarised in Table 1. The results reveal considerable differences in the quality of slurries deposited in individual impoundments (Baic el al. 2012, Baic 2015, Lutynski el al. 2012). The samples taken from twenty impoundments were used for studies and for beneficiation tests.

**Table 1 Summary of coal slurries properties in analytical state at impoundments** (Lutynski el al. 2012)

| Impound. No. | Average ash content [%] | Average total sulphur content [%] | Average volatile matter content [%] | Average net calorific value [kJ/kg] |
|---|---|---|---|---|
| K 13 | 27.47 | 1.90 | 28.50 | 15,096 |
| K 14 | 32.98 | 0.72 | 23.85 | 15,646 |
| K 12 | 41.36 | 0.86 | 21.31 | 14,813 |
| K 18/1 | 63.96 | 0.57 | 14.38 | 9,325 |
| K 18/2 | 63.04 | 0.64 | 14.39 | 10,073 |
| K 11/1 | 49.48 | 0.88 | 18.50 | 13,297 |
| K 3/1 | 60.43 | 0.70 | 16.41 | 9,265 |
| K 3/2 | 45.90 | 2.98 | 18.01 | 14,877 |
| K 2 | 58.34 | 2.26 | 14.29 | 12,304 |
| K 17 | 28.41 | 0.95 | 23.47 | 22,807 |
| K 1 | 26.98 | 0.95 | 23.77 | 23,293 |
| K 4/1 | 27.89 | 0.97 | 23.79 | 22,941 |
| K 4/2 | 47.22 | 0.59 | 18.89 | 15,813 |
| K 4/3 | 31.84 | 0.79 | 23.85 | 20,828 |
| K 5/1 | 53.79 | 1.21 | 16.99 | 12,051 |
| K 5/2 | 42.86 | 1.09 | 16.89 | 17,802 |
| K 5/3 | 37.59 | 0.94 | 20.64 | 19,402 |
| K 5/4 | 35.22 | 0.97 | 21.54 | 20,351 |
| K 11/2 | 37.33 | 0.92 | 20.72 | 19,672 |
| K 6 | 38.83 | 0.94 | 20.16 | 18,887 |

## 3. BENEFICIATION OF COAL SLURRIES DEPOSITED IN THE IMPOUNDMENTS

Beneficiation of coal slurry deposits was studied on a laboratory and semi-industrial scales. The experiments involved flotation process, separation using LD4 Reichert spirals, a hydrocyclone and a centrifugal classifier.

### 3.1. FLOTATION BENEFICIATION

Laboratory tests of coal slurry beneficiation by froth flotation were performed in a laboratory flotation machine with a flotation chamber of 1 $dm^3$. The density of the water-coal slurry was 100g/$dm^3$. Two flotation reagents were used in the study, which are most commonly used in flotation conducted in coal preparation plants at hard coal mines. The actual studies were preceded by experiments aimed at determining the optimal amount of flotation reagent. Further studies were conducted for the flotation reagent concentrations of 0.4, 0.5 and 0.6 kg per 1 Mg of dry material. Analysis of the results showed that the best results of coal slurry flotation were obtained for 0.6 Mg per 1 Mg of dry material. The studies were conducted for all the identified slurry impoundments. The coal slurry flotation test results were considered positive when 80% of the samples taken from a given impoundment gave such results. The flotation results of coal slurry from a single deposit were considered positive when 66% of the samples gave yields above 30% and the ash content in the concentrate was below 25%. Such conditions were fulfilled by samples from 12 impoundments. The test results that were considered positive are specified in Table 2.

### 3.2. SEPARATION WITH REICHERT LD4 SPIRAL SEPARATOR

Separation was conducted in an LD4 type Reichert separator. The test stand consisted of the feed box, an LD4 type Reichert separator with two troughs containing six coils and of a dewatering sieve. Before the actual study, many tests were performed to determine the best range of feed densities, feed supply rates to the spiral trough, the appropriate angle of the product splitters and the position of the separation products outlet ports. The study was conducted on feedstock from twenty slurry impoundments. Two feed densities were used: 300g/dm$^3$ and 400 g/dm$^3$. The results obtained for a density of 400 g/l were found to be the best and are shown in Table 2.

### 3.3. BENEFICIATION IN A HYDROCYCLONE

The possibility of coal slurry beneficiation in a 150 mm diameter hydrocyclone concentrating classifier was studied in a laboratory test stand. The feed of appropriate density was pumped to the hydrocyclone under constant hydrostatic pressure. Before the tests on actual materials, tests were conducted on waste products with similar properties to determine the most favourable feed densities and the best feed rates (feed pressures). It was found that the favourable feed density was 150 g/dm$^3$. The test stand was designed to separate coal slurries with particle sizes smaller than 1(2) mm. The results of slurry beneficiation in the hydrocyclone are given in Table 2. In gravity beneficiation the quality of the concentrates deteriorates in the presence of the smallest rock particles (<0.1 mm) which, predominantly, pass to the concentrate along with coal particles. The quality of the concentrate can be improved by removing sludge from the beneficiation feed or from the concentrates. Sludge removal was performed in a hydrocyclone classifier.

### 3.4. BENEFICIATION IN A CENTRIFUGAL CLASSIFIER

The concentrate obtained during studies with a hydrocyclone was used to find out whether slurries can be beneficiated in a centrifugal classifier. A centrifugal classifier employs centrifugal force for separation. It is used to separate slurries with particle sizes smaller than 1(2) mm. The feed with pre-selected density is pumped into the cone whose overflow ensures constant hydrostatic pressure. The products are discharged by gravitation into two tanks. Slurry beneficiation was studied for feed densities of 100 and 150 g/dm$^3$. Table 2 shows the results for the feed density 150 g/dm$^3$, which was found more advantageous than those of 100 g/dm$^3$.

### 3.5. ANALYSIS OF THE SLURRY BENEFICIATION RESULTS

The beneficiation methods used in this study produced different results. The best results of beneficiation were obtained using froth flotation. The average yield of the concentrate was about 64%. The net calorific value of the concentrate was 25,000 kJ/kg and the ash content was about 22%. Beneficiation by froth flotation showed a relatively high ash content of the tailings. These studies also demonstrated that the two commonly used flotation reagents did not produce good results for all the studied materials, which suggests that the use of flotation for beneficiation of other fine-particle refuse materials produced by other hard coal beneficiation methods requires a search for new, more effective reagents. Much less favourable results of slurry beneficiation were obtained in a centrifugal classifier and in the Reichert LD4 type separator. The material fed to these separation units was pre-treated in a hydrocyclone to remove particles below 1.0 mm in diameter. The average concentrate yield in beneficiation using the Reichert spiral was about 25% and in a centrifugal classifier about 22%. The net calorific values of these concentrates were 22,687 and 22,846 kJ/kg, respectively. The tailings produced by these methods had relatively low average ash content: 48% and 58%, respectively. Beneficiation in a hydrocyclone, owing to the separation of grains below 0.1 mm, gave a considerable yield – about 53%, but at the cost of quality of the concentrate, which had considerable ash content (39.6%) and a low net calorific value (16,950 kJ/kg). Also, the tailings produced in this process had a relatively low ash content (48.11%). Fine particles were unfortunately not separated effectively in that only about 38% of grains below 0.1 mm with high ash content remained in the concentrate.

**Table 2. Comparison of selected parameters of coal concentrate fraction obtained through beneficiation of the coal slurry using different methods** (Szpyrka el al. 2012)

| Impound. No. | Flotation | | | Reichert spiral | | | Hydrocyclone | | | Centrifugal classifier | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | yield | ash content | net calorific value | yield | ash content | net calorific value | yield | ash content | net calorific value | yield | ash content | net calorific value |
| | $\gamma_k$ [%] | $A^a$ [%] | $Q^a$ [kJ/kg] | $\gamma_k$ [%] | $A^a$ [%] | $Q^a$ [kJ/kg] | $\gamma_k$ [%] | $A^a$ [%] | $Q^a$ [kJ/kg] | $\gamma_k$ [%] | $A^a$ [%] | $Q^a$ [kJ/kg] |
| K 13 | - | - | - | 29.05 | 21.10 | 18,825 | 47.50 | 31.64 | 18,121 | 23.22 | 21.14 | 18,916 |
| K 14 | - | - | - | 40.64 | 22.14 | 20,271 | 55.58 | 26.12 | 20,362 | 36.17 | 20.68 | 20,654 |
| K 12 | - | - | - | 17.64 | 21.86 | 21,523 | 49.52 | 39.64 | 17,281 | 9.68 | 20.31 | 22,042 |
| K 18/1 | - | - | - | 4.35 | 22.07 | 21,042 | 50.03 | 57.45 | 9,295 | - | - | - |
| K 18/2 | - | - | - | - | - | - | 59.66 | 71.63 | 8,576 | - | - | - |
| K 11/1 | - | - | - | 14.51 | 22.81 | 20,760 | 51.05 | 42.51 | 15,990 | 4.00 | 22.06 | 21,043 |
| K 3/1 | - | - | - | 22.54 | 21.20 | 25,843 | 77.35 | 43.87 | 16,277 | 14.08 | 21.59 | 25,840 |
| K 3/2 | 44.8 | 22.7 | 24,687 | 5.84 | 23.81 | 24,258 | 63.40 | 56.87 | 12,027 | - | - | - |
| K 2 | 41.5 | 31.2 | 20,670 | 49.85 | 20.47 | 24,335 | 57.89 | 25.13 | 24,234 | 48.44 | 20.47 | 24,104 |
| K 17 | 73.7 | 14.5 | 11,620 | 8.68 | 21.45 | 19,136 | 44.23 | 46.22 | 13,444 | 2.91 | 21.68 | 18,965 |
| K 1 | 79.6 | 15.9 | 27,120 | 14.31 | 22.31 | 24,241 | 52.08 | 43.91 | 17,972 | 8.25 | 19.93 | 25,046 |
| K 4/1 | 81.1 | 16.4 | 26,880 | 30.47 | 20.33 | 24,459 | 50.58 | 32.20 | 24,363 | 28.03 | 20.47 | 24,095 |
| K 4/2 | 65.3 | 29.4 | 21,525 | 49.99 | 22.52 | 23,763 | 59.24 | 24.21 | 24.557 | 46.57 | 21.14 | 24,164 |
| K 4/3 | 41.5 | 31.2 | 24,520 | 52.45 | 21.84 | 24,333 | 57.43 | 22.36 | 25,501 | 51.64 | 21.56 | 24,315 |
| K 5/1 | - | - | - | 29.92 | 23.26 | 23,352 | 45.61 | 30.96 | 21,415 | 24.73 | 20.46 | 24,430 |
| K 5/2 | 58.4 | 23.6 | 24,670 | 27.41 | 22.27 | 23,666 | 47.57 | 34.88 | 21,085 | 21.73 | 21.51 | 24,043 |
| K 5/3 | 71.7 | 19.2 | 25,875 | 29.66 | 21.46 | 24,035 | 50.50 | 34.21 | 21,161 | 25.96 | 21.86 | 23,802 |
| K 5/4 | 71.1 | 19.5 | 25,810 | 26.56 | 21.39 | 24,195 | 50.74 | 35.67 | 21,844 | 21.99 | 21.15 | 24,281 |
| K 11/2 | 70.0 | 19.3 | 25,845 | 7.05 | 20.82 | 18,756 | 43.79 | 48.63 | 12,008 | 2.68 | 22.35 | 18,519 |
| K 6 | 71.7 | 21.3 | 25,465 | 13.97 | 21.35 | 24,256 | 47.50 | 44.06 | 18,022 | 8.37 | 21.82 | 24,124 |

## 4. ENERGY POTENTIAL OF RAW AND BENEFICIATED COAL SLURRIES

### 4.1. METHODOLOGY OF THE STUDY

The practical value of coal slurries depends on their calorific value and the content of ash and moisture. These properties determine whether particular slurry can be used as a source of energy. The impoundments were filled with slurry over many years and thus the deposits reflect changes occurring in individual mines. These involve mining coal from different seams, varying quality of extracted coal, the type of accompanying rock, the beneficiation technologies used and the accuracy of coal separation from refuse achieved in various types of equipment beneficiating raw coal. Consequently, the properties of slurries in large impoundments may be much diversified. This requires appropriate sampling procedures: samples should be taken at many points and at various depths. Some of these results were described in a paper presented at the 17$^{th}$ ICPC (Blaschke el al. 2013). Detailed studies made it possible to assess the weight of slurry in each of the impoundments. Depending on the surface and capacity of the impoundments, several to almost twenty samples were taken. The average net calorific value was determined for each borehole sample together with the maximum and minimum calorific values. The average calorific value was calculated for each slurry impoundments.

### 4.2. DETERMINING THE ENERGY POTENTIAL

The studies made it possible to calculate the energy balance, referred to as energy potential. It is defined as the amount of heat in GJ, calculated as the product of mass and the average net calorific value of the slurries deposited in the impoundments. The average net calorific values of slurries from individual impoundments are listed in Table 3. The maximum and minimum values are also given: they are expressed as deviations from the average value. Three values of energy potential are given: the average, minimum and maximum potential. All the specified values pertain to the analytical state.

**Table 3 Energetic potential of coal slurries in analytical state deposited at impoundments**

| Impound. No. | Approximate capacity of impoundment | Average net calorific value | Deviation from the average net calorific value | Energy potential, analytical state | | |
|---|---|---|---|---|---|---|
| | | | | average | maximum | minimum |
| | Mg | kJ/kg | kJ/kg | GJ | GJ | GJ |
| K 13 | 1,000,000 | 15,096 | 1,509 | 15,095,667 | 16,604,265 | 13,587,068 |
| K 14 | 300,000 | 15,646 | 830 | 4,693,800 | 4,942,657 | 4,444,943 |
| K 12 | 1,000,000 | 14,813 | 581 | 14,812,667 | 15,393,327 | 14,232,006 |
| K 18/1 | 100,000 | 9,325 | 2,052 | 932,547 | 1,137,768 | 727,326 |
| K 18/2 | 100,000 | 10,073 | 2,747 | 1,007,325 | 1,281,976 | 732,674 |
| K 11/1 | 640,000 | 13,297 | 2,413 | 8,509,964 | 10,054,237 | 6,965,690 |
| K 3/1 | 1,521,000 | 9,265 | 3,498 | 14,092,825 | 19,413,371 | 8,772,280 |
| K 3/2 | 176,000 | 14,877 | 5,976 | 2,618,308 | 3,670,019 | 1,566,597 |
| K 2 | 1,117,000 | 12,304 | 2,803 | 13,743,987 | 16,874,910 | 10,613,064 |
| K 17 | 155,000 | 22,807 | 1,538 | 3,535,074 | 3,773,403 | 3,296,745 |
| K 1 | 153,000 | 23,293 | 1,444 | 3,563,810 | 3,784,749 | 3,342,871 |
| K 4/1 | 345,600 | 22,941 | 590 | 7,928,525 | 8,132,297 | 7,224,753 |
| K 4/2 | 163,000 | 15,813 | 937 | 2,577,600 | 2,730,378 | 2,424,822 |
| K 4/3 | 460,000 | 20,829 | 2,065 | 9,581,173 | 10,530,941 | 8,631,404 |
| K 5/1 | 130,000 | 12,051 | 1,504 | 1,566,590 | 1,762,060 | 1,371,119 |
| K 5/2 | 228,000 | 17,802 | 5,351 | 4,058,928 | 5,279,050 | 2,838,807 |
| K 5/3 | 106,000 | 19,402 | 646 | 2,056,612 | 2,125,131 | 1,988,132 |
| K 5/4 | 102,000 | 20,351 | 844 | 2,075,761 | 2,161,898 | 1,989,625 |
| K 11/2 | 176,000 | 19,672 | 767 | 3,462,345 | 3,597,362 | 3,327,329 |
| K 6 | 236,000 | 18,887 | 1,834 | 4,457,435 | 4,890,353 | 4,024,518 |

The concentrate produced by beneficiation has a higher calorific value than the feed, but has a lower mass. Table 4 shows the calculated energy potentials of concentrates obtained by the beneficiation of slurries from individual impoundments. The energy potential is calculated as a product of slurry mass, recovery of the concentrate and its net calorific value.

**Table 4 Energetic potential of coal slurries deposited at impoundments as a result of separation** (Lutynski el al. 2012)

| Impound. No. | Impound. | Concentrate from flotation process | | Concentrate from Reichert spiral | | Concentrate from hydrocyclone classifier | | Concentrate from centrifugal classifier | |
|---|---|---|---|---|---|---|---|---|---|
| | average potential $E_{śr}$ | potential $E_{śr}$ | loss of potential | potential $E_{śr}$ | loss of potential | potential $E_{śr}$ | loss of potential | potential $E_{śr}$ | loss of potential |
| | GJ | GJ | % | GJ | % | GJ | % | GJ | % |
| K 13 | 15,095,667 | - | - | 5,459,250 | 64 | 8,516,870 | 44 | 4,350,680 | 71 |
| K 14 | 4,693,800 | - | - | 2,493,333 | 47 | 3,420,816 | 27 | 2,230,632 | 52 |
| K 12 | 14,812,667 | - | - | 3,874,140 | 74 | 8,640,500 | 42 | 2,204,200 | 85 |
| K 18/1 | 932,547 | - | - | 92,585 | 90 | 464,750 | 50 | - | - |
| K 18/2 | 1,007,325 | - | - | | - | 514,560 | 49 | - | - |
| K 11/1 | 8,509,964 | - | - | 1,992,960 | 77 | 5,219,136 | 39 | 538,700 | 94 |
| K 3/1 | 14,092,825 | - | - | 9,040,657 | 36 | 12,377,730 | 12 | 5,502,369 | 61 |
| K 3/2 | 2,618,308 | 19,55,210 | 25 | 150,564 | 94 | 1,333,553 | 49 | - | - |
| K 2 | 13,743,987 | 94,66,240 | 31 | 12,602,267 | 9 | 9,221,639 | 33 | 11,923,600 | 13 |
| K 17 | 3,535,074 | 31,68,014 | 10 | 266,947 | 92 | 916,880 | 74 | 88,187 | 97 |
| K 1 | 3,563,810 | 33,19,488 | 7 | 519,000 | 85 | 1,429,852 | 60 | 306,000 | 91 |
| K 4/1 | 7,928,525 | 75,24,680 | 5 | 2,535,909 | 68 | 4,294,124 | 49 | 2,331,625 | 71 |
| K 4/2 | 2,577,600 | 22,80,574 | 11 | 1,936,684 | 25 | 2,073,136 | 20 | 1,851,204 | 28 |
| K 4/3 | 9,581,173 | 46,24,472 | 51 | 5,820,454 | 39 | 6,686,362 | 30 | 5,816,148 | 39 |
| K 5/1 | 1,566,590 | - | - | 910,728 | 42 | 1,180,617 | 25 | 793,975 | 49 |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| K 5/2 | 4,058,928 | 32,62,361 | 20 | 1,456,879 | 64 | 2,307,542 | 43 | 1,205,997 | 70 |
| K 5/3 | 20,566,631 | 19,74,780 | 4 | 764,313 | 63 | 1,121,533 | 45 | 655,983 | 68 |
| K 5/4 | 2,075,761 | 18,69,160 | 10 | 666,330 | 68 | 1,136,324 | 45 | 544,865 | 74 |
| K 11/2 | 3,462,345 | 31,84,104 | 8 | 231,074 | 93 | 92,999 | 97 | 65,187 | 98 |
| K 6 | 4,457,435 | 43,27,013 | 3 | 801,000 | 82 | 1,999,000 | 55 | 455,461 | 90 |

## 5. CONCLUSIONS

The results of coal slurry beneficiation using four methods and analysis of their energy potential showed that considerable part of this potential is lost in the course of beneficiation because the smallest grain of coal pass on to the tailings. The most advantageous results were obtained by froth flotation, which is understandable considering the nature of this method. The average loss of the energy potential in flotation tests was 15%. This loss ranged from 3% to 21% depending on individual slurry impoundments. The average net calorific value was 25,057 kJ/kg and was the highest of all the analysed methods. Unfortunately, not all coal slurries were susceptible to flotation process – in terms of the adopted evaluation criteria – with reagents used in this study. The highest losses of the energy potential of coal slurries were observed in the centrifugal separator with preliminary sludge removal. The average loss was 68% and ranged from 13% to as much as 98% depending on individual slurry impoundments. The great differences in the losses between individual ponds clearly demonstrate that the method is not adequate. However, the average net calorific value of the product obtained in the centrifugal classifier was 22,864 kJ/kg, which seems a very good result. These results are similar to those obtained using spirals with preliminary removal of sludge. The average loss of the energy potential of slurries beneficiated with Reichert spirals was 64% and the calorific value of the concentrate was 22,678 kJ/kg. The lowest calorific values were noted for products obtained by beneficiation in the hydrocyclone. In spite of the considerable loss of energy potential of the slurry – which was 44% and ranged from 12% to 97% – the average calorific value of the product was 16,950 kJ/kg and did not increase much relative to the calorific value of unbeneficiated slurry, which was 16,427 kJ/kg.

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# Methods of Increasing the Calorific Value of Fine Coal Waste

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*Abstract*

The most frequent solution of water-slime industry in mines and companies working on coal output enrichment is linking watercourses from individual technological nodes into one stream of water-slime slurry. The outcome of such solutions is water-slime mixture of averaged composition and physicochemical properties.

Depending on the applied slurry dewatering methods, we obtain cakes of slime coal mixtures or/and, in the case of gravitational dewatering of slurry in settlings, slimes diversed in composition in relation to the place of sedimentation.

Among all studied various methods of coal concentrates extraction from slime – water slurries, the most interesting results are these of industrial significance obtained on mesh sieves. The new production of arched and centrifugal sieves (the so-called OSO),with minimum mesh size, enables selective coal concentrates extractions from coal slimes. The recovery of coal concentrates from flotation wastes is more difficult and most often the best results are ensured through application of flotation processes.

The conducted research of fine-grained coal wastes and side products of coal combustion lay a fundament to elaboration of new technology, pilot installations and incorporation completion.

*Key words:* coal slime, flotation waste, fine coal waste, incerasing calorific value

**Introduction**

Growing restrictions in relation to gas emission pollution to environment and the necessity for economic improvement during combustion processes indicate purposefulness of analysis of existing solutions in order to prepare and apply low calorific fuels and to elaborate new approach in actions taken to further



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improvement.

Current processing solutions in mines and enrichment companies do not guarantee substantial increase in power value of produced fine grain coal wastes. Acquiring significant increase of power value of current coal slimes, postflotation wastes etc. requires incorporation of new solutions in technological and organizational procedures.

Material presented constitutes an analysis of selected research results and incorporations as well as studies on possibilities of quality increase of fine grain coal wastes and rationalization of its use in energy industry.

**Research and introduction connected with decrease in moisture of fine coal wastes**

Research and attempts of water content reduction in fine coal wastes were conducted and assessed with filtering devices in the process of feed granulation for filtration, influence of ionic surfactants, application of flexible water permeable tanks and application of chemical dewatering agents.

In the case of selected power coal slime water slurry was concentrated in radial Dorr thickener and subsequently directed into parallel working filtering devices. The yearly average of water content in cakes from the chamber filter press amounted to 29,2%, whereas cakes from belt filter contained 40,9%. Additional application of sedimentation centrifuge sieve-sieveless to dewatering of coal slimes allowed for even greater results, as in parallel with dewatering the process of slime occurs. The recovery of dewatered slime is 75% with calorific value exceeding 14.000 kJ/kg, from feed of calorific value 9.500 kJ/kg.

The use of chemical agents to decrease water content in coal fuels, including coal slimes/postflotation wastes is practical only when it involves obtaining new properties of the product, for instance:

– structure alteration of slimes from paste into partial/solidified,

– adaptation of slimes to safe transportation and storage (petrification, granulation),

– self heat prevention,

– neutralization of pyrite compounds in fuels.

The most often applied reagents for water bindings in coal slimes are: calx (CaO), semiwater plaster/anhydrite ($CaSO_4/CaSO_4.0,5H_2O$), cement, flying and bottom ash of calcium and fluidal kind etc. as well as dry slaked calcium (hydrated)The common feature of all discussed reagents is presence of active calcium compounds (CaO or/and $CaSO_4/CaSO_4.0,5H_2O$).

Based on current studies and trials, techno-economic assumptions has been elaborated and two installations have been introduced for coal slime cake granulation with two rapid reaction mixers.

Application of granulated coal slimes in fluidized-bed furnaces is of particular economic importance, as well as a supplement to coal culms. Fluidized-bed furnaces are currently equipped with option to prepare water-slime pulp injected into furnace bed with average water content of 40%. Balances investigation shows that granulate coal slime combustion, in comparison to pulp, leads to increase in calorific value of combusted slimes of approximately 2.000 kJ/kg and decrease in combustion costs.

**The research regarding increase in content of coal substance in fine grain coal wastes**

On the basis of conducted studies and research on emission of coal concentrates from fine materials and wastes, there were selected and verified through trials among others the following:

– methods of selective emission of courses rich in coal from water-slime circulations,

– methods of grain classification, densimetric and based on differences in surface properties of its components,

– methods of ‘deep’ enrichment of fine grain mining wastes,
– changes in methods hitherto of water-slime slurry storage in settlings.

Selective separation of fine coal wastes from water courses circulations from coal enrichment companies. The water-slime industry in mines is an example of averaging of various quality and composition of courses from individual technological nodes from extraction companies and then dewatering of water-slime slurries, which result in slimes of average properties and composition.

The analysis of solid phase included in water-slime courses show high diversity of their composition and energetic properties regarding the place of their origin.

Separation of fractions rich in coal from fine grain coal wastes. The studies of grain composition of fine coal wastes demonstrate a regularity, that in coal slimes one can explicitly differentiate grain fractions, which are either poor or rich in mineral components (marked as ash). The most frequent inflection point is slime granulation on the border of 30 to 60 micrometres.

The possibility and purpose of grain classification of coal slimes has been confirmed for many years and by many facilities. However, it hasn’t been incorporated into industries in such great extent. The most commonly presented are propositions of water-slime slurries classification in hydrocyclones and then on spiral separators.

The choice of device used for the reasearch on grain classification was based on quality and quantity requirements of coal concentrates emission above 0,05 mm. Among numerous methods of grain classification of coal slimes, the recovery of coal concentrates was carried out on:
– screen type Stack Sizer from Derrick Corporation,
– arched sieve (VariSieve) from Progress Eco SA,
– rapid vibratory screen with textile membrane filter.

Despite satisfactory results of enrichment, there were no studies on application of concentrate tables, as well as of hydrocyclones, spiral separators and gravitational jiggers (TBS Teeter bed separator; hydrosizer). Whereas, flotation wastes underwent separation in centrifuges with high centrifugal force (Falcon) and oil agglomeration (Otisca) not only for assessment of possibilities regarding concentrates emission but also for assessment of economic benefits arising from such incorporation. Moreover, there was also demineralization and coal desulfurization carried out in the gravimelt process. The latter studies and trials and trials on Stack Sizer screen were conducted in cooperation with creators/owners of the mentioned processes.

The trials conducted on Stack Sizer screen demonstrated efficiency of the classification process, as well as the process of unloaming the course of extracted coal concentrate with granulation over 0,075 mm. Coal slimes with combustion heat in the frame from 12.807 to 14.622 kJ/kg enabled to extract coal concentrates with calorific value from 14.864 to 25.932 kJ/kg.

In the case of coal concentrates separation, solution of fine grain coal wastes was applied with rapid vibratory screen with textile filter screen. Depending on the applied membrane, the obtained coal concentrate, from studied slime, varied from 6,9 to 11,6% during the process of singular sift, whereas after secondary sift it varied from 7,01 to 19,24%. The results were verified with industrial device.

On the basis of gathered experience, the installation was built, which aim is to recover coal concentrates from slime extracted from the bottom of lake, where flotation waste and slime from nearby mines have been accumulating over the years. This installation enabled decrease in ash content by ca. 33 to ca. 22% and increase in calorific value from 15.000 to 18.000 kJ/kg.

The choice of arched sieves for coal slime classification resulted from its specific properties. Tangential application of thin layer (film) of water-slime slurry onto surface curve of mesh sieve triggers reduction of active space of sieve mesh s and raise of centrifugal forces on dewatering and grain division from heavy and light. In numerous examples, a process of coal grain flotation in arched sieves is also observed.

The correlation between sieve mesh size ‘s’ and diameter of separated grains ‘$d_{gr}$’, is estimated by the following equation:

$$d_{gr} = 0,5 \div 0,6 \cdot s \tag{1}$$

Regarding that the borderlines between coal fractions and loam fractions are close to the limit of grain diameter, which is 0,05 mm on average, it means that the most relevant is arched sieve with mesh size of 0,1 mm.

The most comprehensive coal concentrates emission was obtained on arched sieve with mesh size of 0,1 mm. There were studied three water-slime slurries derived from slime cakes, which varied with grain fraction content + 0,063 mm, and by this coal concentrates of high heat combustion value (18,6; 20,3 and 27,1 MJ/kg) were acquired, with adequate recovery volume of 25; 17,7 and 13,7% of the feed.

Traditional arched sieves have been substituted with dewatering centrifugal sieves (the so-called OSO sieves) to improve the process of quantitative emission of coal concentrates and to increase capacity of sieves (calculated as $m^3/m^2$). It was especially essential, as the production company Progress Eco SA has developed production of wear-resisting sieves with mesh size of 0,1 mm – figure 1. OSO sieves are features with high efficiency of grain distribution and higher efficiency per unit ($m^3/m^2$), due to action of higher centrifugal force.

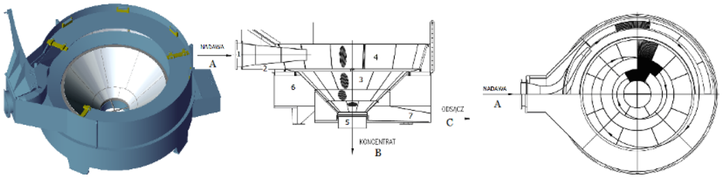


**Fig. 1. Centrifugal dewatering sieve OSO**
**A – feed; B – concentrate; C –filtrate; 1 – feed nozzle; 2 - steering wheel; 3 - slotted centrifugal sieve; 4 - wheel; 5 - outlet dewatered product; 6 - filtrate collector; 7 - outlet.**

This solution is also supported by industrial experiments with the use of OSO sieves applied for dewatering of culm and slimes in mining industry and its application for separation of bottom ash from flying ash from water slurries and water purification from water-bottom ash circulations.

Pilot installation, with industral sieve of diameter of 1200 mm, was constructed in order to verify assumptions regarding separation and selection of grain fractions with granulation over 0,05 mm with the use of OSO sieves with mesh size of 0,1 mm. The OSO sieves were also used to test efficiency and conditions of separation and emission of grain fractions over 0,05 mm.

The aforementioned research and trials, in comparable conditions, enabled us to conclude the following:

1. OSO sieve with mesh size of 0,1 mm and diameter of 1200 mm, in standard construction, offers filtering surface of 2,3 $m^2$, whereas in the case of upgraded sieve it is 2,8 $m^2$.

2. Unit capacity of the upgraded sieve equals 14,3 $m^3/m^2$ with feed processing in the volume of 40$m^3$/h and 11,8 $m^3/m^2$ with feed processing with the volume of 33 $m^3$/h;

3. OSO sieve capacity during gravitational feed input, to high degree, depends on hydrostatic feed pressure and the degree of throttle opening in pipelines providing the feed. After the change of pressure height of slurry onto the sieve in the range from 0,96 m to 1,8 m and the degree of throttle opening from 22 to 100%, the OSO device efficiency increased from 14 to 39 $m^3$/h;

4. The construction and method of installation of the nozzle providing feed onto the sieve wheel have great impact on capacity and quality of feed processing. The task of nozzle is not only to provide the feed but, most importantly, to provide kinetic energy ensuring circulation of the feed on sieves of the wheel, shelf and cone – figure 2.

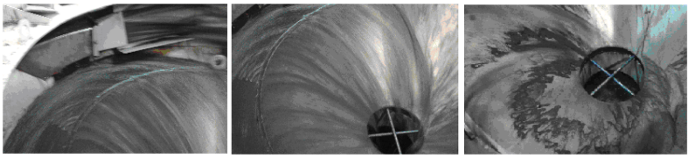

**Fig. 2. Centrifugal dewaterinf sieve in work**

5. The volume of selected grain fraction over 0,05 mm (of concentrate) for 100 μm constituted averagely 24% of the fees, which is demonstrated on mass balance in figure 3.

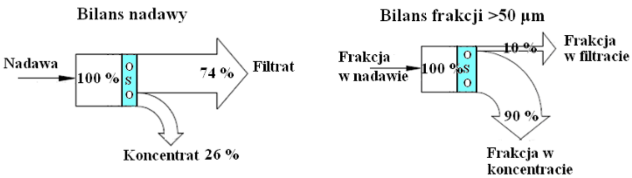


**Fig. 3. Mass balance of feed and >50 μm fraction**

**Conclusion**

The possibilities to improve calorific properties of fine grain coal waste for power engineering have been confirmed by the conducted research and trials and partially by incorporations on industrial scale.

The decrease in water content in fine grain waste can be achieved through the selection of appropriate filters and dewatering centrifuges, as well as with feed granulation and ionic surfactant. Especially efficient method of dewatering and acquiring new structure of coal slimes is their granulation with addition of active agents of calcium.

The best method to increase fine grain coal waste energy value is their selective extraction from the courses of the chosen technological nodes from the water-slime circulation and their grain classification in devices ensuring extraction of grain fraction over 50 μm. From the all analysed and verified grain classification method in pilot installations, the most efficient and at the same time the simplest solution appeared to be application of dewatering centrifugal sieve with mesh size of 0,1 mm.

Fine-grained coal wastes in the form of cakes and/or deposits from settlings are only partially used as independent fuel or as a supplement to basic fuels, whereas the rest most often constitutes waste landed into environment. High content of ash and water in fine grained coal wastes is not advantageous for its rational use as fuels (low calorific value, increase of $CO_2$ emission).

The research carried out on numerous fine grained coal wastes from current production and settlings has shown substantial differences in composition and properties of coal slimes and postflotation wastes. In the case of coal slimes we observe relatively acute separation of mineral part (loam) from coal maceral part in relation to its grain composition. The research conducted on postflotation wastes does not report analogous relationship.

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# Ecological and Economic Aspects of the Management of Mining Waste in TAURON Mining S.A.

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*Abstract*

The article presents the activity of TAURON Mining S.A. in the scope of the management of mining waste generated in mining companies. In addition to brief characteristics of the waste (mining waste, tailings, and previously deposited waste), as well as technologies and installations to generate new useful products on their basis, present and forecasted for next years results obtained from research ecological projects and company plans for the future in this scope were shown. The aim of actions taken in this field is to eliminate the problem of waste in mining companies as a result of their total use.

*Key words:* coal mining waste management, economic benefits, coal slimes granulation, aggregate-ash mix

## Introduction

### TAURON MINING S.A. IN TAURON GROUP

TAURON Group is one of the biggest business entities in Poland which can dispose its own capital in amount of 16.5 billion PLN. Holding employs more than 29 thousand people and is

- The biggest distributor of energy in the country (18.3% of the national area);
- The second biggest producer of energy in Poland;
- The biggest deliver of warmth in Upper Silesia.

 
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Figure 1. TAURON Mining S.A. headquarters in Jaworzno

The Group is active in the Southern part of Poland. The companies of TAURON Group act in such voivodships as: Slaskie, Malopolskie, Opolskie and Dolnoslaskie. The basic activity of TAURON Group is:

- Extraction of coal;
- Production of energy and warmth;
- Distribution of energy;
- Management of energy;
- Distribution and selling of warmth.

As the producer of coal, TAURON Mining S.A. has about 20% of national balance stocks of energetic coal and 7% share in selling market.
The parts of TAURON Mining S.A. are two mining plants:

- *Sobieski* in Jaworzno (ZG Sobieski) – Figure 1
- *Janina* in Libiaz (ZG Janina) – Figure 2

These plants are seen as the modern ones. This is the effect of the deep technical, technological and organizational restructuration conducted here during past years.

Figure 1. Sobieski Coal Mine

Figure 2. Janina Coal Mine

## 1. SECONDARY PRODUCTS

Mining plants TAURON Mining S.A. are producing about 5.5 million tones/year of coal in various sorts. Simultaneously, apart from it they are producing also about 2.0 tons/year of rocks and additional materials.

Rocks and additional materials were treated as wastes for a long time. They were transferred to outer plants as:

- Wastes 01 01 02 – being produced during preparation works, which are directed to the surface without Mechanical Processing Unit (wastes originating from extraction of materials different than metal ores);
- Wastes 01 04 12 – being produced in Mechanical Processing Unit as well Fine Coal and Water-Slurry Management Plant in ZG Sobieski and ZG Janina (wastes being created during washing and cleaning of materials different than these listed in 01 04 07 and 01 04 11), which can be divided into two types:
  - Gangues;
  - Coal slimes.
- Wastes 19 13 06 – sludge originating from underground waters clearing, different than these listed in 19 13 05 – wastes in form of slimes or sludge gathered in bottom waters tank.

The sorts of these rocks and materials differ between themselves not only because of granulation but also consistence and mainly by mineral composition, flammable abilities and physicochemical features.
The main components of the gangue originating from preparation mining works are limestone (10-90%) and loamy rocks. The contents of coal is various (2-8%) while the contents of sulfur is equal to about 3%. The particle size distribution of wastes depends on mechanical properties of the processed rock bodies by means of blasting materials and applied method of processing.
The main component of the gangue from mineral processing plants is 70-90% of loamy rocks slivers. The rest are limestone, slimes and coal outgrowths. The contents of sulfur is equal to about 0.4 till 0.6% and in some cases it reaches even 2.5%.
Coal slimes – both from direct production as well these occurring in the tank – as well fine-grained processing wastes contain mainly loamy rocks and loamy-carbonate rocks what causes growth of share of loamy mineral substance and raises coal contents which can exceed 10% and in some cases even 20% (slime from tanks). These wastes characterize with higher contents of sulfur comparing to coarse-grained wastes.
The conducted tests of leaching indicated that the main ions in the composition of water extract for the gangue originating from mining works (preparation and processing) are sulfates and sodium.
The general contents of pollutants, ability to their leaching and negative influence on environment are slight and are not hazardous for surface and underground waters quality as well soil quality.

## 2. ADDITIONAL MATERIALS – WASTES OR PRODUCT?

One ton of extracted material in mines contains from 60 to 90% of coal, which occurs on the market in various assortments, and 10-40% of accompanying rocks which usually occur on slag heaps instead of the market. These accompanying rocks are being called popularly the gangue, coal slate or extracting waste. In past years they were mainly generating costs which, however, did not affect too much on economic balance of mines. Currently, with law and social changes together with no more space on slag heaps or tanks, the situation caused that these costs raised significantly and will grow even more in future.
The introduced novelizations of the laws and orders limiting amounts of stockpiling made the coal producers to start various activities causing minimization of the mines influence on the natural environment.

In TAURON Mining S.A. the works over successive limitation of growing costs of waste management are in progress. Apart from accepted organization works also the realizations of investments supporting extractive wastes processing into products of the certain application are being done. It would be not possible if there were not researching and developing works allowing determination of properties, chemical composition as well directions of applications of both gangue and slimes being produced during coal processing process. In such way the usability of gangue was determined. It can be useful in:

- Construction of road and railway embankments;
- Construction and strengthening of floodbanks;
- Reclamation, revitalization and leveling of the grounds;
- Funding ground objects.

Coal slimes were properly characterized in purpose of transforming into low-calorific granulated fuel. From the wastes in form of the gangue the aggregates started to be produced for which TAURON Mining S.A. got Technical Approbates as well other documents allowing to use them in the directions mentioned above. The special installments allowing to produce aggregates according to the client's expectations were constructed in Jaworzno and Libiaz. The additional asset of these aggregates is using of binders to their production, which are based on ashes originating from power plants of Group TAURON. In this way, from the moment of starting installment to construct aggregates in 2011 in ZG Sobieski and in 2013 till now, about 1.5 million of tons of aggregate was produced and sold in TAURON Mining S.A., including:
0.3 million of tons of aggregate to construct road embankments;
0.4 million of tons of aggregate to reclaim municipal waste landfill site in Jaworzno-Pieczyska;
0.8 million of tons of aggregate to construct floodbanks on the river Wisla in area of the city Oswiecim.
The construction of installments to produce aggregates constructed by TAURON Mining S.A. are designed in the way allowing the transformation of about 90% of the gangue being produced during coal extraction and processing into the aggregate. The remained part, mainly fine-grained materials will be located on the bottom of the mines by the plants. In such way TAURON Mining S.A. will have potential to manage 100% of accompanying rocks during coal extraction process. So called "waste" will then fulfill the requirements and will be properly prepared in special installments. It can be used as the full product instead of natural materials.

Figure 4. Construction of road with use of aggregates from Sobieski Coal Mine in the phase of construction of the road embankment and after completion of construction

Recently, the problem of coal slimes management remained big and cost generating problem. These slimes were produced during the process of processing of extracted coal. Theoretically, as the flammable material slimes could be used. However, in practice the cakey, wet consistence of the slimes did not allow to use them in processes of combustion or mixing with other fuels. The part of produced slimes in such form was used in the thermal power plants of the Group TAURON. However, their state before transferring it to the combustion chamber must have been changed from the cakey one to the form of watery emulsifier. Such way of preparing slimes causes big loss on calorific value because of the necessity of applying water to water down process. Till 2010 only about 25% of the slimes produced in TAURON Mining S.A. were combusted. The remaining not used part of the slimes had to be traditionally stocked.

The investigative and investment activities were successful. As the result the installment to produce low calorific granulated fuels from coal slimes was built in ZG Sobieski. The obtained powdery product allowed to mix it freely with other fuel in the appropriate proportions. In such way the problem with the consistence of the slimes limiting their applications was solved. Additionally, the granulate produced in the installment did not cause problems with storage and transport what in case of slimes disqualified this material to apply in larger scale.

From the beginning of the production of low-calorific granulated fuels about 150 thousand tons of slimes were processed, which normally would end on the landfill site. From the beginning of 2014 such installment works also in ZG Janina, where apart from granulates for energy also the production of the granulates to develop into soil objects with addition of binding materials produced on the basis of ashes from the plant of the Group TAURON takes place.

In such way even the less energetically attractive slimes are being processed into products.

Similarly as in the case of production of the aggregates, also the production of granulates on the installments being properties of TAURON Mining S.A. allows to transform about 90% of produced slimes into the product. The processing potential will allow in the future to fully use the whole extracted coal limiting negative influence on the natural environment and the costs of conducting mining activity.

For many TAURON Mining S.A. strived toward no-waste production through applying innovative technologies. It was noticed and awarded in the country and abroad.

The technology of producing granulates was awarded on international trade fair of Scientific Research and New Technologies "BRUSSELS INNOVA 2011" which took place in Brussels, where it obtained gold medal. Also, it was awarded with silver medal during 111$^{th}$ international trade fair of inventiveness „CONCOURS LÉPINE*" in Paris.*

In Poland, TAURON Mining S.A. was honoured, for applying technology of granulates and aggregates production, with prestiguous title of the INNOVATION LEADER 2012 which was the highest prize in XI$^{th}$ Polish edition of the contest organized by the Center of Innopvation of Voivodship Club of Technics and Rationalization in Katowice. Furthermore, it was awarded with the highest prize in category "ECOLOGY" granted by the competition jury of XXII$^{th}$ School of Underground Exploitation in Krakow.

**Conclusion**

In the following years, the proecologic activities of TAURON Mining S.A. will focus around using the potential of partnerships of the Group TAURON to work out the model of management of produced "wastes", mainly inside of the Group. In this purpose, the project Grey2Green was initiated between areas of energetic partnerships and TAURON Mining S.A., which idea is based on using the effect of synergy in the direction of wastes management. The realization of the project's assumptions is based also on applying innovative technologies in the planned investments of constructing special installments dedicated to process of various sorts of "wastes" and transform them into products.

Thanks to greener investments, realized in TAURON Mining S.A. it was possible to manage or process both own wastes as well these originating from TAURON Wytwarzanie S.A. Additionally, the planned investments as modernization of installments to direct the slimy-ash suspension in ZG Sobieski to the bottom of the mines in purpose of fire prevention as well implementation of innovative technology of complex "cleaning" of slime tanks will allow to manage totally the coal slimes. This concerns both slimes being produced directly as well these ones which are still stored in tanks what will bring back their functionality.

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# The Impact of Sampling Errors on the Accuracy of Mass Balance in the Coal Enrichment Process

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*Abstract*

Coal is a highly heterogeneous substance in terms of the inorganic and organic constituents and exhibits wide variability with respect to size and chemical composition of the particles. An estimation of the true value of the desired parameters of a bulk material, to certain of degree confidence, through analysis on a few grams of test sample is definitely a daunting problem. The basic purpose of collecting and preparing a sample of coal is to provide a test sample which when analyzed will provide the test results representative of whole material. In the paper, the results of comparison of the of coal sampling made by three independent laboratories (manual and automatically) are shown..

Key words: coal, processing, sampling, automatic sampler, sample error

**Introduction**

Sampling procedures for coal vary considerably depending on what the analyses are to be used for, which may include technical evaluation, process control, and quality control for international trade. Sampling of coal forms an integral part of the coal production route, yet it is often a misunderstood and underrated activity on collieries. It is a vital aspect of ascertaining that the coal provided to the customer is of the correct quality. Most coal quality parameters are determined using ISO (Polish Standards PN-EN).

The benefits of good sampling are difficult to quantify, but there are numerous examples of the costs and losses, in some cases amount of money. Poor sampling practice will influence outcomes, evaluations, estimations, and bottom line cash flows from the earliest stages of mineral exploitation and extraction.

Errors in determining coal quality are compared. For the example of the ash content, it is shown that the error in forming the samples is almost an order of magnitude greater than the error in sample analysis. The metrological characteristics and conditions of correct utilization of results from coal analysis are

 
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considered in the light of polish standard compatible with ISO 11648. The correct comparison of results obtained by different methods is described. An experience of third party sampling and analyzing of coal are shown.

The first step taken in the commercial coal research is collection and reduction of the sample. The appropriate method of sample collection and reduction has substantial influence on the result of the research. The accuracy of numerous research measurement highly depends on conducting proper procedures during sample collection and reduction. The most essential information that should be assessed while sample collection is the size of researched batch, the size of researched general sample and the number of primary samples and correct execution of the method of collection and reduction, in order to achieve laboratory sample.

In the standards of PN regarding coal sample collection, the size of the batches included in the research are given. In the standards a batch is defined as an amount of coal produced at the same time, in conditions accepted as the same. Thus, it is the definition that requires further specification. Although, it's easy to determine even conditions through the process of transformation, nevertheless conditions related to geology of collected deposits may vary significantly. Due to this fact, in situations when accurate quality assessment of the mineral is essential e.g. of its chemical composition, the frequence of primary samples collection should be increased.

Since 1979, Gy's theory is increasingly being used in all areas where sampling plays a role. This theory already plays a role in official sampling standards (e.g. some ISO standards, see e.g. Holmes, 2011) and work is in progress to implement it in the near future in even more official standards (see e.g. Esbensen, et al, 2009; Esbensen & Minkkinen, 2011). Of course it is to be welcomed when scientific theories are implemented in standards, if that helps practitioners to incorporate scientifically-based methods in their practice. But at the same time people need to stay aware that implementation in standards can lead to future uncritical and dogmatic application of the scientific theory concerned (Gy's theory in this case). And that would be unwelcome, especially if the theory still needs to be updated or revised.

**Assessment of measurement uncertainty**

The sampling of coal, performed manually or automatically, must extract a quantity of coal much smaller than the original lot but with proportionately the same characteristics qualities and quantities present in the entire lot.

The many different approaches to mechanical sampling allows, for many different applications and consideration of specific requirements. Although mechanical sampling is more precise than manual, not all mechanical methods collect equally representative samples. The sampler should include a means for taking a sample from the entire cross-section of coal stream.

The aim of mechanical sampling may be the collection of a truly representative sample of the coal.

The modern theory of sampling of particulate minerals essentially consists of a recent generalization of the procedures for applying and calibrating the formula usually known as 'Gy's Formula' for sampling variance control, in which the central concept is a parameter called the liberation factor.

ISO standards have been the principal guides for producers of mineral bulk commodities who produce to customers' specifications, whereas Gy's insights have been most readily accepted by minerals producers.

Sampling practices at different stages of mineral development from exploration, face sampling and grade control, ore processing and handling, metallurgical sub-sampling, point of sale sampling, and sampling in the laboratory are considered in the gold, platinum, ferrous metal, and coal industries. A summary of the impact of poor sampling in these industries is presented. Generally it appears that poor sampling practice is most likely to erode mineral asset value at the early stages of mineral development. The benefits of good sampling are considered, especially with regard to the financial implications of bias and error on large and consistent consignments of bulk commodities.

**Methodology of sample collection and reduction**

The procedure of sample collection variation measurement and the amount of primary samples required is described in the standard PN-EN and is based on calculation and comparison of the variance and standard deviation of repeatability and of variance and standard deviation of the sample. In coal research general sample is initially collected, then it serves to prepare further subsamples to the point when analytic or research sample is received (in the case of the research conducted on several samples).

The definitions of general sample, subsample and analytic sample are given in relevant research standards. Analytic samples are acquired through reduction of general sample, which subsequently are intended for particular research. In laboratory practice of coal enrichment companies, the most frequent operations are analyses of grain composition and ash content designation.

Reduction of laboratory sample to analytic sample intended for chemical analysis should be conducted through material crushing at indirect stages in the manner so that the mass of the sample is not smaller, at each stage, than the right border value for particular size of grains. For instance, if we have a sample of coal with granulation of 0-32mm, then we cannot reduce it (with the given granulation) to subsample with the mass lower than 2 kg. When subsample mass reaches border value, the material must be crushed to smaller size in order to enable further division. This method is used to ensure representativeness of the analytic sample in relation to laboratory sample.

The observation shows that during sample preparations to chemical analysis a common mistake is often made, i.e. reduction and acquisition of subsamples of definitely smaller size than from the standards included in PN-EN. In such circumstance, it is important to first crush the sample material and only later reduce it. Laboratory crushers are very useful for that purpose. It was also observed, that sometimes laboratory technicians, who use the quartering method, take only 1/4 of divided sample to further research, instead of 2/4. As research has shown, such way of conduct triggers substantial errors.

**Description of conducted research**

For coal samples collection there are very often used devices, which complete this activity automatically, along with fixed schedule. In one of the companies, which deals with black coal transformation, the work of a device, used for aggregate sample collection from conveyor belt, was evaluated. The scope of works included manual crushed coal collection from conveyor belt, on which there was an automatic sampler installed and results comparison of their research of collected samples was processed automatically.

An analysis of grain composition and ash, sulphur content and calorific value was carried out. There were 4 series of sampling. About 150 kg mass samples in total were collected in each series, three of them were picked manually from car and the other one was collected automatically in processing plant.

***Sampling systems*** can be relatively simple and inexpensive.

On-line coal analyzers began to be developed in the late 1970’s and early 1980’s in the United States, Australia, and Europe (Woodward).

Most analyzers were one of three types:

- On-line moisture meters employing microwave technology,
- On-line ash gauges using gamma ray attenuation technology (collectively known as either dual-gamma gauges, dual-energy transmission (DUET) or low-energy transmission (LET) gauges), and
- Elemental analyzers used for ash, and sulfur, and sometimes ash constituent information as well.

These analyzers relied on prompt gamma neutron activation analysis (PGNAA) for elemental analysis, and they analyzed sample streams rather than the full process flow. When PGNAA is combined with a moisture meter, as is generally the case, moisture, calorific value and $SO_2$ content per million tonnes can also be determined.

The sampler used for investigation is build up in Coal Processing Plant Jankowice.

Scheme of automatic sampler is shown at Figure 1 and at Photo 1 and 2.

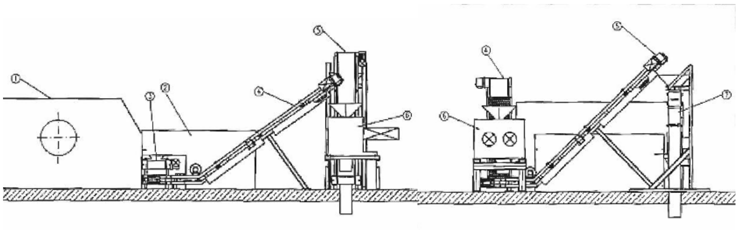

**Fig. 1. Scheme of automatic sampler**

**Photo 1. and 2. View of the on-line coal sampler**

**Results analysis and discussion**

It was concluded, through comparison of grain composition of particular samples, as well as through assessment of its size, that the grain composition of material collected by the automatic sampler varied partly from material grain composition collected manually from conveyor belt. In the material collected by the sampler there was less amount of the smallest and the thickest fractions. It could result from the fact that the vessel of the sampler could not reach the very bottom of conveyor belt trough and secondly – the biggest parts of the material could be rejected by the scoop (it is difficult to get into the inside of a scoop). Standard deviations in values of characteristic diameters d25, d50, d75, and d90, are close to each other, which denotes stabilized scoop work.

Grain composition of the samples gathered in the vessel after the process of quartering differed explicitly from grain composition of the samples collected from conveyor belt and the samples collected by the automatic sampler. The material accumulated in the vessel was smaller than the material located on conveyor belt and the one collected by the sampler. The values of characteristic diameters for this material are the lowest. Standard deviations for the values of characteristic diameters of the material accumulated in the vessel through the process of quartering are significantly higher than for the material samples from the conveyor belt and the sampler. It all indicates unstable and improper work of this part of the installation, where the process of sample reduction is conducted. Moreover, the quantity of the material accumulated in the vessel was lower from standard requirements for preparation of the sample designed for ash content analysis.

All these factors cause the sample designed to carry out ash content analysis to be unrepresentative.

In the process of sample reduction it is also important to maintain appropriate sample size dependantly from material granulation. It is recommended to crush material collected from conveyor belt prior to the process of its reduction.

In the described example, during one rotation of the scoop wheel there was 1,5 kg of material with granulation 0-31mm collected from conveyor belt into the vessel on average. This amount, regarding the data included in the standard PN-EN 932-1:1999, was substantially too low. Additionally, there was no crushing process applied in the described example, which is required before further division of the aggregate with 0-31mm granulation in order to acquire reduced subsample for chemical composition research.

**Own Research**

The aim of the study was to determine the error of analysis of the carbon taking automatically and manually by three independent laboratories. .

The study was power coal produced at the Coal Mine Jankowice and delivered to the IP Kwidzyn.

Quantity of the coal for individual sample was in each case about 2 500 Mg.

Two samples were collected manually from the railway wagon, one by automatical sampler installed in the coal enrichment plant. Samples were taking according to the polish standard,

The samples were collected by three independent (certified) units: KWK Jankowice, CLPB and IP Kwidzyn. Samples were analyzed in independent laboratories KWK Jankowice, Central Measurement and Testing Rovers and IP Kwidzyn, it determine the ash, sulfur constant and calorific value.

**Automatic sampler**

Because of the need to control the quality of the loaded coal is important to maintain full compatibility of the device for automatic sampling during loading of wagons with polish standard PN-90/G-04502. (Coal and lignite. Methods of sampling and preparation of samples for laboratory tests)

For this purpose on mark, annually, preparation and testing of samples in accordance with standard PN - ISO13909-7: 2005 (Coal and coke. Mechanical sampling. Part 7: Methods for determining the precision of the collection, preparation and research samples.) The following table (Table 1) shows the results of precision calculated in accordance with standard ISO 13909-7: 2005 for sampler built-up at processing plant.

**Table 1. Results of precision of automatic sampler in KWK Jankowice coal processing plant**

| **Parameter** | **Wtr** | **Ar** | **Str** | **Qir** |
|---|---|---|---|---|
| **Precision** | **%** | **%** | **%** | **%** |
| **P** | **0,1** | **0,76** | **0,01** | **285** |

**Results of investigation**

The example of results of investigation are shown at Table 2.

**Table 2. Results of investigation. Sampling date 27.05.2014 r. – sample mass 2.329,300 Mg**

| **Sampling place** | ***KWK Jankowice*** | ***KWK Jankowice*** | **IP Kwidzyn** |
|---|---|---|---|
| **Sample taken by** | ***KWK Jankowice*** | **CLPB Jastrzębie** | **IP Kwidzyn** |
| **Laboratory tests made by** | ***KWK Jankowice*** | **CLPB Jastrzębie** | **IP Kwidzyn** |
| | ***1*** | **2** | **5** |
| Calorific value **Qir** | 21 603 | 20 830 | 20 597 |
| Ash content **Ar** | 23,31% | 24,61% | 24,40% |
| Moisture content **Wtr** | 9,00% | 9,40% | 9,30% |
| Sulphur content **Str** | 0,43% | 0,39% | 0,38% |

**Conclusion**

Sampling and analysis of samples were conducted on a large general sample with a mass of approximately 2.500 Mg. All studies were conducted identical according to the polish standards. Research laboratories are certified ones.

Nevertheless, the results obtained are subject to errors - for the calorific value of the relative error does not exceed 4.8%, for ash content in operating condition is within the limit of 4.9–21%, moisture between 10–20%, and a sulfur content of 12–19%.

Designation of each quality feature of studied coal involves collection and reduction of the sample. The method of sample collection and reduction has crucial influence on research result. Specific research standards (PN EN) describe these procedures. Nevertheless, it is very important to have an experience of collection and reduction of aggregate samples in particular production processes. Study procedures give two methods of collection, calculation and interpretation of research results – manual and automatics). In some studies, it is common to average two results varied from each other within the value determined in the norm. In standardized research procedures there are also given standard deviation of repeatability and reproducibility of particular studies. Laboratory, by application of appropriate reference material, can determine its own accuracy measurements applicable in its conditions. The conducted statistical experiments of repeatability and reproducibility constitute crucial part of validation of the given research method.

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International Standard ISO 11648-2 Statistical aspects of sampling from bulk materials:

# Ash melting behaviour with respect to VM content of coal

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**Abstract**
Traditional way for energy production is coal fired power plants. Ultimate product of coal fired power plants is ash. Coal ash is valuable for operations like cement production and in future it might get more importance. Ash properties changes with parent coal and operational parameters. Change in ash properties results in slagging or fouling problems inside the boiler. High temperature of combustion and low melting point of ash are the main reason of these problems. In this study 3 coal samples with moderate to high VM (volatile matter) content are observed in terms of ash melting behaviors. Lower VM content of coal resulted as lower initial ash deformation point. With the increase of VM content, increase in the slagging factor was observed. As temperature inside the boiler increases slagging is more likely to happen, since ash is molt at high temperatures. High VM content of coal is a reason for the accelerated temperature increase inside boiler. Slagging and fouling potential of the ashes of these 3 pattern coal are obtained with different indexes. Ash fusion temperature index of slagging and fouling potential resulted as "low to medium" potential for lower VM content of coal, and "high to severe" for higher VM content coals.
**Keywords:** Ash melting point, Volatile matter, Slagging, Fouling, Coal fired power plant, pulverized coal combustion, ash deposition

## Introduction
Ash deposition, slagging fouling phenomena is getting more and more importance as fuel quality changes due to coal pricing, mine closures, government regulations market forces and etc. (Harding and Connor 2007). Many scientist (Naganuma 2009; Zbogar and Frandsen 2008; Harding 2007; Wigley 2007; Wee and Wu 2005; Wall et al 1993; and Yan et al. 2001) have been working on ash deposition in coal fired boilers. Chernetskii et al (2012) studied slagging inside a pulverized coal boiler and they proposed a mathematical model of slagging. According to their study different coals with different average mineral compositions undergo various transformations and as a result fly ash particles differ in composition aggregate state size and other features. Tillman and Duong (2007) investigated slagging problem at Monroe Power Plant in Michigan. In this study of Tillman and Duong (2007), with the help of on-line analyzer program they evaluate the coal or blend properties. On-line analyzer program analysis results are converted to some characteristics such as fuel volatility ratio, slagging alkalinity and etc. As claimed by Tillman and Duong (2007), volatile matter content of the fuel (coal or blend) is one of the important parameter in terms of slagging and fouling potential. According to Fang et al. (2010) three types of coal with VM content 4.5, 4.83 and 5.47 combusted and deformation temperatures of ash are 1450 C, 1220 C, and 1320 C respectively; which might or might not be a clue to any relation between VM content of parent coal and ash deposition. Mechanism of the ash deposition in the boiler is summarized in Figure 1 (Akiyama et al. 2011)



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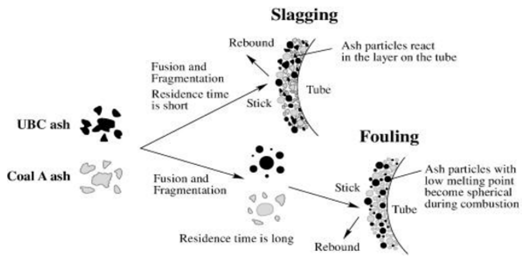

Fig. 1. Mechanism of the ash deposition in the boiler (Akiyama et al. 2011).

In this study, there types of coal with different VM content has been studied. Slagging and fouling potential of these coals have been investigated by employing literature empirical formulas. In order to calculate empirically the slagging and fouling potentials, ash deformation and ash elemental analysis were carried out. These three types of coal had been utilized in a power plant. Possible relations between VM content and slagging and fouling potential was investigated experimentally and plant based.

**Materials and experimental procedures**

The studied coal samples were provided from a local power plant. Combustion tests of the coal provided and experimental analysis were carried out on the samples provided. As experimental procedure, coal samples were analyzed in terms of proximate analysis. Coal samples were ground -75μm before analysis. Leco-S132 sulfur analyzer was used in the determination of sulfur. Moisture analysis was carried out on these samples. Coal samples were dried at 105 C for 24 h. Coal samples are coded C1, C2 and C3. Coal samples are burned at laboratory conditions and ash of the each sample was analyzed in terms of ash composition and ash deformation. Coal is burned at 500 C for 1 h and at 950 C for 2 h. Ash samples obtained after burning were coded as A1, A2 and A3 which corresponding to C1, C2 and C3, respectively. Since coal samples were ground before, grinding of ash samples was not necessary. Ash deformation temperatures were analyzed in the order of ASTM D1857. Ash composition analysis of each ash sample was realized in the order of ASTM D4326.

**Results and Discussion**

Proximate analysis of the coal samples is provided in Table 1. All of the coal samples has moisture content of below 10 %. The C1 coal sample has dry basis ash content of 20.95 % and its upper calorific value 5371 kcal/kg, however the C2 and C3 has lower (14.94 %, 13.152 %) ash content while upper calorific values 5160 and 5394 kcal/kg, respectively. Dry basis volatile matter contents are 28.61 %, 34.60 % and 37.95 % for each coal samples C1, C2 and C3 respectively. In terms of ash content C2 and C3 is very close (16.1, 14.23) however their VM content is almost 10% different than each other. Highest VM content is C3 and lowest one is C1; remarkable change was observed among each sample. In same perspective, the difference of VM content between C3 and C1 is 32 %. Elemental analysis of the ash samples was obtained after the combustion of the coal samples at laboratory conditions and it is provided in Table 2.

Ash melting behavior of the samples is also analyzed. This analysis is also carried out with the laboratory based combustion products of the coal samples. Ash melting temperature analysis result is given in Table 3.

Table 1. Proximate analysis results for each coal sample C1, C2 and C3.

| . | C1 | | C2 | | C3 | |
|---|---|---|---|---|---|---|
| | Dry | Original | Dry | Original | Dry | Original |
| Moisture (%) | | 3.60 | | 7.20 | | 7.80 |
| Ash (%) | 21.73 | 20.95 | 16.10 | 14.94 | 14.23 | 13.12 |
| Volatile Matter (%) | 28.61 | 27.58 | 34.60 | 32.11 | 37.95 | 34.99 |
| Fixed Carbon (%) | 49.66 | 47.87 | 49.30 | 45.75 | 47.82 | 44.09 |
| S (%) | 0.61 | 0.59 | 0.59 | 0.55 | 0.32 | 0.30 |
| UCV (kcal/kg) | 5572 | 5371 | 5560 | 5160 | 5850 | 5394 |

Table 2. Ash elemental composition for each ash sample A1, A2 and A3.

| | $Na_2O$ | MgO | $Al_2O_3$ | $SiO_2$ | $SO_3$ | $K_2O$ | CaO | $TiO_2$ | $Fe_2O_3$ |
|---|---|---|---|---|---|---|---|---|---|
| A1 (%) | 1.72 | 1.85 | 25.50 | 47.48 | 1.16 | 3.33 | 3.66 | 0.91 | 12.17 |
| A2 (%) | 2.06 | 2.69 | 23.57 | 44.73 | 1.51 | 2.81 | 8.64 | 0.87 | 10.20 |
| A3 (%) | 1.79 | 3.36 | 23.53 | 50.16 | 1.23 | 2.85 | 3.24 | 0.99 | 10.73 |

Table 3. Ash deformation temperatures for each ash samples A1, A2 and A3.

| Sample | DT ($^0$C) | ST($^0$C) | HT($^0$C) | FT($^0$C) |
|---|---|---|---|---|
| A1 | 1097 | 1192 | 1291 | 1275 |
| A2 | 1129 | 1156 | 1201 | 1275 |
| A3 | 1141 | 1189 | 1259 | 1362 |

Slagging and Fouling indices can vary with the original coal properties. The ash samples obtained after burning at laboratory conditions are analyzed in terms of elemental analysis and melting temperatures. Then slagging and fouling potential of ash samples of these coals was evaluated regarding the estimation method given in Table 4 Slagging and Fouling potentials were calculated for each ash sample A1, A2 and A3 and provided Table 5.

Briefly referring to Table 5, A1, A2 and A3 ash samples show “low” Slagging or Fouling Propensity with Slagging Factor 0.36, 0,58 and 0.36 respectively. In terms of Slagging Index A1 and A2 show “severe” Slagging or Fouling Propensity with Slagging Index of 1136 and 1143 °C respectively while A3 ash sample shows “high” Slagging or Fouling Propensity with Slagging Index 1165°C.

Combustion of these coal samples at power plant concerned was realized and ash deposition characteristics were observed. Only slagging and fouling trouble has been experienced with the utilization of C3, i.e. the highest VM content coal. According to calculation C2 and C3 has same slagging and fouling potential, but only trouble was with C3. That is why calculations are not always clear in terms of slagging and fouling potential since not only coal characteristics and ash composition are important but also operational conditions of power plant takes role on slagging and fouling problems. There might or there might not be relation between VM content but the only truth is medium VM content is better in order to decrease this slagging and fouling potential.

Table 4 Slagging ve Fouling Index Calculation methods (URL-1, 2015)

| INDEX | FORMULA | Slagging or Fouling Propensity | | | |
|---|---|---|---|---|---|
| | | Low | Medium | High | Severe |
| Ash Fusion | **Sphere (softening) temperature under reducing conditions** | >1350 | >1350 | <1350 | <1350 |
| Base-Acid Ratio | $\frac{Fe_2+CaO+MgO+Na_2O+K_2O}{SiO_2+Al_2O_3+TiO_2}$ | <0,4 veya >0,7 | <0,4 veya >0,7 | 0,4 -0,7 | 0,4 -0,7 |
| Slagging Factor | $\frac{Base}{Acid}\times Sulphur\ in\ coal\ (d\%)$<br>for bituminous ash ($Fe_2O_3$ > CaO + MgO | <0,6 | 0,6-2,0 | 2,0-2,6 | >2,6 |
| $T_{250°C}$ Temperature | $T_{250}\ °C = [\frac{(M\times10^7)}{\log(250)-C}]^{0.5} + 150$<br>where<br>$C{=}0.0415\times SiO_2+0.0192\times Al_2O_3+0.276\times Fe_2O_3+0.0160\times CaO-3.92$<br>$M = 0.00835\times SiO_2+0.00601\times Al_2O_3-0.109$ | >1400 | 1400-1245 | 1245-1120 | <1120 |
| Iron Calcium Ratio | $\frac{Fe_2O_3}{CaO}$ | <0,3 veya >3,0 | 0,3-3,0 | 0,3-3,0 | 0,3-3,0 |
| Iron plus Calcium | $Fe_2O_3+CaO$ | <10% | | | |
| Slagging Index, °C | $\frac{((\max\ HT)+4\times(\min\ IT))}{5}$<br>where max HT is the highest value of the hemisphere ash fusion temperature under reducing or oxidising conditions and min IT is the lowest initial deformation temperature under reducing or oxidising conditions | >1340 | 1340-1230 | 1230-1150 | <1150 |
| Silica Percentage | $\frac{SiO_2\times100}{SiO_2+Fe_2O_3+CaO+MgO}$ | 72-80 | 65-72 | 65-72 | 50-65 |

Table 5 Slagging and Fouling Calculated for each ash sample A1, A2 and A3

| INDEX | A1 | A2 | A3 | Slagging or Fouling Propensity for A1 | | | | Slagging or Fouling Propensity for A2 | | | | Slagging or Fouling Propensity for A3 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Low | Medium | High | Severe | Low | Medium | High | Severe | Low | Medium | High | Severe |
| Ash Fusion | 1192 | 1156 | 1189 | | | x | x | | | x | x | | | x | x |
| Base-Acid Ratio | 0.31 | 0.38 | 0.29 | x | x | | | x | x | | | x | x | | |
| Slagging Factor | 0.36 | 0.58 | 0.36 | x | | | | x | | | | x | | | |
| $T_{250°C}$ Temperature | 3313 | 2111 | 2569 | x | | | | x | | | | x | | | |
| Iron Calcium Ratio | 3.32 | 1.18 | 3.31 | x | | | | | x | x | x | x | | | |
| Iron plus Calcium | 15.83 | 18.84 | 13.97 | | x | x | x | | x | x | x | | x | x | x |
| Slagging Index, °C | 1136 | 1143 | 1165 | | | | x | | | | x | | | x | |
| Silica Percentage | 72.9 | 67.5 | 74.3 | x | | | | | x | x | | x | | | |

## Conclusion

Ash with low melting point or high temperature inside boiler causes slagging and fouling problems in pulverized coal fired boilers. In this study 3 coal samples with moderate to high VM (volatile matter) content were analyzed experimentally and plant based. Increase in VM content in coal might only be effective on Slagging Index, since highest VM content coal sample C3 has high slagging and fouling propensity while other coal samples has severe. This might be due to high VM content of coal results in quick combustion and accelerated increase in boiler zone temperature. Slagging and fouling indices has been widely researched and there are a lot ways to predict the propensity. However, still experiencing different operational conditions and coal characteristics, it is impossible to predict and model the exact slagging and fouling process. In this study, it has been emphasized that VM content of parent coal might be critical parameters in terms of coal characteristics affecting slagging and fouling potential. Ash fusion temperature index of slagging and fouling potential resulted as “high to severe” potential for each coal sample. The future research may be focused on wider focus on coal properties and with several more coal types.

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# Petrographic properties with respect to VM content of coal

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**Abstract**
Petrographic analysis of coal is most critical step that must be taken into consideration regarding pulverized coal combustion. Petrographic constituents of coal play an important role and petrographic constituents have different combustion activeness. VM (volatile matter) content changes with respect to the coal rank. Petrographic composition of coal is also is a function of coal rank. That is why, coal fired power plants not only take proximate analysis of coal on board but also they should conceive the petrographic analysis. In this study, 3 coal samples with different VM content respectively are analyzed in terms of proximate and petrographic analysis. Petrographic analysis resulted as lowest VM content (20.98 %) of coal has the lowest vitrinite percentage (20.8 %). VM content are close for the analyzed two coals (20.98%, 23.38%) and vitrinite composition are close (21% and 32 %) for them. Not only petrographic composition is evaluated but also the reflection measurement of these 3 coals is obtained. In terms of reflection measurement of these coals, two coals with lower VM has the biggest range (0.40-0.94) and the coal with highest VM content has the narrowest range of vitrinite reflection (0.50-0.79) respectively. In addition, microscopic observations are carried out and pictures are provided in the order of better understanding the effect of petrographic composition on combustion.

**Keywords:** Volatile matter, petrographic analysis, coal fired power plant, vitrinite. Pulverized coal combustion, proximate analysis, reflectance measurement

## Introduction

The combustion behavior o coals depend on the nature of the combustion conditions and coal characteristics such as volatile and ash content surface area and active sites minerals in ash and rank of coals (Lee et al 2009). Cloke and Lester (1994) claimed that petrographic analysis of coal can give useful information about the combustion of pulverized coal. According to Cloke and Lester (1994), not all vitrinites are reactive and not all inertinites are inert. Mineral matter, microlithotype and reflectance needs to be taken into account while determining the reactivity of chars (Cloke and Lester 1994). Tang et al (2005) claimed that measurements of the intrinsic reactivity of chars to oxygen are regarded as an indicator of the combustion potential of fuels by many scientist. Several parameters have been used in characterizing coal such as rank (Kleesattel et al 1987), fuel ratio (Thompson et al 1993), maceral composition (Lee and Whalery 1983), reactive maceral content (Furimsky et al 1990), and ash content (Wall et al 1989) (Tang et al 2005). Pulverized coal sample can consist of pure organic particles, pure mineral particles and particles consisting of mixtures of organic material and included minerals with different compositions (Wigley et al 1997, Liu et al 2005) According to Everson et al (2008) this is because of preceding coal preparation such as crushing, grinding and milling during which liberation of minerals from the organic matrix occurs. In the study of Malumbazo et al (2012) lower reactive content coal was

 
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releasing 10 % more of its volatiles in a reactor zone. In the same study of Malumbazo et al (2012) higher reactivity content coal produced a greater proportion of inert chars.

In this study, 3 coal sample which has been used for power generation in local power plant has been analyzed in terms of proximate and petrographic analysis. The effect of the change in volatile matter content, the change in reactivity was observed. Reflectance measurements as well as the petrographic composition analysis was carried out and the data of experimental findings of petrographic and proximate analysis was interrelated. The real life combustion of each coal sample was observed and the effect of either the reactivity or the volatile matter content in combustion was understood.

**Materials and experimental procedures**

The studied coal samples were provided from a local power plant which was to decide on a supplier. Combustion tests of the coal provided by supplier was realized while concerned power plant utilizing the coals and experimental analysis were carried out on the samples provided. As experimental procedure, coal samples were analyzed in terms of proximate analysis. Coal samples were ground -75μm before analysis. Leco-S132 sulfur analyzer was used in the determination of sulfur. Moisture analysis was carried out on these samples. Coal samples were dried at 105 C for 24 h. The difference in weight of dried sample and initial sample was proportioned and the percent moisture contents were obtained. Coal samples are coded as C1, C2 and C3.Samples of coal (C1, C2 and C3) were investigated petrographically. Microscopic studies of polished coal blocks were carried out under incident light by using a Zeiss Axioplan microscope and MPM 400 operating system. The microscope was equipped with a fluorescent light source and a camera. Fluorescent light is used in the identification of liptinites. Maceral group analyses were performed on mineral free basis. Random reflectance measurements were obtained from all the suitable vitrinite macerals. Polished block preparation, maceral group analysis, and reflectance measurement were carried out as described in ISO 7404–2, ISO 7404–3, and ISO 7404–5, respectively.

**Results and Discussion**

Proximate analysis of the coal samples is provided in Table 1. All of the coal samples has moisture content of below 10 %. The C1 coal sample has dry basis ash content of 12.42 % and its upper calorific value 6763 kcal/kg, however the C2 and C3 has 11.54 %, 11.84 % ash content while upper calorific values of C2 and C3 6754 and 6647 kcal/kg, respectively. Dry basis volatile matter contents are 32.49 %, 22.18 % and 25.05 % for each coal samples C1, C2 and C3 respectively.

Each coal sample has high amount of reactive organic constituents. While C1 coal sample has 81.6% total amount of reactive constituent, C2 and C3 coal samples has almost similar and 60-64 %. In terms of reactive composition each coal sample is convenient for utilization in power plants. Petrographic analysis result of coal samples C1, C2 and C3 is provided in Table 2.

Table 1. Proximate analysis results for each coal sample C1, C2 and C3.

| . | C1 | | C2 | | C3 | |
|---|---|---|---|---|---|---|
| | Dry | Original | Dry | Original | Dry | Original |
| Moisture (%) | - | 4.28 | - | 5.40 | - | 6.65 |
| Ash (%) | 12.42 | 11.89 | 11.54 | 10.92 | 11.84 | 11.05 |
| Volatile Matter (%) | 32.49 | 31.10 | 22.18 | 20.98 | 25.05 | 23.38 |
| Fixed Carbon (%) | 55.09 | 52.73 | 66.28 | 62.70 | 63.11 | 58.92 |
| S (%) | 0.41 | 0.39 | 0.32 | 0.30 | 0.35 | 0.33 |
| UCV (kcal/kg) | 6763 | 6473 | 7140 | 6754 | 7120 | 6647 |

Table 2 Petrographic analysis result of coal samples

| Petrographic Constituent | C1 | C2 | C3 |
|---|---|---|---|
| | % Amount (Volume based) | % Amount (Volume based) | % Amount (Volume based) |
| Vitrinite<br>Liptinite<br>İnertinite<br>Semi-inertinite<br>Pyrite<br>Mineral matter | 66.2<br>1.4<br>5.2<br>21.0<br>0.2<br>6.0 | 20.8<br>n.c.*<br>18.4<br>58.8<br>n.c.*<br>2.0 | 32.20<br>0.2<br>16.6<br>47.8<br>n.c*<br>3.2 |
| Total amount of Reactive Components | 81.6 | 60.0 | 64.3 |
| Random Average Vitrinite Reflection (%) | 0.618 | 0.771 | 0.61 |

*Not counted in the analysis.

Microphotographs of various constituents of coal samples are provided in Figures (Fig.1-Fig.14) Fig.1 to Fig.4 stands for the micro photographs of coal sample C1; while between Fig.5 and Fig. 9 are the microphotographs of coal sample C2 and Fig-10 to Fig 14 represents the microphotographs of coal sample C3.

**Reflection Measurements**

Histogram results of the reflection measurement of coal samples C1, C2 and C3 (V-types) are provided in Fig 15 to Fig. 17. Reflection measurements on vitrinite group macerals give information about the rank of coal concerned. In addition to this, histogram results of reflection measurements for coal samples can be helpful to understand whether the coal concerned is a mixture of one or more coal or not. Referring to Vitrinite reflection histograms (Fig. 15 – Fig. 17);

Among coal samples (C1, C2 and C3) C1 is not the mixture, i.e. C1 is one type. Average vitrinite reflection (random measurement) of coal sample C1 is 0.618, C2 is 0.771 and C3 is 0.61 and these coals can be classified as Subbituminous A

Coal samples (C1, C2 and C3) were combusted in power plant and unburned carbon and efficiency was analyzed in the field. Higher the volatile matter content resulted in the less amount of unburned carbon. Since only C1 coal is not the mixture, (referring to reflection measurements), utilization of C1 is the most suitable for power generation in terms of coal characteristics.

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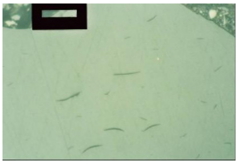
Fig. 1. Pseudo vitrinite

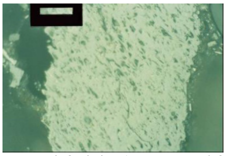
Fig. 2. Semi-fusinite (compressed form)

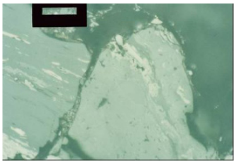
Fig. 3.Vitrinite and semi-fusinite

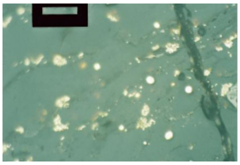
Fig. 4. Orthorombic pyrite in Vitrinit

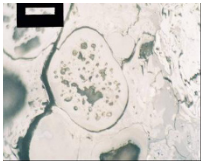
Fig. 5. Semi-fusinite (partially inert)

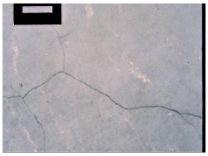
Fig. 6. Vitrinite

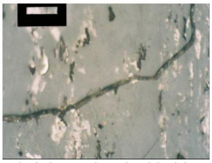
Fig. 7.Vitrinite and embedded inertinites

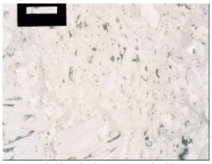
Fig. 8. Semi-fusinite (partially reactive)

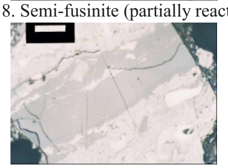
Fig. 9. Vitrinite, semi-fusinite and inetinite

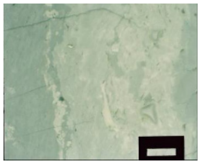
Fig. 10. Vitrinite and reactive and inert semi-fusinites

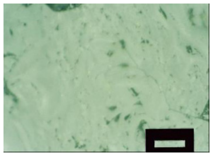
Fig. 11. Semi-fusinite

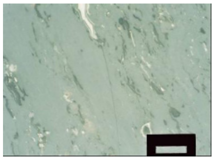
Fig. 12.Liptinite in vitrinite (sporinite)

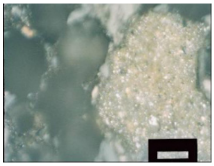
Fig. 13. Mineral matter

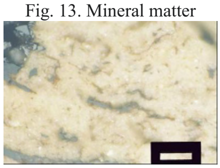
Fig. 14. Mineral matter and vitrinite

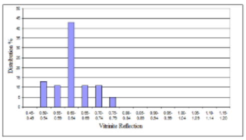
Fig. 15. Vitrinite reflection result for the coal sample C1.

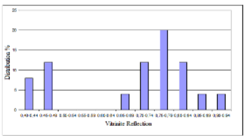
Fig. 16. Vitrinite reflection result for the coal sample C2.

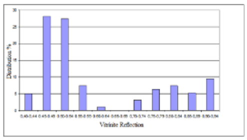
Fig. 17. Vitrinite reflection result for the coal sample C3.

Although operational constraints are still valid and various for each coal sample, higher VM content in other words higher vitrinite content (to some degree) resulted as the best coal characteristics for the utilization in coal fired power plant.

**Conclusion**

VM content and petrographic composition is a function of rank of coal. Although in a traditional way to determine the coal suitability for the utilization in coal fired power plants is proximate analysis, petrographic analysis of coal should be taken into account. In this study, 3 coal samples with different VM content respectively are analyzed in terms of proximate and petrographic analysis. The coals with lower VM content also have lower vitrinite content which also shows lower reactivity to some degree. As suggested, reflection measurement is more acceptable in terms of reactivity determination, reflection measurement for each coal was carried out. Coal combustion in pulverized coal fired boilers is a higgledy-piggledy process which all parameters of coal characteristics and operational parameters should be taken into account. Experimental set up to observe the effect of operational parameters is not possible. That is why each coal sample was combusted in local power plant and comparing out of 3 coal samples, highest VM content coal has the best efficiency and lowest unburned carbon in ash. Understanding of coal characteristics which play role more on combustion efficiency such as petrographic composition and volatile matter and the interrelation between these two was the main objective of this study. Future studies would better focus on this coal characteristics effect on combustion and the relation between proximate and petrographic analysis.

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# FACING THE CHALLENGES OF ULTRAFINE COAL RECOVERY

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**Abstract**

In the past there was often valid economic justification for not processing the ultrafine coal fractions and it was common to simply discard them as tailings. The trend is now to extract as much coal value as possible in the preparation process across all size ranges and to optimise performance and recovery. In addition, social, environmental and legal pressures to address concerns over power and water conservation drive incentives to maximise coal value recovery.

The issues associated with processing, dewatering and handling of fine coal and reject streams are complex and significant and call for better management solutions to qualify admission of fine coal into products. These include, maximised solids capture, improved operating performance, reduced water consumption and increased water recycling. The development of alternatives to existing processing and dewatering technologies along with the better understanding of existing processes are essential to achieving better ultrafine (<50micron) coal recovery.

In this paper, the authors review new and emerging beneficiation options. In addition to these, a number of established technologies have been improved and new versions of processes once considered unviable now have the potential to become solutions for the issues facing ultrafine coal treatment.

Keywords: Clean coal, ultrafines, recovery, optimisation, fine coal cleaning, dewatering, tailings.

**Introduction**

Although it is difficult to estimate the amount of coal lost from inadequate processing or reduced capacity caused by recirculation in coal preparation plants, there have been some reported attempts to quantify and provide cost estimation (Toney and Bozzato; 2014). A significant proportion of such losses occurs in the finer size fractions as the degree of difficulty increases and moisture issues become more significant. In addition to the fine coal permanently lost to tailings, ultrafine coal can often accumulate in fine coal circuits, generating a recirculating load of fine solids which must be accounted for in the metallurgical balance across unit operations and ultimately limits the amount of fresh feed to be processed by the fines circuit. This occurs within fine coal circuits which are all too often overloaded and often represents the limiting unit operation in the entire plant.

Commonly, this situation arises from inadequate input towards determination of the amount of fines likely to be encountered. This may have largely resulted from notional “cost savings” made in exploration and coal testing and analysis during the “front-end” stages of a project. In order to ensure that these errors are avoided, solutions are needed that include the introduction of a sampling protocol that incorporates recovery and testing of samples representing the full spectrum of coal types likely to be encountered by the coal preparation plant. From these programmes are generated things like the particle size distribution family of curves, coal cleaning washability and flotation data, moisture data and a host of other essential “design data” required for plant design. A range of surface area based models have historically been used for the prediction of the product moisture retention properties of various coal types (White, et al, 1996) , but more recently an Australian developed laboratory batch test for determining non-centrifugable moisture (NCM) is gaining increasing acceptance as a robust and reliable technique which has established that there are distinct limits to the final moisture contents that can be achieved for a

 
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coarse coal product (O'Brien, M; 2013). The technique provides quantitative information with regard to the dewatering characteristics coal after having passed through a conventional vibrating basket centrifuge. Research is continuing to extend this procedure for finer coal centrifuges.

Simulation based assessments of plant flowsheets and the associated implications for resource product quality and yield are utilised extensively through the various project study phases as well as the resource evaluation and optimisation assessments. Commonly used simulation tools for coal preparation plant flowsheet assessments are RESOURCE_MASTOR™ http://www.abmylec.com.au/campaigns/resourcemastor.html and LIMN® http://www.davidwiseman.com.au/. By using partitioning models for individual plant equipment items, these models can be used to predict product quality and plant yield for several optional flowsheets. Product moisture is an important consideration for export coals and moisture modelling techniques can be incorporated into these simulations.

A fresh approach is also needed for recovering ultrafine coal by a combination of effective cleaning, improved dewatering, and perhaps size aggregation, such as briquetting to ensure that the ultrafine coal component contributes positively to the total coal product. Many companies prefer to conduct a test sequence, commencing with laboratory or bench-scale, through pilot-scale and eventually to full- or commercial-scale installations. This enables reliable scale up determination often reducing the risk of under-designing the process requirements. In most cases, the adopted approach distributes the effort of designing the flowsheet and sizing the equipment weighted more towards the coarser than the finer fractions, which may not be the most prudent approach. This defines the familiar "top down" approach as seen in a typical flowsheet. The "bottom up" flowsheet, shown in Figure 1, represents a change in thinking that could prove beneficial towards avoiding previously common outcomes of underestimation of fines capacity during the plant design.

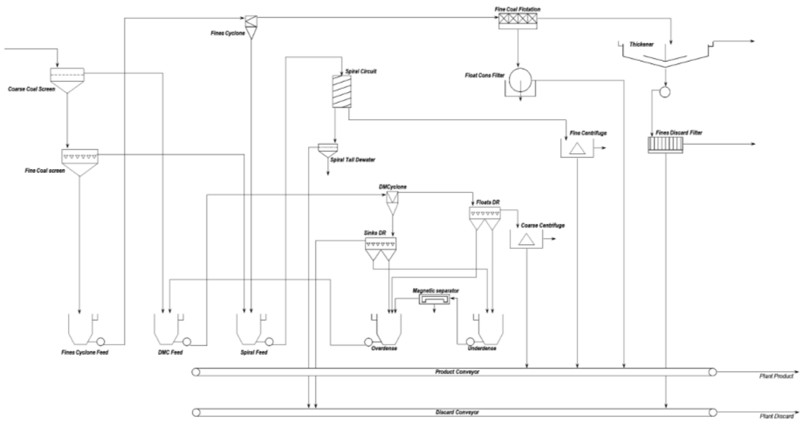


**Figure 1 Coal preparation flowsheet, viewed "unconventionally!** (Courtesy: David Wiseman LIMN)

## Resource Considerations

A root cause of inadequate "front-end" data gathering may be the complexity of the resource itself. There is little doubt that the "easier", deposits have been largely exploited and it is likely that "standards" developed for data gathering and plant design are largely based on these less challenging deposits. In contrast, complex deposits such as those involving multiple seams in a mineable coal sequence require a corporate commitment not to "cut corners" in drill-core processing and testing. Where a single seam deposit with a 5m thick seam will cost about $3,000 USD per hole, the multi seam deposit of perhaps 10 plies or more, similar to the Limpopo measure in Southern Africa, could be more than 10 times this cost. Added to this is the fact that fines generation has been increasing with mechanisation so the working section selection for complex "bar-code" stratigraphies and conducting the required amount of analysis will inevitably involve some cost "trade-offs" for these types of resources (Osborne, Sherritt, et al 2014).

The major sources of coking coal in Queensland are seams in the Moranbah and German Creek coal measures of the Bowen Basin. Many of these represent "easier" deposits and coals that have traditionally been proven as good sources of hard coking coal preferred by coke makers for many years. The Rangal coal measures in the upper part of the geological section, i.e., Blackwater Group, have more recently emerged as lower cost alternatives to the Moranbah or German Creek coals, but are more challenging to characterise and evaluate and exploration costs are significantly higher in gathering the data needed.

## Relevant Technologies

*Coal Cleaning Technology*

Water-based cleaning options available are listed in Table 1. Many of these have undergone a number of "development phases" extending over several decades. Recent advances in "by zero" fines flotation have enabled very high recoveries of fine coal to be achieved by either Microcell column or Jameson flotation technologies. In research aimed at producing micronized coal slurries as fuel alternatives, ultrafine coal sized below 20 microns produced via ultrafine grinding and floated in Jameson cells has resulted in low ash (~1%) and high recovery carbon level ideal for stabilised slurry fuel substitute for diesel (Wibberley, 2014). Another re-emerging process is selective agglomeration for which new techniques have been recently reported (Galvin, ACARP, 2015) promising viable commercial applications.

**Table 1 Equipment used for Fine Coal Cleaning** (Osborne 2012)

| Equipment | Feed Particle Size (mm) | Footprint/ Unit size mm | Through put (dry solids) TPH | Operating condition | Feed Preparation | Comments |
|---|---|---|---|---|---|---|
| Dense medium cyclone (fine coal) | 1.0 – 0.1* | 250 - 450 dia. | 10 - 45/ unit | 70 – 120 kPa | Desliming | Commercialised but medium recovery issues |
| Water-only cyclone | 2.0 – 0.1* | 150 - 660 dia. | 5 – 75/unit | 70 – 100 kPa | Desliming | Needs two-stages or spiral second stage |
| Spiral incl. fine application | 2.0 – 0.05 | 650 - 750 dia. | ~2.5/start | 7 - 8 $m^3$h feed, 25 – 30% solids | Classified feed | Low unit capacity per pole; 2-stage common |
| Teetered-bed Separator | 2.0 – 0.2 | 7.5 $m^2$ | 25 | ~110 $m^3$h; 30-35% solids | Classified feed | High capacity per unit area footprint |
| Reflux Classifier | 3.0 – 0.2 | 2.4 x 1.9 $m^2$ | 30 - 50 | ~110 $m^3$h; 30-35% solids | Classified feed | High capacity per unit area footprint |
| Mechanical flotation cell | <0.5 | 28 - 100 $m^3$ | <5 t/h/$m^2$ | 3 – 10% solids feed | Classified feed | Several stages 4 – 8 are essential |
| Column cell | <0.25 | Up to 4,270 dia. x10,000 | 1.5 – 2 t/h/$m^2$ | 3 – 10% solids feed | Accurate top-size cut | Recirculation effective |
| Jameson cell | <0.25 | Up to 6.5$m^2$ dia. with 24 downcomers | 1.5 – 2 t/h/$m^2$ | 3 – 10% solids feed | Accurate top-size cut | Recirculation effective |
| Spherical agglomeration | <0.25 | No commercial units | No commercial units | High-shear mixer ~5,000rpm | High feed solids conc. ~50% | Chemical cost limit application |

*original DSM proposed range – some Ep overlap finer size encourages alternative solutions to be considered.

Application of dry beneficiation, the nirvana of many mineral processing engineers, has been further stimulated by exploitation of new deposits in arid areas, is. Progress as always, will probably evolve from proven processes in the coarser sizes progressing to the finer end of the particle size range. A number of potential technologies have been researched including triboelectric electrostatic for cleaning highly liberated powders (Honaker, 2007). Evidence of this trend is the growing body of research work underway in China particularly involving variants of fluidised bed, dense medium separators (Chen and Wei, 2003). A dense-media vibrated fluidized bed (DMVFB) for separating dry fine coal particles (minus 5mm) from unwanted gangue particles was reported by Luo et al (2008). The results also show that, in the particulate bed of fine coal, most of −1+0.5mm size fractions of coal and a small portion of −0.5mm size fraction of high ash coal, contributes to the formation of an autogenous medium bed. This could prove to be the breakthrough needed to take this technology into commercial operations.

*Dewatering Technology*

Dewatering options available are listed in Table 2. Recent advances in "by zero" fines dewatering has enabled total moistures of fine coal to be reduced to below ~18%. This is the target whereby it can become viable, to include flotation concentrates into a thermal coal product.

**Table 2 Equipment Used for Dewatering Coal (adapted from Bickert's chapter in Osborne 2013)**

| Equipment | Footprint Size | Throughput (dry solids) | Product Moisture (wt %) | Feed Preparation | Application |
|---|---|---|---|---|---|
| High frequency screen | 0.6-2.4 x 3m | 10-100tph | 15-25% | Cyclone U/F | Fine Coal |
| Screen scroll centrifuge | 0.5-1.5m dia. | 45-100tph | 11-18% | Cyclone U/F | Fine Coal |
| Horizontal vacuum belt filter | 75-150m$^2$ | 50-130tph | 20-30% | Flocculation | Ultrafine Coal |
| Screen bowl centrifuge | 1.1m dia. x 3.3m long | 20-60tph | 16-27% | Thickening | Ultrafine coal |
| Solid bowl centrifuge | 1.1m dia. x 3.3m long | 15-20tph | 15-20% | Thickening | Ultrafine coal |
| Disc filter (including large dia. units) | 120-200m$^2$ | 50-150tph | 20-32% | Thickening - flocculation | Ultrafine coal |
| Hyperbaric disc filter | 70-200m$^2$ | 30-150tph | 17-25% | Thickening - flocculation | Ultrafine coal |
| Paste thickening | 25m dia. x 6-12m high | 100tph | 45-55% | Flocculation | Coal tailings |
| Solid bowl centrifuge | 1.1m dia. | 20-60tph | 30-45% | Thickening | Coal tailings |
| Belt press filter | 3-3.5m wide | 10-20tph | 25-45% | Thickening - flocculation | Coal tailings |
| Filter press | 200-800m$^2$ | 15-30tph | 14-32 | Thickening | Ultrafine coal and tailings |

## Where to from here?

Research addressing the specific coal processing needs, addressed earlier is mostly being carried out in the coal producing countries. Based on the number of peer-reviewed publications, China and India appear to be leading the way, together with Australia and to a lesser extent South Africa and the USA. A useful way of analysing the value proposition is the matrix plot shown in Figure 2 which compares the level of innovation for a research activity with the probable time to implementation (Osborne 2010). Several companies have developed matrixes using for tracking and prioritising projects. Referring to the matrix:

- H1 – S1: derived from a mining company's current projects, including short term R&D programmes such as ACARP (Australia), CoalTech (South Africa) & locally contracted collaborative R&D.
- H2 – S2: Strategic projects impacting on the entire value chain, including some contracted R&D with in-house research groups and external – usually the shorter term. projects
- H3 – S3: Challenging projects impacting an entire value chain including some contracted R&D with the in-house R&D group and external – usually only a few are in progress

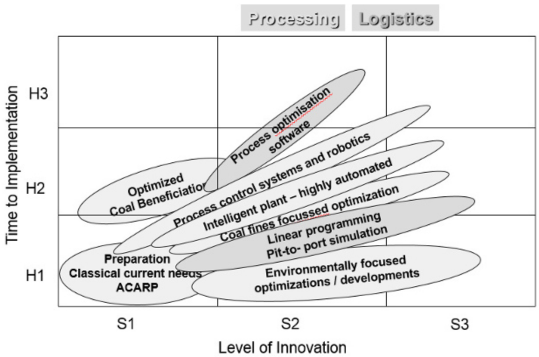


Figure 2: Research and Technology Objectives (including examples)

When progress over the past 5 decades is reviewed, very few new coal cleaning technologies have emerged, just better designs of existing technology. To a lesser extent this is also true for dewatering technologies which have switched from vacuum filtration to centrifugal techniques. This has been enhanced in many cases by the potential introduction of new materials, leading to improved maintenance and longer component life; and the implementation of better control and monitoring systems. Rewards from this have been optimised operation and greater efficiency. Further improvements in both areas remain a major R&D and process operational challenge (Osborne 2010).

Some more specific examples for fine coal circuits include the following:

- Screening / classification dominates plant design; need more efficient, higher capacity units.
- Cost effective and efficient dense medium cyclone (DMC) application, or other accurate processes.
- Improved classification, 0.10 - 0.25 mm. i.e., better hydrocyclones, sieve bends, or something new.
- More predictable and controllable flotation and/or agglomeration technologies.
- All centrifuge technology (screenbowl, solid bowl units) for "by zero" fine coal dewatering.
- Continuously operating pressure filters, screw presses, or centrifuges for tailings dewatering.
- On-line slurry sampling, coal characterization / species recognition with effective feedback control.
- Commercialisation of high capacity, high resolution dry sorting technologies.

## Conclusions

Some of the challenges to creating improvements in fine coal recovery have been with us for a very long time but there are still questions that need to be answered by more innovation, risk taking and further technical development:

- Coal cleaning has tended to match the "cost of use" to customers; and fines recovery has remained in the "too hard" basket.
- Development has stagnated as proven designs and old engineering approaches have been preferred.

- Serious impacts such as water restriction; instead of driving solutions via dry cleaning technologies, or circuits with maximized water recovery, have still not been adopted.
- Emerging clean coal technologies have not been adopted following properly planned plant trials.

Other, more wide sweeping challenges have emerged which pose serious threat to the future for coal:
- Speed to commercialization – ownership and intellectual property issues slow down the process.
- Waste disposal and management – tailings ponds, toxicity, burning discards and run-off impacts.
- The urgent need to ensure the required influx of engineers and scientists to rise to these challenges.
- Climate change implications – impact has largely been passed on to the end-user.

Solutions must include
- A "whole-of-coal supply chain" approach, i.e., all cost components and $CO_2$ impacts considered together, and fully integrated, not individually stated.
- Climate change implications - need to be considered as opportunities, not threats.
- Coal preparation engineers and scientists must recognize and develop new technology to achieve improved energy efficiency.
- In many cases more intensive coal cleaning, via greater liberation can be justified.
- Introduction of "Communities of Practice" to provide a basis for sharing knowledge and working together especially with coal users.

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# Online Slurry Particle Density Meter

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**Abstract**
This paper reports on the long term operation of an Online Slurry Particle Density Meter (OSPDM) at the Bloomfield CHPP in New South Wales, Australia. This unit has been operating continuously for 36 months, providing real time process monitoring data to the operators. OSPDM outputs include slurry density, solids concentration on both mass and volumetric basis and **average particle density.**
The operational reliability of the OSPDM was a major point of investigation. The installation of innovative backflush systems and other measures has allowed the unit to run for the 36 months without operator intervention.
From a plant perspective, the OSPDM has been an invaluable instrument which has enabled irregularities in the flotation circuit performance to be identified which sampling had not and could not readily identify. This was achieved by close monitoring of real time tailings composition on a minute by minute basis. Variations in the tailings stream with respect to particle density and solids concentration were traced back to other issues in the plant. The OSPDM has allowed cell optimisation with respect to reagent dose rate in real time. The OSPDM with online feedback can also negate the need for large sampling campaigns and analysis during plant optimisation programs.
**Keywords**; Online, instrumentation, particle density, measurement, control, CHPP, non-nuclonic

## Background

The Online Slurry Particle Density Meter (OSPDM) is a process measuring device which measures exactly what its name implies; particle density. The OSPDM provides the average particle density by using a patented method. The original theory behind the OSPDM is given in Lambert (2007), but more accurate means are now used and the conversion to ash value is not always performed. This is due to the errors in the determination of ash as discussed in Lambert and Campbell (2012 and 2013).
The OSPDM uses the interaction of the particles and the slurry with vibration frequencies and with a range of microwave frequencies. The equipment can then be calibrated to particle density, after the effect of conductivity and other factors are accounted for. These corrections must be made and these must be taken into account to provide accurate and reliable results.
The OSPDM has gone through many stages since its inception. This has included conceptual design, equipment selection and refinement, laboratory trials, onsite trials and permanent onsite installation.
After much time spent building and rebuilding equipment to suit specific needs, developing more accurate laboratory tests and making the entire unit as robust as possible, the OSPDM is now an accurate and reliable commercial unit.
The OSPDM is a non-nucleonic device and thus there are none of the safety issues concerned with nucleonic devices.

## Bloomfield Installation

The Bloomfield CHPP flotation circuit consists of a single Jameson Cell and the OSPDM was installed on the tailings from this cell. The outputs from the OSPDM provide the operators with the average particle density for the cell tailings material, which can be used to give an indication if saleable coal is

 
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being lost from the flotation circuit. The OSPDM provides online measurement, which can be monitored in real time during circuit optimisation. This quickly allows changes, whether good or bad, to be identified immediately without having to collect samples and wait for laboratory analysis to return results.

**Summary of other Systems**

There is a long history of flotation control systems, but all seem to have been left by the wayside, such that today's coal flotation plants are largely controlled "by eye" and by sampling and laboratory analysis. Often, these analytical results are produced some time after the event and rarely within two hours. This often means that the analytical results may not be useful for controlling the flotation. As flotation has a great many factors that affect the performance, it also means that the time delay may not make sampling very useful in process optimisation.

Other automatic control systems that have application on flotation and fine coal include:

1. Autoflote – a system developed by Century Oils (now Fuch's) that used turbidity to determine the "ash" of particles within a stream. The monitors were used on flotation feed, product and tailings and used to automatically control flotation circuits. Gallagher et al (1988)
2. ASHSCAN – a nucleonic device that used different sources to detect the solids concentration and the "ash" of the particles in slurries. Clarkson et al (1985) and Ellis et al (1988).
3. AMDEL Coal Slurry Ash Analyser (CSA)– a nucleonic device developed by the CSIRO which uses X-ray back-scatter to detect "ash" and solids percent within the presented slurry. (Nicol, 1997).
4. Conductivity determination of ash, as described in Mohanty et al (2013).
5. Froth Camera systems. Developed by JKMRC and Metso.
6. Tailings Colour Detection – several systems. Colour is only determined by the very finest solids and streams can have a "good" tailings colour but still contain large amounts of coal.

Many of the systems above have been installed and tried and many removed. Often the systems had poor reliability and poor accuracy.

**Sampling**

Like any measurement, it is only as good as the sample it receives. Prior to installation, the best possible sample point had to be investigated.

Ideally, a representative sample would be obtained by a full stream cut of the flotation tailings stream, but this is not always feasible in an existing CHPP. Another option is to use multiple sampling spears flowing into a vezin sampler although adequate head room is required for this option.

Another option is to install a single spear into the flotation tailings stream. This is the simplest option, but better sampling will lead to better results.

**Calibration**

Like any measurement device, the OSPDM requires accurate calibration to provide accurate results. If the calibration information and data is poor, then the measurement or instrument output cannot be accurate and in some cases is flawed.

A great deal of time has been spent ensuring the OSPDM receives the most accurate calibration possible. This starts with the sample collection from the OSPDM. If this is done incorrectly, there is no point in proceeding further.

The entire stream which passes through the OSPDM measuring tube is collected for **15 residence times.** All values measured from each instrument in this period are logged, collected and averaged for each sample.

This ensures that **all** material which flowed through the measuring tube during the calibration has been collected and forms the sample to be analysed in the laboratory.

Once a sample has been collected, the laboratory analysis is the next stage of calibration. Laboratory analysis is as important as the sample collection, as errors here can also render the data useless.

The sample is not subdivided after collection. Incorrect laboratory subdivision can be another source of large errors in the sample analysis process. In the CPT method, all of the sample material is analysed.

With the 'Australian Standard' laboratory methods not being accurate enough for calibration requirements, more accurate methods were developed for determining slurry density and solids concentration by volume. These methods were verified by an independent laboratory (Steel River Testing) as being accurate methods for these determinations.

During the verification program, five samples were taken at each plant condition and analysed by three different laboratories. The two laboratories using the 'improved' laboratory method analysed two out of five samples from each run and a commercial coal laboratory performed the 'conventional' particle density (AS 1038.21.1.1 - Relative Density) analysis on the remaining sample from each condition.

**Reliability** – Initial problems, rectifications, duration, extended reliability.

As well as providing accurate results, the unit needs to be reliable and operate with as little user or operator intervention as possible. The initial designs used an orifice plate to control the slurry flow through the OSPDM. This led to constant blockages due to either large, oversize particles or high solids concentrations blocking the orifice. A unique interference valve was therefore designed and incorporated into the OSPDM, which uses clarified water to control the slurry flow rate without using a physical restriction. In addition, physical restrictions (orifices) are also prone to wear and blockages when either oversize material and or high solids concentrations are present. With the use of an interference valve, these shortcomings were removed.

Although the OSPDM at Bloomfield was designed to operate on the flotation circuit, having a nominal particle size of 100 to 300 microns, "stray" oversize material of greater than 20 mm was periodically experienced which the unit was not initially designed to handle. As the unit is required to operate on whatever particle size that maybe be experienced in the fines circuit, changes to the OSDPM feed system were made. These included oversize protection which stops extremely oversize particles entering the feed system and an automatic back flush on the feed, which is also used to flush and clear the sampling spear when blockages occur. Figure 1 below, shows a sample of misplaced, oversize particles that periodically flowed through the OSPDM and were subsequently caught on a grate below the units discharge pipe. Whilst an unusual event, tramp and stray oversize particles are a reality of most fines circuits. These are often experienced after plant maintenance days or screen panel change outs, and equipment used in the fines circuit must be capable of handling and passing extraneous, coarse material on a routine basis. The OSPDM has now proven itself capable of readily coping with this oversize material.

The automatic back flush system senses a low flow condition using a flow meter and automatically forces clarified water back through both the measuring tube and feed system to remove any blockages.


*Figure 1 - Oversize material present in flotation tailings which passed through the OSPDM.*

**Outputs**

The OSPDM measures parameters such as slurry density and solids concentration, as well as particle density and these can also be provided as outputs from the unit. These can also be key indicators for equipment performance and used for general process stream monitoring and control.

Bloomfield have found the solids concentration output allows the fines circuit loading and fluctuations within the rest of the CHPP system to also be monitored as these impact the feed to the flotation circuit and therefore the tailings stream.

Figure 2 shows the OSPDM outputs on a Supervisory Control and Data Acquisition (SCADA) system. The various outputs are shown as trends with varying conditions. It can be seen quite clearly that the tailings particle density; increases when collector dose is increased, it decreases when collector dose is decreased, it decreases when frother is decreased and falls away markedly when the reagents are turned off. All of these are expected changes, but they are monitored in real time. Obviously, cell level, air flow rate, vacuum and other conditions could also be changed and the change in tailings quality similarly monitored.

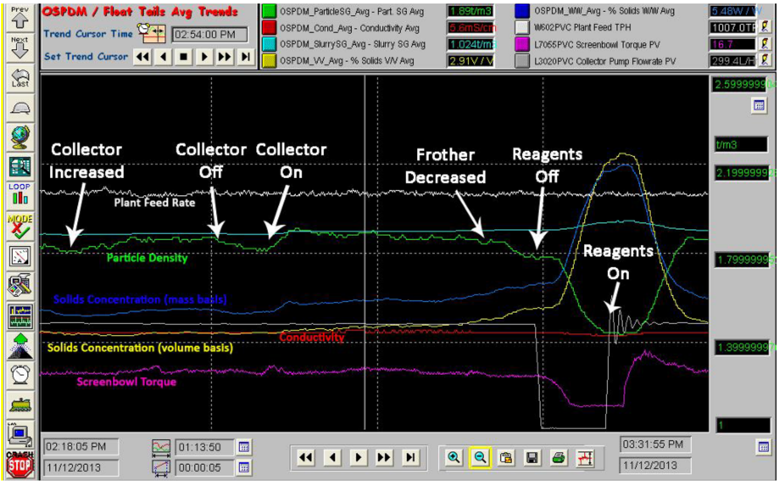


*Figure 2- SCADA system screen showing OSPDM outputs tracking operational changes.*

**Payback calculator**

Payback calculations are given in detail in Lambert et al (2014) and summarized here. The first is for performance optimisation. This is given for a theoretical situation where the average particle density in the tailings is increased from 1.90 to 2.10 t/m$^3$. This can be achieved by monitoring the average particle density and using this information to optimise flotation performance. In this case, for a 160 t/h feed flotation plant, the OSPDM installation would be paid for within six days. This assumes a coal sale price of $100/t.

The second payback calculation is from detecting near flotation failure. This is given for the same hypothetical situation, but where the OSPDM detects that the flotation is "near" failure. "Near" failure is where the flotation is producing froth and visually the flotation appears to be operating normally, but in fact, it is producing very little product coal. Froth may still be produced, but, it may not be heavily laden and operators may not immediately detect that very little coal is being produced. This has been observed to occur when, for example, the collector is not dosing due to a blockage, but the SCADA system still shows that collector is being dosed. In this instance, the OSPDM would pay for itself in 18 hours. This 18 hours could be made from several instances where "near" failure (due to a variety of causes) occurs and the OSPDM alerts the operators in a more timely fashion than current detection techniques.

Bloomfield staff have also noted that simply having a monitor on the flotation tailings has led to increased flotation performance as it is known that the flotation tailings are monitored in real time, 24 hours a day and the flotation circuit cannot simply be ignored even when it is appears to operating satisfactorily.

An important aspect of any process monitoring device is the ability to view the outputs. The OSPDM is captured on the Bloomfield CHPP's SCADA system (again, see Figure 2), which displays the outputs as raw values and also as a rolling three minute average. The raw values can provide information on stream variability over a short period of time, whereas the averaged data shows sustained trends over longer periods of time. Both sets of values are very useful and care should be taken when deciding to average or 'smooth' the outputs from any process measurement device.

**Conclusion**

The OSPDM has helped identify issues in how the flotation circuit was operated with respect to air supply, the operation of control loops, variations in feed solids loading, reagent addition with the loss of collector as a result of operators failing to transfer all reagent lines with a change of plant set up / feed / operating philosophy, and changes that may occur in reagent addition due to irregularities or calibration issues with the reagent dosing system. All the above have been identified as issues when the flotation tails are constantly monitored online by the OSPDM. These are "coarse" operational improvements which when eliminated or managed can allow quantifiable process "refinements" to be made to the flotation circuit.

Although the OSPDM is currently only installed on the flotation tailings stream, it can also be used for monitoring of other fines streams within a processing plant. With the ability to handle large particle sizes and with the incorporation of the back flush system, the OSPDM can also be utilised to monitor all streams around fine coal circuits such as spirals, teetered bed separators and reflux classifiers.

The potential for the OSPDM on these other fine coal processing units is particularly powerful as these units perform density based separations and they are often poorly controlled, if at all.

In a perfect world, all streams within a CHPP would be monitored for all important parameters. In the real world, the majority of process streams have no measurement at all. The OSPDM allows fine coal slurry streams to be monitored constantly and knowing the average particle density of a stream allows the process to be optimised as well as facilitates the monitoring of coal losses and changes in feed quality. This is achieved on a reliable basis without the need for large, costly sampling campaigns and the subsequent wait for the return of the laboratory results.

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# Coal Hydrophobicity and the settling behaviour of coal fines tailings

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**Abstract:**

India is blessed with enormous coal reserves (one fifth of the world reserves) and is characterized as high ash and difficult to wash. With a 66% share of installed power generation capacity (255 GW, plus about 40GW captive power plants), the coal industry has a major role to play in the nation's development. The coal consumption in 2013-14 was 643 MT against the indigenous production of 462 MT (mostly from Coal India Limited). The coal washing capacity from all the 52 washing plants is 131Mtpa, indicating that around 30% of the coal is being washed before it is used for power generation. It is envisaged that by the end of $12^{th}$ five year plan coal washing capacity will be increased to 250 Mtpa.

In India wet coal washing is usually practiced by crushing, sizing followed by beneficiation. The coal fines are usually recovered from the thickener, were flocculants are added to increase the settling of the coal fines. Normally, the coals are hydrophobic in nature and affect settling processes to a large extent. In the present paper four Indian coals of different characteristics are taken and their settling behavior in presence of industrial flocculent was investigated and correlated with the hydrophobicity of coal.

Key words: coal washing, coal fines, effluents, contact angle, hydrophobicity, settling, flocculent, etc



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**Introduction:**

India is blessed with enormous coal reserves (one 5th of the world reserves) and is characterized as high ash and difficult to wash. We have to depend on the coal for power as the reserve indicated. About 85 percent of the Indian coals are of power coal. With a 66% share of installed power generation capacity (255 GW, plus about 40GW captive power plants), the coal industry has a major role to play in the nation's development. About 72 percent of coals required for power generation meet through the Indian coal producing companies and rest of the gape filed up through Import. The coal consumption in 2013-14 was 643 MT against the indigenous production of 462 MT (mostly from Coal India Limited). The coal washing capacity from all the 52 washing plants is 131Mtpa, indicating that around 30% of the coal is being washed before it is used for power generation. It is envisaged that by the end of 12th five year plan coal washing capacity will be increased to 250 Mtpa.

The Indian coals are of Gondwana origin in which the coaly matter and the ash forming minerals are intimately mixed having high NGM and wet coal washing is usually practiced by crushing, sizing followed by beneficiation. In the processes of beneficiation about 10 % of fines were generated. The coal fines generated are usually recovered from the thickener, where flocculants are added to increase the settling of the coal fines [1-2].

Normally, the coals are hydrophobic in nature and affect settling processes to a large extent. Many researchers had worked out to get clarified water to recirculate the washeries water [2-5]. The process of settling is such a complex phenomenon which still needs to study. In the present study four Indian coals of different characteristics are taken and their settling behavior in presence of industrial flocculent was investigated and correlated with the hydrophobicity of coal.

**Materials and method:**

The two borehole core samples were crushed individually at 13 mm top size and screened at -0.5 mm for the study while the other two samples from the washery were collected as per the standard [7-8]. All the four sample were sub sample for the end use. The representative parts of the collected coal fines/ slurry were characterized for proximate analysis as per IS standard [9]. The determine ash range of the coal fines varied from 35.1 to 59.1% while moisture content varied from 1.5 to 3.4%. The contact angle varied from 72.6 to 86.3 degree and the petrographic reflectance value varied from 0.49 to 1.02 [10]. Their general characteristics are depicted in Table 1.

Table 1: Properties of the coal samples

| Properties | Coal-A | Coal-B | Coal-J | Coal-K |
|---|---|---|---|---|
| Ash% | 35.5 | 35.1 | 59.1 | 46.7 |
| Moisture% | 1.5 | 1.7 | 3.4 | 1.5 |
| Contact angle (degrees) | 84.8 | 72.6 | 74.0 | 86.0 |
| Ro Value | 0.85 | 0.76 | 0.49 | 1.12 |
| Surface area ($m^2/g$) | 6.3895 | 13.2931 | 30.3883 | 4.5728 |

* The contact angle measurements were conducted on the powder coal fines sample using capillary rise (Washburn method)

**Flocculants Used**

Nine flocculants that were used in the investigation were collected from manufacturers and also from coal washeries. The details of the flocculants used in the study are given in Table 2.

Table 2: The flocculent used

| S.No. | Flocculants |
|---|---|
| 1 | SUPERFLOC16 (off white granular solid) |
| 2 | LABUFLOCC 300 (off white granular solid) |
| 3 | LABUFLOC N 036 (off white granular solid) |
| 4 | LABUFLOC C 301 (off white granular solid) |
| 5 | CAT floc N-8102 plus (Pale Yellow Liquid) |
| 6 | NALCO N-83384 (white amorphous solid) |
| 7 | NALCO (off white liquid) |
| 8 | CRISTOL T50 (Clear, Pale Yellow liquid) |
| 9 | CRISTOL N50 (Clear, Pale Yellow liquid) |

**Results and Discussion:**

Wet sieve analysis at 63µm size of the four coal fines samples were carried out. Size-wise ash and moisture distributions for the samples and the screened size fractions were carried out following standards, [11] the results are depicted in the Table 3.

Table 3: Size and proximate analysis percentage for the four coal samples

| | Coal A | | | | Coal B | | | |
|---|---|---|---|---|---|---|---|---|
| Size (µm) | Wt% | Ash% | M% | VM% | Wt% | Ash% | M% | VM% |
| +63 | 79.6 | 34.3 | 1.4 | 23.7 | 77.4 | 33.3 | 1.6 | 24.7 |
| -63 | 20.4 | 41.4 | 1.4 | 21.6 | 22.6 | 42.5 | 1.9 | 22.2 |
| -500 | 100.0 | 35.7 | 1.4 | 23.8 | 100.0 | 35.4 | 1.9 | 24.1 |

| | Coal J | | | | Coal K | | | |
|---|---|---|---|---|---|---|---|---|
| Size (µm) | Wt% | Ash% | M% | VM% | Wt% | Ash% | M% | VM% |
| +63 | 66.0 | 54.8 | 3.1 | 21.9 | 63.9 | 43.8 | 1.6 | 18.0 |
| -63 | 34.0 | 67.6 | 2.1 | 16.2 | 36.1 | 51.5 | 1.9 | 17.6 |
| -500 | 100.0 | 59.1 | 3.0 | 20.1 | 100.0 | 46.6 | 1.7 | 17.2 |

**Sedimentation Study:**

The sedimentation behaviour of the coal fines below -500 micron size at 5 percent solid content is cumbersome monitor the settling particles by traditional jar test method. To overcome the problem an automated bottom loading balance was used with suitable cone for the collection of fallen mass of per unit area. The instrument is having automated computerized data recording facilities. The method adopted here was compared on coal slurries with the application of flocculent and confirmed with the traditional jar test [12-13]. The sedimentation test conducted at 5 percent solid content with distilled deionized water. Figures 1 and 2 are the natural sedimentation behaviour of the four coals for the – 500 µm and – 63 µm size fractions respectively.

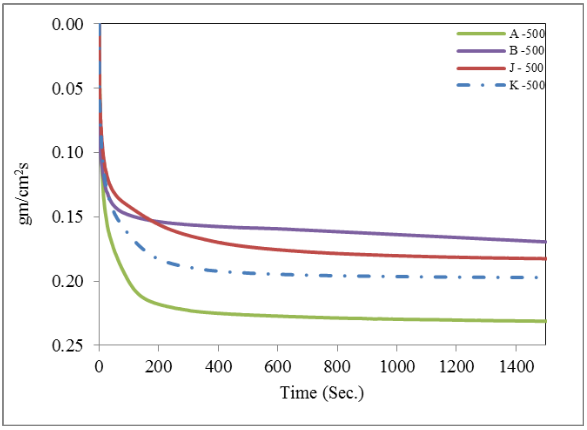


Fig 1: Settling behaviour of all the four coal of -500 µm size without flocculent

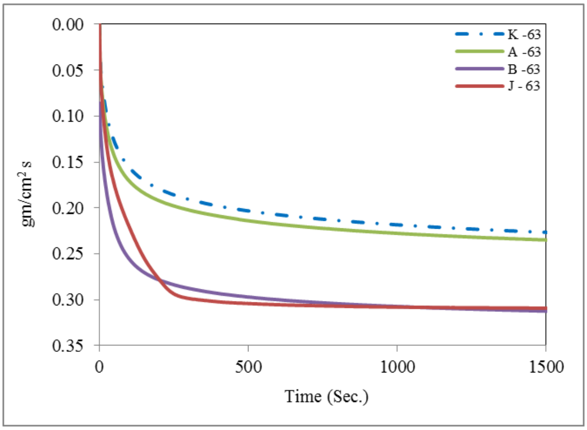


Fig 2: Settling behaviour of all the four coal -63 µm size, without flocculent

The settling behaviour of the four coal samples for -500 and -63 µm size fractions were conducted and is depicted in Figure 1 and 2. The sedimentation trend shows a typical sedimentation behaviour of the particle, that normal sedimentation behaviour is directly proportional to its size, i.e. finer the size, slower the particle will settle, heavier the particle faster the settling rate. Further it may be observed from the figure 1, that the sedimentation behaviour of coal J and B are similar even though they have significant difference in their ash percentage coal J having ash 59.1% is nearly almost double the ash percentage to that of coal B 35.1%. The -63 micron size fraction also varies significantly. The contact angle for coal J and B noted to be 74.0 and 72.6 almost similar which indicate that it plays an important role in the sedimentation of coal fines. On the other hand BET surface area of the two coals varies significantly the coal J having more than doubled the surface area of coal B. The ash and moisture percentage and the BET surface area of the coal J is almost double while the reflectance value of coal J is lower than the coal B. The contact angle of coal B recorded to 72.6 and coal J was 74.0 i.e. nearly same, exhibit similar sedimentation behaviour.

Similarly the coals A and K differ in ash content by 11 units. The petrographic studies of coals A and K show comparable reflectance value. The BET surface area the two coals is not varying much and the contact angle showing 84.8 and 86.0 almost similar the identical coal hydrophobicity, exhibit similar sedimentation behaviour even though the ash content differ significantly.

The sedimentation behaviour of finer size fraction -63 μm of the four coals are depicted in figure 2 showing sedimentation trend in normal condition. One may observe from the figure 2, that the sedimentation behaviour of coal J and B are similar even though they have significant difference in their ash percentage coal J having ash 67.6% while coal B having ash only 42.5%. Although the ash is reported highest with the coal J for finer fraction – 63 μm size observed to have the slowest rate of sedimentation in compared to other three coal fines and exhibit almost similar sedimentation behaviour with coal B – 63 μm size.

Similarly the finer fraction (– 63 μm) of coals K having higher in ash content by 10.1 units than the coal A. The petrographic studies of coals K is in higher side show comparable reflectance value with A. The BET surface area the two coals is not varying much and the contact angle showing 84.8 and 86.0 almost similar, the identical coal hydrophobicity, exhibit similar sedimentation behaviour.

The finer fraction, -63 μm size of coal J may be observed to have the slowest rate of sedimentation in compared to other three coals even though the ash content reported highest foe coal J (Figure -2).

The XRD and ICP data indicate presence of higher percentage of clay mineral in case of coal J and this may be one of the reasons for the slower rate of sedimentation. Further coal B had petrographic reflectance near to that of coal J show, exhibit similar sedimentation behaviour as J (Figure 2). For both the sizes tested, the contact angle (coal hydrophobicity), petrographic properties and the surface area of fines dominated the sedimentation behaviour rather than ash percentage and percentage of finer fraction in the sample.

**Conclusion:**

This observation can lead to the conclusion that the hydrophobicity and coal rank plays an important role in the sedimentation behaviour of coals.

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# Assessment of quarry of reopened mine Čáry based on the mining right of the Slovak Republic and European Union

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This article is aimed to determine the lignite market price in the reopened mine Čáry in Western Slovakia. A new method of quarry assessment based on the mining right, standards of the European Union and the Slovak republic has been used simultaneously. The Mine Čáry with its lignite deposit is a very perspective raw material source to produce power. The Lignite from this deposit with its low sulphur content and stable heating capacity is suitable to burn down in fluid boilers, which meet the ecological criteria. The raw material policy of the Slovak republic reflects the all-society interest to exploit the energetic raw material henceforward. The lignite mining in the mine Čáry started from driving of opening and preparing headings in 1989 Exploitation from the first long wall started in 1991, when 37 495 t lignite was exploited. During the whole existence of the Mine Záhorie, presently Mine Čáry, total 3 838 545 t lignite was exploited from 1991 up to 2006.

**Key words:** Lignite, brown coal, deposit evaluation, cash flow, mining economics, mining company, raw material policy.

**Introduction**

The purpose of the mining proposed activity is lignite extraction. The mine Čáry with its lignite deposits is a very prospective source of raw material for electricity generation. Lignite from this source has very low amount of sulphur and stable heating value. Due to these qualities it is suitable for burning in fluid boilers that meet ecological criteria. The Raw material Policy of the Slovak republic expresses all-society interest in mining of this mineral fuel henceforward. The Government of the Slovak republic, resolutions No.356 / 2005 and No.639 / 2006 approved generating electricity from domestic coal as a general economic interest.

**Physical-mechanical, technological properties and size of the mined mineral**

Chemical – technological analysis of Dubňany seam show, that there are no significant qualitative changes. Lignite is more or less constant concerning not only quality but also thickness, so that the shed occurrence does not interferes significantly the homogeneity of seam structure taken as an entity. Lignite of Dubňany seam is chemically pure with minimal content of heavy metals. (Hrúziková, 2008)

Technological parameters of the seam in the area of interest are documented by the selected qualitative parameters:

- Average thickness 4,70 m
- Seam location depth 194 to 238 m
- water $W^r_t$ 43,44 %
- ash $A_d$ 16,71 %
- calorific value $Q^r_i$ 10,95 $MJ.kg^{-1}$
- density 1,228 $t.m^{-3}$
- Sulphur $S_d$ 0,9-2,64 %
- arsenic $AS_d$ 4,0-43,0 $g.t^{-1}$

According to the latest calculation of the reserves dated 1.1.1997, the state of reserves in blocks are depicted in Table No.1. 9.(Alekseenko, 2014, Hrúziková, 2008)

Table 1: Overall amount of reserves in IX Mining section



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| Reserves | Area ($m^2$) | Geological reserves (kt) | Mineable reserves (kt) |
|---|---|---|---|
| Original | 798 989 | 4 490 | 2 193 |
| Increase | 1 448 852 | 7 584 | 4 399 |
| **Total** | **2 247 841** | **12 074** | **6 592** |

According to § 1 of Mining Act of Slovak Republic, the reserves are divided as follows when talking about the degree of exploration:

*"(1) Reserved deposit reserves are classified according to the degree of exploration of the reserved deposit or its part and according to the degree of deposition relation knowledge, quality, technological parameters and mining – technical conditions into these categories:*

*a) verified reserves Z-1*

*b) probable reserves Z-2*

*c) supposed reserves Z-3." (according to bulletin of acts SR 6/1992)*

Overall reserves in 9th mining section are divided into groups as balance free and balance bound reserve categories Z-1, Z-2, Z-3 (Table 2)

Table 2: Reserve categorization into groups and categories

| Group | Category | Geological reserves (kt) | Mineable reserves (kt) |
|---|---|---|---|
| Balance free | Z1 | 3 048 | 1 890 |
| | Z2 | 7 584 | 4 399 |
| | Z3 | - | - |
| Balance bound | Z1 | 1 442 | 303 |
| | Z2 | - | - |
| | Z3 | - | - |

**Deposit mining**

There were stated 2 mining methods of:

- Main mining method – longwall on swamp by complex mechanization (mechanized shifting brace and mining harvester). This mining method ensures the exploitation of predominant part of lignite seam.
- Additional mining method – corking by punch harvester. This mining method ensures the exploitation of residual lignite seam pillars.

Wall coalface:

A pair of preparation corridors:

- Material corridor,
- belt corridor (collection).

These corridors are interconnected by wall breakthrough, in which mining complex is mounted and in which transversal vent stream is mounted. Mid-pillars left between coalfaces (length cca 20m according to pressure conditions) will be mined by blind wing after material or belt corridor ventilated separately. Protection layer in thill 0-0.2m and min. 1.0m in ceil is proposed to be left from overall seam thickness approximately 4.70m. (Tobisová, 2010)

Mechanization:

Complex mechanized coalface:

- mechanized brace BMV – 1, with long frame,
- mining harvest MB-9VM
- coalface scraper conveyor TH 604

Downstream drill rig:

- shredder DR 2 type
- collection scraper conveyor TH 601 suspended on ZD 24 A, advancing with wall coalface,

• TP 400 C after collection TH 601, with shortened length with advancing wall coalface, this TP transfers muck onto TP line of drill rig.

Punching harvester:

• for mechanized disconnection and charging of disconnected muck on following drill rig punching harvesters 4 PU and GPK are used.

**Current economical state of the Bane Čáry a.s. Company**

In Záhorie mine (previous name of Bane Čáry mine) a mine crash occurred in 2005. After 3 year break in Čáry mine the lignite mining was renewed. The company of Baňa Čáry, a.s. m received a state support in 2008 in the amount of 3.85 mil. €. It was the 1st phase of investment project focused on lignite mining recovery in Záhorie, which is approximately 30% of overall costs needed. Overall expected costs were 12 833 000 €. (These costs will be also used in mining appreciation in the next section). The mine wanted to create more than 6 kilometers of new mine corridors. The current state form profit point of view is not very positive. The receipt reaches 6 641 904 € in average, meanwhile the profit is 57 711€ to 1 048 028 € per annum. (391 141€ in average). The return of investment would be approximately 32 years with this economy when the deposit would be free of any usable mineral. (Kršák, 2012)

**The Results: Mining appreciation and the solution of current state**

**Cash-flow**

Incurred and gained money is cash-flow in general. Cash-flow of the organization, department, area, individual equipment or project is the sum of positive and negative items, receipts and costs joined with mining activities. The sum of financial flows which are the result of the investment into mining project are determined as cash-flow produced by capitalized investments.(Dujčák, 2014) The Cash-flow is a tool of financial analysis. The Cash-flow is an extremely important data for any commercial unit. Each shop or activity can bankrupt regardless of the fact how high profit shows the accounting result if the company cannot pay the invoices. The aim is to have positive cash-flow. The calculation of yearly cash-flow is made by following equation:

CF = RECEIPTS + CREDIT - INVESTMENTS – PRODUCTION COSTS – CREDIT PAYMNETS – INTEREST RATE PAYMENTS - TAXES (FIG. 1) (Cehlár, 2014)

Figure 1: New Cash flow Baňa Čáry

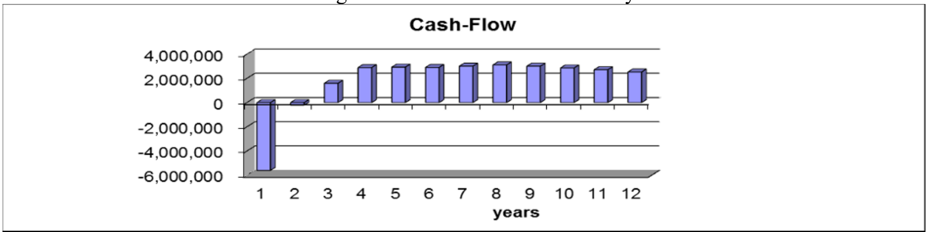
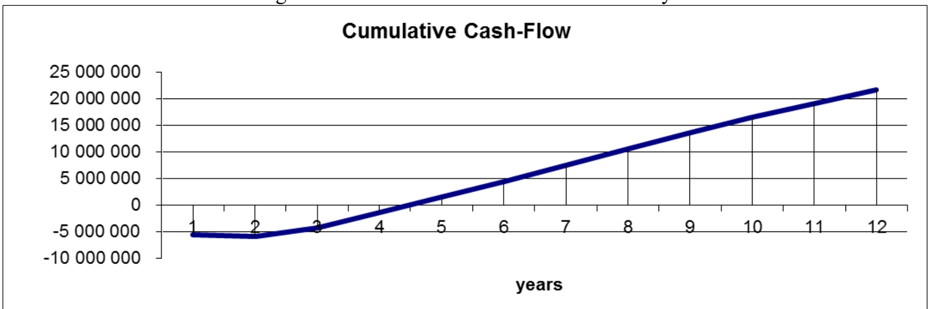



Figure 2: New cumulative cash flow Baňa Čáry



**Payback period (PBP)**

Payback Period is the duration of the project from its beginning until the cumulative cash-flow gets positive. Payback period is the expression of a capital return and for some projects it means the risk measure indicating how long the invested capital is at risk.(Mirdala, 2015) As in previous case, there is no unique full indicator as it informs about the project only until the period of return on investment. Although some projects' results may appear interesting based on payback period, the indicator does not tell about further project continuation from the cash-flow point of view. This can be positive or negative and it is unreadable from payback period. In addition, this indicator does not specify a development of the cash-flow in payback period duration in detail. (Cehlár, 2011)

**Net Present Value (NPV)**

Based on these facts, it is an advantage of finance receipt from the project in the shortest possible period. Money that can be gained by investment in 6 years are more sure than those that can be gained in 32 years. This is the reason why it is good to appreciate the project by yearly calculation of the cash-flow, which depends on project duration and it is determined as Net Present Value - NPV.(Cehlár, 2011)

$$NPV=\sum_{i=1}^{n}\{(-I+CF_i)/(1+a)^i\} \qquad (1)$$

where NPV - Net Present Value,
I - investment,
CF - cash-flow,
a - updating rate,
i - current year,
n - project duration length.

This is a way how to use cash-flow and its certain measures of uncertainty expressed in updating factor (1 + a) raised to the data distance from today. An option "a" depends on the interest rate at which investments can be obtained, or it is also possible to choose this value according to uncertainty of the future predictions. If the future with its effects on finances is possible to predict strictly, the value of "a" is lower. Thus, theoretical limit is „0“ – and update is cancelled. This case happens today.

Net present value can by illustrated by the following:

$$NPV=(R_0-C_0)+\frac{R_1-C_1}{1+a}+\frac{R_2-C_2}{(1+a)^2}+....+\frac{R_n-C_n}{(1+a)^n} \qquad (2)$$

R - receipts ,
C - costs . (Cehlár, 2011)

**Price of raw material**

In this case we are considering the price list which is published on the website of the Baňa Čáry a.s. Company (Table No 3)

Table No 3: The price list of the Baňa Čáry a.s. Company

| | | Quality marks | | | | | |
|---|---|---|---|---|---|---|---|
| **Type** | Granularity (mm) | $W^r_t$ | $A^r$ | $S^r_t$ | $Q^r_i$ | Sale price in EUR/tonne | |
| | | % | % | % | MJ/kg | without VAT | with VAT |
| Cube | 40-120 | 45,8 | 6,6 | 1,06 | 11,1 | 49,92 | 59,9 |
| Nut | 20-40 | 38 | 8,4 | 1,17 | 12,89 | 49,42 | 59,3 |
| Energy | 10-30 | 40 | 11,15 | 1,17 | 10,22 | 57,41 | 71,76 |

**Salary costs**

The salary costs reflect technical part of the project and consist of the average wage calculation according to the structure and number of workers as well as of wage costs. The costs of social and sickness insurance under the present legislation are considered in respect of the relevant legal provisions. The average amount of monthly remuneration is 1 121,12 € for 218 workers.

The valorization of salary represents a part of inflation (4%) and a measure which reflects the strategy to maintain the skilled and trained labour force and represents 1 % and summarily 5%. (Šoltés, 2015)

**Direct costs**

The direct costs of mining and processing amount to 30,7 €. The direct (variable) manufacturing costs in 2014 relating to the raw material processing are dependent on rate of production and processing method. Direct manufacturing costs for processing in accordance with the plan of Baňa Čáry a.s. Company to produce fractions Cube 40-120mm, Nut 20-40 mm and energy 1-30mm represent direct manufacturing costs for processing - 30,7€/t.

**Indirect costs**

The fixed manufacturing costs including administration and overhead costs and determined by the rate of production, the mining technology character for deposit of non-reserved mineral are 1 250 000€ (104 166- € per month). This will represent an annual increase of 6%.

**Other indicators**

This model requires the discount rate of 6% to be considered. The corporate tax in Slovakia currently stands at 22%. The share of investment of total costs of the implementation has been divided into the first three years at a ratio of 65-25-15%. 2% of sales of brown coal and lignite is included among indirect costs, which is a fee for mined mineral in accordance with the Laws in force of Slovak republic. Annual purchase is 250,000 tonnes per year (it is 3,000 000 tonnes for 12 years ). (Janke, 2015)

## Conclusion

In conclusion, the value of net profit would amount to € 13,139,127 for the mining period of 12 years according to the calculations. Return on investment would thus be reached between 4 and 5 year of the mining. The model that has been used deals with all aspects that affect mining. When it is used and be achieved primarily financial goals of the company but also the state. The company's and state's primary financial goals can be achieved by using and following the model. In the future, it is also possible to consider assessment and pricing of raw material deposits in order to support complex decision-making within the sustainable raw materials policy implementation in terms of economic, legislative, environmental and social impact on the feasibility and profitability of deposit extraction.

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# Evaluation and risk estimation by business with the earth's resources

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**Abstract**

The economics evaluation components of the feasibility study, therefore, must ultimately be based on information, which provides an answer to the question, "What is going to cost?" Unfortunately engineers preparing feasibility studies never have all the engineering or economic information they would like or need. Consequently, the economics portion of the analysis can only be performed if estimates of the various costs associated with project are made. Each project has an associated level of risk. For a proposed investment to increase the value of a firm's stock, it should have a higher expected rate of return than shareholders require for assuming that risk. Since investors demand higher potential returns from riskier project, the cost of capital depend on the venture's risk. Therefore, this must be quantified. The costs of capital also reflect interest payments on debt and the level of dividends and capital gains needed to satisfy shareholders. Given a proposed project's expected future cash flows, the risk level and the minimum rate of return required by the shareholders, the value of a potential investment can be calculated. Theoretically, his is accomplished by comparing the investment to others in the financial market.

**Key words:** Evaluation, risk estimation, drilling, earth's resources, management of mining, cost information, drilling machines

**Introduction**

The economic evaluation of a project requires a great deal of diverse information to be brought together in one place. The greatest concern is that there will be an error by omission, so it is useful to have a detailed list of what one needs to know in order to make a thorough evaluation.

There are varying of detail required at the different stages of evaluation in project, from the "quick and dirty" overview to the pre-feasibility study, to a full detailed feasibility study, to a due diligence review. Specially operating costs are too important for valuable judgement not only for future project but also for already existing project. All this problematic is oriented to the economic variables in project and can be used for all levels of studies. Its purpose is to identify a variable or issue and to raise a question, which the review can then pursue in more detail using increasingly more comprehensive checklist for each topic. While developed from the point of view of a new project, this list is equally valid for an ongoing operation.

**Capital and Operating Cost Estimation**

Before useful cost estimating procedures can be applied considerable data must be collected, compiled, and organised. Preparatory work typically includes organisation of cost data available for estimating purposes, development of techniques for updating historical cost, and establishing methods of forecasting prices and costs over the duration of the project. (Gentry, 1984)

**Order of Magnitude Estimates**

These cost estimates are generally intended to assist management in making appropriate decisions regarding potential project feasibility and to justify a further expenditure. of funds for the next stage of the project. Such estimates are sometimes suitable to reject a project, but are seldom adequate for positive project acceptance. The estimates are often based on known costs of similar projects and typically involve little or no design work for the mine and mineral processing facility in question. These estimates seldom become the basis for even conceptual design but may indicate the desirability of expanding work or efforts further (Gentry, 1984).



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**Preliminary Estimates**

The purpose of the preliminary cost estimate is to refine the order of magnitude estimate when additional data become available. These estimates are usually suitable to indicate or determine project feasibility and assist management in estimating a budget for the project. The estimate typically relates to a conceptual design of a mine or mineral processing facility (Gentry, 1984).

**Definitive Estimates**

The purposes of definitive cost estimates are to:

1. provide for appropriation of funds or to establish a contract price,
2. provide the basis for project cost status reports, and/or
3. establish a format for final cost reports to aid accounting and provide feedback information on actual costs (Puzder, 2015)

To use in future estimates and to improve existing estimating methods. An estimate of this type should enable management to authorise expenditures for completion of engineering specifications and drawings, design, and site surveys (Jurkasová, 2014).

**Detailed Estimates**

The detailed cost estimate culminates the estimating procedure. It is based on complete engineering drawings, specifications, and site surveys. This type of estimate is normally suitable for accurate projections and funding for the project and provides a basis for authorisation to proceed with construction and development. with particularly attractive projects, some construction may be authorised prior to completion of the detailed estimate, although this is not a recommended practice. A detailed cost estimate is seldom undertaken unless there is reasonable assurance as to project feasibility (Gentry, 1984).

**Kinds of Cost Information**

Cost information may be classified into three primary categories.

1. historical costs,
2. measured costs, and
3. policy costs.

*Historical costs* are those collected from the literature, accounting records, governmental cost information sources, business reports, trade associations, technical publications, etc. These costs are often obtained internally within the organisation and may represent data from projects previously funded by the firm (Domaracká,2011; Janke, 2015) .

*Measured costs* are defined as time - dollar relationships where direct observational processes and mathematical rules are followed. Measured costs are primarily limited to costs of work. Material quantities determined from drawings and specifications are also a kind of measured data. In general there are four methods normally employed for the determination of time, which are fundamental to determining measured costs (Gentry, 1984).

**Time Studies**

Time requirements for unit operational components of the entire production cycle are observed, recorded, and analysed with respect to $ per hour, $ per ton produced, etc. The following simple example illustrates the concept of measured costs derived from shift reports (Gentry, 1984).

***Example 1:*** A small fleet of 100-ton trucks was observed throughout the following cycle load, flat haul of 3000ft, spotting and dumping, return, waiting, spot and load. The average production for the truck fleet was 12,00tons per shift operating costs for the 100-ton truck fleet were as follows.

*Table 1 Operating costs for the 100 t truck fleet*

| Operating costs | Price/shift |
|---|---|
| Operating labor | $ 800/shift |
| Fuel and oil | $ 1 000/shift |
| Repairs | $ 600/shift |

| | |
|---|---|
| Tires | $ 1 400/shift |
| Total | $ 3 800/shift |

Therefore the cost per ton of material moved is:

$ 3 800/12100 tons = $ 0,314/ton.

**Work Sampling**

Observations are taken pertaining to specific activities of the person or machine at random intervals. Such a figure can be used in conjunction with other work samples to estimate the production capacity of machines in a given operation (Kršák, 2012). The following example shows how work sampling can be used to determine production estimates for a given machine (Jurkasová, 2011).

***Example 2:*** A work sampling study of an electric shovel in an open pit mine was performed. Some 400 observations were made of the unit and the shovel was found to be in a waiting or nonproduction mode during 55 of these observation (Pavlík, 2015). How much of the total time is the shovel expected to be waiting for one reason or another?

*Solution:*

Since work sampling is a statistical technique, the laws of probability must be followed. When *n,* the number of observations, is large the binomial distribution approaches the normal distribution. Therefore, the probability of even *i* occurring can be estimated from the number of observations (Stackhouse, 1979).

$$pi = \frac{ni}{n} \tag{1}$$

where *p* is observed percentage of occurrence of an event,
*i* expressed as a decimal,
*ni* is the number of observations of event *i*,
*n* is the total number of observations. in this case. *pi* =55/400 = 0,14 or 14%.

Using the standard error. of a sample percentage for a binomial distribution and the concept of confidence intervals, the following formula may be used:

$$\mathrm{I} = 2 \left[\frac{pi(1-pi)}{n}\right]^{1/2} \tag{2}$$

where *I* is the confidence interval obtained from the study expressed as a decimal,

$\alpha$ is the factor obtained from tables of probabilities for a normal distribution

The problem can now be expressed as follows for a 90% confidence interval ( $\alpha$=1.645 ):

$$\mathrm{I} = 2(1.645) \left[\frac{0.14(1-0.14)}{400}\right]^{1/2} = 0.057. \tag{3}$$

Therefore the electric shovel can be expected to be in a waiting mode 14% of the time $\pm$ 5.7% /2, or a range of 11.2% to 16.8%. (Oswald)

**Man - Hour Reports**

These reports include tie card reporting of working hours by type of work, activities, etc, cost distribution for labour can be obtained in this manner. It is important, however, that provisions be made in the recorded data for delays (controllable and uncontrollable), weather conditions, equipment malfunctions, work location, etc, if the information is to be of real value to the estimators (Domaracká, 2011).

**The Results**

The manner of data processing is chosen taking into account the data obtained from running records and also taking into account the requirement to find out prescriptive costs in real conditions. As real conditions can be understood the operation of two identical machines in different working sections of the same organisations.

The calculation contains concrete data on activities related to the formation of costs and concrete data on unit prices related to the above mentioned activities. The prices are obtained either directly from the machine operator or from the tables if a certain activity related to the origin of costs have not been realised yet.

Sometimes the costs given by operator contain the data related to repairs or to the solution of a problem and therefore price does not include the wages, it includes only the price of a spare part. The values given in tables are corrected and include wage costs which represent approximately 30% of the price of a spare part.

*Table2 Prescriptive costs*

| List of activities | Activity code | Unit price [Sk] | Number of repairs | Time [h] | Total performance [m] | Prescriptive cost I [Sk/m] | Prescriptive cost II [Sk/m] |
|---|---|---|---|---|---|---|---|
| Repair of Simbo | 1 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Pulling out rods | 2 | 2000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Exchange of hammer | 3 | 5000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Defect | 4 | 3000 | 0 | 563 | 18711 | 0.00 | 0.0 |
| Repair of pipe | 5 | 201.37 | 10 | 563 | 18711 | 3.58 | 0.11 |
| Lubrication of rig | 7 | 42.25 | 21.5 | 563 | 18711 | 1.61 | 0.05 |
| Exchange of adapter | 8 | 12 369.5 | 4 | 563 | 18711 | 87.88 | 2.64 |
| Maintenance | 9 | 2000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| PHM | 10 | 19.76 | 96 | 563 | 18711 | 3.37 | 0.10 |
| Drill rod | 11 | 1677 | 5 | 563 | 18711 | 14.89 | 0.45 |
| Exchange of drill point | 12 | 5000 | 0 | 563 | 18711 | 0.00 | 0.0 |
| Repair of drill mounting | 13 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Broken out | 14 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Hydraulics | 15 | 48.1 | 110 | 563 | 18711 | 9.40 | 0.28 |
| Oil terminal | 16 | 409.5 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Changing of rod | 17 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Broken holender | 18 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Repair of socket | 19 | 3000 | 1 | 563 | 18711 | 5.33 | 0.16 |
| Repair of gear | 20 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| Repair | 21 | 3000 | 0 | 563 | 18711 | 0.00 | 0.00 |
| | | | | | Total | 126.06 | 3.79 |

*Table 3 Drill tools*

| Date | Number of drill holes | Performance [m] | Unit price [$] | Prescriptive cost II [$/m] |
|---|---|---|---|---|
| 20-09-97 | 245 | 724 | 1677 | 2.32 |
| 11-09-97 | 505 | 1515 | 1677 | 1.11 |
| 3-10-97 | 752 | 2256 | 1677 | 0.74 |

| | | | | |
|---|---|---|---|---|
| 27-10-97 | 675 | 2226 | 1677 | 0.83 |
| 6-11-97 | 824 | 2472 | 1677 | 0.68 |
| 19-11-97 | 395 | 1185 | 1677 | 1.42 |
| 25-11-97 | 399 | 1197 | 1677 | 1.40 |
| 2-12-97 | 732 | 2196 | 1677 | 0.76 |
| 12-12-97 | 841 | 2523 | 1677 | 0.66 |
| 6-1-98 | 137 | 411 | 1677 | 4.08 |

| | |
|---|---|
| Average | 1.40 |

*Table 4 Adapter R32*

| Date | Number of drill holes | Performance [m] | Unit price [$] | Prescriptive cost II [$/m] |
|---|---|---|---|---|
| 2-09-97 | | 5162 | 12369.5 | 2.40 |
| 29-10-97 | | 3116 | 12369.5 | 3.97 |
| 13-11-97 | | 2801 | 12369.5 | 4.42 |
| 25-11-97 | | 3507 | 12369.5 | 3.53 |
| 12-12-97 | | 2846 | 12369.5 | 4.35 |

| | |
|---|---|
| Average | 3.73 |

**Conclusion**

The article has been oriented to describe estimation of the costs and specially describing the operating costs of two drilling machines in the iron ore mine. These two machines are new in the process of the extracting of iron ore. For this reason it was necessary to find out all possible costs in order to know influence of using drilling machines to all economy of the mining divisions.

Drilling machines are working in two divisions with not same conditions. So all data were judged from this point of view.

Management of iron ore mine and especially management of mining division now can effectively manage all operations and in same time can have review about discipline of attendance of drilling machines. Like results is decreasing of operating costs of drilling machines and can forecast possible costs in new conditions or in different style of using.

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# Part V
# Coal Deep Processing Technologies

# EFFICIENCY OF LOW- GRADE COAL PROCESSING DURING METAL SELECTIVE EXTRACTION

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Mineral coals are able to accumulate significant amounts of elements and impurities which often reaching significant industrial concentrations. This paper presents the recovery method of the extraction of some oxides of metals from low-grade coal. Available processing technology of low-quality coal with a high volatile and sulfur components are inadequate for metallurgy when using in conventional carbon composite method but for the cleaned coal can be applied to by product extraction of high quality oxides of various metals if treated with organic solvents. Certain coal grades of different deposits have similar composition.
Some of coal grades of different deposits have similar composition present the scientific and technical interest particularly to develop processing technology of metal containing coal. With the introduction of modern technology based on the individual fields one can develop the cost-effective production for rare metals extraction. Experiments were conducted to benefit the coal with high variety components under laboratory conditions and containing the greatest number of metals. Over the interfaced processing of metallic coals by counter current of coal organic acids and the oxides outfall of some rare metals could reach positive results.

**Keywords:** coal mining industry, coal benefication, metallurgy, low-grade coal, rare metals, ash, thermoplastic, coal chemistry, selective extraction.

## Introduction

Nowadays there is unstable situation on the world oil market and coal becomes important basic raw material source for the chemical and especially for the steel industry. Therefore the technologies which are associated with the non-energy use of mineral coal require substantial development. Is known [1] that mineral coals are able to accumulate significant amounts of elements and impurities and the percentage of which often reaches a commercially viable concentration for processing. Some coal deposits have high concentrations of certain metals (Ge, U, Au, Sc, Pt, Nb), the group of lanthanides and other elements. For instance, Russian coals have a higher concentration of metals in comparison with global coal Clarke [2] as well as lithophile elements typical for alkaline rocks (Zr, Hf, Nb, Y, lanthanides, Ba, Sr, and Be) and the siderophile elements (Sc, Fe, Cr, Ni, Co). On the other hand they are differ by the low content of chalcophile elements (Cu, Pb, Zn) and some rare and alkali (Li, Rb, B, V). With using of modern benefication technologies and high effective equipment on the base of the certain coal fields is possible to create a cost-effective production of rare earth elements (REE) by their extraction from the material. The analysis show that the extraction from low-grade coal of Ge, Sc, Au and rare metal complex (Ta, Nb, Zr, Hf, Y, lanthanides) is the most advanced [3, 4].

High concentrations of tantalum, niobium, zirconium, hafnium and REE were found in the coals of Kuznetsk basin in Russia [5]. The abnormal contents of which are also observed in coal and coal ash of Minusinsk and Kansk-Achinsk basin [6, 7]. Such element as scandium is the

 
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most potential to be extracted from coal and is typical for the lignite deposits. In Siberia there are several coal deposits that are suitable for the organization of industrial production of Sc. There is of some interest the developed coal deposit (Kansk-Achinsk deposits (Siberia) [8]. Thus, there are technological and economic preconditions for improving the quality of mineral coal due to its deep processing, using of various materials by selective extraction which allows to obtain additional coal chemical products and raw materials especially for metallurgy branch. The paper presents the selective oxides extraction technology of metals from the low-grade coal with value the highest metals that could be of scientific and technological interest.

## Experimental techniques

For action there are the existing methods of coal processing in the vertical shaft furnace (blast furnace) using the solvents that contain a hydrogen donor which require the carbonaceous material and has a high load resistance and thermoplasticity [9]. On the other hand hyperactive inorganic solvents are inappropriate for industrial applications as its quickly lose their property of hydrogen extraction while transferring. Nitrogen-containing solvents can also have selective compatibility with coal but nitrogen and extracted coal make stable connections between them. Thus, it is impossible to extract the solvent, and there is no possibility for its recycling. As result, there is existing technology of the low-grade coal benefication with high volatile components and sulfur are inadequate for metallurgic for using in conventional carbon composite method but for the cleaned coal acquisition may be used for the by-product extraction of high quality oxide indifferent kinds of metals if treaded with the organic solvents.

Experiments were conducted to deep processing the low-grade coal with high variety components under laboratory conditions which containing the greatest number of metals. The low-grade coal was used as a starting (primary material) because of the low thermal plasticity such as bituminous coal and anthracite coal. Five samples from each line on several Russian coal deposits were selected. The main selection criteria were a high content of certain elements and the use of which is rational for the production of ligatures and the alloys especially in nonferrous metallurgy. Table 1 shows the contents of the most important elements for extraction and the content of major elements for recovery.

Table 1 Content of some rare elements in different batches of coal.

| Coal batches | Content of elements g/t | | | |
|---|---|---|---|---|
| | Sc | Y | Sm | Yb |
| 1 | 6,4 | 35,5 | 5,4 | 1,5 |
| 2 | 8,7 | 33,4 | 7,8 | 5,7 |
| 3 | 3,9 | 15,4 | 2,6 | 1,3 |
| 4 | 8,2 | 13,76 | 2,2 | 1,1 |
| 5 | 16,0 | 17,0 | 2,5 | 2,1 |

Incurred samples (150-200 g) were processed according to the developed flow chart (Fig. 1) by an organic solvent-based naphthenic acid to give a carbonaceous material for later oxides recovery to metallized state. In some areas of the production cycle counter-current filtration technology was used.

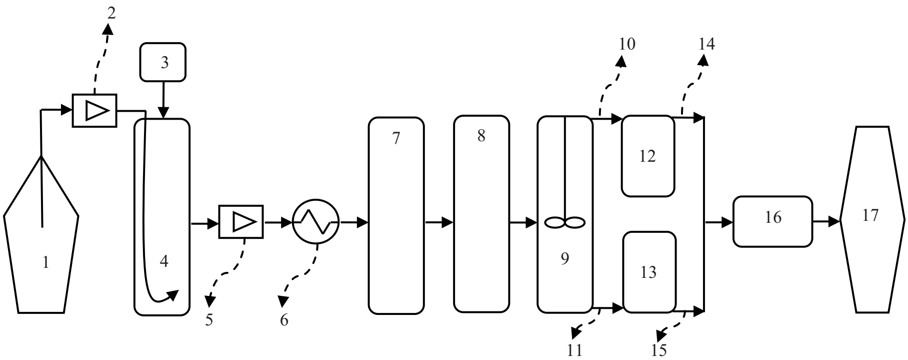


Fig. 1. Flow chart showing coal cleaning and metal flux preparation.

1 – solvent; 2,5 – pump; 3 - coal; 4 - slurry; 6 – heat interchanger; 7 - heater; 8 – holding capacity; 9 - sedimentator; 10- supernatant; 11- residual coal slurry; 12,13 – solvent removal device; 14 – counter-current filtration; 15 – coal extract and residual coal; 16 – clean coal 17- shaft furnace.

In the opening stage, the slurry of various coal grades was prepared (1-4) to determine recovery ratio for certain metals. After demetallisation, clean coal properties and the degree of its relevance for metallurgical production were studied. In the extracting stage, the solution was cured at a preset temperature and heating rate (6).

Throughout the process, the slurry was blown through by oxygen-air mixture, (7) (with an oxygen content of 35-40%) for components recovery and solvent removal from extraction mixture by evaporating (8) to give clean coal in solid shape (11). It was noted, that the elemental part of metallic oxides reacted to the form of extract. In the sedimentation stage (9), hardly soluble coal components were precipitated. Extraction slurry standing after counter-current filtration was conducted until solution liquid-liquid partitioning.

The upper layer was removed from the holding capacity (9) into the tank (12), wherein the solvent is generally concentrated, and residual coal slurry was evaporated until the clean coal in solid shape was given (15). In addition, the mixture underwent compounding procedure, whereon extracted coal and residual coal were mixed to give clean coal with high level of thermal plasticity. Useable organic naphthenate solvent contained aromatic compound with two rings and had boiling temperature of 250-300 ° C under normal pressure. The organic solvent was removed by evaporation, and then returned for re-using at the stage of the slurry preparing. Test portion of coal 100 ± 1 g, reduced to <1.5 mm was taken for coals cleaned out of metals testing for plastometry indicators. Thickness of plastic layer determination was conducted by plastometer [10].

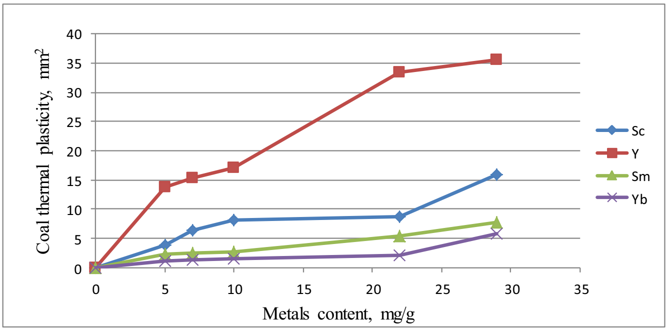


Fig. 2 Changing of coal thernal plasisity dependence on metal content.

Method of coal thermal plasticity determination is based on closed thermal heating and changing of coal properties to soften which it being heated airtight and to react to plastic state within the temperature range 350-470 °C and to form coke (coal carbonizing process) at further heating temperature lift. Figure 2 shows thermal plasticity level increasing interdependencies for different incurred samples. The highest thermal plasticity changing is observed when recovering yttrium and scandium to some degree. Ytterbium content changing slightly influences the coal properties.

Cleaning coal samples with metal oxides mixing were conducted for metals winning and concentrating by X-ray phase analysis. The resulting mixture was reduced in the laboratory revolver furnace. The following processes were undergone in this home straight: reduced mixture heating in the furnace for metal oxides agglomerating and then mixture transporting and loading into the shaft furnace, and at the ending melting to metal state for further dross separation in subsequent casting-form and separation. When it is using complex technology the maximum scandium recovery ratio were for example 1.2 mg/100 g, yttrium - 2.5 mg/100 g, samarium - 0.54 mg/100 g, ytterbium - 0.15 mg/100 g.

## Conclusions

One of the problems of coal industry are providing with non-ferrous and rare metals in Russia compound by difficult social and economic situation. It is possible to create profitable rare metals recovery production based on separate basis with using modern technologies. Scandium and some rare-earth elements extraction from the coal presents the advanced branch. Scandium resources only in Kansk-Achinsk deposits (Siberia) are able to provide its global needs for several decades.

The proposed process flow diagram of low-grade coal preparation may be adapted for any metallic coals. Complex coal mixture processing with organic solvents in counter-current filtration under various heat treatment conditions allows to concentrate metals in solvent along with their deep processing and to recover them in the other stage. Selected modes provide for oxides recovery and improve the properties of the coals especially their thermal plasticity.

Available resources are enough for creating large-scale production involved with preprocessing mining materials of Y, Sc and other elements and other elements based on any coal deposit.

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# Technological researches of coal mining waste with its processing and utilization to build-up production of constructional concrete in the north

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**Annotation**

The article reviews a current state of waste use in the coal-mining industry to produce porous fillers of light concrete, ceramic materials and products taking into account the modern market relations in the conditions of Far North territory development and the Arctic sea shelf as well. It solves an important environmental problem such as the reduction of harmful emissions of greenhouse gases from the smoking waste heaps of the coal mines which work with thermal power plant relating to monotowns of the Pechora coal basin, Vorkuta and Inta. The construction industry, considering the exhaustion of the mineral deposits, is able to recycle the sub-standard clay soils which litter the surrounding nature and it is useful to use them with technology cost reduction.

Agloporite is a material in the form of crushed stone, gravel and sand. It is received by means of agglomeration, but not by blowing-up of clay breeds and waste after production, processing and burning of fossil coals. On porous filler, agloporite, we receive constructional light concrete of 300-500 brands to produce covering and overlapping plates, building walls including monolithic housing construction, wall concrete blocks at the reconstructed operating production industries with batching plants. Their application will give the chance to increase heat-shielding qualities of buildings and to cut down transportation costs.

**Keywords:**

Agloporite, flyash concrete, waste of coal mining and coal preparation, processes of agglomeration and blowing-up.

Production of modern construction materials and the products is connected with mineral development including the rocks which compose the crust. They are divided on three genetic types. According to the formation, sedimentary breeds are subdivided into three groups and subgroups: a) detrital; b) clay; c) chemical and organogenic including aluminous (bauxites) and caustobioliths (peat, coal, sapropelite, combustible slate). At production and processing we have waste which is called mining waste. It accommodates at the related enterprises, waste heaps and in dumps. Its utilization promotes to develop new technological processes of mining branches.

In the second half of 20th and the beginning of 21st century the advanced countries use a high-strength light concrete with durability on compression of 50 … 150 MPa. It promotes to decrease a material capacity and increase a durability of building and construction designs in comparison with designs from heavy concrete of the classes B15 … B30. High-strength concrete appeared in Great Britain, the USA, Norway, Germany, Finland for high-rise buildings, bridge designs, oil sea platforms with a durability of 40 … 70MPa.

In Russia concrete of high durability are used insufficiently. Average values of durability at compression are twice lower, than in the USA, and 30-40% lower, than in countries of Western Europe. To construct buildings the maximum durability of concrete according to Construction Norms and Regulations 2.03.01-84 corresponds to the class B60 and in accordance with GOST 25820-2000 "Concrete light" it goes to the class B40.



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Many scientists of Russia carried out a lot of researches on receiving constructional light concrete on the basis of artificial porous fillers, for example, expanded clay, agloporite, bloated slag and others. They also studied their properties and their application in oil and gas, mine, road, hydrotechnical, industrial and civil types of construction. They produced cases of sea vessels for their service and development in severe conditions of the European North and the Arctic. These works were performed by I.N. Akhverdov, V. V. Babkov, Yu. M. Bazhenov, O. Ya. Berg, G. A. Buzhevich, A. I. Vaganov, A. A. Gvozdev, V. G. Dovzhik, I. A. Ivanov, I.G. Ivanov - Dyatlov, S. M. Itskovich, M. A. Kornev, P. G. Komokhov, B. A. Krylov, O. V. Kuntsevich, A. F. Polak, N. A. Popov, M. Z. Simonov, B. G. Skramtaev, M. P. Elinzon and others. Their theoretical prerequisites and the studying of problems of structurization and physical and chemical properties of light concrete have promoted the optimization of products` static durability, freezing resistance and crack resistance as well [1].

The offered technological researches consider and justify the complex utilization of silicate mining formation on the basis of coal washing waste at concentrating mill and waste heaps of coal mines in the Pechora pool. As new types of mineral components together with clay breeds in raw mixes to produce porous fillers, products of construction ceramics and portland cement, they viewed waste rocks of lithocomplexes of Devonian lateritic bauxite ore of Timan minerogenic province (Middle and Southern Timan). They are presented by low-modular bauxites, allites, siallites on the basis of water aluminosilicates and the sub-standard containing bauxites [2].

Modern requirements of resource and energy saving define the search of materials with high rates of thermophysical properties. In the northern climatic zone a very effective gas and foam concrete with the low density don't take place in single-layer execution according to the thermal resistance indicator. The use of porous fillers (agloporite, slag pumice, thermolith and others) allows to utilize production wastes, not to develop natural material, to reduce the designs` mass. Producing artificial fillers the roasting of the prepared semi-finished product is the most responsible operation which defines the quality and technical and economic indicators of the enterprise work, especially in the conditions of the market relations.

The cheapest way of their production at heat treatment of clays, loams and waste of coal mining and coal washing is a roasting by sinter machines and in fluidized layer furnaces. These thermal units provide higher efficiency per unit volume of oven space, the heat utilization during the roasting. Agloporite quality improvement goes along with the production technology cost reduction, i. e. the exception of burned product crushing onto the crushed stone due to receiving an agloporite of a gravel-like form.

The Central Laboratory of construction materials PECHORNIUI (Ukhta, Vorkuta in Komi republic) carried out a research work on use of not burned mine breeds and waste of coal washing in the Pechora pool as raw materials to produce an agloporite [3].

The main component of these breeds are aleurolites, argillites in the number of 70 … 80% and carbonaceous substance upto 25%. The mineral part of coal waste tests is presented by dense, fine-grained aleurolites of clay and quartz and feldspathic structure.

During the laboratory and technological tests it was developed the copyright certificate on the USSR invention No. 439484 relating to compositions of light artificial filler for concrete, agloporite [4,5]. Its purpose was to increase the durability, vertical speed of filler agglomeration because the weight including clay raw materials and coal contains also the rock, for example bauxite. This is the following ratio of components, in weight. %:

Clay raw materials – 75 … 85,

Coal - 4 … 5,

Rock, for example, bauxite - 10 … 25.

Then the invention was developed according to which at production of porous filler, concrete (expanded clay, agloporite, thermolite and haydite) the granules of a semi-finished product are dusted with refractory or fire-resistant powders after their granulation or directly in the roaster furnace by means of the special device which gives powder before a blow-up zone of the thermal unit. The purpose is to decrease bulk density, increase the coefficient of constructive quality and expand blow-up interval of filler according to the USSR copyright certificate No. 1066967 [6]. In researches they used waste rocks of bauxites of Vezhayu-Vorykvinskiy field of Middle and Timan bauxite mine of JSC "Boksit Timana" (Ukhta, Komi Republic). The surface of gummy granules from low blown-up raw materials on the basis of coherent soil with water colloidal linkages in the form of mineral clay formations is processed by floured waste rocks of bauxite ores, allites and siallites or their mix with fire resistance 1500 … 1600 $^{o}$C and high dispersion. Thus, in a semi-finished product blanket when roasting there are high-temperature fine-crystalline new growths, such as hematite, mullite, feldspar, spinel and ulvospinel characterized by the increased durability, deformability and heat resistance at the time of pyroplastic state of a firm phase of minerals [7].

Pilot-scale tests of these rocks were executed at Research Institute of Construction Materials BSSR (Minsk) by semi-industrial agglomerative drawing machine of continuous running. They were carried out with rocks, sample No. 3 in the mine No. 9 at "Intaugol" plant and with washery refuse sample No. 6 of washing plant of mine No. 40 at "Vorkutaugol" plant, dusting the gummy granules with bauxite rocks. On the basis of pilot lot agloporite they received a constructive flyash concrete of class B15 … 20 (M 200 … 300) with an average density of 1600 … 1700 kg/m3.

Agloporite pilot lot crushed stone was used to select the flyash concrete structure according to the technique of professor N. A. Popov [8]. The results of flyash concrete sample test are given in table 1.

Table 1

Physics and technology properties of flyash concrete

| Class (brand), durability on compression | cement consumption, kg/m3 | Water consumption, liter | Filler consumption, kg/m3 | | | | Average density, kg/m3 | Strength on compression, MPa |
|---|---|---|---|---|---|---|---|---|
| | | | Quartz sand | Agloporite sand | Crushed stone, **фр.**, mm | | | |
| | | | | | 5-10 | 10-20 | | |
| 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 |
| After steaming | | | | | | | | |
| B3,5 (M50) | 200 | 180 | - | 240 | 130 | 250 | 1000 | 5,5 |
| B5 (M75) | 220 | 200 | - | 280 | 150 | 360 | 1210 | 6,8 |
| After steaming +28 per day of normal concreting | | | | | | | | |
| B3,5 (M50) | 200 | 180 | - | 240 | 130 | 250 | 1020 | 6,6 |
| B5 (M75) | 220 | 200 | - | 280 | 150 | 360 | 1220 | 7,8 |
| After steaming | | | | | | | | |
| B15 (M200) | 380 | 220 | 320 | - | 165 | 530 | 1620 | 17,5 |
| After steaming +28 per day of normal concreting | | | | | | | | |
| B15 (M200) | 380 | 220 | 320 | - | 165 | 530 | 1640 | 21,2 |

According to the data of table 1 it is established that the durability of flyash concrete is defined by the quantity and quality of the cement stone providing the set degree of filler grain separation, considering a ratio of strength and stress-related characteristics of the high-strength flyash concrete [9].

The subsequent researches in NIISM BSSR and NIIZhBe (Moscow) allowed to develop standard designs from a dense flyash concrete of class B30 … B40 on a local agloporite. On its basis they produce the manufacture panels of overlappings and coverings of residential, civil and industrial buildings, monolithic walls in high-rise buildings in Minsk, considering influence of structural and mechanical characteristics of filler and concrete mix.

In the Komi Republic when houses were being built the flyash concrete had no distribution because ceramsite gas concrete was used in the single-layer protecting designs with the density of 850-1000 kg/m3. It had the best heat-shielding characteristics. Now according to Construction Norms and Regulations 23.02-2003 "Thermal protection of buildings" it is also impossible to make single-layer walls of buildings without the facade warming [10].

In recent years instead of mass large-panel housing construction we have been producing a construction and front ceramic brick, hollow stones with chromophoric colourants and wall blocks from heavy and light concrete with different fillers of 390*190*188 mm in size in the way of volume vibrocompression. Agloporite occupies 60-75% of concrete volume.

Trial production of products made from constructional concrete with local fillers with a productivity of 10 … 25 thousand sq.m per year can be in the operating productions of the monotown, Vorkuta, at development of Far North territories and fields [11].

If there is a need to have agloporite or ceramsite concrete of high brands, they can be received with the use of quartz sand and to form blocks with internal emptiness. It will improve a heat-shielding of walls. Thus it can be developed the production technology of ceramsite gravel from the coal preparation waste including high-strength one. The problem of their utilization and ecological safety of the surrounding nature including Bolshezemelskaya Tundra will be solved. Powder preparation of breed, oxidizing roasting at temperature 800-1000$^{o}$C in furnaces with the fluidized boiling layer and the dust application at blow-up can be put in a basis to produce technology of expanded clay from waste of coal preparation and waste heaps of mines [12].

Thermotechnical calculation of heat-insulation layer thickness for the northern cities of KR is given in the table 2.

If it is necessary to manufacture flyash concrete light blocks with the thickness 190mm, it is possible to carry out blockwork from two blocks with inside warming between them. As a heater we offer the heat-insulating mats from basalt superthin fiber which is produced at JSC Lotos (Syktyvkar). The thickness of these mats is 60mm. If it is necessary to have a heater layer of 120mm, it is possible to use the double layer. Consequently, the wall will conform to the requirements on heat resistance. In addition it is possible to cover a building facade with hydrophobic plaster from dry construction mixes with the painting pigments.

Table 2

Insulation thickness at different thickness of external walls

| Thickness of blockwork, m | Effective thickness of plate from basalt fiber, $\lambda$ = 0,042 W/m·K, m | Thickness of blockwork, m | Effective thickness of plate from basalt fiber, $\lambda$ = 0,042 W/m·K, m | |
|---|---|---|---|---|
| | Hollow brick, density, $\rho$<1600 kg/m$^3$, $\lambda$ = 0,56 W/m·K | | Agloporite block | |
| | | | Density, $\rho$<900 kg/m$^3$, $\lambda$=0,27 W/m·K | Density, $\rho$<1000 kg/m$^3$, $\lambda$=0,29 W/m·K |
| 1 | 2 | 3 | 4 | 5 |
| Vorkuta | | | | |
| Rтр=4,3, 0,90-0,77 | 0,088 0,101 | 0,38 | 0,121 | 0,125 |
| Inta | | | | |
| 0,90-0,77 | 0,088 | 0,38 | 0,12 | 0,125 |

All these problems are devoted to development, investigation and operation of mineral raw material resources of the North, the Polar Ural and Subpolar Mountains and future implementation of the Belkomur project.

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## Registration form (data) of the participant of the XVIII International Congress on coal enrichment (28.06-01.07.2016, St. Petersburg, Russia, Gorny University)


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# NEW APPROACHES FOR COAL OXIDIZATION PROPENSITY ESTIMATION

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**Abstract**

Studying of coals propensity to oxidation nowadays is current and urgent. It is reasoned by the fact that coals oxidation processes, happening during the whole life-cycle (from mining to utilization), inevitably lead to quality loss and, what is even more important, to occurrence of fire (self-ignition) hazards. At the moment, there exist no unity of opinions regarding classification of coals by their propensity to oxidation. In this paper, newly developed approaches for coals propensity to oxidation are introduced based on complex study methods, such as metamorphism degree characterization, chemical activity evaluation, characterization of fracturing occurrence and tendency, thermal methods of different types. Combination of such coals behavior and properties characterization methods allowed to classify their propensity to oxidation at least by three groups: 1st group – coals with minimal propensity to oxidation, 2nd group – coals with pronounced tendency to oxidation and 3rd group – coals with the highest propensity to oxidation. The first group contains high-rank coals and anthracite, the second and third groups included lignites, oxidized and bituminous low-ranked coals depending on parameters combination.

**Keywords**

coal, oxidation, propensity, metamorphism degree, chemical activity, ozone, endogenous and exogenous fracturing, heat flow, kinetics

Since its mining up to utilization, coal passes through multiple stages connected with different impacts of natural and technological origins. Coal's life cycle since its extraction from bed is defined by contacts with oxidation environment as air oxygen and moisture. This leads to initiation of processes of organic and inorganic coal matter oxidation. Unity of processes accompanying coals oxidation is a reasons for risks of fire events and products quality loss. Therefore, the coals oxidation problem is very current and urgent nowadays.

According to the world and Russian practice, coals tendency to oxidation is determined by different methods. The most widespread in the USA and Australia is method based on temperature increase effects for coals loads between 40 -70ºC in adiabatic conditions under constant oxygen concentration. Such method was called "R70" [1-4]. Also, a "crossing point temperature" [5-8] method has become increasingly popular in the world. Such method is based on determination of time corresponding to initial stage of temperature increase within the center of coal load with respect to the walls temperatures under the condition of coal load heating with constant temperature. In the Russian Federation, coals propensity to oxidation and self-ignition is traditionally characterized by kinetic indices (velocity constant) of oxygen adsorption by coals at room temperature. Many researchers also connect coals tendency to self-ignition with sulfites and other mineral components in coals [9-12].

In the current work we propose to classify coals by their propensity to oxidation with help of the following criterion groups:

1st group – genetic characteristics connected with coal matter origin and their genesis and metamorphism processes;

2nd group – coals chemical activity with respect to oxidation agents;

 
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3rd group – characteristics of endogenous and exogenous fracturing of coals defining accessibility of oxidation agent to free surface, gas permeability and diffusion coefficients;

4th group – effective kinetic parameters characterizing intensity of exothermal processes leading to coals self-heating.

Analysis of literature data, also results of preliminary studies demonstrated that part of the listed characteristics may be determined on the basis of standard methods (petrographic composition of coals, metamorphism degree indices, carbon, hydrogen, sulphur and nitrogen contents). As for the rest of indices, an adaptation of existing or construction of new methods is mandatory. Development of new methods as well as their verification has become possible due to a representative collection of coals samples at hand. This collection contains coals of different origin and rank, also having various material composition on mineral components contents and presence. As a result of experimental and analytical studies we developed and verified the following methods allowing to adequately evaluate criteria characterizing coals propensity to oxidation:

1. Method for coals tendency to oxidation estimation by interaction with ozone. The developed method regulates determination of coals chemical activities indices in processes of low-temperature oxidation. Parameters calculated with help of the proposed kinetic model allowed to introduce classification of studied coals by their tendency to oxidation (Fig. 1). Because of the fact that at oxidation the number of active centers decreases, coals chemical activity with respect to oxidation may be evaluated by ability to irreversibly sorb ozone. Such parameters are determined by velocity of integral activity alteration (here, characteristic value is the maximal velocity of KL(t) variations). At development of such method we utilized new approaches and technological solutions, specifically, utilization of more active oxidation agent (in comparison with oxygen) and original system of data registration, allowing to perform investigation at room temperatures by on-line writing the ozone concentrations.

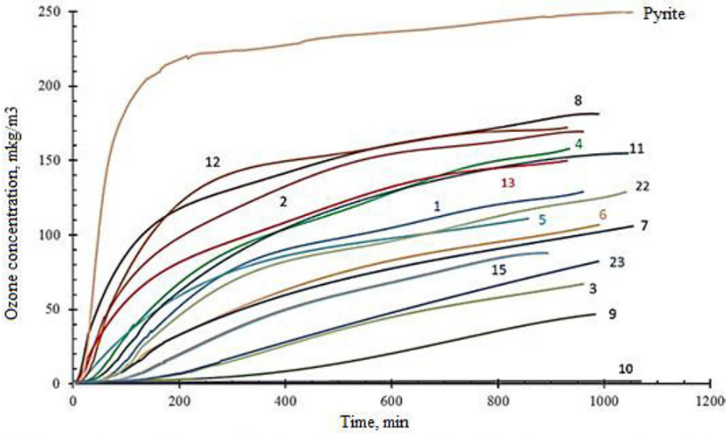


Figure 1. Modeling of coals oxidation processes by ozone treatment (1,2, 3 –low-volatile bituminous coals, 4,7,8,15 – high-volatile bituminous coals, 10- anthracite, 9,11,12,13,22,23 – lignites of different rank, 5 and 6 – oxidized bituminous coals).

2. Method for determination of effective kinetic parameters characterizing intensity of exothermic processes leading to coals self-heating. Evaluation of effective heats of coals oxidation was performed by isothermal calorimetry by TAMAir isothermal microcalorimeter. The conducted studies allowed to determine differences between kinetic parameters of oxidation processes (heat release velocity, total heat) of coals of different types. Comparison of the results with data on coals chemical activity by total active

groups (TA-total acidulous) and indices characterizing coals rank (reflectance index R0) demonstrated a good correlation (see Figures 2-3).

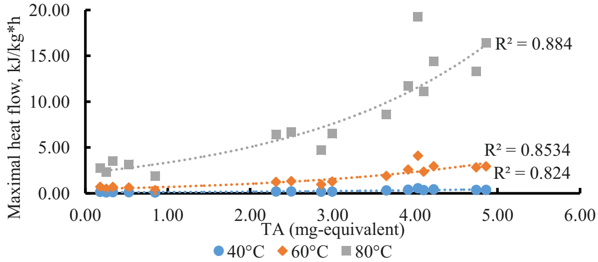


Figure 2. Maximal heat flow (heat release velocity) vs TA at different temperatures of experiment

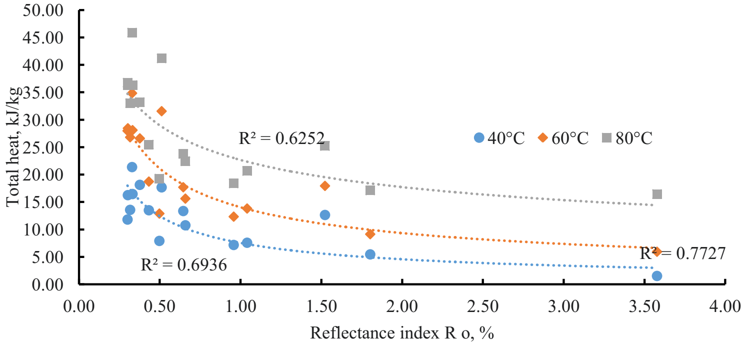


Figure 3. Total heat vs coals rank for different temperatures of experiment

3. Determination of coals propensity to oxidation by physical and chemical methods. For evaluation of total active groups contents in coals a method was developed based on potentiometric titration of alcohol-alkaline extracts of coals. Processing of potentiometric titration results along with chemical analysis data allowed determination of total acidulous groups (TA) (sum of phenolic and carboxyl groups), carbonyl, phenolic and carboxyl groups for different coals types. It was found that by character of active oxygen contents alteration all the studied coals may be divided into two groups: 1$^{st}$ group – all bituminous coals except of oxidized one; 2$^{nd}$ group – lignites and oxidized bituminous coal.

In order to evaluate the mechanism of coals interaction with air oxygen the TGA studies of coals in inert and oxidizing environments were performed. Comparison of TG curves in the aforementioned environments (by the value of mass change in low-temperature zone – 'mass delta') also allows dividing of the studied coals by the same groups: 1$^{st}$ type – bituminous coals and anthracite; 2$^{nd}$ type – lignites and oxidized coal of low rank (Figures 4-5). Connection of 'mass delta' parameter with coals chemical activity related to ozone is shown in Figure 6.

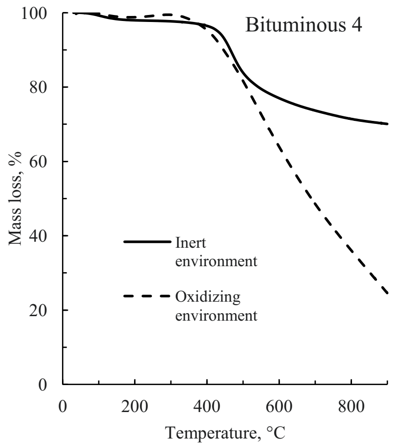


Figure 4. Comparison of TG curves of bituminous coal in different environments

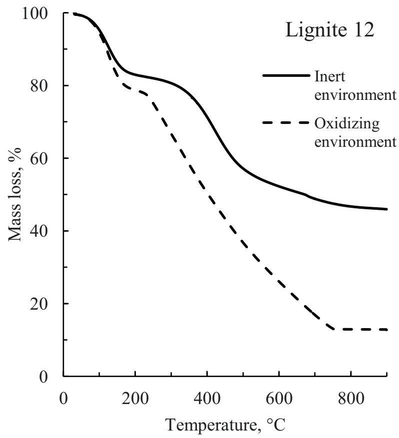


Figure 5. Comparison of TG curves of lignite in different environments

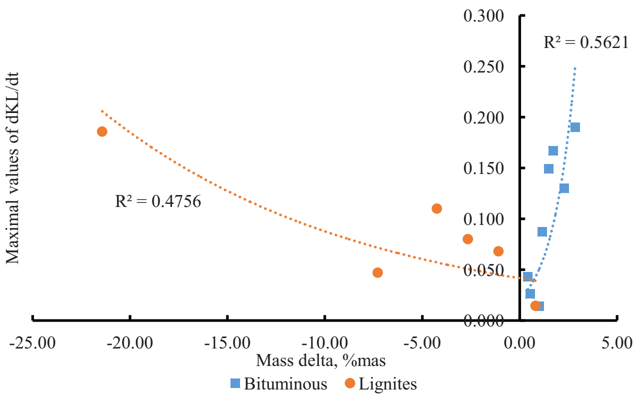


Figure 6. Ozone adsorption kinetic parameters vs TG-derived 'Mass delta'

4. Method for determination of mineral and valence iron forms in coals. As a result of the aforementioned studies for coals of different origins by element analysis and Mossbauer spectroscopy, a variation within sulphides (as pyrite) and carbonates (as siderite) and oxides of iron were found. A method for determination of such values and their variation was developed, the corresponding values were found for initial coals and coals after different terms of storage.

5. Determination of coals propensity to oxidation by thermostimulated acoustic emission. A new method is developed for quantitative exogenous and endogenic fracture estimation for coals based on effects of thermostimulated acoustical emission (TAE). TAE laws experimentally found and interpreted for coals as functions of their strength at thermal impacts, and also material composition of inclusions, oxidization degree and features of samples individual structure [13]. Taking into account the found

sensitivity of thermal destruction coefficient ($k_{td}$) as such classification parameters as material and chemical composition, it should be admitted that the developed method could be adapted also for quantitative evaluation of the aforementioned classification indices determining coals propensity to oxidation.

Characteristics obtained by the aforementioned methods allow to classify coals propensity to oxidation at least by three groups: 1$^{st}$ group – coals with minimal propensity to oxidation, 2$^{nd}$ group – coals with pronounced tendency to oxidation and 3$^{rd}$ group – coals with the highest propensity to oxidation. The first group contains high-rank coals 1-3 and anthracite, the second and third groups included lignites, oxidized and bituminous low-ranked coals depending on parameters combination.

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# Potentially valuable microelements in coal balance reserves of JSC SUEK mining units

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**Summary**

There are no coal deposits whose content of potentially valuable microelements (PVE), except for uranium (U), is high enough to ensure economically viable production of commercial PVE compounds. This process should be based on by-products which are generated as a result of conventional use of coals – combustion, coking, gasification, etc., which concentrate PVEs. By-products which accumulate certain PVEs are called primary concentrates. These can then be used to produce commercial compounds or industrial PVE concentrates.

The PVE content in coals from various deposits or different areas (seams) of the same deposit is considerably variable. For example, there are some deposits where the PVE content is several scores of times higher than their percentage abundance in coals. Coals of this type are called metal-bearing and serve as the main raw material for production of commercial PVE compounds.

JSC SUEK is striving for integrated development of coal deposits.

Areas containing metal-bearing coals have been identified in deposits which are operated by JSC SUEK in the Russian Far East, Kuzbass, Kansk-Achinsky coal basin, Zabaikalye and other basins where concentrations of some PVEs are several scores of times higher than their average content in coal. To date, we have acquired extensive information on patterns of formation and location of metal-bearing coals and their PVE compounds within coal-bearing structures in these basins.

For instance, it has been determined that a major part of gold contained within brown coals of Borodinsky-1 and Borodinsky-2 seams occurs as micro and nanoparticles of solid solutions of Au-Cu-Zn, Au-Cu-Ag, and Au-Cu-Zn-Pb whose gold content is over 50% by mass.

We have also conducted preliminary studies of distribution of various PVEs in washed products, products of combustion, gasification and hydrogenation of coals as well as studies of the most advanced technologies for generating certain PVEs. We have discovered that industrial-scale combustion of coals and coal-bearing products has the highest potential in terms of producing primary PVE concentrates.

Certain PVEs (Ge, Mo, W, Ag, Au, Zn, Re, Se, and possibly Ga and Rb) which form gaseous compounds within the high temperature zone, condense on fly ash particles once the gas flow cools down. This gas flow carries PVEs out of the furnace, thus the captured fly ash becomes the primary concentrate. For other PVEs (rare-earth elements, strontium, Nb, Hf, Zr) the primary concentrate is a mixture of fly ash and slag.

Primary concentrates of both types, especially the fly ash one demonstrate much higher PVE content than the burnt fuel. Additional increase in the PVE content in primary concentrates can be achieved through preliminary processing of coals before combustion and through separation of fly ash either based on the particle size or various magnetic parameters.

**Keywords**

Potentially valuable microelements, content and occurrence forms, distribution during processing and combustion, primary concentrates, commercial compounds, integrated utilisation.



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Elements whose content in solid fossil fuels is ≤0.1% by mass are called microelements (ME). Among MEs there is a special group of potentially valuable elements (PVEs): Ge, Ga, U, rare-earth elements, Y, Au, Ag, Mo, W, Pt and other platinoids, Re, Se, Co, Ni, V, Zn, Pb, Cr, Bi, Sr, Zr, Nb, and Ta.

There are no coal deposits whose content of PVEs, except for uranium (U), is high enough to ensure economically viable production of commercial PVE compounds. This process should be based on by-products which are generated as a result of conventional use of coals – combustion, coking, gasification, etc., which are called primary concentrates and where certain PVEs concentrate.

High PVE concentrations in products of coal utilisation depend on their content in raw coal, which in turn depends on geochemical patterns of their accumulation. In terms of the average content calculated using coal or coal ash, we identify so-called typomorphic MEs whose content is more than 1.4 times or 2.0 times higher, respectively, than their average content (percentage abundance) calculated using dry coal or ash of all coal deposits in the world: Au-Ge-Re-Se or Au-Ag-Be-B-Ge-Mo-Pb-U-W, respectively [1-3, 5, 6].

PVE content in coals from various deposits or different areas of the same deposit varies considerably. For example, there are areas or seams in the Russian Far East [1, 3, 5] and Kuzbass [2] deposits where the content of Ge, Au and non-typomorphic rare-earth elements is several scores of times higher than their percentage abundance in coals. Coals of this type are called metal-bearing and serve as the main raw material for production of primary PVE concentrates which are suitable for production of their commercial compounds or industrial concentrates.

Distribution patterns, conditions of migration, peculiarities of localisation in coals and methods of extracting PVEs and their compounds have not yet been properly studied while methods of analysing Au in coals and derivative products have not been tested. Therefore, the search and identification of promising areas of coal accumulations containing PVEs at concentrations which ensure profitable production of their commercial compounds remains a very important task for efficient development of coal deposits.

Since its early days, JSC SUEK has been striving for integrated development of coal deposits, which includes utilisation of their organic substances and concurrent mineral components, including microelements. We have acquired more than 10 years' experience of studying patterns of formation and location of metal-bearing coals within the coal-bearing structures in the Russian Far East, Zabaikalye and other basins. We have determined that metallic coal-bearing strata are formed in places of prolonged intersection between areas of accumulation of organic substance and alimentation zones of large ore metallogenic belts. These structures interacted for tens of millions of years via a network of water transport passages and draining uplifts which provided transportation of PVEs both as solutions and clastogenic forms and as colloids and dissolved chemical compounds.

Coals with higher content of gold and some other PVEs have been found in the deposits in Primorye as well as in some seams of the Kansk-Achinsky coal basin, which are being extracted by JSC SUEK’s mining units, in metal-bearing ore alimentation zones [1-3, 5, 6].

Along with studies to identify areas of potential PVE accumulation in coal-bearing structures, JSC SUEK’s objective has been to substantiate the accuracy of the obtained results using the existing methods for determining the gold content in coals. An assay coal testing method has been selected. We have identified forms of occurrence of PVEs in coals and their derivatives using electronic microscope and X-ray spectral studies of coal samples from Pavlovsky deposit in the Russian Far East, Nazarovsky and Borodinsky deposits of the Kansk-Achinsky coal basin, and Leninsky and Erunakovsky deposits in Kuzbass.

For instance, it has been determined that a major part of gold contained within brown coals of Borodinsky-1 and Borodinsky-2 seams occurs as micro and nanoparticles of solid solutions of Au-Cu-Zn, Au-Cu-Ag, and Au-Cu-Zn-Pb whose gold content is over 50% by mass. Finer loose particles occur within caverns and cavities and form intimate interpenetrations with matrix organic substance. We have also discovered gold sulphochloride particles. Silver occurs in native state, as sulphides and possibly as chlorides.

Studies of the degree of mineralisation of brown coals from Rybinsky-1 and Rybinsky-2 seams have shown that noble metals occur in the form of disseminated microphases of solid solutions: with copper and zinc for gold; with iron and copper for platinum; and with tin for silver. The particle size is of micronic and sub-micronic levels. The particles are scattered within the coal and do not have a marked association with other minerals, for instance with quartz or sulphides.

In terms of commercial-scale production of industrial PVE compounds, the by-products of processing of coals which contain two or more PVEs demonstrate the highest potential due to geochemical factors and parameters of coal processing and utilisation technologies.

Prospecting and exploration activities for determining the content of PVEs within coals of Kuznetsk basin, which are mined by JSC SUEK have shown that coals from certain seams can be considered to be metal-bearing because their PVE content is many times higher than their percentage abundance in coals. Among the potentially valuable are some rare-earth elements like Nb, Au, Ag and others.

For example, Au whose content in coals is several scores of times higher than its percentage abundance (0.0056 grams per tonne for hard coals and 0.018 grams per tonne for coal ash) has paragenesis with Ag, Fe, Sn, Cu, Pt and some other elements in various mineralogical forms. Gold content of coals from Breyevsky seam of Kirova Mine is 27.2 grams per tonne, which is to be confirmed using newly obtained samples. Gold has been discovered in clastogenic native and clastogenic autogenic forms. There are predominantly small-size forms of native gold, whose particle size ranges from several tens to a hundred microns. Correlation between the list of elements which form natural associations in Kuznetsk basin coals and the sales potential for this product means that the following elements are of primary interest in terms of commercial operation: gallium (Ga), lanthanides (La, Tb, Yb), scandium (Sc), yttrium (Y), Zr, Hf, Nb, Au, Ag, and Ti.

A significant number of natural associations with higher PVE content is typical of various coal seams of Kuznetsk basin: Breyevsky seam (Au, Ba, La, Ce, Nd, Sm, Eu, Yb, Lu, Hf, Th, U), Seam No. 52 of Kotinskaya Mine (Ga, Sr, Y, Nb, Hf, Au, U) and others.

Many PVEs within metal-bearing coals are mainly concentrated within their organic substances. Tests have shown that the following elements belong to these: Ge, Ga, Be, Mo, W, Sr, Sc, Y, La, etc. Thus, in the course of coal processing these microelements mainly transfer into the concentrates which are processed further – for combustion, coking, gasification, hydrogenation to produce synthetic liquid fuels or chemicals.

After coking most PVEs except for mercury are transferred into the main product, which is coke. Up to 12% of Ge contained within the raw blend is concentrated within tar water and to a lesser extent within tar itself, which is used to produce its commercial concentrates. Possibly, cokes obtained from coal blends with higher PVE content can be used in production of preliminary alloys used for production of special steels enriched with PVEs.

In the process of hydrogenation of coals PVEs together with other mineral components concentrate in the solid residue – slag, which is used as a fuel for combustion or gasification. After combustion or gasification of coals or solid products which are produced as a result of hydrogenation or production of waxes, PVEs are transferred into by-products – the so-called ash and slag products (ASP) which are primary concentrates of PVEs and are regarded as a raw material for production of commercial compounds or industrial PVE concentrates.

However, the forms of PVE compounds in the ASP which are captured after combustion are more favourable in terms of performance indicators associated with production of such commercial products compared to ASP compound forms after gasification. Thus, commercial combustion of coals and coal-bearing products is of highest interest as a process of producing primary PVE concentrates.

Technologies of recovering rare elements from thermal coals should be based on the following key principles:

- PVE content in ash and slag products should not be lower than in conventional raw materials;
- production of PVE commercial compounds from coals should be accompanied by the fullest possible use of their energy constituent;

- ash and slag products of coal combustion should be used as an individual raw material or a component of a crude ore blend at the existing operations which produce commercial compounds of rare elements.

The content of PVEs in ASP depends on their content ($C_0$) in the burnt coal (concentrate), its ash ($A_0$) or ($A_к$), distribution between slag and fly ash, i.e. their normalised concentration ($Y_s$) and ($Y_f$), respectively, slag capturing factor ($K_s$) and combustion mode.

Distribution of various PVEs during combustion of coals is not uniform because some of them which belong to Group 1 of PVEs (Ge, Mo, W, Ag, Au, Zn, Re, Se, possibly, Ga, and Rb) form gaseous compounds in the high temperature zone, which condense on fly ash particles after combustion products cool down and are captured with them. Because of this their primary concentrate is fly ash where concentration of such PVEs is many times higher than their content in burnt coal, which is especially true for furnaces with high slag capturing factor.

Other PVEs which belong to Group 2 (rare-earth, strontium, Nb, Hf, Zr,) are carried out of the high temperature zone as finely dispersed particles which are captured together with the fly ash particles. Their primary concentrates are a mixture of slag and fly ash, so for them the combustion mode must be associated with generation of such compounds within the fly ash and the slag, which will ensure their hydrometallurgical processing into commercial compounds [1, 3]. PVEs, with a possible exception of gold, are concentrated in such primary concentrates as oxygen compounds.

Studies which have been performed by SUEK and other organisations show that fly ash consists of particles mainly within the 10-300 micron size range, which possess some peculiar physical properties like density and magnetic susceptibility, besides size. The content of PVEs in particles which possess these physical properties is variable.

Therefore, separation of fly ash in magnetic or electrostatic fields to produce part of fly ash significantly enriched with certain PVEs might be potentially productive. This will provide an additional economic benefit from producing commercial compounds of valuable microelements. For example, separation of fly ash which is carried out of the furnace combined with capturing of coarsely dispersed particles with cyclones and finely dispersed particles captured with bag filters and electric filters may produce an increase in concentration of, say, germanium by more than 5 times compared to non-separated fly ash.

In some thermal coals of Primorye, Kuznetsk and other basins which are operated by JSC SUEK-Kuzbass mines, and sometimes in their refuse coal the PVE content is high enough for their concurrent recovery [5, 6]. If washing is the first stage of processing coals which contain PVEs, then in most cases the PVE content within the primary concentrates increases compared to combustion of non-washed coals. Further methods of PVE recovery depend on the type and content of the element.

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# PERSPECTIVES OF COMPLEX DEVELOPMENT OF COAL DEPOSITS

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**Abstract**

For the first time the system approach to the solution of a problem of complex development of coal deposits for the purpose of creation of production engineering of non-polluting safety manufacture of the thermal and electric energy, valuable chemical and fuel products is offered. The complex of technologically interconnected manufactures will represent the closed system. Designing of systems components is carried out at a stage of geochemical and chemistry-technological researches of coals and embedding rock, taking into account the value of valuable and toxic chemical elements revealed in them.

In Institute of Coal of Siberian Branch of RAS it is designed renewed automatic Database "Geochemistry of coals of the Kuzbass", containing information on concentration before 70 chemical elements. Designed technology protected by the patents to Russian Federation on inventions: extractions toxic chemical element from gaseous products underground gasification of coal; complex development of coal deposits; the selective mining of coal seam with raised by contents valuable chemical element; shaping technogenic deposits useful fossilized from waste of the incineration and enrichments of coals. They are created and are scientifically motivated the new geo-technology structure of the opening, preparation and mining, directed on complex development of coal deposits.

**Key words:** coal deposits, complex development, coal energy-chemical clusters, chemical trace-elements, selective mining of coal, underground gasification of coal, technogenic deposit, complex coal processing.

Russia is ranked 6th in the world for coal production, the volume of which in 2014 amounted to 357 million tons, including 203 million tons - in Kuzbass. The program of development of coal industry of the Russian Federation assumes that by 2020, production will reach 380 million tons, and by 2030 - 430 million tons. More half of this volume is planned mane in Kuzbass.

Kuznetsk coal basin possesses unique resources of hard coal of different grades and here industrial mining of coal is accomplished in a big scale. High quality of kuznetsk coal has defined a sphere of its wide application. Coal is used as the energy resource for production of metallurgical coke and also as the technological feedstock. Total resources of kuznetsk coal till the depth of 1,800 meters quantify 524 bln. tons.

One of the directions of the development to technologies coal mining from thick flat and pitching seams is connected with controlled issue underroof or between layer mass. This opens the new possibilities of the underground development coal seams on get fat thickness [1]. The Other direction of increasing to efficiency, technical and ecological safety of the underground development high gas storage capacity coal deposits indissoluble is connected with development of the complex of the special facilities of the stimulation gas yield of un relieve coal seams with the following salvaging the mine methane [2].

However these directions do not solve the problems of the complex developing coal deposits Kuzbass. Conventional technologies of coal mining and processing in a more or less extent entail technogenic de-



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struction of natural environment, appearing to be dangerous production processes, don't allow to recover associated minerals and don't ensure a minimal loss of coal.

Integrated development of coal deposits implies two key aspects in addressing a number of issues considered in this study. The first one relates to the definition of quantitative and qualitative indices of the coal, by a complex of precious and toxic mineral components accumulated in coal mining and coal consumption wastes. The second one assumes new development trends of technologies for mineral goods production, including rare, noble, nonferrous metals recovery from coal and coal waste, as well as toxic components utilization at stages of coal mining, processing, use and waste products outputting, accumulation and storage [3].

Ecological balanced, resource-saving technologies of extraction and complex coal processing in all coal-mining countries of the world are engaged in creation. The main directions of research are: creation of low- and waste-free production and complex use of raw materials and energy resources; developing a radically new technological processes without waste; recycling all kinds of waste production and consumption to obtain marketable products; creation and production of new products with the requirements of reuse; creation of closed cycles.

It is installed that in kuznetsk coals, product of their conversion and consumptions contains valuable and toxic chemical elements and components, which reasonable at the same time extract not only to achieve the additional goods product, but also for the reason guard surrounding ambiences [4].

For example, only in the Kemerovo region has accumulated more than 77 million tons of ash waste from large power plants, to which is added to 7 million tons/year. These wastes, on the one hand, represent an environmental hazard, on the other - it is a concentrate of valuable chemical elements, components. In them, except the basic components of a mineral part of coals (Si, Al, Fe, Ca, Mg, Na, K), can be present up to 60 chemical elements, including at the concentration exceeding industrial conditions on extraction. Kuzbass coal ash contain, for example, to 36.7% iron, gold - 28 g / t silver - 770 g / t, antimony - 2% titanium - 1.5% and other elements is very valuable [5].

The composite author has developed a complex of new and specific technologies, aimed for recovery of chemical admixture-elements in the course of integrated development of coal deposits [6]. Thus, quality of marketable coal may be improved due to discrete (selected) mining accomplishment directly in the mine, when run of mine coal of different composition is delivered to the surface. Dedicated technologies, excluding a gross mining method of coal seams, should ensure the preservation of coal natural grade in various layers; discrete formation and surface delivery of several flows of homogeneous quality rock mass; separate processing of coal of different grades; simultaneous operation of some technological lines–mining-processing, integration of additional technological operations into the main process [монография, нетрадиционные технические решения, горнотехнические требования [7].

Due to dedicated formation of technogenic deposits of minerals with specified geochemical composition that defines a main trend of its further processing, it becomes possible to extract different mineral resources more complete. For instance, during regulation of coal delivery to processing and power plants, the recorded and primarily prepared material aimed for further industrial processing will accumulate in coal concentrate, setting dumps and waste banks.

It is known that technogenic deposits are formed mainly on the lands of the federal subjects and individual municipalities. Thus, a feature of the modern paradigm of state (municipal) government is internationally recognized form of interaction which is a state-private partnership [8]. In the context of this article the authors define a state-private partnership as an institutional alliance, backed by the political will, between the state and private business in order to improve governance in the field of integrated development of natural resources in a particular area with the transfer of business responsibility for the performance of certain public functions [9].

At the same time it should be provided to establish closer links between research institutions and the productive sector for a better understanding of their technological challenges, as well as the need to adjust the regulatory framework for the implementation of the principles of partnership with the private business sector.

The principles of state-private partnership bodies, businesses and research organizations can effectively ensure the comprehensive development of coal deposits by creating coal, energy-chemical clusters.

Principle novelty proposed scientific and technological decisions is concluded in that that is expected unite on one territory, drawn near to place of the mining useful fossilized, complex interconnected on source raw material and technologically complementing each other production, allowing make the most energy and chemical potential of the gained solid fuel, provide get fat salvaging of the technogenous waste, provide safety water and land resource of territory.

We shell considers complex development new coal deposit, for example Uvalinoe in Tersinsky geological-economical region of Kuzbass. Coal energy-chemical cluster mast include (fig.) [10]:

- enterprise for coal mining;
- preparation plant;
- shop on clear slime;
- boiler-house;
- enterprise for underground coal gasification;
- enterprise for remake bottom ash;
- storehouse of waste of preparation – technogenic deposit of minerals;
- enterprise for remake of waste of preparation.

The accounting amounts of the mining, preparation, remake and god marketable products are shown on fig. The sub-standard reserves of coal, concentrated in parts of coalfield with complicated geological conditions of the bedding the seams (Uvalinie 5-6, 7-8, 9-10), is offered to perfect by underground gasification with passing extraction valuable elements. This will allow to get not only combustible gas, but also valuable chemical elements – a rhenium, quicksilver, selenium, arsenic and platinum.

Conventionally the recovery of iron from composition of ferromagnetic minerals is performed with magnetic separators. Iron contained in siderite undergoes complete oxidation under water vapor treatment and then is extracted into a concentrate by an ordinary or HGM separator. The authors suggest to apply this method for iron recovery from friable dumps left after coal and ore processing. In Kuzbass ore resources incorporated within above said dumps and in coal combustion waste products amount to at least 180.0 mln. tons [5].

The complex of interrelated plants on the base of coal field aimed at more full use of coal potential as a mineral deposit will allow for obtaining product of high added value and ensure rational nature management.

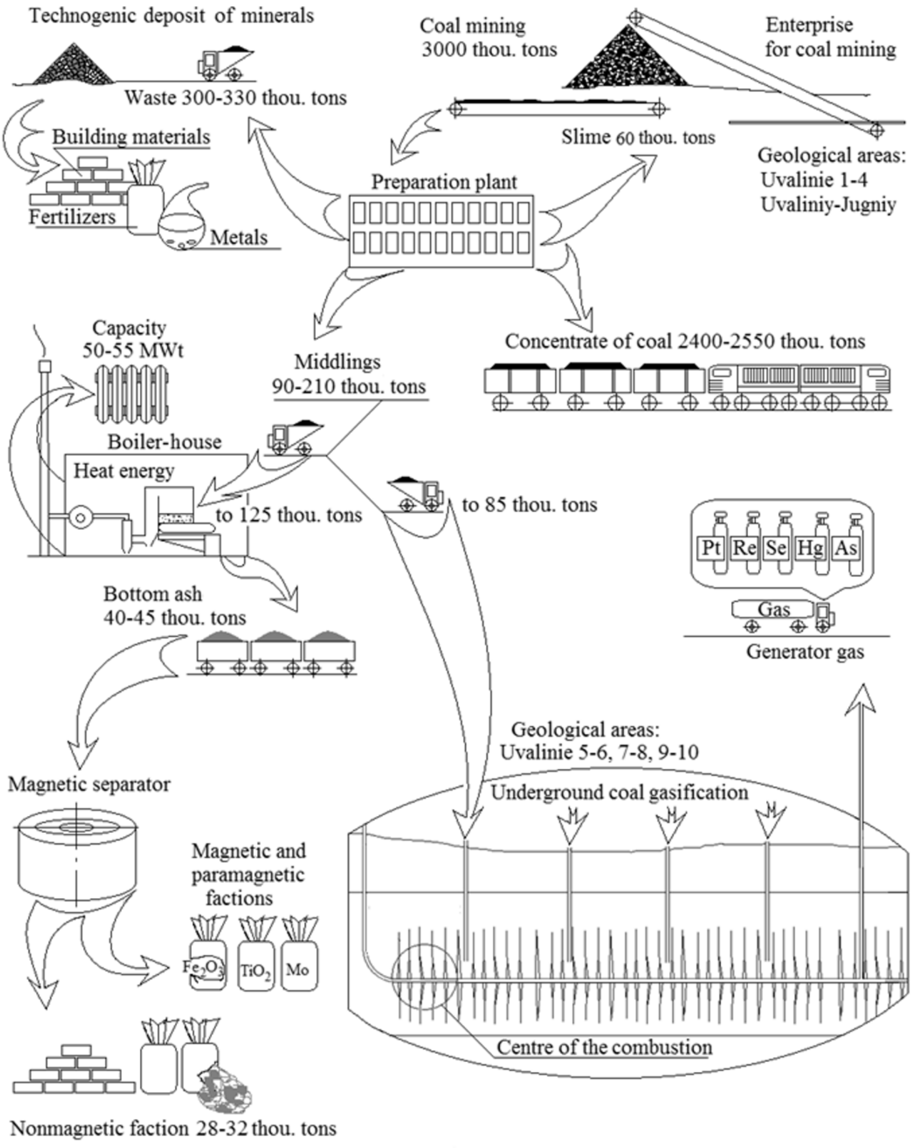


**Fig. Structure of coal energy-chemical cluster to the developing coal deposit Uvalinoe in Kuzbass**

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# Development of Technology for Producing Compound Coal Binder

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The paper discusses the need of developing an alternative binder to produce anodes. The possibility has been studied of using the tars from semi-coking coal for the production of pitch. It has been shown that the pitch produced from the above tar does not meet the requirements for the pitch used to produce anodes. A technology has been suggested to produce compound pitch by combined heat treatment of low-temp coking coal tar with high-temp coking coal tar. The resulting product is, then, oxidized to produce pitch of anode quality. For combined heat treatment, in addition to the tar obtained by high-temperature coking, it is feasible to use either tar from semi-coking coal or a heavy residue obtained by distilling such tar. The share of the tar from semi-coking coal can be 40-50% in the above mixture. The share of the heavy residue can be 20-30%. The resulting compound pitch meets the requirements for high-temperature pitch of anode quality. Moreover, it shows low carcinogenicity.

Key words: coal tar pitch, semi-coking coal tar, anodes, heavy residue, distilling, compound pitch, benz(a)pyrene

Coal tar pitch and petroleum pitch are the main raw materials used to produce carbon anodes for electrolysis. Both coal tar pitch and petroleum pitch are by-products, and their quality varies. High-temperature coal tar is a by-product generated in the process of producing metallurgical coke from coal, so the amount of tar produced and the quality of such tar are determined by the demand for metallurgical coke and the coal coking technology used. A decrease in the amount of coal tar has already made the Russian aluminum smelters purchase imported pitch. The second issue related to pitch is the content of polycyclic aromatic hydrocarbons (PAHs). The environmental authorities are concerned with the issue, and make smelters reduce PAH emissions into the atmosphere, including benzo(a)pyrene [1, 2]. So, the above is the reason to look for alternative raw materials and develop new binder production technologies.

One of the solutions of the above pitch deficit problem is to develop an alternative method for pitch production.

Coal tar pitches are unique in that they give a high coking value during carbonization due to their polyaromatic nature. Coal tar pitches contain more than 10,000 different compounds [3]. Such compounds have a wide range of molecular weights: 180-2,500 and 12,000 AMU [4, 5].

Alternative pitch production abroad is based on the use of aromatic petroleum products. Petroleum pitch itself is not applicable for the production of anodes and anode paste due to a low coking value and, as a result, unsatisfactory mechanical and physical properties of pre-baked anodes. The common practice is to mix petroleum pitch with coal tar pitch. Industrial tests of mixed pitches show a decrease in PAH emissions by 40-50% vol.; the cell technical & economic indicators, in the meantime, are standard [6, 7].



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Today, petroleum-coal tar pitch is not produced in Russia. The price for aromatic petroleum materials is high to arrange for cost-effective hybrid pitch production.

One of the practical ways of solving the pitch deficit problem is the use of other products obtained in the process of coal processing. There is data [8] that using the tars, methanol-extracted from gasified coal, helped obtain medium-temperature pitch with a high coking value. Other researchers [9] developed a process for producing pitch by low-temperature oxidation of anthracene oil. Combining air oxidation and thermal treatment of oxidized products helped obtain pitch with a coking value of up to 55%.

This paper discusses the results of using tar from semi-coking coal to produce pitch of anode quality.

Semi-coking of coal, compared to high-temperature coking, is performed at 550-700°C, which leads to a higher tar yield and almost no carcinogenic PAHs, including benzo(a)pyrene. Both brown and black coals can be used for semi-coking. Tars are produced as a result of the primary decomposition of coal: individual units are split off from the organic mass of coal, and transferred to the liquid phase. The resulting tar includes hydroaromatic, aromatic and naphthenic compounds with a large amount of side chains [10], which, in turn, explains a low C/H atomic ratio, as well as 30-40% of phenolic compounds. Currently, the tars from semi-coking coal are generally used as fuel and barely used for other purposes.

The objective of our work was to determine the suitability of using tars from semi-coking coal to produce pitch of anode quality.

The tars obtained by both coal low-temperature pyrolysis (semi-coking) & high-temperature coking were used as raw materials. The tar characteristics are given in Table 1.

Table 1 – Characteristics of Initial Tar Samples

| Tar | Density, g/cm$^3$ | Coking value, % | Moisture, % | T.I., % | Q.I., % | Ash, % |
|---|---|---|---|---|---|---|
| Semi-coking coal tar | 1.033 | 4.9 | 2.3 | 1.4 | 0 | 0.4 |
| Coal tar | 1.182 | 30.4 | 2.2 | 10.1 | 2.3 | 0.1 |

**Method for Producing Pitch & Distillate Fractions**

Pitches were produced as a non-distillable residue in the course of distilling tars (separately or together.) Distillation was carried out by using a 500-ml three-neck flask with a single ball reflux condenser (tar sample weight – 318-350 g.) The tar was separated into distillate fractions (by temperature in the vapor phase.) The pitch yield was determined by residue weight after distillation vs. sample weight. The rate of collection of distillate fractions was 2 drops per second; temperature was controlled in both the liquid and vapor phases. Distillation was carried out in several modes:

1. With heat soaking – heating in the still pot up to a liquid phase temperature of 390-421°C, retention within 60-120 minutes at the same temperatures.

2. Without heat soaking – heating in the still pot up to the target temperature, no retention.

Pitches were taken out at a liquid phase temperature of 280-300°C. The loss was less than 5%.

**Method for Thermal Oxidation**

Pitch samples (100 g in total) were treated in a cylindrical reactor with air, including mixing with propeller stirrer at 340-370°C.

**Pitch from Semi-coking Coal Tar. Results**

The results of semi-coking coal tar distillation (up to 390 and 405 degrees C) are given Table 2.

Table 2 – Balance of Distillation of Semi-Coking Coal Tar

| Designation | Final temp, ºC | Fractional yield, % wt. | | | |
|---|---|---|---|---|---|
| | | Up to 210ºC | 210-230 ºC | More than 230 ºC | Pitch |
| DR405 | 405 | 4.5 | 7.0 | 46.8 | 39.3 |
| DR390 | 390 | 5.8 | 6.5 | 37.6 | 46.7 |

Table 3 – Elemental Analysis and Characteristics of Tar Distillation Residue

| Sample | Content, % wt. | | | | | | Characteristics | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | C | H | N | S | O | C/H | d, g/cm$^3$ | $T_{sp}$, ºC | α,% | $α_1$,% | $V^r$,% |
| DR405 | 83.3 | 7.4 | 2.3 | 1.2 | 5.8 | 0.94 | 1.10 | 65 | 12 | 0.2 | 84 |
| DR390 | 83.4 | 7.5 | 2.0 | 1.4 | 5.7 | 0.93 | 1.09 | 56 | 10 | 0.2 | 88 |
| Coal tar pitch w/ softening point of 70°C | 91.9 | 4.3 | 1.7 | 0.6 | 1.8 | 1.78 | 1.28 | 70 | 27 | 5.0 | 59 |

d-density, $T_{sp}$-softening point, ring-and-pin method, α– T.I., $α_1$— Q.I., $V^r$ – volatile yield @ 850ºC.

As compared to coal tar pitch, such pitches (semi-coking coal tar distillation residues) have a lower carbon content, a low C/H ratio, a rather high content of heteroatoms, nitrogen, sulfur and, especially, oxygen. Such pitches are substantially free of quinoline-insoluble substances (Q.I.), have a low content of toluene-insoluble substances (T.I.) and a high volatile content. According to the thermogravimetic analysis (TGA), the coking value @ 850 °C for these pitches is 12-15%. The above characteristics will negatively affect the mechanical strength of the binder matrix in the course of manufacturing anodes.

Increasing the temperature and duration of treatment of the semi-coking coal tar distillation residue leads to a jump in the softening point, making such pitch unsuitable for the use as a binder. When obtaining pitch with the required softening temperature, the content of T.I. & Q.I. is below standard.

Therefore, the pitch obtained by heat treatment of semi-coking coal tar, does not meet the requirements for pitch of anode quality.

The following studies are related to the combined heat treatment of low-temperature coal tars with high-temperature coal tars.

For combined distillation, either semi-coking coal tar or a semi-coking coal tar distillation residue (with a liquid phase temperature of up to 390ºC) were used together with the tar obtained by high-temperature coking.

Additionally, pitches were subject to air oxidation after combined distillation. The oxidation conditions varied within the range of values that are possible to be reached by using a particular coal tar processing plant. Mainly, it is related to the temperature range of oxidation, which is 340-370 °C. The duration of oxidation was 4 to 6 hours. The air feeding rate was slightly higher than it normally is in the industry, but this parameter is quite variable, depending on a particular industrial unit.

Table 4 lists the conditions for obtaining compound pitches, and their characteristics.

Table 4 – Conditions for Obtaining Compound Pitches, and Their Characteristics

| Coal tar pitch : semi-coking coal tar | Thermal treatment | Pitch yield, % | Characteristics | | | | |
|---|---|---|---|---|---|---|---|
| | | | $T_p$, °C | α,% | $α_1$,% | $V^r$,% | Benz(a)pyrene, mg/g |
| 45:55 | distillation | 43 | 104 | 38 | 4.5 | 57 | - |
| 50:50 | distillation | 46 | 93 | 35 | 5.1 | 59 | 4.9 |
| 60:40 | distillation | 48 | 88 | 30 | 4.4 | 58 | - |
| 50:50 | distillation, oxidation | 46 | 96 | 35 | 5.1 | 58 | 5.0 |
| 60:40 | distillation, oxidation | 49 | 89 | 30 | 4.5 | 59 | 5.6 |
| Coal tar pitch : semi-coking coal tar distillation residue | | | | | | | |
| 60:40 | distillation | 57 | 112 | 39 | 5 | 54 | - |
| 70:30 | distillation | 61 | 94 | 35 | 4 | 56 | 5.1 |
| 80:20 | distillation | 57 | 90 | 36 | 6 | 56 | 6.4 |
| Requirements for the quality of pitch of anode quality | | | 85-95 | ≥ 31 | ≤ 12 | 53-58 | |

The results show that:

1. An increase in the content of semi-coking coal tar in a mixture of tars – under same conditions of heat treatment – leads to the growth of the softening point and the content of T.I. However, there is no significant change in the content of Q.I. and the volatile yield.
2. An increase in the content of a semi-coking coal tar distillation residue in a mixture, containing the tar obtained by high-temperature coking, also leads to the growth of the softening temperature and the content of T.I. And, there is a decrease in the content of Q.I and the volatile yield.
3. The compound pitch yield is high when a distillation residue is used.
4. Additional oxidation of compound pitch leads to an increase in the softening point without any substantial change in other characteristics.

The obtained results show that chemical reactions take place between compounds of high-temp and low-temp coking coal tars. As a result, there is a non-additive increase in the α- and $α_1$-fraction content and a reduction in the volatile yield, and there is an increase in the softening point. For example, when distilling semi-coking coal tar, the α-fraction yield is 12%. When distilling high-temperature coking coal tar, the yield is 27%. Under combined distillation (50:50), the α-fraction yield is 35%.

It should be noted that the content of benzo(a)pyrene in the obtained compound pitches is significantly less than in coal tar pitch, where it is in the range 8.5-1.1mg/g. [11, 12]. Due to the fact that semi-coking coal tar is substantially free of benzo(a)pyrene, compound pitches, which use such tar, show low carcinogenicity.

Table 3 shows that the pitches with a ratio of tars of 50:50, 60:50, as well as the pitch obtained by combined heat treatment of a mixture containing both high-temp coking coal tar and a semi-coking coal tar distillation residue (80:20), meet the requirements for pitch of anode quality. Increasing the share of semi-coking coal tar or the above semi-coking coal tar distillation residue leads to a dramatic increase in the softening point and, as a result, a reduction in the wettability of the filler's grain (during anode paste mixing).

The content of Q.I. in compound pitches fully depends on the Q.I. content in high-temp coking coal tar. So, therefore, for compounding (combining), it is preferable to use coal tars with a high content of Q.I. in order to compensate for the absence of such compounds in semi-coking coal tar.

The elemental analysis of compound coal pitches shows that the C/H atomic ratio is 1.5. This value is closer to the parameter for coal tar pitch than for the pitches obtained by heat treatment of semi-coking coal tar (Table 2). The content of oxygen, sulfur and nitrogen is 2, 1 and 2%, respectively (which is at the level of coal tar pitch.)

Compared to the pitch produced from semi-coking coal tar, the Fourier IR spectra of compound pitches show no absorption related to the phenolic groups in the following areas: 3,400 and 1,150-1,280 $cm^{-1}$, and there is a higher ratio of the absorption bands of aromatic hydrogen to hydrogen in the aliphatic groups (Fig. 1).

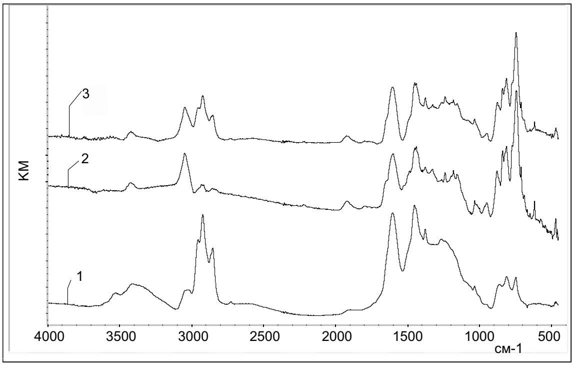


Fig. 1 - Fourier IR spectra of semi-coking coal tar distillation residue (1), coal tar pitch (2) and the compound pitch from coal tar pitch :semi-coking coal tar distillation residue in ratio 50:50 (3)

Thus, under combined distillation of high-temp coking coal tar and either semi-coking coal tar or a semi-coking coal tar distillation residue, phenols are removed from the residue of combined distillation. And, under certain conditions, the technical parameters of compound pitch are close to the characteristics of conventional coal tar pitch.

The proposed technology will help expand the raw materials' base (the base of binder pitches), ease the tension in the market of coal tar (which is produced in a conventional way), and reduce PAH emissions (during preparation & carbonization of binder pitch in Söderberg cells.)

Low-temperature coal pyrolysis products are cheap and affordable. The above process may be performed ether continuously or in batch mode by using the existing coal tar processing plants.

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# Complex Technology of Underground Coal Gasification and Coal-Based Methane Recovery Using Geodynamic Zoning

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## Abstract:

The article describes the complex technology of coal methane recovery, which involves underground coal gasification of lower beds in series of gas-rich strata liable to rock-bumps, coal based methane recovery and mechanized coal mining. Geodynamic zoning is used for selection of places for well drilling.

The application of the suggested technology enables to solve a range of tasks, such as

- unloading of workable beds and reduction of their rock-bump hazard due to protective seam burnout;
- increase of workable beds degasification due to unloading;
- increase of workable bed methane desorption rate due to conductive and partly convectional heat transfer through interbeds;
- reduction of workable bed coal strength and rock-bump hazard by passing-through combustion gases (CO and especially $CO_2$);
- degasification of coal bearing layers and sublayers of interbeds due to their partial burnout during gasification and accelerated degassing during intensive heating;
- increase of gasification products heat value due to their diluting with methane recovered from upper beds degasification;
- reduction of methane outburst to the atmosphere, which has greater greenhouse effect in comparison to carbon dioxide.

**Key words:** series of coal strata, gasification, complex technology, degassing, geodynamic zoning, methane.

The significant amount of ultimate coal reserves is located in the series of gas-bearing strata liable to rock-bumps and could not be recovered with conventional mining methods. During ascending exploitation mining the extraction of coal from the lowest seam is quite challenging or impossible due to high geodynamic risks. During descending mining the overworking has low impact on processes of

 
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unloading and gas emission from lower beds, thus mining can be done only up to the first thick interbed or until reaching very hazardous beds.

The conventional method of mining series of strata prone to rock- or outbursts involves initial extraction of protective seam. [1, 2]. The advantages of this method are parallel extraction of protective and main seams during the period of the highest unloading and its maximum effect due to relatively small advances.

However, firstly, when mining methane rich coal beds the extraction of protective seam using oil mining becomes highly risky in every aspect – outbursts, rock-bumps, intensive gas emission, and methane explosion risk, which prevents protective seam extraction under extremely hazardous conditions at great depths.

Secondly, the non-pillar extraction is quite challenging within the framework of conventional mining methods, and if there are difficult conditions in the protective layer, the mining process could not be carried out at all. The alternative to non-pillar extraction is creation of yield (long-holed) pillars, but even they cause high-pressure zones in the overlying beds though not very intense. Besides it is very challenging to provide yielding property of pillars created not for technical purposes (between extraction columns, panels, etc.), but for passing extremely hazardous tectonic disturbances (faults, flexures, folds) and formation morphological damages (swelly, balk, sandstone inclusion, offset, fault wash, etc). These pillars can be large in size, have complex shapes, rapid changing physical-mechanical properties, and produce significant burst (stress) pressure on joint coal seams.

Thirdly, the unloading of a hazardous seam due to extraction of protective layer only partially solves the issue of its degassing and increases gas recovery factor by reducing rock pressure. Moreover, the extraction of protective layer using conventional mining methods leads to intake of additional gas through interbeds, as well as from development and working faces during mining and later from worked-out areas. The gas from goafs may come not only from "fast sources" (not-mined layers of the face, rocks of immediate mine roof) but from "slow sources" as well (incomplete long-holed pillars, pillars around extraction column, zones of geological faults, off-spec coal bearing seams, sublayers and interbeds).

We consider underground gasification of protective layer to be the most advanced method. It was first studied in publications [3-5], several suggestions for developing this method were given in [6-8] and other papers.

In this paper the underground gasification of lower protective layer in series of strata is described not only as a method of unloading and reduction of rock- or outbursts hazard during overlying beds mining, but as a method of highly dynamic intensification of overlying beds degassing and methane recovery and more complete extraction of energy from coal-based methane deposits.

The complex technology combining coal-based methane recovery and underground gasification of coal from lower beds in series of gas-rich strata liable to rock-bumps solves several tasks:

1. unloading of workable beds and reduction of their rock-bump hazard due to protective seam burnout;
2. increase of workable beds degasification efficiency due to unloading;
3. increase of workable bed methane desorption rate due to conductive and partly convectional heat transfer through interbeds;
4. reduction of workable bed coal strength and rock-bump hazard by passing-through combustion gases (CO and especially $CO_2$);
5. degasification of coal bearing layers and sublayers of interbeds due to their partial burnout during gasification and accelerated degassing during intensive heating;
6. increase of gasification products heat value due to their diluting with methane recovered from upper beds degasification;
7. reduction of methane outburst to the atmosphere, which has greater greenhouse effect in comparison to carbon dioxide.

One of the main arguments against using in-place gasification (it is similar to arguments against shale production) is pollution of ground waters, reservoirs and withdrawal points. The suggested technology reduces this type of risk since gasification occurs at great depth and bulk of water from burnout zone is pumped out with mine systems of degassing and drainage, which enables to perform water treatment process.

The experience of underground gasification [9-10] has shown that the technology is most efficient for coal seams having the following characteristics:

- seam lies at the depth from 30 to 800 m;
- seam has great thickness not less than 5 m;
- ash content should not exceed 45%;
- part of the seam undergoing the gasification should not have evident faults
- high seam permeability (≥ 0.1-0.5 mD).

The presented example intentionally does not have the second characteristic as it is economically profitable to use conventional methods for mining thick seams, it is suggested to use thin seams (from 0.5-0.7 to 1.5-2 m) as protective layers. In some cases, it is difficult to meet the 1st condition as well since the lower seam can lie at great depth of 1 km and more. The last characteristics are very difficult to follow since under conditions of vertical and horizontal stresses the deep seated seams liable to rock-bumps have massive structure and low permeability (~ 0.01 mD).

That is why the rational selection of the starting place for gasification is very significant. This place should be located in tectonically unloaded zone with excessive fissuring and preferably having low water and high methane contents. In other words, when using other mining methods, the gasification should start with the places most favorable for degassing.

The selection of tectonically unloaded sections is made using geodynamic zoning. The application of this method entails reconstruction of block structure of coal lease, creation of joint map of geodynamic and gasodynamic factors, calculation of stress inside and at the edges of blocks, and evaluation of filtering and collecting properties of the section using connection between stress state and permeability.

The example of selecting the places favorable for degassing or initiating in-place degasification is shown at figure 1.

The second stage involves drilling, this technology requires three types of wells: draught, gas drawoff in burn area and degassing in workable seam (fig. 2). With further advancement the functions of the wells are switched hence all of them are used as draught, gas drawoff or degassing ones. The experience has shown that it is more efficient to drill wells with relatively large diameter for installing tubes with relevant capacity for gas drought and draw off. The drill rigs are equipped with control units and gyroscopic inclinometers for precise drilling.

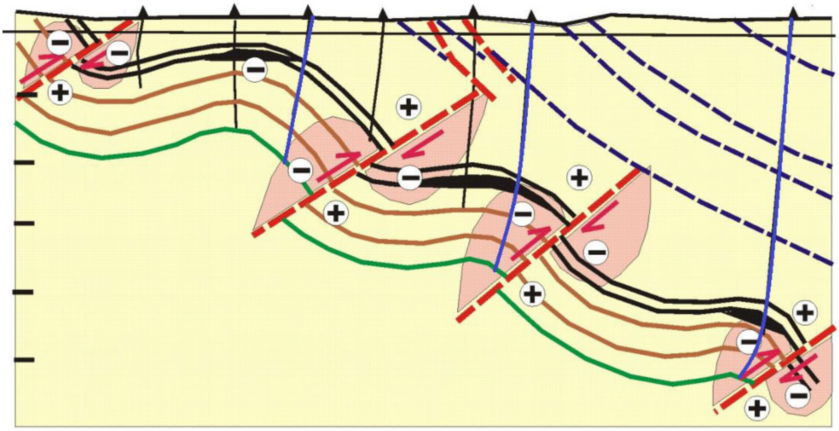

Figure 1 – Selection of places favorable for initiating degassing and underground gasification processes (based on materials of N.I. Mishin and A.L. Panfilov). The faults define stress condition of block structure and form compression and tension zones. The latter are the most favorable for degassing. The optimal positions of degassing wells are marked with black color, blue is used to show places for degassing and underground gasification of lower seam (green color).

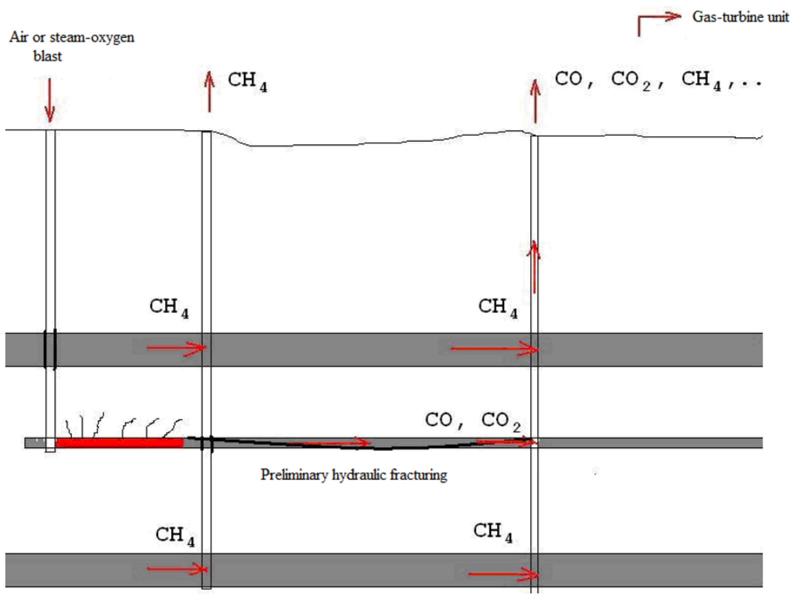


Figure 2 – Degassing by means of underground gasification of one of low thin beds.

The third stage involves creation of gas-permeable channels between draught and gas drawoff wells for preparation of underground gas generator and can be carried out using the following methods:

– linking of wells by burning filtering;
– hydraulic fracturing of coal seam with fluid or gas and burning of the created fracture (fig.2);
– drilling and burning of inclined and horizontal burn path.

The method selection depends on the existing mining and geological conditions and can be specified during design process. The temperature in the coal burning zone can be very low when there is a high water inflow to gasification area. The experience has shown that underground generator works efficiently with water flow up to 0,6 $m^3$ per 1 ton of gasified coal. Thus it is necessary to perform dewatering of the gasification place in case of high water flow in the gasified seam and/or when using hydraulic fracturing. The removal of water from the burn zone is done by permanent air injection under pressure exceeding underground water seam pressure.

The key element of the suggested technology is heating of interbed and working seams to increase degassing efficiency. In order to estimate the influence of heating process on methane desorption we can use experimental graph of I.L. Ettinger [11] for coal from Karaganda basin (fig.3) and empirical equation of G.D. Lidin [12]:

$$x_p = 65.5 \Big/ \left[ V^{0,146} (a/P + b) \exp\frac{0.02t}{0.993 + 0.007P} (1 + 0.31\,W) \right] \tag{1}$$

where $x_p$ – sorption methane retention capacity of coal, $m^3$/t, $V$ – volatile content per combustible mass, %, $P$ – pressure, atm, $W$ – water content, %, $a$ and $b$ – coefficients depending on coal rank. It is important that dependency of sorption methane retention capacity of coal on temperature $t$ ($^{o}$C) is quite general and similar for different mining-geological conditions, and temperature influences the process greater than pressure (depth of formation).

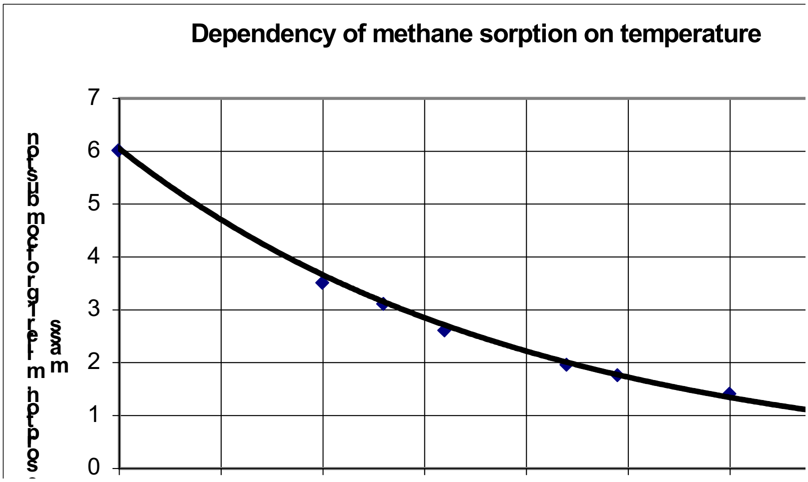


Figure 3 – Dependency of coal sorption capacity on temperature based on data [11]

For maximum heat transfer, on one hand, it is necessary to have the maximum possible combustion temperature. On the other hand, high speed of burning and pumping of coal-derived gas leads to reduction of heat amount transferred to enclosing rock. There is also a risk of burning a thick working bead if the interbed is thin.

On this basis we suggest three variants of underground gasification technology:

1. In case of thin interbed (up to 30-35 *m*, where *m* – produced seam thickness), it is recommended to use air blasting providing less rapid combustion. The air blasting could be used in case of low depth of production seam, thus heat loss will be relatively low and coal gasification paths will be formed in a more efficient way.

2. In case of thick interbed the risk of burning the working seam is quite low and it is more important to provide efficient heating. Thus the optimal solution is to use steam-oxygen blast providing higher burning temperatures (up to 800-1000 °C). At the same time the burning intensity should be kept at the minimal level supporting the stable process.

3. In case of over-thick interbed the direct heating of the working seam by heat transfer becomes very slow and inefficient. The produced gas could be used for additional heat impact on "seam-gas collector" through degassing wells. As shown on figure 4, in order to heat the thick seam and intensify degassing one or several wells were occasionally switched from withdrawal of combustion products to gas-turbine unit to passing through the seam. The combustion products CO and especially $CO_2$ facilitate coal strength reduction and accelerate methane desorption [13].

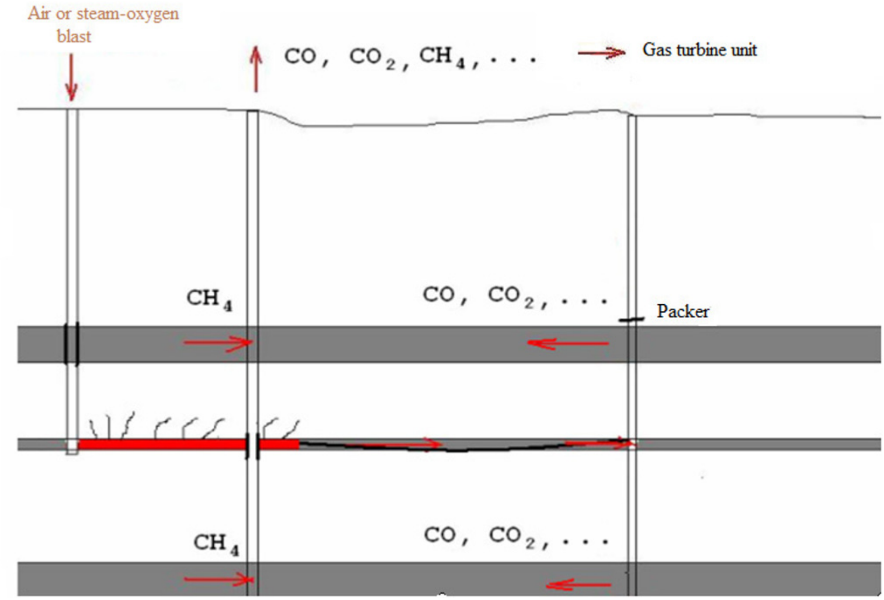


Figure 4 – Occasional intensification of degassing process by passing combustion products through seam

During gasification the burning intensity could become lower, especially in the zones of geological disturbances including minor faults and plication dislocations, balks, gaws, and seam inclination changes. In these cases, the intensification could be carried out by the inverse process – feeding recovered methane to combustion source. In order to stabilize and control the gasification process the authors of this paper developed and tested in laboratories and actual environment (Gas turbine unit at Angrenskaya plant) the techniques based on injecting liquid additives to expectable attenuation zones and squeezing them to the coal formation.

The gasification contours and volumes are monitored by the following:

- the volume of gaseous product;
- the results of observation over surface deformation;
- the control of burning face position using electro-magnetic survey methods developed by the authors of this paper.

The composition of the produced gas and its heating value are very dependent on type of combustion air and coal properties. When using air blow for gasification the produced gas has low heating value of about 4-5 MJ/m$^3$. This gas is suitable for gas turbine units producing electricity for drilling degassing wells, feeding power to compressors and pumps. When using steam-oxygen blast the produced gas has intermediate heating value of 10-13 MJ/m$^3$. The combination of gasification and degassing of working beds processes enables to produce the intermediate heating value gas by air blow and high heating value gas (from 20 MJ/m$^3$ and higher) by using steam-oxygen blast.

The underground gasification product generator gas used for production of heat and electric power also has valuable chemical raw materials: tar, phenol, hyposulphite, sulfur, etc. It is economically feasible to use chemical raw materials form gasification products recovered from condensate resulting from gas purification and cooling, including ammonia, phenols, tars, which can be extracted as saleable products (table 1). Depending on economic conditions the tar disposal can be done in two ways:

– total residue (mechanical additives and heavy tars – “heavy coal-tar products”) and coal are burnt in boilers;

– total residue is used in asphalt production for road construction.

Table 1 – Content of chemical components during underground gasification per 1 m$^3$ of product gas

| Gasification products | Volume, g/m$^3$ |
|---|---|
| Gas condensate | 140-150 |
| Ammonium | 2,0-2,5 |
| Benzene hydrocarbons | 1,0-2,0 |
| Tar | 0,3-0,6 |
| Pyridine bases | 0,5-0,7 |
| Sulfureted hydrogen | 0,3 |
| Tar camphor | 0,1-0,9 |
| Ethyne | 0,003-0,1 |
| Hydrocyanic acid | 0,007 |

The purification of product gas shows that efficiency of separation of condensate from ammonium is about 98% in the form of aqua-ammonia 25% solution. After this process the phenols are captured from waste water in the form of sodium phenate. The dephenolized condensate is sent to bio-chemical purification and phenole purification degree is about 90%.

Thus the suggested technology will provide an opportunity to mine industrial coal reserves in series of gas-bearing strata liable to rock-bumps and complex usage of energy and chemical resources of coal-methane deposits.

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# The technology of the oil shale processing to obtain products and semi-products for the chemical and metallurgical industries

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This paper is directed at deepening of processing of oil shale by thermal processes (pyrolysis, coking and gasification), products and semi-products of their thermal processing (oil, gas, semi-coke, coke). The main purpose was research of physicochemical properties, composition and thermal processing of oil shale. Modern laboratory equipments and methods of examination were used to achieve this goal. The physical-and-chemical properties and compositions (elements, chemical, phase and mineral) of oil shale (moisture content, ash content, porousness, and density) have been investigated. The optimal conditions of oil shale briquetting (composition, pressure, temperature, moisture content and material fraction) have been defined. The mineral components effects on fuel shale preparation and processing (yield, quality and properties products) have been researched. The operating practice depending on composition of the oil shale has been defined. The obtained results of these investigations allow developing new methods of oil shale processing. These methods allow significantly to expand the resource base and ensure rational use of nature resources.

**Word keys:** hydrocarbon recourses, oil shale, semi-coke, coke, chemical and mineral composition, physical-and-chemical properties, briquette

At present, coal and oil shale are the most common and reliable energy sources. The wide applicability of oil shale calls for study of their physicochemical properties to permit their effective use in industry as an energy source and a raw material for the production of graphite components. World reserves of oil shale (oil tar and gas) significantly exceed reserves of oil and natural gas (fig.1). Russia has extensive reserves of oil shale, exceeded only by those of the United States and Brazil. At present, with rising oil costs, the processing of oil shale is of increasing interest [1-3].

Improvements in the processing of oil shale require a better understanding of pyrolysis. Currently, no theory permits the explanation of the thermal processes and determination of the product composition on the basis of the chemical and petrographic composition and structure of the oil shale. In most cases, therefore, the pyrolysis of oil shale is studied experimentally for each specific deposit. The influence of the parameters of the process on the product composition and yield is determined. Research on the rational use of oil shale from different countries indicates that different processing technologies must be developed for different geological location and chemical composition of the oil shale [4-9].

Experience in Russia and Estonia is noteworthy. These countries lead the world today in the extraction, processing, and combustion of oil shale.

Current economic conditions call for innovative products such as coke for the electrode industry, graphite, and fullerenes. To that end, the physicochemical properties, composition and thermal processing of oil shale must be determined.

 
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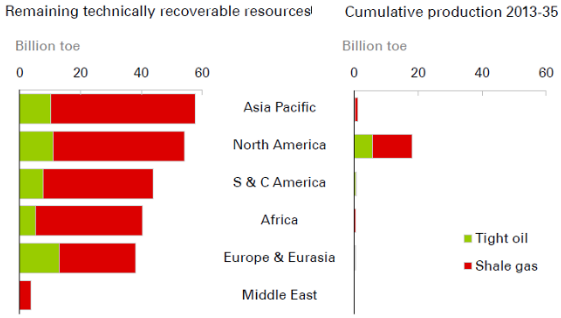


Fig.1 The reserve of oil shale and tight oil

The main purpose was research of physicochemical properties, composition and thermal processing of oil shale. The following tasks were decided:

– The physical-and-chemical properties oil shale (moisture content, ash content, porousness, and density) have been investigation;
– The compositions (elements, chemical, phase and mineral) of oil shale have been researched;
– The optimal conditions of oil shale briquetting (composition, pressure, temperature, moisture content and material fraction) have been defined;
– The processing of oil shale by thermal processes (pyrolysis, coking and gasification), products and semi-products of their thermal processing (oil, gas, semi-coke, coke) have been studied.

## The research methods

The object of investigation was oil shale Baltic basin. Chemical composition of oil shake was researched by energy – dispersive X-ray fluorescence spectrometer Epsilon3 PANalitical. Mineral compositions of oil shale and dross were investigated by X-ray diffractometer DRON-6. The thermal processing of oil shale was explored by pipe-furnace PT-1.2-40 and PTK-1.2-40. Briquets were produced on a PVL laboratory hydraulic press from two shale fractions (<125mm and from 125 mm to 2 mm) at 10 and 15 MPa, with and without preliminary wetting; in some cases, coal fines are added. The strength of the briquets is determined by the standard method: dropping from a height of 1 m until the briquet has completely disintegrated.

## The results of discussing

Obtained physicochemical properties of oil shale were compared with properties coals. Results of experiments are shown in table 1 [10].

**Table 1**. Physicochemical properties

| **Property** | **Brown coal** | **Black coal** | **Oil shale** | |
|---|---|---|---|---|
| | | | **Leningrad deposit** | **Estonian deposit** |
| Moisture content, % | 15,7 | 9,26 | 11,6 | 10,2 |
| Ash content, % | 18,0 | 6,0 | **50,5** | **42,0** |
| Volatile content, % | 45,0 | 290 | **41,43** | **40,0** |
| Apparent density, кг/м$^3$ | 1240,0 | 1223,0 | 1243,0 | 1476,0 |
| Actual density, кг/м$^3$ | 1452,0 | 1350,0 | 1643,0 | 1908,0 |
| Porousness, % | 14,0 | 10,0 | **24,0** | **23,0** |
| Moisture of analysis sample, % | 3,7 | 0,9 | 1,8 | 1,2 |

It can be seen that oil shale have value of ash content, porousness and volatile content more than coals.

The results of optimal conditions of briquetability of oil shale determination are the optimal moisture content of the material (a 1 : 1 mixture of the <125 mm fraction and 125 mm–2 mm fraction) is 23–25%; when a pressure of 15 MPa is used, the briquets withstand four drops. Increasing the moisture content from 11.6 to 37% reduces the calorific value of the briquets, while adding coal fines increases the calorific value [10].

The chemical composition of oil shale mineral part is shown in table 2 [11].

**Table 2.** Chemical composition of oil shale

| Number | Component | Content | Number | Component | Content |
|---|---|---|---|---|---|
| 1 | $Al_2O_3$ | 7,691 % | 6 | MnO | 0,106 % |
| 2 | $SiO_2$ | 33,220 % | 7 | $Fe_2O_3$ | 4,258 % |
| 3 | $P_2O_5$ | 0,470 % | 8 | Br | 0,191% |
| 4 | $SO_3$ | 4,499 % | 9 | CaO | 39,946 % |
| 5 | $K_2O$ | 7,147 % | 10 | $TiO_2$ | 1,318 % |
| 6 | MgO | 0,981% | | | |

According to the data obtained it can be seen that content consists mostly of oxides of calcium (CaO-39,946%) and silicium ($SiO_2$- 33,220%). The element composition of oil shale organic part is shown in table 3.

**Table 3**. The element composition of oil shale organic part

| Element | C | H | O | N | S |
|---|---|---|---|---|---|
| Content, % | 77,0 | 9,0 | 11,3 | 0,2 | 1,5 |

The experiment results of chemical composition determination of oil shale mineral part are shown in table 4 (diffractogram of oil shale is shown on figure 2) [11].

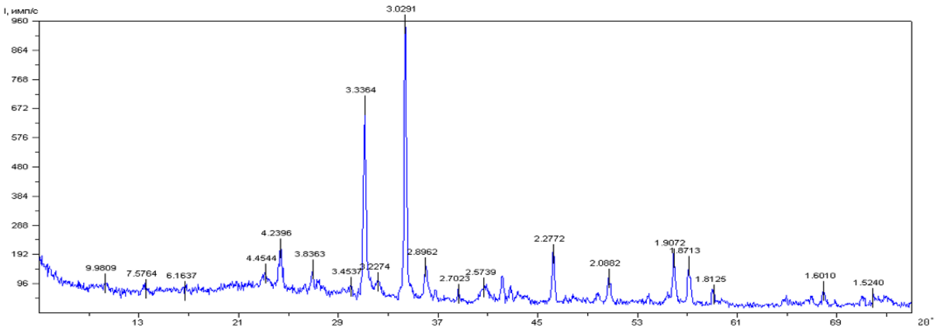


Fig.2 Diffractogram of oil shale

**Table 4.** Mineral composition of oil shale

| Mineral composition | | Content, mass. % |
|---|---|---|
| **Name** | **Formula** | |
| Calcite | $CaCO_3$ | 28±4 |
| Quarts | $SiO_2$ | 25±4 |
| Microcline | $K[AlSi_3O_3]$ | 11±3 |
| Illit | $(K_{0.75}(H_3O)_{0.25})Al_2(Si_3Al)O_{10}((H_2O)_{0.75}(OH)_{0.25})_2$ | 17±6 |
| Chlorite | $(MgFe)_3(SiAl)_4O_{10}(OH)_2 \cdot (MgFe)_3(OH)_6$ | 2±1 |
| Dolomite | $(CaMg)(CO_3)_2$ | 6±3 |
| Pyrite | $FeS_2$ | 2±1 |
| Goethite | FeO(OH) | 2±1 |
| Gypsum | $CaSO_4 \cdot 2H_2O$ | 2±1 |

It is evident from Table 4 that the dominant minerals in the initial oil shale include calcite $CaCO_3$ (28%), quartz $SiO_2$ (25%), illite $(K_{0.75}\,(H_3O)_{0.25})_{Al2}(Si_3Al)O_{10}((H_2O)_{0.75}(OH)_{0.25})_2$ (17%), and microcline $K[AlSi_3O_3]$ (11%).

It is found that on heat treatment, the mineral composition of oil shale changes, with the formation of rankinite $Ca_3Si_2O_7$ (17%), anhydrite $CaSO_3$ (10%), magnesite $MgCO_3$,periclase, MgO, and lime CaO. In the presence of oxygen bearing compounds such as illite and chlorite, pyrite $FeS_2$ is oxidized to hematite $Fe_2O_3$. Basic and acidic oxides react to form new minerals, such as larnite $Ca_2SiO_4$ and rankinite $Ca_3Si_2O_7$. The transformation of the shale's mineral composition on heating is accompanied by the liberation of CO, $CO_2$, $O_2$, $H_2O$, and $H_2S$ [11].

Figure 3 shows the mass loss of the oil shale in the given temperature range, at a heating rate of 18°C/min. Can identify five stages, corresponding to the composition of the organic and mineral components and the change in sample structure and properties with increase in temperature.

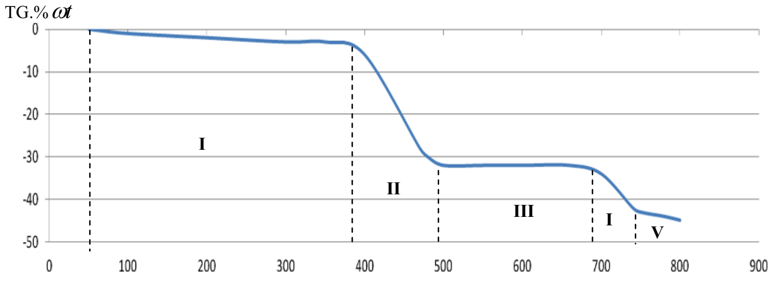


Fig.3 The mass loss of oil shale under heat treatment

In stage I (25°C –370°C), gases are liberated from the organic molecules of the oil shale (kerogen). Carbon dioxide and hydrogen sulfide predominate. At 270°C –290°C, active liberation of pyrogenic water begins. Further heating leads to more profound changes. In stage II (370°C –500°C), the liberation of tar begins; at 350–380°C, the shale's solid component passes to semi-liquid state. This stage is usually associated with bituminization, when the basic mass of the shale tar begins to form but there is insufficient heat for its vaporization. To reduce the bituminization, the heating rate must be increased, since the surface tar begins to evaporate with increase in the temperature, without allowing the shale pieces to liquefy [3, 4]. At 400°C, some quantity of tar is formed, but at 450°C tar liberation is practically over. Above 550°C, a small quantity of gas is liberated, since the hydrogen and oxygen content in the semi-coke is small. Almost no tar is liberated. Therefore, the yield of volatile components in stage III is slight. In stages IV and V (at 700°C –950°C), there is another abrupt change in the mass loss. That may be attributed to active decomposition of dolomite $(CaMg(CO_3))_2$ whose content in the mineral component of the shale may reach 50% [10].

The Figure 4 shows the change porousness in products and semi-products of oil shale thermal processing in a nitrogen atmosphere and in air in the range 25°C - 900°C. It can be see that the change in porousness may be divided into four stages. In the first stage (25°C –200°C), the porosity increases on account of the liberation of water, carbon dioxide, and hydrogen sulfide. In the second stage (200°C – 400°C), the porosity declines, on account of the bituminization of the shale. In the third stage (400°C – 600°C) and the fourth stage (600°C –900°C), the porosity increases on account of semi-coking and the decomposition of carbonate minerals [12].

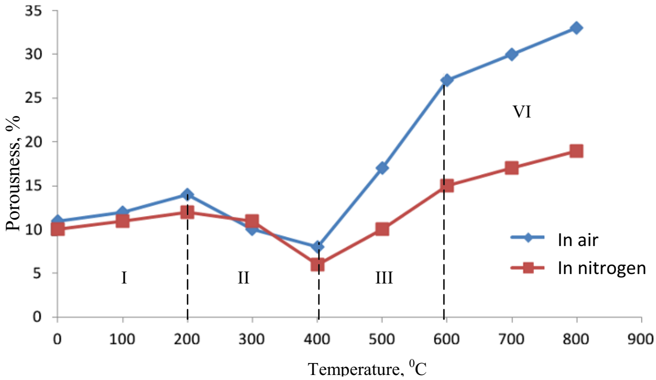


Fig.4 The change of porousness of oil shale thermal processing products and semi-products

**Conclusions**

1. Oil shale has value of ash content, porousness and volatile content more than coals;
2. The optimal conditions of briquetability of oil shale determination are the optimal moisture content of the material (a 1 : 1 mixture of the <125 mm fraction and 125 mm–2 mm fraction) is 23–25%; when a pressure of 15 MPa is used, the briquets withstand four drops. Increasing the moisture content from 11.6 to 37% reduces the calorific value of the briquets, while adding coal fines increases the calorific value;
3. Mineral part of oil shale consists mostly of oxides of calcium (CaO-39,946%) and silicium ($SiO_2$- 33,220%);
4. In oil shale dominant minerals in the initial oil shale include calcite $CaCO_3$ (28%), quartz $SiO_2$ (25%), illite $(K_{0.75}\ (H_3O)_{0.25})_{Al2}(Si_3Al)O_{10}((H_2O)_{0.75}(OH)_{0.25})_2$ (17%), and microcline $K[AlSi_3O_3]$ (11%);
5. It is found that on heat treatment, the mineral composition of oil shale changes, with the formation of rankinite $Ca_3Si_2O_7$ (17%), anhydrite $CaSO_3$ (10%), magnesite $MgCO_3$,periclase, MgO, and lime CaO. In the presence of oxygen bearing compounds such as illite and chlorite, pyrite $FeS_2$ is oxidized to hematite $Fe_2O_3$. Basic and acidic oxides react to form new minerals, such as larnite $Ca_2SiO_4$ and rankinite $Ca_3Si_2O_7$. The transformation of the shale's mineral composition on heating is accompanied by the liberation of CO, $CO_2$, $O_2$, $H_2O$, and $H_2S$;
6. By studying the mass change of the fuel shale on heat treatment in the range 25°C –950°C, we identify five stages: (I) 25°C –370°C; (II) 370°C –500°C; (III) 500°C –700°C; IV, (V) 700°C – 950°C. The maximum mass change (25%) is seen in the range 370°C –500°C;
7. Our experiments on the change in shale porosity in a nitrogen atmosphere and in air in the range 25°C - 900°C show that it may be divided into four stages: (I) 25°C –200°C; (II) 200°C –400°C; (III) 400°C –600°C; (IV) 600°C–900°C. The shale porosity is higher after heat treatment in an oxidizing atmosphere than in neutral nitrogen.

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# Progress of Steam Coal Clean and Efficient Jigging Technology in China

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**Abstract:** Under the conditions of a two stage fixed-sieve jig rejecting refuse simultaneously in the steam coal separation process, the variation in low quality coal feeding (raw coal ash fluctuating range from 30% to 55%) leads to considerable fluctuations of washing clean coal (-1.8kg/L ) ash, which affects jigging separation performance seriously. To solve this problem, the correlation between Automatic Tracking Ash Closed-loop Control and the five control factors (jig, air, water, feeding and discharge) is analyzed in this paper, and then the viewpoint that jigging separation performance depends on stratification performance and automatic tracking secondary optimal cut point is proposed. In order to put this new technology into practice, on-line identification technology is applied to measure the ash distribution of washing clean coal. According to it, an appropriate control method is selected and the cleaned coal ash index interval is amended for systematic adjustment. In this way, the contradiction among the washing clean coal ash variation, efficiency and quality of cleaned coal products is harmonized and clean, efficient steam coal jigging is performed.

**Keywords:** jigging control, washing clean coal ash, washing clean ash fluctuations interval， cleaned coal ash, cleaned coal ash index interval， ash closed-loop control, optimal ash cut point， optimal cut point.

## 1 Introduction

Low-quality coal makes up about 40% of the coal reserves in China. The uneven washing of low quality coal[1][10][7] leads to considerable fluctuations of washing clean coal ( -1.8kg/L ) ash, which is out of the control range of the traditional five control factors (jig, air, water, feeding and discharge，referred to as the five control factors) [3][9], causing significant fluctuations of cleaned coal products quality or coal lost in refuse. This has become a key problem in steam coal clean and efficient jigging.

The site validation shows that jigging ash closed-loop control has some unique advantages, such as fast feedback control response and small system lag, which are unmatched by any other coal preparation methods. Automatic tracking ash closed-loop control (referred to as "new technology" or ATACC) by using these features can break through the above obstacles and improve steam coal jigging preparation performance significantly[4][5][6][8]. For an easy illustration, the definitions of the terms used in steam coal jigging in the following analysis are listed below:

(1) Washing clean coal, cleaned coal, washing clean coal ash distribution and washing clean coal ash. In the feeding raw coal, the material which the density is less than 1.8kg/L is called "washing clean coal"; the product of washing clean coal excluding the coal lost in preparation is called "cleaned coal"; washing clean coal ash and ash content constitute "washing clean coal ash distribution"; the integrated ash of washing clean coal ash distribution at any moment is called "washing clean coal ash".

(2) Optimal separation density, optimal ash cut point, secondary optimal cut point. Under the conditions of a two stage fixed-sieve jig rejecting refuse at the same time, the two physical quantities (separation density and ash cut point) which correspond to the optimal separation status at the end of the second stage are the secondary optimal separation density and the secondary optimal ash cut point, both of which can be collectively referred to as the secondary optimal cut point.

## 2 Two key factors for improving steam coal jigging separation performance

Cleaned coal recovery depends on the separation performance. Considerable random fluctuations of

 
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washing steam coal quality make the secondary optimal cut point of a jig a random variable, so the improvement in separation performance depends on two key control elements: stratification performance and tracking secondary optimal cut point (as shown in Figure 1)── there is no good separation performance without good stratification performance; an excellent stratification performance will not produce the best separation performance if the jig does not run at the secondary optimal cut point .

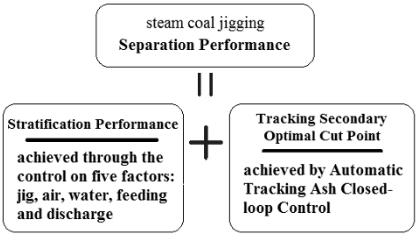


Figure 1 Two key factors for improving steam coal jigging separation performance

Figure 1 shows that the stratification performance is determined by the control on five factors: jig, air, water, feeding and discharge; the tracking secondary optimal cut point is conducted by ATACC. Therefore, it is impossible to obtain desired overall separation performance in steam coal jigging merely by focusing on the stratification performance without addressing the issue of tracking secondary optimal cut point.

## 3 Jigging stratification performance analysis

(1) Monomer distribution of washing clean coal ash. Assuming that the curve Q of washing clean coal ash and content at all levels show a normal distribution during a short period of time (5~10mins) in jigging, as shown in Figure 2a, it is known as the monomer distribution of washing clean coal ash. The abscissa is ash in ascending order from left to right; the ordinate is the grading ash content at all levels; the envelope of curve Q and the abscissa may be considered to be the total coal content, that is:

$$S = \int_{A=0}^{+\infty} Q(A)dA\,,$$ where A is ash.

(2) Misplaced area and optimal cut point. The stratification after the raw material coal is fed into a jig is a transition process during which coal is separated from refuse and the misplaced area is narrowed under the action of the five major factors. When reaching the end of the second stage of a jig, the shaded area formed with two boundaries **a** and **c** (as shown in Figure 2a) is the misplaced area where coal and refuse fail to be clearly separated. The optimal ash cut point is the left boundary Line **a** of the misplaced area. Left to the cut point is the overflow cleaned coal product and right to the cut point the discharge of waste. There a few losses of high ash coal at point **a**, but it ensures that the jigging overflow coal contains no refuse; as the cut point shifts right into the misplaced area and moves to Line **d**, refuse may enter into cleaned coal, and the farther the Line **d** is from Line **a**, the more the refuse enters into cleaned coal (this feature allows for an easy and quick artificial positioning of the optimal cut point). Conversely, when the cut point shifts left to Line **b**, high or medium-ash coal will enter into refuse, resulting in over-discharge, and the coal over-discharged is determined by the area of envelope $S_b$ of the right of Line **b**/ Curve **Q**/ the horizontal axis (see Figure 2a), that is:

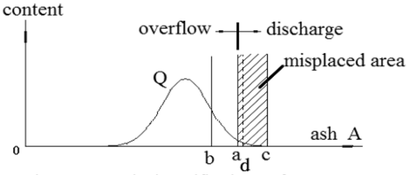


Figure 2a Desired stratification performance with narrow misplaced area

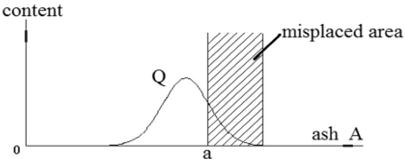


Figure 2b Undesired stratification performance with wide misplaced area

$S_b = \int_{A=b}^{+\infty} Q(A)dA$, where the farther the Line **b** is from Line a, the more the coal contained in refuse.

(3) The effects of jig, air, water, feeding and discharge on stratification performance. Figure 2b shows the poor stratification performance and too wide misplaced area when washing raw material coal arrives at the end of the second stage of a jig under the control of the five major factors. The comparison with Figure 2a shows that the poor stratification performance will lead to a large amount of coal lost in refuse and poor separation performance even if the jig runs at the secondary optimal ash cut point Line **a**.

## 4 Tracking of jigging secondary optimal cut point

### 4.1 Multi-body distribution of washing clean coal ash and the changes of optimal cut point

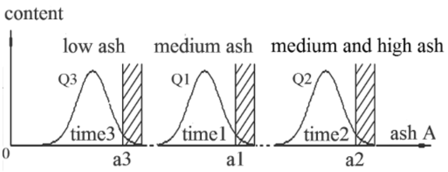


Figure 3 Multi-body distribution of washing clean coal ash and the changes of optimal ash cut point

During a longer period of time in jigging separation process, the changes in washing clean coal ash and content distribution can be decomposed into the combinations of N monomer distributions of washing clean coal ash, which is known as the multi-body distribution of washing clean coal ash. Taking N=3, as shown in Curve Q1, Q2 and Q3 in Figure 3, the corresponding misplaced area changes accordingly.

Assuming that the site coal quality change process is as follows: at moment time 1, washing clean coal ash distribution is as Curve Q1, called "medium ash" state, and the corresponding secondary optimal ash cut point is **a**1; at moment time 2 after 10mins, washing clean coal ash distribution is as Curve Q2, showing a medium and high ash, called "medium and high ash" state, and the corresponding optimal ash cut point is **a2**; at moment time 3 after another 10mins, washing clean coal ash distribution is as Curve Q3, showing a low ash, called "low ash" state, and the corresponding optimal ash cut point is **a3**. Figure 3 shows that the position of optimal ash cut point varies with the distribution locations of washing clean coal ash at different times, and it is necessary to make tracking and adjustments with the locations of the optimal ash cut point to achieve high separation performance.

### 4.2 Simplifying issues

Given favorable jigging stratification performance, cleaned coal containing no refuse and refuse containing little coal, the instant ash and changes of cleaned coal are approximately equal to those of washing clean coal ash, while the cleaned coal ash separated is measurable in real time. Therefore the implementation of "tracking and adjustments with the changes of optimal ash cut point location" can be elaborated as follows: "After calibrating the optimal cut point, making tracking and adjustments with the changes of cleaned coal ash", and thus a theoretical problem is turned into a technique that can be implemented in production. Conversely, if this control method can be maintained at the favorable run status, the correctness of the theory and techniques can be verified.

As for the jigging technologies, the conception of "the secondary optimal separation density" is intelligible, but it is unable to achieve the detection and control of automatic tracking; "The secondary optimal ash cut point", a conception proposed from the perspective of jigging control, is combined with the ash closed-loop and has excellent practicality in measurement, control and tracking.

### 4.3 Automatic tracking ash closed-loop control and application methods

#### 4.3.1 Automatic tracking ash closed-loop control

For a long time, the jigging ash closed-loop control has been confined to the mode of "constant value adjustment" which is not applicable to the cases where washing clean coal ash varies greatly. But with adding the "setting ash automatic tracking adjustment control" unit upto "constant-value-adjustment jigging ash closed-loop control", the combination constitutes " ATACC " (as shown in Figure 4), thus the tracking control on jigging secondary optimal cut point can be achieved, which significantly improves the jigging separation performance.

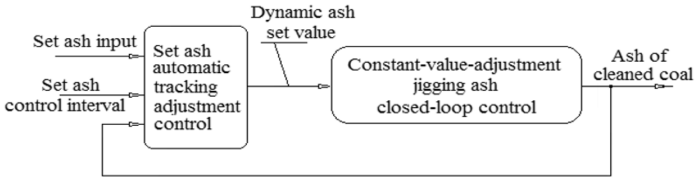


Figure 4 Automatic tracking ash closed-loop control （ATACC）

4.3.2 Selection of application methods

■ Under condition of that washing clean coal ash distribution is unknown, using "detecting washing clean coal ash interval control method" with an opening cleaned coal ash index interval after aligning the optimal cut point， the fluctuation interval of washing clean coal ash can be ascertainable.

■ When washing clean coal ash fluctuation interval is less than 8% of ash, by using "reducing cleaned coal ash interval control method " after matching the cleaned coal ash index interval with washing clean coal ash fluctuation interval, a double control effect of "efficient and quality jigging" can be acquired.

■ When washing clean coal ash fluctuation interval is greater than 8% of ash, the application of "expanding cleaned coal ash interval control method" after aligning the optimal cut point will allow for "efficient jigging". This method should combine with the process of " bunker blending and coal blending" to achieve quality optimization。

**5 On-line evaluations of changes in washing clean coal ash fluctuations interval and systematic regulation**

5.1 Tripartite contradiction of steam coal jigging and coordination

Due to the changes in washing clean coal ash, the realization of "quality" is always at the expense of "efficiency", or the realization of "efficiency" is always at the expense of "quality"(as shown in Figure 5). This contradiction can be resolved through selecting a control method and making systematic regulation of cleaned coal ash index interval according to washing clean coal ash fluctuation interval， to realize the corresponding follow-up coordination between efficiency and quality， to maximize the benefits of steam coal jigging.

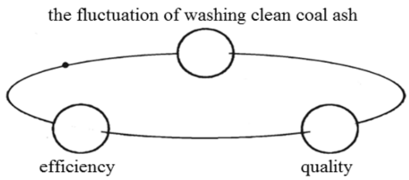


Figure 5 Tripartite contradiction of steam coal jigging and coordination

It is known that the fluctuation interval of washing clean coal ash cannot be directly detected. As the coal lost in refuse is the minimum and the cleaned coal ash is approximately equal to washing clean coal ash at the optimal cut point, the current washing clean coal ash fluctuation interval can be approximately determined through analyzing the 24-hours cleaned coal ash mathematical statistical curve.

5.2 On-line analysis of washing clean coal ash fluctuation interval and its application

5.2.1 Application to expanding cleaned coal ash interval control

Figure 6a presents the washing lump cleaned coal ash curve within 24 hours in one day before the

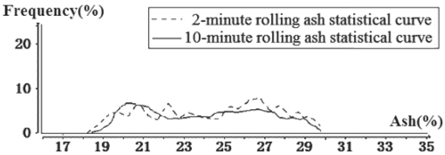


Figure 6a Washing lump cleaned coal ash curve within 24 hours before the application of ATACC

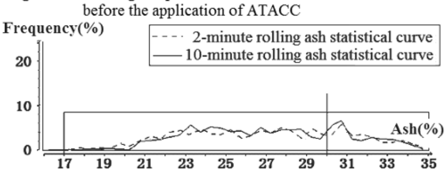


Figure 6b Washing lump cleaned coal ash curve within 24 hours after the application of ATACC

application of ATACC in Xiaoqing Preparation Plant[2]. The ordinate "Frequency (%)" in Figure 6 represents the frequency of ash appeared at the points of the abscissa and has the same meaning as the ordinate "content" in Figure 2, the envelope area of the curves is the quantity of cleaned coal. In Figure 6a, although the upper limit of statistical ash of washing lump cleaned coal has reached 30.00%, the site test results show high coal lost in refuse and it is estimated that the upper limit of cleaned coal ash index needs to be raised.

Figure 6b shows the washing lump cleaned coal ash curve within 24 hours after the upper limit of cleaned coal ash control index is extended to 17.00%~35.00% (the Linear square area in Figure 6b) and the ATACC is applied. Compared with Figure 6a, in Figure 6b, the area increment in the cleaned coal ash interval of 30.00%~35.00% makes up about 1/4 of envelope area of the cleaned coal ash statistical curves of that day, indicating a significant increase in middlings coal recovery.

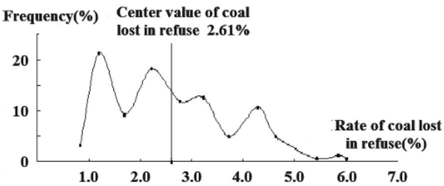


Figure 7 Statistical curves of coal lost in refuse in expanding cleaned coal ash interval control

Meanwhile, the 15-day test of integrated coal lost in refuse in the Primary and secondary of a jig after expanding the upper limit of cleaned coal ash index was made. The statistical analysis results are shown in Figure 7. The center value of coal lost in refuse decreased substantially to 2.61%.

5.2.2 Application to reducing cleaned coal ash interval control

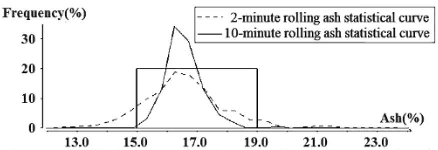


Figure 8a Matching between washing lump cleaned coal ash curve and cleaned coal ash index interval in the 1st day of statistics

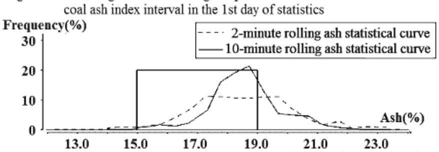


Figure 8b Washing lump cleaned coal ash curve shifted out of index interval in the 2nd day

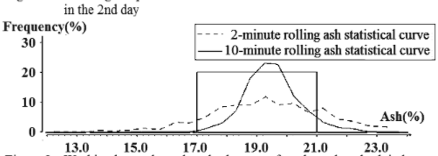


Figure 8c Washing lump cleaned coal ash curve after cleaned coal ash index interval adjustment in the 3rd day

This method requires no manual calibration of the optimal cut point. But requires tracking and capturing the main fluctuation interval of washing clean coal ash to make timely adjustment to the cleaned coal ash index interval. It works through compressing cleaned coal ash to the specified cleaned coal ash index interval, which presents a convergence effect. When the fluctuation interval of washing clean coal ash is less than 8%, the goals of both quality and efficiency can be fulfilled. This method has been applied in 5 jigging preparation plants in Tiefa coal mining area.

The 24-hours washing lump cleaned coal ash statistical curve on the first day of a certain period in Daxing Preparation Plant is shown in Figure 8a. The cleaned coal ash index interval is 15.00%~19.00%. Most 10-minute washing lump coal statistical ash falls within the cleaned coal ash index

interval, presenting the desired effect. Figure 8b presents a 24-hours lump cleaned coal ash statistical curve on the second day, and the envelope of 10-minute statistical curves exceeds the cleaned coal ash index interval of 15.00%~19.00% and significantly shifted in direction of the increasing ash, and it can be thus concluded that the changing of increase occurred in the fluctuation interval of washing clean coal ash. The absence of timely corresponding increase in the cleaned coal ash index interval will cause consider-
able coal lost in refuse. Figure 8c shows that the cleaned coal ash index interval was adjusted to 17.00%~ 21.00% on the third day. The 2-minute and 10-minute curves show that some medium cleaned coal ash exceeds the upper limit of cleaned coal ash index interval, and the cleaned coal ash index of 18.00%~ 22.00% is more suitable，that will further lower coal lost in refuse and improve cleaned coal recovery.

## 6 Summary

The new technology of ATACC solves the problem of real-time tracking control of the secondary optimal cut point and is an indispensable control factor for steam coal jigging to overcome the changes in washing clean coal ash and to improve the separation performance. An appropriate control method is selected depending on washing clean coal ash fluctuation interval and systematic adjustment is made in conjunction with the on-line analysis of washing clean coal ash fluctuation interval, to improve and support the new technology of ATACC. The recent development on jigging separation technology effectively mitigates the tripartite contradiction among the fluctuation of washing clean coal ash, efficiency and quality, further reveals the inherent law of steam coal jigging control, and provides new development space for cleanness, efficiency, quality and increasing benefits of steam coal jigging.

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# The Golden Decade of the Rapid Development of China's Coal Preparation Equipment

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**Abstract:** The period from 2003 to 2013 was the *golden decade* of the development of China's coal industry. The rapid technological advances of coal cleaning processes and equipment had led to doubled and even redoubled increase of the capacity of per unit cleaning machine, as well as the rapid enhancement of equipment operating performance and reliability and manufacturing level. These enabled China to become a world powerful country in terms of cleaning technology and advanced process equipment. Backed up with the advanced equipment, China's coal cleaning facilities have become larger in scale and are operating with highly efficient larger process equipment. For instance, the H.M.cylone reached a diameter up to 1500mm, the flotation machine has a pulp throughput of 2400m$^3$/h, and the maximum handling capacity of crusher reaches 10000t/h. Now almost all of the plants are operating with simplified cleaning systems and more diversified cleaning process under specialized management mode. The paper points out that only through continuously enhancing equipment stability and realizing intelligentization and more elaborate design of equipment, can China develop more and better coal cleaning equipment with worldwide influence.

**Keywords:** China coal preparation Large-sized coal cleaning equipment Rapid development Golden Decade Crusher Flotation Machine Screen Fine Management

## 1. A Decade's Rapid Development

Through the high-speed development over the past decade, China's coal output increased sharply from one billion tons in 2000 to 3.68 billion tons in 2013. Now the largest coking coal cleaning plant has reached a capacity up to 30.0Mt/a while the largest power coal cleaning plant has topped the capacity of 40.0Mt/a. Such a scale and developing speed deserve to be acclaimed as an historically unparalleled feat both in China and in the world. The period that saw the rapid growth of China's coal industry is generally termed as the *golden decade*.

China's coal cleaning equipment industry benefits considerably from the tremendous headway made over the *golden decade*. The treating capacity of per unit equipment is sharply increased along with the rapid enhancement of its technical performance, operating reliability and manufacturing level (See Table 1). The progress of coal cleaning equipment serves in turn as a indispensable backup for the accelerated development of coal cleaning technologies, making, as a result, a revolutionary change of such technologies. The rapid progress as stated above enables China to leap to a global powerful country in the field of coal cleaning technology and process equipment.

**Table 1 Changes of Capacities of Main Coal Cleaning Equipment**

| Equipment | Technical Parameter | 2000 | 2014 |
|---|---|---|---|
| **H.M. Cyclone** | Diameter, mm | 1200 | 1500 |



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| | | | |
|---|---|---|---|
| | Capacity, t/h | 350 | 600 |
| **Jigging Machine** | Area, $m^2$ | 30 | 36 |
| | Capacity, t/h | 240~360 | 500~650 |
| **Flotation Machine** | Cell Volume, $m^3$ | 16 | 90 |
| | Capacity, $m^3$/(h·bank） | 750 | 2400~3000 |
| **Dry Cleaning Machine** | Area, $m^2$ | 12 | 48 |
| | Capacity, t/h | 120 | 480 |
| **Screening Machine** | Area, $m^2$ | 3.0×7.3 | 4.3×9.1 |
| | Capacity, t/h(Calculated on the basis of 13mm) | 400 | 700~850 |
| **Belt Conveyor** | Belt Width, m; | 1.6 | 2.6 |
| | Belt Speed, m/s | 3.15 | 7.0 |
| | Capacity, t/h(coal) | 800 | 10000 |
| **Crusher** | Length of Roll, m | 2.0 | 5 |
| | Capacity, t/h | 300 | 10000 |
| **Horizontal Vibrating Centrifuge** | Diameter of Basket, m | 1.0 | 1.5 |
| | Capacity, t/h | 120 | 350 |

## 2. Development of Process Equipment Benefits from Enlargement of Plant Scale

As stated above, the new coal cleaning plants are all high-output, high-efficiency plants operating with simplified processes and large-capacity equipment. This calls for the availability of large-capacity and highly reliable process equipment, including the primary equipment like large-diameter cyclone, heave-medium shallow-bath separator, jigging machine, flotation cell, large-capacity compound pneumatic cleaning machine, etc, as well as the auxiliary equipment, such as large-capacity sizing crusher, vibrating screen, flip-flop screen, belt conveyor, smart rapid-open membrane filter press, horizontal vibrating and screen bowl centrifuges, and pressure filter.

With the requirements placed on the equipment as described above as an impetus, China has made much headway in the development of large-sized and highly reliable equipment, making the equipment manufacturing industry to enjoy an unprecedentedly prosperity. During the *golden decade*, in addition to the increase of number, technical capability and marketing scale of equipment producers, the overall equipment manufacturing level and quality also saw a significant improvement.

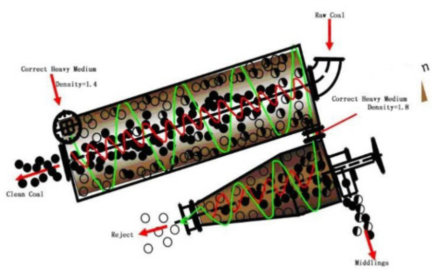


**Fig.1 Working Principle of the Gravity-fed 3-product H.M. Cyclone** **Fig. 2 A Picture of a Mechanically-agitated Flotation Machine in Operation**

## 2.1 Primary washing equipment

### 2.1.1 Large-diameter 3-product H.M. cyclone

The large-diameter gravity-fed 3-product H.M. cyclone, has particular originality and is unparalleled worldwide for its distinctive features (See Fig.1). The largest cyclone currently in successful operation has a diameter of 1500mm and a capacity of 550~650t/h. According to report, the diameter of the cyclone has now increased to 1600mm.

### 2.1.2 Jigging machine

The jigging machine with air chamber beneath screen-plate has become the key equipment in coal jigging process for its remarked working performance and easiness to undergo enlargement in size. The SKT version as a primary washing equipment in China has now reached an area of $36m^2$, a capacity up to 500~650t/h and an imperfection of $I\leq0.16$.

### 2.1.3 Flotation machine

As the coals produced in China are diversified in kinds and ranks, they vary in washability. Therefore, the requirements and conditions of flotation process are comparatively large in range. When used for treating long-flame middlings, the XJM-S flotation machine has a pulp throughput of $2400m^3/h$, with the ash of fine coal reduced from 26.60% to 13.24%, an increase of calorific value (Qnet, ad) from 3250kcal/kg to 4660kcal/kg and a recovery of combustible matters up to 87%. Shown in Fig. 2 is a mechanically-agitated flotation machine.

## 2.2 Auxiliary equipment

### 2.2.1 Crusher

A decade ago, crushers were regarded as equipment of no importance in coal cleaning process design. The crushers used were those not specifically designed for coal cleaning plant. Right now, the large-sized coal cleaning plant all place a specially high requirement on crusher performance. Listed in Table 2 are the changes of technical demands placed on crushers a decade ago and right now.

**Table 2 A Comparison of the Changes of the Demands Placed on Technical Performance of Crushers A Decade Ago and Nowadays**

| Item | Requirement | |
|---|---|---|
| | Medium-and Small-sized Coal Plant A Decade Ago | Large-sized Plant |
| **Capacity,** Mt/a | $<3$ | 5~40 |
| **Capacity of Crusher,** t/h | 300 | 8000 |
| **Material Crushed,** MPa | 40 | 200 |
| **Associated Process Required** | Hand picking of dirt (Only crushing of coal is made) | No hand picking operation (Coal, dirt and rock materials all crushed) |
| **Stability Requirement** | Not strict | Very strict |
| **Sizing Necessity** | Not required | Generally required |
| **Requirement on Product Size** | Not strict | Very strict |

It can be seen from Table 2, currently the large-sized plants all place a higher requirement on the capacity, crushing strength, operating reliability, ability for product size control and degree of optimization of a crusher selected for use. The conventional double-toothed crusher, ring roll crusher and jaw crusher are found incapable of meeting the above-mentioned requirements. Crushing is a key initial process link after raw coal goes into a cleaning plant, and the crushing station can be likened to a man's throat. The performance of crusher exerts a direct bearing on the smooth running of a plant.

The main technical data of the TCC series sizing crusher are as follows: capacity – 10000t/h; maximum feed size – 2m; maximum crushing strength – 240MPa; maximum installed capacity – 800Kw (See Fig. 3).

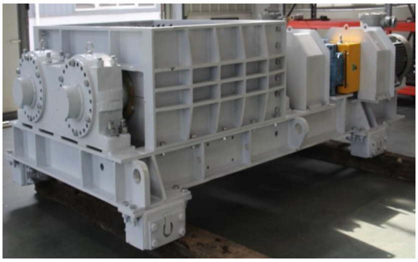

**Fig. 3 The TCC Series Sizing Crusher and the Replaceable High-strength Teeth**

#### 2.2.2 Screening machine

A great number of various kinds of screening machines is used in cleaning plants. Now, the most widely used is the 3661 version and nearly 1000 such screens are in operation in China. The use of the world largest banana screens have also met with success. For instance, a 4.9m wide and 7.3m long version is now in operation at Jinhaiyang Coal Mine while another 4.3m wide and 9.1m long version is in operation at Zhougezhuang Coal Mine.

The flip-flop screen is suitable for treating -6mm sticky and difficult-to-screen materials. The linear-motion version (See Fig. 6) works at a standard speed of 800r/min, and a screen deck possibly up to 4.3 in width. When used for sizing sticky wet material at a size of 13mm, its capacity reaches around 560t/h with a screening efficiency of 90%~95%. The mechanical version (See Fig. 5) can effectively treat a sticky wet material having a surface moisture of 15% with a material throwing force up to 50g. A screen with a width of 3m and a length of 12m can reach a capacity of 800t/h in this case.

Fig. 5 Mechanical Flip-Flop Screen

Fig. 6 Linear Vibrating Flip-Flop Screen

### 2.3 Dewatering centrifuge

The centrifuge as a dewatering equipment possesses the technical features of being high in efficiency and capacity, and less in floor space occupation. The centrifuges have seen a rapid development technologically. As compared with vertical centrifuge, the horizontal version is high in efficiency, low in generation of fines and is adaptable to a feed with a higher top size. There are only few of easily-worn parts which can be replaced in horizontal direction, facilitating, therefore, maintenance work. With the use of the centrifuge, the height of workshop can be much reduced. The ideal small coal and coal slurry dewatering equipment has different versions which work on different principles and fit for different applications. Those include horizontal vibrating-discharge version, scraper discharge version and screen bowl version, etc.

#### 2.3.1 Horizontal vibrating-discharge centrifuge

The diameter of the larger end of the centrifuge's basket is up to 1500mm. When used for treating a feed with a top size of 50mm, its maximum capacity ranges from 300 to 350t/h.

#### 2.3.2 Horizontal screen bowl centrifuge

This kind of centrifuge (Seen Fig. 7) is now widely used for coal slurry dewatering operations in coal cleaning plants. The specifications of the largest version are as follows: ID of larger end of basket: 1400mm; length of basket: 2000mm. When used for treating -0.5mm coal slurry having a density of 20%~40%, the product moisture is in a range of 14%~24% (equivalent to a treating capacity of 40~50t (dry coal)/h.

**Fig. 7 The LWZ Horizontal Screen Bowl Centrifuge**

Quick-open filter press is also a highly efficient energy-saving equipment widely used for dewatering of flotation concentrate and tailings. A large-sized version can produce each cycle a volume of 20m$^3$ of filter cake (Number of cycles: Max. 6 per hour. Filter Area: 550m$^2$). The cake moisture varies, depending on the materials treated: 17.5% (Concentrate) or 20% (Tailings).

## 3. Development of Coal Preparation Industry Promoted by Improvement of Process Equipment

The rapid development of coal preparation industry is embodied by the following aspects: 1) Simplification of cleaning systems and enlargement of plant scale; 2) High efficiency and reliability; 3) Higher degree of fine management; 4) Specialization of management; 5) Use of highly intelligent cleaning systems and equipment; 6) Diversification of cleaning methods.

### 3.1 Simplification of cleaning systems realized through using large-sized equipment

As stated above, China has currently nearly 50 over 10Mt/a-capacity super-large coal cleaning plants. Enlargement of plant scale has become a general trend in plant design. For instance, for a 12Mt/a-capacity plant, if calculated on the basis of 5280h/a, the capacity of cleaning systems is 2500t/h in total. It can be seen that the capacity per unit equipment in current use is much higher than 10 years ago. Obviously, without the large-sized equipment successfully developed in recent years, enlargement of plant scale is out of the question, to say nothing of simplification of cleaning systems.

### 3.2 High-output high-efficiency operation made possible through use of highly reliable equipment

It is stipulated by design code that a cleaning plant works 330 days a year and 16 hours a day. However, the actual work hours of many new plants exceed 6500h/a, with the capacity per unit system far greater than the designed capacity. The majority of the plants are working with a single system without stand-by system. This places an extremely high demand on reliability of process equipment.

## 4. Conclusions

4.1 The period from 2000 to 2013 my be regarded as the *golden decade* of the development of China's coal industry. During this period, a series of high-capacity highly reliable primary and auxiliary coal cleaning equipment was successfully developed and put into operation. The use of the equipment all holding independent intellectual property rights enables China to become technologically a powerful nation in the field of coal preparation.

4.2 The coal cleaning equipment and technology help each other forward and the former plays a supporting role for China's coal industry to realize enlargement of plant scale, high-output, high-efficiency operation, simplification of cleaning system, diversification of technological processes and specialized management.

4.3 Sustained efforts should be directed toward the development of elaborately designed coal cleaning equipment capable of working with higher reliability and under fine management, in a bid to produce more, better brandname coal cleaning equipment with international influence.

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## Preparation of High-Strength Gasified Coke Used for DRI from Low Rank Coal

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**Abstract**
This paper describes attempts to produce high-strength gasified coke for the production of direct reduced iron (DRI) from low rank coal. The method involves combining modified lignite with original low rank bituminous coal, briquetting and then carbonizing. The effects of content of modified coal and carbonization temperature on the qualities of coke were investigated. The results showed that the change of carbonization yield and density of coke was not obvious with the variety of modified coal content from 0 to 20%. The compressive strength of coke ascended with the increase of proportion of the modified coal. The yield of carbonization decreased gradually with temperature increased from 350 ℃ to 850 ℃, and the compressive strength and density of coke showed a trend of decreasing firstly and then increasing. The resulting carbonized briquettes had a compressive strength as high as 2.05 ~ 9.10 MPa and density of 0.87 ~ 0.96 g/cm$^3$. The process of preparated gasified coke from low rank coal, its not only realized the low rank coal clean efficient utilization, but also had a significant economic benefit from Heat - Electricity and Metallurgy coproduction.

**Keywords:** low rank coal, fine coal, clean and efficient utilization ,briquetting, gasified coke, compressive strength , DRI, metallurgy

### 1 Introduction

Low rank coal such as lignite, low rank bituminous coal resources in China are rich reserves, accounting for coal reserves and coal production more than 55 %. Generally, high moisture and low heating value of low rank coal, and 70~80 % is inferior fine coal (Qu et al.,2012; Wang＆Zhao,2012). In China, there is no report on the operation of the large scale of clean efficient utilization of inferior fine coal. The technology of gasified coal/coke briquetted and carbonized from low rank coal can get gas (180~340 m$^3$/tons of dry coal) and tar (5~9 %) and other chemical products (Chen et al.,2013; Xu et al.,2013), also can produce clean gasified materials used in fixed bed (fixed carbon 62~85 % and reactivity >50) (TYUT,2013). It is widely recognized as a science, technological and reasonable approach about conversion of low rank coal. Based on the production of gasified coal/coke from low rank coal, gasification gas coupling with coke oven gas replace of traditional reductant for the production of direct reduced iron (DRI), can reduce the total cost of DRI about 9 %, and have a significant economic benefit.

At present, the technology of producing high-strength coal/coke briquetted and carbonized from low rank coal had been widely researched (Gu＆Perry,1992; Mollah et al.,2015). French et al. (2012; Mori et al.,2013; Chow et al.,1981) were able to produce a high-strength product by hot briquetting and then carbonizing from several low rank coal. Mori et al. (2011) briquetted Victorian brown coal at range of temperatures from 25 ℃ to 230 ℃, then carbonized and obtained a coke of density of 1.1~1.3 g/cm$^3$ and strength of 28~37 MPa. However, high qualities of equipment required for hot briquetting, and poor adaptability to the quality of coal. The high moisture content and volatile in the low rank coal formed a large number of cracks during carbonization progress. Taylor＆Hennah (1992) had be researched the char of non-coking coal combining with tar and pitch in cold briquetting and carbonizing, and noted that



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char of non-coking coal combining with tar and pitch in cold briquetting and carbonizing, and noted that tar-derived binder had displacements and adhesion of char particles, the strength of carbonized briquettes reached 12 MPa. This process requires the pyrolysis for producing char and tar-derived as binder before briquetting and carbonizing. Bayraktar＆Lawson (1984) applied of mechanical pressures of 113~212 MPa on briquetting in Turkish lignites, then carbonized and obtained high-strengh coke, but parts of raw lignite demineralized with HCl. Based on those problems, we attempt to produce high-strength coke from lignite and low rank bituminous coal. The method involves modified lignite with original low rank bituminous coal mixing, briquetting and carbonizing. This method have low cost and simple progress, and the coke with less cracks.

## 2 Experimental Section

### 2.1 Materials

Low rank bituminous coal (SM) was obtained from Shen Mu, crushed and sieved to ＜3 mm. Modified coal (MC) was produced by lignite that modified in CO and subcritical $H_2O$ system, crushed and sieved to ＜0.125 mm. The analyses of samples of the SM and MC are shown in Table1.

Table1. Proximate analysis and ultimate analysis of materials

| Sample | proximate analysis/ wt-%, ad | | | | ultimate analysis/ wt- %, d | | | | | G |
|---|---|---|---|---|---|---|---|---|---|---|
| | *M* | *A* | *V* | FC | C | H | O | N | S | |
| SM | 5.13 | 3.88 | 25.72 | 65.27 | 79.75 | 4.22 | 10.66 | 0.97 | 0.29 | 0 |
| MC | 0.96 | 13.46 | 27.49 | 58.09 | 74.75 | 5.73 | 4.93 | 1.46 | 0.40 | 92 |

### 2.2 Preparation of carbonized briquettes and measure of compressive strength

The mixture of SM, MC and water was employed as feedstock. About 40 g as feedstock was placed into a cylinder shaped die set of 43.3 mm internal diameter pressed over 3 min under vacuum until the force reached 100 KN and the force held for 10min in TYE-2000B briquetting setup. The briquettes were disc shaped with the diameter of 43.5 mm and height of 22.0 mm. The briquettes dried at room temperature oven for 48 h, then heated in atmospheric flow of $N_2$ at a rate of 3 ℃/min up to 350 ~ 850 ℃ with a holding period of 1h and then cooled to ambient temperature.

The mechanical strength of the coke was measured at ambient temperature by means of compression tests on a testing apparatus. Three to five samples prepared under the same conditions were subjected to tests. The compressive strength $\sigma_C$ was calculated using Equation (1), as used by Mollah et al. (2015):

$$\sigma_C = (4F / \pi D^2)(H / D)^{0.5} \qquad (1)$$

Where force, F, is determined from the maximum load the coke withstood, D is the diameter and H is the height of the coke.

## 3 Results and discussion

### 3.1 Effects of the modified coal content

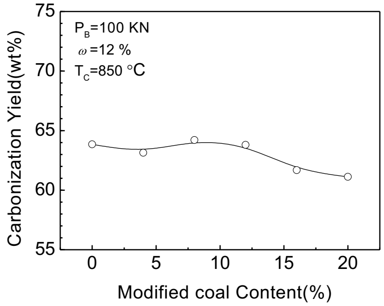


Fig.1. Effect of modified coal content on carbonization yield.

Fig1 showed the yield of coke from carbonization with $T_C$ =850°Cas a function of modified coal content. The carbonization yield, $Yield = (m_C / m_B) \times 100$ , where $m_C$ is the final weight after carbonization and $m_B$ is the weight of briquette. As seen in Fig1, the change of carbonization yield was not obvious at the content of 0~12%, further but gradually to 61.1% at the content of 20%. Carbonization yield is far lower than that of industrial 72%~78%. This mainly depends on the quality of coal, and also be influenced of carbonization conditions and carbonization furnace. With the increase of carbonization temperature, followed by drying, decomposition and degassing progress (Xu et al.,2013). Due to the difference of moisture and volatiles between modified coal and low rank bituminous coal, the increase of modified coal content led to decrease of carbonization yield. As the effects of the donor-hydrogen and synergetic effects of binder, formed coke yield decreased gradually(Song et al.,2015).

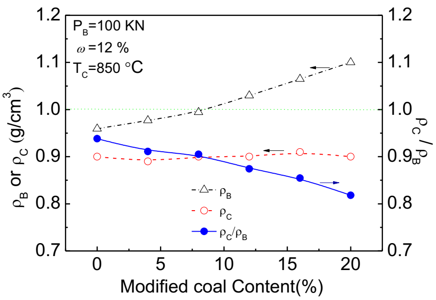


Fig.2. Effect of modified coal content on $\rho_B$, $\rho_C$ and $\rho_C/\rho_B$.

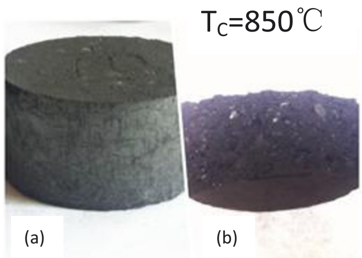


Fig.3. The images of carbonized briquette(a)and its fracture surface(b)

The effects of modified coal content on $\rho_B$, $\rho_C$ and $\rho_C/\rho_B$ is demonstrated in Fig.2. The density of briquettes, $\rho_B$, was calculated on a moisture-free basis by an equation, $\rho_B = (1-\omega)\rho$, where ω and ρ were moisture content and measured apparent density of the briquette, respectively. As seen in Fig.2, $\rho_B$ greatly increased from 0.96 g/cm$^3$ to 1.10 g/cm$^3$ at the content of 0~20%. Due to increase of content of modified coal, the briquetting at such a press effectively eliminated interparticle spaces and increased contact area between adjacent particles. The density of coke, $\rho_C$, kept 0.89~0.91 g/cm$^3$. With the increase of carbonization temperature, the briquettes would soften, melt and curing process, led to keep the density of coke about 0.90 g/cm$^3$. The ratio of the density, $\rho_C/\rho_B$, decreased gradually from 0.94 to 0.82. Fig.3 displayed the images of carbonized briquette (a) and its fracture surface (b).

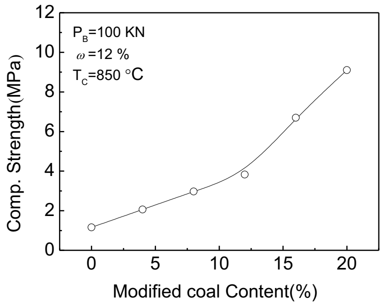


Fig.4. Effect of modified coal content on compressive strength of carbonized briquettes.

The compressive strength of coke as a function of modified coal content is shown in Fig.4. The strength increased linearly with increasing modified coal content. It increased from 1.16 MPa to 3.83 MPa at the content of 0~12%, and then quickly increased to 9.10 MPa at 20%. We can find that the density of coke kept 0.90 g/cm$^3$ with the variety of content of modified coal, but the strength of coke improved obviously. It was believed that, during the carbonization, a large amount of colloid come from modified coal and adhesion adjacent particles together (Arslan, 2006), then solid lead to improve compressive strength significantly.

**3.2 Effects of the carbonization temperature**

Briquettes were prepared with the content of modified coal was 16% and carbonized at $T_C$=350~850 °Cfor investigating the effects of $T_C$ on the properties of the resulting coke. The results are summarized in Table2. The density of coke and $\rho_C/\rho_B$ showed a trend of decreasing firstly and then increasing at 350~850 ℃ The minimum$\rho_C$ was 0.87 g/cm$^3$ in 450 ℃ and $\rho_C/\rho_B$ was 0.89. The carbonization process of briquette includes depolymerization stage, polycondensation stage and crack formation stage (Wang et al., 2014). The macromolecular structure of briquettes has been broken and turned to be free radicals and other small molecular compounds, and led to the expansion of briquettes. Further increasing of temperature, the structure is compact and the volume decreased. It decreased from 2.49 MPa to 2.05 MPa at 350~450 ℃,and then quickly increased to 7.22 MPa at 850 ℃ It seemed that an increase to briquette/coke strength occurred in $T_C$ > 450 ℃ The yield of carbonized decreased gradually from 87.4% to 61.6% with the variety of temperature of 350~850 ℃

Table2. Effects of the carbonization temperature on the properties of coke

| $T_C$ (°C) | $\rho_B$ (g/cm$^3$) | $\rho_C$ (g/cm$^3$) | $\rho_C/\rho_B$ | $\sigma_C$ (MPa) | Yield (%) |
|---|---|---|---|---|---|
| 350 | 0.97 | 0.92 | 0.95 | 2.49 | 87.4 |
| 450 | 0.98 | 0.87 | 0.89 | 2.05 | 78.9 |
| 550 | 0.98 | 0.88 | 0.90 | 3.63 | 73.8 |
| 650 | 0.97 | 0.89 | 0.92 | 4.10 | 69.4 |
| 750 | 1.00 | 0.94 | 0.94 | 5.64 | 65.1 |

| | | | | | |
|---|---|---|---|---|---|
| 850 | 1.00 | 0.96 | 0.96 | 7.22 | 61.6 |

**4 Analysis of the benefit of Heat - Electricity and Metallurgy (DRI) coproduction based on briquette/coke gasification**

The technology of Heat - Electricity and Metallurgy (DRI) coproduction, gasified coke produced from low rank coal as carrier, realized the low rank coal clean efficient utilization about “hierarchical utilization, energy integration, integrated production”. In addition, the cost of traditional reductant is about 15% of the total cost during the process of DRI. The replace of traditional reductant with gasification gas coupling with coke oven gas for DRI, can reduce the cost of reductant about 50 ~ 60%. There have a significant economic benefits, the total cost without tax is 1140 yuan/ton and marginal cost is 1070 yuan/ton during DRI.

**5 Conclusion**

Briquettes from lignite and low rank bituminous coal were prepared with press of 100 KN and modified coal content of 0 ~ 20% but without binder and carbonized by heating to 850 ℃can produced density of 0.88 ~ 0.96 g/cm$^3$ , high-strength gasified coke for the production of direct reduced iron (DRI) . This process not only realized low rank coal clean efficient utilization, but also had a significant economic benefit from Heat - Electricity and Metallurgy (DRI) coproduction. The results show that the change of carbonization yield was not obvious with the variety of modified coal content from 0 to 20%. The density of coke as high as 0.89 ~ 0.91 g/cm$^3$ and the compressive strength ascended with the increase of the proportion of the modified coal. Its strength up to 9.10 MPa when modified coal content was 20%. The yield of carbonized briquettes decreased gradually with the temperature from 350 ℃to 850 ℃, and the compressive strength and the density of coke showed a trend of decreasing firstly and then increasing. The resulting carbonized briquettes had a compressive strength as high as 2.05 ~ 9.10 MPa and density of 0.87 ~ 0.96 g/cm$^3$.

**Acknowledgements**

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# Efficient & Rational Utilization of Low-Rank Coal Resource by Using Dadi New Type Coal Deep Processing Technology

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**Summary:** Efficient and rational utilization of low-rank coal, especially lignite, was always a worldwide problem. With its sufficient experience accumulated in the field of coal beneficiation and deep processing, Dadi Engineering Development Group developed a series of advanced technology named TDM Lignite Low-Temperature Pyrolysis. With such series of technology, it can not only realize low cost and high efficiency low-temperature pyrolysis deep processing of lignite, but it is especially suitable for low-temperature pyrolysis deep processing of granulum long flame coal and oil shale resource.Our new technology can create quite a good favourable opportunity on safety, economical, efficient and reasonable quality-based utilization of lignite, granulum long flame coal and oil shale, on realizing healthy development of upstream and downstream production chain, on developing the resource value, and on upgrading competition and anti-risk capacity of enterprises. This article introduced Dadi TDM lignite low-temperature pyrolysis technology, detailing main characteristics and advantages of this technology, proposed cohesion of lignite low-temperature pyrolysis technology and upgrade coal preparation and processing, thought that utilizing lignite low-temperature pyrolysis technology to realize lignite deep processing will be right way that improve the value and competitiveness of lignite processing products, quality-based, high-efficient and rational use of lignite resources in the future.

**Key Words:** Low-rank coal, Lignite, Deep processing, Low-temperature pyrolysis, TDM technology, quality-based utilization in high efficiency, upgrade coal preparation and processing

Efficient and rational using of low-rank coal, especially for lignite, has been becoming a worldwide problem need to be solved. In China, Along with the up and down change of international energy situation recent years, because of the lack of effective technical support of upgrading deep processing and conversion, together with the restriction from the downstream coal chemical industry, pithead power generation and other industry chains which are affected by technical adaptability, regional environmental carrying capacity, industry overcapacity and other factors, also with the weaker macro economic situation, how to development and utilization of the lignite resources which have been invested heavily and large-scale developed in the past ten years, has become a heavy burden to related energy companies and local governments. Therefore, to develop an advanced, safe, environmental protection, economic coal deep processing technology, to realize the efficient and rational utilization of low-value resources, to ensure the long-term sustainable development of energy development enterprises and mining areas, and then to use these effective new technology, to promote the industrialization development in China and in the worldwide other undeveloped low-rank coal resources domains, all these are the issues to be faced in international coal field.

With the proved application of practice in recent years, based on the low-temperature pyrolysis deep processing technology, firstly at a lower cost from lignite extract the rich and valuable oil and gas

 
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products before the coal combustion, to realize the quality-divided efficient and rational use of lignite, it has become the right direction and consensus of solving the low value of lignite direct use, weak market competitiveness and difficult to achieve the development of industry chain and other difficulties. [1] [2] However, how to choose the low-temperature pyrolysis technology, how to improve the efficiency and quality of raw materials, how to improve the yield and quality of pyrolysis products, how to optimize the cohesion of low-temperature pyrolysis and the coal preparation technology in view of the medium to high ash and sulfur content elderly lignite, how to promote the integration development of the upstream and downstream industry chain by reasonable selecting technology, optimizing the industrial layout, all these are worth for studying and thinking. This article will combine with the Dadi new type lignite low-temperature pyrolysis technology to discuss in-depth.

**I. Development and influence of lignite low-temperature pyrolysis technology**

Lignite pyrolysis reaction process includes the first pyrolysis reactions and second pyrolysis reactions under the first reactions' condition within the device, in these reactions lignite were heated isolating air in the pyrolysis device to generate the primary thermal decomposition products and with the inter reaction between primary products. The feed coal with different coal quality characteristics, under the condition of particular conditions, after a series of thermal decompositions and synthesis reactions, eventually get the products with different composition, yield and quality, mainly for the high carbon content semicoke, light coal tar products fulling of hydrocarbon and phenolic, as well as the burnable ingredients coal gas products such as carbon monoxide, methane, hydrogen gas.

Lignite low-temperature pyrolysis is usually proceeded under the atmospheric conditions, but there are many factors which affect the pyrolysis process, mainly are the feed coal property (coal quality and composition, particle size and chemical properties, etc.), pyrolysis reaction conditions (heating mode, heating rate, heating time and heating final temperature an atmosphere conditions, etc.), [3]. These factors interact with each other in the pyrolysis device, mutual effect, restrict each other, interaction, constituting a complete pyrolysis reaction process.

Different pyrolysis technology not only affects the utilization rate of raw coal, product yield and composition, but also brings important influence to the project economy, subsequent use of products and industry chain development.

According to the different classification, lignite low-temperature pyrolysis technology mainly have the low-temperature pyrolysis types of gas heat carrier, solid heat carrier and other special type (external heat, heat accumulating type); According to the different types of heat carriers, they can be divided predominantly into porcelain ball or semicoke solid heat carrier, internal or external heated types gas heat carrier, external heat accumulating heat pipe and heat conduction oil and other indirect pyrolysis types; According to the different device types, it can be divided into vertical tower type, air tube fast hybrid type, rotating drum-type, horizontal rotating bed, van mesh belt furnace, circulating fluidized bed and other technology types.[4]

In recent years, the above various types technologies were carried out small and pilot scale tests project in China. Each process has unique advantages and characteristics, the vertical tower furnace process uses the traditional gas heat carrier, the system is simple, using the successful application experience and the foundation of the traditional vertical heater for decades, technical threshold is low, but it is sensitive to the particle efflorescency and fragmentation, low gas calorific value, lower oil yield than solid heat carrier technology[5] [6]; Solid heat carrier technology, having higher theory oil yield, higher gas calorific value, 100% used of raw materials, but the system is complex, immature, hard to separate to dust, oil and gas, all feed coal should be broken to under 6 mm , upgrade coal or semicoke usually need to be briquetted; Rotating drum-type needs uniform heating, having higher theory oil yield, higher gas calorific value, the advantages of 100% using raw coal, but also with the problems of high dust content of oil and gas, hard to separate to dust, oil and gas, system is complex, large scale area covering, and the higher investment; Horizontal rotating bed has potential advantage of large scale, there are two ways of internal heating type gas heat carrier technology and external heat accumulating pipe, internal heating type has been industrialized in the United States and China, but the investment is high, the oil yield is not high. And the external heat accumulating pipe type has the advantages of higher oil yield, higher gas calorific value, good security, but it has not been large-scale operation. In general, rotating bed process has a higher investment cost, raw material utilization ratio is relatively low; Van mesh belt furnace process is heated for the internal heating type fixed bed section, it is sensitive to fine material, large scale cost is too high. It has not been industrialized; BT circulating fluidized bed technology is the effective combination of solid heat carrier pyrolysis and circulating fluidized bed power generation, solving the difficult problems of the hard fine and carbocoal handles of traditional solid heat carrier technology, and the poor comprehensive benefits, shortening the industrialization process, it is a kind of reasonable industrial chain integration, it is more valuable for the "coal mining-pyrolysis – electricity Generation – chemical engineering" integration project, but it has not yet been achieved a large-scale industrialization. [7] [8]

summarized all kinds of technical progress of the pyrolysis process, there is not one technology in the field of lignite low-temperature pyrolysis to get breakthrough success and realize the large-scale engineering application. Although the similar technologies and devices which are applied in long-flame coal, are all made a success of industrialization in small particles oil shale and other material, but in the field of lignite low-temperature pyrolysis it has not achieved the same results, which is mainly determined by the characteristics of lignite, due to in different areas the lignite characteristics and coalification degrees have great difference, their coal and rock characteristics, coal compositions, particle space structures, particle size distributions, thermal stability and so on are the important influence of pyrolysis technology selection and application. Also, any pyrolysis technology and devices must be combined with its own technical characteristics and the application ranges, to ensure its adaptability of processed coal resource quality, product structure, market demand, to improve the comprehensive technology economy of project and long-term competitiveness, and to achieve the technical advantages and project construction goal.

**II. Brief introduction of Dadi TDM lignite low-temperature pyrolysis technology**

Dadi TDM lignite low-temperature carbonization process, its basic pyrolysis principle and main device design are fully learned from the advantages of the traditional vertical furnace most widely used in China and abroad, and from the Lurgi three sections furnace. Through further studying the inherent characteristics of lignite, the advanced, reasonable and efficient internal structure innovation and system technology optimization, it can effectively solve the low-temperature pyrolysis problem of traditional vertical furnace which cannot adapt to lignite in small particles long-flame coal and oil shale area.

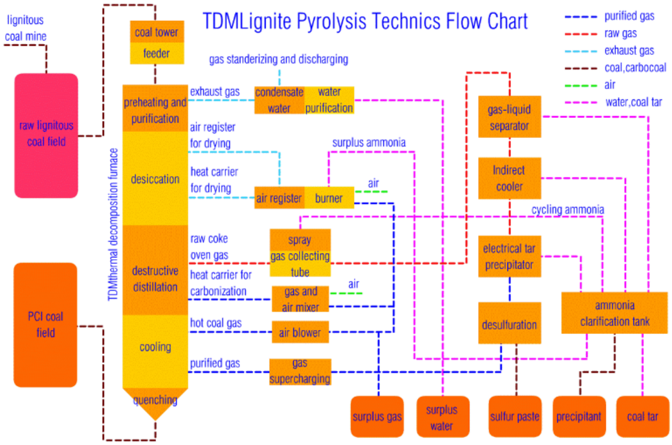
Dadi TDM lignite low-temperature pyrolysis process is based on the internal heating type vertical shaft furnace structure, although compared with solid heat carrier process, external heated process, there are some disadvantages on lower oil yield, raw gas heat value, but it inherits the advantages of the China vertical furnace and Lurgi three sections furnace, realistic and feasible to resolve the pyrolysis problems of small particle materials especially lignite. As with obvious comprehensive technical and economic advantages, it is worth for popularization and application.

**III. Main characteristics and advantages of Dadi TDM low-temperature pyrolysis process**

A. The feed particle size of feed material particles can be as lower to 1 mm, ultrafine pulverized coal can achieve nearly 100% raw material used directly.

B. Integration of external combustion and internal heat drying, internal combustion and internal heat pyrolysis, semicoke cool coal gas quenching. Tower type furnace structure. From the top to the ground, materials continuously from the top to down directly pass through the preliminary drying section, dry distillation section, dry quenching section, short process path, high efficiency and energy saving.

C. Adapt to the different raw materials, it can stably operate and adjust under the condition of generating surplus gas or not. For the high-quality raw coal with low ash, low sulfur and low phosphorus, and other raw materials such as oil shale, the semicoke volatiles can be controlled below 10%; For the lignite with high content of ash and sulfur content, it can realize to produce products under the condition of absence of surplus gas, the upgrading coal volatiles can be retained more than 25%, as it is prime electricity-coal, its market space is very large.




Flow diagram of TDM lignite low-temperature pyrolysis process

D. Using cold coal gas to realize dry semicoke quenching, it can reduce the energy losses from the drying process of wet semicoke quenching. If needed can be blown into a proper amount of steam in pyrolysis section to improve quality and yield of gas, with low oxygen content, high hydrogen content.

E. Tar recovery is more than 70%, and high light oil yield.

F. It can make full use of the drying exhaust gas and waste heat of heat semicoke, high heat utilization rate, increasing pyrolysis efficiency. Excretion water vapor in dry section is recycled condensation easily and also easy to use as resource. Dry quenching is greatly reduces the waste water quantity.

G. coal particle with carbonization upgrading has low damage rate, good thermal stability, it reduces the product molding processing costs.

H. It is easy to realize automation with the advanced, safe and controllable control technology.

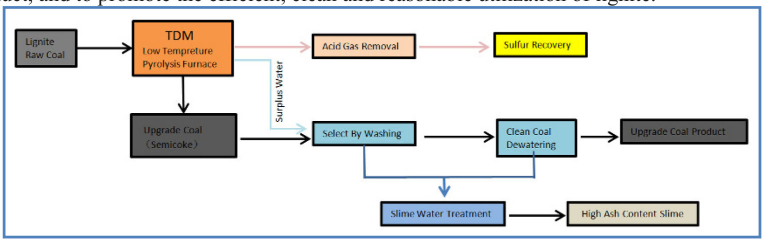
I. The tower furnace structure is most saving land occupation, and the large-scaled furnace of 40t/h has a big unit capacity, it can greatly simplify the production system. Advanced technology, low investment, low operation cost can guarantee the comprehensive technical economy of project.

Main device structure image of Dadi TDM low-temperature pyrolysis furnace
(single furnace 40 t/h)

**IV The cohesion of lignite low-temperature pyrolysis technology and upgrade coal preparation and processing**

China's lignite is mainly the elderly lignite, the ash content is usually about 20%, sulfur content is higher than the younger lignite and mainly is organic sulfur. According to experiences, the mixing of the roof and floor coal gangue from the coal mining procedure made the ash content becoming higher, lignite ash content is mainly the internal ash and prone to argillization easily, usually the ash content in the duff dust coal is higher than in the lump coal, this is mainly caused by lignite efflorescency and fragmentation, the internal ash retained in the lignite particles and interlayer is stripping into the duff dust coal. To stabilize the lignite quality, reduce product ash and sulfur content, it is usually only through discharging gangue of lump coal to handle the mixed coal gangue, but for the internal ash of lump coal, high ash content of duff dust coal, and organic sulfur, it has a poor direct washing effect, the slime water system is difficult to function efficiently. If adopt the air separation ash fall, although to avoid the effect of argillization, but air separation has no separation effect to fine particles and low-density lignite, and it will cause the further breakage of lignite and serious dust pollution [9].

To solve this problem, in addition to improve the traditional coal preparation process, increase the capacity of slime water treatment system, implement partial ash fall upgrading, the better way is combined with the low-temperature pyrolysis technology and the coal preparation processing technology, reaching the effective combination of lignite drying modification and ash reduction, desulfurization upgrading, so that to obtain the lower content of moisture, ash and sulfur, higher calorific value quality coal product, and to promote the efficient, clean and reasonable utilization of lignite.




Cohesion of lignite low-temperature pyrolysis and upgrade coal preparation and processing

**V Conclusion**

Lignite low-temperature pyrolysis deep processing is the right way to improve the lignite value-added and competitive ability of products, quality-divided, high efficient and rational use of resources. By adding the lignite low-temperature pyrolysis link, the coal mine can be free from the extensive pattern of just selling simply the low added value raw coal, and the upgrading coal (semicoke), coal tar, coal gas, that all obtained from the low-temperature pyrolysis are the important raw materials to realize deep conversion and industrial chain extension in generate power, coal chemical industry application.

Pyrolysis furnace with China's traditional vertical furnace structure can only handle the big granularity lump coal, it can't handle the poor thermal stability, efflorescency and fragmentation lignite. Dadi TDM lignite low-temperature pyrolysis technology, through a series of technological innovation and development, is becoming better and approaching perfection day by day. Since 2012, using the 0.15 t/h, 1 t/h test device for a series of small industrial operation tests have proved that it can better solve lignite low-temperature pyrolysis problem and the granular coal, oil shale. It can not only gain higher yield of coal tar, coal gas and high quality upgrade products, but also has the advantages of cool coal gas dry quenching, waste heat and water recovery, less pollutants emission, it also has a significant comprehensive technical and economic advantages. For the low heat value by-product coal gas, except using in power plant directly, we can utilizing coal gas nitrogen removal technology to increase the coal gas heat value, and a more improve on the utilizing value of coal gas [10]. For the high ash and sulfur content of lignite, through the effective combination of low-temperature pyrolysis and upgrade coal preparation and processing, it can effectively solve the problem of lignite ash reduction and desulfurization, and can provide the condition for lignite to realize a high quality, efficient, clean and reasonable usage.

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# Novel additives obtained from low grade biomasses for coke-making

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**Abstract**

In this study, a rice straw (RS) and a fir sawdust (SD) were extracted by 1-methylnaphthalene (1-MN) in an specially designed autoclave at different temperatures (250-350$^{o}$C). Two extracts, low molecular weight extract (termed as Soluble) and high molecular weight extract (Deposit), were obtained. The feasibility of using these two extracts as additives in coking coal blends was investigated for the first time. The Soluble and Deposit addition into the coke-making coal blends obviously increased the resolidification temperature and plastic range, thus improving the thermoplastic properties of the coal blends, especially for the Soluble and Deposit obtained at higher temperature. Solubles and Deposits contained significantly higher carbon content, lower oxygen groups (such as –OH groups) and extremely lower ash content in comparison with raw biomass. These unique properties of the Solubles and Deposits could explain their good performance in thermoplasticity modification of the coal blends. The addition of Deposits and Solulbles also markedly enhanced the quality of coke with increased coke strength after reaction (CSR) and reduced coke reactivity index (CRI). Therefore, it is highly feasible to use Soluble and Deposit prepared from degradative solvent extraction of biomass as additives to coking blends.

**Key words**: additives; solvent; extraction; biomass; coke; thermoplasticity; coal blending

## 1. Introduction

Coke, which is an important industrial material in iron and steel industry, acts as carbon skeleton , main fuel, reducing agent and carburization carbon in blast furnace. The coking coals need to have special physical and chemical properties, resulting in their much higher price than other coal used for electricity generation. Due to the rapid consumption and increasingly high price of prime coking coals in past decades, coal blending is a common practice in coke-manufacturing industry to lower the cost of coke-making [1, 2]. However, the addition of non-caking coals into coal blends usually causes an obvious reduction in the thermoplastic properties of coal blends and the quality of subsequently produced coke. For these reasons, various carbon-containing materials such as pitch, tyre recycling residue, plastics and petroleum coke have been used as additives in coke-making [3-5]. However, these additives usually contain high sulfur content, emit high amounts of pollutant and result in a significant reduction in the quality of coke. Therefore, it is of great importance to find or prepare suitable additives for coke-making.

The authors have presented a degradative solvent extraction method to upgrade and convert various types of biomass wastes and low-grade coals [6-9]. This method treats the carbonaceous resources in a non-polar solvent below 350 $^{o}$C under an inert atmosphere and then obtains three solid fractions after separation, which are low molecular weight extract (termed as Soluble), high molecular weight extract (Deposit) and residue (Residue).The two extracts, namely Soluble and Deposit, have rather high carbon content and low oxygen content. In addition, Soluble and Deposit are almost free from water and ash. Furthermore, all Solubles can completely soften below 120 $^{o}$C, and part of Deposits derived from woody biomass could completely soften below 300 $^{o}$C. These unique properties may make Soluble and Deposit highly potential as additives for coke-making. These biomass-derived additives also has the advantage of reducing the $CO_2$ emission over other additives derived from coal and petroleum, because biomass is carbon-neutral.

In this study, the feasibility of using these two extracts (Soluble and Deposit) obtained from



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biomasses as additives for coke-making were examined for the first time. The influence of the extracts on thermoplastic properties of coal blends and the quality of subsequently produced coke were determined and discussed.

## 2. Experimental

**Samples and solvents used.** Two typical biomass wastes, a rice straw (RS) and a fir sawdust (SD) from China, were extracted by 1-methylnaphthalene (1-MN) at 250, 300, and 350°C with the residence time of 30 min in a specially designed batch autoclave. After a series of separation procedure, three solid products, namely two extracts (Soluble and Deposit) and the unextractable fraction (Residue), were obtained. The gaseous products (Gas, mainly consisting of $CO_2$) were collected in a gas bag and quantitatively analyzed by a gas chromatograph. The yields of liquid products (Liquid, mainly $H_2O$) was determined by difference. The detailed procedure of the extraction experiments has been described in our previous works [6, 7].

**Gieseler test.** Gieseler plastometry is a widely used method for evaluating coal thermoplasticity. In this study, a gas coal (GC, from China) with 36.74 wt% volatile matter on a dry and ash free basis was employed as the base of the coal blends, which contains 77.28 wt% carbon, 1.68 wt% hydrogen, 3.64 wt% nitrogen and 17.40 wt% oxygen (dry basis), respectively. The addition rate of the Soluble or Deposit to coal KZ was 2.0 wt%. The thermoplastic properties of the raw KZ and blends were measured by a constant-torque Gieseler plastometer, according to the GB/T 25213-2010 standard procedure. The fluidity of samples were recorded in dial divisions per minute (ddpm) with the temperature heated up from 300 °C to 550 °C at a heating rate of 3 °C/min. Four important parameters reflecting the thermoplasticity of the coal or blends were obtained: (i) Ts: softening temperature (ii) Tf: the temperature corresponding to maximum fluidity (iii) Tr: resolidification temperature (iv) plastic range (pr): Tr–Ts (v) MF:maximum fluidity (ddpm).

**Coke quality determination.** The carbonization experiments were conducted in a laboratory scale crucible coking oven. On each run, 100 g samples were heated from ambient temperature to 300 °C at a rate of 10 °C/min and then to 1050 °C at a rate of 3 °C/min. After the temperature was held at 1050 °C for 30 min, the crucible was cooled down to room temperature to obtain crucible coke. 20 g coke with the size in the range of 3-6 mm was reacted with $CO_2$ at a flow rate of 5 L/min for 2 h at 1050 °C. The weight percentage of weight loss of coke after reaction was designated as coke reactivity index (CRI). The micro-strength of coke after reaction with $CO_2$ was designated as coke strength after reaction (CSR) [10].

## 3. Results and Discussion

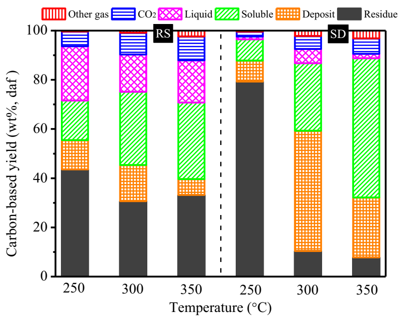


**Fig.1** The yield distributions of products obtained from degradative solvent extraction of RS and SD

**Yield distributions of products.** The carbon-based yield distributions of products obtained from degradative solvent extraction of RS and SD is shown in Fig.1**.** It shows that the yield of Soluble increased with temperature for both RS and SD, while those of Deposit reached a maximum at 300 °C. The sum of yields of Soluble and Deposit was rather high at elevated temperature (300, 350 °C), and especially reached as much as 81% for sawdust at 350 °C. Most of carbon in raw biomass was retained in Solubles and Deposits, especially for SD.

**Table 1** Ultimate and proximate analyses of biomass and extracts obtained at 250-350 °C

| T(°C) | Sample | Ultimate analysis(*wt*%, daf) | | | | Proximate analysis (*wt*%,db) | | | Atomic ratio | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | C | H | N | O[b] | VM | A | FC | H/C | O/C |
| | RS | 49.01 | 5.54 | 0.81 | 44.64 | 75.39 | 9.98 | 14.63 | 1.36 | 0.68 |
| RS-250 | Soluble | 72.76 | 6.82 | 1.29 | 19.13 | 77.46 | 0.14 | 22.40 | 1.12 | 0.20 |
| | Deposit | 67.33 | 6.11 | 1.47 | 25.08 | 66.76 | 0.36 | 32.88 | 1.09 | 0.28 |
| RS-300 | Soluble | 75.78 | 6.75 | 1.43 | 16.04 | 76.49 | 0.00 | 23.51 | 1.07 | 0.16 |
| | Deposit | 73.29 | 5.91 | 1.54 | 19.26 | 62.75 | 0.36 | 36.89 | 0.97 | 0.20 |
| RS-350 | Soluble | 78.80 | 6.73 | 1.47 | 12.99 | 68.57 | 0.00 | 31.43 | 1.03 | 0.12 |
| | Deposit | 79.68 | 6.23 | 1.81 | 12.27 | 46.88 | 1.54 | 51.57 | 0.94 | 0.12 |
| | SD | 53.11 | 5.84 | 0.08 | 40.97 | 84.39 | 1.23 | 14.37 | 1.32 | 0.58 |
| SD-250 | Soluble | 74.12 | 6.97 | 0.64 | 18.26 | 81.49 | 0.00 | 18.51 | 1.13 | 0.18 |
| | Deposit | 69.45 | 6.06 | 0.43 | 24.06 | 72.71 | 0.00 | 27.29 | 1.05 | 0.26 |
| SD-300 | Soluble | 75.97 | 6.46 | 0.45 | 17.12 | 79.06 | 0.25 | 20.68 | 1.02 | 0.17 |
| | Deposit | 74.93 | 5.77 | 0.19 | 19.11 | 68.56 | 0.75 | 30.70 | 0.92 | 0.19 |
| SD-350 | Soluble | 76.60 | 6.17 | 0.45 | 16.78 | 73.19 | 0.00 | 26.81 | 0.97 | 0.16 |
| | Deposit | 73.54 | 5.52 | 0.40 | 20.54 | 56.15 | 0.16 | 43.69 | 0.90 | 0.21 |

**Ultimate and proximate analyses**. Table 1 displays the ultimate and proximate analyses of biomass and extracts obtained at 250-350 °C. Compared to raw biomasses, Soluble and Deposit contained obviously higher carbon content and lower oxygen content respectively, especially at high temperature. Furthermore, the ash content of Soluble and Deposit were extremely lower than those of RS and SD, accordingly. This suggests that the proposed method is very effective to deoxygenate and upgrade biomass wastes.

**FTIR analysis:** The chemical structures of raw biomasses, Solubles and Deposits were determined by FTIR analysis, as shown in Fig. 2. The abbreviation "RS-250-S" denotes Soluble produced from RS at 250 °C, and by this analogy. Compared to raw biomasses, the peaks attributed to oxygen-containing groups (such as -OH stretching at 3406 $cm^{-1}$ and C-O stretching at 1030 $cm^{-1}$) dramatically weakened. It can be observed that the spectra of Solubles and Deposits were similar respectively at different temperatures, indicating their similar chemical structures. However, the intensities of oxygen groups (especially for -OH) in Solubles and Deposits decreased with increasing temperature, which is consistent with the ultimate analysis (Table 1). The oxygen groups, including hydroxyl, in coal are deleterious for coke-making since they can deplete the donor hydrogen and leading to the formation of cross-linked chars [11]. Hence, the lower oxygen groups content in Solubles and Deposits is beneficial for its use in coal blends.

**Thermoplastic properties and coke quality evaluation.** The quality of produced coke depends greatly on the thermoplastic properties of coal. In this study, the influence of the extracts addition on thermoplastic properties of coal blends was measured by Gieseler test. Table 2 shows the thermoplastic parameters of coal GC and its blends with 2 wt% Solubles or Deposits addition. The plastic range of raw coal GC was only 74 °C. For Solubles addition, the softening temperature of coal blends decreased slightly, while the resolidification temperature of blends increased significantly. These leads to the enhancement of plastic range, especially for Soluble prepared at elevated temperature. The plastic range of blends with incorporation of Solubles prepared from both RS and SD increased with the temperature rising, which may be due to the lower oxygen content in Solubles at higher temperature [1]. In contrast, the softening temperature of coal blends with Deposits addition was higher than those of raw coal GC and Solubles addition. This may be because that Deposits contain relatively more heavy components than Solubles [8]. Still, the resolidification temperature and plastic range also increased like Solubles addition. Generally, the extracts (both Solubles and Deposits) obtained from SD behaved better in the fluidity development of coal blends than those from SD. The difference may be due to their different chemical structure and decomposition behavior, which will bring about their different interaction with the base coal (GC).

**Fig.2** FTIR spectra of raw biomasses and extraction products

**Table 2** Thermoplastic parameters of coal GC and its blends with 2 wt% Solubles or Deposits addition

| Blend | Ts(º) | Tf(ºC) | Tr(ºC) | Pr (ºC) | MF(ddpm) |
|---|---|---|---|---|---|
| Coal GC | 402 | 439 | 476 | 74 | 1159.5 |
| RS-250-S | 399 | 443 | 475 | 76 | 1201.8 |
| RS-300-S | 403 | 444 | 482 | 79 | 739.4 |
| RS-350-S | 398 | 436 | 481 | 83 | 768.8 |
| SD-250-S | 403 | 446 | 479 | 76 | 284.6 |
| SD-300-S | 399 | 439 | 480 | 81 | 970.6 |
| SD-350-S | 400 | 445 | 483 | 83 | 1084.6 |
| RS-250-D | 403 | 442 | 485 | 82 | 899.1 |
| RS-300-D | 404 | 445 | 480 | 76 | 453.3 |
| RS-350-D | 407 | 443 | 482 | 75 | 539.1 |
| SD-250-D | 402 | 444 | 481 | 79 | 800.2 |
| SD-300-D | 403 | 445 | 483 | 80 | 897.3 |
| SD-350-D | 400 | 444 | 483 | 83 | 885.6 |

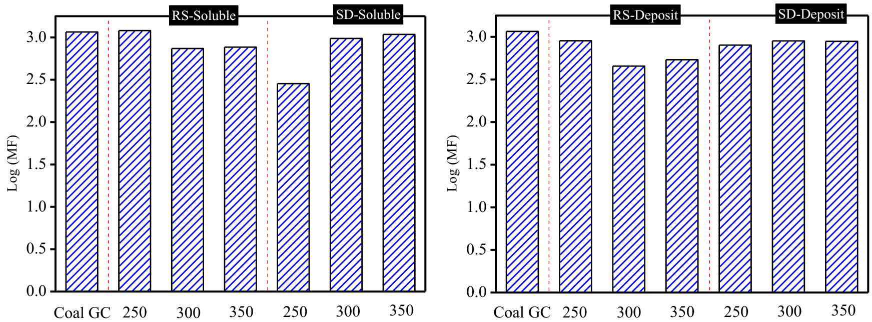


**Fig.3** Variation of log (maximum fluidity) of the blends with addition of Solubles and Deposits

The variation of log (maximum fluidity) of the blends with addition of Solubles and Deposits is illustrated in Fig.3. The MOF model establish the optimum window for log(maximum fludility) between 2 and 3 for coking coals to yield a strong coke [12, 13]. It can be observed that after these extracts addition the log (maximum fluidity) nearly all falled within the optimal window and maintained the fluidity the coal blends. So It is expected that the coal blends with Soluble and Deposit addition can produce coke with good strength.

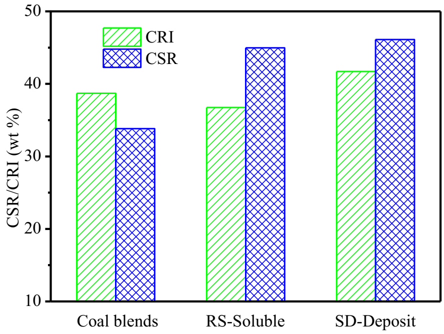


**Fig.4** The quality of coke produced from coal blends and coal blends with Solubles and Deposits addition

Coke reactivity index (CRI) and coke strength after reaction (CSR) are known to be the two most important parameters to assess the coke's quality and its behavior in the blast furnace. Generally, a coke with good quality is supposed to possess a lower CRI and a higher CSR. In this research work, the quality of coke was measured by crucible cooking. The RS-350-S and SD-350-D were chosen as the representatives for Solubles and Deposits (1 wt% addition) respectively, due to their comparatively high yields and good performance in thermoplasticity, and the results are presented in Fig.4. The raw coal blends was an industrial coal blends provided by an iron and steel group. When RS-350-S was added, compared to raw coal blends, the CSR of the coke obtained increased significantly from 33.8% to 45.0% and the CRI decreased. For SD-350-D addition, the CSR of the coke obtained also increased obviously by 12.3% although the CRI was slightly higher than those of raw coal blends. In general, it can be concluded that the addition of Solubles and Deposits from biomass could enhance the coke quality, thus opening the coking coal resources.

## 4. Conclusion

The yield of Soluble increased with temperature increasing for both rice straw and sawdust, while those of Deposit reached a maximum at 300 $^{o}$C. The sum of yields of Soluble and Deposit was rather high at elevated temperature. The Soluble and Deposit addition into the coke-making coal blends obviously increased the resolidification temperature and plastic range, thus improving the thermoplastic properties of the coal blends, especially for Soluble and Deposit obtained at higher temperature. Compared to raw biomasses, Soluble and Deposit contained significantly higher carbon content, lower oxygen groups (such as –OH groups) and extremely lower ash content in comparison with raw biomass, which should be responsible for its good performance in thermoplasticity modification. The addition of Deposits and Solulbles also markedly enhanced the quality of coke with increased coke strength after reaction (CSR) and reduced coke reactivity index (CRI). Therefore, it is highly feasible to utilize Soluble and Deposit prepared from degradative solvent extraction of biomass as additives to coking blends.

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# Study of Lignite Upgrading by Using a Pulsing Air Riser

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**Abstract:** The lignite cannot be separated by traditional net coal preparation techniques for its easy sliming and difficulty of products dewatering subsequently. A dry process using the cylindrical pulsing air riser is tried to decrease the ash content, and then upgrade the lignite to effective utilization. The lignite sample was collected from a coal mine in Inner Mongolia, China. The proximate analysis of the lignite was tested, and the separation experiment of the -6+3mm size fraction lignite was conducted by the designed pulsing air riser system. The results show that the ash content of feed is reduced by 12%−15% and the separation efficiency is up to 43.11% under the conditions that air flow rate: 87m$^3$/h; pulse frequency: 40Hz; feed speed: 400g/s and it can be realized by using cylindrical pulsing air Classifier in relatively short period of time The results show that CPAR could be efficiently used for the beneficiation of -6+3mm lignite of Inner Mongolia,China.

**Keywords**: lignite; upgrade; pulsing air riser; dry coal separation; the air flow; the pulsing frequency; the feeding speed

## 1.Introduction

The lignite reserves which have has been explored in China are more than 200 billion, accounting for 12.7% of the explored coal resources [1,2]. Reserve of lignite is rich and most of China's lignite is distributed in the eastern area of Inner Mongolia. In the structure of Chinese energy, the coal is always the main body and the demand and supply of coal obviously increase with lignite. The supply of high rank coals, such as anthracite and bituminous coal, is gradually in short. Reserves of high rank coal are less gradually with the increase of exploitation, and energy problem is more serious. So the effective utilization of low rank coal of lignite is particularly important [3,4]. Lignite, with the features of low calorific value(about 13.0~27.2J/ kg), high volatile content (15%~40%), poor thermal stability, easy to spontaneous combustion, high moisture content and difficult storage and transportation, is not conductive to use [5]. 90% of China's lignite is now used for power generation. Lignite has the features of easy to weathering and mud, and the price is low. Lignite is used directly as the fuel of power plant, without any separation process to upgrade its quality. Therefore, problems regarding the combustion and use of lignite should be solved and conducted in a more efficient and environmentally friendly way [6]. The processing and utilization of lignite would give priority to power generation. Countries with rich lignite resources like Australian, Germany, United States, Russia and Indonesia etc. all have successively carried out the study of processing and utilization technology of lignite [7,8]. Technologies of processing and utilization of lignite mainly include the lignite fluidized bed technology of Japan's Mitsubishi heavy industries company [9],German tubular type dryer lignite technology[10], America "K fuel process" [11] and so on.

Considering the associated minerals in lignite have a negative effect on the efficient utilization, de-ashing processes should be introduced. However, there is few reports about dry separating technology for lignite. In this case, an innovative separation method called cylindrical pulsing air classifier (CPAR) is introduced to conduct this target [12-14]. The pulsing air used in this classifer is produced by a pulsing air generator. Separation experiments of the -6+3mm lignite, sampled from Inner Mongolia,China are conducted. Separation process of different particles, interaction between gas and particles and particle dynamics are studied intensively.

## 2. Separation Principles and Apparatus

 
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### 2.1 Methods and theories

When a particle moves through a rising vertical air stream, it would get gravity, air buoyancy, and an aerodynamic drag. Active pulsing air classifier can produce pulsing air by a pulsing air generator, where particles can be accelerated and decelerated by the accelerating and decelerating air current. The dynamic model for particle moving through the active pulsing airflows is deduced as Equa (1):

$$m_p \frac{dv_p}{d_t} = -m_p g + \frac{\rho}{2}\frac{\pi d^2}{4} \cdot (v_A - v_p)^2 \left[\frac{3.6*10^{-4}}{d(v_A - v_p)} + \frac{9}{2}\right] - \frac{1}{2} m_{df} \left[\frac{\partial v p}{\partial_t}\right] + \frac{1}{2} m_{df} \left[\frac{\partial_v A}{\partial_t}\right] \quad (1)$$

where $m_p$ is the particle mass (g), $v_A$ is the airflow velocity(m/s) , $v_p$ is the particle velocity(m/s), d is the particle diameter(mm), g is the acceleration of gravity(m/s²), $m_{df}$ is the quality of the displaced fluid particles(g).

It is clear that heavy particles with higher density and light particles with lower density show different movement paths,and high-density particles tend to sink. Conversely, the low-density particles tend to rise. Separation experiments of cylindrical pulsing air classifier technology for -6+3mm lignite were conducted in this work.

### 2.2 Experiment Apparatus

The structure of CPAR is illustrated in Fig.1. Air is induced into experimental system by the air blower 1 and stored provisionally in the buffer tank 2. control flow of gas pipeline through flow control valve 3, display the gas flow of the whole line with gas flow meter 4 in time; the frequency converter is controlled by pulse valve, when flowing through the pulse valve in the line, air also produce the corresponding airflow ripple effect, the pulsating air flow enter into separation cylinder 6 through the wind plate of the bottom of separation cylinder. The lignite entering into separation cylinder through the star feeder 7 are oscillated and layered under the action of the pulsating flow.

Based on the differences in density, heavy particles gradually move to the discharge outlet at the bottom of separation column. However, the light particles move to the opposite discharge outlet in the top of separation column. The results indicate that it can obtain a better performance than that of the conventional air classifier in lab scale. Light material and dust are further separated after they flow through the cyclone separator 8. The dust after separation discharges from the upper vent-pipe and enters into the bag-type dust remover. Concentration then enters into the bottom of cyclone and finally collected by collecting device.

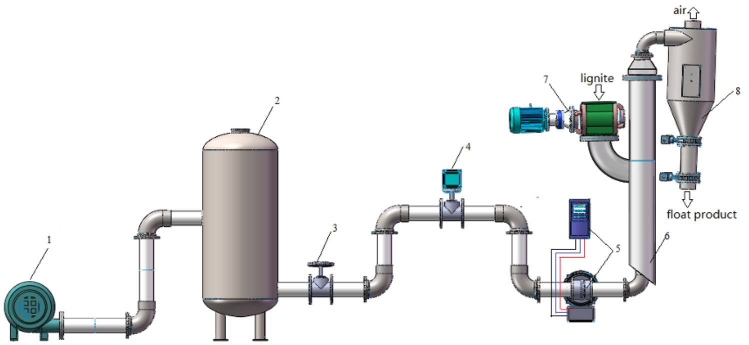


Fig1. Component of lab-scale Cylindrical Pulsing Air Classifier:1. air blower;2. buffer tank;3. control valve;4. flowmeter;5.pulsating air generator;6. separation cylinder;7. star feeder;8. cyclone separator.

### 3.Experimental material and process

Proximate analysis is conducted for lignite and results are shown in table 1. Note that $M_{ad}$, $V_{ad}$, $FC_{ad}$ and $A_{ad}$ represent for the moisture content, volatile content, fixed-carbon content and ash content of air-dried coal, respectively. $V_{daf}$ represents for volatile content of dry ash-free coal and $FC_{ad}$ is the fixed-carbon content of dry coal. It shows from table 1 that the $M_{ad}$ of coal sample is 22.32%, belonging to the high moisture coal; air dry ash content $A_{ad}$ is 26.08%, belonging to medium-ash coal; dry ash free basis volatile $V_{daf}$ is 49.04%, belonging to high volatile coal. Washability curves of the lignite is shown in Figure 2. where β is the cumulative float, λ is the characteristic ash,θ is the cumulative sink, δ is the relative density and ε is the near density. The result indicates that the contents of <1.4 g/cm$^3$ and >1.8 g/cm$^3$ particles in feed are 33.15% and 23.66%, respectively.

Table 1 Proximate analysis of lignite

| Coal sample | $M_{ad}$(%) | $V_{ad}$(%) | $F_{cad}$(%) | $A_{ad}$(%) | $V_{daf}$(%) | $F_{cd}$(%) |
|---|---|---|---|---|---|---|
| lignite | 22.32 | 26.70 | 27.74 | 26.08 | 49.04 | 34.06 |

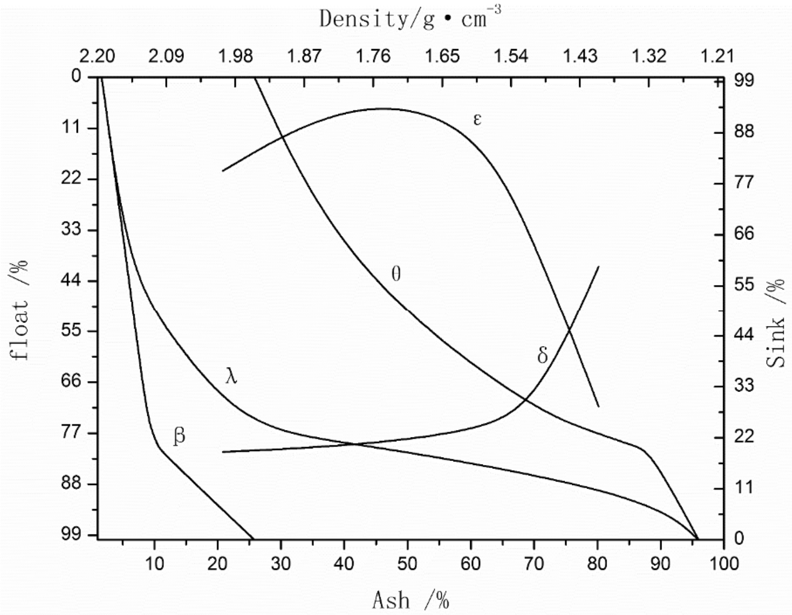


Fig.2. Washability curves of the lignite

The separation performance was assessed by separation efficiency E, and this parameter was calculated according to the following formula (2):

$$E = \frac{\gamma_j}{1-Ad_j} \cdot \frac{Ad_y - Ad_j}{Ad_y} \times 100\% \tag{2}$$

where E is the separation efficiency,$\gamma_j$is the yield of concentration, $Ad_y$is the ash content of feed coal, $Ad_j$is the ash content of concentration.

**3.1 The Effect of air flow rate**

Figure 3 shows the effect of air flow rate, which ranges from 60 to 100 $m^3$/h, for feed speed of 300g/s and pulse frequency of 25Hz. It indicates that increasing air flow rate from 60 to 80 $m^3$/h, the separation efficiency E would increase significantly from 11.87% to 43.11%. With a further air flow rate increasing from 80 to 100 $m^3$/h, the separation efficiency decreases. The best separation result are found at the air flow rate of 87$m^3$/h and ash content of light product and separation efficiency of the lignite separation test were 14.42% and 43.11%, respectively. On the contrary, Ash content of light product appears continuously increasing all the time.This is due to the more heavy density of particles are blown up with the increase of wind speed.

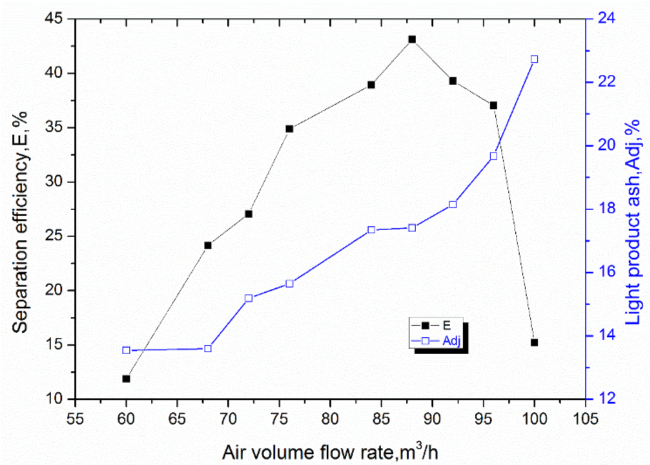


Fig.3. Effect of air volume flow rate on lignite separation.

### 3.2 The Effect of pulse frequency

The effect of pulse frequency was established. Test results are given in Figure 4. Separation efficiency E increases with the increasing of pulse frequency before the pulse frequency reaches to 40 Hz. After that, the fluctuation trend is gradually slow. A higher frequency provides a larger acceleration of the air, however, the pulse frequency should not be too high otherwise intensive collisions of particles are believed to reduce the efficiency. It is clear that when pulse frequency exceeds about 47.5 Hz, separation efficiency begins to decline. Similarly, the best pulse frequency for separation is found to be 40Hz in the study, and separation efficiency, E is 43.04%. It can be seen that ash content of concentration sees a decrease trend with the increase of pulse frequency. By increasing the pulse frequency to 40Hz, ash content of light product decreases inversely to 13.71%.

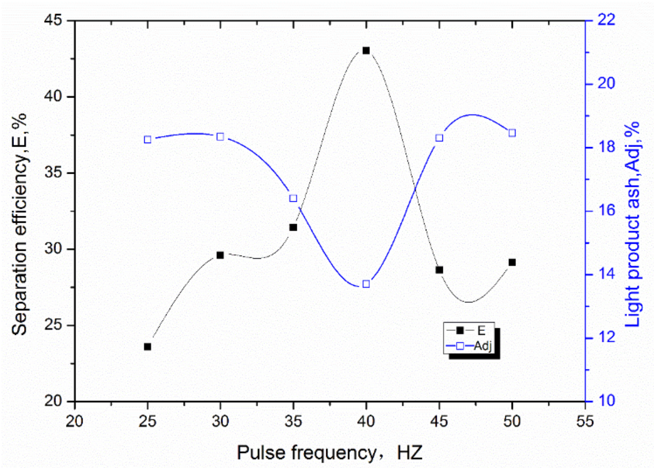


Fig.4. Effect of pulse frequency on lignite separation

**3.3 The Effect of feed speed**

Fig.5 shows the separation efficiency reduces after the rising trend, while ash content of light product increased first and then decreased. Feed speed varies between 300 and 700 g/s, and feed rate of 400g/s was established as being the best because of the high separation efficiency, E (42.27%) and low ash content (14.09%) of the light product obtained.

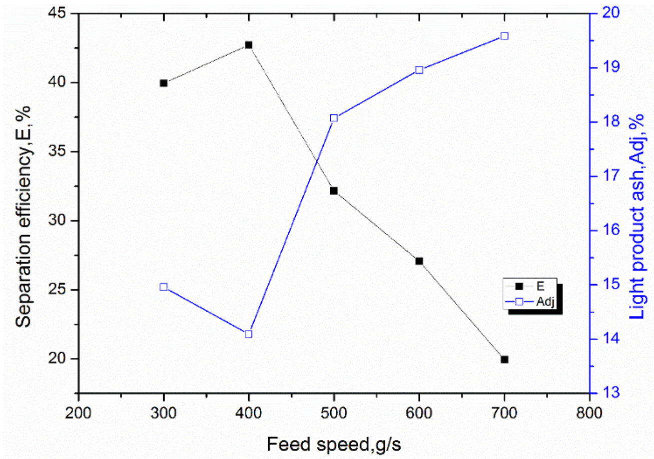


Fig.5. Effect of feed speed on lignite separation

**4. Conclusions**

(1) The air flow, pulsing frequency and feeding quantity are the key factors which influence the separation of -6+3mm lignite in the pulsation airflow. The ash content is reduced by 12%−15% and the

separation efficiency is up to 42.27%-43.11%. The best separation result is obtained at the experimental condition of air flow rate: 87$m^3$/h; pulse frequency: 40Hz; feed speed: 400g/s.

(3) It is further verified that the pulsing airflow force acting on the particles of low density (rise as the drag force, falls as resistance force) is more obvious than that on the particles of high density. Also, the acceleration of absolute value is bigger. When these two kinds of particle are driven by airflow to rise at the same time, the low density particles with bigger acceleration of absolute value rise faster and the high density particles with smaller acceleration of absolute value rise slower. When they fall, the results are on the contrary. So the two different particles can be separated according to the difference in density.

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# Low Relative Density Processing of Fine Coal

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**Abstract**

It is anticipated that much of South Africa's coal will in the future be mined from the Waterberg and Limpopo coal fields which are situated in the north of the country. The coal from these areas contains some bright coal and can be processed to produce a semi-soft coking coal which constitutes a valuable resource for the country's iron and steel industry. Production of a semi-soft coking coal requires the coal to be processed at very low relative densities in order to obtain low-ash coking coal products and it is furthermore required to crush the coal to about 20 mm top-size in order to liberate the coking coal. This results in a significant amount of fine coal reporting to the plant which also requires to be processed at low relative densities in order to yield coking coal. Processing of the coal at low relative density is very difficult to achieve, especially on the fine coal (-1+0.1 mm size fraction). This paper illustrates the importance of fine coal processing for the Waterberg and Limpopo coal fields and compares the currently available fine coal processing technologies.

**Key words:** Fine coal, low-density processing, coking coal, spirals, teetered-bed separators, reflux classifiers, fine coal dense medium cyclones.

## 1. Introduction

Run-of-mine coal typically contains some 10% to 15% of fine coal sized between 1 mm and 0.10 mm. In South Africa, beneficiation of this size fraction has largely been carried out with spirals since their introduction in the mid 1980's. Spirals are inexpensive to install and to operate and this has made their application in the South African coal industry widespread. Spirals replaced froth flotation as the fine coal beneficiation process of choice largely as a result of the low operating cost of spirals compared to flotation which is a very expensive process. Flotation is currently only employed at a few mines in South Africa to process ultra-fine coal (the minus 100 micron size fraction)

In recent years, production of coal from the Waterberg and Limpopo coal fields as well as from neighbouring Mozambique has increased. The coal from these areas differs from the coal traditionally mined from the South African Highveld and new processing challenges arose as a result. One of these is the need to process fine coal at low relative density.

The coals from the Waterberg, Limpopo and also from neighbouring Mozambique all contain coking coal which constitutes a potentially high value product. The production of coking coal is, however, complicated by the fact that the coal has to be processed at low relative densities to extract the coking fraction, which is required to have low ash content – typically 10%. The yield of coking coal is also low – usually between about 10 and 15%. The coals are furthermore friable which implies that a significant proportion of the coal reports to the plant as fine coal. Much of the coking coal is concentrated in the finer size fraction. In order to produce a coking coal with 10% ash content, the fine coal should ideally also be processed to yield a product with 10% ash or lower. This is, however, not easy in practise as processing of the fine coal is difficult and also because the near-dense content of the coal is high.

 
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When the fine coal cannot be processed to an ash content of 10% or lower, the coarse coal has to be processed to an ash content of below 10% in order to compensate for the fine coal and this can result in a significant yield reduction – often to the extent where the contribution of the fine coal becomes negative. This is illustrated in Table 1 which shows the combined yield of coarse and fine coal for varying fine coal product ash values. Note that in the table, the theoretical (washability) ash and yield values of the fine coal are used. The data in the table are also shown graphically in Figure 1.

**Table 1: Yield and quality for varying fine coal product ash content**

| Fine coal product ash | Combined yield % | Combined product ash content % |
|---|---|---|
| 4.0 | 15.36 | 10.00 |
| 5.0 | 17.26 | 10.00 |
| 6.0 | 17.76 | 10.00 |
| 8.0 | 18.07 | 10.00 |
| 10.0 | 16.74 | 10.00 |
| 12.0 | 12.04 | 11.05 |

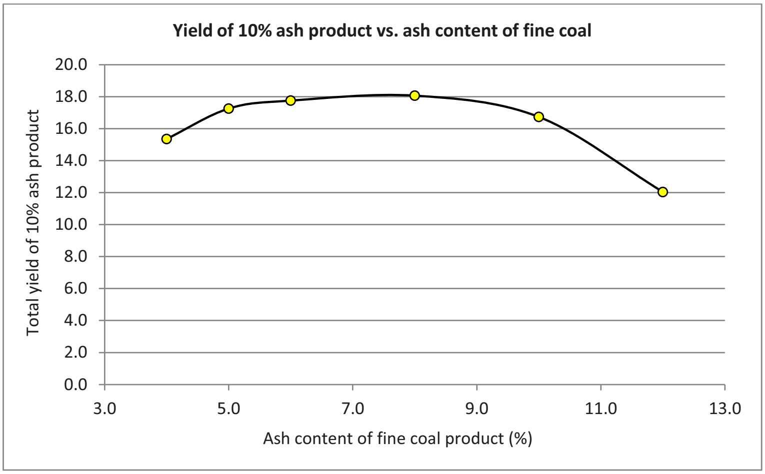


**Figure 1: Influence of fine coal product ash on overall coking coal yield**

It can be seen from Table 1 and Figure 1 that the highest coking coal yield is obtained when the fine coal product ash content is 8 %. When the fine coal product ash increases to 10%, the overall product yield is reduced by 1.33% and when the fine coal product ash increases to 12%, it is not possible to produce a combined coking coal product with a 10% ash. In addition to this, the overall yield is reduced. This illustrates the need to be able to process fine coal at a low relative density in order to produce a fine coal product with maximum 10% ash content and if possible, 8% ash.

## 2. Fine coal processing equipment options

### 2.1 Spirals

Spirals have been used extensively in the South African coal industry since their introduction in the mid 1980's. The units are inexpensive to install and operate and for most of the applications in the existing coal processing plants, serve the purpose of upgrading raw fines to thermal coal quality fairly well. The main obstacle to employing spirals in the production of coking coal from the Waterberg and Limpopo coal is the high cut-point density of these units which is typically above 1.70. The cut-point density can be lowered somewhat by using two stages of spirals in series but even this is usually not enough to obtain fine coal product ash values as low as 10%. The separation efficiency of spirals is furthermore not very good and typical EPM values obtained on spirals are about 0.150 at best.

Mineral Technologies, an Australian company, recently announced the introduction of a low-density cut-point spiral, the LC3, to the market. An evaluation of the LC3 spiral was carried out by Coaltech, a collaborative research program, in co-operation with Anglo American Coal, which proved that the LC3 spiral can cut at low densities. EPM value obtained from the LC3 spiral is similar to that of conventional spirals.

Due to the high cut-point density and high EPM values, spirals are not able to process the fine coal from the Waterberg and Limpopo areas to ash content values of below 10% and in fact cannot achieve product ash values below about 15% on this coal.

### 2.2 Teetered bed separators (TBS) and Reflux classifiers (RC)

The application of teetered bed separators is relatively new in the coal industry and only three full-scale TBS units are in operation in South Africa at present. There are currently only one RC unit in operation in South Africa but two RC 2020 units have been in operation at a coal processing plant in Mozambique since early 2012.

The TBS and RC differ only in the fact that the RC is equipped with a set of lamella plates on the upper end to improve the separation. In both the TBS and the RC, the separation between coal and impurities is affected in the hindered settling zone where an upward flow of water causes the fine coal and sand to behave like a fluidised bed. This results in a pseudo-dense medium which enables a separation between lighter particles (coal) and denser particles (sand) to occur.

The RC units in Mozambique have been subjected to extensive testing and the results obtained confirmed that the RC is able to cut at very low relative densities. EPM values obtained are approximately 0.100. The TBS units in operation in South Africa have also been evaluated and proven to be able to cut at a low relative density. The performance of the TBS, measured in terms of Imperfection, is very similar to that of the RC.

Both the RC and the TBS is very sensitive to particle size and any large particles in the feed will report to the reject stream irrespective of the quality or density of the particles. This is mainly a result of the fact that TBS and RC separators normally receive feed in the size range 1 mm to 0.15 mm or even 2 mm to 0.15 mm. It is difficult to optimise the flow rate of the fluidisation water for such a wide size range. It would be more advantageous in practice to split the feed to TBS or RC units at approximately 0.5 mm and feed the plus 0.5 mm coal to one unit and the minus 0.5 mm coal to a separate unit. This will allow the fluidisation water flow in both units to be optimised for the specific feed size range and improve the separation efficiency. Another problem specific to the RC is blockages occurring between the lamella plates. Frequent cleaning of the plates is required which is disruptive to production.

In coal applications, it is unfortunate that any ultra-fine material (slimes) in the feed, reports to the TBS or RC overflow, together with the product. Unless the slime fraction is removed from the product during dewatering, it has the potential to lower the quality of the product. In mineral applications, for example in chromite or fine iron ore processing, the slimes report with the gangue material and the product, in this case the RC underflow, is very well de-slimed. This makes the RC or TBS very well suited to mineral processing applications.

### 2.3 Fine coal dense medium cyclones

Dense medium cyclones have been used to process fine coal as long ago as 1957 when the process was first employed in Belgium for this purpose. It has, however, not gained popularity for a number of reasons with the main issue being high magnetite consumption. Recent advances in magnetic separator technology have now made it possible to use dense medium cyclones to beneficiate fine coal whilst maintaining acceptable magnetite consumption levels. Coaltech demonstrated the viability of dense medium cyclones for fine coal processing by building and operating a 25 t/h pilot plant. Coaltoll Pty. Ltd., a private company, built two 50 tonne per hour fine coal dense medium plants. The first of these plants has now been in successful operation for two years and the second, more recent plant, for a few months.

With the permission of the owners of the plants, Coaltech conducted a series of efficiency tests on the plants which confirmed that the plants are able to cut at low relative density and also that the process is more efficient than spirals and TBS or RC units. Typical EPM values for the fine coal dense medium process varies between approximately 0.070 and 0.100. Dense medium processing of fine coal offers further advantages in that the process is not sensitive to the particle size of the feed coal and is easily controllable through accurate control of the circulating medium density.

### 2.4 Two processes in series

Due to the difficult washability characteristics of the fine coal from the Waterberg and Limpopo coal fields, it is not possible to produce a fine coal product with an ash content of lower than 10% with either TBS or RC units or with fine coal dense medium cyclones. It is, however, possible to lower the ash content of the product by combining processes in a serial fashion. Although spirals are not capable of producing a low ash product on their own, they can be gainfully employed to affect a primary, de-sanding operation on the fine coal. The product from the spirals can then be sent to a second-stage unit which could be a TBS, RC or dense medium cyclone for further beneficiation. In this manner, it is possible to obtain a product with ash content below 10%. This allows the coarser coal to be processed to an ash value above 10% which optimises the overall yield of coking coal as shown in Table 1 and Figure 1.

The predicted practical yield/ash relation for the different fine coal processing options when processing fine coal from the Limpopo coal field is shown graphically in Figure 2.

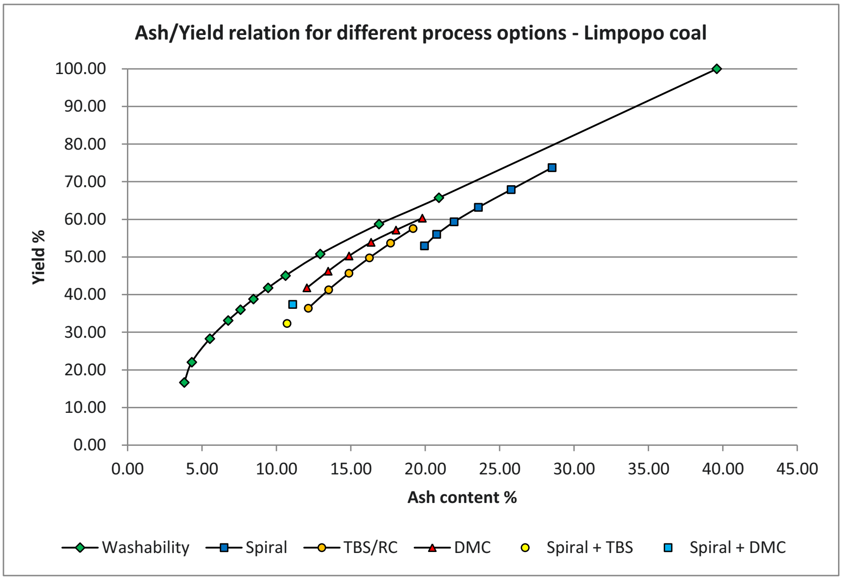


**Figure 2: Ash/yield relationships for different process options**

It can be seen that the lowest product ash will be obtained when using a spiral and TBS or RC combination. The product ash obtained from a spiral followed by a dense medium cyclone combination is slightly higher but the yield of product is also higher due to the higher processing efficiency of the dense medium cyclone.

## 3. Conclusion

Successful processing of Waterberg and Limpopo coals, when the aim is to produce a coking coal, requires a fine coal processing technique which can yield a product with a low ash content – ideally below 10% ash. It is not easy to process the fine coals from the Waterberg and Limpopo to low ash contents and the only technologies currently available capable of this are TBS or RC and dense medium cyclones. Spirals are not presently a contender although the new LC3 spiral could potentially change this. It still remains to be proven though.

Dense medium cyclones are more efficient than the water-only TBS and RC units but uses magnetite which increases the cost of operation. On the other hand, a dense medium cyclone plant is easier to operate than a RC or TBS unit and is not sensitive to particle size. Any large particles in the feed to a dense medium cyclone will be correctly beneficiated whilst the RC will reject such particles.

It therefore seems that processing of fine coal from the Waterberg and Limpopo to obtain low-ash products will require a two-stage approach with any one of the currently available technologies (spirals, TBS/RC or dense medium cyclones) as a primary, high-density stage. A TBS, RC or dense medium cyclones could be used in the second, low-density stage.

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# COAL UNDERGROUND LIQUEFACTION AS ALTERNATIVE TO NATURAL OIL PRODUCTION

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**Abstract**

With the oil crises as a turning point, the development of oil-alternative fuel, particularly coal utilization technology, came into the spotlight amid calls for a diversification of energy sources.

One of the potential technologies should be underground coal liquefaction. The liquefaction of coal, which had been positioned as the strongest oil-alternative energy contender because of huge coal reserves, was undergoing development in many countries.

Research in Germany, the United States and Japan involved pilot plants with the capacity to treat hundreds of tons of coal per day.

A new approach was suggested by authors - underground coal liquefaction. Up-to-date mining and chemical science and technology provide such opportunities. In particular, the process of coal liquefaction in the immediate vicinity of the mining face, allows not only obtaining artificial liquid fuel in underground conditions at a quite lower price in comparison with natural petroleum, but also rejecting the use of railway services for delivering the hard coal as a mine production to its destination place.

**Keywords**

Coal, Underground, Direct Liquefaction, Natural oil, Hydrogenation, Beneficiation, Conversion, Coal Processing, E-Trans Technology

## Introduction

Analysis of the cost of synthetic oil produced in the chemical industry, specializing in coal liquefaction, shows that about half of all costs, in the total cost of the production falls on coal extraction and its transportation to the site of liquefaction.

For this reason, it is required to look for ways to radically reduce the cost of underground mining first, combining with the elimination of produced (now at much lower cost) coal delivery to the earth's surface and the use of highly efficient and compact methods of its direct underground processing to synthetic oil close to the working face.

Such process of advanced mining and precise beneficiating with subsequent underground liquefaction of a cheaper coal produced in this manner is offered by us.



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## The object of study, the materials used and methodology

In contrast to silica sand or other oxygen-containing rock formations, coal is not a chemically inert material and its contact with oxygen irreversibly leads to gradual processes of endogenous oxidation, resulting in loss of not only calorific value, but also in decrease of synthetic oil yield during liquefaction of the solid fuel.

Therefore, the main object of the study was not only finding new, non-energy-intensive ways of underground coal mining, but also development of compact and precise methods for ultra-clean coal concentrates production directly in underground conditions, and identifying the correlation between the contact of freshly mined coal with oxygen in air and the effectiveness of its liquefaction.

Coal processing was carried out in the laboratory by the following methodology. Freshly mined from the working face regular raw coal (ash content 8.9%, volatile matter content of 45.2%, and Sulfur content of 0.8%) was immediately immersed in a jar with an aqueous solution of Lewis acids. The delivered to the laboratory coal sample, isolated from oxygen in the described manner was loaded into a steel mortar filled with a Lewis acid of density of 1.417 g/cm$^3$. Under such conditions the most clean coal particles in the raw material, which are not contaminated by gangue, remained afloat of such liquid while the rest sank to the bottom of the vessel. The material that sank was crushed down manually into smaller pieces using a steel pestle. During this operation, through the layer of such aqueous solution, a continuous bubbling of methane, which is insoluble in such environment, occurred. Also, under the impact of the crushing pestle, the pure combustible minerals were automatically released from their aggregates with gangue and floated to the surface of the liquid, thus avoiding additional irrational energy costs for their further destruction. Accordingly, at the bottom of the mortar less and less solid material remained with each time, which continuously reduced the efforts invested into the crushing.

When combustible minerals with density less than 1.417 g/cm$^3$ no longer floated on the liquid surface, the contains of mortar was poured into container with stirrer, from which the stirred-up suspension was passed through a hydrocyclone by means of a centrifugal pump, to make more intensive separation of the coal from the gangue. The emerging from the conical part of the hydrocyclone heavy product of the first step of crushing, which is a mixture of gangue minerals, was dewatered on a Nutsche filter. The light product suspension of the first stage of crushing collected into the bucket at the outlet of the cylindrical part of the hydrocyclone was heated to 100°C and fed into insulated ball mill equipped with methane collector and mechanical seal, flooded with hot (100 °C) Lewis acid solution. In such manner, in the second stage of wet grinding, even cleaner coal extraction from aggregates with waste rocks occurred, also accompanied by methane release. That is because with decreasing density of the liquid phase, only practically clean from mineral impurities pure coal will float to the surface of the heavy liquid medium in the thermally insulated mill.

The hot suspension of the grinded coal discharged from the thermally insulated ball mill, was separated on a high-speed (4000 rpm) laboratory centrifuge. The cake, pressed by centrifugal forces into the surface of the inner rotating rotor, was discharged from the centrifuge by rotating screw, cooled, and returned to the steel mortar for mixing with the next batch of the raw mined mass. The centrifuge centrate, which is a suspension of deeply purified from mineral impurities coal, was separated using a filtration centrifuge.

Without waiting for the wet cake pressed on the filtration centrifuge to cool down, 108 grams of the material was collected, which corresponded to a net of 100 grams of dry, demineralized solid carbonic material, and mixed with 200 ml of hot (160 °C) paste-forming agent, which is a mixture of (90: 10 by volume) tetralin with anthracene oil. In addition, during the mixing process almost complete removal of water vapor from the prepared paste occurred and was evacuated with a vacuum pump.

Then, to the paste obtained in the above manner and cooled to 80°C, 100 ml of isopropyl alcohol was added, and a dilution of the mixture was carried out in a high-speed mixer.

The diluted with isopropyl alcohol paste was then sent to a centrifugal disperser in which additional grinding of the solid phase of the system was carried out until a colloidal particle size. After this, the content of the disperser was reloaded into a steel autoclave, placed in a muffle furnace with a tangential inlet of compressed hydrogen (for a more efficient mixing), carrying out hydrogenation of coal in the medium. For this purpose, the mixture in the autoclave was gradually heated from 80°C and initial hydrogen pressure of 2 MPa to the temperature of 405°C and 11 MPa hydrogen pressure.

After 2 hours of hydrogenation, the autoclave was removed from the muffle furnace, cooled, unsealed, and the formed synthetic oil was separated from solid particles using the filter centrifuge. The extracted solid precipitate from such liquid was washed from the residues of synthetic oil with petroleum ether and dried with hot air, then sufficiently washed with warm distilled water, dried, and weighed. The dry residue weighed 2.74 grams. This material was directed to mixing with the cake received from pressing of the heavy product of the first stage of grinding on the Nutsche filter. The obtained tailing mixture of solid minerals was washed with hot fresh water using the Nutsche filter and removed from the process as waste. The formed washing water was mixed with the discharge of washing of the solid residue separated from the synthetic oil and evaporated to adjust the density of the salt-water solution to its initial value of 1.417 g/cm$^3$.

## Results and discussion

Since the autoclave was loaded with paste prepared according to the procedure described above, containing 100 grams of pure feedstock, the degree of liquefaction of the material which entered to hydrogenation was 97.26%.

Similar experiments were carried out exactly with same procedure and for the same coal, but already delivered to its destination by a railway from the place of mining. There, the degree of liquefaction was equal to 82.48 - 83.56%, that is 13 - 15% lower than of coal freshly extracted in the mining face.

## Conclusions

The use of the proposed method in the coal industry as energy-saving and environmentally friendly, using underground, and not above ground production of synthetic liquid fuel from such carbonaceous raw materials, promotes significant improvement of technical and economic efficiency of processing of such fossil fuels, particularly characterized by high content of non-combustible mineral impurities in it.

At the same time the method provides a comprehensive utilization of the produced coal (due to deep extraction from the raw material and extraction of methane) and increase in the degree of its liquefaction (due both to the elimination of contact with oxygen in the air and addition of ultra-high purity coal concentrate, thoroughly soaked with catalyst of the hydrogenation process, to the hydrogenation). This significantly reduces primarily the specific consumption of raw solid fuel for each tonne of liquid product.

In addition, the harmful effects of the whole complex of underground mining and production of synthetic liquid fuels on the environment are greatly reduced, as all final tailings of coal beneficiation, as well as solid waste from the production of synthetic liquid fuels, will automatically remain in underground mined-out space.

Moreover, the rejection of lifting raw solid fuel to the earth's surface from the coal mines, but lifting of finished products in the form of synthetic oil (or even the primary products of its processing) obtained in the underground conditions, allows not only to annihilate such powerful consumers of electricity as the cable-skip hoist (power of modern mine hoist machines reach 15,000 kW) and coal preparation plants from mine facilities (total electric power of all kinds of basic technological equipment drives, installed on this kind of ground production facilities, might reach up to 10 000 kilowatt), but also completely abandon railway transport services.

In this case, for coal producers it is much more profitable to use coal pipeline for delivery of their product to the place of its destination, which is about three times cheaper than coal transportation by railway (not taking in account the mechanical losses of solid fuel due to the coal dust blowing by wind from the wagons).

In general, the main advantage of the proposed combined technological process is steady improvement of the technical and economic efficiency of underground coal liquefaction with increasing depth of coal mining. While the traditional approach to synthetic liquid fuels production, with above ground coal liquefaction, far away from the coal supplier and steady deepening of underground mining leads to irreversible increase in the final cost of the synthetic liquid fuel production, and sharp increase in harmful effects of this type of human industrial activity on the environment.

Thus, to the top political leaders of any leading coal-mining country of the world (China, USA, Canada, India, Australia, Indonesia, Russia, Ukraine, Poland, Kazakhstan, South Africa, Germany, Mongolia, Mozambique, the United Kingdom, Spain, Turkey, Brazil, Argentina etc...) a direct technical possibility of synthetic oil production at price at least twice lower (about 19 - 27 dollars per barrel) than barrel of natural oil currently purchased in the Middle East and the Persian Gulf countries is offered.

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# DIVERSIFICATION OF JSC "SHUBARKOL KOMIR" PRODUCTION WITH OBTAINING PRODUCTS OF COAL DEEP PROCESSING

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*The article bring some results of industrial-innovative development of JSC "Shubarkol komir", carried out at the expense of diversifying production and obtaining new products from coal: 1) production of special coke - reducing agent, which is used for full (in the production of siliceous ferroalloys and ferrosilicon manganese) and partial (production of high-carbon ferrochrome, phosphorus) replacement of metallurgic coconut nut (0-10 mm; 10-25 mm and 25-40 mm; 10-60 mm); 2) production of light coal tar, which is used as a boiler fuel and as a raw material for the chemical industry; 3) production of coal oil brand M2 for impregnate the wood; 4) production of electricity, the purposes – reduce of greenhouse gas emissions in the atmosphere and obtain additional profit, through the use of associated coke oven gas in gas reciprocating installations, which produce electricity.; 5) production of activated carbons from fine fraction of special coke on the experimental-industrial device of continuous with rotating reactor; 6) production of polyfunctional sorbents from weathered coal of Shubarkol deposit.*

SHUBARKOL KOMIR, COAL, DIVERSIFICATION, SPECIAL COKE, COAL TAR, COKE OVEN GAS, ELECTRICITY, ACTIVATED CARBON

**Introduction.**

JSC "Shubarkol Komir" – a company known not only in Kazakhstan, but, also in many countries of near and far abroad by extensive market supplies of unique quality coal. However, in these latter days the company is developing new repartitions, developing relationships with other industry structures, research institutions and foreign partners for projects of deep processing of coal [1].

JSC "Shubarkol Komir" in its activities is guided by the principles, laid down in the Government Program of forced industrial-innovative development of Kazakhstan, who proclaim the priorities for the development of diversification and creation of products with high added value. In order to implementation these principles, the company was adopted strategy of development until 2022, which optimally combines the direction the increasing coal production and diversification of production [2].

Therefore, JSC "Shubarkol komir" has put and realize goal – to develop the basic activity – production of high quality coal, diversify production and obtain new products from coal – special coke, activated carbon, polyfunctional sorbents, coal tar and products based on its processing, to introduce a new energy-saving technology – based on the of utilization of associated coke oven gas of producing electrical energy.

 
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**The main part.**

**Production of special coke.** On the industrial site of JSC "Shubarkol komir" in 2006 introduced in commissioned a factory for the production of special coke with 6 rectangular, vertical single-chamber furnaces type «SJ» by technology of low-temperature dry distillation, total production capacity of 210 thousand tons of gross product [3].

These furnaces are characterized by high thermal efficiency, high productivity, low temperature on the furnace mouth, high yield of tar, convenient operation at low the value of investments.

Special coke production is unique for Kazakhstan and has no equals.

Produced special coke in comparison with similar products (carbonaceous reducing agents used in metallurgy) [4-5]:

- on technical composition most similar to the level of Russian coke (Altai Coke) and Chinese coke from gas-coal with of batteries JV "Asmare – koksohim, LTD", differing from them less ash content at practically the same level of volatile substances (4-6% vs 2-3 % in coke). The sulfur content (up to 0.3%) and phosphorus (0.02%), not higher than Chinese coke, and the level of fixed carbon not less than 84%, that is close to the level of this parameter for the coke;
- has enough high characteristics in terms of reactivity and the specific electrical resistance, standing in line with already in use types of carbonaceous reducing agents;
- in terms of structural strength (85.8%) coked material occupies an intermediate position between the coke production of the Russian and Chinese.

By its upon indications, special coke used:

- for full (in the production of siliceous ferroalloys and ferrosilicon manganese) and partial (production of high-carbon ferrochrome, phosphorus) replacement of metallurgic coconut nut;
- for the production of calcium carbide;
- for the production of non-ferrous metal as a carbonaceous reducing agent;
- agglomeration of ore;
- the production of briquettes.

In the coke ovens, coal successively going through the stages of drying, pyrolysis at a temperature of 700-750°C and cooling of water vapor, coming from bath quenching. Heating the coal to a pyrolysis temperature, is carried out by the oxidation reaction products cooled and purified coke oven gas and air, fed to the furnace tuyere by ventilators combustible gas and air.

The escaping coke oven gas with temperature 90-150°C, removed from the ovens and comes to the purification and cooling. Purification and cooling of coke oven gas is carried out in several steps: on the furnace mouth, the towers of direct and indirect cooling, in electrostatic filters.

Established as a result pyrolysis of coal tar, accumulates in pools of hot and cold water. In pools the tar separated into two fractions: the light and heavy. Light resin has a density of 940-1000 kg/$m^3$, heavy - 1040-1070 kg/$m^3$ [6]. Excess water, produced as a result the process of semi-coking coal and containing phenol and ammonia, are directed to the installation of utilization (incineration) of contaminated water.

**Production of electricity.** The purposes of the project – reduce of greenhouse gas emissions in the atmosphere and obtain additional profit, through the use of associated coke oven gas in gas reciprocating installations (GRI), which produce electricity. The project is without precedent in the CIS countries and included in the list of regional projects of the Industrialization Map of Kazakhstan.

The volume of coke oven gas, which has so far burned in the atmosphere, amounted to 21 thousand $m^3$ per hour, which led to additional environmental payments in the budget for emissions. At the same time, coke oven gas is a good, though a low-calorie fuel, burning that can receive electricity.

Electricity production is planned in three phases (Table 1). Since in 2012 JSC "Shubarkol komir" implemented primarily project "The organization production of energy and technological complex on the utilize coke oven gas of installed electrical capacity of 2.0 MW". The novelty of the project lies in the fact that to date, low-calorie and highly contaminated coke oven gas is not used in gas-of reciprocating power installations on the territory of CIS countries. Therefore, the specialists of JSC "Shubarkol komir" carried out extensive work to adjust the operating conditions and improvement of TPP (thermal power plant) equipment in accordance with the existing conditions of the coke oven gas.

Table 1 - Characteristics of the stages of the project for the utilization of coke oven gas

| Indicator name | Unit of measure | Phase 1 | Phase 2 | Phase 3 |
|---|---|---|---|---|
| The installed electric power | MW | 2.0 | 5.5 | 40 |
| Power generation | Million kW-h/year | 6.8 | 30.8 | 238 |
| Number of generating plants | pcs. | 4 GRI with 0.5 MW | 11 GRI with 0.5 MW | 5 GRI with 8 MW |
| Deadline for completion | months and year | December 2011 | December 2017 | December 2022 |
| Number of employees | Person | 10 | 16 | 120 |
| Number of Utilized gas | thous. $M^3$/h | 4.0 | 11.0 | 60 |
| Reducing greenhouse gases in the atmosphere | thous. tonnes / year | 13 | 50 | 450 |

First of all TPP coke oven gas is represented by four modular installation, consisting of a gas-a reciprocating engine, the generator installed capacity of 500 kW, equipment of filing and cleaning of coke oven gas, equipment of power transmission and control installations. The station consistently generates 500-600 million kW-hours of electricity per month, providing fully internal needs production plant of the special coke.

The second phase of coke oven gas TPP with installed capacity of 5.5 MW will cover the internal needs of JSC "Shubarkol komir" Designing the second phase of the station is planned from 2016. In the future, it envisages the construction of the third stage with the issuance of electric power stations on the domestic market of Kazakhstan.

**The implemented research and development.**

**Obtaining of activated carbons based on the special coke.** To date in Kazakhstan, activated carbons has not produced, but imported from Russia and China and other foreign countries. JSC "Shubarkol komir" decided to establish domestic production of activated carbon from fine fraction of special coke.

The main properties of the obtained of activated carbons and primarily porous structure, determined by the type of the original carbonaceous raw materials (special coke) and the method of processing. As part of the research work was obtained party of activated carbons on pilot plant continuous operation with rotating reactor [7].

The end result of this project are activated charcoal the crushed stone (Table 2) (analogue coal of brands AG) and activated carbon a finely dispersed (Table 3) (analogue coal of brands UAF). The product is characterized by a highly developed system of pore with a large inner surface of 570 $m^2$/g. The porous surface determines the sorption activity of coal, i.e., absorption ability of any substance from solutions and gases.

Table 2 - Physical and chemical parameters of activated carbon crushed stone

| № | Indicator name | Results |
|---|---|---|
| 1 | Outward appearance | Granules is black color |
| 2 | Moisture content, % | 2.24 |
| 3 | Adsorption activity on iodine,% | 40.47 |
| 4 | The total pore volume of water, $cm^3/g$ | 0.59 |
| 5 | Ash content,% | 6.24 |
| 6 | Bulk density $g/dm^3$ | 490 |
| 7 | Fractional composition, weight content of residue on the sieve with the web:<br>№36,%<br>№10,%<br>On the pallet,% | <br><br>11.6<br>81.6<br>6.8 |
| 8 | Mechanical (structural) strength,% | 87.1 |
| 9 | Granule strength to abrasion,% | 79 |
| 10 | The pH of water extract | 7.1 |
| 11 | Adsorption activity on the indicator methylene blue, mg/g | 114 |
| 12 | Adsorption activity on molasses,% | 68 |
| 13 | Adsorption activity on acetic acid, units (mg/g) | 46 |
| 14 | Adsorption activity on active chlorine, mg/g | 15.6 |
| 15 | Static adsorption capacity of the chlorine, mg/g | 45 |
| 16 | Specific surface, $m^2/g$ | 361.377 |
| 17 | Dynamic activity on benzol, min | 39 |
| 18 | The adsorption capacity for copper, mg/g | 46.2 |
| 19 | The adsorption capacity of iron, mg/g | 59.3 |
| 20 | Weight content of volatile substances,% | 0.1 |

Table 3 - Physical and chemical parameters of activated carbon finely dispersed stone

| № | Indicator name | Results |
|---|---|---|
| 1 | Outward appearance | Powder is black color |
| 2 | Moisture content, % | 3.16 |
| 3 | Adsorption activity on iodine,% | 53.7 |
| 4 | Ash content,% | 6.24 |
| 5 | Bulk density $g/dm^3$ | 668.5 |
| 6 | Fractional composition, weight content of residue on the sieve with the web:<br>+ 0.4 mm,%<br>+ 0.1 mm,%<br>On the pallet,% | <br><br>2.39<br>33.27<br>64.34 |
| 7 | The pH of water extract | 7.1 |
| 8 | Adsorption activity on the indicator methylene blue, mg/g | 114 |
| 9 | Adsorption activity on molasses,% | 68 |
| 10 | Static adsorption capacity of the chlorine, mg/g | 367 |
| 11 | Specific surface, $m^2/g$, granules | 570.635 |
| 12 | Adsorption activity on acetic acid, units (mg/g) | 59.2 |
| 13 | Adsorption activity on active chlorine, mg/g | 71.4 |
| 14 | Weight content of volatile substances,% | 0.1 |

Application: desalination of water, water purification, enrichment, air-gas cleaning, catalytic systems [8-9].

Obtained activated carbons were successfully tested in the production of car chemicals to remove the smell of MPCB (mixture of propane and commercial butane) in the filtration plant, work is underway to test of activated carbons in the process of purification of household and industrial waste water, as well as in the process of flotation of minerals and to extract the gold. Currently, work is underway to develop a feasibility study for the creation of pilot production of activated carbons with capacity of 500 tons per year, using as a fuel element of coke oven gas. This production will be economically profitable, environmentally friendly and technologically simple.

**Obtaining polyfunctional sorbents from weathered coal of Shubarkol deposit.** Weathered coal of Shubarkol deposits contain on conversion to organic matter up to 91% of humic acids, which are the products of ancient organics recycling. Humic acids in their turn are divided into fulvic acids and humic acids. The mechanism of their effects in the biosphere is different.

So according to Holin Y.V. [10] migratory ability of elements in natural waters as a result of the complexation with fulvic acids increases sharply, which allows the use of this property for the plant nutrition trace elements. As contrasted fulvic acids, humic acid is soluble only in highly alkaline solutions. In natural systems, such conditions are not occur, and humic acids behave as complexing sorbents, hold and concentrating heavy metal elements in soils, suspensions of water, sediment, carbon rocks.

By modifying the humates for transporting microelements (Fe, Zn. Cu, Mo, Mn, Co, B, Si), were synthesized humic preparations, known as "Poligum". Their composition and properties are given in the following table:

Table 4 - The composition of the fertilizer series "Poligum" (a liquid form of the polymer humic fertilizer with microelements)

<table>
<tr><th rowspan="3">№</th><th rowspan="3">Indicator name</th><th colspan="7">Composition of the preparation</th></tr>
<tr><th colspan="2">sodium humate</th><th colspan="3">microelements</th><th colspan="2">polymer</th></tr>
<tr><th>g</th><th>%</th><th>g, overall</th><th>%</th><th>Additional</th><th>g</th><th>%</th></tr>
<tr><td>1</td><td>Poligum - 21</td><td>250</td><td>87.0</td><td>37.24</td><td>13.0</td><td>-</td><td></td><td></td></tr>
<tr><td>2</td><td>Poligum - 22</td><td>250</td><td>86.0</td><td>40.28</td><td>13.9</td><td>silicon</td><td></td><td></td></tr>
<tr><td>3*</td><td>Poligum - 23</td><td>250</td><td>74.1</td><td>37.24</td><td>11.1</td><td></td><td>50</td><td>14.8</td></tr>
<tr><td>4*</td><td>Poligum - 24</td><td>200 SHNa,<br>50 HNa</td><td>87.0</td><td>37.24</td><td>13.0</td><td>-</td><td></td><td></td></tr>
<tr><td colspan="9">Note: The content of microelements,%: Fe - 3.6-3.8; Zn - 2,1-2,2; Cu - 2,1-2,2; Mo - 2,1-2,2; Mn -1,7-1,8; B - 0,5-0,6; Co - 0,21-0,22; Si - 1,05<br>* Exposure by ultrasound, SHNa – sulfate humate sodium</td></tr>
</table>

For the development the methods and the use of doses of humic preparations, and to determine their impact on the chemical properties of soil and productivity wheat on southern chernozems of Northern Kazakhstan LLP "Scientific-Production Center of Grain Farming of the Barayev A.I." (Shortandy village, Akmola region) were laid small plot field and pot experiment.

The results showed a clear tendency improve germination of grain treated with humic substances.

We have also are conducted research to develop sorbents for the purification of industrial waste water and contaminated soils with heavy metals. In this case, we used the complexing properties of humic acids for the production of sorbents based on them, adsorbed from solution heavy metal ions.

Thus humic preparations modified mineral and organic compounds. Currently, these sorbents are tested for industrial wastewater and contaminated soils of the metallurgical enterprises of Kazakhstan.

These tests allow us to speak about getting a positive effect their sorption characteristics.

**Conclusion.**

Thus, JSC "Shubarkol" has put and implement goal - to develop the basic activity - production of high quality coal, to diversify production and to obtain new products from coal - special coke, activated carbon, polyfunctional sorbents, coal tar and products of its processing, to introduce a new energy-saving technology - based on the of utilization of associated coke oven gas producing electrical energy.

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# COAL GASIFICATION AND GAS OBTAINING FOR SYNTHESIS OF MOTOR FUELS

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Abstract. For the first time research of steam and air gasification of coal of Kazakhstani deposits (Shubarkol, Zhalyn, Maikube, Saryadyr), and Mongolia (Nalaikha, Bakaguur) was conducted in this paper. Technical and elemental composition of these coals was determined as well as output of produced gas. Dependence of concentration of basic combustible gas components on temperature in the range of 600-1000 $^0$C. Description of the layered gasification process and corresponding equations were given in three main areas of reactions. Also the method to increase concentration of carbon monoxide and hydrogen in composition of generating gas was shown and described by adding various catalytic additives. During experiments the series of tests were performed on steam and air gasification of coal of some Kazakhstani and Mongolia deposits using potassium carbonate or sodium carbonate as catalyst. The process of coal gasification was carried out at enlarged laboratory setup, its scheme and description are shown. The received results are of some scientific and practical interest for the introduction of coal gasification technology to similar deposits in coal processing industry for production of gas for further synthesis of motor fuel and other valuable chemical products.

**Key words:** coal, coal processing, layered coal gasification, steam and air blow, coal analysis, gas, combustible gas components.

Coal remains the main and the most reliable strategic fuel for the Republic of Kazakhstan, ensuring the development of electric power industry and coal processing industry of Kazakhstan. Coal use in industrial and other sectors of the economy increases annually and according to analysts ' forecasts it will grow up to 121.3 million tons by 2020.

Currently Kazakhstan occupies 8th place in the world in proven reserves of coal of all kinds, which are estimated at 150-160 billion tons (4% of the global volume), 62% of them are brown coal, and 38% are stone coal. We can consider this amount of coal reserves not only as raw material for coal power plants but also for use in thermochemical and catalytic processes for obtaining of a wide range of valuable chemical products [1,2].

 
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One of the promising methods of deep processing of coal through clean coal technology is gasification. The main advantage of the process of coal gasification is that it can remove pollutants from coal before the coal will be burned, thus preventing the emission of harmful substances into the atmospheric air [3,4].

Currently, coal gasification processes are multipurpose. The main products of coal gasification are:

1. The synthesis gas with subsequent conversion to valuable chemicals, including motor fuel;
2. The fuel gas as a substitute for natural gas;
3. The product gas as a fuel for power plants.

During coal gasification the processes take place in three main areas of reactions (equations 1-9) Reactor zones location diagram is shown in Figure 1.

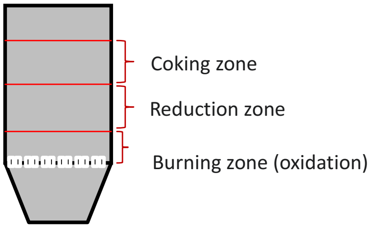


Figure 1 – Location of main zones of coal gasifying process

Reactions 1-3 occur in the combustion zone, reaction 4-9 – in the reduction zone.

1) $C + O_2 = CO_2 \Delta H = -404{,}7$ kJ/mole
2) $2C + O_2 = 2CO \Delta H = -246{,}4$ kJ/mole
3) $2CO + O_2 = 2CO_2 \Delta H = -567{,}3$ kJ/mole
4) $CO_2 + C = 2CO$ $\Delta H = 161$ kJ/mole
5) $C + 2H_2 = CH_4 \Delta H = -83{,}8$ kJ/mole
6) $C + H_2O = CO + H_2 \Delta H = 118{,}5$ kJ/mole
7) $C + 2H_2O = CO_2 + 2H_2 \Delta H = 16{,}3$ kJ/mole
8) $CO + 3H_2 = CH_4 + H_2O \Delta H = -206{,}7$kJ/mole
9) $CO + H_2O = CO_2 + H_2 \Delta H = -42{,}4$ kJ/mole

The coal gasifying process was observed based on data from the composition of gasifying products received in different temperature intervals. The analysis results of the generator gas are shown in Figure 1 [5-7].

Observations were started after achieving the temperature of 500°C in the gasifying zone. Components concentration of gas mixture has sufficiently high values, which is obviously connected to the development of thermal decomposition reactions on the surfaces of coal particles in the coking zone and as a result removal of volatile components: $H_2$, $CH_4$, CO and $CO_2$.

Reactions 1 and 3 prevail in the temperature range from 500 to 750 °C, which results in increase of volume ratio of $CO_2$ in the gasifying products and the reduction of CO concentration [6, 8].

During arising up through the coal layer, incandescent gases reach reduction zone. But how we can see from the gas composition of the components concentration (CO, $CH_4$, $H_2$) which should prevail in the reduction zone had low values. Possibly, it is due to potassium carbonate molecules penetrate into coal pores, thereby blocking an access of gas components to active carbon centers, which in its turn prevents the reaction processes 4-9.

The staged increase of volume ratio of carbon oxides, methane and hydrogen occurs at temperature rise. After achieving 900 $^0$C in the gasifying zone, the temperature rise stops till the end of the process, and it was kept in this range with some minor changes ±10 $^0$C[9-11].

Staff of "Institute of coal chemistry and technology" LLP conducted researches on fixed bed coal gasification in laboratory installation of periodic action. As the objects of study coals of different Kazakhstani deposits were selected as well as coals of Mongolia for comparison.

Technical and elemental analyzes were carried out on Eltra Thermostep coal thermogravimetric analyzer (Germany), in accordance with ASTM D7582-12 «Standard Test Methods for Proximate Analysis of Coal and Coke by Macro Thermogravimetric Analysis») and «EURO EA 3000» elemental analyzer, the results of which are shown in Table 1 [12].

Table 1 - Technical characteristics of gasified types of fuels

| № | Coal deposits | Coal composition, % | | | | | |
|---|---|---|---|---|---|---|---|
| | | Coal grade | $W^r$ | $A^r$ | $V^{daf}$ | $S^r$ | Combustion value, kcal / kg |
| 1 | Shubarkol | Д | 11,42 | 2,76 | 57,48 | 0,29 | 6127 |
| 2 | Maykube | Б | 9,50 | 13,00 | 49,99 | 0,55 | 5150 |
| 3 | Saryadyr «Nadezhniy» layer | Г | 2,94 | 46,47 | 27,68 | 0,27 | 3771 |
| 4 | Zhalyn (Kazakhstan) | Г | 7,60 | 5,62 | 47,70 | 0,77 | 5600 |
| 5 | Bagakuur (Mongolia) | Б | 8,10 | 18,63 | 40,00 | 0,90 | 3513 |
| 6 | Zhalyn (Kazakhstan) | Г | 7,60 | 5,62 | 47,70 | 0,77 | 5600 |

One way of in creasing carbonmonoxideandhydrogenconcentrationwithintheexhaustgasis the utilization of different catalytic additives. "IHUT" LLP staffmembershave performedaset of experiments using potassium carbonate as gasifying catalyst.

Coal gasifying process was performed in the laboratory unit, scheme of which is shown in Figure 2. Preliminary fractionated coal sized 5 - 10 mm, with the weight 1,5 kg, was separated into two parts. Some part of the coal was used for preliminary ignition and the remaining part was drenched with 10% solution of $K_2CO_3$ and put on the live coals in the gasificator. Fixing temperature in the area of gasification was performed with chromel-alumel thermocouples.

On the cover of gasificator there installed a heating component used for resin afterburning which is evolved in the process of coal thermal destruction. To reduce excess heat and sustain required temperature regime the gasificator is equipped with water jacket.

In the lower part of the gasificator there installed two pipes for the delivery of gasifying agent into the area of coal gasification. Overheated steam and air are used as gasifying agents. The steam is delivered from steam generator 2. The steam generator is equipped with heating element and manometer for measuring steam pressure. Temperature of steam generator heating is regulated with laboratory transformer 11. Steam supply is regulated with needle valve. Air goes from compressor 3 into tube air heater 9. Tube air heater is a brass tube, 40 cm long with the diameter 2 cm filled with ceramic material. To heat the air nichrome spiral is coiled on the tube the heating speed of which is regulated with laboratory transformer 12. Air supply is regulated with rheometer 10.

Generative producer gas received in the gasification process is supplied to separator 4, to separate moisture and residual resin. From the separator gas is transported to refrigerator 5. The quantity of generator gas is registered with gas meter 6. With compressor 7, gas mixture is pressed and delivered into gasholder 8.

On the gas line behind the gas meter there installed a three-way cock from which there was performed gas sampling for chromatographic analysis.

Blend composition of gas is determined with gas chromatograph «Crystalux-4000M», with the module 2 DTP. The method of analysis carrying out is described in [13]. Carrier gas is helium, consumption of carrier gas is 30 ml/min., columns temperature is 90 $^{0}$C, detector temperature is 210 $^{0}$C, evaporator temperature is 150 $^{0}$C, separation of components was performed on NaX and HayesepR columns.

Coalgasifyingprocesswasconducted during the following parameters:

Air consumption – 5 l/min,

Steam consumption - 0,52 kg/ kg,

Coal mass – 1,5kg,

Coal fractional composition – 5-10 mm,

10% potassium carbonate solution - 200 ml,

Process time – 90 min.

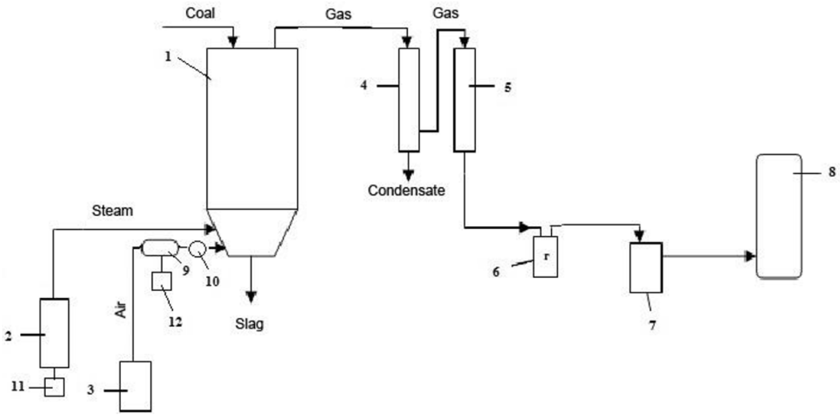


Figure 2 – Coal gasifyingplant process scheme

*1 - Gasifier; 2–steam generator; 3 – compressor; 4 – separator; 5 – cooling unit; 6 – gas meter; 7 – compressor; 8 – gas collector; 9 – air heater; 10 – rheometer; 11, 12 – laboratory transformer*

After fixed bed coal gasification obtained syngas contained the following total output of gases shown in Table 2.

Table 2 - Total output of combustible gas components ($CO$, $H_2$, $CH_4$) at temperature from 600 to 900 $^{0}$C

| № | Coal deposits | Total output of $CO$, $H_2$, $CH_4$ ( %) | | | |
|---|---|---|---|---|---|
| | | 600 $^{0}$C | 700 $^{0}$C | 800 $^{0}$C | 900 $^{0}$C |
| 1 | Shubarkol | 18,71 | 25,48 | 36,16 | 43,65 |
| 2 | Maykube | 10,85 | 17,44 | 22,43 | 34,55 |
| 3 | Saryadyr «Nadezhniy» layer | 6,70 | 10,40 | 18,08 | 23,65 |
| 4 | Zhalyn | 4,86 | 8,44 | 14,86 | 24,59 |
| 5 | Nalaykha | 13,19 | 17,97 | 23,82 | 27,18 |
| 6 | Bagakuur | 13,92 | 20,69 | 27,14 | 40,84 |

As can be seen from the presented experimental data, there is a significant increase in the concentration of CO, $H_2$, $CH_4$ for all types of coals, with increasing temperature from 600 to 900 $^0$C, with a gradual decrease at temperature above 900 $^0$C.

Figures 1-3 show the dependence of output of main combustible gas components (CO, $H_2$, $CH_4$) on temperature range of 600-1000 $^0$C, their total yields are represented in Table 2.

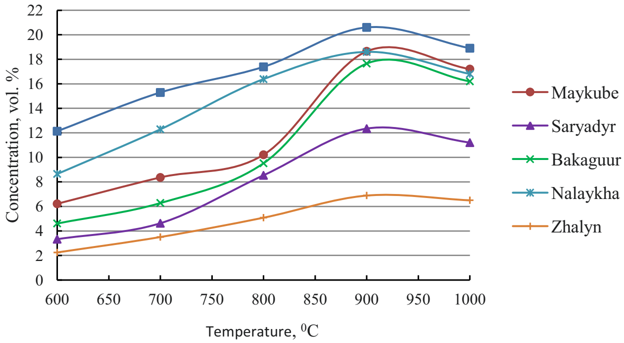


Fig. 3 – Dependence of CO output on temperature

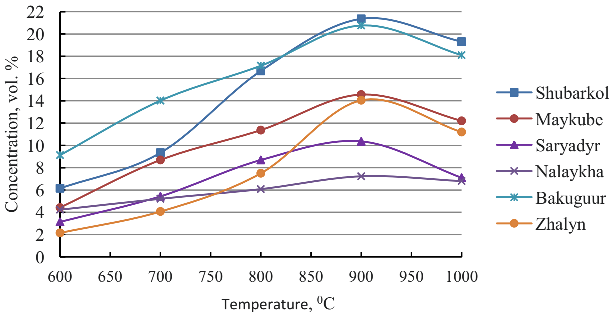


Fig. 4 – Dependence of $H_2$ output on temperature

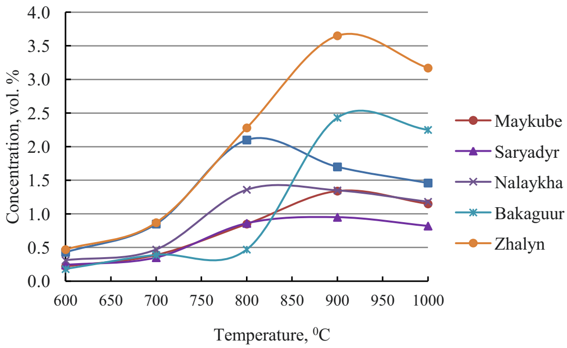


Fig. 5 – Dependence of $CH_4$ output on temperature

The researched coals are suitable for coal gasification process, but the most attractive are coals of Shubarkol deposit, Bagakuur (Mongolia) and Maikube, with total yield of combustible gas components more than 30% (Table 2). However, there is a strong separation of resinous substances of Shubarkol and Nalayha samples (which are long-flame) and small separation of resinous substances of Maikuben and Zhalyn coal samples, which will require further purification of the product gas for its further processing into the desired products.

Thus, a study of steam gasification of Kazakhstani and Mongolia coals was conducted for the first time with determination of optimal parameters of the process. The results are of particular practical interest for implementation of coal gasification technologies in coal processing of similar deposits.

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# PRODUCTION OF GASOLINE FROM COAL BASED ON THE STF PROCESS

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**Abstract**

The growing demand for motor fuels, particularly in a number of countries which own coal but not crude oil, is promoting a rising interest in converting coal into synthetic liquid fuels. The formation of valuable liquid products from coal may be an option for using coal deposits located in remote areas, since the long-distance transport of liquids with high energy content is more economical.

The coal liquefaction process includes the production of synthesis gas from coal and the subsequent conversion of synthesis gas into liquid fuels (gasoline, diesel, jet fuel). Several well-established technologies are known both for coal gasification and fuel synthesis.

At the beginning of this decade, a novel process was developed for the production of high-octane gasoline from synthesis gas (“Syngas To Fuel” – STF®). A STF demonstration plant is operated by Chemieanlagenbau Chemnitz GmbH and Technische Universität Bergakademie Freiberg. It includes the conversion of syngas into methanol and the subsequent conversion of methanol into gasoline in isothermal reactors, which are advantageous in terms of selectivity compared to adiabatic systems. After the STF plant was started up in Freiberg in June 2010, the whole process was systematically optimized and successfully operated in a series of long-term test campaigns, using synthesis gas generated by the autothermal reforming of natural gas. The STF process allows gasoline to be produced from different feedstock including coal.

Based on the test results, technical and economical parameters were calculated for the overall coal-to-liquid conversion. For flow sheet simulations, the commercial software Aspen Plus was used. Gasoline output and energy efficiency were the main optimization parameters. Investment costs and operation expenses were assessed for different scale-up scenarios. Using the economic characteristics and the calculated gasoline output, the production costs were estimated and compared with current world market prices. The influence of the variable boundary conditions on the final product price was considered in order to evaluate the potential for cost optimization. The determined technical and economic parameters for the CtL process under consideration, based on STF technology, were benchmarked against commercialized CtL processes. The study also included other technologies for the generation of valuable products based on different feedstocks.

**Keywords**

syngas to fuel; gasoline synthesis; isothermal reactor; high-octane gasoline; methanol; partial oxidation; autothermal reforming

 
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**Technology chain and synthesis gas production**

Technologies for the conversion of synthesis gas into liquid hydrocarbons may be divided into two main groups: the direct catalytic conversion of the synthesis gas into liquid hydrocarbons on the one hand and indirect catalytic conversion via methanol on the other. The characteristic shared by these technologies is that the capital expenditure for the large-scale production of gasoline or diesel, respectively, is very high. The products of Fischer-Tropsch synthesis or methanol-to-gasoline synthesis (MTG) have to be re-treated in downstream processes (hydrocracking, oligomerization, distillation) to achieve product qualities which meet the specifications. These downstream processes include reforming, isomerization or hydro-isomerization to remove undesirable components such as durene.

The novel “Syngas-To-Fuel” process (hereinafter the STF process) is managed without this usual expensive product re-treatment. This process was designed on the joint research project “Development of a new technology for the production of high-octane gasoline from synthesis gas” by the project partners Chemieanlagenbau Chemnitz GmbH (CAC) and TU Bergakademie Freiberg (TU BAF) with support from the Russian company SAPR-NEFTEKHIM and the Kazakh company Techno Trading. Implemented on pilot plant scale, it was successfully trialled at the test plant of TU BAF’s Institute of Energy Process Engineering and Chemical Engineering (IEC).

The synthesis gas for the gasoline synthesis plant can be supplied by the IEC's HP POX test plant, which can deliver synthesis gas on spec by means of autothermal catalytic reforming. The HP POX test plant is unique in terms of its operating conditions and operation modes: the reactor design allows for catalytic and non-catalytic natural gas reforming as well as the gasification of liquid feeds (oil). The feedstocks are converted by adding oxygen and water steam at gasification temperatures up to 1450 °C and at pressures up to 100 bar. The plant has a maximum thermal capacity of 5 MW and generates up to 1500 scm of raw gas per hour.

To deliver synthesis gas for gasoline production, the autothermal catalytic reforming of natural gas (ATR) was used exclusively. To achieve this, the reactor was operated at low capacity, because only 700 scm of synthesis gas per hour at 58 bar are consumed under the design conditions of the STF methanol synthesis reactor. The percentage of carbon contained in the natural gas in relation to the gasification agents ($O_2$, $H_2O$) determines the synthesis gas composition and the reaction temperature. In the catalyst layer which is located downstream of the flame zone, the reforming reactions reach thermodynamic equilibrium at temperatures between 800 and 1100 °C (measured at the reactor outlet before quenching). The raw gas leaving the reactor is quenched by water injection to prevent reverse reactions as the gas cools down. The quench water is recycled after re-cooling, with a minor amount being discharged. The cooled raw synthesis gas passes through a droplet separator and enters the gasoline synthesis plant.

**Gasoline synthesis**

The gasoline synthesis plant converts the synthesis gas produced in the HP POX plant via methanol into high-octane gasoline The methanol synthesis is designed for an input of up to 700 scm of synthesis gas per hour and the gasoline synthesis for an hourly capacity of 120 l of gasoline. The essential process features such as the isothermal reactors, the means of heat removal, the reaction conditions and the specially conditioned catalyst differ from the properties of competing processes such as the MTG process developed by ExxonMobil or the TIGAS process used by Haldor Topsoe. The chemical structure of the primary gasoline product complies with the EU and GOST standards after simple post-treatment by means of distillation. The effluent-free production enabled by integrating the post-treatment of methanolic water is worth emphasizing. The method by which the gasoline synthesis plant functions is represented below (Figure 1):

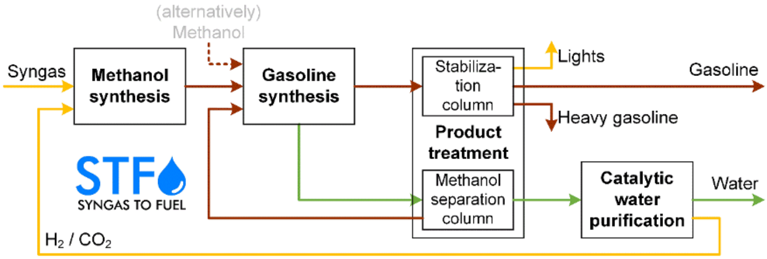


*Figure 1: Balance flow chart of the gasoline synthesis plant*

The HP POX synthesis gas has to be further cooled, and the condensed water has to be separated. Then the synthesis gas enters the isothermal methanol synthesis reactor (pressure 50–55 bar, temperature 230–250 °C) together with the recycling gas (non-converted synthesis gas components). The generated methanol-water mixture (approx. 90 % purity) has to be condensed and separated from the non-converted gas components, which are recompressed and recirculated. The methanol-water mixture passes a multi-stage preheating and an evaporation step and is fed into the gasoline synthesis reactor (pressure 4–6 bar, temperature 290–320 °C). For comparison, the MTG process[1] can be considered.

The STF process uses an isothermal fixed-bed reactor. The heat dissipates in a molten salt heat cycle. The selected reactor system differs significantly from the MTG process, and furthermore the pressure and the temperature level are considerably lower[2].

The synthesis product is cooled and fed into a three-phase separator in order to separate the condensable components and allow the gas phase to be recycled. In addition, the gasoline phase is separated from the methanol-containing aqueous phase. The gasoline (> 96 % yield in relation to the lower heating value, LHV) enters the gasoline stabilization column, and the methanol-containing water arrives at the methanol recovery. If the Freiberg gasoline synthesis plant – designed as a test plant – is to be operated without synthesis gas and without methanol synthesis, purchased methanol is used.

In the gasoline stabilization column, the gasoline is conditioned to the required quality. In this process the light components have to be removed at the column head and the heavy components at the bottom. After condensation, some of the light components are buffered in a reflux drum and recycled in order to adjust the column head temperature. The high-octane gasoline product is withdrawn at the lateral outlet and added to the product tank.

In the methanol column, water and methanol are separated. The methanol can be returned to the gasoline synthesis process. The effluent of the methanol column contains traces of methanol, which have to be removed in the catalytic wastewater treatment unit. The methanol-containing water runs through multiple preheating stages and has to be evaporated. The methanol is broken down in a catalytic process (pressure 50–55 bar, temperature 320–340 °C) with steam to excess (steam reforming of methanol) and delivers mainly $H_2$ and $CO_2$. This gas mixture can be returned to the methanol synthesis process. The re-condensed purified water (> 99.97 %) may be used to generate steam or added to the municipal waste water system (in compliance with the limits).

[1] The MTG process for the production of hydrocarbons from methanol was developed by Mobil Oil and others in the 1970s (methanol-to-gasoline, Patent: US 3,928,483 (1975) [1]) and is based on knowledge about zeolithic catalyst systems. Methanol is dehydrated to produce dimethyl ether (equilibrium reaction). Next, primary C2–C5 olefins are formed, which react to alkanes, (methyl-substituted) aromatic compounds and higher olefins in subsequent steps.

[2] The MTG process works between 300 and 450 °C at elevated pressures of up to 25 bar. Both fixed-bed and fluidized-bed reactors are used.

**Test campaigns and gasoline quality achieved**

After the STF plant was started up in Freiberg in June 2010, gasoline synthesis was tested in seven campaigns, each of them with continuous plant operation for two to six weeks. The tests were focused on optimizing the reaction conditions in the methanol reactor and the gasoline reactor. All in all, 79.053 kg of gasoline was produced in these long-term campaigns, which included heating-up and cooling-down phases lasting several days. After distillation, the stabilized gasoline satisfies or even exceeds most of the requirements of the technical specification DIN EN 228 [2] for gasoline, as illustrated in Table 1.

*Table 1: Characteristics of the STF gasoline (data from [3])*

| | | STF gasoline | gasoline specification acc. to DIN EN 228:2013-01 |
|---|---|---|---|
| Density (15 °C) | kg/m³ | 720 – 760 | 720 – 775 |
| Molar mass | kg/kmol | 95 – 96 | |
| Alkanes | vol% | 58 – 66 | |
| Olefins | vol% | 4 – 5 | max. 18 |
| Cycloalkanes | vol% | 6 – 9 | |
| Aromatic compounds | vol% | 22 – 30 | max. 35 |
| Benzene | vol% | 0.2 – 0.4 | max. 1 |
| Durene | vol% | max. 3 | |
| Research octane number (RON) | | 93 – 95 | min. 91 |
| Motor octane number (MON) | | 84 – 85 | min. 82.5 |
| Vapour pressure | kPa | 50 – 80 | max. 100 |
| Boiling range | °C | 20 – 200 | 20 – 210 |

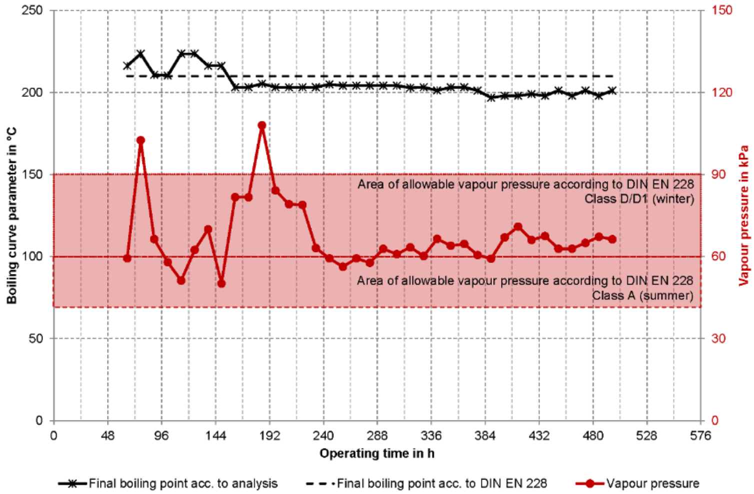


*Figure 2: Boiling curve and vapour pressure of the stabilized gasoline (test campaign 7)*

The separation of light and heavy components via distillation is necessary to fulfil the gasoline specification DIN EN 228. As an example, Figure 2 shows the conformity between the boiling curve and the vapour pressure for gasoline produced during the last test campaign with the standard specification. After the plant had been broken in, after approximately 200 h, the analysed gasoline samples matched the corresponding requirements.

**Study of technical and economical parameters of coal-to-liquid conversion using STF technology**

Besides the process chain from natural gas to gasoline, the combination of the STF process with some other feedstocks and syngas production technologies was also investigated. The gasification technology chosen for each feedstock was based on the market analysis of existing commercial-scale gasification plants [4]. Each of these process chains was adapted to specific technological peculiarities (e.g. gas purification and gas conditioning) and optimized in terms of maximizing the product yield and heat integration. Material and energy balances were calculated with the commercial software Aspen Plus and subsequently used as basis for assessing cost effectiveness. The process chains investigated are summarized in Fig. 3.

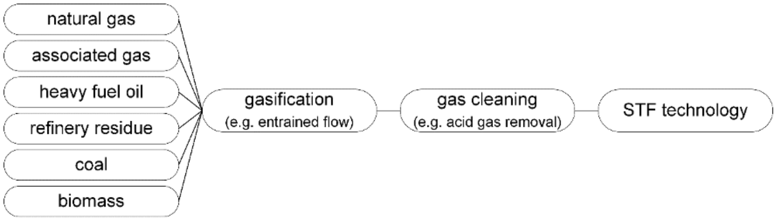


*Figure 3: Routes for synthetic fuel production using the STF technology*

*Table 2: Process chains considered to assess cost effectiveness*

| | Feed / Location | Gasification technology | Size | Raw gas treatment |
|---|---|---|---|---|
| 1.x | natural gas /<br>gas field, USA | Lurgi HP POX | 1000 MW | case 1: quench cooling<br>case 2: raw gas cooling<br>case 3: case 2 + syngas drying |
| 2.x | associated gas /<br>oil field, Russia | GE gasification technology | 200 MW | desulphurization, CO shift<br>case 1: quench cooling<br>case 2: raw gas cooling + drying |
| 3.x | heavy fuel oil /<br>refinery, Saudi Arabia | Lurgi Multi-Purpose Gasification | 500 MW | CO shift, sour-gas removal<br>case 1: quench cooling<br>case 2: raw gas cooling + drying |
| 4.1 | refinery residue /<br>refinery, Netherlands | Shell gasification process | 750 MW | CO shift, sour-gas removal |
| 5.1 | coal /<br>China | Siemens Fuel Gasification Technology | 1000 MW | CO shift, sour-gas removal |
| 6.1 | biomass /<br>Sweden | Pressurized circulating fluidized bed | 200 MW | CO shift, sour-gas removal |
| | **global properties:** purge and flare gases used with heat recovery steam generation, steam turbine for electricity generation if steam not used for heating or as a process steam, separated sulphur components processed in Claus plant | | | |

The plant capacity, which was selected as a reference point when considering the respective process route, was fixed depending on the typical quantitative availability of the feedstock, see Table 2. For example, biomass utilization plants are conceivable up to a thermal power of 200 MW [5]. Otherwise, in the case of natural gas reforming with the subsequent production of liquid fuels, several commercial plants with a thermal power higher than 1,000 MW have been successfully operated worldwide [6]. On the whole, the practical relevance of the chosen boundary conditions was foregrounded when assessing cost effectiveness. The process chains for different feedstock were deliberately spread across different world regions to take into account local specifics.
As depicted in Table 2, different cases were investigated for some of the feedstocks; these varied with respect to how the raw gas was handled (kind of gas cooling, gas drying). The corresponding case studies for the last three feedstocks led to similar results, thus only one case is shown.
Special attention was paid to the specific costs for the fuel feedstocks due to their strong influence on the operating costs. The data used for assessing cost effectiveness are listed in Table 3.

*Table 3: Specific price of different feedstock (April 2014)*

| **Feedstock** | **Specific price** | |
|---|---|---|
| Natural gas | 1.90 EUR/GJ | [7] |
| Associated petroleum gas | 0 EUR/GJ | Utilization of flare gases |
| Heavy fuel oil | 420 EUR/t | [8] |
| Visbreaker residue | 25 EUR/t | [9] |
| Coal | 58 EUR/t | [10] |
| Biomass | 65 EUR/t | [11] |

Assuming that construction starts in 2015, the main parameters used to assess cost effectiveness are an economic utilization period of 20 years, a plant capacity factor of 91.3 %, a mean common inflation rate of 2 %, a mean real cost increase of 1 %, a calculated interest rate of 10 %, and an insurance fee of 0.5 % of the plant costs.
The total investment costs include the onsite costs based on CEPCI values (Chemical Engineering Plant Cost Index) of 2014, offsite costs (2 % of onsite costs), indirect costs for engineering and supervision (5 % of direct costs) and for planning and construction (8 % of direct costs), costs for the initial plant start-up and interest costs for the plant construction phase. Table 4 summarizes the total investment cost of each selected process route.

*Table 4: Calculated investment costs (reference year: 2015)*

| **Route** | **Capacity (based on LHV)** | **Fixed capital costs (FCI)** | **Total capital costs (TCI)** |
|---|---|---|---|
| 1.1 | 1000 MW | 699 mill. EUR | 834 mill. EUR |
| 1.2 | 1000 MW | 791 mill. EUR | 932 mill. EUR |
| 1.3 | 1000 MW | 746 mill. EUR | 882 mill. EUR |
| 2.1 | 208 MW | 275 mill. EUR | 321 mill. EUR |
| 2.2 | 208 MW | 286 mill. EUR | 334 mill. EUR |
| 3.1 | 500 MW | 508 mill. EUR | 626 mill. EUR |
| 3.2 | 500 MW | 520 mill. EUR | 640 mill. EUR |
| 4.1 | 750 MW | 679 mill. EUR | 802 mill. EUR |
| 5.1 | 1000 MW | 907 mill. EUR | 1078 mill. EUR |
| 6.1 | 200 MW | 271 mill. EUR | 322 mill. EUR |

The levelled production costs are calculated from the regular payments of the cash value of the necessary annual revenue based on the amount of gasoline produced and the plant capacity factor. They are summarized in Table 5 and shown in Figure 4. It can be seen that the range of production costs is broad due to the widely differing boundary conditions for the various routes.

*Table 5: Production costs for stabilized gasoline*

| Route | 1.1 | 1.2 | 1.3 | 2.1 | 2.2 | 3.1 | 3.2 | 4.1 | 5.1 | 6.1 |
|---|---|---|---|---|---|---|---|---|---|---|
| ct/kWh | 6.64 | 9.61 | 5.49 | 8.03 | 7.48 | 15.31 | 15.02 | 6.15 | 8.20 | 22.10 |
| EUR/GJ | 18.45 | 26.69 | 15.26 | 22.30 | 20.77 | 42.53 | 41.72 | 17.09 | 22.77 | 61.38 |
| EUR/l | 0.55 | 0.80 | 0.46 | 0.66 | 0.62 | 1.27 | 1.24 | 0.51 | 0.68 | 1.83 |

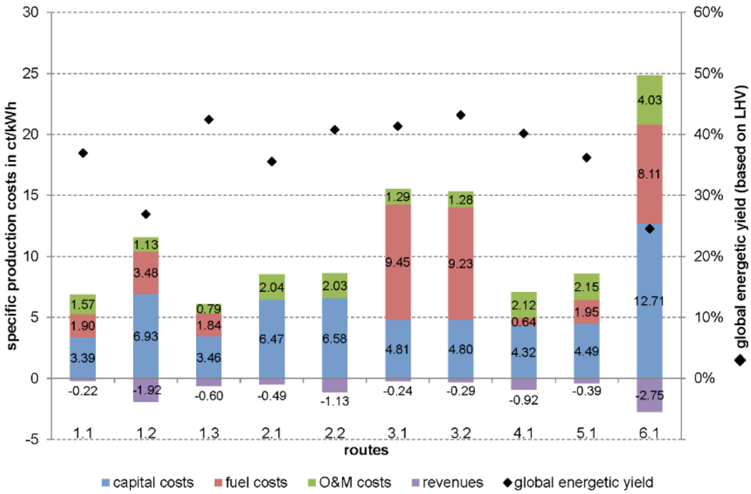


*Figure 4: Specific production costs*

It should be noted that the production costs for the synthetic gasoline generally significantly exceed the conventional gasoline price (in Germany, listing in Rotterdam, 0.519 EUR/l in March 2014 [12]). Individual exceptions are routes 1.3 (natural gas → Lurgi-ATR → STF) and 4.1 (Visbreaker residue → Shell gasification process → STF), which enable gasoline to be produced on a competitive basis under current boundary conditions.

## Summary

A novel process for the production of high-octane gasoline from synthesis gas (Syngas to Fuel – STF®) was developed by Chemieanlagenbau Chemnitz GmbH and Technische Universität Bergakademie Freiberg. It includes the conversion of syngas into methanol and the subsequent conversion of methanol into gasoline in isothermal reactors which are advantageous in terms of selectivity compared to adiabatic systems. After the STF demonstration plant was started up in Freiberg in June 2010, the whole process was systematically optimized and successfully operated in a series of long-term test campaigns, using synthesis gas generated by the autothermal reforming of natural gas. The STF process allows gasoline to be produced from different feedstocks including coal.
Investment costs and operation expenses were assessed for different scale-up scenarios. Using the economic characteristics and the calculated gasoline output, the production costs were estimated and

compared with current world market prices. The routes “Natural gas → Autothermal reforming → STF” and “Visbreaker residue → Shell gasification process → STF” enable gasoline to be produced on a competitive basis (under 2014 economic conditions).

The joint research project – finished in September 2013 – was supported with public funding by the Saxon State Ministry for Economic Affairs and Labour (SMWA), represented by the Sächsische AufbauBank (SAB), and financed by the European Regional Development Fund (ERDF) as well as by private investors.

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## Environmentally friendly energy production technology based on water-coal fuel


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### Abstract

Coal, for several centuries is the main source of primary energy in the world economy. Poland is a country very rich in carbon. Taking into account a number of technical, economic, social and political Poland should use the opportunity for a more rational and efficient use of coal. The article presents an innovative technology for the energy production based on water-coal fuel CWL (Coal-Water Liquid), which will allow its production in an efficient ecologically and economically manner at every stage in the life cycle.

The proposed technology is a new solution in Poland, and in the most recent national literature in this field is a little publication. Its main advantages are:

increased ability of coal burning (reactionary) compared to eg. the combustion of pulverized coal in traditional boilers,

- reduction from 2 to 2.5 times the emission of gaseous pollutants (NOx, SOx), and particulates compared to conventional coal combustion technologies,
- reduces the capital and operating costs of flue gas desulphurization,
- improve combustion efficiency up to 99%
- the possibility of storage in containers and transport in tankers or pipelines,
- lack of fire and explosive hazards in the manufacturing process (wet milling contributes to the reduction of coal dust).



Keywords: coal, energy, water and coal fuel, water and coal slurry, environment, clean coal technology, low emission


### Introduction

The purpose of fundamental research leading to making a concept for CWF is based on the assumption that liquid fuel is easier to handle in distribution and transportation and is also safer because there is no possibility of self-ignition or self-explosion. For CWF, the transportation system and coal distribution from mining location to the port or the user can be accomplished using piping system because it is not necessary to construct an expensive solid fuel infrastructure. CWF can be used as oil substitute in existing boiler industries. The advantages of using liquid CWF in a boiler over pulverized coal are a cheaper capital cost, it is more compact in operation, it is easier in fuel handling, and it contribute to significant reductions of carbon footprint and emission type dust, $SO_X$ and $NO_X$.

So far developed technologies for conversion of coal in the gaseous and liquid products allow you to increase energy efficiency by receiving a product that can be transported by pipeline and stored in tanks. But these processes usually have a negative impact on the environment (generation of hazardous by-




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products, high energy consumption, Loon, 2008). Work on the new technologies that allow use energy benefits of carbon, minimize the cost of its use and risks resulting from its combustion, contributed to the orientation of scientific research towards to fuel consisting of coal and water CWF. CWF fuel because of its potential as an alternative fuel for existing boilers, can be used as future clean fuel in new power plants or coal-fired gas turbines has been and is the subject of intense research and interest since 1980. China, Australia, Russia and the USA are the countries which focus on research and development of fuels based on coal-water mixture (Wibberley, 2008). However, current preparation of CWF is based on a conventional milling techniques, which are technologically complex and characterized by low efficiency (he average particle size is about 50 microns).

A significant improvement of CWF production can be made by applying a new micronization technique based on high pressure liquid jets. The equipment based on this concept allows the generation of an average particle size around 10 to 15 microns in a single pass through disintegrator. The average coal particle size produced by conventional means is around 50 microns. The specific surface of 10-15 micron coal particles is 25 times greater than for 50 micron particles. This is very important for fuel burning dynamics. Smaller particles, means easier access to impurities buried inside the particles and a more efficient process of their removal during a clean-coal preparation process. Smaller particle size also means smaller impurity inclusions and their full oxidation during the burning process. This is relevant for eliminating ash generation during combustion. Ecologically, a CWF combustion product is highly favourable.

Using micronization technique based on high pressure liquid jets variable grain-size distribution of coal in the coal-water slurry (CWS) in the range 5-30μm and water content in CWS in the range 25-50% can be reached. Such properties will increase combustion effectiveness to 99% and thermal effectiveness to >80%, which will be a landmark addressing one of the major challenges for the next decade in the EU. Economic and society needs for innovative and ecoefficiency solutions in energy sector are underlined in many EU and other countries policies. The advantages of the proposed technology and the CWF production are following: very low electricity consumption figure for production of 1Mg CWS – ~40kWh; grain-size of coal grains suspending in CWF even <10μm. This results in higher combustion and thermal efficiencies when compared to current CWF containing coal grains with a grain-size >50μm; all technological properties of liquid fuels, especially regarding its suitability for being pumped with typical pumps, and transported in tanks or through pipes, or stored in typical tanks; retention of physical and chemical properties over long periods for e.g. transportation and storage; no fire and explosion hazards; safety for the environment during production and use; possibility of decreasing 1.5-3.5 times of carbon footprint, $NO_X$, $SO_X$ and dust emissions in exhaust gases by comparison with coal dust firing; decreasing operation costs of the carbonizing systems in heat-generating plants by 15-30%; combustion effectiveness to 99%, thermal effectiveness >80%.

## CWL characterization

Ground coal, water and various additives (stabilizers, reducing viscosity and antifoam components) are the basic components of coal-water fuel (*CWF, CWS*).

Suspended coal fuel is composed of finely milled coal dispersed in one or more liquids such as water, oil or methanol. Suspension or coal mixture are interchangeably used terms. Studies of this type of fuel were carried out for its application in furnaces and boilers so far fueled with oil, in cement rotary kilns, gas turbines, internal combustion engines or as an energy feedstock for co-firing in boilers (Hycnar, 2001; Zimny, 2001; Zhang, 1977).

Calorific value, particle size distribution, stability, viscosity and solids content are parameters that characterize coal-water slurry and determine the possibility of its use.

The calorific value that determines the energy coal-water slurry (for a suspension containing 49-75% of coal) varies from 15.9 to 26.3 MJ/kg and it depends on the amount of water and the type of coal from which a suspension was made of (Hycnar, 2001). Grain size of slurry coal has a significant impact on the rheology and therefore constitutes the main part of all suspensions proprietary technologies. The most economical coal-water slurry is one that contains the maximum amount of coal at the lowest possible viscosity, which meets the requirements during storage, transport and fuel atomization. The solids content is a parameter that specifies the amount of coal in suspension, and influences its viscosity and determines the calorific value of the fuel. On the one hand, the increased solids content increases the calorific value of the fuel, on the other, it leads to obtain higher suspension viscosity, which makes it difficult to pump and injection into the burner.

The viscosity of the coal-water slurry gradually increases with the content of coal particles until the critical point in which a large number of particles causes their mutual friction and rapid increase in viscosity - the slurry stops flowing (Turian, 2002). Fuels having the same coal content and smaller grains have a higher viscosity which results from an increase in the resistance of fine particles against to the movement in water, which in turn is caused by the increase in density and total surface area of the grains. In order to allow pumping fuel at a lower energy demand it is required to obtain slurries with lower viscosity (e.g., by adding a coarser grains fraction).

## CWL preparation

The stability of the suspension is determined by its composition and grain size and it is the main parameter that determines the quality and decides on further processing and use of suspension. If the system consists of particles smaller than 1 μm than in order to prevent its sedimentation, Brownian motion take place. For particles larger than 1 μm, suspension is thermodynamically unstable hence the it tends to form deposits caused by the action of particles gravity (Lee, 2007). Consequently, the coal-water slurries are the dispersions and the coal grains have a relatively large size, even if there is no aggregation, the sedimentation of particles takes place. In order to stabilize the dispersion of coal-water, dispersing agents such as surfactants and electrolytes are used (Aktas, 2000). Process for the preparation of coal-water slurries consists of three stages: a preliminary crushing coal, milling and homogenization of the resulting suspension. Crushing of material is carried out by using jaw crusher, hammer or tapered, so that the material is prepared for further grinding. In order to obtain a particle size less than 150 μm, fraction of coal less than 12 mm is subjected to wet milling. During the wet milling stabilizers and plasticizers may be introduced into the suspension. Their addition at this stage of milling results in that the resulting new surface of the particles are immediately coated with a substance before the coal is exposed to weathering factors (Turian, 2002; Tiwari, 2003). For coal wet milling roller mills and ball mills coupled with air classifiers are used. Figure 1 shows a flow diagram for the production of coal-water slurries, that consists of two types of mills, which allows to obtain a suspension which is characterized by a bimodal grain size distribution.

A milling device used for the production of water-coal slurries comprises of a system of two blades, where one is movable and the other is immobile. The second blade has an opening in the middle through which the suspension is dispensed for grinding. The arrangement of the grinding elements of the mobile blade and the immobile blade are mirror images of each other, and the space between the blades is adjustable.

The grooves in the shape of “swallow tails” are the grinding elements of the blades. They have a 30° axial tilt and are centrically arranged on the blade. The advantage of this device is its simple, compact design which makes the grinding process effective by the possibility of changing the grinding parameters, especially the amount of the dosed feed (by adjusting the space between the blades) and the speed of rotation of the mobile blade, which affects the grain size of the particles in the coal-water slurry.

Innovative is also a method of producing water-coal slurries on that device. It is characterized by the fact that the feed - the suspension prepared by mixing water and coal, in which the maximum coal content in the slurry directed to grinding does not exceed 91% by weight - is ground by two grinding blades. Coal particle size in the feed is less than 6 mm. The distance between the blades is regulated in the range of 0.5 to 1.5 mm, and the rotation of the mobile blade is regulated in the range of 300-400 rev / min.

The effectiveness of the grinding can be controlled both by changing the distance of the two blades and the rotation of the mobile blade. The suspension inserted between the two blades is subjected to intensive grinding, as a result of which the coal particle size in the ground coal-water slurry is below 20 microns. Such a deep fragmentation of coal, and consequently its large specific surface area, opens up very practical opportunities, in particular it makes atomization of water-coal slurries easy and the process of combustion more dynamic. It also contributes to the more efficient oxidation of mineral fraction, minimizing the amount of ash and reducing emission of toxic products in the exhaust gases, while maintaining the simplicity of the device and better economics.

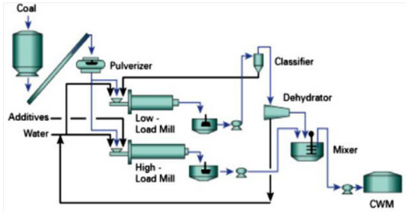


Fig.1. Diagram for the production of coal-water slurries

## CWL combustion

Coal - water fuel combustion in boilers and furnaces processes have already been carried out since 70s of the last century. The low stability of the flame, incomplete combustion and corrosion of equipment caused by gases emitted are the main difficulties emerging during combustion of CWL. Incomplete combustion of coal fuel may be due to the introduction of suspensions into insufficiently heated chamber or delayed ignition caused by evaporation of waters. The deposition of pollutants, slag, and corrosion of boilers and furnaces due to the presence of sulfur in the coal and mineral parts are the additional difficulties occurring during combustion of CWL. However, the combustion of this type of material is easy to achieve in cement kilns and fluidized beds. Coal - water fuel combustion is a multistep process. In order to describe this process one should consider the phenomena that occur on the surface and inside the droplets of sprayed suspension. On the surface of the drop, heating and evaporation of moisture takes place. A low activation temperature prior to combustion reaction and combustion of the coal microparticles on the drop surface are observed. The group of phenomena occurring inside the drop includes the gradual heating, evaporation of water contained in its interior, degassing, and then combustion of volatiles in the vicinity of droplet, transfer of coal microparticles to the droplet surface, forming of spherical recesses and burning of coal microparticles in the presence of air and steam (Mochalov, 2012; Burdukov, 2002). There are three methods of combustion: classic flame combustion, fluidised bed combustion and co-incineration with gas, oil or coal. Flashpoint occurs at 450-650 ° C and the combustion temperature is 950-1050 ° C. According to some researchers, combustion of the carbon particles takes place before the evaporation of water, at the temperature of approx. 1000 ° C, in which the suspension of combustion and decomposition of hydrogen to ions H + and OH- occurs.

Due to the amount of impurities that result from traditional coal mining, power generation based on this raw material is considered to be "dirty energy". The ideal would be to refine so as not to affect the environment. Achieving such a state is not feasible, but it does not bother in striving and introducing more and more new technological solutions (Nikodem, 2009). The use of coal - water fuel instead of conventional fuels reduces the impact on the environment, and is therefore a solution to the problem of further use of coal.

Therefore, one of the main parameters determining the use of coal-water slurry as a fuel is the environmental aspect. It is important to accurately test the combustion of CWL due to the emission of $CO_2$, $SO_2$ and $NO_x$. Based on the study of emissions from the preliminary tests burning of the water-coal slurries on the experimental installation, it might be said that their burning allows to minimize emission of $SO_2$ to the level of the emissions equal to the burning of natural gas. Very low contents of sulfur dioxide are, according to the literary data, due to the fact of the oxidation of $SO_2$ to $SO_3$ under specific conditions of combustion of coal-water slurry and then the formation of sulphates as a result of their reaction with a mineral phase of coal. These sulfates remain in the ashes.

Based on the state of art it may be assumed that using of coal slurry as a substitute for coal or heavy fuel oil in the processes of firing of power boilers reduces emissions of nitrogen oxides (approximately 2-times), sulfur oxides (about 5- times) and dust (about 1.2- times, Hycnar, 2001).

The technology for production of coal-water slurries in conjunction with proper energy policy may allow for the introduction of an efficient low emission economy. An excessive burden on the environment resulting from the emission of gases produced during combustion of fossil fuels justify the need for research in the field of receiving and combusting of coal-water slurries.

## Analysis of cost effectiveness

In order to determine the economic benefits of the use of the proposed technology, economic analysis was performed using analysis of financial flows and calculating the net present value NPV and the financial internal rate of return IRR. The economic analysis was performed for an exemplary system with several boilers, one of which has a power of 5 MW. It was estimated that the average sales value per unit of heat power was 6500GJ / MW. It was assumed that the installation would be led by the crew working in the boiler room / industrial plant, which would not increase the cost of heat production. The price of chemical energy used in coal mules was estimated to be 7 PLN / GJ. The investments include the construction of mills for hot water preparation directly next to the boiler room. The analysis refers only to the modernization of one boiler 5MW that uses hydrocarbon CWL fuel.

**Table. 1 Analysis of cost effectiveness of CWL technology (payment unit PLN)**

| | | | |
|---|---|---|---|
| **Variable costs** | 4 572 804 | *the cost of manufacturing - PLN per 1 GJ | 22,64 |
| **Including:** | | | |
| Coal | 312 963 | Revenues | 1 137 500 |
| Electricity – boiler | 75 833 | Sales of heat | 1 137 500 |
| Electricity – grinding | 137 588 | Other revenues | 0 |
| Water consumption – boiler | 3 340 | **The quantitative data** | |
| Water consumption – grinding | 1 267 | Sales of heat in GJ | 32 500 |
| The cost of spare parts for mills | 23 757 | Adopted price of heat for 1 GJ | 35,00 |
| The additional cost of servicing mills | 0 | **EBITDA** | 401 696 |
| Environmental tax | 18 056 | Capital expenditures for the mill | 1 000 000 |
| Fixed costs | 163 000 | Capital expenditures for the boiler | 300 000 |
| Including: | | Capital expenditure for the burner | 300 000 |
| Personnel costs | 0 | Other investments | 500 000 |
| Other costs | 100 000 | The sum of investments | 2 100 000 |
| Local taxes (3% of the investment) | 63 000 | Depreciation 10% | 210 000 |
| The sum of costs | 735 804 | **EBIT** | 191 696 |

The analysis shows only the first year of exploitation of the installation, and taking into consideration the increases in inflation at 3% per annum, the results are as follows: IRR -18%, NPV (rediscount rate 8%) 1 135 742 PLN.

The obtained results demonstrate economic profitability of the analyzed system and are most welcomed in the energy sector enterprises. The introduction of hydrocarbon CLW fuel to the basis of heat production, and thus the increase in sales of heat within the same investment costs, further contributes to the improvement of financial parameters of the project. It should be noted, however, that a more accurate economic analysis can be performed only after conducting an extensive research during quarter- and half industrial phase. The conducted analysis indicates that the best location for installation CWL will be small and medium-sized heating plants, where costs of the production of heat are relatively high, but the economy of scale is already clearly visible in the investment.

## Conclusion

Due to its characteristics CWL can be successfully used as a substitute for coal or fuel oil for combustion in industrial boilers. Laboratory tests indicated the potential environmental benefits arising from the use of this fuel. Emissions of NOx, SOx, CO produced in the combustion of CWL fuel are 2 to 5 times lower than in conventional coal technology. This is mainly due to the increase of combustion efficiency to 99%, and effective oxidation of the mineral fraction. Moreover, due to the rheological properties of the fuel, it can be safely transported by pipelines and stored in tanks. Additional environmental benefits can be achieved by using fuel production waste materials, such as coal muds, without the need for prior drying. Production of hydrocarbon CWL fuel may be a chance for the Polish economy, both in terms of increasing the country's energy supply security and reducing the negative influence of energy on the environment. It is further demonstrated by the possibility of utilizing raw materials from the domestic resource base, including wastes (sludges and muds), increasing the efficiency of the production potential of active mines, adjusting functioning coal-fired boilers with a power of 5-100 MW to new emission standards. In industrial practice, aqueous coal fuels are much safer than solid fuels because there is no risk of self-ignition or explosion, which contributes to the possibility of safer storage and transportation (compared with fuel oils, etc.). The transportation and distribution system based on ordinary pipeline, eliminating the need for building other infrastructure for the transportation of solid fuels.

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**STUDY OF THE POSSIBLE USE OF PRODUCER GAS OF COAL GASIFICATION AS FUEL**

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**Key words:** solid fuel, gasification, gas generator, combustion calculation, temperature, carbon dioxide, water, dissociation, calorific value.

Energy consumption around the world constantly increases for a number of reasons, which are connected with the population growth, industrialization and rapid economic growth of developing countries. At the same time reserves of the main energy sources, at least, explored and easy mined with use of the existing technologies, remain for 40-70 years by different estimates on Earth nowadays. Whereas reserves of solid fossil raw materials, such as coal, combustible shales, peat, sapropelite, etc., should be enough for mankind not less than for one thousand years. Volumes of the raw materials relating to renewables in general are considered unlimited [3].

In one scenario according to the international non-governmental organization – the Club of Rome – by the middle of the 21st century global reserves of raw materials will have been decreased three times, and the volume of industrial production will have fallen repeatedly. In another scenario, the resources will have been reduced only by one third, and the volume of industrial production will have remained at the level of beginning of the century. Both the first scenario and the second one provide for industrial stagnation. For the dynamic and sustainable development of technological civilization it is unacceptable, and vigorously growing economy of China, India, Brazil and Russia deny forecast of the Club of Rome, since the possibility of scientific and technological progress to reduce energy consumption and the use of alternative sources have not been taken into account in the forecasts, the increased technological potential of humanity has not been considered also.

Advantages of gaseous fuels are well-known: they are well adapted to the transport, ash, soot and large volumes of flue gases aren't emitted during combustion, the combustion process is easily automated, the temperature of the combustion process is usually much higher than that obtained with the direct combustion of liquid and solid fuels. Artificial combustible gases are valuable raw material for the chemical industry.

The main advantage of producer gases of solid fuel is relatively low level of negative impact on environment. First of all it is caused by long enough presence of gaseous products at first in zone of oxidation at temperatures from 1000-1200°C and higher, and then in a reducing zone of producer gas formation. It prevents the formation of various oxides of nitrogen and sulfur, and toxic substances such as dioxins, furans, polychlorinated biphenyls, benzo(a)pyrene, and other polycyclic aromatic hydrocarbons are subjected to thermal decomposition and reductive dechlorination.

Another advantage of gasification compared to direct combustion of solid fuel is the formation of much smaller volumes of gas to be cleaned. Furthermore, as a result of more complete combustion of gaseous fuel much less environmentally harmful chemical compounds are produced,

 
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which saves on scrubbing equipment of flue gases emitted into the atmosphere (cost of such equipment is more than 50% in composition of incineration plants), and secondary solid waste decontamination equipment.

The presence of significant volume of ballast inert components also has its positive side, as the producer gas compared to natural gas is less explosive.

Finally, almost 100% conversion of the carbon in its transition from the solid to the gaseous state takes place during gasification, and unreacted carbon is practically absent in the producer gas and ash.

Almost all carbonaceous materials of natural and anthropogenic origin could be used as raw material for gasification.

Due to depletion of the main sources of energy (oil and gas) many experts link the future of the global energy industry with the capabilities of coal gasification. Since the calorific value of the producer gas obtained from the gasification is relatively small in comparison with natural gas, the research on the possible use of producer gas in industrial plants of Russia as an alternative to natural gas, access to which is not available in all regions, have been carried out in Mining University.

For example, natural gas, fuel oil and coal are known to be used as fuel for the sintering drum furnaces in the alumina production by sintering method [8]. The above-mentioned disadvantages of conventional sources of energy and advantages of the producer gas of coal gasification indicate the prospectivity of using the latter as alternative energy source in such energy-intensive enterprises as alumina plants processing bauxite by sintering method.

Since the calorific value of the producer gas obtained from the gasification is relatively small in comparison with natural gas it was necessary to study in detail the process of producer gas combustion.

For the study the producer gas has been chosen of composition, $\%_{(vol.)}$: 5,0-$CO_2$, 0,3-$H_2S$, 0,3-$C_mH_n$, 26,5-CO, 13,5-$H_2$, 2,3-$CH_4$, 51,9-$N_2$, 0,2-$O_2$ and moisture W=5,7 g/m$^3$.

The main characteristics of producer gas combustion in the excess air ratio $\alpha$ = 1,10 are presented in Table 1.

*Table 1.*

The calculated values of producer gas combustion

| Value | Magnitude |
|---|---|
| Calorific value $Q_{lower}$, kJ/m$^3$ (kcal/m$^3$) | 5866 (1400) |
| Theoretical amount of air $L_{theor.}$, m$^3$/m$^3$ | 1,22 |
| Practical amount of air $L_{pr.}$, m$^3$/m$^3$ | 1,35 |
| The amount of producing combustion products $V_g$, m$^3$/m$^3$ | 2,15 |
| Composition of combustion products, % vol. | |
| $CO_2$ | 15,92 |
| $H_2O$ | 9,23 |
| $N_2$ | 73,52 |
| $O_2$ | 1,19 |
| $SO_2$ | 0,14 |
| Density of combustion products ρ, kg/m$^3$ | 1,33 |
| The heat content of the combustion products $i_0$, kJ/m$^3$ (kcal/m$^3$) | 2728 |
| Calorimetric temperature of combustion process, $t_c$°C | 1646 |

Since the calorimetric temperature of combustion is practically unattainable because of the expense of thermal energy for a partial dissociation of the combustion products, the theoretical temperature or the temperature of the combustion products at partial dissociation of carbon dioxide and water vapor are calculated [2, 4]. The process of dissociation of the combustion products are described by thermochemical equations:

$$2CO_2 \leftrightarrow 2CO + O_2 \text{ -3045 kcal/m}^3 \quad (1)$$

$$2H_2O \leftrightarrow 2H_2 + O_2 \text{ -2580 kcal/m}^3 \quad (2)$$

Determination of the theoretical temperature is harder then determination of the calorimetric temperature. Quickly, but roughly, the theoretical combustion temperature can be determined using a schedule shown in Figure 1, where carbon dioxide dissociation degree and water vapor dissociation dissociation degree in dependence on temperature and partial pressures of the components are presented [2, 4, 6].

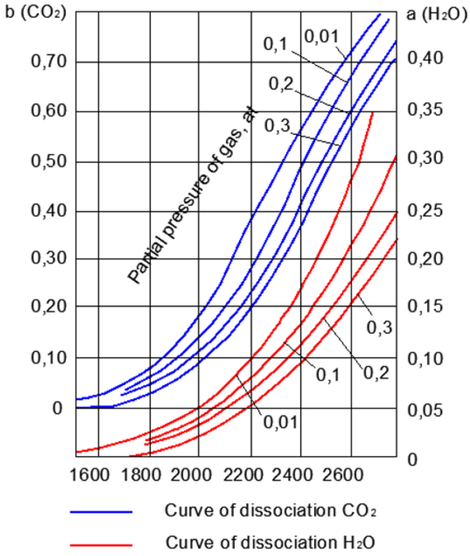


Figure 1. **The graph for determining the dissociation degree**

For accurate determination of the theoretical temperature it should be taken into account that the combustion products contain simultaneously $CO_2$ and $H_2O$, and balance should be between the five components:

$CO_2$ $CO$
\ /
$O_2$
/ \
$H_2O$ $H_2$

The problem can be solved in the following order. By specifying different values of the coefficient of dissociation temperature of the combustion products is determined appropriate to the value of the coefficient of dissociation. According to the obtained points curve is made in the coordinate axes (coefficient of dissociation – temperature). Then, by specifying values of temperature values of coefficients of dissociation are determined from the equilibrium equation. On the same grid new data is applied. The intersection of the two curves gives values of the theoretical temperature and coefficient of dissociation (Figure 2).

This graphic-analytical method was proposed by Academician N.S. Kurnakov and recommended by Professor B.V. Stark as the only possible method in case when the combustion products contain at the same time $CO_2$, $H_2O$, $CO$, $H_2$ and $O_2$ [4].

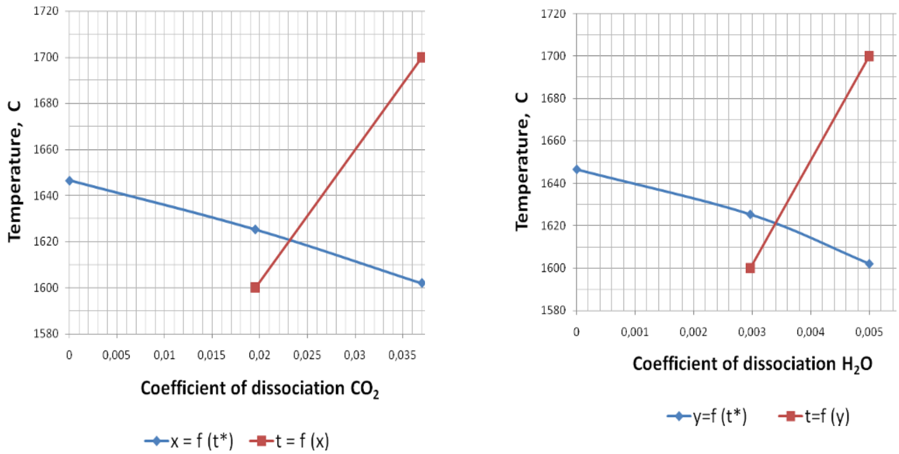


Figure 2. **Determination of the theoretical combustion temperature of the producer gas**

Following the main principle of N.S. Kurnakov's school: "Theoretical questions should be studied in the technically important objects" [5, 7], analytical method of calculation has been developed in Mining University, different from previously proposed method by higher accuracy, since there is no need for constructing charts, which is difficult to determine the exact temperature on. We illustrate the proposed method of calculation of the theoretical combustion temperature on the example of the producer gas.

During dissociation of carbon dioxide and water vapor the reactions are conducted:

$$2CO_2 \leftrightarrow 2CO + O_2 \qquad (1)$$

$$2H_2O \leftrightarrow 2H_2 + O_2 \qquad (2)$$

Subtracting reaction (1) from the reaction (2), we obtain:

$$CO + H_2O \leftrightarrow CO_2 + H_2 \qquad (3)$$

The equilibrium constants of reactions (1) and (3) are equal to:

$$K_1 = \frac{(CO_2)^2}{(CO)^2 \cdot O_2} = \frac{(1-x)^2(1+0{,}079x+0{,}046y)}{x^2(0{,}079x+0{,}046y)}$$

$$K_3 = \frac{(CO)\cdot(H_2O)}{(CO_2)\cdot(H_2)} = \frac{x\cdot(1-y)}{(1-x)\cdot y} \rightarrow y = \frac{x}{K_3(1-x)+x}, \text{ where:}$$

$x$ – coefficient of dissociation $CO_2$;

$y$ – coefficient of dissociation $H_2O$.

We use the method of successive approximations for solving systems of nonlinear equations [1]. We set up a system of equations having a sign of convergence, and introduce the new value $z$ to simplify the calculations:

$$\begin{cases} z = \dfrac{K_1 \cdot x^2}{(1-x)^2} = \dfrac{(1+0{,}079x+0{,}046y)}{(0{,}079x+0{,}046y)} \rightarrow x = \dfrac{-z \pm \sqrt{K_1 z}}{(K_1 - z)} \\ y = \dfrac{x}{K_3(1-x)+x} \end{cases}$$

**Set the equilibrium temperature 1600°C,** which $K_1=1{,}568\cdot 10^6$ and $K_3=6{,}69$ for.

Take $x = 0{,}02$, then

$$y = \frac{0{,}02}{6{,}69(1-0{,}02)+0{,}02} = 0{,}003$$

$$z = \frac{1{,}568 \cdot 10^6 \cdot 0{,}02^2}{(1-0{,}02)^2} = 0{,}5831 \cdot 10^3$$

$$x = \frac{-0{,}5831 \cdot 10^3 \pm \sqrt{1{,}568 \cdot 10^6 \cdot 0{,}5831 \cdot 10^3}}{\left(1{,}568 \cdot 10^6 - 0{,}5831 \cdot 10^3\right)} = 0{,}01892$$

Take $x$ = 0,01892 → y = 0,0028713 → z = 0,6158·$10^3$

Similarly, we calculate the $x$, $y$, and $z$ as long as the resulting values do not coincide with the set. The calculation results are shown in Table 2, which also shows the changes in the values of $x$ and $y$ with respect to the initial value.

*Table 2.*

Calculation results with using method of successive approximations

| Step | $x$ | $\Delta x = x_n - x_{n+1}$ | $y$ | $\Delta y = y_n - y_{n+1}$ | $z$ |
|---|---|---|---|---|---|
| 1 | 0,02000 | | 0,0030000 | | 0,5831·$10^3$ |
| 2 | 0,01892 | +0,00108 | 0,0028713 | +0,0001287 | 0,6158·$10^3$ |
| 3 | 0,01944 | -0,00052 | 0,0029553 | -0,0000840 | 0,5992·$10^3$ |
| 4 | 0,01918 | +0,00026 | 0,0029148 | +0,0000405 | 0,6074·$10^3$ |
| 5 | 0,01931 | -0,00013 | 0,0029350 | -0,0000202 | 0,6033·$10^3$ |
| 6 | 0,01924 | +0,00007 | 0,0029240 | +0,0000110 | 0,6054·$10^3$ |
| 7 | 0,01927 | -0,00003 | 0,0029290 | -0,0000050 | 0,6045·$10^3$ |

Composition and amount of combustion products:

| | | $m^3$ | Volume shares |
|---|---|---|---|
| $CO_2$ | 0,1592(1-0,01927) | 0,1561 | 0,1558 |
| CO | 0,1592·0,01927 | 0,0031 | 0,0031 |
| $H_2O$ | 0,0923(1-0,002929) | 0,0920 | 0,0918 |
| $H_2$ | 0,0923·0,002929 | 0,0003 | 0,0003 |
| $O_2$ | 0,079·0,01927+0,046·0,002929+0,0119 | 0,0141 | 0,0136 |
| $N_2$ | 0,7352 | 0,7352 | 0,7340 |
| $SO_2$ | 0,0014 | 0,0014 | 0,0014 |
| Σ | 1+0,079·0,01927+0,046·0,002929 | 1,0017 | 1 |

Heat content of the combustion products:

| | |
|---|---|
| Combustion of producer gas | 2728 : 1,0017 = 2723 kJ/$m^3$ |
| Dissociation $CO_2$ | -3045·4,19·0,0031 = -39,6 kJ/$m^3$ |
| Dissociation $H_2O$ | -2580·4,19·0,0003 = -3,2 kJ/$m^3$ |
| Σ | $i_0$ = 2723 - 36,9 - 3,2 = 2680 kJ/$m^3$ |

Find the temperature of a given heat content $t = \frac{i_0}{\sum v_r \bar{c}_r}$, where

$i_0$ – heat content of combustion products, kJ/$m^3$;

$v_g$, - volume share of the combustion product;

$\bar{c}_g$ - mean heat capacity of the combustion product, kJ/($m^3$·K).

Sequentially taking temperature, substitute values of heat capacity at taken temperature until identity is appeared.

$$t_i = \frac{2680}{0{,}1558\bar{c}_{CO2} + 0{,}7340\bar{c}_{N2} + 0{,}0136\bar{c}_{O2} + 0{,}0918\bar{c}_{H2O} + 0{,}0031\bar{c}_{CO} + 0{,}0003\bar{c}_{H2} + 0{,}0014\bar{c}_{SO2}}$$

Take $t_i = 1600°C \xrightarrow{step1} 1634°C \xrightarrow{step2} 1632°C \xrightarrow{step3} 1633°C$.

**Set the equilibrium temperature 1633°C**, which $K_1$=0,8054·$10^6$ и $K_3$=7,0 for.

Take $x$ = 0,02 → $y$ = 0,003 → $z$ = 0,5831·$10^3$.

*Table 3.*

Calculation results with using method of successive approximations

| Step | $x$ | $\Delta x=x_n-x_{n+1}$ | $y$ | $\Delta y=y_n-y_{n+1}$ | $z$ |
|---|---|---|---|---|---|
| 1 | 0,02000 | | 0,0030000 | | $0{,}5831\cdot 10^3$ |
| 2 | 0,02619 | -0,00619 | 0,0038305 | -0,0008305 | $0{,}4464\cdot 10^3$ |
| 3 | 0,02299 | +0,00320 | 0,0033501 | +0,0004804 | $0{,}5083\cdot 10^3$ |
| 4 | 0,02450 | -0,00221 | 0,0035758 | -0,0002257 | $0{,}4772\cdot 10^3$ |
| 5 | 0,02376 | +0,00174 | 0,0034641 | +0,0001117 | $0{,}4921\cdot 10^3$ |
| 6 | 0,02411 | -0,00035 | 0,0035186 | -0,0000545 | $0{,}4847\cdot 10^3$ |
| 7 | 0,02394 | +0,00017 | 0,0034915 | +0,0000271 | $0{,}4887\cdot 10^3$ |
| 8 | 0,02403 | -0,00009 | 0,0035065 | -0,0000150 | $0{,}4864\cdot 10^3$ |
| 9 | 0,02398 | +0,00005 | 0,0034975 | +0,0000090 | $0{,}4875\cdot 10^3$ |
| 10 | 0,02400 | -0,00002 | 0,0035020 | -0,0000045 | $0{,}4870\cdot 10^3$ |

Composition and amount of combustion products:

| | | $m^3$ | Volume shares |
|---|---|---|---|
| $CO_2$ | 0,1592(1-0,02400) | 0,1554 | 0,1558 |
| CO | $0{,}1592\cdot 0{,}02400$ | 0,0038 | 0,0038 |
| $H_2O$ | 0,0923(1-0,003502) | 0,0920 | 0,0918 |
| $H_2$ | $0{,}0923\cdot 0{,}003502$ | 0,0003 | 0,0003 |
| $O_2$ | $0{,}079\cdot 0{,}02400+0{,}046\cdot 0{,}003502+0{,}0119$ | 0,01400 | 0,0140 |
| $N_2$ | 0,7352 | 0,7352 | 0,7336 |
| $SO_2$ | 0,0014 | 0,0014 | 0,0014 |
| Σ | $1+0{,}079\cdot 0{,}02400+0{,}046\cdot 0{,}003502$ | 1,0021 | 1 |

Heat content of the combustion products:

| | |
|---|---|
| Combustion of producer gas | 2728 : 1,0021 = 2722 kJ/$m^3$ |
| Dissociation $CO_2$ | $-3045\cdot 4{,}19\cdot 0{,}0038 = -48{,}5$ kJ/$m^3$ |
| Dissociation $H_2O$ | $-2580\cdot 4{,}19\cdot 0{,}0003 = -3{,}2$ kJ/$m^3$ |
| Σ | $i_0 = 2722 - 48{,}5 - 3{,}2 = 2676$ kJ/$m^3$ |

Find the temperature taking into account the calculated heat content.

$$t_i = \frac{2676}{0{,}1558\bar{c}_{CO2} + 0{,}7336\bar{c}_{N2} + 0{,}014\bar{c}_{O2} + 0{,}0918\bar{c}_{H2O} + 0{,}0038\bar{c}_{CO} + 0{,}0003\bar{c}_{H2} + 0{,}0014\bar{c}_{SO2}}$$

Take $t_i = 1600°C \xrightarrow{step1} 1632°C \xrightarrow{step2} 1629°C \xrightarrow{step3} 1629°C$.

**Set the equilibrium temperature 1629°C**, which $K_1=0{,}872\cdot 10^6$ и $K_3=6{,}963$ for.

Take $x = 0{,}024 \rightarrow y = 0{,}0035 \rightarrow z = 0{,}4871\cdot 10^3$.

*Table 4.*

Calculation results with using method of successive approximations

| Шаг | x | $\Delta x=x_n-x_{n+1}$ | y | $\Delta y=y_n-y_{n+1}$ | z |
|---|---|---|---|---|---|
| 1 | 0,02400 | | 0,0035000 | | $0{,}4871\cdot 10^3$ |
| 2 | 0,02308 | +0,00082 | 0,003381 | +0,000119 | $0{,}5062\cdot 10^3$ |
| 3 | 0,02352 | -0,00044 | 0,003449 | -0,000068 | $0{,}4967\cdot 10^3$ |
| 4 | 0,02331 | +0,00021 | 0,003416 | +0,000033 | $0{,}5014\cdot 10^3$ |
| 5 | 0,02341 | -0,00010 | 0,003432 | -0,000016 | $0{,}4992\cdot 10^3$ |
| 6 | 0,02336 | +0,00005 | 0,003425 | +0,000007 | $0{,}5001\cdot 10^3$ |
| 7 | 0,02338 | -0,00002 | 0,003428 | -0,000003 | $0{,}4996\cdot 10^3$ |
| 8 | 0,02337 | +0,00001 | 0,003426 | +0,000002 | $0{,}4999\cdot 10^3$ |
| 9 | 0,02338 | -0,00001 | 0,003426 | -0,000000 | |

Composition and amount of combustion products:

| | | $m^3$ | Volume shares |
|---|---|---|---|
| $CO_2$ | 0,1592(1-0,02338) | 0,1554 | 0,1552 |
| CO | $0{,}1592\cdot 0{,}02338$ | 0,0037 | 0,0037 |
| $H_2O$ | 0,0923(1-0,003426) | 0,0919 | 0,0918 |

| | | | |
|---|---|---|---|
| $H_2$ | $0,0923 \cdot 0,003426$ | 0,0003 | 0,0003 |
| $O_2$ | $0,079 \cdot 0,02338+0,046 \cdot 0,003426+0,0119$ | 0,0139 | 0,0139 |
| $N_2$ | 0,7352 | 0,7352 | 0,7337 |
| $SO_2$ | 0,0014 | 0,0014 | 0,0014 |
| Σ | $1+0,079 \cdot 0,02338+0,046 \cdot 0,003426$ | 1,002 | 1 |

Heat content of the combustion products:

| | |
|---|---|
| Combustion of producer gas | $2728 : 1,0020 = 2722$ kJ/m$^3$ |
| Dissociation $CO_2$ | $-3045 \cdot 4,19 \cdot 0,0037 = -47,2$ kJ/m$^3$ |
| Dissociation $H_2O$ | $-2580 \cdot 4,19 \cdot 0,0003 = -3,2$ kJ/m$^3$ |
| Σ | $i_0 = 2722 - 47,2 - 3,2 = 2672$ kJ/m$^3$ |

Find the temperature taking into account the calculated heat content.

$$t_i = \frac{2672}{0,1552\bar{c}_{CO2} + 0,7337\bar{c}_{N2} + 0,0139\bar{c}_{O2} + 0,0918\bar{c}_{H2O} + 0,0037\bar{c}_{CO} + 0,0003\bar{c}_{H2} + 0,0014\bar{c}_{SO2}}$$

Take $t_i = 1629°C \xrightarrow{step1} 1627°C \xrightarrow{step2} 1627°C$.

Temperatures adopted 1629°C and obtained 1627°C differ by two degrees, that for thermal calculations is acceptable. Thus, the number of steps will determine the accuracy of the calculations.

## Conclusion

1. The method of successive approximations in the calculation of the theoretical fuel combustion temperature is more accurate and less time consuming as compared to the graphic-analytical method proposed by Nikolay Semenovich Kurnakov, due to lack of the need for constructing charts, which doesn't allow to determine the exact temperature.

2. Acceptable calorific value – $Q_{lower}$=5866 kJ/m$^3$ and temperature – 1627°C of producer gas combustion together with low level of negative impact on the environment allow to use this fuel as alternative source of energy in industrial enterprises of Russia in general, and energy-intensive enterprises as alumina plants processing bauxite by sintering method in particular.

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# Part VI
# Environmental Protection

# To the question of complex use of ashes and slag waste CHP plant

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**Abstract**

Existing technology for the processing and utilization of ash and slag material are devoted to the use of ash in construction and production of building materials to a greater extent. Technology for production of concentrates from ash and slag materials containing various metals, have significant drawbacks. They are connected with the quality of the resulting concentrates mainly. The current state of the problems associated with the accumulation, processing and disposal of coal combustion was studied in the work. The main directions of of ash and slag recycling materials include: magnetic separation, removal of unburned carbon and precious metals by a flotation method, acid leaching of aluminum components and the study of the distribution of rare and trace elements.The analysis of the practice enrichment of technogenic material by gravity, magnetic and flotation methods was carried out. The objects of the study were the ash and slag wastes CHP of Far East region and the ash from coals burning of Far Eastern Fields. Qualitative and quantitative microscopic analyses of material were presented. The same technological scheme which includes flotation, gravity separation, magnetic enrichment and the high-gradient magnetic separation followed by acid extraction of aluminum components was substantiated for complex processing of technogenic carbonaceous material. The indicator of complexity for the proposed technology has proven the effectiveness of complex processing.

Key words: ash and slag material, high-gradient magnetic separation, complex processing, acid extraction, underburning, qualitative indicator, resource conservation.

**Introduction**

The efficiency of all branches of industry should be assessed from the point of view of the balance between the weight of the final product and the amount of technological waste generated. Enterprises of fuel and energy complex, namely, thermal power plants are the most disadvantaged in this respect. It is the source of massive atmospheric emissions of large solid waste (ash and slag materials). Currently, in connection with annual decrease of proven mineral reserves, the ash can be a source of minerals during recycling [1-6]. The bulk of the ash is oxides of silica, alumina, iron, calcium, potassium and titanium. Also, the presence of valuable components such as gold, platinum group metals, rare-earth is noted with content that is, often reaching values that are optimal for industrial processing. However, a complicated material composition, the presence of unburnt coal (underburning), fine particles of the valuable components and their partial presence in colloidal form in the pore space of the material requires additional operations to classic technological processing schemes.

The basic properties of ash and slag materials are determined by the following factors:

- peculiarities of formation of coal seam (natural factors);
- peculiarities of the burning process of solid fuels, ash and dust removal (technological factors);
- peculiarities of storage of ash and slag waste (environmental factors).

 
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**The research**

The objects of the study were the ash and slag wastes (ZSHM) CHP of Far East region and the ash from coals burning of Far Eastern Fields. They are potential sources of metals.

The chemical composition of ash and slag materials can vary widely, but in general, it is considered as stable for the same type of fuel at a certain mode of combustion. The crystalline, glassy and organic components are distinguished in the composition of ash and slag materials.

The crystalline material consists of primiry minerals of fuel and new formation, which are obtained in the combustion process, during hydration and weathering in the ash dump. Up to 150 minerals has been found in the crystalline component of ash and slag materials. Predominant minerals are meta- and orthosilicates, aluminates, ferrites, spinels, dendritic clay minerals, oxides (quartz, tridymite, cristobalite, alumina, alumina, and others). Native elements and intermetallic compounds (lead, silver, gold, platinum, mercury, iron, nickel iron, copper gold and various alloys of copper, nickel, chromium (silicon) and others) are very interesting. The vitreous material is a product of unfinished transformations during combustion. It composes essential part of the ash. Mostly, it is presented black glass with submetallic, variety of spherical vitreous microspheres (beads) and their aggregates (figure 1). On structure it is alumina, potassium, sodium and, less calcium.

*a)* *b)*

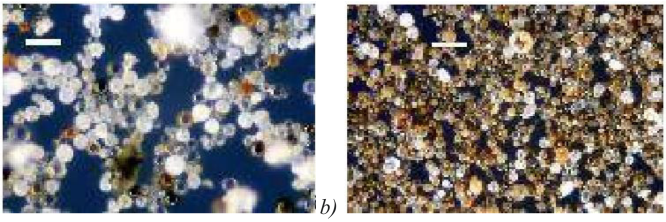

Fig.1 Microscopic image of finely divided technogenic materials

a) microspheres of fly ash (datum mark - 25 microns);

b) the average fraction of lignite ash (datum mark - 100 microns).

Organic material is presented burnt fuel particles (underburning) (figure 2). Organic material after combustion is found in the form of coke and coal char with a very low hygroscopicity, and is quite different from the original material. Quantity of unburned carbon in the of ash and slag materials is an average 10-15%.

*a*) *b)*

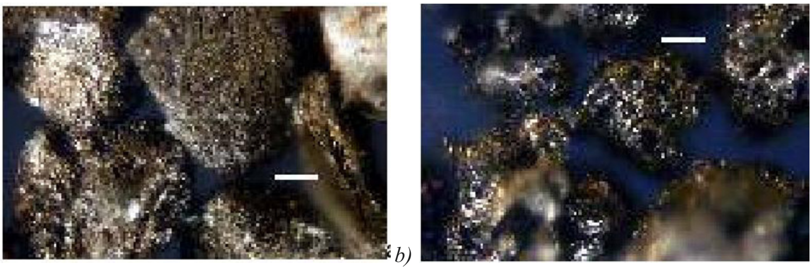

Fig. 2 - The organic component of the ash and slag materials

*a)* carbon particles in brown coal ash (datum mark 100um);

*b)* coke from fly ash.

Electromagnetic fraction is composed of oxide scale, differently colored spherical aggregates of quartz-ferrous composition to 75-100% (figure 3).

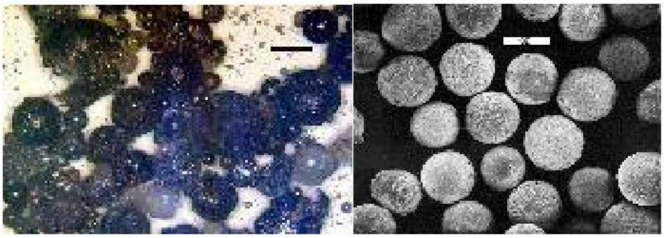

Fig.3 Magnetic fraction of the fly ash from the combustion of coal

The magnetic fraction is composed of magnetic spherical aggregates predominantly of black color, nuts, magnetite, pyrrhotite, limonite, hematite, pyroxene, chlorite and epidote. It is from 70 to 98.5% of the heavy fraction. Rare grains of platinum and various alloys of iron - nickel-chromium composition sporadically are observed in magnetic fraction.

A similar set of elements normally contains the host rocks, but in lower concentrations. The bulk of the metals is localized in coal and ash of coal in a fine mineral form (1-10 microns). Various native metals and intermetallic compounds, sulphides, carbonates, sulphates, tungstates, silicates, phosphates, rare earths, niobates were found. Each of the metal forms several mineral phases, for example, tungsten is marked in the form of wolframite, stolzite, ferberite, scheelite and as impurities. The composition of the compounds varied as the morphology of the grains: well defined and skeletal crystals, aggregates and delicate clusters of crystals (Fig. 2), doubles, pieces of crystals; druze, globules and microspherules; porous shapes flocculent and lamellar packs, lumpy clusters and et al.

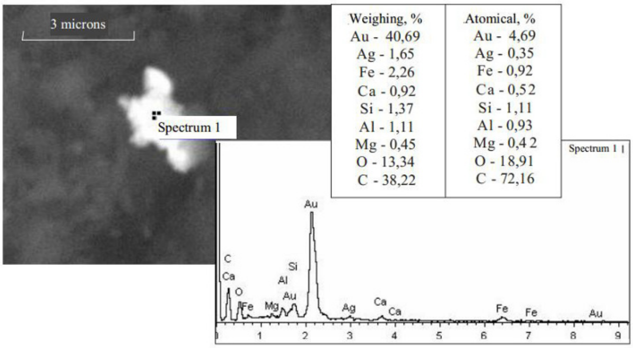


Fig. 4. Grain of native gold with admixture of silver and iron (3 microns) of friable form in the form of a loose form or agglomerated lumps splice crystals. The gold content in the sample is 0.6 g per t.

Material ashes and CHP slag waste were introduced by a thin class size with a share of class -40 microns up to 20%. The studies of technological properties of ash waste dumps from the viewpoint of the recovery of noble metals have shown that gold mass fraction in the studied samples ranges from 0.1 to 0.9 g per t. Part of gold is in free form and can be extracted by gravity methods. Quantity of gravity recoverable gold is ranged from 5 to 45%, sometimes reached 70%. Material, selected from stale waste dumps, contains more free gold than the current material waste dumps. A minimum quantity of free gold contains in samples are taken at unloading incinerators, meanwhile, "hurricane" gold contents (up to 3-20 g per t) are fixed in some samples. Presence of the largest gold particles was revealed in these samples.

Electronmicroscopic examination of dressing products (removal of unburned carbon, flotation) shows micron grains of native gold in the samples, sophisticated man -made alloy of gold and silver grains were marked, as well.

Petrochemical characteristics of the material are calculated on the basis of chemical analysis of the contents of the main components of ash and slag materials.

Silicate module ranges from 1.82 to 1.99, the mean value is equal 1.882. Lime factor varies from 0.076 to 0.096, the mean value is equal 0.086. The quality factor varies from 0.02 to 0.04, the mean value is equal 0.03. In general, ashes are acidic, a sufficiently high content of aluminate.

The research results have become the basis for designing industrial scheme of processing of ash and slag materials [6-10]. Machine flow sheet of the complex processing is shown in figure 5.

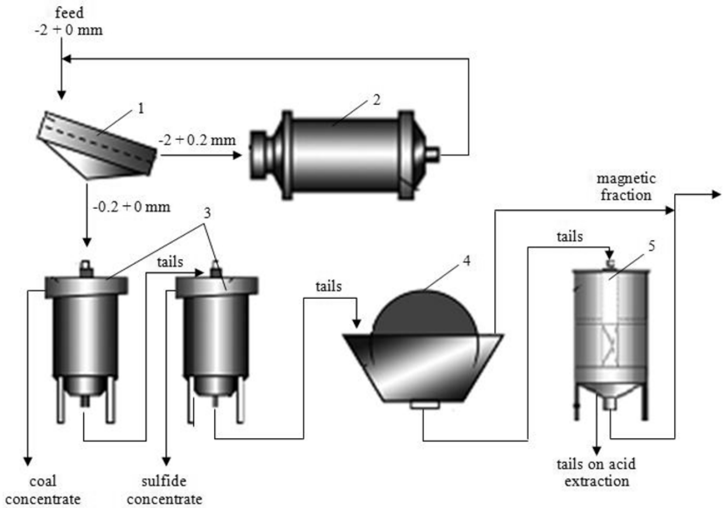


Fig. 5 Machine flow sheet for processing of ash and slag materials CHP
1 - screen 2 - mill, 3 - flotation machine, 4 - magnetic separator, 5 - high-gradient magnetic separator

The resulting coal concentrate can be used as an additive to obtain coal briquettes hereinafter. Iron concentrate (containing chromium, nickel, molybdenum, vanadium) may be used as raw material for the metallurgical industry to obtain alloys with specific properties. Significant amounts of magnetic microspheres are used in the cement industry to adjust of iron module clinker. The resulting aluminum-containing product can be used as a coagulant for water purification.

REE may be isolated from an extract of the acid leaching of aluminum components using selective solvents (high molecular weight acids: stearic, oleic, naphthenic, as well as primary and tertiary amines) at complex processing of of ash and slag materials.

**Conclusions**

The same technological scheme which includes flotation, gravity separation, magnetic enrichment and the high-gradient magnetic separation followed by acid extraction of aluminum components was substantiated for complex processing of technogenic carbonaceous material. The indicator of complexity for the proposed technology has proven the effectiveness of complex processing.

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# ABANDONED COAL MINES INFLUENCE ON ATMOSPHERE , ENVIRONMENTAL MONITORING OF COAL MINING AND PROCESSING TERRITORIES AND PREVENTIVE MEASURES RESOURCES OPTIMIZING

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***Abstract:*** *Generalizing field observation results of gas changing open areas with atmosphere on Earth surface of mining lease liquidated mines territories in Kuznetsk and Donetsk Basins illustrates that improving regularities of this process is necessary for providing environmental safety of undermining territories. It's shown that different gases filtration to Earth surface from undermining rocks realizes as result of these gases excessive pressure in open areas. Neglecting mining operation safety by gas factor is cause of major accidents, which giving heavy losses. On the other hand, reliable forecasting gas emission can provide high level of mining safety and create technical measures by aerological safety of liquidated mines mining leases. Scientific-practical results of monitoring anthropogenic influence upon environment of mining-industrial territories were discussed. Evaluating risk of negative influence is very important phase of environmental monitoring. The first is evaluating probability of appearance negative anthropogenic influence to environment, which more then maximal level. The second is evaluating probability of failure at the environmental protection system. The third is evaluating detriment of negative anthropogenic influence upon environment.*

***Key words:*** *monitoring, forecasting, environmental protection, mathematical model, optimizing, anthropogenic influencing, pollutant, atmosphere, optimal distribution, mining enterprises.*

## 1. INTRODUCTION

Global environmental problems of the present are connected just with anthropogenic polluting atmosphere. Protection of atmospheric air is key problem of environmental sanitation, because air has special rule among others components of biosphere. Atmospheric air realizes different protective ecological functions too. Sphere of influence upon atmosphere and increasing negative changing environment becomes wider with development of social production. It is especially typical of mining region territories, as well as for post-mining abandoned coal mines areas worldwide [1-3]. One of modern society most topical problem is improvement of management and industrial efficiency rise. Industrial rise and material welfare already aren't considered without taking into account environmental influencing these processes [4]. Important conception environmental-economical system appeared. It is combination of interconnected economical, technical, social and natural factors in visual environment.

Increasing scale of environmental influencing imparts special topicality for problem of creating adequate environmental-economical mathematical models. Increasing scale of anthropogenic influence upon natural environment and its negative consequences and else possibilities of optimizing this influencing require in-depth study. The search of scientific substantiated forms and scale of industrial human activity providing rational using natural resources and getting necessity useful productions without pernicious influence upon natural environment acquires specific significance [5]. Provisional evaluations show that long-range planning without taking into account environmental effects is impossible. On the other hand available environmental models have specific character and are impossible to reflect complex interconnections but can show consequences of anthropogenic influences. For example, Kuznetsk Basin coal industry influences upon environment essentially. Retrospective analyzing and statistical evaluating show

 
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that intensity of anthropogenic influencing Company “Prokopievskugol” enterprises upon atmosphere is equal to 6 – 7 kg of dust-gas emission per 1 ton of produced coal [6].

## 2. ENVIRONMENTAL DANGER OF WORKED AND LIQUIDATED COAL MINES OPEN AREAS

**Methane emitting in Earth Surface from worked mines.** Generalization of field observation results by gas changing worked mines open areas with atmosphere surface layer shows that specifying this process regularities are necessity for safety of undermining territories [7]. Methane filtration process through undermining rocks to surface layer is realizing as result of excessive pressure of methane containing in coal-bearing series. Mathematical model of vertical isothermal field of methane pressure and methane filtration flow have got follow view [8-9]:

$$p^2(z,t) = p_a^2 + \left(p_0^2 - p_a^2\right)zH^{-1} + 1{,}273\, p_0^2 \sum_{n=0}^{+\infty} (2n+1)^{-1} \exp\left[-(2n+1)^2 \pi^2 \kappa_m H^{-2} t\right] \times$$

$$\times \operatorname{Sin}\left[(2n+1)\pi z H^{-1}\right], \tag{1}$$

$$j_m = 0{,}5\, k_n \left(\mu p_a H\right)^{-1} \left\{\left(p_0^2 - p_a^2\right) + 4\, p_0^2 \exp\left[-9{,}87 \kappa_m H^{-2} t\right]\right\}, \tag{2}$$

where $p$ is pressure of free methane in pores and interstices of undermining rock massif; $\kappa_m$ is methane piezoconductivity of undermining rocks; $z$ is vertical coordinate with reference point locating in Earth Surface; $t$ is time of considering process; $H$ is coal seam depth of occurrence; $p_0$ is initial pressure of methane; $p_a$ is atmospheric pressure; $j_m$ is methane filtration flow to earth surface; $k_n$ is average volume of undermining rocks gas permeability; $\mu$ is dynamic viscosity of methane.

Analyzing dependence (2) is showing that with the course of time the methane filtration flow to Earth surface tends to stationary value:

$$j_\infty = \lim_{t \to \infty} j_M = 0{,}5\, k_n \left(p_0^2 - p_a^2\right)\left(\mu p_a H\right)^{-1}. \tag{3}$$

where $j_\infty$ is stationary value of methane filtration flow to Earth surface.

The dependence (2) visually demonstrates that limited value of the filtration flow is depending from coal seam depth of occurrence, filtration properties and natural gas content of underworking rocks, which influencing upon initial pressure. It is obvious that measures by degassing underworking rocks allow decreasing methane emission into Earth surface. Let’s introduce follow designation: $Fo_f = \kappa_m t H^{-2}$, where $Fo_f$ is filtration Fourier criterion. Then we get follow dependence for forecasting methane emission dynamic into Earth surface on the territories of liquidated mines mining leases:

$$I_{e.s}\left(Fo_f\right) = 0{,}5\left(j_m - j_\infty\right)\mu p_a H \left(F_{e.s} k_n\right)^{-1} p_0^{-2} = \sum_{n=0}^{+\infty} \exp\left[-9{,}87 (2n+1)^2 Fo_f\right], \tag{4}$$

where $I_{e.s}$ is dimensionless value of methane emission into earth surface from underworking rocks; $F_{e.s}$ is underworking earth surface area.

Geotechnological process life is usually very large until unprofitable mine complete liquidation and methane pressure stationary distribution along underworking rocks is formed. Then methane filtration flow will correspond with the formula (3). Consequently, forecasting methane emission at the territory of liquidated mines mining leases reasonably realize with using follow formula

$$I_\infty = 0{,}5 F_{з.п} k_n \left(p_0^2 - p_a^2\right)\left(\mu p_a H\right)^{-1}, \tag{5}$$

where $I_\infty$ is stationary methane emission.

Thus it’s proved that methane emission velocity on earth surface from underworking coal-bearing series is the function of filtration Fourier criterion, which tending to asymptotic value. This value dependences from coal seam depth of occurrence, filtration properties and natural gas content of underworking rocks too.

**Methane emitting in Earth Surface from undermining coal-bearing series after mining.** Methane filtration from undermining rocks takes place as a result of excessive methane pressure, which locating in coal-bearing series. Then with taking into account of one-dimensionality moving gas mixture and some assumptions we can to write follow [7]:

$$P(z,t)=\frac{1}{2\kappa_m\sqrt{\pi t}}\left[p_a^2\int_0^{H_0}F(z,\xi)\,d\xi+\alpha\int_{H_0}^{\infty}\xi F(z,\xi)\,d\xi+K_P^2\int_{H_0}^{\infty}\xi^2F(z,\xi)\,d\xi\right], \tag{6}$$

where $P=p^2$ ; $\alpha=2p_aK_P$; $K_p$ is angular coefficient at the "hydrostatic" low of changing methane pressure in coal-bearing series (in compliance with theory of professor Leonid Bikov);

$$F(z,\xi)=\exp\left[-\frac{(z-\xi)^2}{4\kappa_m^2t}\right]-\exp\left[-\frac{(z+\xi)^2}{4\kappa_m^2t}\right]. \tag{7}$$

Using Darcy low we can write that methane emission velocity into undermining Earth surface $j_{e.s}$ is calculating by formula:

$$j_{e.s}(t)=-\frac{\langle k\rangle}{\mu}\left.\frac{\partial \mathrm{p}}{\partial \mathrm{z}}\right|_{z=0}=-\frac{\langle k\rangle}{2\mu p_a}\left.\frac{\partial P}{\partial \mathrm{z}}\right|_{z=0}, \tag{8}$$

where $\langle k\rangle$ is average value of gas permeability for undermining methane-bearing rocks.

Therefore general gas emission $I_m^{e.s}$ from undermining Earth surface area, which is equal to $F_{e.s}$, can be defined by formula:

$$I_m^{e.s}(Fo_f)=\frac{0.282\langle k\rangle F_{e.s}}{\mu p_aH_0\sqrt{Fo_f}}\left\{p_a^2\left[1-\exp\left(-\frac{0.25}{Fo_f}\right)\right]+2p_aK_p\left[1+\frac{0.564}{\sqrt{Fo_f}}\exp\left(-\frac{0.25}{Fo_f}\right)-erf\left(\frac{0.5}{\sqrt{Fo_f}}\right)\right]+\right.$$

$$\left.+2.26K_P^2H_0\sqrt{Fo_f}\left(1+\frac{0.25}{Fo_f}\right)\exp\left(-\frac{0.25}{Fo_f}\right)\right\}. \tag{9}$$

Calculate experiments were realized for follow functions forming dependence (9):

$$f_1(Fo_f)=\frac{1}{\sqrt{Fo_f}}\left[1-\exp\left(-\frac{0,25}{Fo_f}\right)\right];\ f_2(Fo_f)=\frac{1}{\sqrt{Fo_f}}\left[1+\frac{0,564}{\sqrt{Fo_f}}\exp\left(-\frac{0,25}{Fo_f}\right)-erf\left(\frac{0,5}{\sqrt{Fo_f}}\right)\right];$$

$$f_3(Fo_f)=\left(1+\frac{0,25}{Fo_f}\right)\exp\left(-\frac{0,25}{Fo_f}\right)\Bigg\}.$$

Analysis of calculate experiments results show that dependence (9) has got the asymptote:

$$I_\infty^{e.s}=\lim_{Fo_f\to\infty}I_m^{e.s}(Fo_f).$$

Thus methane emission at Earth surface from undermining coal-bearing series is monotone decreasing function, which tends to asymptotic value $I_\infty^{e.s}$ for large time periods. Consequently steady methane emission at undermining territory will be observed. The methane emission can be calculated by follow formula:

$$I_\infty^{e.s}=\frac{0.637\langle k\rangle F_{e.s}K_P^2}{\mu p_a}. \tag{10}$$

Therefore methane emission velocity from undermining Earth surface will be constant value large of time period. The methane emission velocity will be depending only gas permeability undermining rocks and changing natural gas content with depth. Apparently those measures by degassing undermining rocks will allow reducing the gas emission.

**Filtration blackdamp through undermining rock massif.** It is known that blackdamp is mixture of nitrogen and carbonic acid. Consequently it is partially but sometimes entirely of deoxygenating gas mixture, which is unfit for respiration. Mining air oxygen is absorbed by coal and transformed into carbonic acid in open areas. Filtration of the blackdamp realizes trough mining rocks by decreasing atmospheric pressure.

The equation of filtrating blackdamp in open area is similarly of equation (1), where pressure of blackdamp in open area will be already considered and filtration properties will be characterized by undermining rocks piezoconductivity for blackdamp.

Solving equation of blackdamp filtration for those conditions was gotten in follow view [8]:

$$p_b(z,t)=\sum_{n=1}^{\infty}T_n(t)\ \mathrm{Sin}\left(\frac{\pi \mathrm{z}}{\mathrm{H}}\right), \tag{11}$$

where $\kappa_b$ is undermining rocks piezoconductivity for blackdamp;

$$T_n(t)=(-1)^n\frac{2p_0}{\pi n}+(-1)^n\frac{2\alpha_P}{\pi n}\left\langle t-\left(\frac{H}{\pi n\kappa_b}\right)^2\left\{1-\exp\left[-\left(\frac{\pi n\kappa_b}{H}\right)^2 t\right]\right\}\right\rangle.$$

Then quantity of blackdamp, which is equal to $j_b$ ($m^3/m^2c$), will be arriving at unit of time to perpendicular to filtration flow specific area. Using Darcy low we can get follow formula:

$$j_b=\frac{2\langle k\rangle}{\mu_b H}\left(p_0+\alpha_P t\right), \tag{12}$$

where $\mu_b$ is blackdamp viscosity; $\alpha_P$ is velocity of decreasing atmospheric pressure.

Its follow detecting that structure of formula (12) and results of calculating experiment well agree with qualitative picture of gas emission from open areas. It’s known that gas emission velocity, which conditional on decreasing atmospheric pressure, is proportional of decreasing atmospheric pressure velocity. Proposed dependence (12) a first confirms conclusions about influence of changing atmospheric pressure upon gassing open areas in territories of closed mines and a second can use for solving problems of forecasting gas situations arising in basement floors of buildings and different constructions locating over undermining rocks massifs. Thus created mathematical models of filtration gas changing undermining rocks with Erath surface allow forecasting methane emission from undermining coal-bearing series and blackdamp emission. Forecasting gas changing open areas with Erath surface makes possible evaluating level of environmental protection by gas factor for territories closed mines.

## 3. SCIENTIFIC-PRACTICAL RESULTS OF MONITORING ANTHROPOGENIC INFLUENCE UPON ENVIRONMENT OF MINING TERRITORIES

Practice of life and sensible mentality show, that environmental condition and necessity getting useful properties from it are two basic factors, which influence upon Human life activity. As a rule, we perceive environmental condition like consumers of environmental properties. Effective society and economy are impossible if using natural resources contradict principals of sensible self-restraint and environment deteriorates. It was clear at last decades. Using natural resources is anthropogenic influence upon environment to get materials and energy for own life activity. Scientific tools for studying consequences of anthropogenic influence upon environment are basic part of geoecology. Geoecology is theoretical foundations of environmental (or geoecological) monitoring.

Geoecology is interdisciplinary scientific direction, which connecting researches by composition, constitution, properties, processes for physical and geochemical fields of Earth geospheres as human environment and the environment of other organisms. Basic problems of geoecology are studying variation of life-support resources for geospheres under the influence natural and anthropogenic factors, environmental protection, rational using geospheres and controlling there conditions in order to save productive natural environment for our and future generations.

Basic goals of geoecological monitoring are observations, evaluating and prognosis of environmental condition for studying variations produced of anthropogenic activity.

Basic process of this activity is evaluating influence upon environment. That is why geoecological monitoring is basic part of rational using natural resources. Abstract theorems of geoecological monitoring are based on fundamental physical principals [4]. Detailed analysis of Russian environmental situation has shown [10] that necessary taking into account follow important system principals:

- The territories of industrial regions are complex socio-economical and technical systems.
- Management of environmental situation of territories demands efficient information about consequences of made decisions.

Industrial-technological complex is interconnection open technological and socio-economical subsystems hierarchically connecting with all complex ecological systems by vertical and horizontal levels.

Idea of proposed concept is follow. Authentic evaluation of environmental condition under consideration region bases at the regularities existing between environmental and population life quality indexes, which reflecting socio-economical background of the specified territory.

Solving this problem makes with using follow methods:

- Algorithm is created for territorial structure of region as finite aggregate of territorial subdivisions, which entered into regional composition and where environmental and demographic indexes evenly distributed and depending only time.
- Compartmental systems of mathematical models are created for describing population size, changing age and sexual structure and lifetime with taking into account environmental indexes reflecting anthropogenic influence in even subdivision.
- Computing experiments are organized and made with using government statistical information.
- Legislative acts are created and practical tested for limitation level of anthropogenic influence with using evaluation method of environmental condition by physical-chemical, biological, demographical and epidemiological indexes at the specified territory of mining region.

Evaluating influence upon environment (EIE) is practical base of geoecological monitoring. Evaluating influence upon environment consists of nine parts. EIE involves follow basic phases:

- assembling and analyzing necessary information;
- identifying sources, kinds and objects of influence;
- forecasting condition variation of natural environment;
- evaluating probable emergency situations and there consequences;
- evaluating environmental, social and economical consequences;
- identifying prevention or decreasing negative anthropogenic influences methods and substantiating of methods there controlling;
- realizing environmental-economical evaluation of projects or operating enterprises;
- analyzing and choosing alternative variants for realization under consideration project;
- generating new variants of under consideration project.

Evaluating risk of negative influence is very important phase of environmental monitoring. Algorithm of evaluating risk of negative anthropogenic influence upon environment consists of follow parts. The first is evaluating probability of appearance negative anthropogenic influence to environment, which more then maximal level. The second is evaluating probability of failure at the environmental protection system [6]. The third is evaluating detriment of negative anthropogenic influence upon environment. Conceptual formula has got follow form: $R_{ENV} = P_{ANT} D$, where $R_{ENV}$, $P_{ANT}$ are risk and probability negative anthropogenic influence to environment accordingly; $D$ is detriment of negative anthropogenic influence to environment. This formula can be written in follow form: $R_{ENV} = P\{I_{INF} > MPEL\} P_{GL} D$, where $I_{INF}$ is intensity of negative anthropogenic influence to environment; $MPEL$ is maximal level of influence to environment; $P\{I_{INF} > MPEL\}$ is probability of appearance negative anthropogenic influence to environment, which more than maximal level; $P_{GL}$ is probability of failure at the environmental protection system.

## 4. CONCLUSION

In conclusion we want to say follow. People are major population of the Earth. Our planet is Human ecological niche. Conditions of Human Environment define quality of their life. Improving environmental monitoring system and creating effective technological actions by neutralization wastes are based on adequate mathematical models of environmental condition for a territory. These models have to reflect connection between indexes of public health, influence upon environment and basic demographic characteristics. Consequently, it is environmental rationally and economical reasonably considering atmospheric air as natural resource of mining-industrial region. Therefore optimizing technical-economical indexes by atmospheric factor must be based at principals of regulating by institutes of agreement and consolidating mining enterprises with enterprises of others industrial brunches. Proposed methodical principals of complex evaluating atmosphere of mining-industrial region allow realizing integrated approach to forecasting intensity of polluting atmosphere, economical efficiency of production and atmospheric air condition control with using basic demands of environmental imperative for considered territory.

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# New Progress of Coal Preparation Technology in China

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**Abstract**

This paper introduces three development potentials of the new coal preparation technology: "Steam coal fines dry removal preparation process", "coal cleaning technology by muddy water" and “lignite pelitizing separation technology ". "Steam coal fines dry removal separation process" is used to simplify the steam coal and chemical coal separation process, the output and the heating value are higher by this process than that by the traditional separation process. "Steam coal muddy water washing process" compared to the traditional "coking coal clean water washing process ”can save a lot of valuable water resources, reduce water consumption in circulation, further reduce power consumption. “Lignite pelitization preparation process” will help people change the traditional concept that lignite may not be washed, improve the coal quality and improve the calorific value of lignite as well. The engineering of these new coal preparation processes are now wide spread and applied for the steam coal preparation plants in China.

**Key words**

coal preparation technology steam coal lignite fines dry removal washing with fines removal muddy water processing pelitizing processing

## 1. Introduction

In China, the high ash, high moisture, high sulfur content, low heat value and low metamorphic degree of low-grade coal production capacity is greatly increased.Per Coal resources situation and the state demand on control of the smog weather, the coal processing is substantially increased in the proportion, and the coal preparation process technologies are constantly improved, innovated and advanced. This paper introduces three kinds of new coal preparation technologies with potential of development.

## 2. Coal washing technique with fines dry removal separation

### 2.1 background information

Shenhua group is in Shendong mining area which is one of the main coal producing areas in China, with an annual output of 150 million tons of raw coal. The coal products of weak sticky coal, non sticky coal and other low metamorphic coal are good for power generation and chemical application. In order to further improve the coal recovery rate and meet the requirements of users on the quality of coal, the coal preparation process in the coal preparation plants of Shendong mining area were transformed technologically by adding the fine coal separation system, saying change the original classified separation with "+13mm lump coal washed, and -13mm fine coal not washed" - part of the processing into whole washing processing of the "+13mm of lump coal and -13mm fines". The change of coal preparation technology brings forth effective improvement of the calorific value of coal products. However it results in a large number of coal slime that is difficult to deal with the slurry of high moisture content and low calorific value.

 
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slurry of high moisture content and low calorific value.

Dadi Engineering Development Group Co., Ltd. and China University of Mining and Technology jointly research and propose the comprehensive and integrated solutions in utilization of coal slime in Shendong mining area. One of the most important and simple measure is as recommended: -6mm or -3mm coal is not washed, raw coal "dry removal separation" is used. To improve the coal preparation process can greatly reduce the amount of slime [1].

### 2.2 Theory basis

The theoretical basis of the "Coal washing technique with fines separation" mainly relies on three points:

(1) Coal characteristic: +6mm or +3mm raw coal generally high ash content, low calorific value, if not washed it does not meet market requirements; -6mm or 3mm raw coal is generally low ash, high heat value, it even without washing can also meet the requirements of the market.

(2) The heat value variation of the coal before and after washing: coal all washed, +6mm or +3mm product ash content is reduced, heat value greatly increased. -6mm or -3mm products get low ash content, but water content increased (especially -0.5mm slime water up to 30%), resulting in calorific value decreasing.

(3) Reliability and efficiency of coal fines dry separation equipment: production practice proves that satisfactory results can be achieved by using the flip flow screen for coal size of 6 mm or 3 mm; 6mm screening efficiency is 60-80%; 3mm screening efficiency 50-70%. From the engineering point of view, even though the 3mm screening efficiency is only 40%, the technical and economic benefits are still feasible. In addition, the wind blowing equipment is also promising as fine classification equipment in dry processing.

### 2.3 Project Case

The design capacity of No. 2 coal preparation plant is 10 million per annum. It was put into operation in 2014. According to the coal washing test and result analysis per table 1, a dry processing for 3mm cut design is adopted. (May change to 6mm cut according to the changes of coal properties), 200-13mm coal is processed by heavy medium scraper separator, 13-3 (6) mm with heavy medium cyclone. The product heat value is greater than 19.23MJ/kg (4600kcal/kg) for the power generation. Among them, 3mm or 6mm separation equipment may use the flip flow screen made by China (Tianjin) Aury industrial technology limited [2][3].

It is proved through over a year practice that the plant's technical and economical benefits are significantly better than other coal preparation plants in the surrounding areas either with the "coal all washing technique" and "lump coal washed, fines not washed coal separation technology". Compared with the "Whole coal washing process", the all 3mm fines processing technique has four major advantages:

(1) Less work load in the end coal separation system, equipment size is reduced and/or the number of equipments is reduced;

(2) The amount of slime is significantly reduced, so the size of the slime water treatment equipment significantly reduced;

(3) Capital investment and production costs are significantly reduced;

(4) The coal slime volume is small so it is easy to sell by mixing into the steam coal products.

**Table 1. Raw Coal Washed Result Analysis**

| Size mm | Before Washed | | | | After Washed | | | | Added |
|---|---|---|---|---|---|---|---|---|---|
| | γ % | $A_d$ % | $M_t$ % | Qnet,ar Kcal/kg | γ % | $A_d$ % | $M_t$ % | $Q_{net,ar}$ Kcal/kg | $Q_{net,ar}$ Kcal/kg |
| 200-100 | 7.60 | 26.33 | 733 | 4033 | 5.72 | 12.38 | 18.55 | 4895 | 862 |
| 100-50 | 11.63 | 25.55 | 594 | 4061 | 8.99 | 12.11 | 19.00 | 4879 | 818 |
| 50-25 | 16.32 | 25.43 | 19.50 | 4047 | 12.44 | 12.56 | 19.45 | 4835 | 788 |
| 25-13 | 14.61 | 24.10 | 19.85 | 4109 | 11.15 | 11.71 | 19.90 | 4842 | 733 |
| 13-6 | 16.57 | 22.26 | 20.20 | 4203 | 12.79 | 12.16 | 20.35 | 4797 | 594 |
| 6-3 | 14.25 | 21.54 | 20.55 | 4227 | 10.61 | 10.23 | 21.70 | 4795 | 568 |
| 3-0 | 19.01 | 22.75 | 21.25 | 4109 | 21.05 | 16.93 | 31.50 | 3852 | -257 |
| Total | 100.00 | 23.73 | | 4105 | 82.75 | 13.13 | | 4584 | 479 |

## 3 muddy water washing process

### 3.1 background information

The traditional coal preparation process originated from metallurgical demand of coking coal washing.

In low density separation, the raw coal washability is generally difficult. Coking coal products requires not only low ash ($A_d$<12.5%), but it also requires the less difference in ash from grades of coal. Ash increased or decreased by 0.5%, the sales price is in relatively big difference. Therefore, high degree of emphasis on separation efficiency and strict control of the mismatch are well noted in coking coal preparation process. The "Clean water washing" can make sure that the pollutant in high ash coal will not damage the clean coal products quality so as to avoid affecting the grade and price.

In the high dense separation, the coal washability is relatively easy. The steam coal has higher ash content ($A_d$>13%), the big difference of the heat value is among the grades, and the heating value is the base for pricing. Calorific value increased or decreased by 100Kcal/kg, bears no big difference in sales price. The system is simple, low cost, even if there is clean coal polluted by "muddy water washing", it will not affect the sale price basically.

### 3.2 Theoretical basis

Generally, the solid content in coal circulating water below 50g/l belongs to "Clean water washing ". If it is higher than 100g/l, it basically "muddy water washing ".

Previous preparation researches show that when using water as the medium to low density jigging separation, the solid content in the circulating water is increased from 50g / L to 200g/l, circulating water viscosity from the 0.0010N.s/m$^2$ to 0.0012N.s/m$^2$. The down size limit of separation is increased from 0.5 mm to 3 mm and coal ash in 0.5 ~ 1mm grade fines from 9.5% to 13.5% [4].

The above data show that, increasing the solid content in the circulating water (i.e. "muddy water washing") has obvious influence on the separation for coking coal. However from other perspective, although the coal washing with muddy water jig sorting has impact on the down size limit and the ash content in the fines, its adverse effects are only on –3mm fines, there is no adverse impact on the coarse size (+3mm) of coking coal and steam coal in the preparation process.

When the solid content is increased in the circulating water, the above size at the dewatering screen sieve will get more amount of slime, which results in high ash content on the above size material. As long as reasonable measure is taken to water spray over the screens that help eliminate the influence of slime pollution, and control ash content in the clean coal effectively.

The production practice in the thermo coal preparation plant also proved that it is feasible both technologically and the economically to use the muddy water washing process in the steam coal preparation.

### 3.3 Project case

The original design capacity of Daliuta Coal Preparation Plant is 12.00Mt/a. The process technology is Jigging Washing for 50~13mm coal , -13mm coal is not washed; the design washing capacity for +13mm coal is 6.00Mt/a, by using 2 sets of 30m diameter thickener for coal slurry treatment, thickeners' overflow is enclosed circuit in coal preparation plant.

After technological modification, the actual production capacity of the coal preparation plant has reached 34.00Mt/a. 200~1.5mm coal is processed by the heavy medium processing (part of Jigging Washing). -1.5mm coal is not washed. The +13mm coal washing system is divided into two parts: the HMS new system design and the original design of the jig separation system. The Part of the original design for the jig separation has no substantial changes, so it is comparable. The original design capacity is 6.00Mt/a by the jig process system while 8.00Mt/a. for the new design. The actual density of circulating water is increased by more than 1 / 3 from the original design. Although "muddy water washing" is used, there is no side effect either in raw coal separation or on the quality of the products [5][6].

## 4 Lignite pelitizing separation processing

### 4.1 background Information

The characteristics of lignite in China:

(1) High ash, $A_d$=20%~40%;

(2) High moisture, $M_{ad}$=10%~20%, $M_t$=30%~55%;

(3) Low heat value, $Q_{net,\ ar}$=6.9~14.6MJ/kg (1650~3500Kcal/kg).

The traditional concept for long time is: Lignite prone to mud, not suitable to water washing. Therefore, people tend to study on how to reduce the moisture content of the lignite, other than to research on how to reduce the ash content of the lignite.

### 4.2 Theoretical basis

By studying the characteristics of the hardness of the coal particles and the mudding characteristics, the theory and method of the "Pelitizing separation" of the lignite are put forward in this research [7][8].

The major theoretical basis are as below:

(1) The macro particle size characteristics of the lignite: the hardness and mechanical strength of the coal particles are much greater than those of the gangue particles. The macro results of the group behavior show that: The smaller of the raw coal size, the higher of the ash content, while the bigger of the raw coal size the lower of the ash content.

(2) The micro particle size characteristics of lignite: in the coal, the coal particles and the gangue particles have obvious differences in the degree of pelitization. Coal particles are easy to be fractured,

and the gangue particles are easy to mudding. The results of the performance of the group are:

Typical lignite pelitization test results are shown in Table 2. After 30 minutes of raw coal in the water, the +13mm large grain size is still dominated by the grain size, even if the coal particles are reduced into fine particles, yet it remains the particle state in millimeter size, it will basically not become "silt" in micron meters. Reject is flipped in the water for 30 minutes, -0.5mm size becomes dominant, easy to become "mud" shape in micron meters. In particular, the -0.045mm particles are basically gangue enrichment.

**Table 2. Raw Coal & Reject Slime Test Results of Lignite**

| Raw Coal Yield,% | | | | | | Reject Yield,% | | | |
|---|---|---|---|---|---|---|---|---|---|
| +13 mm | 13-0.5 mm | 0.5-0.045 mm | -0.045 mm | Total | Ash,% <45μm | +500 μm | 500 -10 μm | -10 μm | Total |
| 47.66 | 12.15 | 10.63 | 29.56 | 100.00 | 81.25 | 15.73 | 60.69 | 23.58 | 100.00 |
| Note: Coal samples for the slime test were taken from a large sample of screened raw lignite13-100mm。The samples were taken from the lignite reject,sieved 6-3mm。 | | | | | | | | | |

The different granularity characteristics and the characteristics of the pelitization of the lignite coal and gangue laid the theoretical foundation for the water washing separation of the lignite. The different characteristics between coal and gangue in the petilization reveal that the brown coal with high ash content is in the process of the petilization, in fact, it is the separation of coal ash separation process. When Lignite is fully pelitized, coarse grains are low ash coal, coal slime after sedimentation is high ash gangue. This process constitutes a new theory and new method “Pelitization separation” for the separation of brown coal.

Based on the theory of "Pelitization separation", we don't have to worry about the effect on mineral pelitized may impact the mineral separation process. Further, the different characteristics of pelitization can be actively utilized, the pelitization process cannot be over emphasized. Make the easy mudding materials to be fully pelitized in the “Pelitization Processing”. Positive results may be achieved as long as we choose the suitable pelitization process and the proper sizes in the separation for the easy mudding mineral (not limited to lignite).

“Pelitization separation” mainly uses the friction among particles to realize the full pelitization, and the process needs less water, and “clean water" is not required, that is a benefit to the closed circulation system, especially suitable for dry locations of the lignite areas.

**4.3 Project case**

Inner Mongolia lignite demonstration project using the typical "lignite pelitization process" .The Lignite mine belongs to high ash content ($A_d$ 38.41%), high moisture ($M_{ar}$ 27.8%), high sulfur content ($S_{t,d}$ 2.62%), low calorific value ($Q_{net,ar}$ 10.77MJ/kg) lignite ($P_M$ 31%), is typical of the high ash, high moisture and easy mudding lignite. Lignite washing results show that petilization separation process in the technology is feasible, after washing the heat value of the product can be increased by more than 1000Kcal/kg, coal slurry clarification treatment meets the production requirements in the industry[9][10].

The lignite pelitizing separation process includes the following four steps:

Scrubbing (Petilizing): the raw coal particles are rubbed one another in a small amount of water to moderately dissociate and fully mudding, to get the low ash particles, middle and high ash fines.

Washing (desliming): after the full pelitization of the coal which will be sieved through the vibration sieve with overhead water spray to remove the high ash refuse; high ash fine size is 0.5mm or 0.3mm.

Dehydration: the granular coal product after the vibrating screen is going through cyclone to remove the surface water. The final surface moisture of lignite product can be controlled at 12%.

Thickener: the overflow of slime water sedimentation tank is enclosed circuit; the slime after settling is high ash content with low calorific value, as tailings to be discharged for comprehensive utilization (such as raw materials for construction). If the amount of the additives, the inorganic coagulant and the high polymer flocculent to be added can further strengthen the coal slurry concentration, precipitation and clarification process.

## 5. Conclusion

"Steam coal fines dry removal separation process" is used to simplify the steam coal and chemical coal separation process, the output and the heating value are higher by this process than that by the traditional separation process. "Steam coal muddy water washing process" compared to the traditional "coking coal clean water washing process ”can save a lot of valuable water resources, reduce water consumption in circulation, further reduce power consumption. “Lignite pelitization preparation process” will help people change the traditional concept that lignite may not be washed, improve the coal quality and improve the calorific value of lignite as well. The engineering of these new coal preparation processes are now wide spread and applied for the steam coal preparation plants in China.

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**Study on Multi-model roof safety warning based on decision fusion**

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**Abstract:** Analyzing the currently existing problems of pressure manifestation prediction in our country in the case of deep mining, the increasingly intensified roof pressure, the deterioration of roadway maintenance and the increased frequency of coal mine accidents, putting forwards a method to predict roof pressure based on multi-model fusion. First, the monitoring data from each sensor is mode decomposed respectively through ensemble empirical mode decomposition(EEMD), getting multiple mode function sequence(IFM) of each sub-model. Then the prediction on the nonlinear sequence is conducted via support vector machine(SVM) according to the characteristics of IMF. While for the linear parts, the trend of the sequence, the auto-regressive integrated moving average model(ARIMA) is applied to conduct the prediction. According to the former two steps, the prediction output of each sub-model results from the reconstruction of the respective predictive values which are processed by normalized decision fusion through statistical recognition model and then represented in the same coordinate system to study on dynamic law of roof movement and improve its predictive ability. The practical application shows that this method can reflect the law of large deformation of dynamic pressure change and capture the roof disaster warning information and meet the needs of production safety.
**Keywords:** ground pressure; EEMD; SVM；ARIMA; statistical recognition model; prediction；statistical recognition model

## 1 Introduction

With the transfer to deep mine exploitation and the influence by abutment pressure on the working surface, the roadway surrounding rock stress increases, the mine pressure behavior intensifies, looseness range expands and the roadway deformation increases, especially in the area with partings along the fault near empty roadway, which bring huge difficulties to roadway control and have serious impact on normal advance of the working surface. There is a necessity to study in-depth on the dynamic pressure of large deformation regularity of the roadway in the case of the situation that the continued large deformation of special thick coal seam roadway influenced by mining is hard to control[1].Therefore, the establishment of the mine roof safety warning model has great significance on the status quo to change coal mine production safety.

At present, many large coal companies in China have also established a series of mine roof model and disaster warning system, but the effect is not satisfactory. That is mainly because of the following four reasons[2-5]:First, the ways to monitort the mine roof are diverse, such as electromagnetic radiation monitoring, micro vibration monitoring, roof abscission layer monitoring ,the roof pressure monitoring, which have reflected the mine roof safety status from different perspectives. However, the monitoring system lacks of early warning information fusion and is unable to provide the comprehensive decision-making, which will cause false alarms, missed alarms and is unable to evaluate roof safety conditions especially when the predicting results are not uniform; Second, the current mine roof monitoring is only displayed as monitoring charts, the achievements of study on rock strata movement and ground control is limited to experience and the monitoring systems haven’t reached a high intelligent degree, which is unable to provide a reliable scientific basis for roadway support and selection of rock



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stress parameters; Third, the researches on multi-sensor information fusion models and algorithms have received fruitful results, but lots of which cannot reflect the actual needs in industrial production; Fourth, affected by the production process, on-site operation and other complex factors, the single model of software measurement method cannot reflect the dynamic information and global characteristics of coal production process, which makes the prediction have a poor adaptability, and cannot be used for long-term.

In order to solve the above problem, a comprehensive decision early warning model based on stope roof of roadway surrounding rock and multi-factor of multi-model fusion is proposed on the foundation of the practical mine pressure theory. The prediction and early warning of the roof through the prediction on changes in the mining overburden movement and pressure distribution based on the monitoring data from multiple sensor fusion.

## 2 Information integration of roof prediction model

There are two main mine roof disasters: stope roof disaster and roadway surrounding rock disaster, thus roof safety warning can be realized by monitoring the stope roof and the roadway. The support resistance of mechanized mining face, including real-time pressure of anterior column, posterior column and anterior ground beam, provides the basis for reasonable support-surrounding rock relationship study and roof design and management. Roadway condition monitoring is realised through separation instrument and advanced pressure sensor. The separation instrument mainly observes the abscission of the rock adjacent to the stope roof and provides the basis for the prediction on abscission layer caving conditions of immediate roof, main roof and Overburden movement condition. Advanced pressure sensor mainly monitors the relatively stable and significant movement whole process of roof strata and the changing process of support pressure in the internal and external stress field, which provides the basis for decision-making on support pressure distribution and transfer changes.Mine overburden movement and pressure distribution are predicted through software measurement method based on decision fusion of the various strata pressure information from each sensor. While, with the the change of stope in time and space propulsion, the single model of software measurement method cannot reflect the dynamic information and global characteristics of coal production process affected by the production process, on-site operation and other complex factors. Software measurement method[6] based on decision fusion can significantly improve the prediction and generalization ability of the model by adding a few models and decision fusion.

Multi-model modeling generally involves three steps[7]: data set dividing, sub-models' establishing and multi-model output. Figure 1 shows the structure of the roof safety prediction model based on multi-model fusion. The training sample monitoring data of support resistance of fully mechanized working face, advanced pressure sensors, roof abscission layer sensor are respectively X1， X2 and X3, which forms three sub-models: support pressure sub-model, advanced pressure sub-model and roof separation sub-model. The analysis and forecasting process is as follows:First, for signal interference suppression and noise pollution considerations,surrounding rock stress monitoring data, the training data from each sub-model, is mode decomposed through ensemble empirical mode decomposition (EEMD)[8], getting several intrinsic mode components (IMF) respectively and a trend item. Second, the paper predicts the nonlinear sequence via support vector machine (SVM))[9-11] according to the characteristics of IMF. While for the linear data, which can be approximated as linear stationary time series, the paper applies auto-regressive integrated moving average model(ARIMA)[12]to conduct the prediction. Then According to the former two steps, the corresponding prediction values Y1，Y2，Y3 result from the reconstruction of the respective sub-models' predictive values. Finally, predictive values of three kinds of sensors are processed by normalized decision fusion through statistical recognition model and then represented in the same time and space coordinate system to study on dynamic law of roof movement and analyze statistical and forecasting recognition results, which provides data judgment basis for the establishment of mine pressure expertise system and effective predictive data for roof safety monitoring and control system.

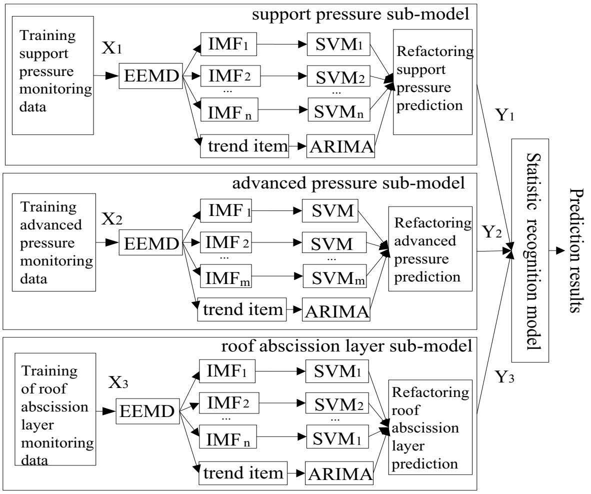


Figure1 predictive model structure

## 3 Case Study

Take Shendong Coal Group Sahara 12101-2 mechanized coal mine working face for case study. The advance length of the working face is 100 m, the length of the working face is 280m. The average coal thickness is 1.6m, the designed mining height is 1.8m and there is 0.01～0.15m thick at the bottom of the coal seam.The seam floor is fine sandstone whose thickness is 5.98～8.14m. The direct roof is siltstone whose thickness is 1.2～5.18m. Working face advances to 686m and enters one coal pillar section. Working face advances to 706m and enters the below in 12101 auxiliary circular lane. Working face advances to 736m and exits one coal pillar section and enters the bottom row in 12101 gangue lane. The roof is in good condition, coal rock fissures and there's no watering phenomenon. The first 60 brackets locate below the working face. The roof historical data is collected from monitoring data hydraulic support 25 measuring points. Taking a weighted average of the initial resistance and maximum resistance as training resistance data p.

The data is from 8:00 on April 19 to 8:00 on April 30 in 2013, the cumulative advance of the working face is 100m. Actual, there are totally 8 pressure cycles during the recovery. In view of the face advance by meters is more realistic in mine work, every 1m is a sampling point, i.e, sampling 100 experimental samples, the first 90 points is seen as the experimental sample, comprising six pressure cycles. The last 10 points is viewed as the output samples. The specific modeling steps are as follows:

(1) First, EEMD adaptively decomposes inherent fluctuations of different frequencies into different components of the IMF according to mine pressure signal itself. The number of the IMFs is related to the

signal itself. The amplitude coefficient of the added white gaussian noise is 0.2 during the decomposition. The support pressure monitoring signal is decomposed into five orthogonal signal components (IMF1, IMF2, ..., MF5) and a trend item, as well as the advanced pressure monitoring signal and the roof abscission layer monitoring signal.

(2).Then, the component of the IMF of the three predictive values is studied by SVM, according to IMF component of its own characteristics, and then the model parameters are determined. Select the appropriate SVM model fuctions and its parameters according to roof pressure data complexity. The IMF1, IMF2, IMF3 in support pressure sub-model,in advanced pressure sub-model, in roof abscission layer sub-model have a large fluctuation frequency and a high complexity, the paper applies radial kernel function to conduct the prediction. The IMF4、IMF5 in support pressure sub-model,in advanced pressure sub-model, in roof abscission layer sub-model are relatively stable, and belong to the low-frequency component, the paper applies polynomial kernel function to conduct the prediction. All items with clear linear trend characteristics, the paper applies ARIMA to conduct the prediction. Figure 2 shows the error between the predictive values and the 10 actual experimental data of the three different test methods of the three sub-models. It can be seen from the three figures that the predictive values in the prediction model accord with the actual values when the working face advances 10m. The predictive results is completely consistent with the actual meters the pressure cycle advances.

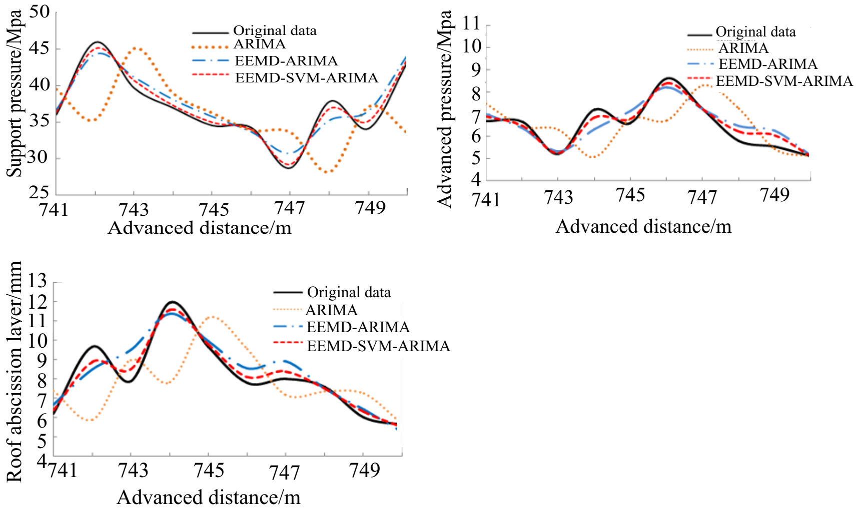


Figure 2 the error between predictive values and actual values of the three different models

In order to verify the effectiveness of the proposed algorithm, Table 1 shows the root mean square errors (RMSE) and mean percentage errors (MAPE) between SVM prediction model, ARIMA prediction model, EEMD-SVM prediction model and the actual pressure data. Error indicator of the three models is shown in Table 1. As can be seen from the comparison chart, the proposed multi-model soft measurement method based on EEMD-SVM- ARIMA has improved the prediction accuracy significantly, especially in the most fluctuating positions of the signal. As can be seen from Table 1, the root mean square errors (RMSE) and mean percentage errors (MAPE) of EEMD-SVM-ARIMA forecasting method are smaller than other measurement model, especially the percentage error is significantly less than ARIMA, EEMD-ARIMA method, which shows the whole sequence prediction is close to the actual data on the highest level. Thus, the prediction model presented in this paper has been greatly improved in terms of performance, the model has a better prediction accuracy.

Table 1 Error indicator of the three models

| Prediction model | RMSE | | | MAPE | | |
|---|---|---|---|---|---|---|
| | ARIMA | EEMD-ARIMA | EEMD-SVM-ARIMA | ARIMA | EEMD-ARIMA | EEMD-SVM-ARIMA |
| Support pressure | 4.61 | 4.55 | 4.41 | 0.131 | 0.041 | 0.016 |
| Advanced pressure | 4.37 | 4.37 | 4.12 | 0.163 | 0.078 | 0.056 |
| roof abscission layer | 4.96 | 4.87 | 4.32 | 0.129 | 0.071 | 0.014 |

The support pressure predictive value, the advanced pressure predictive value and the roof abscission layer predictive value *Y1*，*Y2*，*Y3* are gotten through each sub-model respectively. Establish the roof pressure prediction and management system according to roof dynamic system integration principle and law of mine pressure appearance. The normalization information curve is shown in Figure 3 by decision fusion. At the same time and space coordinate system, we can see that the predicted peak of abscission layer appears in 744m where is the support pressure peak of the sress field, indicating that the basic roof beams will be transferred from relatively stable condition to end fracture and will have significant movement. The advanced pressure peak appears in 746m, the top beam turns into the significant movement. The support pressure peak appear in 742m and 750m, after the a cycle pressure, the basic roof rock beam has a significant movement, the working face endures another cycle pressure. The peak of abscission layer is 6m away from the peak of support pressure, which can be used as a long-term basis to the pressure. Measures to cope with the pressure should be taken while the working face advance is accelerating. The peak of advanced pressure is 4m away from the peak of support pressure, which can be used as a short-term basis to the pressure. Measures to cope with basic roof rock beam pressure should be taken, such as processing of coal wall rib spalling, reducing the scope of overhead roof, ensuring support resistance. When the face advancing to 750 meters, indicating that the pressure cycle's coming, we must strengthen and perfect the ground beam support near the coal wall to prevent local roof caving, machinery and life accident. The predictive results is completely consistent with the actual meters the pressure cycle advances. The normalized graph is shown in Figure 3.

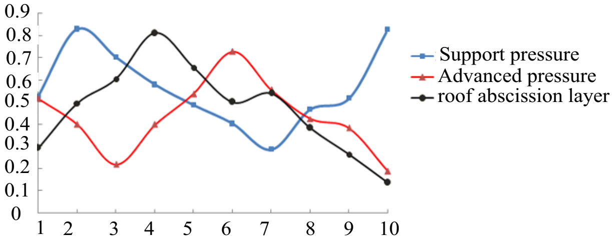


Figure3 The normalized graph

## 4 Conclusion

Because of the nonlinear, non-stationary randomness of roof pressure signal itself, single model modeling needs to consider all the pressure training sample, which makes the accuracy of the model unsatisfactory. Although the traditional multi-model approach to predict the pressure divided training data in modeling and established the corresponding sub-models, the consideration layed on interval unit for pressure monitoring data sequence and the influence of multi-model affected by clustering division accuracy is quite limited, making the prediction inefficient. Multi-model soft measurement method presented in this paper decomposes the nonlinear, non-stationary random signal and gets various intrinsic mode components of the respective sub-model. Then the paper predicts the nonlinear sequence IFM via support vector machine (SVM) and applies auto-regressive integrated moving average model(ARIMA) to predict the trend item of each sub-model. The prediction output of each sub-model results from the reconstruction of the respective predictive values which are processed by normalized decision fusion through statistical recognition model. Finally, the measured data from Shendong Coal Group Sahara Coal Mine verifies the forecasting accuracy, the practicality and the promotion of universal significance of the model.

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# EXPERIENCE OF WAREHOUSING OF WATERY AND JELLIED WASTE OF COAL PREPARATION

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**ANOTATION**

Article contains results of the analysis of the existing ways and technological solutions of joint warehousing of waste of coal preparation factories: watery (slurry), jellied slurry and firm (gravitational) in the way of formation of external dumps.

The reasons of the knitting properties of watery waste of coal preparation and possibility of their interaction with a surface of the waste stored in rock dumps are established. These properties of joint types of waste allows coal preparation factories to provide work without operation of external slurry settlers for a turn of the clarified technical water.

Using properties of an optimum ratio of participation of different types of waste at mutually dense formation circles of storages when forming dumps becomes possible to solve a problem of stability of slopes in time. Change of activity puzzolanovy is considered at change of chemical and mineralogical structure of rock. It is established that the general for rock of coal preparation factories is availability of active alumina in the form of radicals of the dehydrated clay minerals or in the form of active alumina, silicon dioxide and ferruterous connections, are characterized by high sorption ability. These conclusions are confirmed by justification of indicators: basicity modules, aluminous, silicate and coefficient of quality.

**Keywords: the coal preparation, watery waste cementing properties, a rock dump, technology of warehousing, viscous mass, filter presses, hydraulic activity**

## 1. INTRODUCTION

When developing projects of coal preparation factories the most technical and economic tasks are: storages of watery waste and creation of rock dumps not inclined to self-ignition. The first problem was solved, generally by means of disposal effluent, the second – system of the dumping of rock dumps including material consolidation, creation of fire-prevention layers, an irrigation, etc.

Now the solution of the first task by means of application of disposal effluent as difficult hydraulic engineering constructions is practically excluded for the following reasons:

- land branch is necessary that in most cases it is forbidden at the state level;
- considerable costs of design, construction, operation and recultivation;
- difficulties with operation during the winter period of time;



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- existence of technical and technological ways of short circuit of water and slurry schemes of coal preparation factories without application of disposal effluent.

## 2. PRACTICAL EXPERIENCE OF JOINT WAREHOUSING OF WASTE OF VARIOUS TYPE

Recently short circuit of water and slurry schemes of coal preparation factories is carried out by means of filter presses [8], and, first of all, the chamber. Experience of such technologies on COF "Chumakovskaya", COF "Kievskaya" and UPC № 2 of Avdeevskiy KHZ showed that the deposit received in the form of cake layers with moisture from 22% easily is transported independently by motor transport on a rock dump and there is stored together with rock. Thus receive clear technical water with the contents firm to 5 g/l [5].

At application on this operation of tape filter presses moisture of a deposit is unstable and fluctuates within 30-40% (COF "Svyato-Varvarinskaya", COF "Octyabrskaya", COF "Pavlogradskaya", JSC Mospinskoye UPP) that predetermines need of its transportation on a rock dump only together with rock of gravitation. Thus receive dirty technical water with the contents firm to 20 g/l and more.

The way of warehousing of tails of flotation together with shredded large rock which is applied by concentrating factory at Yorkshire Meyn mine (England) [1] is known. When crushing large rock the opened again formed surface of grains with great value of surface area which plays a moisture absorber role due to puzzolanovy activity becomes more active. At a ratio of shredded rock in 45 t/h and tails of flotation condensed to 350 g/l that corresponds to a ratio in 74 kg of tails of flotation on 1 t of rock. The received mechanical mix quite satisfied to norms on moisture at joint storages. Also on COF "Uglegorsk" with shredded rock mixed waste of concentration tables on which slimes [2] were enriched. On 10 t of shredded rock fineness of +25 mm and moisture from 7-8% dosed 0,6-0,7 $m^3$ of water, $1m^3$ the thin Ils condensed to 450 g/l, or 1,4 $m^3$ of Ils condensed to 900 g/l. Moisture of mix in this case makes 14-16%.

## 3. HARAKTERISTITKA OF WASTE-GENERATING MINE DEAD ROCKS

The most characteristic components of dead mine rocks are soapstones and aleurolites, and in a smaller measure sandstones, limestones and clay slates [3, 5].

Soapstone (from Greek argillos — clay and lithos — a stone), the sedimentary rock formed as a result of consolidation, dehydration and cementation of clays; differs from the last in the bigger hardness and inability to razmokat in water. Aleurolite (Greek aleuron — flour and lithos — a stone) — solid sedimentary rock. It is formed of an alevrit in the course of lithification. Consists of grains of the wrong form, from 0,01 — 0,1 mm in size (on other these 0,005 — 0,05 mm). There are three kinds of aleurolites in a form of the composing particles: coarse-grained, raznozernisty and fine-grained. A basis of aleurolite is quartz, there can be also particles of feldspar, clay minerals, sometimes there are carbonates and hydroxides of iron.

Metamorphosed soapstone's, alevrita and sandstones possess high density and, as a rule, difficult soak in water. They can be carried to low-plastic or nonplastic clay raw materials.

In comparison with clays soapstones possess higher durability which makes 2–4 MPas at a natural bedding. Alevrita in comparison with soapstones have more coarse-grained structure.

Unlike dump rocks of coal mines waste of coal preparation is characterized by higher content of coal, stabler material structure, the smaller content of sandstones and high content of soapstones, increase in the contents is gray also reduction of mechanical durability [6].

Products of the empty rocks accompanying fields of coals are gliyezhi-clay and argillo-arenaceous rocks are.

The analysis of components of the stored furnace charge defines existence of the following main rocks:

— in large rock aleurolites at the small content of carbonaceous substances are presented mainly;

— in a fine earth the main rocks are alevrita and aleurolites, partially, carbonaceous slates.

Table 1.

Typical chemical composition of waste of coal preparation

| Compound | Joint gravitation rock (ash-content $A^d$ = 85%) | Cake (viscose mass) (ash-content $A^d$ = 65%) |
|---|---|---|
| | Content, % («atmospheric dry» condition) | |
| $SiO_2$ | 40-50 (mean 45,6) | 30-45 (mean 39,3) |
| $Al_2O_3$ | 8-20 (mean 17,3) | 5-17 (mean 12,6) |
| $Fe_2O_3$ | 4-12 (mean 8,5) | 2-10 (mean 5,6) |
| MgO | 1,5-3 (mean 2,4) | 1,5-3 (mean 2,6) |
| CaO | 0,5-2 (mean 1,4) | 0,5-2 (mean 2,2) |
| $K_2O$ | 1-3 (mean 1,4) | 1-3 (mean 1,8) |
| $Na_2O$ | <1 | <1 |
| $TiO_2$ | <1 | <1 |

The chemical and mineralogical structure of rocks is various, however the general for them is availability of active alumina in the form of radicals of the dehydrated clay minerals or in the form of active alumina, silicon dioxide and ferruterous connections, are characterized by high sorption ability.

The X-ray phase analysis of rock of coal preparation factory confirms the statement about multiphase of connections. In a large number there are $SiO_2$ at a look α-кварца and hydroxides of aluminosilicates of iron and aluminum of variable structure of family of chlorites – (Mg, Fe, Al) $_x$ (Al, Si) $_yO_z(OH)_w$, kaolinites - $Al_2Si_2O_5(OH)_4$ with $TiO_2$ impurity, magnesian shamozit - $Fe_2O_3$, $K_2O$, (Fe, Mg) $_5$ (Fe, Al) $(OH)_8$ ($Si_3$ (Si, Al) $O_{10}$). At much smaller quantities there is one of kinds of white mica – $KAl_2(OH)_2$ [$AlSi_3O_{10}$]. Also at small quantities there are $FeS_2$ pyrites, and in very small – $Fe_2SiO_4$ and $Mg_2SiO_4$.

## 4. THE CHARACTERISTIC OF GOODNESS OF WASTE OF COAL PREPARATION WHEN WAREHOUSING

For an assessment of goodness of waste of coal preparation, on the basis of data of the chemical analysis, use the following indicators [4]: module of basicity (Mo), aluminous (Mg), silicate module (Ms), coefficient of quality (Кк). Value of silicate and aluminous modules is the main indicators of the cementing activity, for empty mine rocks these values are in limits: Ms = 1,5-2,5 and Mg = 1-2 that points to the knitting properties.

Hydraulic activity is estimated by quality coefficient. Hydraulic activity of mine rocks is in average values therefore these materials possess the average knitting properties.

Table 2.

Indicators of quality of a chemical composition

| Sample | Index | | | |
|---|---|---|---|---|
| | Mo | Мг | Мс | Кк |
| Joint gravitation sample | 0,10 | 2,04 | 1,77 | 0,45 |
| Cake (viscous mass) | 0,15 | 2,25 | 2,16 | 0,43 |
| Mean | 0,14 | 2,22 | 2,10 | 0,43 |

In general, on all structure of technological test of waste of coal preparation the ratio of three main lithologic groups of rocks is shown: alevroargillitovy, silicon dioxide - the containing and clay and carbonaceous slates.

## 5. PRACTICE OF WAREHOUSING OF JELLIED WASTE OF COAL PREPARATION

Application the filter - press department of external moisture from fine waste of coal preparation in the present is dictated by restriction of possibility of dumping of the last in external disposal effluent. The filter - press department is the final product a fine product, moisture of 35-40%.

The technology of formation of a rock dump is provided with actions for the prevention of self-ignition of dump mass.

The technology of formation of a rock dump with actions against self-ignition represents the consecutive performance of operations on formation of a circle directed on creation of dense air-tight beds.

Stability of slopes of a dump is checked by calculation.

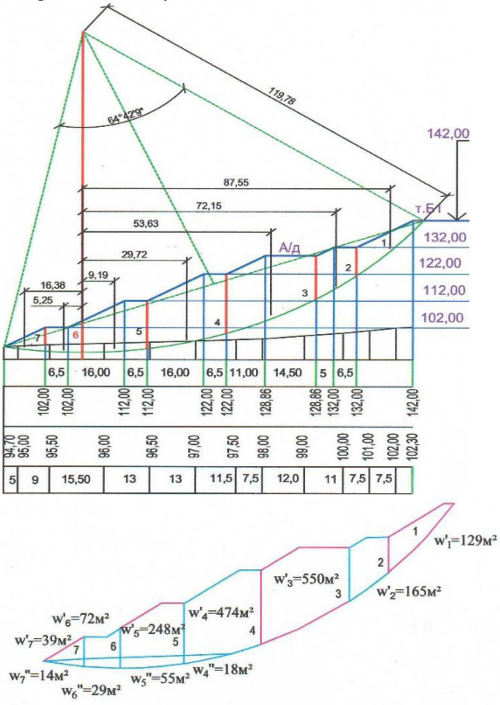


Fig. 1 Graphic display of model of calculation of stability of slopes of a dump

The formation of a rock dump directed on creation of dense air-tight beds represents consecutive performance of a number of operations on creation of the condensed air-tight, isolating, fireproof layers, formation of a circle 10,0 m high from five fireproof layers by thickness 2,0m with isolation of a contour of a circle the isolating soil with a width of a strip of isolation of 13,0 m.

The recommended time of isolation of slopes of the created fireproof layer has to be no more 33 days.

One of conditions of joint warehousing of the dehydrated slime of press filters, the forecast of warehousing of rock of COF "Pavlograd" executed by GVUZ "Donetsk National Technical University" of NICh "Uglekhimchesky Laboratory" Mineral processing chair [9, 10] and the report on the carried-out research work "Development of recommendations about fireproof formation of a rock dump of COF

"Pavlograd" developed by NIIGD and PB "Respirator" moisture of dump weight shouldn't exceed 17,3% for preservation of stability of slopes of a dump 320.

## 6. EXPERIENCE OF WAREHOUSING OF WATERY WASTE OF COAL PREPARATION

Experience of GOF "Hookovskaya", with the simultaneous solution of these two tasks is interesting.

It is known that receiving watery waste in the look allowing their transportation on a rock dump consists of two main stages:

- condensation of watery waste;
- dehydration of the condensed watery waste on filter presses.

It is known also that the factory incurs the main expenses on preparation of watery waste for warehousing at the second stage (more than 80%).

Dehydration of the condensed watery waste on filter presses are replaced with their dehydration by a drainage through a layer of rock of gravitation with the subsequent natural drying before material cementation.

The condensation of watery waste of GOF "Hookovskaya" is carried out in a gravitational thickener of 500 $m^3$ which drain product is technical water with the contents firm to 10 g/l.

The condensed product of a gravitational thickener downloads in tankers, and is transported on a rock dump for in common warehousing with rock of gravitation.

Prior to dumping of rock weight (figure 1) on all perimeter of a rock dump the air-tight shaft is formed of clay, 1,2 m wide and up to 1,0 m high, with the subsequent advancing building up to the design height. In a consequence of that, the external slope closed from a wind is formed.

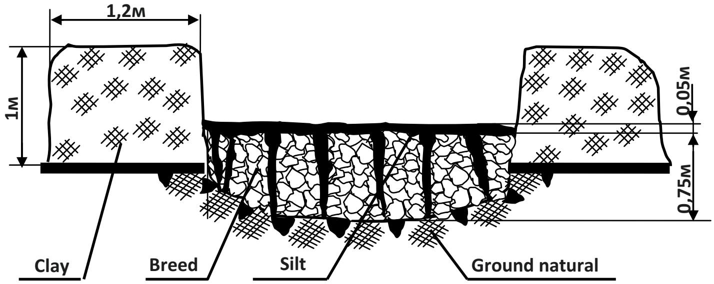


Fig. 2 Section of a Rock Dump when forming a layer together with watery waste

The way of warehousing of rock includes preparation of the basis of a dump, rock delivery by the motor transport, dumping of the rock inclined to self-ignition in a layer of 0,75 m which is planned and condensed with the bulldozer.

The stored rock from above fill in with a layer of high-ash silt (kl.0-0,05 mm an ash-content of 55-65% and moisture of 50-60%), for the purpose of prevention of intake of oxygen in interfractional hollowness of rock, thereby we receive the rock weight, not inclined to self-ignition, of which form a rock dump.

The subsequent layers of rock mass on a rock dump form by the same principle, layer-by-layer - circles, until final formation of a rock dump.

Recultivation of the created surface of a rock dump is carried out in two stages: technical and biological.

## 7. CONCLUSION

Formation of rock dumps with a jellied mass of the dehydrated slimes or watery slurry waste allows to solve problems with use of a way of joint warehousing of gravitational rock: release of land resources from the areas of disposal effluent [7, 11], quickly to operate one object on storage of production wastes, quickly to return reverse waters in technological process.

The offered ways and measures exclude burning of a rock dump as all emptiness between pieces of rock are filled with silt. After drying silt cements pieces of rock that doesn't allow further a rock dump to be deformed – it becomes the uniform condensed weight.

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## Mercury reduction in hard coal cleaning processes in Poland – the technology and the environmental impacts

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**Abstract:**
Coal cleaning is listed among the methods of mercury emissions reduction from the point of view of coal utilisation processes. The article quantifies the content of mercury in Polish hard steam coals, the charge of mercury and, for around 20 Polish Coal Preparation Plants (CPP), the split of mercury charge in raw coals between commercial coal products and waste. The mean mercury content in Polish hard steam coal is presented also. The differences between the washability (in terms of mercury behaviour) of different size fraction of raw coal are discussed and emphasized. Coal cleaning, giving us the benefit of cleaner fuel production, is accompanied by waste release. Waste from coal cleaning may be regarded as one of the sources of the release of mercury to the environment, what wasn't the object of in-depth analyses so far. The annual charge of mercury in waste from hard steam coal cleaning in Poland was assessed and emissions of mercury from waste dumps as well as the increase of the mercury concentration in the surrounding soils are discussed.

**Key words:** Polish hard steam coal, coal cleaning, mercury, mercury concentration, mercury charge, mercury reduction, waste, environmental impacts, mercury emissions from waste dump

## Introduction

Coal consumption is listed among the activities responsible for dominant part of anthropogenic mercury emissions to the atmosphere (UNEP, 2103). There are several techniques available to reduce mercury emissions on the coal consumer side (Pavlish et al., 2003; Yudovich et al., 2005; UNEP, 2010; SLOSS, 2012). Coal cleaning gives also additional benefit of reduction the charge of mercury in commercial coal products, so is also regarded as the method of mercury emissions reduction (UNEP, 2010; SLOSS, 2012).
Mercury reduction in coal cleaning processes varies due to different technological characteristics of coal as well as the range of coal cleaning (DAS et al., 2013; OZBAYOGLU, 2013; Pyka et al., 2010; Pyka et al., 2015, Quick et al. 2002 WENFENG WANG et al., 2006; WENFENG WANG et al., 2009). The employment of techniques, concerning the reduction of mercury emissions, both in collieries and on the coal consumers' part, creates waste streams contaminated with mercury.
In Table 1 the data on hard coal produced in Poland sales structure for 2012 and 2013 are given. The production of around 70-75 millions metric tons of hard coal annually is accompanied by the release of around 35 millions metric tons of waste from coal exploitation and cleaning. In Poland, 84% of electricity production is based on coal and coal consumption secures 54% of whole energy needs of Polish economy. Coal (hard coal and lignite) gives Poland relatively low level of energy dependence among European Union members (European Union, 2014). The recognition of the scale of all potential risks, created by the content of mercury in exploited and utilised coals is vital for the economy based in such a big way on coal.
Lignite exploited in Poland is not treated in any way before utilisation. Only hard coal is cleaned. The article concentrates on hard steam coal only so further discussion doesn't concern lignite and coking hard coal production in Poland.
The article deals with three topics:

- the assessment of mean mercury content in Polish steam hard coals as well as the charge of mercury,



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- the differences in technological characteristic of different size fractions obtained from single ROM coal in terms of mercury reductions,
- the quantification of the split of mercury charge in raw coal between products and waste streams and the discussion of the potential environmental impacts of mercury in waste.

Table 1. The sales structure of hard coal produced in Poland in 2012 and 2013 [13].

| Specification | | **2012** | | **2013** | |
|---|---|---|---|---|---|
| | | million tons | % | million tons | % |
| **Total sales** | | **71.17** | **100.0** | **75.74** | **100.0** |
| **Domestic sales to** | | **63.79** | **89.6** | **65.28** | **86.3** |
| of which: | commercial power industry | 35.36 | 49.7 | 35.65 | 47.1 |
| | industrial power plants | 1.48 | 2.1 | 1.76 | 2.3 |
| | industrial and municipal heating plants | 5.56 | 7.8 | 4.44 | 5.9 |
| | other industrial recipients | 0.43 | 0.6 | 0.57 | 0.8 |
| | coking plants | 9.77 | 13.7 | 10.36 | 13.7 |
| | other domestic recipients | 11.19 | 15.7 | 12.49 | 16.5 |
| **Export** | | **7.38** | **10.4** | **10.46** | **13.7** |

## Hard steam coal cleaning in Poland

Hard steam coal is not cleaned in Poland in full way. For the purpose of this article we can generalize that two basic models of steam coal cleaning technologies are employed in Poland. First one consists in cleaning only +20 mm (roughly) size fraction to produce cleaned coarse and medium size grades. Smalls are raw coal then. The second model involves also cleaning of -20 mm size fraction (smalls) – including, or not, the cleaning of the finest coal. Both models are presented in Figure 1 with the amount of CPPs fitting each one.

The range of hard steam coal cleaning in Poland can be characterized by two numbers. For +20 mm size fraction (the share in ROM coal equals 30-40%) 100% of ROM coal mass is cleaned. For -20 mm size fraction (the share in ROM coal equals 60-70%) only around 40-50% of ROM coal mass is cleaned. The rest of -20 mm ROM coal is used to compound the commercial products – smalls (different steam coal blends) as raw coal (Dubiński 2011).

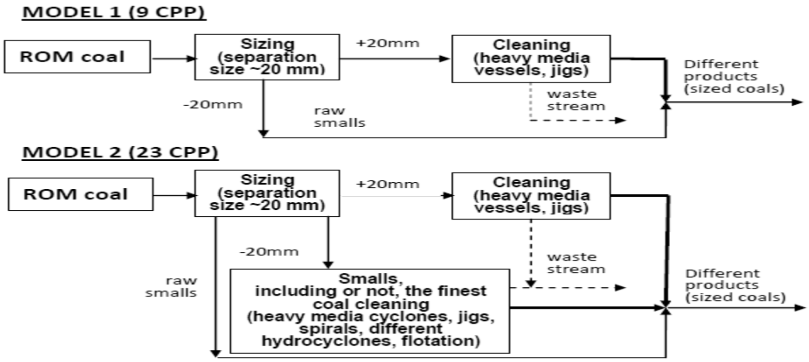


**Figure 1. Generalized models of hard steam coal cleaning in CPP in Poland.**

## Methods of research

The assessment of mean mercury content in Polish steam hard coals and mercury charge as well as the split of mercury charge in raw coal between commercial products and waste, were assessed based on analyses of samples taken from all commercial products of all CPP in Poland. It means: coarse size grades (lumps, cobbles, nuts), middle size grades (different pea coals) and different smalls and slimes (the finest coals) produced in 2014. In many cases waste products were also taken. The samples, one or two, were collected in each case for at least one – two weeks. Parameters determined (including mercury) were attributed to the annual production (2014) of given sized coal in given CPP and weight mean mercury content and mercury charge were calculated. Basing on these results the split of mercury charge from raw coal between commercial products and waste was assessed.

<u>The differences in technological characteristic of different size fractions obtained from single raw coal in terms of mercury reductions,</u>

Float and sink tests were performed for size fraction: 120 - 20 mm, 20 – 0.5 mm and 0.5 – 0.045 mm, separated from the one ROM coal, using the standard method (PN-ISO 7936). Ash determination, sulphur content and calorific value results from applying standard methods, respectively: PN-ISO 1171, PN-G-04584 and PN-ISO 1928. Mercury content was determined according to the certified internal procedure elaborated in GIG No. SC-1.PB.23 applying the Cold-Vapor Atomic Absorption Spectrometry.

<u>The potential environmental impacts of mercury in waste.</u>

The tests have dealt with monitoring of soils in the surrounding of the waste dumping ground site and the atmosphere contamination with the mercury compounds, originated from the burning dumps. Soils sampling was based on Polish standards' procedures concerning soils and lands (PN-ISO 10831-5 and PN-R-04031) and burning areas of dumps tests comprise the surface temperature measurements applying contactless method, internal temperature measurements, fire gases measurements (Hg, CO, O2, CO2).

## The results and discussions

<u>Mean mercury content and mercury charge in Polish steam hard coals.</u> The weight mean value for mercury content in Polish steam hard coals for 2014 was assessed for 107 μg/kg (as received). It means that the charge of mercury in all commercial products of steam hard coal amounted around 6500 kg in 2004. In Table 2 the weight means for mercury content (as received), and three simple statistics for ash (air dried) in groups of size grades of commercial products are presented with their annual production.

Table 2 The results of assessment of mean mercury content and charge in Polish steam hard coals

| | Coarse size grades | Middle size grades | Small (and other) | Slimes (the finest coal) |
|---|---|---|---|---|
| Annual production, millions metric tons | 5.2 | 2.3 | 51.2 | 0.5 |
| Mercury, weight mean, μg/kg (as received) | 64 | 87 | 113 | 81 |
| Ash, mean, %, (air dried) | 5.2 | 5.9 | 19.5 | 25.2 |
| Ash, quartile I, %, (air dried) | 4.0 | 4.4 | 9.2 | 16.4 |
| Ash, quartile III, %, (air dried) | 6.8 | 7.0 | 26.7 | 44.9 |

The coarse and middle size grades are cleaned coals only so ash in this products is low, around 5-6% , and the variation of ash in all products is not large. 50% of results of ash fit the range from 4-7% for both. Smalls and the finest coals are partly raw so they are characterized by higher ash, respectively 19.5 and 25.2 % and the variation of ash in these products is much greater then in the case of the coarse and middle size grades. Mercury differs also between analysed groups of commercial products and is visibly lower in cleaned coals (coarse and middle size grades) than in partially raw smalls. The relatively low mercury in the finest coals results from the huge share, in the total production of this size grade (almost 30% of mass), of the product form one colliery, in which mercury content is very low, around 40 μg/kg.

<u>The differences in technological characteristic of different size fractions obtained from single raw coal in terms of mercury reduction</u> describe the results of float and sink tests given in Table 3.

Coal under consideration was subbituminous one so ashes for the lightest density fractions (-1.4 g/cm$^3$) are relatively high (around 6-7 %) and calorific values are relatively low (around 25.5 MJ/kg). The variations of ash and calorific value in density fractions for all size fractions tested are comparable. In the case of mercury content we could identify the lack of the similar tendencies. Mercury content is different for different size fractions and although, for this coal, the increase of mercury content with the raise of the density of density fractions occurs, the ranges of mercury content are different for all tested size fraction. The greatest mercury content - 233 μg/kg – was found for the size fraction 0.5-20 mm (around 47 % of mass of ROM coal), comparing with 137 μg/kg in size fraction +20 mm (around 37 % of mass of ROM coal) and 116 μg/kg in size fraction 0.045-0.5 mm (around 6 % mass of ROM coal). So the greatest part of raw coal is characterized by the highest mercury content. The best results, from the point of view of mercury in coal reduction, were obtained for size fraction 0.045-0.5 mm – less than 50 μg/kg of mercury in size fraction -2.0 g/cm$^3$. Unfortunately the share of this size fraction in ROM coal is the lowest and this size fraction is cleaned in a few Polish CPP only (Figure 1.). The cleaning of dominant part of smalls – size fraction 0.5-20 mm – leads to clean coal with mercury content around 170 μg/kg (density fraction -2.0 g/cm$^3$). The differences observed, concerning mercury distribution in density fractions, can have other character but they were noticed for many of Polish hard steam coals (Pyka et al. 2010, Pyka et al. 2014).

Table 3. The results of float and sink tests for different size fractions separated from one ROM coal.

| Density | Yield | | Ash content, Aa | | | Calorific value, Qia | | | Mercury content, Hg[a] | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| fractions | Fraction | Cumulative | Fraction | Cumulative | Cumulative | Fraction | Cumulative | Cumulative | Fraction | Cumulative | Cumulative |
| δ, g/cm3 | γ, % | γ, % | % | floats, % | sinks,% | kJ/kg | floats, kJ/kg | sinks, kJ/kg | % | floats, μg/kg | sinks,μg/kg |
| Size fraction +20 mm | | | | | | | | | | | |
| - 1.5 | 58.5 | 58.5 | 8.7 | 8.7 | 68.5 | 25 126 | 25 126 | 6 693 | 107 | 107 | 180 |
| 1.5 - 1.6 | 5.1 | 63.6 | 31.1 | 10.5 | 73.8 | 18 229 | 24 571 | 5 066 | 160 | 111 | 183 |
| 1.6 - 1.8 | 5.1 | 68.7 | 43.6 | 12.9 | 78.8 | 14 392 | 23 816 | 3 546 | 265 | 122 | 170 |
| +1.8 | 31.3 | 100.0 | 78.8 | 33.5 | | 3 546 | 17 481 | | 170 | 137 | |
| Razem | 100.0 | - | 33.5 | - | - | 17 481 | - | - | 137 | - | - |
| Size fraction 0.5-20 mm | | | | | | | | | | | |
| - 1.4 | 65.1 | 65.1 | 7.0 | 7.0 | 59.0 | 25 515 | 25 515 | 8 776 | 151 | 151 | 386 |
| 1.4 - 1.5 | 3.6 | 68.7 | 21.1 | 7.7 | 64.5 | 21 634 | 25 310 | 7 292 | 316 | 160 | 394 |
| 1.5 - 1.6 | 3.5 | 72.2 | 27.0 | 8.7 | 69.0 | 18 736 | 24 992 | 5 855 | 238 | 164 | 413 |
| 1.6 - 1.7 | 2.2 | 74.4 | 37.0 | 9.6 | 71.8 | 16 764 | 24 743 | 4 897 | 289 | 168 | 424 |
| 1.7 - 1.8 | 2.1 | 76.5 | 45.4 | 10.6 | 74.2 | 13 861 | 24 446 | 4 099 | 293 | 171 | 436 |
| 1.8 - 1.9 | 1.3 | 77.8 | 49.5 | 11.2 | 75.7 | 11 507 | 24 224 | 3 653 | 328 | 174 | 442 |
| 1.9 - 2.0 | 1.5 | 79.3 | 57.8 | 12.1 | 76.9 | 9 297 | 23 951 | 3 258 | 389 | 178 | 446 |
| + 2.0 | 20.7 | 100.0 | 76.8 | 25.5 | | 3 254 | 19 658 | | 445 | 233 | |
| Razem | 100.0 | - | 25.5 | - | - | 19 664 | - | - | 233 | - | - |
| Size fraction 0.045-0.5 mm | | | | | | | | | | | |
| - 1.4 | 48.5 | 48.5 | 6.2 | 6.2 | 62.5 | 25 865 | 25 865 | 8 599 | 21 | 21 | 205 |
| 1.4 - 1.6 | 12.8 | 61.3 | 17.5 | 8.5 | 77.5 | 22 325 | 25 125 | 4 051 | 97 | 37 | 240 |
| 1.6 - 1.8 | 4.6 | 65.9 | 41.6 | 10.9 | 82.3 | 14 895 | 24 408 | 2 580 | 123 | 43 | 256 |
| 1.8 - 2.0 | 2.1 | 68.0 | 54.4 | 12.2 | 84.1 | 11 052 | 24 005 | 2 036 | 154 | 46 | 263 |
| + 2.0 | 32.0 | 100.0 | 84.1 | 35.2 | | 2 036 | 16 973 | | 263 | 116 | |
| Razem | 100.0 | - | 35,2 | - | - | 16 973 | - | - | 116 | - | - |

The quantification of the split of mercury charge in raw coal between products and waste streams and the discussion of the potential environmental impacts of mercury in waste.

The values for mercury charge in commercial products and wastes result both from mercury concentration as well as mass of product (waste) – Figure 2. Summarizing these results and comparing with the charge of mercury in steam coal commercial product it can be assessed that the total charge of mercury in waste streams, for hard steam coal production, amounts around 3000 kg per year.

The mercury content in waste, dumped on the landfill, leads to the increase of the mercury concentration in the surrounding soils, particularly in the direction in which the winds blow. Moreover, it was shown that, analysing the soil profile, the amount of the cumulated mercury decreases with depth, what can be related to the sorption properties of the soils.

In Table 4 the results of field tests, targeting at defining the concentration of mercury in the gases, released from the burning waste dumps, are given. Tests were performed over holes done in the surface of a thermally active dump (Białecka 2015). Assuming that the current surface of the objects formed from coal waste, which are thermally active only, amounts around 40 ha, the estimated value of the mercury emissions to the atmosphere from this area can equal 140-150 kg per year.

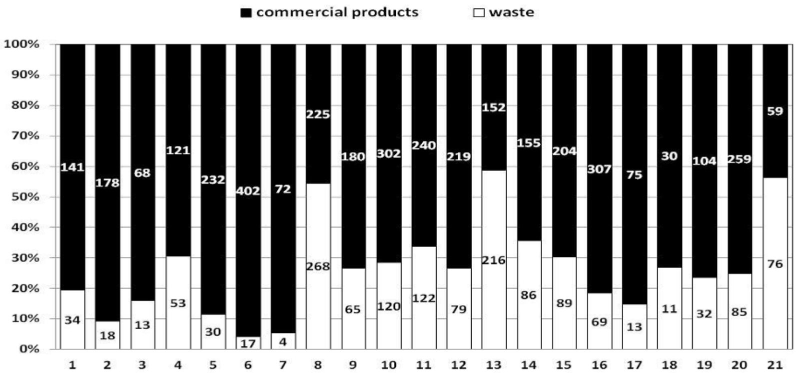


**Figure 2. The split of mercury charge in raw coal between products and waste streams for 21 collieries and masses of mercury charge in kilograms.**

Table 4. Mercury concentrations in exhalation gases from the burning mining waste dump. Atmospheric condition: cloudy, light wind, temperature 5°C.

| Point No | Ground surface temperature, °C | Hg Background (2 meters from a hole, 1 m above ground level), $ng/Nm^3$ | Hg (2 cm above a hole), $ng/Nm^3$ |
|---|---|---|---|
| P1 | 55 | 14 | 324-463 |
| P2 | 61 | 80 | 280-490 |
| P3 | 34 | 66-83 | 140-260 |
| P4 | 50 | 50-60 | 167-280 |

## Conclusions

1. The annual charge of mercury in commercial products from hard steam coal mining in Poland was estimated for around 6500 kg in 2014, while, as the result of coal cleaning, wastes carry the charge around 3000 kg of mercury per year.
2. Hard steam coals are not cleaned in Poland in full way. Taking into consideration the results of float and sink tests performed already, as well as the results of analyses of mercury content in commercial products, the charge of mercury in Polish hard steam coal can be further decreased by broadening the scope of smalls cleaning. The partial cleaning of smalls in 14 CPP gives the reduction of mercury charge in commercial products, comparing it with ROM coal, from around 11% (only +20 mm size fraction is cleaned) to over 30%.
3. The assessment of mercury reduction in coal cleaning processes requires detailed analyses of washability of whole ROM coal. Analyses limited to single (narrow) size fraction, separated from ROM coal, can not be (and in the most cases are not) representative for the whole ROM coal.
4. The monitoring of soils on the areas of mining activities have revealed low threat for the soil-water environment, originated from dumping of wastes from hard coal mining.
5. The burning coal mining waste dumps are a serious threat to the natural environment. The conducted analyses of the burning coal mining waste dumps leave no room for doubts: the burning dumps are also the source of mercury emissions. It should be stressed that the conducted tests are one of the first attempts aiming at the presenting of the problem of mercury emissions from the burning coal mining waste dumps in Poland

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# Modelling of coking coal preparation taking into account mercury content variability

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Mercury (Hg) is one of the most toxic elements contained in coal released to the atmosphere during the combustion and coking processes. Compounds of these element are easily emitted to the atmosphere thanks to its very low boiling point which equals 356,7 °C. The Minamata Mercury Act of United Nations in 2014 obliged its signatories to introduce national programmes to reduce emission of Hg compounds.

Clean coal technologies consist of means enabling to lower the amount of toxic compounds emitted to the atmosphere. Precombustion techniques and methods focus on removal of undesired impurities from fuel. Coal preparation is the first stage when Hg content can be reduced from fuel before it will be released to the atmosphere during combustion or coke production. The data acquired from Polish coking coal preparation plants (where all the feed is being enriched) will allow to build frames of process model taking into account mercury content variability helping in:

- determination of factors affecting the mercury load in coal,
- choice of enrichment range and processing infrastructure,
- definition of production policy and needs of future investments in coal industry,
- elaboration of tools supporting clean coal production based on available equipment.

**Key words: coal preparation, mercury removal, mining waste, precombustion, coking coal, air protection, modelling of coal processing, Poland**

**Precombustion – the idea**

Mercury (Hg) is considered to be one of the most dangerous elements for human health and environment. Its content in coal seams usually varies between 50 – 150 ppb (Klojzy – Karczmarczyk, 2013). Despite the fact this is relatively low value, global use of this resource may cause up to 30 % of global Hg emission worldwide according to United Nations Environmental Programme. Its low boiling point which equals 356,7 °C, volatility and high toxicity focused the global attention. In order to solve the problem of mercury in 2013 Minamata Convention was signed as the agreement to minimize the global emission of this element. Each signatory of this document is responsible for introduction of local measures to minimize mercury release according to ways of pollutions caused by its industry (Minamata, 2013).



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In case of Poland it is believed that up to 90 % of Hg emission comes from coal combustion and coke production. This is a result of carbonization of economy which is characterized by major utilization of this fuel in power engineering. Though the further minimization of emission by introducing flue gas cleaning facilities should be continued, mineral processing according to the research seems to be at least considerable support. The precombustion stage is based on technological capabilities to prevent from occurrence of such compounds in the final fuel (Blaschke, 2008).

Another option for minimization of mercury release is fuel switching which is not reasonable for Poland because of limited amount of available deposits as well as economic issues do not allow Poland to abandon exploitation in some areas and replace it with new production capacity. Thanks to research we now know more about mercury abundance in coal mining products although the further steps to take over control on impurities migration has not been already taken. The industrial simulation and modelling seem to be rational first step to determine the capabilities of mercury reduction in coal preparation (King, 2001).

## Coal processing – mercury removal effectivity

Global research on mercury problem in coal mining products has been concerned about assessment of mercury compounds concentration in raw coal, products and refuse. The less attention was given to process parameters determining the Hg content in commercial products (Dziok, 2014), (Michalska, 2012). In case of coal preparation its actual range is the basic limitation of mercury removal. Model of preparation plants depends on the requirements of customers, energy policy, technological development and other factors. In the previous article – the authors basing on Polish research made an assessment of mercury removal in the different preparation range. The results of static calculation are presented in the table 1.

***Table 1 Mercury removal effectivity in different models of preparation plants (own elaboration based on 1)***

| Preparation model | Thermal coal 1 | Thermal coal 2 | Thermal 3 / Coking coal |
|---|---|---|---|
| Minimum size of enriched coal [mm] | 20 (10) | 1 (0,5) | Complete enrichment (depends on capacity) |
| Mercury reduction rate [%] | 40 | 24 | 12 |
| Mercury reduction rate with middlings removal [%] | 46 | 29 | 14 |

For the calculation we used the data about average content of mercury in inputs and outputs of the preparation processes taking into account the total mass of products and refuse. The presented assessment seem to be comparable with other works concerned about mercury removal during coal washing. The results of calculation for independent processes taking into consideration flow of feed and outcome of product and refuse indicated the decrease in mercury load (the numbers in brackets refer to disposal of middlings):

- 51 % (59%) – for dense media treatment,
- 30 % (39%) – for processing in jigs,
- 45 % - for flotation. (Kurus, 2015)

Significant percent of mercury in raw coal is located in middlings and yield of higher density what makes it possible to reduce the load of impurities in washed product using the conventional ways of coal processing. Although it is estimated in Poland only 30 – 40 % of coal mass is somehow cleaned so the current potential of these methods remains limited. Only coking coal is completely enriched in relation to the requirements of customers. Several collieries are not equipped with infrastructure to clean coal over 20 mm fraction. Actually enrichment comes to coarse and medium assortments provided for private household users as well as coking coal. Additionally some part of enriched product is blended with the raw one to ensure the requested quality. The situation and specific approach to coal preparation is the result of long-term policy of using raw coal in power engineering and local demand for product intended for private users.

The number of Polish works were related to content and migration of mercury in products and waste but only some of them could give advices what process parameters may enable to take over control of mercury in products (Aleksa H., 2007), (Wierzchowski K.,2010). Accurate prognosis based on general data does not seem to be relevant to all seams in Coal Basin. In each colliery the modelling and then control processes should be developed independently. The article presents research which is part of Doctoral Thesis "Modelling of coal processing taking into account reduction of certain impurities". The work is to answer a question what steps should be taken to indicate the potential of coal preparation in decreasing the load of mercury in products and how it could be controlled with use of existing infrastructure. The main aim of Polish coal preparation plant static modelling has been focused on general effectivity and processes capacity. The impurities has not been regarded as an aim of simulation or process control.

## Coking coal preparation plant – object of the analysis and modelling

The individual mercury removal rate in different separators is not enough to describe the migration of the impurities during coal preparation. The full modelling of coal processing taking into account mercury content variability requires taking into consideration all processes that may affect the outcome like dewatering and blending. The review of available infrastructure of Polish coal mines, literature studies and mercury research enabled to formulate assumptions for selection of the right facility to take analysis in:

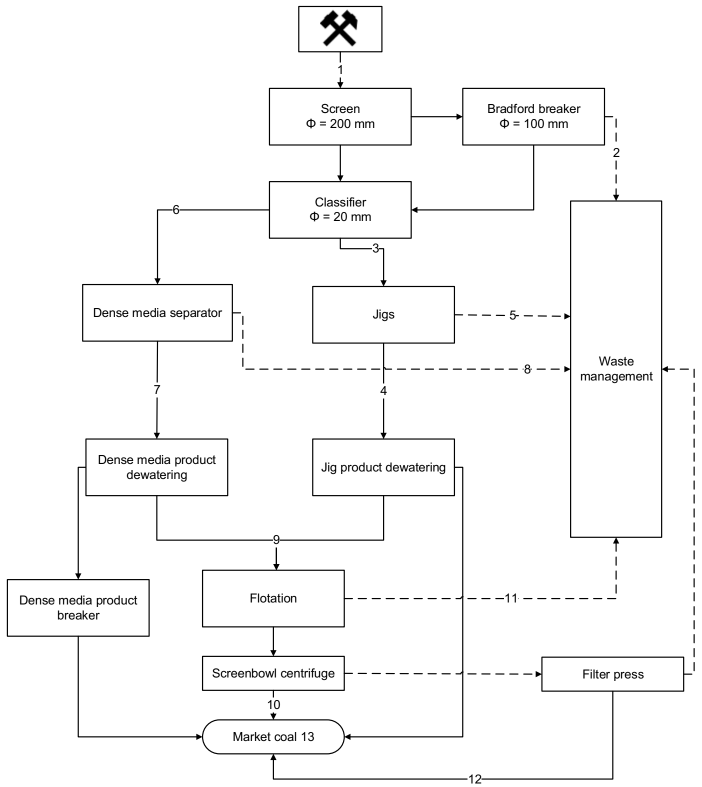
- the preparation plant should be equipped with infrastructure capable of enriching all sizing fractions,
- more types of processing units will allow to conduct more complex analysis,
- the less commercial products made the better calculation of impurities migration,
- the more typical infrastructure the more representative assessment,
- the most representative data could be acquired during processing feed from one seam.

On the basis of the assumptions the coking coal preparation plants seem to be the right choice to gain data regarding to mercury content variability in different processes. The chosen plant is characterized by daily gross capacity up to 12 000 tonnes and it is equipped with separators capable of enriching sizing fractions: over 20 mm – dense media washery; between 0,5 and 20 mm – jigs; below 0,5 mm – flotation cells.

The table 2 presents the information about capacity of the chosen preparation plant. The picture 1 presents simplified flowchart presenting the objects that may decrease the Hg content in products as well as measurement points (1-13).

*Table 2 Basic information about capacity of preparation plant (own elaboration)*

| Capacity | [t/h] |
|---|---|
| Primary screen | 1600 |
| Preliminary classifier | 900 |
| Dense media separators | 325 |
| Jigs | 475 |




| | |
|---|---|
| Flotation cells | 140 |
| Loader | 1200 |

***Picture 1 Simplified coking coal preparation flowchart with selected measuring points (own elaboration)***

The measurements are not limited to input and output of separation processes which have already been the subject of research but also:

- hard sandstone refuse from Bradford (drum) breaker,
- products of filter press (recovery of flotation product dewatering suspension),
- refuse of all processes.

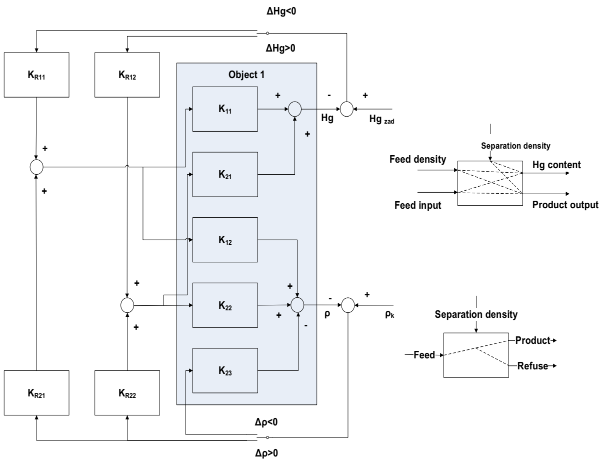
Acquiring information from measurement points will enable to simulate the migration of impurities such as mercury during complete coking coal processing. The full analysis gives also the opportunity to compare different processing models by simple deleting separators of sizing fractions to obtain the lower one. The analysis in these points will include:

- A – ash content [%],

- Q – calorific value [kcal/kg]
- S – sulphur [%]
- W – total moisture [%]
- Hg – mercury [ppb] – considered as market quality specification.
- output and input [tones]
- separation parameters [cut point: kg/dm$^3$]

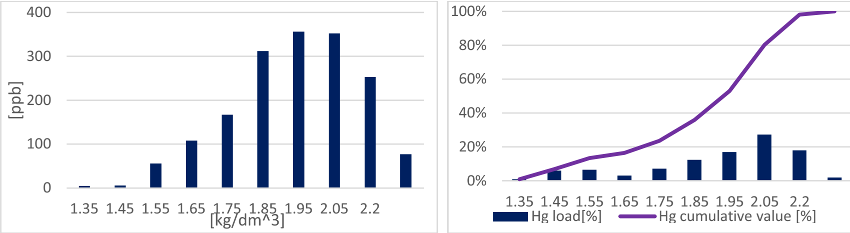
**Process model and mercury control**

The coal preparation modelling will be based on data acquired from measuring points including typical coal quality parameters such as ash content and sulphur. Additionally mercury will be considered as a trade parameter which should be reduced according to customer needs. For this purpose the best solution would be to find the correlation between Hg and traditional market parameters to enable control over its stream using conventional controllers like ash meters. Direct real – time control of mercury in feed and product is impossible due to lack of available measures. Also to describe the processes we should consider the output and input to the separators and outputs. The picture 2 presents the general model of dense media separator controlled according to 2 trade parameters as well as quantity of feed and output. The regulation of cut point is changed by the signals from radiometric ash meters.




***Picture 2 Regulation model based on 2 parameters - ash and mercury – the idea (own elaboration)***

In some cases available data confirms an existence of relationship between mercury content and density/ash that may enable to control Hg by setting cut point as for the trade parameters. Picture 3,4 present author's analysis based on available data how mercury could be controlled using gravity separation in dense media and jigs which analyzed coal preparation plant is equipped with (Chugh, 2013).




***Picture 3,4 Relation between density and mercury content in floats (based on Chugh,2013)***

If the washability curve in analyzed colliery would be comparable mercury content could be controlled by changing the cut point according to signals from ash meters as on picture . Due to the fact mercury in coal occurs with mineral organic fraction – its removal can be limited and significantly differ among extracted seams. To control the process it would be required to elaborate special washability curves for mercury in particular mines. Flotation would require more research because it is not known how its work parameters affect Hg content in concentrate.

**Summary**

Modelling of coking coal preparation may give answer what are the most effective methods of enrichment to reduce mercury load in the fuel. The article presented what parameters of processing plants should be taken into consideration to assess its potential in Hg reduction in final product. It is likely there are some capabilities of indirect control of mercury load in coal although they depend on: processing range, infrastructure and other factors. The initial modelling research should be concerned about possibilities of taking control over the mercury flow to indicate the proper ways of its reduction. Because of complete feed enrichment coking coal preparation plants seem to be the best source of information to indicate Hg variability.

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# Technological and ecological aspects of briquettes production on the basis of coal wastes in Upper-Silesian Coal Basin

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**Abstract**

The problem of coal waste management has been a local issue for many years, due to the updates of environmental laws that enforce the fulfillment of stringent requirements. Recent years have brought fundamental changes in the approach to waste management, which result in implementation of new technologies and methods, which allows re-use more coal waste. Making efforts to minimize the deposit of waste in settling ponds or in spoil tips are resulting in growing tendency to use coal mining waste in production of alternative fuels (including the briquettes).

This paper will present an analysis of the current situation and scenarios that will allow to fulfill the directives related to coal waste management. Selected evaluation criteria, covering the most important technological and environmental issues, from the point of authors view, will identify the possibilities of implementing different types of solutions in a variety of conditions, e.g.:

- the possibility of coal wastes preparation into a briquette,
- the ways of using briquettes in industry and households,
- required parameters of coal waste fuel,
- products of burning briquettes with the chosen technology,
- ability to implement solutions in local conditions.

**Key words:** *recovery, fine coal, briquettes, fuel, granulation, settling ponds, coal slime, steam coal.*

## Introduction

Production of environmentally friendly fuel which is a mixture of wastes is a required direction of sustainable development. Available technologies allow to produce high energetic alternative fuels with required  caloric value. Additionally, generated briquettes of fuels are dedicated to burn in conventional boilers and furnaces .

Up-to-date regulations are enforcing the manufacturers to proper waste management making the recovery a priority action. This efforts result in decrease of landfills. Therefore, presented in the global market innovative technological solutions allow to build a rational economy based on sustainable development.

 
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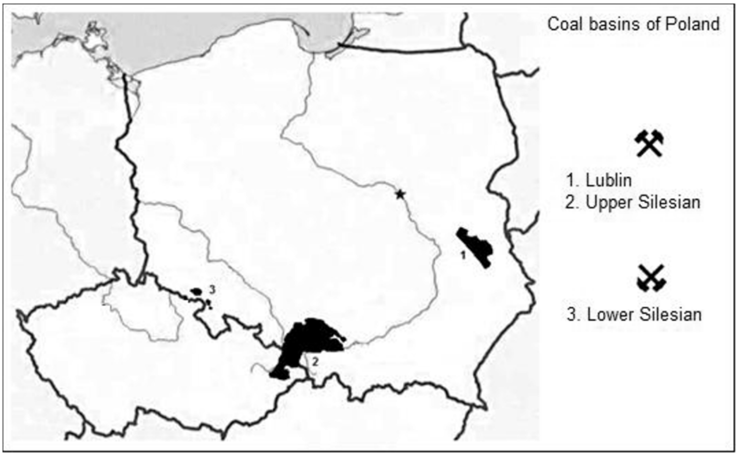
Briquetting is the one the most known ways of processing coal especially peat and lignite. This group of technologies allowing to produce fuel in shape may appear useful in Polish conditions of shortage of medium and coarse coal sizing assortments. Also it allows to transport dried slime from the settling ponds without causing dustiness. For a long time coal slimes were not useful in the industry so they were considered to be waste. Thanks to briquetting technology waste coal may be used in power engineering and households instead of being deposited in the environment.

Special attention is given to the area of Upper Silesian Coal Basin which is characterized by still unsolved problem of mining waste management. It is estimated that in Silesia district, in dumps with an area of 3.5 thousands of ha, there are deposited more than 100 million Mg of mining waste. The consequence of long-term and mass exploitation of Silesian basin are large antropogenical deposits of coal possible to utilize in briquette production.

Performed the review of methods in the area of waste management, along with the assessment criteria elaborated by the authors allow to determine the group of processing technologies, as a result of the implementation in Silesia can increase the recovery of artificial deposits.

## 1. Mining waste management in Upper Silesian Coal Basin

Upper Silesian Coal Basin is located both in Poland and Czech Republic (about 30% of its territory). Currently Poland mines coal also in Lublin Coal Basin. Due to economic issues and depletion of available deposits the coal production has been constantly decreasing since the early 90's. The current production is approximately 63 millions of tons a year (2014) of thermal and coking coal. About 200 years of intensive extraction of deposits resulted in dumping big amounts of gangue and fine coal into spoil tips and settling ponds. The problem exists also in Lower Silesian Coal Basin where production ceased in early 2000's. The picture 1 shows the location of mentioned mining areas.




*Picture 1. Coal basins of Poland. Source:own elaboration based on (Kurus, 2014)*

It has been calculated the total mass of anthropogenic deposits in Upper – Silesia may exceed 100 million of tones in settling ponds and 4 billion of tones in spoil tips. The availability of them is constantly changing because of terrain reclamation, spatial planning and extraction. Relatively high calorific value of dumped material in settling ponds (calorific value varies from 3000 to 6000 kcal/kg) enables use of it as fuel after enrichment in hydrocyclones or flotation cells. Utilization and transport of dried coal slimes may cause harmful, local dustiness. Also some of the medium and coarse waste remains valuable because of past imperfection of processing techniques that could not prevent from high concentration of carbon in tailings. That fact is the reason of spoil tips fires occurring in the coal mining areas.

Still about 70 % of single – family Polish households uses coal and wood for heating. Due to the reduction of coal extraction – the amount of required sizing assortments – coarse and medium is not high enough to meet the local demand. The mechanization and domination of longwall systems contributed to growth of fine coal in feed. The most of imported coal are coarse and medium assortments.

The popularity of semi – automatic boilers equipped with retort burner resulted in growth of demand on medium coal assortments. The boilers are normally designed for sizing assortments between 5 and 25 mm called "eco-pea" on Polish market. On the other hand Polish coal mining produces too fine assortments being replaced by cheaper imported ones. Briquetting may appear a solution of simultaneous shortage and surplus of coal.

## 2. The current purpose of briquetting coal

A key task of the forming process of the fuel is to identify the components and their properties, which are directly affected to the combustion efficiency. An interest in waste treatment and reuse them as fuel is associated with a flexible approach to compounding and molding. Fuel properties, as well as their chemical composition must be adapted to the needs and therefore fuel production is a complex process. The ability of molded fuel's adaptation to the expectations of the recipients is one of the many advantages of waste reuse as a energy products, other benefits may include (Wandrasz, 2006):

- the high calorific value that is comparable with other traditional fuels,
- ability to control the composition of the fuel, which affects the carbon efficiency of its combustion/ co- combustion,
- fuels derived from waste treatment, exhibit storage properties, thereby reducing the negative impact on the environment,
- the granulation of fuel is directly connected with the expectations of consumers and transportation properties.

The fuel pellets production technology (pellets, briquettes, bales) is based on the control process of mixer, which is connected with the weighting dispensing flammable substances and stabilizers (binder) . Modern pro-environmental trends point to the need to treat waste as a potential valuable raw materials for recovery and development. According to this assumption, waste from coal mining are a high energy potential sources.

In order to wastes manage, it is necessary to processed them into the homogeneous form. This is very important due to the mixing process of fuel components with the binder and their formation. Steam coal is material which has a low susceptibility to merge in the briquetting presses. Therefore it is desirable to prepare the appropriate mixtures to improve the ability of connection the briquettes components (Hycnar, 2005).

Another important feature of wastes is their reduced fuel properties caused by inter alia high humidity. Previous studies and researches have shown that the most effective method for increasing the calorific value of coal wastes (including sludge) is to reduce the water content and ash content by the granulation. The obtained granulate is not only characterized by increased fuel properties (up to minimum 2 MJ/kg) but also better transport and storage properties. Economic and ecological benefits of waste recovery by their granulation became a source of research and innovative implementations of waste fuels briquetting technologies.

## 3. Briquettes production on the basis of coal wastes in Upper-Silesian Coal Basin

Nowadays, in Upper Silesian Coal Basin there are only 2 applied different technologies for production of briquettes:

- Coal pellet production – innovative way of manufacturing fuel that consists of: blending flotation concentrate, additives and binder to prepare a mixture. Then hot fuel is extruded into pellets of desired diameter. The humidity of pellets is reduced thanks to belt dryer. (picture 1)
- Briquetting of coal slimes for transport purposes – forming the briquettes using coal slimes from old settling ponds with addition of CaO as a binder. (picture 2)

*Picture 1 Coal pellets extrusion (by Krzysztof Kurus)*

*Picture 2 Mobile briquetting plant. Source:haldex.pl*

Analysis of technological and ecological solutions implemented in Silesia, allowed to develop comparisons of products recovered from wastes, through its granulation and parameters'evaluation (table 1).

**Table 1. Fuels comparison**

| **Producer** | **Polski Koks S.A.** | **Haldex S.A.** |
|---|---|---|
| Product | Fuel - Varmo | Steam coal, granulated coal slime, fine coal with the granulated coal slime |
| Product picture |  |  |
| **Technical parameters** | | |
| Composition | Flotation waste, brown coal, bio-component binder | Waste coal slimes from current production , as well as from settling ponds exploitation<br>The calcium oxide serves as a binder, improving the durability of the granules |
| Trading purpose | It is designed for modern boilers with automatic feeding of fuel (boilers with feeding screw or piston) | For use in power plants and households with appropriate boilers |
| Calorific value [MJ/kg] | 26-27 | 10-14 |
| Granulation / Device | Pellets Ø 12 –16 mm or 20 mm, length 10 – 30 mm, produced on the extruder | Mixing and granulating devices.<br>Granulated coal slime production on installation DW-29/5CONTI, stationary and mobile.<br>Manufacturer: Eirich |

| | | |
|---|---|---|
| Moisture [%] | 6,3 | 17-25 |
| Ash [%] | 11 | 30-40 |
| Capability | 60k tones/ year | Stationary unit granulates about 80 tones of coal slime per hour, while the capability of the mobile unit is 40 Mg/hour.<br>10 000 tones/month to 24 000 tones /month. |
| **Ecological parameters** | | |
| The emission factor [MgCO2/TJ ] | 94,32 | Lack of data |
| Sulfur content | max. 1/0,8% | 0,6-0,8% |
| Chlorine content | max. 0,33 % | Lack of data |
| Nitrogen content | max. 1,25 % | Lack of data |

***Sources: own elaboration based on (Kugiel,2012; products data sheets)***

**Technology assessment**

Production of Varmo fuel by the company Polski Koks SA - is based on the extruder, which originally was intended for the production of ceramics elements. Formed fuel production based on extrusion process is the unique example in Europe, which also allows to adjust shapes according to requirements. A high calorific value of fuel, which is directly connected with a low moisture and ash content is a key advantage of this product. The main defect of this fuel is its fragility. The transport of fuel causes fragmentation of product. For the final recipient it means the considerable amount of loosely fuel.

Much better transport and storage properties are characterized by granulated coal slimes produced by Haldex. It is directly related with the design of the technological granulation process, which was aimed to transform untransportable and very moist waste into the full calorific value fuel with reduced moisture content and reduced sulfur content. Haldex is the single holder of the license to use the granulation technology based on coal slimes in pursuance of patent no. 207431 from the 1.06.2004 (Kugiel, 2012). Except the stationary installation of coal slime granulation in the Haldex plant is possible to use a mobile version, which can be settled in close distance to settling ponds or places of mixtures' production.

Importantly, owned by the company technology is not limited only to the production of granulated coal slimes, but it also enables to produce other granular products- for example the humus substitute called the BioCarbohumus (comprising i.a. mud and sludge).

**Ecological assesment**

The use of energy products based on listed technologies undoubtedly was aimed to achieve a ecological effects, including primarily the reduction of waste amount deposited in landfills. In both cases, the effect has been fully achieved and theirs rate is calculated according to plant capability.

Coal slimes granulation form the settling ponds using Haldex's technology, each year helps to reduce the amount of waste in landfills. Also coal slimes granulation from current production prevents the generation of new waste and extending waste landfills. From an ecological point of view, the reclamation of settlers ponds allows to restore the utilitarian and nature value, which leads to green areas recovery or to redevelopment them as a industrial areas. In the other hand granulation of steam coal prevents dusting and facilitates transport of final products.

VARMO fuel technology also is aimed to reduce emissions of sulfur dioxide, nitrogen dioxide, carbon monoxide, dust and highly carcinogenic benzoapyrene in total amount more than 90%. According to the producers calculations, using the new fuel enables to reduce emission of about 600 tones of particulate matter per year; sulfur dioxide emissions is reduced by 98 tones, nitrogen oxide by 340 tones and nitrogen oxides by 4 thousand tones.

## 4. Summary

One of the key areas of research carried out in Silesian district is to minimize the negative impact of coal mining to the environment. Implementation of innovative technologies which granulate waste energy products have a significant contribution to achieving the environmental effect by minimize the waste stream deposited in landfills. Each of these technologies has to focus on its own properties of the final product, which confirms the wide application of the granulation process. Undoubtedly a highly urbanized Silesia region helps with the exploration and implementation of new solutions aimed at the recovery of waste from coal mining.

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# Selection of Parameters of the *Support Vector Machine* Method to the Problem of Subsidence Modelling due to Drainage

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**Abstract:**
This article presents the problem of modelling drainage subsidence that accompanies the mining of solid minerals. Rock mass drainage causes a change in pressure in the aquifer, and thereby initiates the compaction process. On the surface we can observe the effect in the form of a wide drainage basin, which adds to the direct impact of mining operations. The article presents the research stage associated with the use of artificial intelligence in forecasting the indirect impacts of (drainage) in mining areas. This article also outlines the Support Vector Machine (*SVM*) method and its use based on the example of underground coal mining. For the purpose of calculations, the data from altitude surveying conducted on the terrain surface, and information from the network piezometric boreholes installed in subsequent aquifers were used. Used in the analysis was *ε-SVM* method for regression tasks with the use of radial basis function. The calculations were performed with an integrated software package for support vector regression (*LIBSVM*) and the obtained results were presented. The process of selection of parameters in different variants, and obtained discrepancies in the process of research and testing were described. Cross-Validation and generalization of the knowledge processes necessary for future forecasting the process of drainage subsidence were characterized. The summary includes opportunities for further research as well as analysis using artificial intelligence.

**Key words:**
Subsidence, Dewatering, Rock Mass, Deformation, Artificial Intelligence, Support Vector Machines (SVM), LIBSVM

## 1. Defining research problem

Fluctuations in water levels in the deep aquifers accompany underground and open-pit mining, both during making the deposits available and its exploitation, as well as during the liquidation of the mine. Displacements of land caused by changes in the aquifers should be modelled and tested, just like the displacements caused directly by exploitation of solid minerals. The complexity of the development process of the land surface subsidence due to drainage, and a review of existing methods of forecasting (Galloway & Burbey, 2011; Witkowski, 2014) indicate the possibility of using artificial intelligence tools in the modelling of compaction process of aquifers accompanying mining of mineral raw materials (Witkowski, 2014). Both neural networks, as well as Support Vector Machines (*SVM*) method are well known computational techniques (Bishop, 1996; Vapnik, 1998; Osowski, 2006; Steinwart & Christmann, 2008; Chih-Chung & Chih-Jen, 2011). These methods are used successfully in issues of protection of mining areas (Ambrozic & Turk, 2003; Kim at all., 2008; Zhi - xiang at all., 2009; Kumar at all., 2010; Lee at all., 2012; Witkowski 2015). However, these methods were not used previously in drainage subsidence modelling in mining areas. Under the project "*New algorithm for modelling of land surface subsidence due to the rock mass drainage*" implemented by the AGH University of Science and Technology in Cracow, studies are being conducted on the use of artificial intelligence in forecasting drainage displacement in mining areas. Previous studies have focused on the possibility of applying artificial neural networks of multi-layer perceptron type (Hejmanowski & Witkowski, 2015). This paper discusses preliminary results of studies on the use of machine learning method and its parameterization.

 
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## 2. Research Area

In the analysis, data from an underground coal mine was used. Changes on the ground surface are recorded by the observatory network existing from the beginning of the operation of the mine. Analysis of the vertical displacement increments recorded on the network not directly related to mining operations made it possible to determine the extent and magnitude of the impacts of drainage (Hejmanowski at all., 2013). In turn, aquifers are observed by a network of boreholes, where piezometric head of the water level are recorded few times a year (Hejmanowski at all., 2013). A more detailed description of the research area is presented in another work by the Authors (Hejmanowski & Witkowski, 2015). Data obtained from the research area shall hereafter be called *Data I*. They represent the state of the drainage basin in *2010*, and the observations of changes in head of piezometric level together with thicknesses of aquifers. In addition, in analyses the data from another period of time, called the *Data II* was used, needed to determine the degree of generalization of the learning process.

## 3. Support Vector Machines Method

In the early '90s a method of machine learning, i.e. Support Vector Machines (*SVM*) (Vapnik, 1998) was developed, free from limitations that neural networks of Multi-Layer Perceptron (*MLP*) and Radial Base Function (*RBF*) are characterized with. *SVM* method is used mainly for classification and regression tasks. Method of learning of *SVM* networks comes down to square programming by minimizing the objective function for measurement data $(\boldsymbol{x}_i, d(\boldsymbol{x}_i))$ in the domain of real numbers (Osowski, 2006; Chih-Chung & Chih-Jen, 2011) defined by the function:

$$min\{\phi(\boldsymbol{w}, \xi_i, \xi_i^{'})\} = \frac{1}{2}\boldsymbol{w}^T\boldsymbol{w} + C\left[\sum_{i=1}^{p}(\xi_i + \xi_i^{'})\right] \quad (1)$$

with limitations:

$$d_i - \boldsymbol{w}^T\varphi(\boldsymbol{x}_i) - b \leq \gamma + \xi_i \quad (2)$$

$$\boldsymbol{w}^T\varphi(\boldsymbol{x}_i) + b - d_i \leq \gamma + \xi_i^{'} \quad (3)$$

$$\xi_i, \xi_i^{'} \geq 0, i = 1, \dots, p \quad (4)$$

and solving the dual problem using Lagrange multipliers:

$$max\{Q(\boldsymbol{\alpha}, \boldsymbol{\alpha}^{'})\} = \sum_{i=1}^{p} d_i(\alpha_i - \alpha_i^{'}) - \gamma\sum_{i=1}^{p}(\alpha_i + \alpha_i^{'}) - \frac{1}{2}\sum_{i=1}^{p}\sum_{j=1}^{p}(\alpha_i - \alpha_i^{'})(\alpha_j - \alpha_j^{'})K(\boldsymbol{x}_i, \boldsymbol{x}_j) \quad (5)$$

with limitations:

$$\sum_{i=1}^{p}(\alpha_i - \alpha_i^{'}) = 0 \quad (6)$$

$$0 \leq \alpha_i, \alpha_i^{'} \leq C, i = 1, \dots, p \quad (7)$$

where:

$\boldsymbol{x}_i$ - input vector of learning data;
$d(\boldsymbol{x}_i)$ – expected response;
$\boldsymbol{w}$ – scale vector;
$(\xi_i, \xi_i^{'}), (\alpha_i, \alpha_i^{'})$ - complementary variables and the corresponding Lagrange multipliers;
$C$ - regularization constant;
$K(\boldsymbol{x}_i, \boldsymbol{x}_j) = \varphi^T(\boldsymbol{x}_i)\varphi(\boldsymbol{x}_j)$ - kernel function;
$b, \gamma$ - polarization and error tolerance function.

After solving the problem (5) the output network signal is described by the equation:

$$y(\boldsymbol{x}) = \sum_{i=1}^{N_{sv}}(\alpha_i - \alpha_i^{'})K(\boldsymbol{x}_i, \boldsymbol{x}_j) + b \quad (8)$$

$N_{sv}$ - number of supporting vectors.

A *LIBSVM* library was used in the study (Chih-Chung & Chih-Jen, 2011), which solves the task of the dual problem. Its use requires the designation of hyper parameters values C and γ, and an additional two parameters necessary to perform calculations of *p-value* and *k-fold* (Chih-Chung & Chih-Jen, 2011).

## 4. Testing of Machine Learning Methods

### 4.1 Determining Optimal Parameters

The simulation involved calculations using the Cross-Validation (*CV*) method to check the stability and accuracy of obtained results. In the first stage, the simulations were carried out by dividing the data set (*Data I*) into *k-fold* subsets (learning and testing). Calculations were carried out for each of them with the given parameters of the kernel function in order to minimize the problem of network overlearning and better generalization of knowledge. For the needs of simulation radial kernel function form was assumed (Osowski, 2006; Chih-Chung & Chih-Jen, 2011). For each division, from *2-fold* to *20-fold* a set of parameters was prepared:

- $log_2(C)$ from *-10* to *10* with step *+1*;
- $log_2(\gamma)$ from *-10* to *10* with step *+1*;
- *p* from *0,02* to *0,36* with step *+0,02;*

The *p* parameter is associated with the use of an algorithm designed for solving the dual problem (5) in the *LIBSVM* library (Chih-Chung & Chih-Jen, 2011). For the evaluation of parameters obtained in the *CV* process, in each *k- fold* division test, data sets from the same mining region was used (*Data II*). The value of mean squared errors (*MSE*) and squared correlation coefficient ($R^2$) both for the set *I* and *II* were designated. Figure 1 presents the obtained values of average errors *MSE* (solid lines), and the standard deviation of these results (dotted lines) resulting from the *k-fold* divisions. Intersection of curves for the value p=0.1 is visible (Fig. 1). In the interval $p\in(0;2)$ a more precise calculation of the designation of the value of squared correlation coefficient $R^2$ was made (Fig. 2). Stabilization of results begins with a p-value of *0,09*. It was further noted, that the *MSE* error values depend very little on the distribution of the learning set. As an example the *MSE* errors depending on the *k-fold* of the division of learning set at a fixed value of p=0,1 were presented (Fig. 1).

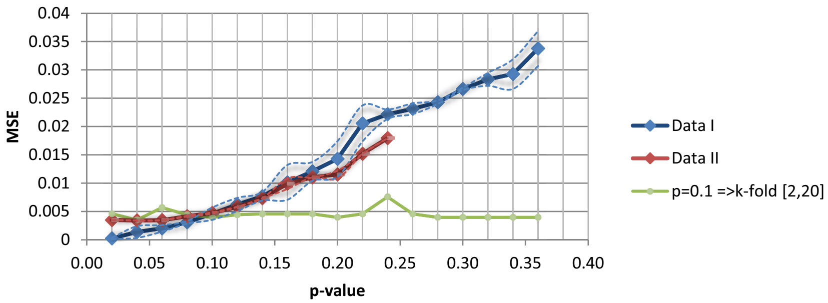


Fig. 1. Dependence of *MSE* error from the *p* parameter

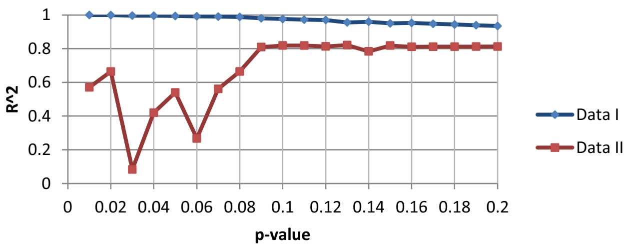


Fig. 2. Dependence of squared correlation coefficient $R^2$ on the *p* parameter

The following charts show the optimal distribution of hyper parameters ($C, \gamma$), depending on the value of the $p$ parameter. The average values and the standard deviations of amount resulting from the *k-fold* divisions of set $I$ were designated too. The graph (Fig. 3) shows a periodic waveform of the $C$ parameter and approximately linear dependence of the $\gamma$ parameter from the $p$ amount. They take into account previous observations relating to the $p$ parameter and a multiplicity of set division, and optimum parameters for the first stage as mean values from the graph (Fig. 2) were established respectively:

$$log_2(C)=5,7 \qquad log_2(\gamma)=-3,2$$

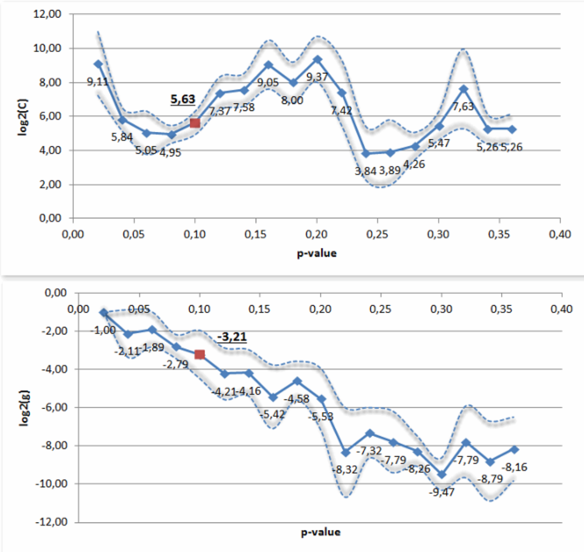


Fig. 3. Distribution of $C$ and $\gamma$ hyper parameters from p parameter

### 4.2 Execution of Cross - Validation Process

The analysis conducted in the first stage allowed to establish the $p$ parameter value at 0,1 as the optimal one, and to assume for further analysis the default *5*-fold division of a set, due to negligible dependence on the results from the degree of division of a set. In the second stage of the analysis, a calculation for so accepted values was performed using the *CV* method obtaining optimal values of hyper parameters $C$ and $\gamma$. *MSE* error distribution and correlation of learning and testing $R^2$ data for optimal parameters is shown in Figure 4. After approximate determination of position of a minimum of error function, the calculation process was concentrated in this region (Fig. 4). In *Cross-Validation* procedure an optimal $C$ and $\gamma$ values were achieved:

$$log_2(C)=5,0 \qquad log_2(\gamma)=-2,9$$

The average amounts of $C$ and $\gamma$ obtained in the first stage of analysis for $p=0,1$ are indicated in Figure 3:

$$log_2(C)=5,7 \qquad log_2(\gamma)=-3,2$$

The calculations were made for these two hyper parameters and the results presented in the table and the histogram of obtained differences (Fig. 5). The results in the case of mean squared error *MSE* vary by about *0,00002* in the case of squared correlation coefficient $R^2$ by *0,009%*. The number of support vectors $N_{SV}$ in both variants is similar, and amounts to *66* and *65* from *701* learning sets respectively.

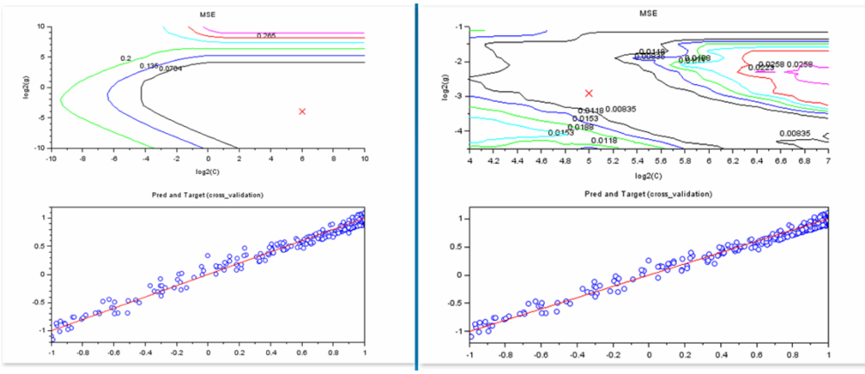


Fig. 4. Distribution of MSE error and correlation of results in the Cross-Validation process

| *p=0,1* <br> *k-fold=5* | *Stage I* <br> $2^C$*=5,7* $2^\gamma$*=-3,2* | *Stage II (CV)* <br> $2^C$*=5,0* $2^\gamma$*=-2,9* |
|---|---|---|
| *MSE* | *0,00438* | *0,00436* |
| $R^2$ *[%]* | *97,629* | *97,638* |
| $N_{sv}$ | *66* | *65* |

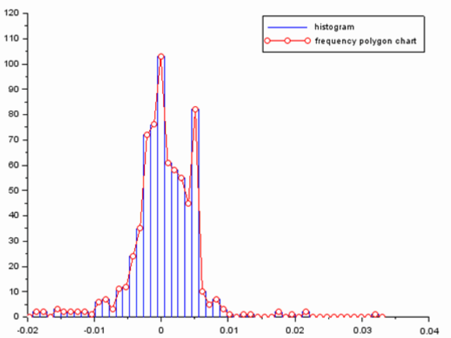


Fig. 5. Comparison of the results of calculations from stage *I* and *II* and the histogram of differences between the stages

## 5. Summary and Conclusions

Support *Vector Machines Method* is one of the methods of artificial intelligence that can be used for regression tasks. Adequate selection of the method parameters allows generalization of acquired knowledge in the learning process. The SVM method has not been previously used in the drainage displacements issues, therefore it was necessary to analyse the selection of optimal parameters. General conclusions are as follows:

- method of data division into subsets does not significantly affect obtained mean squared error *MSE* (Fig. 1);
- above value of *p=0,1* an acceleration of accumulation of *MSE* error occurs (Fig. 2);
- below the value of *p=0,09* the problem of network overlearning can be noted (Fig. 2);
- with assumed value of *p=0,1* selection of parameters using *CV* method (Fig. 4) and the one obtained from the first stage of analysis (Fig. 3) give similar results final (Fig. 5);
- due to the complexity of calculations with different multiplicity of division of learning set, we can assume the default *p* and *k-fold* parameters of Cross-Validation method to determine *C* and *γ* hyper parameters.

Presented research results represent the great potential of the *SVM* method and the possibility of its use in drainage subsidence problems that have been observed in the areas of deep mines extracting minerals. Work in this field will be continued within the framework of the project "*New algorithm for modelling of* *land surface subsidence due to the rock mass drainage*" at the AGH University of Science and Technology in Cracow in Poland in relation to other solid raw materials exploited in Poland.

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# Desulfurization of High Rank Coal (TKI-Husamlar, Mugla, Turkey) using Humic Acid

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**ABSTRACT:** Oxides of sulfur created when coals are burned have been recognized as a real threat to both the ecosystem and human health with regard to the environmental aspects. Turkish lignites usually contain a high percentage of ash and both organic and inorganic sulfur. The ash and sulfur contents can be reduced using physical-chemical and chemical methods. Desulfurization of a high rank coal (TKI-Husamlar, Mugla, Turkey) with total sulfur of 3.65% was extracted using humic acid. Effects of reaction time and temperature on the removal of sulfur were investigated using 10%wt. humic acid. The leaching reaction time was ranged from 15 to 60 minutes with an interval of 15 minutes. It was determined that the highest sulfur removal (57.70% of total sulfur) with 17.78% ash removal and 81.15% coal yield were achieved at 90$^{o}$C with 45 minutes of reaction time using 10%wt. humic acid with -1 mm high rank coal.

Keywords: Coal; Desulfurization; Humic acid; leaching; sulfur; ash removal; coal yield

## INTRODUCTION

Coal is an significant source of energy in the world (Jorjani, 2004). It is not a clean fuel since it contains ash and sulfur. $SO_x$ as a pollutant are a real threat to both the eco-system and to human health (Ambedkar, 2011).

Sulfur levels in Turkish coals are variable and changed from 0.5% to approximately 9% (Özbayoğlu, 1999). Sulfur in coal occurs in two forms; organic that is chemically bonded with carbon; and inorganic (pyrite is the most common form of inorganic) (Golshani et al., 2013). Sulfur cause environmental or process problems in its usage cycle. One of the major pollutants is the sulfur which produces hydrogen sulphide and sulfur dioxide when the coal is burned. Acid rains are after effects of these gases.

Sulfur content of the coal can be reduced by applying different techniques. Leaching is one of the common methods employed for coal desulfurization. Coal is mixed with acid or alkali and its sulfur is extracted while being heated or stirred (Abdollahy et al., 2005). Different types of reagents are used in the sulfur removal from the coal.

Humic acid is a principal component of humic substances which are the major organic constituents of soil (humus), peat, coal, many upland streams, dystrophic lakes and ocean water (Stevenson, 1994). It is not a single acid; rather, it is a complex mixture of many different acids containing carboxyl and phenolate groups. Humic acids can form complexes with ions that are commonly found in the environment creating humic colloids (Yamauchi et al., 1984). It has a high cation exchange capacity and not soluble in water under acidic conditions (pH < 2) but it is soluble at higher pH values. Furthermore, the humic acid has a binding capability of several cationic compounds such as Ca, K, Na, Mg, Zn and Fe. As a result of the interaction of the humic acid and coal, inorganic matter in the coal could be removed with adsorption.

Ratanakandilok et al. (2001) applied leaching with methanol/water and methanol/KOH. Methanol/KOH enhanced the desulfurization process in which the inorganic and organic sulfur were removed preferentially. KOH addition was improved the sulfur removal. Desulfurization was about %58.

 
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Alam et al. (2009) studied for desulfurization of coal to remove total sulfur and ash using a combination of flotation and leaching with potassium hydroxide/methanol. The total sulfur and ash contents were reduced by 82.50% and 82.34%, respectively.

Longjun et al. (2012) investigated the effects of grinding and pre-treating with nitric acid on the inorganic sulfur content of coal and also desulfurization of organic sulfur using propylene-glycol-KO on pre-treated coal with nitric acid. The desulfurization of inorganic sulfur and total sulfur was 99% and 70%, respectively.

In this study, humic acid was used to study the desulfurization of Husamlar-TKI high rank coal at medium temperature. Reaction variables included reaction temperature and reaction time. The effect of each variable on the desulfurization of the coal was evaluated by measuring the sulfur content, the ash and the coal yield.

## Materials and Methods

### Materials

The bulk coal sample (approximately 100 kg) was collected from all active stopes in Husamlar-TKI coal mine in Mugla, Turkey. The sampling techniques similar to those of Jones riffles and conning and quartering methods were adopted and representative samples were prepared for further studies. Proximate and ultimate analysis of representative sample has been carried out according to standard methods. The total sulfur of coal sample was measured using an ELTRA CS 580 model instrument. After delivery to the laboratory, the sample was left to dry at room temperature. The air-dried sample was crushed to -1 mm size then it was dried in an oven at 105$^{o}$C for 3h to use at the experimental work. Tables 1-2 present the proximate and ultimate analyses of the coal sample.

**Table 1. Proximate analysis of coal sample (dry basis).**

| Component | (wt.%) |
|---|---|
| Ash | 39.81 |
| Volatile matter | 40.37 |
| Fixed carbon | 19.82 |
| Total | 100.00 |

**Table 2. Ultimate analyses of coal sample (dry basis).**

| Component | (wt.%) |
|---|---|
| Carbon | 32.21 |
| Hydrogen | 4.43 |
| Nitrogen | 1.64 |
| Oxygen | 18.26 |
| Sulfur | 3.65 |
| Ash | 39.81 |
| Calorific value (kcal/kg) | 2740 |

**Methods**

In this research, leaching experiments were performed in two stages. The first stage of desulfurization experiments were carried out in a 250 ml glass reactor equipped with a thermocouple, pressure gauge and vented valve. For each experiment, 10 g of dried coal and 40 ml of humic acid/water solution were loaded into the reactor and the heater and stirrer were turned on at ambient pressure. At the end of each experiment, the reactor was rapidly cooled and the residue (the leached coal) was collected by filtration, washed trough with distilled water and dried in an oven at 105°C for 3h. The raw and leached coals were analyzed for proximate and total sulfur. A series of experiments was carried out to determine the effects of temperature and reaction time.

For the total the sulfur reduction (I), coal yield (II) and ash reduction (III) the following equations were used;

$$\%\, sulfur\ reduction = 100\ (x_1 - x_2\,(m_2/m_1))/x_1 \qquad \text{(I)}$$

$$\%\, coal\ yield = 100\ (m_2/m_1)$$
(II)
$$\%\, ash\ reduction = 100\ (y_1 - y_2\,(m_2/m_1))/y_1$$
(III)

where;

$m_1$ = weight of the original sample (g)
$m_2$ = weight of the sample after leaching (g)
$x_1$ = sulfur percentage in the original sample (%)
$x_2$ = sulfur percentage in the sample after leaching (%)
$y_1$ = ash percentage in the original sample (%)
$y_2$ = ash percentage in the sample after leaching (%).

## Results and Discussion

Humic acid (10%wt) was used as solvent under the reaction conditions of 24±2°C and 15-60 minutes to study the sulfur removal at 20% solids. The effect of the reaction time on desulfurization of Husamlar-TKI high rank coal is presented in Table 3. The sulfur removal increases to 36.43% within 45 minutes. But, it does decrease significantly after 45 minutes. These results indicate that the efficient ash removal which gives a high coal yield and total sulfur reduction is rapid in 45 minutes but the efficiency decreases after 45 minutes of reaction time. Therefore, 45 minutes of reaction time was accepted as the appropriate time.

The effect of the reaction temperature on desulfurization of Husamlar-TKI high rank coal with 10% humic acid at 45 minutes of reaction time is presented in Table 3. The results showed that total sulfur reduction increases with temperature and reaches a maximum value at 90°C. The temperature has a positive effect on desulfurization and it can be said that the higher temperature not only promotes the decomposition of humic acid but also increases the reactivity of the coal and the sulfur.

When the reaction temperature or time increases, the rate of sulfur removal improves which gives a higher selectivity of desulfurization, but also leads to more coal degradation and solubilization resulting in reduced the yield of the desulfurized product. Therefore, it is very important to balance the amount of sulfur removed and coal dissolved for estimating the cost of the desulfurization process where the high concentration is not recommended (Ratanakandilok et al., 2001).

**Table 3. Effect of reaction time on desulfurization of coal leached with 10%wt. humic acid and 10 g coal at 24±2$^{o}$C.**

| Time (minutes) | Sulfur reduction (%) | Ash removal (%) | Coal yield (%) |
|---|---|---|---|
| 15 | 32.05 | 12.26 | 92.13 |
| 30 | 31.23 | 11.27 | 93.25 |
| 45 | 36.43 | 10.12 | 93.25 |
| 60 | 30.13 | 7.98 | 87.64 |

**Table 4. Effect of temperature on desulfurization of coal leached with 10%wt. humic acid and 10 g coal for 45 minutes.**

| Temperature ($^{o}$C) | Sulfur reduction (%) | Ash removal (%) | Coal yield (%) |
|---|---|---|---|
| 30 | 46.33 | 22.07 | 86.12 |
| 60 | 46.86 | 21.30 | 83.16 |
| 90 | 57.70 | 17.78 | 81.15 |

## Conclusions

In the present work, the desulfurization of Husamlar-TKI, Mugla, Turkey coal with 10%wt humic acid and 10 g coal at 90$^{o}$C appeared to be an optimum condition removing about 58% of total sulfur with 18% ash removal at a moderate time (45 minutes). The results suggest that humic acid leaching has a potential for a sufficient desulfurization of lignites with high sulfur content.

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# Removal of Sulphur from Sorgun Yozgat Lignite Coal with $HNO_3$ Leaching

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**ABSTRACT:** Coal desulfurization prior to usage is a pre-processing in order to achieve clean fuel and reduce environmental impacts such as acid rain. The main purpose of this study is desulfurization of lignite coals obtained from Yozgat- Sorgun is the main purpose. In the first stage, untreated lignite samples were leached with $HNO_3$ acid in aqueous solution in different operation conditions; solid ratios, acid concentrates and times. Obtained desulfurization results are compared and optimum conditions are determined for direct $HNO_3$ leaching. Applying of the leaching technique significantly reduced the time and cost required for the experimental investigation. The final sulphur content results of direct $HNO_3$ leaching method are compared to the before $HNO_3$ leaching method. Accordingly, the optimum condition was obtained as, temperature: 60 °C., $HNO_3$ acid concentration: 3 M., and stirring rate: 300 rpm. The total sulfur and coal yield after leaching at optimum condition reached to 57.44% and 83.41% respectively.

*Keywords: Desulfurization; Leach; HNO3; Lignite coal; Solid rate; Coal yield; Pre-processing*

## INTRODUCTION

Coal is used as an economical energy resource in the world. One of the disadvantages associated with the use of coal for energy in recent years is sulfur oxides and hydrogen sulfide emissions in the environment. These problems in the industry to control the sulfur emission, has led to the use of fluidized bed combustion and flue gas desulfurization process. On the other hand, problems with the current technology which are costs, efficiency, applicability and waste disposal, have causes an increase in research of potential solutions to this problems such as pre-combustion desulfurization process (Coal power for progress, 2000; Jorjani at al., 2004).

Generally coal includes approximately more than 10% amount of different impurities like sulfur, silicates, carbonates etc (Bolat et al., 1998; Alam et al., 2009). Sulfur and its compounds found in coal cause undesirable adverse effects on agricultural products, corrosion of metal buildings and breathing problems of humans and animals. Thus, in many applications, before using reducing sulfur and mineral substances (ash) contained in coal is essential and prior (Alam et al., 2009; Mukherjee et al., 2001).

Removal of sulfur in the coal and demineralization can be obtained both by physical and chemical methods. Some demineralization may be obtained by simple physical operations based on the differences in mineral physical characteristics and carbonaceous parts of the coal. The physical processes are cost effective procedures yet it is not effective to separate finely dispersed minerals and those connected the coal structure. On the other hand, Chemical process that includes dissolving process of the inorganic component in different solvents is known as solution for achieving clean demineralized and desulfurized coal (Mukherjee et al., 2001). In many authors has reported coal can be treated with aqueous or fusewd sodium hydroxide alone, or followed by mineral acids for obtain demineralization and removal of sulfur in the coal (Bolat et al.,1998; Araya et al., 1981; Kara et al., 1988).

In the coal industry, desulfurization is one of the most significant topics. Sulfur in the coal is composed of organic and inorganic forms. Fundamentally, coal treatment methods are listed under category of the physical and chemical treatments. During physical process, like flotation or magnetic

 
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separation inorganic sulfur, which exists mainly in two ways disulfides (pyrite and marcasite) and sulfate (mainly calcium, iron and barium), is treated. On the other hand, during chemical process, organic sulfur could be treated (Coal power for progress, 2000; Jorjani at al., 2004).
The organic sulfur is usually formed in form of thiols, sulfides, disulfides, thiophenes and cyclic sulphides (Calkins, 1994). The most recent researches show, the iron could be bonded with secondary organic sulfurs in aromatic and aliphatic systems (Baruah et al., 1998)

Alam, 2009; studied desulfurization of Tabas Mezino coal with $HNO_3$/HCL leaching. It was found that in optimum operating conditions which were 30% acid concentration, 1000 rpm stirring speed, 90 min treating time and 90 °C temperature, nitric acid was effective based on the nitric acid leaching test results. Based on test results, in optimum conditions after flotation and leaching, total sulfur and ash removal was achieved to 75.4 and 53.2% respectively. Compared to the previous works, this is a considerable result.

Jorjani, 2004; examined removal of sulfur in the coal by the effects of grindind and pre-terating with nitric acid on the inorganic sulfur content of coal. It was used propylene-glycol-KO for desulfurization of organic sulfur in coal pretreated with nitric acid. Based on the test results, it was found after nitric acid pretreatment, the desulfurization rate of inorganic sulfur raised to over 99% and the total sulfur desulfurization rate to over 70%.

It is very important to achieve low cost, efficient and applicable desulfurization method. Due to the cost of chemical methods, leaching methods of nitric acid was used in the present research for desulfurization of Yozgat- Sorgun coal.

## Materials and Methods

### Materials

The coal samples collected from Yozgat-Sorgun, Turkey. After delivery to the laboratory, sample was left to dry at room temperature. The dried sample was crushed to -0,500 mm size for desulfurization tests. The fraction < 0.106 mm is selected. Table 1 and table 2 presents proximate and ultimate analyses of the coal sample. Analyzes were performed by MTA.

**Table 1. Proximate analysis of Yozgat-Sorgun coal sample.**

| Component | (wt.%) |
|---|---|
| Ash | 8.05 |
| Moisture | 14.17 |
| Volatile matter | 34.63 |
| Fixed carbon | 43.15 |
| Total | 100.00 |

**Table 2. Ultimate analyses of Yozgat-Sorgun coal sample.**

| Component | (wt.%) |
|---|---|
| Carbon | 61.35 |
| Hydrogen | 3.62 |
| Nitrogen | 1.42 |
| Oxygen | 8.16 |
| Sulfur | 3.84 |
| Ash | 9.38 |
| Calorific value (kcal/kg) | 6394 |

**Methods**

A series of leaching tests were carried out at 5%, 10% and 20% pulp density (w/v) using the coal sample in 500 ml stainless steel reactor equipped with a thermocouple. Each experiment was carried out 200 ml of $HNO_3$. The experiment was performed at $HNO_3$ concentration of 1M, 2M and 3M, at temperature of 60°C, at reaction times 30, 60 and 90 min., at stirring speed of 300 rpm.

After leaching, the result solid filtered and washes regularly with distilled water. The remained coal was dried. The calculate the total the sulfur reduction (I) and coal yield (II), the following formula was used;

$$\%\,Coal\ yield = 100(m_2/m_1) \qquad \text{(I)}$$

$$\%\,sulfur\ reduction = 100\,(x_1 - x_2\,(m_2/m_1))/x_1 \qquad \text{(II)}$$

Where:

$m_1$= weight of the original dried sample (g)
$m_2$= weight of the original dried sample after leaching (g)
$x_1$= sulfur percentage in the original samples (%)
$x_2$= sulfur percentage in the original samples after leaching (%).

## Results and Discussion

The best result of experiments was determined at 20 % solid rate, at 90 min., at 3M acid concentration, at 60°C . The desulfurization results, it can be seen that sulfur reduction ranges from 12.50 to 57.44%, coal yield from 69.80 to 95.47%, depending on the experimental conditions.

| **Solid rate** % | **Nitric Acid Concentration** M | **Leach Time dk** | **Temperature** °C | **Sulfur Reduction** % | **Coal Yield** % |
|---|---|---|---|---|---|
| **5** | 1 | 30 | 60 | 35.80 | 85.28 |
| **20** | 1 | 30 | 60 | 12.50 | 91.65 |
| **5** | 3 | 30 | 60 | 55.69 | 86.86 |
| **20** | 3 | 30 | 60 | 49.97 | 95.47 |
| **5** | 1 | 90 | 60 | 17.58 | 90.01 |
| **20** | 1 | 90 | 60 | 53.89 | 90.34 |
| **5** | 3 | 90 | 60 | 16.64 | 81.24 |
| **20** | 3 | 90 | 60 | 57.44 | 83.41 |
| **10** | 2 | 60 | 60 | 40.49 | 69.80 |
| **10** | 2 | 60 | 60 | 24.20 | 86.00 |

## Conclusions

Nitric acid was effective on desulfurization of Yozgat - Sorgun coal and the test results of nitric leaching revealed the following as optimum operating conditions: acid concentration of 3M, stirring speed of 300 rpm, treating time of 90 min and temperature of 60 °C. The findings indicated that acid concentration, with a high contribution, had the most dominant effect on the desulfurization performance, followed by treating time and solid rate. During the desulfurization process, the total sulfur content was reduced by 57.44% and coal yield 83.41% under the optimum conditions.

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# Alternative storage of ashes to reduce the impacts to the environment

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**Abstrakt**

Presently, the power production by coal incineration connected with the production of huge amount of ashes as a by-product. The production of gashes is approximately 100 million tonnes a year integrating this product into the global problem. The ash is stored in selected spaces, on stacks, occupying valuable soil. Ash storing on the surface allow the maceration and windy erosion increasing the load even in vertical directions. Nowadays, the ash is started to use as an additional material for new products causing to recycle and to process of huge amount of waste. This does not cover the ash production and so it is necessary to create new deposits which might load the environment in the scope known until now.

**Key words:** black coal, brown coal ashes, coal incineration, environment, deposits, alternative storge,

**Introduction**

Ash as a solid by-product of combustion takes the form of slag, cinder, ash, fly ash and fume. Currently, the world produces 100 million tonnes of fly ash annually and its disposal is increasingly becoming a global problem.

Fly ash needs to be stored in designated areas (dumps, storage sites, tailing ponds), which occupy land that could otherwise be used to serve an agricultural function. All fly ash related costs are incorporated into the price of electricity and thus the significantly affects the whole economy.

Fly ash that is stored above ground is subject to being windblown increasing dust pollution which poses health risks and harms vegetation. These adverse effects not only exaggerate the quantity of secondary dust in the immediate vicinity many times over, but also contribute to the degradation of vegetation and soil and negatively affect agricultural and forest productivity.

**How fly ash is formed**

Ash material consists of minerals that are finely dispersed in a carbonaceous matter. Primary ash material is already present in the original mass, while secondary ash material permeates coal deposits via flood applications or infiltration. Secondary ash material consists of silicates, silica's, carbonates, and sulphur. Ash material is defined as a solid residue obtained by burning a solid fuel in an oxidizing atmosphere at 800 ° C.

**Combustion**

During the combustion process, changes occur in the ash material that depends on the properties of the ash material, the combustion technology, the temperature and atmosphere of the furnace. When these changes occur crystalline bound water of ash is released even at low temperatures. The initial reactions occur during the heating up phase, which generate gaseous products while the ash material creates new matter, which is the basic component of the ash. At lower temperatures endothermic reactions take place, in which the carbonate decomposes into carbon and carbon dioxide. These reactions occur at 610 - 900 ° C. Ash material of some coals containing alkali salt begin to smelt at relatively low temperatures between 800 - 900 ° C. (Spišák, 2011)

 

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**Black coal**
The combustion of black coal in a smelting furnace occurs at temperatures of 1 400- 1600 ° C. At this temperature, it is melted to a molten ash. The predominant particle shape is spherical. Fly ash from the flue gases is collected by electrostatic precipitators, with effectiveness in the rangeof 99.7 to 99.9%.
**Brown Coal**
Brown coal is burnt in granulation furnaces at temperatures ranging between 1100-1300 ° C. This type of incinerated ash contains particles with a predominantly cellular structure. Combustion in fluidized bed furnaces is carried out at 800-850 ° C. Fly ash due to lower temperatures is created out of the skeletons of original coal mineral grains from the remnants of unburned coal.
**The classification of ash**
For solid post combustion residues, are classified as:

- Slag - minerals have been subjected to smelting - glassy, metallurgical
- Cinder - minerals during the burning process is softened and then baked to produce a porous material
- Ash - minerals that did not smelt or soften and remained loose
- Fly ash - the fine particles of solid residues which is created by the flow of flue gas
- Fume- the finest particles that escaped to the chimney

For the operation of various types of solid fuel furnaces, the smelting temperature of all the components of ash is important. The properties of fly ash depend on its origin, properties of the coal, and combustion process. The utilization of fly ash depends on their physical and chemical properties and petrographic composition, determined by the second combustion of black coal, brown coal and lignite. Fly ash is a heterogeneous material, composed of particles with different physical, chemical and mineralogical characteristics.
**Chemical Properties**
The chemical composition of the coal can to a certain extent be estimated from the composition of the fly ash released during the combustion process. During the combustion process, the chemical composition and properties of coal combustion change. After combustion, the residue ash contains mineral components that are either original or newly created. Ash material in the carbonaceous matter is partly linked to the flammability of the carbonaceous matter, how finely dispersed it is whether the carbonaceous matter can be separated after breaking up. Fly ash generated in thermal power plants are characterized by their physical, chemical and mineralogical properties of which the most significant are unburnt residues of coal, size distribution, surface area, density, content of microspheres, representation of the crystalline and amorphous phase, representation of magnetite phase, a specific dust resistance of the particle, content of major and minor chemical elements.
From the results of the tests it is evident that lignite fly ash is non-porous and therefore cannot absorb water. Black coal fly ash from smelting furnaces is slightly absorbing to absorbing. Sorption is mainly caused by the presence of residues of unburnt coal that have a large porous surface.
Electrical conductivity of solids are categorised as conductive (metals), semiconductors and insulators. Fly ash can fall into anyone of these categories and can exhibit surface conductivity. The properties of fly ash properties are affected by:

- Petrographic or maceral composition and mineral admixtures of coal,
- Processing of coal during the mining extraction process or at the power plant,
- Conditions of combustion

**Uses of fly ash**
Globally, there is insufficient use for fly ash is and depending on the country only between 1-40% of fly ash produced is processed. Unprocessed fly ash needs to be adjusted for storage so that it can be used in the future. Fly ash and slag mixtures are generally stored in tailings ponds.
**Tailing ponds**
Tailings ponds that store fly ash and pore water using self-weight sediment induce risks on the subsoil. The continuous raising of these dikes causes sediment to rise increasing the risk that creates deformations.

(Taušová, 2011) Large-scale deformations may damage the dikes, drainage system and other pond facilities. Another problem with the large tailings ponds is the quantity of flowing water. Flooded hydro mixtures may leak if the hydro insulation layer is damaged. This leak can cause waterlogging of the area, and a groundwater regime change. The pond is sensitive to rainfall which leaches chemicals and risks contaminating ground and other surface waters. (Kuzevič, 2001., Horbaj 2007)
Tailing ponds expose the environment to a range of risks such as

- Occupation of agricultural land
- Placing on land that has been mined
- Located in protected areas and buffer zones. (Michalíková, 2014)

The greatest danger to the environment is the risk of a catastrophic failures and accidents. Another problem is dustiness when fine dry particles of ash are swept away by winds from exposed portions of the tailings pond. Dust forms around the cover, which is hygienically permanently harmful due to their chemical composition. Dustiness can be influenced in several ways:

- Keep the fly ash in the pond submerged below a thin water layer,
- Increasing the moisture content of fly ash by adding water,
- Increasing the moisture content of fly ash by adding chemical additives,
- Cover with a coarser material
- Cover with geotextile
- Create a vegetation cover (Michalíková, 2014)

Tailings ponds disrupt the natural character of the surrounding landscape and can only be incorporated or tolerated in the natural environment after appropriate remediation. This option requires a considerable financial cost, which does not safeguard the environment in the locality.

**Solution**

Storage of fly ash in mined areas
One option for processing fly ash waste is to store the waste inunderground shafts. Current solutions can be divided into several groups. According to the nature of the material we distinguish basic technologies:

- loose poured
- landfills
- blown
- flooded
- flooded

For loose poured piles, the heap corresponds to the basic cone angle of the material (range 40-55 °). Therefore, deposits for pouring must have a greater angle of repose than the material and typically stipulates 65 °. Vibrators may be used for the transportation of poured materials. There the fill material is poured from the gangue to the vibrators, where one of the two wings alternatively fills.
Due to the low performance of casting machines, casting fill are only used used in a very small scale, because it slows down the work. For similar reasons the use of blown fills are not significantly applied. Adjusted flotation waste may be transported to the coal face either directly by pipeline from the treatment plant via the surface drains and drainage as sand is transported to the mine, where before loading again it is slurried with water to the desired density and piped directly to the containers. (Pavolová, 2014., Pavolová 2011)
Before fly ash is transported to the underground spaces it is processed by

- Stabilizing without the ability to bind water
- Stabilizing with the ability to bind water
- Stabilized by solidification
- Zeolitification of fly ash (Michalíková, 2013)

**Processing of fly ash**

Fly ash from individual furnaces is gathered in the mechanical and electrostatic precipitators and mixed with water pumps and conveyed in the form of slag mixture to a tailings pond. Thickening of hydro mixtures is carried out in a 1:20 weight ratio of ash, slag and water. Installations for the production of stabilizer facilitate the mixing of all products from furnaces - ash-slurry-limestone / gypsum from desulfurization equipment for the creation of a harmless storable product. Stabilization is the result of a one-step process, while the installation of a fluidized bed furnace takes place in a two-step process. (Sabo, 2014)

**Zeolitification of fly ash**

Zeolitification of fly ash occurs when fly ash precipitates at the outlet of the hopper into the combustion pan while still hot (temperature of 150-140-120 ° C) which is then conveyed to wet-storage. The zeolite neoplasm mineral phillipsite exhibits high sorption properties that are often 2 - 2.5 times greater than that of natural phillipsite zeolite minerals. It contains 40-60% neoplasm of zeolite mineral and the sorption capacity varies up to 35 mg of NH4 and has an ion exchange capacity of about 30 mg.g-1. Natural clinoptilolite from Nizhny Hrabovec showed an ion exchange capacity of about 15 mg.g-1. Dry fly ash collected from the hoppers at the accredited workplace had a pH in the range of 11.2 to 12.1. - Fly ash removed from the tailings pond, which has undergone zeolitification, had a pH ranging from 8.12 to 8.81. After 24 hours the same samples demonstrated lower leaching values of between 7,47 - 8,3 pH.

The test results of tensile strength, bending strength and the strength after 7, 28 and 60 days showed that the raw materials - ash, gypsum, desulfurization, waste water and quicklime are suitable for the use of a material composition in the manufacture of various types of stabilizers. Another possibility is the exchange of added stabilizer materials that need to be stabilized. If lime is replaced with cement the stabilization becomes more expensive, but its strength characteristics when evaluating the tensile strength for bending are 2 to 3 times higher than for the stabilizer manufactured using lime.

Of the annual production of thermal power plant in Nováky ENO (SK), which produces 1 million tons of ash and slag 35% is processed for lightweight construction materials. The rest is deposited in tailings ponds that occupy 40 hectares of land and forests. This waste can be stored long-term in unused mining areas. The appropriate application of a stabilizer can achieve the processing of waste while avoiding undesirable elements from leaching into groundwater and aquifers.

**Conclusion**

Fly ash is a burden on the environment around the landfill. The surface layer dries out due to wind erosion, increased dust pollution, threatening the vegetation. The utilization of fly ash as a material that can be incorporated as a component of building materials, is limited by its physical, chemical and mineralogical characteristics. Because of the excess traces of unburned coal it only has limited use in the construction industry. Unburnt residues of coke in black coal behave like an inert material. Fly ash improves the compact ability of soil when mixed in appropriate ratios to form powders, and embankments. For incorporation into the soil of the ash portion, mitigate the degree, retention of soil moisture as a result of the high surface area and porosity, which is expressed by capillary force. Properties ashes are the subject of extensive research. Their recoverability is based on an appropriate application of lessons learned.

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# Novel environmental management application for lignite.

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**Abstract**

Mining and mineral processing operations are usually generating huge amount of tailings. These tailings often gives lots of maintenance work in order to protect environmental resources from heavy metals slowly leaching out of tailing are due to Acid Rock Drainage (ARD) process. Handling leakage could be active or passive such as using of Permeable Reactive Barriers (PRB). Using PRB technology water is not extracted, however PRB's are built underground in the way of underground waters where contaminants are extracted or conversed into harmless components on the surface of the PRB matrix material (Madarász, Tóth, 2010). One of the most important criteria for using PRB to select or design the proper material for matrix material on which the contaminant adsorbs, precipitate or degrade. Water is able to going through the barrier and on the downstream side, clarified water is going to be released PRB's are used worldwide treating underground contaminations, using different type of matrix materials. All generally used matrix material has its advantages and disadvantages such as selectivity, sensitivity of precipitation generation or not satisfying hydraulic conductivity. Nowadays cheap and effective matrix material which is ignorant to pore plugging caused by precipitation within the PRB could be one solution to the demands of the industry. The authors have been carried out an investigation at University of Miskolc on lignite as high humid acid containing material for determining the ability of heavy metal adsorption and pore plugging resistance. Moreover, using hydraulic and chemical simulation methods, complete PRB dimensioning protocol based on new lignite matrix had been introduced. Adsorption capacity of tested lignite samples were determined for toxic transient and heavy metal cations. Static and dynamic adsorption tests had been carried out parallel in order to investigate adsorption kinetics, while hydrodynamic and transport modelling also had been done. As a result of the research, a complex design protocol for lignite based PRB system had been established of which are being discussed within the frame of this paper.

**Keywords: Lignite, Permeable reactive barrier, Environmental management, Cation adsorption, Mineral processing**



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## Introduction

The so called „Pump and Treat" technologies are the most generally used methods treating contaminated underground water sources. Using water extraction and intrusion methods contaminated water is extracted, than treated on site and the clear water is introduced back to the proper layer. It is always an uncertainty to ensure contaminated water is intercepted from the total contaminated area and there is no contamination remains in separated plumes during this active operation. According to the method it is an active method which means continuous operation and monitoring, not rarely at high costs in months or years (J. Bőhm, Á Debreczeni, I. Gombkötő, 2003). This is one of the reason for non active technologies, such as permeable reactive barriers (PRB's) are making increasing progress. Using PRB technology water is not extracted, however PRB's are built underground in the way of underground waters where contaminants are extracted or conversed into harmless components on the surface of the PRB matrix material (Madarász, Tóth, 2010). One of the most important criteria for using PRB to select or design the proper material for matrix material on which the contaminant adsorbs, precipitate or disintegrate. Water is able to going through the barrier and on the downstream side, clarified water is going to be released (Filep et al, 2002).

In generally, regular PRB's, Zero Valent Iron (ZVI) and activated carbon (AC) matrix materials are used. These materials are able to treat waters contaminated organic and non – organic compounds, such as hydrocarbons, chlorinated hydrocarbons, pesticides, heavy metals and some nitrate and sulphate compounds, and for contaminated post mine and mineral processing running waters as well, where ARD causes high heavy metal and sulphate contaminations (Lakatos, Szabó, Csőke, 2007). Our research team uses matrix material with high humic acid content, of which has a price at least one magnitude lower than AC, and however its contamination retention capability (CRC) is lower than AC has, our laboratory experiments are prove that cheaper materials has CRC approx. 30% of AC has. AT the end of life, instead of expensive regeneration, energetic application and introducing of new matrix portion is easily possible. If PRB's become cost effective, cheap water treating technologies are going to be introduced at in some cases 50% lower specific costs than Pump and Treat technologies has (Szűcs, Madarász, 2006). Another innovative approach is that the matrix materials inserted into easily removable cages. These cages are possible to change anytime; therefore old matrix can be removed at no costs or significant efforts.

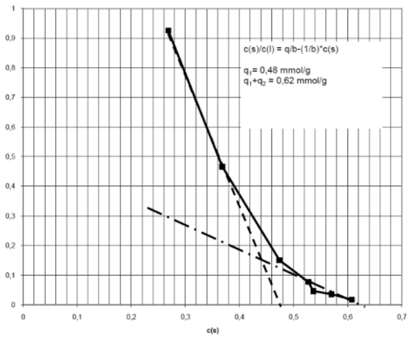
## Development Of Matrix Material Based On Lignite

Within the frame of a research project new generation permeable reactive barrier system had been developed. The reactive matrix is lignite as high humid acid containing material with decent adsorption potential. Application of easily removable multi matrix layers is also one of the novelties of the system. During matrix development, lignite samples of four different particle sizes were used (0/5 mm; 0.125/0.25mm; 0.25/0.5 mm and 0.5/1mm) mixing them with sand with the same particle size fraction at different portions. Knowing the ions in exchange position was determined, because during the process some ions could get advantage on other which should be important to adsorb, and in the otherwise different ions could leach into the groundwater from the matrix. In order to determined the importance of mineral processing techniques are being used, we determined the ion contents of the tail rocks (Ca, Mg, Na, K, Al, Fe). As a result, iron and transition metals are strongly adsorbed in lignite surface which could prevent toxic ions to adsorb. If there is the possibility for this process, application of secondary matrix layer is necessary to remove iron first.

Sorption capacity of the lignite had been also determined. For this, using of Langmuir isotherms is widely accepted (Lakatos, Snape, Ulmanu, 2003). The isotherm describes the sorption process not only mathematically, but parameters like sorption capacity and sorption strength also can be determined. For this purpose, Langmuir isotherm can be used.

$c(s) = q*(c(l)/(c(l)+b))$.

For example linearized Langmuir isotherm for Cu(II) sorption on lignite (Bükkábrány, Hungary) can be seen in fig. 1. and 2.



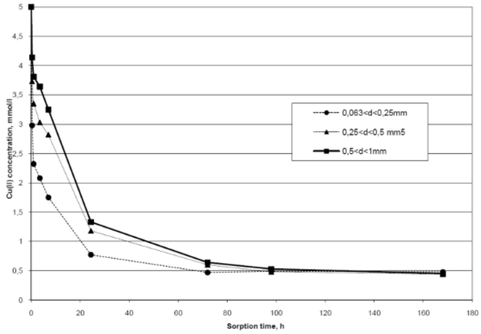




Figure 1 and 2 .linearized Langmuir isotherm for Cu(II) sorption on lignite (Bükkábrány, Hungary) - Sorption velocity of Cu(II) at lignite with different particle size.

Linearized Langmuir Isotherm is one of the easiest to use for this approach.

$$c(s)/c(l) = q/b-1/b*c(s). \quad (2)$$

According to the data visualisation, it is possible to determine of which interval Langmuir Hypothesis can be used. The data revealed, that only “two layer” Langmuir equation can be used, which provide information about strong and weak adsorption. In order to safely dimensioning a PRB “wall” strong potential can be used (Lakatos, Brown, Snape, 2002). Results also revealed, that lignite from different origins shows only differs in weak sorption capacity and the total sorption capacity is 1 to 10 % of sorption capacity of ion exchange resins and is the same magnitude comparing it to activated carbon. The sorption velocity is low; balance is going to be reached treating 1 total pore volume of contaminated solution per four days. After this time period, balance is provided and there is no noticeable difference of sorption capacity depending on the used particle sizes.

Breakthrough curves of different ions at different matrix were also determined. Iron ions (either Fe(II) or Fe(III)) are capable to depress Cu(II), however lignite was proven to be able to adsorbs Cu(II) and other toxic heavy metal ions. According to figure 2.; velocity of the sorption is slow. Investigating it at static system approx. 100 hours was necessary for reaching equilibrium. This value is corresponding for 0.01 pore volume/ hour flow rate. This means that for PRB’s one pore volume/ 4 days is required for the equilibrium, but after that, sorption is not depending on the particle size of the matrix material. For PRB’s, choosing contact time lesser than this have sense.

Ion concentration of downstream solution of the model PRB has been also investigated. Na ion is reaching its breakthrough point after one pore volume, which means Ne ion is nat capable of sorption. There are mostly Ca ions at sorption points in Lignite, therefore because of the sorption of Cu(II) and Fe(II), Ca has higher concentration at downstream than upstream. The area under the curve of 5mM should be the same as the amount of sorbed Cu(II) and Fe(II) ions.

**Hydraulic Modelling Of The PRB Matrix And Introduction Of It Into The Environment.**

In order to prevent groundwater flow by-passing the PRB or to prevent groundwater to be dammed, hydraulic conductivity of PRB have to be larger than its environment, where it is installed. The hydraulic properties of the matrix material were investigated at a flexible permeability meter. During the experiment mixture of lignite and sand were produced (at the rate of 90/10%; 50/50%; 30/70%; 10/90%

in every examined particle size range) with keeping sorption capacity at constant rate and hydraulic conductivity had been determined. Hydraulic conductivity of the matrix variants listed in table 1 and 2. were inserted into matrixes and signed them with proper coloured codes.

Table 1. Hydraulic conductivity of pure lignite at different particle size.

| Particle size | k hydraulic conductivity |
|---|---|
| 0.5-1 mm | 1.39 10-4m/s |
| 0.25-0.5 mm | 5.3 10-5m/s |
| 0.063-0.25 mm | 5.86 10-7m/s |

Table 2. Sample table for introducing Hydraulic Conductivity has been measured at different Lignite/Sand mixtures.

| Hydraulic conductivity (k) [m/s] | | Lignite Sand mixture rate | | | |
|---|---|---|---|---|---|
| | | **90% : 10%** | **50% : 50%** | **30% : 70%** | **10% : 90%** |
| **Particle size of Lignite (d)** | 1 – 3 mm | 2.11E-04 | 8.08E-05 | 6.25E-04 | 5.44E-05 |
| | 0.5 – 1 mm | 1.40E-04 | 1.35E-04 | 1.27E-04 | 1.08E-04 |
| | 0.25 – 0.5 mm | 4.79E-05 | 5.32E-05 | 5.54E-05 | 5.83E-05 |
| | 0 – 0.25mm | 3.60E-06 | 4.55E-06 | 3.60E-06 | 4.55E-06 |

For mathematical modelling, Processing ModFlow™ software was used. PRB's can be described with η – intercept efficiency which is the percentage of the trajectories arriving to the dam actually flowing through it. Sample of result data can be seen on fig. 3.

| **η (%)** | | **Hydraulic conductivity of 0.5 m wide PRB ($k_{PRB}$ [m/s])** | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | **$5\cdot10^{-3}$** | **$10^{-3}$** | **$5\cdot10^{-4}$** | **$10^{-4}$** | **$5\cdot10^{-5}$** | **$10^{-5}$** | **$5\cdot10^{-6}$** | **$10^{-6}$** |
| **Hydraulic conductivity of surrounding soils [m/s]** | **$10^{-3}$** | 100 | 100 | 100 | 86 | 73 | 40 | 20 | 6 |
| | **$10^{-4}$** | 100 | 100 | 100 | | | | | |
| | **$10^{-5}$** | 100 | 100 | 100 | | | | | |
| | **$10^{-6}$** | 100 | 100 | 100 | | | | | |

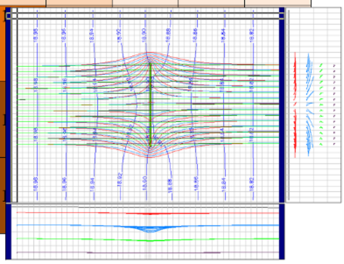


Figure 3. Sample for visualisation of η – intercept efficiency as a function of different hydraulic conductivity of the environment and the PRB

Transport modelling of break through curves using break through curves determined by the dynamic batch experiments, optimal transport parameters were identified. By this way, a reliant tool for PRB dimensioning was created which is able to predict break through time, so called matrix change cycle time as well.

**Methodology Of PRB Dimensioning.**

PRB dimensioning factors, cannot be determined alone as single factors, they affecting each other. These factors are competing in some cases (eg.: quantity of the matrix material, required hydraulic conductivity), Therefore independent factor modules and their effects on each other required a development of a well described complex dimensioning protocol. The dimensioning protocol is divided four main parts as: 1. Check list, 2. Mass Balance, 3. dimensioning by parameters, 4. Pilot scale test.

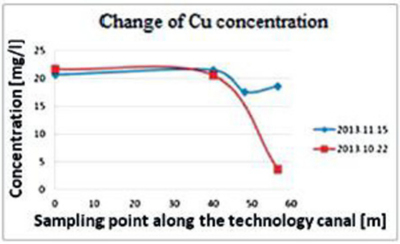
**Passive AMD treatment technology – pilot test**

One of Northern Hungary's abandoned ore mines is undergoing a complex rehabilitation program, which involves the passive acid mine drainage treatment pilot site at one of the former mine shafts. The effluent water is characterized by acidic pH level ranging between 2–4 and high dissolved metal contents (Fe(II) concentration) varying from 589–914 mg/dm3 and Cu concentration ranging up to 30 mg/dm3. The discharge is moderate, ranging between 1.5–2.5 m3/day, but even with the limited discharge it causes significant environmental load on the area. The operator of the site decided to develop a passive treatment technology and our research team offered a feasible technology as a test operation.

The designed system was based on previous research using lignite as a reactive barrier material (Bőhm et al. 2003; Madarász et al. 2011) and on preliminary lab measurements of both the hydraulic parameters and chemical functionality of the applied materials. For the adsorption of high heavy metal content of the drainage, lignite reactive material was applied. Literature studies showed that lignite has favourable properties and its adsorption capacity can reach up to 30% of that of active coal (Lakatos et al. 2007) while its cost is an order of magnitude lower. Also, the expensive regeneration costs can be eliminated if the exhausted lignite material is reused. After testing several options the team concluded that a grain size of 11–22 mm is appropriate to meet the hydraulic demand on the site test. A 56 m-long concrete test channel was installed at the site which was separated into three cascades to support the treatment technology steps. The system consisted of 3 successive steps: 1st step: 40 m-long neutralization section filled with limestone (11–22 mm grain size); 2nd step: 8 m-long section with a 1:1 mixture of limestone and lignite (11–22 mm grain size); 3rd step: 8 m-long lignite field for adsorption of dissolved metals (Figs 4). During installation of the on-site test operation, deflector walls were placed in the channel to increase the interaction time between the drainage and reactive material.

Figure 4. Materials used in the passive AMD treatment pilot test

The performance of the treatment technology was monitored by installing sampling points and on-site passive samplers at various points of the system and in the discharging stream. Water samples were analysed in laboratory for the relevant metal content of the drainage water (Al, Fe, Cu, Ca, Mg, Zn, Ni, Pb, As, Cd). Table 3 and Fig. 5 shows the change of iron and copper concentration at two sampling time and four sampling points along the system.



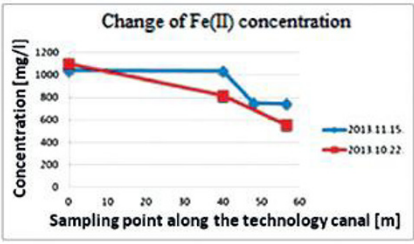




Figure 5. Change of Fe(II) and Cu concentration along the channel in two testing period

Table 3: Summary of lab results for Fe (II) and Cu

| Sampling point along the technology canal [m] | Time of sampling | | | |
|---|---|---|---|---|
| | 22.10.2013. | | 15.11.2013. | |
| | Concentration of Cu [mg/l] | Concentration of Fe(II) [mg/l] | Concentration of Cu [mg/l] | Concentration of Fe(II) [mg/l] |
| 0+000 | 21.65 | 1098.5 | 20.65 | 1039.5 |
| 0+040 | 20.55 | 816.5 | 21.5 | 1033.5 |
| 0+048 | – | – | 17.55 | 748.5 |
| 0+056.5 | 3.65 | 555.5 | 18.6 | 742.5 |

The obtained results demonstrate the decreasing concentration of both components along the treatment channel. The concentration of iron decreases through the full length of the channel, while in the case of cupper only the last lignite-based section was effective.

The background for this phenomenon is due to the increase of pH, which causes Fe(II) to be already transformed and precipitated as Fe(III) in the first half of the treatment channel. In the second half adsorption can further decrease the concentration thanks to the lignite barrier. The change in trends between the two-time periods is due to decreased performance of the treatment technology barrier. Several factors contribute to this decreasing efficiency. As Fe (III) precipitates on the surface of limestone a crust is formed, blocking the neutralizing effect of the material. This has an impact on the adsorption capacity, as the acidic pH is not ideal for the adsorption on lignite.

The described crust formulation has a negative effect on the hydraulics of the entire system as well, although it did not cause any problem at this stage of the test operation. The team had to conclude that the neutralization of the AMD discharge must be implemented with some other mechanism, and is shifting the study into a new line of research.

## Conclusions

During the development of the New Generation Permeable Reactive Barrier (NGPRB) system, it was proven that lignite is able to adsorb toxic heavy metal ions from groundwater plumes at a stable rate. It was also determined, that at certain circumstances, its sorption potential has to be defend against competitive ions using protective multi-layer shields in NGPRBs. During the development a dimensioning tool also had been introduced, and a design protocol was created Pilot test has indicated limitations of the application of which can be predicted on preliminary data, however concept has been proved in relevant environmental circumstances.

## Acknowledgements

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# Part VII
# Dewatering, Drying and Briquetting of Coal

# DETERMINATION OF RATIONAL PARAMETERS OF BROWN COAL BRIQUETTING

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**Abstract**
Brown coal is unstable during the storage and disposed to self-ignition. A great quantity of fine leads to the difficulties of transportation and the impossibility of burning in standard fire grate furnaces. The solution of a problem of underutilization of coal resources as a result of direct fuel utilization of run-of-mine coal is briquetting. New briquette composition including binder and filler was developed. Technological approaches include sizing of run-of-mine coal and briquetting of coal fine with the refinery waste and anthropogenic carbon-containing waste products of hydrolytic industry were designed. Technological and granulometric characteristics of brown coal and hydrolytic lignin were researched. Optimal parameters of coal briquetting were established. Influence of moisture content, binder proportion and briquetting pressure on briquette compressive strength were studied. Fundamental flow-sheet of fine coal briquetting with the addition of technical hydrolytic lignin as a fuel briquette filler and binder was developed. The technology was adopted for one of the deposits of Middle-Amur basin. It can be varied to fit another objects.
**Key words:** coal briquetting, briquette composition, coal briquetting parameters, moisture content, binder proportion, briquetting pressure, briquetting flow-sheet.

**Introduction**

The production of fine coal in mines and coal preparation plants is increasing. A fraction of this coal can be blended with the larger-size clean coal and shipped to the user. However, a convenient means to handle, store, transport, and use the balance must be devised. Coal reconstitution, encompassing briquetting, disk pelleting, extrusion pelletization and roller-and-die pelletization, is a means frequently proposed for this service [1].

Reasonability of fine coal briquetting is caused by its fine-disperse state and the difficulty of transportation, by the impossibility of burning in standard fire grate furnaces. Coefficient of efficiency of utilization of fuel briquettes is 75 % in comparison with 46.7 % for run-of-mine coal [2].

In developed countries, there is an increasing interest in the combustion of coal and biomass mixtures. There is also an increase in the biomass cultivated for energy. The technologies utilizing renewable energy sources have been known well enough, and some were tested in developed countries [3].

The coal processing industry usually discards fine-size (−150 microns) coal because of its high-moisture content and handling problems. Compacting and briquetting of the fines with and without adding of biomass (sawdust) can be solution of the problem [4].

The object of research is coal of one of deposits of Middle-Amur basin [5]. The goal of research is substantiation of technological parameters of brown coal briquetting.

**Role of coal in fuel-energy balance of Far Eastern region**

Structure of power system of Far Eastern Federal District consists of 30 % hydrogeneration and 70 % thermal power-stations. By-turn the portion of coal in fuel balance of thermal power-stations is 72% [6]. The basic mass of run-of-mine coal is burnt in furnaces without of any preliminary treatment: this fact causes enormous losses. These losses are especially large if coal contains a lot of fine particles and dust.



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Mechanization of second working, heaping and transportation of coal leads to rapid growth of fines in run-of-mine coal. Fine coal comes up to 60 % today. Consumers can't efficiently utilize fine coal. We should take into account the fact that oxidation, spontaneous ignition and dusting during coal storage leads to 5-7 % coal losses. The increase of complexity of utilization of coal resources of Russian Federation Far Eastern Region can be reached by run-of-mine coal sorting and briquetting of fine [7].

The research was carried out with the application of the following methods: technical analysis – moisture determination according State Standard (SS) 27314 – 91; solid fuel ash content according SS 11022-95; total sulfur according SS 8606 – 93; volatile substances according SS 6382-2001. Organic nitrogen was detected according SS 28743-93. Calorific value of solid fuel was identified according SS 147-95 by Bertlo calorimeter. Organic carbon content was measured with the application of TOC-V (Shimadzy).

**Development of technology of fine coal briquetting**

Formation of briquette takes place as a result of adhesion of coal particles with binder. Briquetting process consists of three stages:

- adsorption of binder by coal surface and formation of thin film of binder on the coal surface;
- mixture pressing;
- briquette cooling.

As the particles coarseness decreases, the strength of adhesion inside the briquette increases. When coal moisture is in excess supply, the adhesion of binder to particle surface is complicated and briquette strength decreases. When coal is extra dry, surface wettability makes worse and binder consumption increases. Thereby, the main goals during the technology development are:

- selection of a proper granular composition;
- elimination of undesired admixtures and pollutants;
- determination of the most advantageous range of humidity of the material;
- determination of the type and content of the binding material;
- selection of the type of briquetting machine and briquetting parameters [8].

**Technological characteristics of the coal**

Technological indicators of Usumunskoe deposit coal are given in table 1. Abbreviations are W – coal moisture, V – volatile substances output, A – coal ash content (r, a, daf – run-of-mine, analytical and dry ash-free state respectively), $S_{tot}$, N – sulfur and nitrogen content respectively, $Q_s^a$ – high heat value.

Size test analysis revealed that initial coal samples are characterized by fine dust (-0.2 mm) content at the average 8 % and varies from 3 up to 14 %.

High output of fine fractions predetermines the reasonability of Usumunskiy brown coal utilization for briquette production.

**Substantiation of filler choice**

It is considered that in the case of co-combustion of biomass whose content is higher than 5%, there are necessary technological lines that would provide loading of this fuel to the boiler in a way independent of coal. In such a way, it is possible to increase the content of biomass from 10 to 15% of the calorific value of a fuel flux. At the "Ekokarbotech" enterprise, the technology of producing coal–biomass briquettes was developed. In such briquettes, biomass made approx. 20 percentage by weight [3].

Lignin is researched as briquette filler in this study. Lignin is a waste product of hydrolytic factories. It has a porous structure that improves combustion kinetics of fuel briquette.

Low ash content and high calorific value (table 1) are also factors improving quality characteristics of fuel briquette.

**Table 1** Technological characteristic of coal briquette components

| Material | $W^a$, % | $V^a$, % | $A^a$, % | $S_{tot}$, % | N, % | $Q^a_s$, MJ/kg |
|---|---|---|---|---|---|---|

| | | | | | | |
|---|---|---|---|---|---|---|
| Ushumunskoe deposit coal | 24,32 | 29,37 | 17,59 | 0,37 | 0,43 | 23,39 |
| hydrolytic lignin | 61,8 | 29,5 | 0,65 | 1,1 | 0,57 | 16,6 |

Sulfuric acid content is 1,78 kg per ton (0,17 %). Consequently lignin dumps are potential source of sulfuric acid and it is necessary to organize the processing of the waste. Lignin dumps are prone for spontaneous ignition in hot and dry weather.

**Substantiation of binder choice**

Search of new, economical and high-quality binding materials is one of the main briquetting problems.

Approximately 50 potential binder formulations can be listed. Some researchers consider guar gum and wheat starch to be the most suitable for the pulverized coal market while, lignosulfonate/lime, was targeted for the stoker market [9].

Conventional coal binders such as coal tar pitch and petroleum refinery residue provide agglomerates of satisfactory quality. It is proved that environmentally sound binders such as molasses, dextrin, etc. provide poor water-resistant agglomerates [10].

Alternative is a copolymer binder which consists of molasses and lime; 10% copolymer binder content provides water-resistant briquettes [11].

We propose to apply oil bitumen (BND 90/130) as a binder for brown coal briquettes processing. This binding material meets all claims for binder. Composition for fuel briquettes production was developed in Laboratory of mineral processing of Mining Institute FEB RAS [12, 13].

**Determination of optimal parameters of coal fine briquetting**

The effect of moisture content to strength characteristics of fuel briquette was established. Tests parameters are: pressure 160 MPa, fuel briquette composition – coal 74 %, technical hydrolytic lignin – 11 %, binder – 15 %. Relation between moisture of working mass and briquette compressive strength (RC) is shown in the fig. 1.

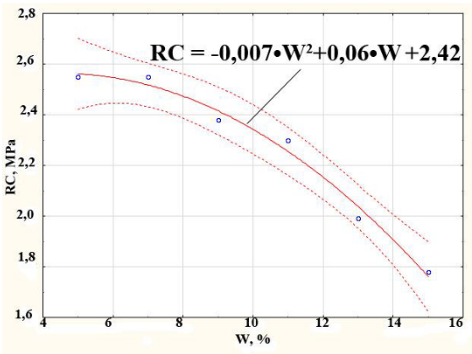


**Fig. 1** Relation between moisture of working mass and briquette compressive strength

Correlation coefficient is R=0.9. Function was researched by differentiation and optimal moisture value was determined as 4.5 %.

Briquette testing was carried out for binder content from 5 up to 25 % (lignin content L = 11 %). Tests revealed that optimal binder content is 15-17 %; further growth of binder content is unreasonable according ecological and economical points.

**Impact of pressure to briquette strength properties was defined**

Relation between pressure of formation (P) and briquette compressive strength (for different binder content B = 10 and B = 15 %, L=11 %) is shown in fig. 2.

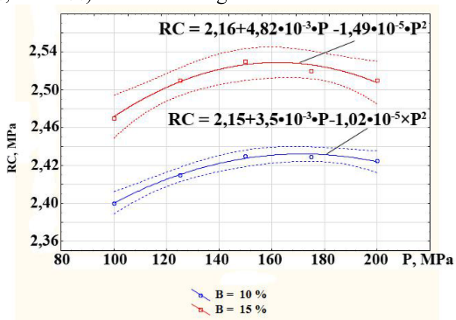


**Fig. 2** Relation between pressure of briquette formation (P) and briquette compressive strength for different binder content

Correlation coefficient is R=0.9. Maximum was identified for P = 160 MPa. Binder content 15 % provides larger briquette compressive strength than 10 % portion. Optimal pressure of briquette formation is 160 MPa; higher pressure causes material overpressing, lower pressure doesn't provide necessary briquette compressive strength.

Principal **flow-sheet of briquette processing** was developed (fig. 3). Coal fine (- 2 mm) is dried and screened. Oversized material is sorted and utilized in heat electropower stations. Technical hydrolytic lignin is dried and mechanoactivated with the addition of hydrated lime. Residual stock of oil distillation is melted and all components proportioned and mixed, briquetted and cooled (up to 45 °C). Briquettes are packed in big bags.

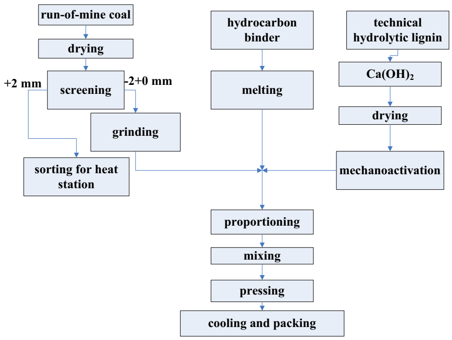


**Fig. 3** Principal technological flow-sheet of fuel briquette production

Rise of complexity of utilization of Far Eastern region resources for the proposed approach is reached at the expense of screening and briquetting of coal fine and processing of waste product of hydrolytic and oil-refining industries.

**Conclusion**

On the basis of technological characteristics of coal the technology of fuel briquettes production was developed. Optimal component composition supplying necessary compressive strength and low moisture absorption of briquettes was established. New innovative composition of fuel briquette providing utilization of coal, hydrolytic and oil-refining industries products was developed. Production is characterized by high qualitative indicators. In a way as a result of research new technological decisions for production of high-quality lump solid fuel from coal fine and waste products were substantiated.

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# The analysis of process flowsheets and selection of equipment for coal fines dewatering

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**Abstract**

In the design of coal preparation plants the selection of methods and equipment for coal dewatering has always been one of the challenges of the process flowsheet development. Currently, the increasingly greater attention n is paid to qualitative indicators of coal products, such as ash content, moisture, sulfur content, and others. Usually the ash content of produced coal is much higher than it is required by consumers, that is why it is necessary to wash it in order to receive low ash concentrates. In order to minimize e the losses in the coal beneficiation, it is necessary to use the most effective wet washing processes, resulting in increase of concentrate's moisture compared to raw coal. Wet technology involves the use of a special dewatering equipment to reduce products moisture. Furthermore, in the process of machine mining, transportation, and washing, there occurs coal degradation and there significantly increases the content of fine particles in the final product, which contributes to its moisture increase. The main challenge is the fine and ultrafine concentrate dewatering. When selecting the dewatering equipment is necessary to comprehensively evaluate both its performance specifications, and capital and operating costs, as well as possible additional safety requirements for the operation of the equipment. The article describes the existing flowsheets of coal fines dewatering and justification of the choice of the most efficient scheme.

**Keywords:** coal moisture, coal fines dewatering, centrifuge, chamber filterpress, vacuumfilter, hyperbaric filter, slurry, ash content, clean coal, concentrate

Modern coal preparation plants are designed to meet maximum possible extraction of combustible mass of raw coal, which involves most effective wet beneficiation processes. Such highly efficient processes of heavy medium separation, jigging, hydraulic classification and flotation allow receiving low ash clean coal product, while providing high product recovery and minimum loss with tailings. However, deep (up to 0 mm) coal washing at the obvious advantages to achieve qualitative and quantitative characteristics of the process has its disadvantages, one of which is high moisture of coal concentrate.

In this paper, technical and economic parameters of the most common schemes of fine concentrate dewatering have been comparatively analyzed, resulting in determination of optimal flowsheet of coal fines dewatering at the current level of development of the technical capabilities of the equipment and equipment prices formed on the market.

The largest part of coal moisture in total concentrates of processing plant is provided by coal particles the size of which is not more than 0-1(2) mm. Such particles have a developed surface area and the channel system between the particles. A significant amount of moisture is retained in pores due to the intermolecular forces (adsorption). It is a common practice in the absence of requirements for the size grade of products to combine fine concentrates with coarse ones. Excess moisture in fine and ultrafine concentrates leads to increase of moisture in total commercial concentrate, decrease of its calorific value, causes problems during transportation and unloading of vehicles due to freezing of coal during the winter season, increases transport costs for the transportation of excess water contained in coal.

The following are the traditional mechanical methods of fine size dewatering: centrifugation, filtration, vacuum or hyperbaric dewatering. The choice of dewatering method depends on the particle size of dewatering slurry.

 
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At old plants, built in the last century, the slurry schemes were limited to grade 0-0.5 mm flotation and meant the dewatering of flotation concentrate on disk, drum and belt vacuum filters. The final water content of flotation concentrate was high - 23-35 %, which presupposed the equipment of processing plants with a drying and heating sections. These are, for example, such large processing plants in Russia as "Pechorskaya", "Neryungrinskaya", "Tomusinskaya", "Siberia", "Berezovskaya".

The slime schemes of modern processing plants wash much broader grain size grade of 0-1(2) mm, which is usually divided into two grain sizes: coarse-grain slurry of 0.15-1(2) mm and fine slurry of 0-0.15 mm [1,2]. In some flowsheets it is even divided into three grain sizes: coarse-grain slurry of 0.15-1(2) mm, fine slurry of 0-0.15 mm and ultrafine slurry of 0-0.04 mm which are processed separately [3]: coarse-grain slurry - in spiral separators (or hydrosizers) and fine and ultrafine slurry - in flotation machines of different types [4,5]. Share of slimes of 0-1 mm particle size at the output of the plant may make from 30 to 60 % of the run-of-mine coal at the input. Material with particle size larger than 0.15 mm (spiral concentrate and hydrosizers) is being dewatered in filtering centrifuges with screw residue discharge. Material of size below 0.15 mm ( flotation concentrate) is being dewatered on filter surfaces (wire mesh, filter cloth) under the influence of the vacuum or hyper pressure in vacuum-filters, hyperbaric filters and filter presses, or in screenbowl centrifuges.

In the course of moderately and highly- metamorphized coal processing the modern mechanical dewatering equipment allows meeting the requirements to the product moisture. So, in construction of new plants, in most cases, it is possible to avoid the use of thermal drying of concentrate. However, the practice of operation of new plants that are processing steam coal with high inherent moisture, as well as plants processing coking coal with a high amount of ultrafines, showed the need to include the drying equipment in new projects.

In the Russian Federation from among the last processing plants with the use of thermal drying are "Neryungrinskaya", built in 1984, and "Pechorskaya" - in 1993. After that in Russia there haven't been implemented any project using thermal drying. At the same time the raw material supplies base of many existing plants is deteriorated. In the run of mine coal there increases the total amount of coal slimes, that directly affects the final product moisture. The owners of coal washing plants, without thermal dryers, face the need to reconstruct the existing plants with a modernization of slimes dewatering schemes and install the additional equipment for fine slurry dewatering to be able to satisfy the total salable concentrate moisture requirements - not more than 7 % in the winter and 8-9 % - in summer. The main limiting factor - coal freezing-up in the winter season. Therefore, for the plants located in regions with the central-continental climate with a predominance of negative average annual low temperatures, such as in Yakutia and Tuva, the thermal drying has no alternative. In other regions, where the problem of product freezing-up is not as acute, modernization of old plants and construction of new ones provides the use of the most effective range of mechanical dewatering equipment for fine concentrate with particle size less than 2 mm.

As it has been mentioned above, the main equipment of the vast majority of modern coal fines dewatering flowsheets are screenbowl centrifuges. The main advantage of this type of equipment is its ability to dewater material of wide size range of 0-3 mm, although the quantity of micron size material in the feed has a strong negative effect both on the final cake moisture and the performance of the centrifuge. Due to the relatively high centrifugal separation factor screenbowl centrifuges are able to provide lower moisture content of cake than other equipment. With favorable feed size distribution in feed, when the content of size 0-0.04 mm is less than 50 %, it is possible to dewater the fine concentrate by using screen bowl centrifuges only. However, the use of screenbowl centrifuges involves some loss of ultrafine coal component of size less than 30-40 microns with bowl effluent. Such flowsheet is implemented on several coal preparation plants in Kuzbass region - "Antonovskaya", "Severnaya", "Raspadskaya", "Shchedrukhinskaya" and others. The achievable centrifuge cake moisture for coal of rank K makes 10-14 %. However we should keep in mind that about half of the concentrate with grain size of less than 0.04 mm goes to centrifuge bowl section effluent which is usually fed to the high-ash slimes thickener and after dewatering on belt filter presses transferred into the waste. Depending on the proportion of ultrafine coal to the volume of plant production, the losses can reach substantial amount of 3-5 % of the raw coal. In order to prevent fine concentrate losses it is necessary to provide a flowsheet scheme of its extraction. The most cost-effective method is to direct the centrifuges effluent and filtrate (sometimes in conjunction with the flotation concentrate) into a sedimentation low-ash slimes thickener with the selection of a special mode of flocculation.

When new processing plants are designed and existing ones are modified, to avoid significant loss of coal fines, there often realized three main combined schemes for coal slimes and froth product of flotation concentrate dewatering [6-10]:

1. A screenbowl centrifuge and a disc vacuum-filter;
2. A screenbowl centrifuge and a hyperbaric-filter;
3. A screenbowl centrifuge and a chamber filter-press.

As practice shows, in most cases, in selection of the dewatering flowsheet, the attention is drawn both to the technological parameters of a particular type of equipment (achievable fines moisture performance), and the equipment price. And it is often a lower price can be the deciding factor when one or another flowsheet is selected. However, it should be understood that when choosing a scheme it is necessary to take into account not only the cost of the main equipment, but also the additional ones (compressors, receivers, pumps etc.) as well as a capital cost of necessary construction projects; maintenance costs, replacements and consumables; the frequency of routine inspections and routine scheduled maintenance and many other moments. In the absence of proper comprehensive analysis of the cost of this or that dewatering scheme variants, there are often taken in the draft costly schemes, increasing the cost of production.

Specialists of "Coralina Engineering" set out to analyze most common flowsheets of fines dewatering at existing plants based on such parameters as capital, operating costs, technological parameters, to take into account the factors of safety and serviceability, and to determine the optimum scheme of coal fines dewatering.

At such plants as "Bachatskaya-Koksovaya", "Krasnobrodskaya-Koksovaya" and after the reconstruction of the "Pechorskaya" coal preparation plant, where the flotation is being used, the fines dewatering scheme No.1 is applied (with screenbowl centrifuges and disc vacuum-filters).

At the "Svyato-Varvarinskaya", "Kuzbasskaya", "Mezhdurechenskaya" and "Matyushinskaya" plants the scheme No.2 is being used including screenbowl centrifuges for coarse slimes dewatering and hyperbaric filters for flotation concentrate dewatering.

At "Raspadskaya" plant for coal fines dewatering the scheme No.3 is being applied (screenbowl centrifuges and chamber filter presses).

Basic technological performances of dewatering schemes work are shown in the Table 1.

**Table 1. The practical results of the chamber filter presses, hyperbar filters and disc vacuum filters performance at the coal preparation plants**

| No. | Plant name | Feed characteristics, size | Type of dewatering equipment | Specific capacity, t/m$^2$*hour | The achievable avg. moisture of cake, % |
|---|---|---|---|---|---|
| 1 | Kuzbasskaya (RF, Kuzbass) | Raw coal 0-0.2 mm, content of 0-0.04 mm up to 20 % | Hyperbar- filter | 0.42 | 19-22 |
| 2 | Mezhdurechenskaja (RF, Kuzbass) | 0-0.2 mm, content of 0-0.04 mm up to 15 % | Hyperbar- filter | 0.75 | 19-22 |
| 3 | Svyato-Varvarinskaya (Ukraine) | 0- 0.15 mm, content of 0-0.04 mm up to 50 % | Hyperbar- filter | 0.36 | 19-22 |

| | | | | | |
|---|---|---|---|---|---|
| 4 | Matyushinskaya (RF, Kuzbass) | 0-0.2 mm, content of 0-0.04mm up to 20 % | Hyperbar- filter | 0.50 | 20-22 |
| 5 | CPU Elga (RF, Yakutia) | 0-0.2 mm, content of 0-0.04 mm up to 50 % | Chamber filter-press | 0.04 | 20-22 |
| 6 | Raspadskaya (RF, Kuzbass) | 0-0.15 mm, content of 0-0.04mm up to 50 % | Chamber- filter-press | 0.04 | 20-23 |
| 7 | Bachatskaya-Koksovaya (RF, Kuzbass) | 0-0.2 mm, content of 0-0.04mm up to 30 % | Disc vacuum-filter | 0.35 | 22-23 |
| 8 | Krasnobrodskaya (RF, Kuzbass) | 0-0.3 mm, content of 0-0.04mm up to 35 % | Disc vacuum-filter | 0.35 | 23-24 |
| 9 | Pechorskaya (RF, Vorkuta) | 0-0.2 mm, content of 0-0.04mm up to 75 % | Disc vacuum-filter | 0.25 | 25-30 |

The comparative analysis of these dewatering schemes was conducted for the same particle size distribution of the feed material (coal concentrate of spiral separator of size 0.2-1 mm and flotation concentrate of particle size 0-0.2 mm) and at the same capacity of the dewatering circuit - 130 t/h. The content of micron size 0-0.04 mm particles in the feed particles size distribution was assumed of 50 %.

The comparison of certain significant technological and economic characteristics of slime dewatering schemes at the same feed is shown in the Table 2.

**Table 2. The comparison of the technological and economic parameters of slime dewatering schemes at the same feed**

| **Item** | **Measure unit** | **Scheme 1. Disc vacuum-filter** | **Scheme 2. Hyperbar-filter** | **Scheme 3. Chamber filter-press** |
|---|---|---|---|---|
| Quantity of equipment units | pcs. | 3 | 2 | 5 |
| Footprint of equipment unit | $m^2$ | 32.4 | 41.4 | 38.6 |
| Total equipment installation area | $m^2$ | 97.2 | 82.8 | 193.0 |
| Cake specific load for 1 $m^2$ of filtration surface | ton/ $m^2$ | 0.54 | 0.45 | 0.07 |
| Cake moisture | % | 25-27 | 19-22 | 20-22 |
| Specific (per 1 unit) energy consumption (incl. auxiliary equipment) | kW/hour | 680 | 700 | 205 |
| The total cost of the equipment, including auxiliary (average) | thousand ruble | 250 000,0 | 200 000,0 | 300 000,0 |
| Energy costs (at cost of 1 kW*h 3.5 ruble) | thousand ruble per year | 49 980,0 | 34 300,0 | 25 112,5 |
| The cost of spare parts, consumables | thousand ruble per year | 25 000,0 | 17 900,0 | 16 000,0 |
| Payroll budget | thousand ruble per year | 3 000,0 | 3 000,0 | 3 000,0 |

| | | | | |
|---|---|---|---|---|
| Amortization expenses (10 years) | thousand ruble per year | 25 000,0 | 20 000,0 | 30 000,0 |
| Operating and service costs, Total: | thousand ruble per year | 102 980,0 | 75 200,0 | 74 112,5 |
| Specific costs per 1 ton of final product (dry): | | | | |
| excluding amortization expenses | ruble per ton (dry) | 85.69 | 60.66 | 48.47 |
| including amortization expenses | ruble per ton (dry) | 132.84 | 82.64 | 81.44 |

The analysis of the technological performance of various dewatering flow sheets allowed us to make the following conclusions:

- The best and similar in value of the final product moisture results are achieved by using a chamber filter-press and hyperbaric filter;
- Chamber filter presses require a larger installation area (footprint) within the perimeter of the plant;
- Smallest power consumption of chamber filter-presses completely compensates the cost of a larger installation area.

The main objective of economic calculations was to detect the slime dewatering flowsheet variant with the lowest capital and operating costs. The calculations were made based on the average prices of the equipment at market, taking into consideration the cost of not only the major equipment, but also support ones - pumps, compressors, receivers, and others. Calculation data on processing in slurry scheme of 910 000 ton a year for mining and beneficiation complex Elegest, as an example, are shown in Table 2.

The analysis of economic indicators of three dewatering schemes operation led to the following conclusions: operating costs and, consequently, the specific cost for 1 ton of fine concentrate dewatering is actually the same for flowsheets with a chamber filter-press and hyperbaric filter, and is less if to compare with the disc vacuum filter scheme.

It should be noted that within the conditions of existing plants, the final product moisture will greatly depend on the type of coal, changes in qualitative and sizing characteristics of feed to fine concentrate dewatering circuit as well as a timely instrumental control over the operation of processing equipment in the plant.

The analysis of existing plants performance shows that the final concentrate moisture depends on the content of -0.04 mm size slimes in the feed. The more fine components in the feed, the higher is the moisture of the final product. Besides the increased ratio of micron size component reduces the specific capacity of the equipment. For example, increasing the content of 0-0.04 mm size slimes in the feed from 50 % to 75 %, hyperbaric filters capacity (based on the dry solids) decreases from 0.750 to 0.360 ton/$m^2$.

In addition to the technological and economic factors affecting the choice of a dewatering scheme, there are legal regulations that could limit the use of one or another type of equipment. In accordance with applicable Federal Rules and Regulations in the field of industrial safety (FRR) "Rules of industrial safety of hazardous industrial facilities that use equipment operating under pressure", item 3, hyperbaric filters are equipment operating under pressure. In accordance with these rules (item 64) vessels installation is possible in open areas or in separate buildings only. In addition, (item 65) it is possible to install them in areas adjacent to industrial buildings (annexes), under condition they are separated by capital wall.

It should be noted that the requirements to the products and prices at the coal market undergo drastic changes and often coal mines and processing plants have to work at reduced capacity. In this case, the presence of several units of dewatering equipment would allow the flexibility in switching to a more economical mode of operation with reduced capacity when a part of the equipment is disconnected from the technological circuit. The flowsheet with a number of chamber filter presses allows regulating plant capacity quite easily.

**Conclusions**

1. In practice, the identical results on the moisture content of the final product are achieved with the use of a chamber filter-press and hyperbaric filter.

2. Chamber filter press has larger installation area, but lower power consumption if to compare with other dewatering flowsheets.

3. The lowest capital and operating cost of 1 ton of fine concentrate dewatering is observed in schemes with chamber filter presses and hyperbar-filter compared to disc vacuum-filter scheme.

4. Calculations made for mining and beneficiation complex "Elegest" show that the flowsheet with a number of chamber filter-press units is similar in value if to compare t with the scheme with fewer number of hyperbaric filter units. The scheme with chamber filter presses provides a significant safety margin of the dewatering circuit compared to the scheme with hyperbaric filters.

Thus, taking into consideration the existing prices at the world market of coal processing equipment, which determine the capital costs and necessary operating costs, the coal slime of size 0-2(3) mm dewatering scheme, including combination of screenbowl centrifuges and chamber filter-presses is optimal from technological and economic indicators point of view. Furthermore, this scheme is easy to use, able to ensure non-stop operation and does not require additional measures to ensure industrial safety.

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# Innovative Drying Technology "Chronos". Deep Non-Thermal Dewatering Of Coal And Mineral Fines

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**Abstract**
Viewed in the article method of coal slime dewatering using dry sorbents allows safe wet materials, including solid fuels, due to the absence of heat. In the article are marked the requirements for properties of the sorbent, the fulfillment of which makes the process of wet materials drying an efficient and cost-effective. The authors developed a method of short cycle adsorption in contact with granules of wet coal sorbent based on the difference in the dynamics of sorption and desorption of moisture by sorbents. As part of the regeneration of sorbents crucial to the successful implementation of the process of moisture desorption is such an intensity of exposure to the sorbent drying factor, which is enough to dry the 40-50 % of the pore volume of the sorbent. Application of the new method significantly reduces the time and energy consumption for wet coal slimes drying. The proposed method of dewatering allows to safely drain the coal with a high volatile content (over 30 %) and oxidized coals. New solutions to reduce the time of drying cycles of coal and sorbent regeneration formed the basis for the name of drying technology - Rapid Nano Drying (RND) CHRONOS.

**Keywords:** sorbent, moisture, moisture sorption, moisture desorption, coal, dewatering, drying.

In most cases, after the processes of coal washing, moisture content of dewatered fine concentrate and thin slimes is high, which leads to freezing of the product in the haul vehicles during the winter [1]. For this reason, after mechanical dewatering occurs the need for thermal drying of the material, at least in the winter season.

When heat drying is possible to dry the material rapidly, but the use of high temperatures makes the process of drying coal and minerals fire- and explosion- dangerous, and implies significant additional financial costs for the organization of the drying and binding measures aimed at industrial safety and environmental protection [2].

In order to achieve the contract specifications are sufficiently to dry slightly slimes from centrifuges or filters by 5-7 %. The use of expensive technologies of thermal drying at small performance preparation plants would be economically impractical. One of the solutions to the problems of safety and cost-economical drying technology is the fine and thin coal, metallic and non-metallic minerals dewatering using sorbents. Upon contact of a wet material with sorbent due to the presence on the sorbent surface of metal (Al, Na, Ca, etc.) active cations, dipolar water molecules are attracted to the cation, penetrate and fill the structured pore space of the sorbent [3]. Since pure sorbents adsorption strength of water retention is higher than that drained material has, the sorbent is pulled out water from the material and absorbs it into itself. Sorbents power in sorption activity is such that for a short period of time it takes from the material free condensed and surface moisture and even the portion of the tightly bound interior moisture. Thus, the wet material in contact with the sorbent is not only better dewatered, as compared with mechanical means, but also the material drying without using of high heat or microwave radiation as well.

Safety of sorbents drying technology is that the process of the material dewatering by sorbent does not use high temperature, that is especially important in the dewatering of combustible materials, such as high volatiles coal. The relatively high temperature (180 °C) can only be used in the process of the sorbent regeneration, which is non-combustible material.

The technology is environmentally friendly because the coal drying takes place at room temperature and during the drying of the sorbent occurs minimal environmental pollution, since the sorbent is heat-resistant and during it regeneration in atmosphere evaporates water contained therein only.

 
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The technology is also cost-effective because after the contact of absorbed moisture sorbent granules and dried coal these are simply separated by screening. Sorbents are sent for regeneration (the process of removing moisture from it), and then repeatedly used for the following batches of drying material as long as the strength of the sorbent allows.

**The history of the idea of drying by sorbents**

The idea of using sorbents for drying wet and sticky materials offered to engineers back in the 1960s. One of the famous patents at the time is Nelson Severinghaus patent [4], to offer the world an idea of sorbent drying wet dust-sized materials in the transport stream. By the beginning of the XXI century this idea received another boost development by the US company Nano Drying Technologies (NDT), offers theoretical and laboratory solutions for the continuous dewatering of mineral and coal slimes [5, 6]. On the basis of the decisions laid down researches conducted by Professor G. Luttrell and his colleagues from Virginia Tech University [7].

Obvious reasons for still non-active development of this technology include: insufficient knowledge about the properties and theoretical foundations of coal interaction with sorbents; the lack of suitable sorbents in the market, as well as the lack of effective and economically affordable methods of sorbents regeneration.

Specialists of the Russian company "Coralina Engineering" also drew attention to an interesting solution to the problem of deep dewatering and drying of coal slimes. Research conducted in the company's lab and results of the studies [8] have become the basis for a Patent [9].

**Sorbents. Characteristics and technological requirements**

One of the limiting factors in the development of the technology is the lack of sorbents with properties that meet the requirements when working with wet coal slimes. Currently, the traditional area of the use of sorbents is the gas drying. The vast majority of existing on the market types of sorbents, theoretically suitable for drying bulk solids are intended for the sorption of water vapors or other undesirable substances from the gases. The gas passes through the bed of sorbent smoothly without breaking its structure. When drying of bulk solids is realized in processes of the mixing, the sorting and transporting there is a mechanical affect on the sorbent granules present. Resistance to abrasion of granules determines the useful life of the sorbent, and the cost of the material drying process as well.

Another selection criteria is the size of the inlet window of sorbent's pore, the diameter of which depends on not only internal pores volume of the sorbent, but also the speed of water absorption and subsequent desorption processes. The large size of the inlet window allows water molecules to penetrate into the sorbent more freely, be adsorbed on the walls of the micropores, and easy to leave the sorbent in the regeneration process.

To compare the effectiveness of different sorbents authors have conducted cyclic tests to dry the coal slime. For an objective evaluation of test results there were selected the drying sorbent weight changes on the sixth cycles of regeneration. That has allowed to compare the results with test data of sorbents with the size of the inlet window of 0.3 nm indicated by the American company NDT in U.S. Patent # 20140144072 [6]. Time of drying is selected from 2 to 5 minutes because the greatest desorption process dynamics fall in the specified range. The test results were extrapolated to the value of the sorbents initial weight before the drying process mentioned in the U.S. Patent above.

In drying tests there was used the concentrate of coking coal of size 0-1 mm and a total moisture content $W_t$ = 30 %, by weight. The weight of each coal sample prior to contact with the sorbent was 50 grams. As a drying agent there were used sorbents of Russian (RUS) and Chinese (CH) production with the size of the inlet window of 0.9 nm, as well as a China-made active alumina oxide (AA) with slit pore structure, the average size of which was greater than 0.8 nm. The weight of the sorbents samples was also 50 grams.

Concentrate and sorbent are mixed together for 1 minute until the concentrate moisture drops below 5 %. After separation on a laboratory sieve, adsorbent is dried by atmospheric air; heated to a temperature of 180 °C for five minutes. During the sorbent drying the weight been measured every minute. The dried sorbent is re-directed to the mixing with the coal again. Thus, it was done in six cycles of sorption / desorption of the sorbent with the same weighed samples of wet concentrate. The results of weight change during the sixth regeneration cycle are shown on Figure 1.

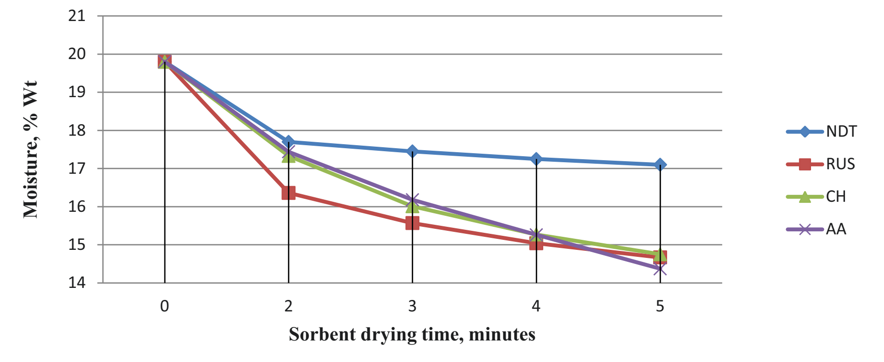


**Figure 1. Dynamics of different types of sorbents moisture loss**

A significant reduction of sorbent contact time with a wet material samples with equal amounts of adsorbed moisture - from 60 minutes at 0.3 nm [6] up to 1-2 minutes at 0.8-0.9 nm is confirmed effectiveness of the use for the sorption process of sorbents with the size of the inlet window of pore more than 0.8 nm.

Curves of sorbents with the size of the inlet window of 0.8-0.9 nm desorption already on the second minute of sorbent drying process show that the effectiveness of such sorbents is higher by 10-30 %. By the fourth minute gap in efficiency rises up to 45 %. These results demonstrate the high efficiency of moisture desorption by such sorbents.

It is necessary to note that the size of the inlet window defines also the filtering capacity of the sorbent. By increasing the diameter of inlet windows besides water molecules into the pores of the sorbent can enter molecules of other substances that could negatively affect the subsequent regeneration.

One problem with the use of sorbents for coal fines drying is the specificity of the process of moisture mass transfer between the material and the sorbent. A significant amount of the condensed and film moisture in the drying material due to the reaction of the water sorption characteristics affects the strength of the sorbent granules. Besides, it can promote instantaneous granules destruction of some kinds of sorbent due to turbulent flow of the exothermic reaction. This feature imposes on the drying technology the limit of the feeding material maximal moisture content, and also shows the points in technological circuits of coal processing plants, where such technology can be effectively used.

Thus, from the technological characteristics required of sorbents, it is necessary to mention two:

- The ability of the sorbent to absorb quickly a large amount of condensed moisture (high sorption dynamics);

- The ability of the sorbent at low energy consumption to give quickly water back during the regeneration (high desorption dynamics).

In turn, the drying technology of such wet materials like coal fines and thin slime with using sorbents must solve the following problems:

- To reduce sorbents wear caused by abrasive coals and the mechanical action of the executive bodies of the drying machine;

- To provide the drying process run at a high content of condensed moisture, promoting structural sorbent granules destruction;

- To reduce time and energy consumption of the process of the sorbent sorption and desorption in removing from the drying material a large amount of water.

To address these challenges and to achieve the best technical and economic results of the dryer, under order of "Coralina Engineering" by Russian manufacturer of catalytic systems was developed the sorbent with high wear resistance, which best meets the listed above requirements of the sorption/desorption activity, and is specially dedicated for coal fines of 0-3 mm size and moisture content of up to 30 % drying use.

**Impact properties of materials on the drying process**

Conducted by the Company research has shown that a high ash content of the drying material promotes the rapid contamination of sorbent granules by micro-particles of clay and other minerals contained in the coal. This process may require additional sorbent cleaning operations. But in most cases after a certain number of adsorption / desorption cycles on the surface of the sorbent is formed a thin layer of mineral micro-particles to prevent a further build-up and have little effect on the efficiency of coal fines drying.

From the particle size distribution of coal and its ash content depends on the efficiency of separation of a mixture of coal and sorbent after the step of mixing. The large content of fine component in the mass of coal and its high ash content contribute to forming coal fines lumps due to non-uniformity of the process of mass transfer of moisture. Lumps in size can reach and even exceed the dimensions of the sorbent granules, while having considerable strength. Without special measures the dried material will inevitably go along with the sorbent in the regeneration unit, clogging it and reducing the effectiveness of the further sorbent action.

The ratio of the surface and interior moisture in coal directly affects the process of its drying and, in order to achieve optimal results, requires determining individual modes for drying coals of different grades.

**Technical and technological solutions to increase the efficiency of coal slime dewatering**

To increase drying efficiency and reduce the time for coal fines drying it is necessary to maximize the surface of sorbent contact with the material to be dried, which is achieved by intensifying the mixing process [9]. This can be done by use of the mixer (Fig. 2), the design of which takes into account changes in the characteristics of the material and the sorbent stirred mass.

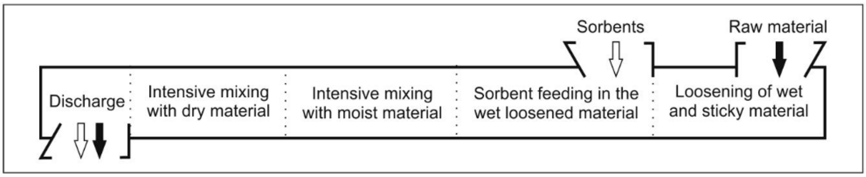


**Figure 2. The movement of wet material and sorbent**

Increasing the intensity of mixing allows to reduce the time of sorbent contact with the material to be dried and reduces the formation of lumps of dried material, provides a positive influence on the efficiency of the subsequent separation of the sorbent and coal concentrate on a separating device (vibrating screen).

In addition to the development of special methods of contacting the dried material and the sorbent, the authors have conducted research and developed the theory of sorbent, used for condensed and film moisture removing, regeneration. Based on this theory for use in the unit for coal fines dewatering by sorbents is proposed the use of short-cycle regeneration, significantly reduces the time and energy consumption to recover the properties of the sorbent in the regeneration process.

**Short-cycle regeneration of sorbents**

Kinetics of condensed moisture desorption process during the sorbents regeneration characterized by non-linear dependence of the volume of liquid removed from the sorbent by the exposure time of drying agent at a constant of its intensity. For example, during the thermal influence on sorbent granule the amount of removed moisture increases with time of high temperature exposure of granules. From the standpoint of the relation of time and energy consumption, this process is effective as long as the amount evaporated from the sorbent moisture does not exceed 50 % of the volume of water absorbed by the sorbent. Subsequently, as the front of dried layer moves to the center of the sorbent granule, spent time and energy in respect to the increment of the volume of evaporated water increase considerably.

A clear evidence of moisture desorption process non-linearity is different amount of removed moisture during regeneration of the sorbent at quite low for regeneration temperature of 180 °C (Tab. 1). For example, the regeneration of the sorbent with the size of the inlet window of 0.9 nm, designated as Sorbent-1, for as increasing of time of stay in the sorbent regeneration chamber, the intensity of the desorption process starts falling considerably after the second minute of drying. At this moment it is about 50 % of the initial volume of water evaporated from the sorbent. A similar process occurs in the regeneration of other types of sorbent - active alumina, designated as Sorbent-2. In the latter case, there is an intensive evaporation of moisture in the range of 2.5 minutes of thermal exposure and further evaporation rate is stabilized, but remains higher than that of Sorbent-1, due to the physical-chemical and structural characteristics of this sorbent.

**Table 1. Dynamics of sorbents moisture loss**

| **Regeneration time, minutes** | **0** | **2** | **3** | **4** | **5** |
|---|---|---|---|---|---|
| **Sorbent type** | **Sorbent-1** | | | | |
| **Moisture content, %** | 35,68 ($W_t$ max) | 22,18 | 18,22 | 15,32 | 13,20 |
| **Moisture loss, %** | 0,00 | 48,63 | 59,79 | 67,33 | 72,54 |
| **Moisture loss to previous, %** | | -48,63 | -11,16 | -7,54 | -5,21 |
| **Sorbent type** | **Sorbent-2** | | | | |
| **Moisture content, %** | 31,11 ($W_t$ max) | 21,81 | 15,70 | 10,65 | 5,11 |
| **Moisture loss, %** | 0,00 | 38,20 | 58,75 | 73,55 | 88,07 |
| **Moisture loss to previous, %** | | -38,20 | -20,55 | -14,8 | -14,52 |

Based on the experimental results presented in Table 1, reflecting the dynamic of sorbents moisture loss depending on the time of temperature affect, it can be concluded that the process of low-temperature regeneration is characterized by the maximum rate of desorption, was observed until the end of the second minute after the start of the sorbent drying. After the second minute the drying efficiency of the desorption process is somewhat reduced, and the specific energy cost per unit of volume of evaporated moisture increases. Time period corresponding to the maximum rate of desorption may vary depending on the intensity of exposure to the sorbent by drying agent.

At the sorbent dynamic capacity of water vapor up to 25-28 %, sorption capacity of some sorbents when these contact with condensed and the film moisture can reach values of 35 % and above (up to 0.55 g of water per 1 g of sorbent). Thus, certain types of sorbents are capable to absorb water in an amount of up to and even more than a half of its weight. This property is due to the presence in the sorbent granules of transport pores, transferring water from the granule surface to the cavities inside the lattice structure of the sorbent. Upon sorbents contact with the condensed and the film moisture pores are filled with water easily due to the high adhesiveness of the sorbent material to water molecules. When the sorbent is used in the gaseous medium the concentration of water vapor is insufficient for the active accumulation of water in the internal pore channels. This sorbents feature allows to reduce the time of condensed moisture sorption process without significant reduction of its dynamics due to filling only the surface layer of the granule, consisting 40-50 % of the total pore volume, even with water fully filled internal pore space of granule.

Based on the studied dependences of the different types of sorbents regeneration dynamic when these are used for the condensed and film moisture remove and subject to standard regeneration mode recommended by the manufacturer of sorbents, the authors propose a method to short-cycle regeneration of sorbents, allowing to extract repeatedly moisture from the sorbent not completely, but in an amount sufficient to repeat the following sorption cycle without loss of efficiency.

Method of the sorbent short-cycle regeneration in terms of cyclic adsorption of condensed moisture can be realized upon exposure to the sorbent of various drying agents - thermal drying of the sorbent by heated gas, power of microwave radiation, etc., as well as the combined agents with high pressure or vacuum in the desorption chamber. The decisive condition for the successful implementation of the short-cycle process of moisture desorption is such intensity of exposure to the sorbent of drying agent of which is sufficient for drying from 40 to 50 % of the pore volume of the sorbent.

New solutions to reduce the drying time of coal and sorbent regeneration cycles created an opportunity to develop the technology of short-cycle adsorption, which formed the basis for the name of drying technology - Rapid Nano Drying (RND) CHRONOS.

**Methodical support**

For an objective assessment of opportunities of sorbents work on coal slimes and influence the properties of the dried material in the process technology has been developed a technique of laboratory cyclic tests, which allows to simulate the behavior of the sorbent during the achievement of running conditions and assess the impact of the specific properties of the dried material on the performance of the sorbent. The technique's essence is to repeat many times of mixing, sieving and regeneration cycles at a variety of pre-defined conditions. The test results allow to determine accurately the technological parameters of installations for coal of fine and ultrafine size drying.

**Conclusion**

The proposed dewatering method allows to dry safely coals having a high volatile matter (over 30 %) and oxidized coals. In applying the developed by authors method of short-cycle adsorption of sorbent granules in contact with wet coal the mixture is heated to a temperature not higher than 100 °C (including those due to the exothermic reaction upon absorption of a large amount of moisture by sorbents), which is insufficient to ignite the coal. Thus, installation is explosion and fire proof in respect of all types and grades of coal. Implementation of the developed technical requirements for sorbents and short-cycle technology of sorbent regeneration will allow to make the drying process a cost-effective and competitive with other types of drying.

Due to the major advantages of the proposed drying method - the absence of thermal effects on the material to be dried - this technology can be successfully applied not only in the coal and mineral industries, but also in other industries.

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# Production of fuel briquettes from carbon containing materials

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**Summary**

Most often for production thermal energy use oil, gas, and coal is more rarely. Also for receiving thermal energy it is possible to use other carbon materials, including the waste. The other side during a woodworking a large number of wastes is formed. The similar side is observed and when processing combustible slates. Both wood and slate waste can be used for production energy resources. One of ways of processing of the above waste is their briquetting. The complex of researches which showed possibility of receiving fuel briquettes from products of a woodworking and a shale processing was carried out. The optimum ratio of waste from a wood content and shale products are ranging from 10:40:50 to 20:60:20. Briquetting was made without binding substances since wood products acted as some kind of binding. It significantly simplifies and reduces the price of techniques. Briquettes were formed by both as uniform pressing as and extrusion. Possibility of production fuel briquettes from coal slimes and paper waste is also shown. Use as carbonaceous components of the coal slimes such as shale waste and paper waste acting as binding substance gives the chance to receive idle time in production and on structure a fuel briquette and at the same time to utilize coal and paper waste. The flammable briquettes having a special incendiary layer were made. Such layer easily ignites from low-energy sources of heat (for example, matches) then burning is goon the main part of a briquette.

**Keywords:** thermal energy, fuel briquettes, carbon materials, wood waste, shale processing, coal slimes, paper waste.

Traditionally for receiving thermal energy use such carbon materials as oil, gas and coal is more rarely In spite of it is possible the receiving of thermal energy to use also other carbon and coal materials including various waste.

It should be noted that during processes of a wood processing a large number of small products (shavings, sawdust, waste pollute products, etc.) is formed. It actually such products are practically not used, respectively pollute environment and create fire danger. The other side is observed and when processing combustible slates. Both wood and slate waste can be used for receiving energy components. One of ways of processing of the above mention waste is their briquetting [1–5].

Every year the scope of fuel briquettes and granules with using of wood waste are increases. At the present time recycling has not less important character for the production than receiving finish products. On the other hand there is a problem of recycling of fuel shale production but briquettes producing only from shale waste have insufficient mechanical durability. Therefore, represents scientific and technical interest studying of possibility of briquetting of a slate trifle with waste of wood processing and use of the received briquettes as the alternative heat carrier. For decision of these problems was carried out the complex of researches on receiving fuel briquettes from products of a wood processing and fuel shale processing [6–8].

 
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As furnace charge for receiving briquettes were used wood waste and also products of pollute processes of shale processing in various proportions. Pilot studies showed that the optimum ratio of wood waste product and shale waste is from 10:40:50 to 20:60:20.

The content of wood wastes and shale waste products quantity smaller, than 10% doesn't allow to utilize them in enough the contents more than 20% reduces durability and caloric content of briquettes. The high content of shale waste products reduces durability of briquettes more than 40% and at contents of shale more than 60% reduces the caloric content of briquettes without increasing durability. However the content of shale products in small quantity than 20% reduces the caloric content of briquettes and the contents more than 50% reduces durability of briquettes.

As components of furnace charge were used briquettes producing from wood waste and products of gas treatment system of fuel shale processing. Briquetting was made without binding substances since wood products acted as some kind of binding. It significantly simplifies and reduces the price of techniques. The process flow diagram o of briquette production is given in figure 1.

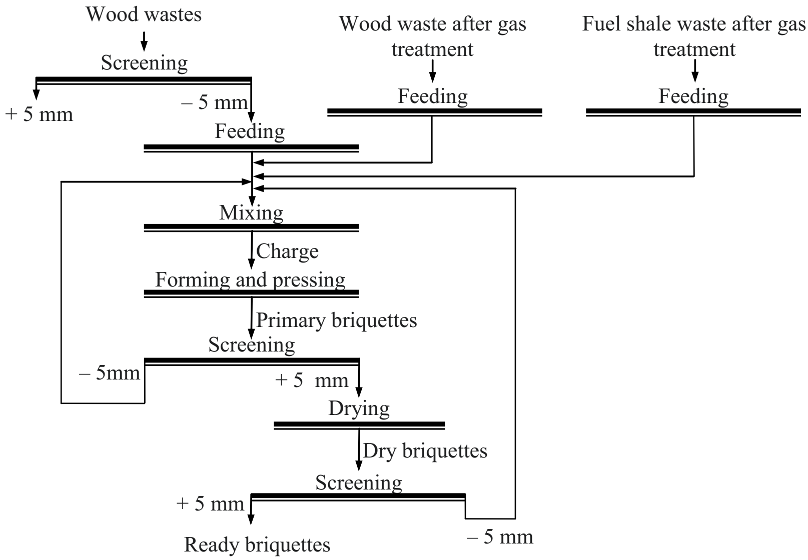


Fig. 1 Technological scheme of processing wood and shale wastes

It should be noted that drying isn't obligatory operation and is applied only when products for some reasons are damp. Briquettes were formed as well pressing as extrusion. The pressed briquettes were made with a diameter of 10 mm, length to 50 mm. Extrusive briquettes were also made with a diameter of 10 mm, length usually was to 100 mm but at an transporting they broke and ready briquettes had length no more than 50 mm. The concrete sizes and a form of briquettes can change in very wide limits according to requirements of the consumer. Mechanical durability of briquettes was defined as number from height of 1,5 m which maintained briquettes. Results of tests are given in table 1 and 2.

Table 1

Results of tests of the wood and shale briquettes by pressing

| No. | Quantity of wood waste, % | Quantity of wood waste after gas treatment, % | Quantity of fuel shale waste after gas treatment, % | Number of falling without essential destruction |
|---|---|---|---|---|
| 1 | 100 | 0 | 0 | 8 |
| 2 | 0 | 100 | 0 | 9 |
| 3 | 0 | 0 | 100 | 1 |
| 4 | 5 | 15 | 80 | 3 |
| 5 | 10 | 40 | 50 | 9 |
| 6 | 15 | 40 | 45 | 11 |
| 7 | 20 | 50 | 30 | >12 |
| 8 | 20 | 60 | 20 | >12 |

Table 2

Results of tests of the wood and shale briquettes by extrusion

| No. | Quantity of wood waste, % | Quantity of wood waste after gas treatment, % | Quantity of fuel shale waste after gas treatment, % | Number of falling without essential destruction |
|---|---|---|---|---|
| 1 | 100 | 0 | 0 | 9 |
| 2 | 0 | 100 | 0 | 9 |
| 3 | 0 | 0 | 100 | 1 |
| 4 | 5 | 15 | 80 | 4 |
| 5 | 10 | 40 | 50 | 10 |
| 6 | 15 | 40 | 45 | 12 |
| 7 | 20 | 50 | 30 | >12 |
| 8 | 20 | 60 | 20 | >12 |

Series of experiences was carried out in which of products of fuel shale waste after gas treatment previously was removed the granular fraction of the first stage of shale processing. Thus there is an additional beneficiation when in large (granular) products is significantly higher then content of inorganic components.

The ready briquettes are kindled as usual firewood and at burning allocate not enough smoke. They are significantly more compact than usual firewood which has specific caloric content and burns more time.
Thus we can produce very effective fuel from products of a wood wastes after gas treatment processes and shale processing. It is give possible to receive full-fledged energy carriers at their simultaneous utilization that partially solves also an environmental problem.

In other experiments were used as carbonaceous components of furnace charge coal slimes and paper waste. Using as coal slimes and paper waste acting as binding substance gives the chance to receive idle time in production and on structure a fuel briquette and at the same time to utilize coal and paper waste. Coal slimes increase calorific ability of a fuel briquette. Paper waste at the same time is both a binding and combustible component [9-12].

Briquettes were made by extruding and had the central opening. Existence of the central opening increases completeness of combustion of a fuel briquette and reduces emissions of harmful substances in the atmosphere. The area of section it is central openings was about 30% of the area of cross section of a briquette. Reduction of this area reduces completeness of combustion of a briquette because of an insufficient air flow. The increase in the area of cross section doesn't increase completeness of combustion of a briquette any more but conducts to decrease in specific caloric content and lowers briquette durability.

Briquettes were made of slimes (– 0,5 mm) long-flame coal with an ash-content of 12,4%, humidity – 25,4%, humidity of paper waste of 2,4%. The main technological operations it is a dosage of components of furnace charge and then their mixing, formation of briquettes, elimination of a trifle which comes back to mixing operation. Standard briquettes on size, dry and from them the trifle which comes back to mixing operation is also eliminated. Briquettes are stored and then shipped to the consumer. At production of a briquette there is a hashing of damp coal slimes to rather dry paper waste thus slimes partially are dehydrated and paper is on the contrary moistened. Necessary humidity of paper-coal furnace charge is regulated by change of the contents in it of slimes also if necessary slimes are dehydrated previously.

Briquette diameter of an internal opening of 12 mm, length of briquettes from 180 to 220 mm were formed by extrusion in the form of the cylinder with a diameter of 40 mm. Data on tests of briquettes are provided in table 3. Mechanical properties of briquettes were defined as number of falling from height of 1,5 m which maintained briquettes.

Table 3

Results of tests of paper-coal briquettes

| № | Quantity of slimes, % | Quantity of paper waste, % | Number of falling without essential destruction |
|---|---|---|---|
| 1 | 30 | 70 | 11 |
| 2 | 35 | 65 | 11 |
| 3 | 40 | 60 | 10 |
| 4 | 45 | 55 | 8 |
| 5 | 50 | 50 | 8 |
| 6 | 55 | 45 | 6 |
| 7 | 60 | 40 | 4 |

Also briquettes in which there was a special incendiary layer were made. The incendiary layer contains nitrates and easily ignites from low-energy sources of heat (for example, matches) then burning is overeaten on the main part of a briquette. Briquettes were created by the extruder having zones of hashing and pressing.

Nitrates which pressing in the main layer in a dry form on external surface of a briquette due to operation of the additional screw. In a pressing zone the briquette is strengthened and leaves an extruder which having strength sufficient for its transportation on drying. Briquettes after drying easily burning and actually didn't allocate a smoke so such briquette is very convenient in use. It is also possible using briquettes with an incendiary layer and without it approximate ratio 1:10. In this case burning briquette with an incendiary layer ignites briquettes without incendiary layer. Thus there is the essential economy because briquettes with an incendiary layer are more expensive than without it.

Thus technologies of receiving fuel briquettes from different types of low-demanded carbon and coal materials and carbon waste are developed. Briquettes intended for household coppers, fireplaces, various household furnaces for cooking, heating of inhabited and economic rooms, for change houses, etc.

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# Parametric Optimization of Coal Concentrate Thermal Drying (in case of Elegest Mining and Processing Complex

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**Annotation**

A comparison of the structural, technological and operational characteristics of different driers has been made: gas-tube driers, direct-fired rotary driers, and steam tube driers; microwave driers, fluidized bed driers. The main parameters restricting the use of the driers have been detected: significant capital and operating costs; high explosion and fire hazard due to the high yield of volatile matter; environmental restrictions. CJSC TEIC is implementing a construction project of the Elegest Mining and Processing Complex, located at the Elegest field of the Ulug-Khemsk Coal Basin in the Tuva Republic. Production output will make up 17.4 million tons of raw coal per year. Production output of semi-hard coking coal enriched concentrate will make up 15.0 million tons per year. The coal from the Elegest field is characterized by good caking properties, homogeneous petrographic composition, and low ash content. At the same time, this coal is soft, spalt and easily milled. The economic and technological feasibility of thermal drying incorporation in the operation schedule of the projected GOK Elegest enterprise has been justified. It has been suggested to include a vertical tube dryer into the process system scheme together with a quick-response hot gas generator, operating on powdered coal, with a dry exhaust gas cleaning system which ensures the safe drying of high-volatile coal. Application of optimal conditions for dehydration and drying in the process system scheme depending on operational modes of the complex will allow to receive total concentrate moisture within 7.0 - 8.0%, and to avoid high losses - up to 1.5 mln. tons of coal per year in the processing of 17.4 million tons of raw coal.

**Key words:** coal, slagging, thermal drying, gas generator,gas recirculation, sock filter.

In the operation schedules of the coal washing plants when flotation is used for the fine slurry treatment, the thermal drying is, as the rule, the final operation of coal processing. The driers constitute a complex of technological units, consisting of drying unit, combustion equipment, dust-recovery devices and fans, metering transport and shut-off equipment. [1]

The scope of this work is to optimize the parameters of thermal drying of GOK Elegest coal concentrates by the following parameters: industrial safety, capital expenditures, operating costs, efficiency, and environmental safety.

The necessity of applying the thermal drying at the projected GOK Elegest enterprise is stipulated by relatively high concentrate moisture after dehumidification as well as by production capacity of this enterprise – 15 mln. tons of cocking coal concentrate per year.

The direct-fired rotary driers have the following advantages compared to the other type driers: performance reliability when drying bulk materials containing lumps of size up to 250 mm, as well as when drying fines, viscous, sticky materials, such as flotation concentrates and slurries. Among disadvantages of the direct-fired rotary driers are: large overall dimensions and low tension of the drum volume of evaporated moisture, significant metal intensity (4 – 5 tones per 1 ton of evaporated moisture per hour), a large amount of the material continuously remaining in the drum during its operation, and sticking of wet material on the internal devices of the drying drum.

 
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per hour), a large amount of the material continuously remaining in the drum during its operation, and sticking of wet material on the internal devices of the drying drum.

In the gas-tube driers drying of the materials occurs under conditions of convective heat transfer and aerodynamic interaction between moving particles of the material with the gas flow. Terminal velocity, which is the main aerodynamic characteristic, is considered to be the main criteria of the gas flow rate evaluation. Among disadvantages of this type of driers are: inability to dry the fine coals of 0 - 0.5 mm and 0.5 - 2 mm (this is mainly due to the construction of feeders), high abrasive wear of vertical gas-tube driers and cyclones shells by coal particles, the complexity of drying handling associated with the appliance of the inertial brick furnaces with forward or back stroke grating.

Use of relatively low hot gas temperatures in the fluidized bed driers compared to direct-fired rotary driers and vertical gas-tube driers, reduces the thermal efficiency of these devices and increases in proportion the volume of flue gas in off-gas cleaning system and exhauster. This leads to high capital costs for the purchase of gas cleaning equipment and high cost of electricity for the transportation of gases during operation. Low temperatures of the hot gases make it difficult to provide self-initializing of drying tract due to return (recycling) of the part of the exhaust gases.

Application of industrial microwave emitters for coal drying is safe, however the cost of evaporation of one ton of moisture from the coal using industrial microwave emitters is by 9-10 times higher than with the convective thermal drying.

The cost of steam tube driers is about three times higher than the cost of similar convective dryers described above. However, when applied in thermal station scheme in combination with vapor cooling after the turbines[7], these devices have an absolute advantage as soon as practically "free" source of heat is used.

In the process of the coal industry restructuring in the 1990s the number of operating driers in Russia significantly decreased. The main reason of it was the high cost of coal drying and maintaining of safe working conditions of drying units.

In the 90s, new technologies for mechanical dewatering of coal at the coal washing plants in Russia were actively introduced. Application of screen bowl centrifuges of "Decanter" type and filterbelt sludge presses for the sludge dewatering significantly reduced the load on the thermal drying. At the same time, in order to reduce the flotation volume, the plants began to apply screens of dry classification of raw coals of 3 - 4 mm class, thus the conditions were created for the design of the coal processing plants of new generation with a significant reduction of capital costs for the construction and operation, since the processes of flotation and thermal drying of coal were totally or partially excluded. [4]

Starting from year 2000, the installation of thermal driers at the newly designed plants in Russia has not been foreseen. All new plants have been equipped with the equipment for intensive mechanical dewatering of coal with application of the precipitation filter centrifuges of "Decanter" type, chamber filter presses and belt filter presses. At the most of the newly built plants the processes of fine coals washing (flotation) have not been foreseen, and part of the sludge in form sediments of filter-pressing particles was directed straight into the waste or combined with the marketable coal concentrate, whenever quality characteristics of concentrate allowed it. A positive aspect of these changes is the lack of thermal drying. However, exclusion of fine coals washing process leads to the losses (sometimes significant) of coal with tailings.

In total since 2001, 17 coal washing plants were built and put into operation in Russia. The currently applied technologies of sludge dewatering do not provide obtaining a marketable coal with the moisture characteristics that meet customer requirements and correspond to applicable conditions of carriage by rail, especially in the winter season. After dewatering at the most modern equipment sludges (of 0-0.2 mm class) have 20-22% moisture, and after mixing of the entire slurry with the concentrate the total moisture of marketable products reaches 11-12%. As a result, today the coal washing plants designed and built without the use of the coal drying process, increasing their losses of coal containing in the waste. Thus, to ensure the quality indicator of coal moisture equal to 9% in the summer time and 7% in winter a part of the waterlogged sludge has to be sent to the waste. The appliance of thermal drying will help to avoid such losses, which can be seen from Table 1, were operation of coal washing plants with the driers and without them is compared.

Table 1 – Characteristics of coal washing plants

| Coal washing plant name | Date of completion/ Reconstruction | Feed stock | Cleaningd depth, mm | Project/ Actual capacity mln.t/year | Losses, % | Dryer type | Number of driers |
|---|---|---|---|---|---|---|---|
| CWP "Krasnobroskaya-Koksovaya " | 2011 | charred coal | 0 | 3 / 3.5 | 3.8 [8] | n/a | -- |
| OMPM “Krasnogorskaya” | 2002/2005 | energy | 0.15 | 1.5 / 2.5 | 2.6 [8] | n/a | -- |
| CWP "Severnaya" | 2006 | charred coal | 0 | 3 / 3 | 6-10.3 [8] | n/a | -- |
| CCWP “Sibir” | 1974 | | 0 | n/a / 3.8 | 0.7 [8] | SB3,5*22 | 3 |
| CCWP “Pechorskaya” | 1993/2003-2010 | charred coal | 0 | 7.1/ 6.7 | 1 | TS-1100 | 5 |
| CWP of “Severnaya” coalmine | 1970/2009 | charred coal | 1 | 3.4/4.2 | 0.5 | TS-1100 | 2 |

In TEIC conditions drying is necessary because the coal from the Elegest field has considerable fragility: sludging indicator (K = 38%) is almost 2 times higher than that of the coal of similar ranks from the Kuznetsk and Pechora basins (K <20%). It is intended for the production of coke concentrate and it is washed up to "0 "mm. Its low structural strength results in high secondary slugging during the washing process, which causes the presence of large amounts of sludge in the raw coal. The total moisture of coal concentrate will be 11.0%, which is higher than the threshold limit values for the transportation of bulk cargo in winter time by rail (8.0%).

A comparison of convective driers with direct contact of hot gases with a wet material has been made in order to select a variant of a thermal dryer (shown in Table 2). [6] The comparison has been made for the driers with capacity of 223 tons of finished product per hour with the initial moisture of coal equal to 14.0% and final moisture of coal equal to 6%.

The performed analysis allows considering the vertical tube driers with simultaneous use of quick-response hot gas generators, working at burning of coal dust, as the most efficient devices for drying the fine coals and slurries.

The vertical tube driers ensure: a) minimum coal-burning rate and electric energy consumption; b) the most secure coal drying process, including drying of coals with a high volatile matter content; c) simplifying of the housing seal due to the absence of moving parts (except coal feeders); d) the air suctions of atmospheric air at gas rarefaction put together less than 3 % in drying section and about 5 % in gas cleaning system; e) the required oxygen content (less than 9%) in the drying section during coal drying; f) a short time of material presence in the dryer; j) explosion pressure resistance.

Table 2– Characteristics of drying complexes applied in coal processing

| Process parameters | Unit of measure ment | Vertical gas-tube dryer | Fluidized bed dryer | Rotary dryer |
|---|---|---|---|---|
| Hot gas temperature | $^0C$ | 700-800 | 250-350 | 400-500 |
| Waste-gas temperature | $^0C$ | 90-100 | 90-100 | 90-100 |
| Quantity of evaporated liquid | kg/hr | 20.72 | 20.72 | 20.72 |
| Moisture evaporation in 1 m3 of drying chamber internal volume | $kg/m^3$ | 500-1500 | 200-300 | 70-100 |
| Dryer thermal output, winter | MWt | 28.5 | 41 | 32.7 |
| Heat rate for drying (evaporation of 1 ton of water) | MWt /t $H_2O$ | 1.27 | 1.59 | 1.37 |
| Volume of exhaust gases | m3/hr | 186 000 | 463 000 | 275 000 |
| Volume of exhaust gases per ton of evaporated moisture | $nm^3$.hr/t $H_2O$ | 9000 | 23000 | 13300 |
| Cumulative suctions of atmospheric air | % | Less than 10 | 15 | 20 |
| Quantity of the material in the dryer during the drying process | kg | 200 | 10 000 – 15 000 | 20 000 – 30 000 |
| Safety from explosion and fire hazard | | high | low | low |
| Area arrangement of the dryer, taking into account the gas cleaning system, fans, exhaust fans and flues | m x m | 12x40 | 15x80 | 15x80 |
| Specific amount of metal | tonn | 70 | 150 | 190 |
| Total electric power consumption | kWt | 635 | 2150 | 1390 |
| Installation term | months | 2-3 | 3-4 | 4-5 |

The interest of specialists to tube dryers re-emerged in the 90s at drying the granulated metallurgical slags and manufacturing the wood chipboards. The new design and schematic solutions had been developed that delivered the comprehensive tube driers from the majority of the shortcomings. The fundamental alterations have been made in design of the driers: the casing of tube dryer and cyclone elements were made with protection against abrasive wear: ceramics, stone casting, and special coatings. Comprehensive coal feeders are used to work with a material with a grain size of 0-2 mm; quick-response hot gas generators, working at burning of coal dust, are applied; their combustion chamber is made without the refractory lining for operation at the temperature of hot gases of 700 - 900°C.

The following serves as the basis for safe coal drying: a) reduced oxygen content inside the dryer due to its self-inertisation; b) return of some flue gases into the dryer; ensuring of a balanced load of the dryer with the wet coal and constant control of the load volume and the moisture content; c) utilization of quick-response hot gas generators, providing accurate and fast control of the drying process; d) ensuring of safety from explosion and fire hazard by choosing the optimal apparatus construction and its strength enhancement; e) air tightness securance, f) equipping the apparatus and the gas cleaning system with the protective explosive valves, as well as with the discharge chamber of sufficient volume to extinguish coal tanned. [7] The technologic complex on the basis of vertical tube dryer with quick-response hot gas generator, operating at burning of coal dust, with the dry gas cleaning system (fig. 1), ensures the safe coal drying, including the coals with a high content of volatile substances.

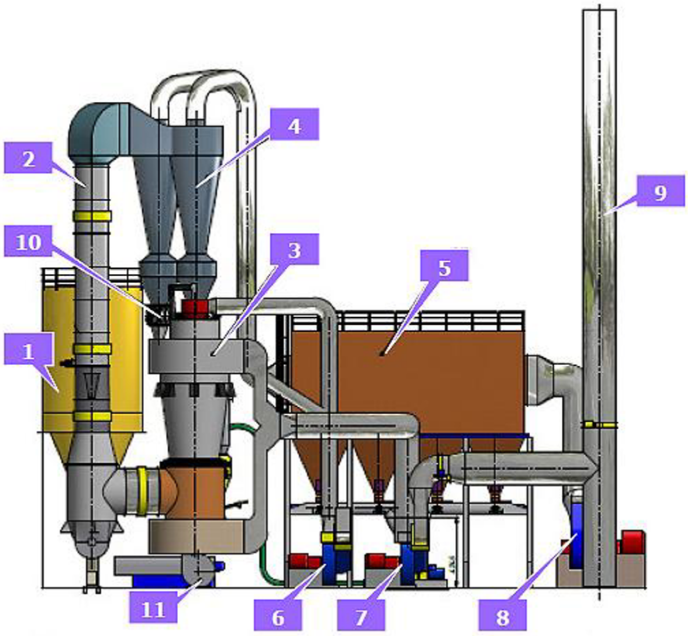


Fig. 1
1 – feed stock hopper; 2 – vertical gas-tube dryer; 3 - hot gas generator; 4 – off-load cyclones; 5 – sock filter; 6 – combustion air fan; 7 – mixing air fan; 8 – main smoke exhauster; 9 - waste-gas flue; 10 - rotary valve; 11 – fan for supply of coal dust to the burner.

The dryer has a cylindrical shape, the diameter of cylinder is 2800 mm, and the casing is made of steel elements. The moist material is supplied into the feed section of the tube drier by two sluice feeders which seal apparatus. The dryer is provided with an opportunity to pick up part of the dried product from under the cyclone and return it via the elevator into the feeding subassemblies of vertical tube dryer which can improve the fluidity and other rheological properties of the initial moist material. This is especially true when highly slugging substandard materials or materials with moisture content higher than estimated are fed for drying. [8]

The sock filters are applied in the gas and dust cleaning unit. Some of the coal dust taken from the sock filters before the exhauster goes for combustion into the hot gas generator, thus the unit supplies itself with the fuel. In addition, the use of sock filters for fine gas cleaning instead of wet dust collectors eliminates the circulation of fine coal slimes through flotation.

In the vertical tube dryers specific charge of coal dust makes up 14-15 kg/t of dry product when the moisture of coal is reduced by 10-12%. Depending on the cost of electricity and the cost of coal burned the cost of drying makes up 40-70 rubles per ton of dry product. Unit production costs for drying make up 1-3% of the cost of the dried coal.

**Conclusions**

The comparison of the structural, technological and operational characteristics of different driers has been made: gas-tube driers, direct-fired rotary driers, and steam tube driers; microwave driers, fluidized bed driers. Their constructive, technological and operational advantages and disadvantages have been detected. Parameters restricting the use of the driers have been detected: significant capital and operating costs; high explosion and fire hazard due to the high yield of volatile substances (up to 40%); environmental restrictions - the volume of gases emitted into the atmosphere at the average makes up more than 450 thousand m3/h.

The economic and technological feasibility of incorporating of thermal drying in the operation schedule of the projected GOK Elegest enterprise has been justified. It has been suggested to include a vertical tube dryer into the process system scheme together with a quick-response hot gas generator, operating on powdered coal, with a dry exhaust gas cleaning system which ensures the safe drying of high-volatile coal. The optimization of the processes of dewatering and drying has been performed: pre-dewatering of sludge concentrate in the precipitation filtering centrifuges and of low-ash sludge cake in the chamber filter press; the thermal drying of the 0 - 2 mm grade concentrate was provided in the drying room, consisting of 4 drying units (vertical tube dryer with 2800 mm diameter). Technical solutions have been applied that provide technological and environmental safety - optimum circulation of inert gases and cleaning of gases emitted into the atmosphere after the sock filters.

The chosen solution of process layout of coal washing allows to obtain, depending on the operating modes of the complex, the total concentrate moisture within 7.0 - 8.0%, and to avoid high losses - up to 1,5mln. tons of coal per year in the processing of 14.8 million. tons of coal.

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# The Usage of flocculants for the Processes of Thickening and Dewatering of thin coal sludges

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**Abstract**

The Usage of flocculants in Processes of Coal Sludges Thickening and Dewatering is an important factor for a stable CPP performance. A presence of fine clay particles greatly complicates Solid and Liquid Phases Separation Processes. An Effective Work of flocculants is complicated by harsh climate of the region (West Siberia).

A large number of different Dewatering methods require a special approach to an Application of Flocculants in each process. Existence of flocculants with various molecular mass and ionic activity allows distinguishing different polymer groups that performance effectively on any type of equipment.

The Authors have been researching a Performance of flocculants Magnaflock produced by the “BASF” Company. Conducted researches showed us that flocculants helps to intensify Dewatering Processes not only in Radial Thickeners and Belt Filter-Presses, but also in “Decanter” Centrifuges, Hyperbar-Filters and Chamber Filter-Presses.

Groups of flocculants have been selected which works effectively in different Processes of coal sludges Dewatering.

A possibility to apply a Selective Flocculation of coal-clay particles in Classifying Hydrocyclone has been shown.

Laboratory and industrial tests of a Usage of the rheology Reomax ETD modifiers in a Process of Water Clarification of Flotation Wastes in Hydraulic Mine Dumps have been conducted.

**Keywords: Flocculant, sludge, dewatering, thickening, filter-press, coal, clay**

## Introduction

Flocculants became widely used on the Coal Preparation Plants since the end of the 90s. The Process of Introduction of flocculants has been taken place primarily on the CPP of Kuzbass [1]. Here, in 1994 has begun its work the Technology of Thickening and Dewatering for a flotation wastes, without the Usage of Sludge Ponds (CPP “Belovskaya”). Technology of Thickening and Dewatering for the flotation wastes on the CPP “Belovskaya” consists of the following: flotation wastes with an ash component value 60-70% and density value 40-50 g/l are proceeding to the working chamber of a radial thickener. After that the prepared solutions of the anionic flocculant Magnaflock 156 and the cationic polyelectrolyte Magnaflock 1597 (produced by the “BASF” Company) are added to the working chamber for an intensification of the sedimentation process.

Formed precipitate with density value 300-500 g/l is transported by a pump into a buffer tank, which is designed for its averaging, and from there it’s proceed on Belt Filter-Presses.

A necessary condition for a stable and effective operation of a Belt Filter-Presses is also an application of flocculants: Magnaflock 156 and Magnaflock 1597. Moisture component value of dried wastes is 30-35% which allows their transportation by car to a Flat Rock Spoil Heap. A filtrate is headed back to a radial thickener.

Implemented technology allows the Closure of a circulating Water Supply of CPP and the Exclusion of a Hydraulic Mine Dump Usage. Dried flotation wastes are transported to a Flat Rock Spoil Heap into pre-prepared trenches. After that the wastes are piled with large particles from a gravity concentration to a

 
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height of 2 m, and then the Waste Disposal Process repeats. That method of Flotation Waste Disposal with the Usage of Flat Rock Soil Heaps eliminates dusting and environmental pollution [2].

In the early 2000s a significant part of the CPP of Kuzbass has accepted the Technology with the Usage of Filter-Presses instead of Sludge Ponds as a result of successful Technology Implementation. In the same time have been started the Process of a building of a new CPP, without the Usage of External Sumps. All this have been resulted in the increasing Importance of flocculants for a modern CPP [3, 5].

**The Necessary Conditions for an Application of Flocculants**

At present time the Basic Operations of Application of polymers in a Coal Preparation Processes are: a Thickening of coal sludges and flotation wastes, a Dewatering of coal sludges and flotation wastes on a Belt Filter-Presses, a Dewatering of a coal concentrate and low-ash sludges on a Disc Vacuum-Filters, Chamber Filter-Presses and Hyperbar-Filters [4].

Nowadays one of the Characteristics of the Application of flocculants is a constant Degression in particle size of a processed material. But what are the preconditions for such phenomenon? Here we must mark a few reasons. The Changes in the Technology of a Coal Preparation have been resulted in a significantly thinner flotation feed. The Changes in the Mining Technology have led to the great amount of fine coal particles in the feed of a CPP. A general drop in quality of a raw coal has also been the cause for the increase in the clay particles content in the feed of a CPP. All of this demands very high efficiency rate for the Application of flocculants in a Process of a thin coal Preparation [5].

Climatic conditions are very important in Siberia. CPP need to receive a product with the minimal moist content. Considering that particle size of a processed material, which usually is less than 100 micron, and very often the 70% of it is ever less than 40 micron, we can say that it is a very difficult task [9, 10].

The Authors have been researching the Possibility to apply anionic and cationic flocculants “Magnafloc”, which have been produced by the “BASF” Company (Germany), in a various Dewatering Processes. Most effective flocculants in Processes of Thickening in a radial thickeners and Dewatering on a Belt Filter-Presses and Disc Vacuum-Filters have been identified.

**A Process of Thickening flotation slimes and tailings in a Radial Thickener**

In the thickening of slimes in a radial thickener, the main aim is to obtain an overflow with as little as, less than 1 g/l, solids possible with a high solid content, 250 - 450 g/l, 600 g/l maximum, in the underflow. Thus, it is important to receive a high rate of sedimentation. As the thickeners in modern plants are of rather small diameters, 12 - 18 meters, specific productivity of a thickener depends on the work of flocculants. As a rule, at coal-clay preparation plants, anionic and cationic flocculants are both jointly used, since plenty of fine clay particles cannot be flocculated by an anionic flocculant alone. There are exceptions. For example, some types of coals with low ash slimes are effectively clarified by anionic flocculants alone as the amount of clay particles in them is insignificant. For example, CPP « Krasnogorskaya» and CPP «Mezchdurechenskaya», coal types "A" and «T» have the insignificant amount of clay particles. Thus, an anionic flocculant, Magnafloc 155, is enough to thicken the slimes. The use of cationic products in this case is not required.

It is important to note, that, usually, for thickening, high-molecular weight anionic flocculants, MW more than $15 \cdot 10^6$, are used. Large flocs with a higher settling rate are created by high-molecular weight polymers [4].

**A Dewatering of flotation slimes and tailings on Belt Filter-Presses**

Unlike processes of clarification, during the Process of Dewatering slimes on Belt Filter-Presses the primary goal of polymer is to provide as much as possible a fast feedback of water from the thickening sediment. It allows receiving the minimal moisture in the cake. Thus, cleanliness of a filtrate is not the final result in itself, but only an indirect indication of effective work of polymers. Joint application of anionic and cationic products, is necessary in a case of thickening of slimes. Flocculant should form stronger flocs compared to the thickening as they experience more mechanical forces.

It is possible to compare the work of flocculants of identical charge, but of a different molecular weight. As an example we shall take Magnafloc 919, ionic activity of 45 %, molecular weight 25 million and Magnafloc 345, ionic activity of 30 %, molecular weight 10 million. Tests were carried out in the flotation tailings of CPP «Belovskaya» and the thickening product of radial thickener of CPP «Belovskaya».

Tables 1, 2 evidently show that high-molecular weight flocculants work more effectively in thickening and middle-molecular weight ones worked in dewatering of the flotation tailings.

Therefore, anionic products are the most effective flocculants with molecular weight 10 - $15 \cdot 10^6$ during dewatering in the belt presses. They easily give enough moisture and at the same time, form strong flocs. Application of cationic products, first of all, is caused by the reception of a good branch of a cake from a tape and a high water-feedback at an initial stage of dewatering. At coal preparation plants of Kuzbass, Magnafloc 345, Magnafloc 156, Magnafloc 5250 are most effective for processes of dewatering [4].

Table 1. Results of thickener tailing of flotation in CPP «Belovskaya»

| № Experiences | Flocculant | Dosage, g/t | Rate of sedimentation, sm/c | Maintenance of a particle overflow, g/l |
|---|---|---|---|---|
| 1. | Magnafloc 156 | 100 | 0,4 | 3,0 |
| 2. | Magnafloc 156 + Magnafloc 1597 | 80<br>40 | 0,7 | 0,4 |
| 3. | Magnafloc 919 | 80 | 0,6 | 1,0 |
| 4. | Magnafloc 919 + Magnafloc 1597 | 80<br>40 | 0,8 | 0,2 |

Table 2. Results of dewatering tailing of flotation in CPP «Belovskaya»

| № Experiences | Flocculant | Dosage, g/t | Maintenance of a particle filtrate, g/l | Moisture cake, % |
|---|---|---|---|---|
| 1. | Magnafloc 156 +Magnafloc 1597 | 60<br>30 | 0,6 | 32 |
| 2. | Magnafloc 919 +Magnafloc 1597 | 60<br>30 | 0,9 | 41 |

**The Research of a Possibility to apply flocculants on different types of a Dewatering Equipment**

Usually, the Producers of some types of a Dewatering Equipment believes, that some types of equipment can achieve required technical and economic performance without the Usage of flocculants. But, because of a high clay particles content and particle size distribution, Dewatering Equipment cannot achieve required performance. It is an actual problem for a Hyperbar-Filters, “Decanter” Centrifuges and Chamber Filter-Presses. We have selected the most effective groups of flocculants for a Dewatering on a Hyperbar-Filters and Chamber Filter-Presses. We have received good technical and economic parameters when flocculants have been introduced to CPP of Kuzbass. An Implementation of polymer Magnaflock 1597 in a Process of Coal Sludges Dewatering on Chamber Filter-Presses was carried out on some CPP. The Application of polymer allowed increasing productivity of an equipment by 25-40% with a consumption rate 50-70 g/t. As a result of the Application of flocculants the moisture content has been reduced and the efficiency of coal preparation equipment has been increased.

The researches of the Application of flocculants have been continued in the Processes of a Coal Dewatering in a “Decanter” Centrifuges. The results of industrial tests have shown to us the possibility to use flocculants on this type of equipment. Some of the polymers are resistant to the destruction in a centrifugal field, and the unit consumption of these polymers are not as high, as it was before [6, 7].

The most effective high-molecular products are: Magnaflock 919, 336, 338. They sufficiently lower a value of a solid content in fugate, to 5-8 g/l and increase Centrifuges productivity to 25-30%.

**The Usage of the Selective Floculation for Coal Sludges Processing**

A Selective Flocculation of coal sludges is not a completely researched process. Many scientists have been developing this method, but they never got to an industrial application of this method [6].

On CPP of Kuzbass the Selective Flocculation on an industrial scale took place for the first time on the CPP “Kuzbasskaya”.

This was preceded by the conducting of extensive laboratory researches []. The feed on the CPP “Kuzbasskaya” was consisted of a coal from the “Raspadskaya” Mine. That coal was oxidized and their flotation conducted poorly. Laboratory researches have shown a possibility of a Coal Sludges Processing through the Selective Flocculation Method. As a result the CPP was reconstructed and a Flotation Process was replaced by the Selective Flocculation. During construction of the CPP “Raspadkaya” the Selective Flocculation was already a part of a Coal Sludges Processing Project. For an effective separation on the CPP were used flocculants Magnaflock of the “BASF” Company. Nowadays the CPP “Raspadskaya” using flocculants Magnaflock 155, Magnaflock 5250 and Magnaflock 1597 [8].

The Authors for the first time have been able to research the possibility to selectively flocculate coal sludges on hydraulic cyclones under laboratory conditions. Some of the CPP, in example, CPP “Sputnik”, have been implemented the fourth stage of a coal sludges hydrocycloning. The main purpose of this operation is to reduce the amount of coal particles, which have proceeding into a Thickening Process together with coal sludges. We have lined up the group of flocculants, which allows us to increase the selectivity of a Hydrocycloning Process. As a result, ash content of a suldges in the feed of a Thickening Process has been increased from 26% to 56%. Also yield of a concentrate have been increased, and a flocculants consumption rate have been decreased.

**The Usage of the rheology Reomax ETD modifiers for a Storage of Coal Processing Wastes**

Some of the CPP are still using an External Sumps. Among them are CPP “Sibir”, CPP “Kuznetskaya”, CPP “ZSMK”, CPP “Procopyevskugol”, CPP “Koksovaya”.

Because of modern requirements, which are forbidding a Building of new Hydraulic Mine Dumps, CPP are forced to search some technological methods in order to increase a life time of already existing Hydraulic Mine Dumps. Among those methods there are Intensification of Water Purification and Recycling Processes on CPP.

The Chemical Concern “BASF”, Germany, has developed new reagents, modifiers of the rheology Reomax ETD. Those products are designed especially for a Storage of Wet Processing Products in External Sumps.

A Pulp, which has been processed with Reomax reagent, is able to quickly lose large volume of water and a precipitate dewaters intensively. As a result of Reomax reagent Usage on solid particles, it is possible after just 10-14 days to use heavy machinery for the purpose of Dewatered Sludges Exportation from a territory of a Tailing Pond.

We have conducted laboratory and industrial tests on some of the CPP of Kuznetsk Coal Basin.

As an example, let's take one of the CPP. A main Processing Method on that Facility is Dense Medium Separation. Sludges with density value 80-100 g/l are headed to Hydraulic Mine Dump. Those sludges are almost not clarified, which leads to quick filling of a Hydraulic Mine Dump.

An efficiency of a Reomax reagent is estimated by a Spreading Angle of a Precipitate and Obtaining Rate of a Filtrate.

Research data are listed in table 3.

Table 3. Results of a Reomax reagent Estimation

| Product | Reagent consumption, g/t | Polymer concentration, % | The Number of mixings | Spreading angle of a Precipitate , % | Amount of filtrate obtained, ml |
|---|---|---|---|---|---|
| REOMAX ® 9010 | 100 | 0,1 | 7 | 3 | 88 |
| | 125 | 0,1 | 7 | 5 | 90 |
| | 150 | 0,1 | 7 | 6 | 94 |
| REOMAX ® 9060 | 100 | 0,1 | 7 | 2 | 80 |
| | 125 | 0,1 | 7 | 4 | 85 |
| | 150 | 0,1 | 7 | 5 | 90 |
| REOMAX ® 9050 | 100 | 0,1 | 6 | 4 | 88 |
| | 125 | 0,1 | 6 | 6 | 94 |
| | 150 | 0,1 | 6 | 7 | 96 |

Data in the table shows us, that the best result is reached with a Usage of Reomax 9050 reagent. That reagent has most the widest optimal operating range along with quite effective results. And so, precisely that product was recommended for industrial tests.

Conducted industrial tests showed good results, and currently the CPP purchases this reagent for an industrial use.

**Conclusions**

Performed laboratory and industrial tests show us, that modern polymeric flocculants allow achieving of a high Technological Parameters in the Processes of Thickening and Dewatering of thin coal-clay sludges on a various Dewatering Equipment.

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# Granulating of coal fines using peat

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**Abstract**

The paper determines the optimal composition of fuel to use in solid-fuel boilers. The primary goal of this report is the development of the combination of coal fines with peat without additives and binders for producing a more stable and cohesive granules. Another objective is a peat that to reduce the emission of sulfur dioxide during the coal peat fuel combustion and increasing the efficiency of the briquetting process. Finally, an objective of the report is a peat that can be produce inexpensively. Therefore, according to these and other objectives, the present report consists of the combination of coal fines with peat. The coal fines with peat are mix and the resulting well mixed mass is then pelletizing by the application of pressure in conventional equipment – stiff extrusion. The coal fines and peat was conditioned at various moisture contents to observe the effects on the finished product. The material then agglomerated into a granular product. The content of the granular product was 2:1, 1:1, 1:2 by volume. When the moisture content was increase to 65% by weight, moist pellets were formed. The next step in the evaluation was to determine what actual condition was acceptable to blast furnaces. A simple test was performed on the agglomerated coal fines to confirm the dedusting efficiency.

**Keywords**: coal fines, peat fuel, mixing, pelletizing, granular product

## Introduction

Considering important message of world standard day, 2012, "less waste, better result", improving the ways to decrease waste is too valuable. Briquetting or pelletizing is a method to utilize coal fines, as a byproduct of mining, to decrease loss of this energy source [1].

There are a large number of facilities throughout the world, which handle coal, such as preparation plants. Many other facilities use coal as a fuel. Coal material handling, crushing operations, air tables, drying, and storage is by far the most polluting in terms of emissions, releasing high levels of coal fines and dust. In coal mining processing and handling, enormous tonnages of coal fines are created. Coal fines whether cleaned or reclaimed from waste ponds, are typically in the form of a wet cake that is difficult to handle, store and transport [2].

Typically, after handling and cleaning is completed, about fifteen to twenty percent of the tonnage mined consists of fines ranging in size from powder to small granules. For the most part, these fines are not directly usable, thereby leaving great quantities of material that is wasted and represents a hazardous and expensive disposal problem. Accordingly, coal-fines handling, storage and disposal operations represent a significant and unproductive expense for the industry.

Even when blended with cleaned coarse coal product, the fines can cause plugging in chutes and hang-ups in transport systems. If stored in piles, contamination of water runoff is possible. Thermal drying of fines is prohibitively expensive, and the product can create a dust problem or even an explosion hazard.



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Perhaps of greater significance is the loss of fines that may occur during rail transport, and the resulting environmental damage that affects areas near these transport systems. One solution to problems associated with coal fines is reconstitution via pelletization. It has been shown that binders such as corn starch, lignosulfonates, Shur Bond, asphalt emulsions, and lignocellulosics, can be employed to produce a strong and durable final product [3].

As a result of these problems, as well as of strict customer demands with respect of coal quality and of increasingly stringent regulation of mine waste disposal practices to satisfy environmental standards, coal fines utilization has been recently reexamined by the industry. In the past, fines have been used mostly for manufacturing briquettes for home and commercial heating. Coal briquetting technology focused on low pressure agglomeration of coal fines, using a binder, typically of coal tar origin, to hold individual particles together. The fines material containing between about twenty and thirty percent moisture, depending upon its size distribution and ash content. In a dry state, the fines are generally predominantly passable through a 28-mesh screen, a size that may be used for pelletizing or briquetting purposes. As used in this disclosure, the terms pellet, briquette, log and block are used interchangeably and are intended to refer to all forms of pellets, briquettes, logs, blocks and other coal agglomerates produced by binding coal fines into a concrete material [4].

However, most binders studied to date significantly increase the cost of coal reconstitution, thus limiting the use of this technology. Therefore, a low-cost, or even negative cost, binder and pelletization process is needed [5].

The positive properties of the granules, such as stability during transport, high bulk density, low dust formation etc., at minimum costs for process. On the other hand, peat derived from peat deposit is using as a local fuel [6].

Peat is generally regarded as a low quality fuel source because of its high water content in addition to lower fuel efficiency. On average, peat contain up to about 80% moisture by weight, which causes a substantial percentage of the energy produced by these peat being used to dry them prior to full combustion.

Peat is particularly interesting due to its excellent binding properties, which makes a combination with all materials possible. The primary goal of this report is the development of the combination of coal fines with peat without additives and binders for producing a more stable and cohesive granules. Another objective is a peat that to reduce the emission of sulfur dioxide during the coal peat fuel combustion and increasing the efficiency of the pelletizing process. Finally, an objective of the report is a peat that can be produce inexpensively. Therefore, according to these and other objectives, the present report consists of the combination of coal fines with peat. The coal fines with peat are mix and the resulting well mixed mass is then pelletizing by the application of pressure in conventional equipment – stiff extrusion. The coal fines and peat was conditioned at various moisture contents to observe the effects on the finished product. The material then agglomerated into a granular product.

## Materials and methods

Materials used in the research were low-caking coal taken from Kuzbass and peat that used as binder were also found in Karelia Republic (Table 1). Peat decomposition degree is H4 on the von Post scale. Why peat used as a binder is supported by the fact that peat contains a bitumen’s and humus that is possible to act as the binder, besides, the peat itself is renewable energy source.

**Table 1** Characteristics of the coal and peat

| Sample | Calorific value, kJ/kg | Moisture, % | Volatile matter, % | Ash, % | Elemental analysis, % | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | | C | H | N | O | S |
| Low-caking coal | 23.4 | 10.2 | 40.9 | 16.3 | 84.6 | 5.1 | 2.1 | 8.2 | 0.60 |
| Grass moss peat | 12.8 | 76.2 | 70.0 | 6.5 | 57.4 | 6.0 | 1.4 | 35.1 | 0.14 |

Study was performed at the following conditions. Product coal/peat graniles have a cylindrical form with diameter 16 mm and height 25 mm, extrusion pressure was 40 MPa. In experiments on the piston press force of replacement of weight was registered as function of time and movement of the piston. The test of replacement lasted 10 seconds for steady formation. Ratios of the coal/peat was 1:2; 1:1 and 2:1(v/v).

The peat sample was thoroughly mixed by hand and placed in a plastic bag for granulating experiments. After preliminary separation were removed large wood inclusions from peat samples. Preliminary dewatering of peat was carried out on a fabric filter press. The separated and dehydrated peat were subjected to mechanical processing - attrition. Attrition of peat raw materials is carried out for the purpose of preliminary mechanical processing on laboratory chopper for a further extrusion.

In the raw material preparation step, coal was crushed and sieved with passing the sample through a 24-mesh sieve (0.707 μm) a size that may be used for pelletizing purposes. The procedure begins with passing the sample through a 24-mesh sieve (0.707 μm) and then mixing the minus 20 mesh material with peat for produce homogenous mixture. After stirred the mixture then put into the piston press to produce the granules. Figure 1 presents the steps of the granules preparation.

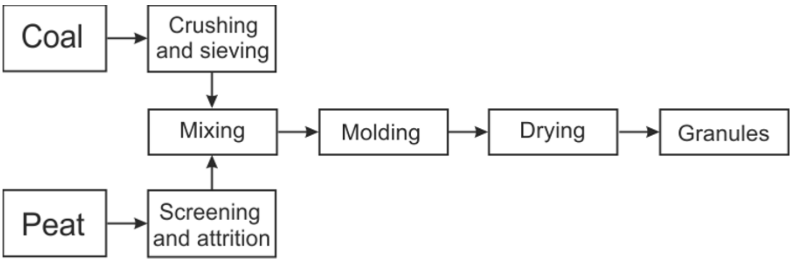


Fig. 1 Steps of granules preparation

Stiff extrusion technology for coal waste recycling combines high torque and high pressure to process coal fines with low moisture content, requiring no additives and fewer processing steps.

Extrusion of coal-peat mix was carried out on a piston press with a diameter of 32 mm and 300 mm long, established by universal Zwick/Roell Z100 Machine. Breakdown of mix via dies with a length of calibrating part of 60 mm and with a diameter of 15 mm at a speed of 0,0125 m/s and ambient temperature +20 °C. The experimental installation is submitted in figure 2.

Fig. 2. Experimental installation on universal Zwick/Roell Z100 Machine and granulation process

Experiment on drying of granules is made on experimental drying installation in a motionless layer with a constant speed and air temperature (+35 °C) with an atmospheric pressure. Air in a drying chamber moved an electric fan heater.

The drying chamber 300 mm high and with a diameter of 85 mm was also isolated 10 mm by a layer of mineral wool and wrapped up by aluminum foil, for elimination of radial temperature gradients. Air temperature in a drying chamber was taken by the thermometer with an accuracy of ±1 °C.

Experiments on drying were made at relative humidity of air of 30% and atmospheric pressure of 758 mm of mercury column and temperature of air of +23 °C.

Damp granules kept within the cartridge, which was suspended in a drying chamber at the height of 100 mm from an output nozzle of a fan heater. The area of the cartridge is 0.08 $m^2$.

The mass of granules before drying, and also the mass of container decided on laboratory scales of VLKT-500 on accuracy ± 0.01 g.

Granule strength was evaluated by single granule compression tests using a «Tinius Olsen» Universal Testing Machine. The «Tinius Olsen» Universal Testing Machine allows recording a maximum force, sufficient to break all of the tested granules.

Fixation of the loading operating on a sample and the deformation arising thus was carried out in the automatic mode.

At an assessment of physic mechanical properties of granules determined the sizes, density, durability in accordance with Russian Standard [7].

**Results**

Based on the theory of binding, binding force is the sum of physical and mechanical binding mostly is the mechanical binding, also the results of mechanical combination when the binder comes into the internal void of the binding substance [8]. Peat as a binder is to use such ways to process natural high polymer organic compound raw material. Peat is with the property of high surface activity lowered tension strong capability of dispersion.

Fine colloidal particles of peat promotes process of granules consolidation. From factors of group composition of peat, the humic acids (HA) and the highly hydrolyzable (HH) have the greatest impact on strength of granules (Fig 3).

Fig. 3. Experimental coal/peat granules (moisture content 25%)

In the process, a mixture of peat and coal is extruded into pellets, which are dried and cured to a durable form. The key element in the process is peat that is composed of bitumens humus, cellulose fiber and lignins. These components give peat special characteristics for binding materials (Table 2).

**Table 2** Components of organic matter of peat

| **Peat composition** | **Dry (average), %** |
|---|---|
| Bitumens | 7.3 |
| Water soluble matter, easily hydrolyzed matter | 33.7 |
| Humic acids | 26.1 |
| Fulvic acids | 16.3 |
| Cellulose | 6.2 |
| Lignin | 9.2 |

The presence of peat natural binders decrease sulfur and residual ash of granules, while they make a negligible decrease in calorific value. Advantages of peat-based materials include low sulfur content, and lower NOx production.

The main functional components in humic substances, including bitumens, humic acids and fulvic acids, could provide sufficient strength for the wet and dry granules [9]. Content of humic acids is higher than 15% and bitumens more than 3-4% increase the inhibiting and consolidating abilities of peat binding. Strengthening of rheological, structural and mechanical properties of mix is observed.

The jointing and density which in turn depend on extent of mechanical processing of material, and also technological indicators of processes of formation and drying has essential impact on strength properties of fuel granules. Table 3 show the parameters of cylindrical granules.

**Table 3**. Parameters of cylindrical granules

| Sample coal:peat (volume) | Average (wet granules) | | | | | Average (dry granules) | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | diameter, mm | length, mm | volume, $m^3 \cdot 10^{-6}$ | moisture, % | mass, g | diameter, mm | length, mm | mass, g | volume, $m^3 \cdot 10^{-6}$ | density, $kg/m^3$ | shrinkage, % |
| 2:1 | 16.0 | 25.0 | 5.026 | 34,4 | 6.14 | 15.5 | 24.0 | 4.18 | 4.54 | 922 | 9.73 |
| 1:1 | 16.0 | 25.0 | 5.026 | 49,7 | 7.24 | 14.9 | 22.8 | 3.23 | 3.98 | 812 | 20.81 |
| 1:2 | 16.0 | 25.0 | 5.026 | 60,5 | 8.14 | 14.3 | 20.3 | 2.48 | 3.26 | 761 | 35.07 |

The organic functional components in peat are chemically adsorbed onto surfaces of coal particles so that the granules strength is improved.

A suite of tests, which showed, characterized these granules:

- Dry granule strength measurements show that, these coal-peat granules have sufficient strength for production. The normal dry granule strength is 1.9–3.0 MPa to crush the granules.
- Experiments on mechanical strength of granules were make, when granules could be dropped from a height of 2.0 m on a concrete floor and survive. The mechanical strength of cylindrical granules made 96.5% [10].

## Conclusion

This research focused on the evaluation the feasibility of coal fines with peat granulation. Fuel granules were prepared using coal and peat, as main components.The results showed that the best composition and process condition to achieve high physical stability of granules would be coal to peat content ratio 1/1.

The organic functional components in peat are chemically adsorbed onto surfaces of coal particles so that the granules strength is improved.

The commercialization of pelletizing technology that incorporates coal fines should sufficiently prove the economics and emissions reduction potential of coal/peat fuels to justify continued research into alternative pelletized fuel feedstocks such as peat.

The equipment and the technology of the stiff-extrusion make it possible to produce the extrusion granules having high strength.

Thus, the conducted preliminary researches showed prospects of coal fines with peat granulation method. Further study is being undertaken to analyze coal/peat granules in a evaluating the combustion and emissions of coal/peat granules tests and of the water resistance defined as the time required for fuel destruction and dispersion in the water medium.

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# DEVELOPMENT & APPLICATION OF HIGH EFFICIENCY THICKENER WITHOUT MOVING PARTS

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## ABSTRACT

Conventional thickeners and high capacity thickeners currently dominate coal preparation industry in the world in spite of the other kinds of thickeners, such as deep cone thickeners and lamella type thickeners, being used in coal industry as well. However, either large amount of maintenance or small capacity is involved in those thickeners mentioned above in spite of their extensive use in coal preparation industry currently.

This paper will introduce a new high efficiency thickener recently developed by Beijing Guohua Technology Group (BGTG) in China and proven with its simplicity and high efficiency from its plant tests and industrial application. Compared to the conventional and namely high capacity thickeners with motor-driven rakes, the newly developed high efficiency thickener has much higher unit handling capacity and much less maintenance due to its non-moving parts and high thickening efficiency; industrial application of the new high efficiency thickener showed that its throughput reached over 9 $m^3/m^2/h$ with overflow solid content of 0.14% or 1.4 g/L at feed (+90% of -125 micron) solid concentration of 3.68%, thickening efficiency of 90% and underflow solid recovery of 96.6%. Footprint area of the high efficiency thickener is reduced considerably by over 75% compared to the conventional rake thickeners.

**Keywords**: Conventional, High capacity, High efficiency thickener, no-moving parts.

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## INTRODUCTION

Thickeners are widely used as sedimentation equipment in coal preparation industry for fine coal slurry before dewatered. The thickeners used in coal preparation industry can be classified into conventional thickeners, high capacity thickeners, deep cone thickeners and other specially structured thickeners such as multiple-plate Lamella thickener.

Wherever in China or the other coal producing countries the conventional thickeners are still predominantly utilized in coal preparation plants in spite of some industrial applications of high capacity thickeners and deep cone thickeners. Application and performance of the high capacity thickeners in China coal industry was rarely reported but there is no detailed application information available. Attempt of modification of conventional to high capacity thickeners was reported in China (Lu, Z., 1995 and 1996). The deep cone thickeners are not very popular either in China coal preparation industry (Guo, Z., 2015). Most of deep cone thickeners are currently used in the front of belt filter presses in order to obtain a high solid content of the underflow in coal preparation plants in China. Besides the conventional thickeners, however, another type of thickeners, multi-incline-tube thickeners, are being widely used in China coal industry since around 2000 (Huang G. and et al., 2011, Ma, S. and et al., 2012).

It is well known that the conventional thickener occupies a very large footprint due to its low throughput for a large capacity coal preparation plant. Typical capacity of the conventional thickeners is



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designed at 1 gallon/min/ft$^2$ or 2.4 m$^3$/h/m$^2$. The capacity will vary with coal tailings particle size distributions and solid contents. The capacity, regulated for the conventional thickener in the "Code of Design of Coal Cleaning Engineering" in China, is 1.6 – 2.4 m$^3$/h/m$^2$ for flotation tailings (Deng X. and et al., 2006). High capacity thickeners are characterized as being modified conventional thickeners which usually larger and deeper feed wells (Leonard J., 1991). The throughput of the high capacity thickener in the plant design is generally considered at 3 times of the conventional thickener in USA and overseas. However, there is common characteristics for the both of conventional and high capacity thickeners, i.e., both of them are manufactured and equipped with rotating rakes at the sloped bottom to push the settled fine coal particles toward the central discharge outlet. There is major maintenance and operating cost always involved with rotating rake mechanism.

Beijing Guohua Technology Group (BGTG) developed the multi-incline-tube thickeners in 2000. There are over 500 units of the multi-incline-tube thickeners used in coal preparation plants in China and oversea due to their smaller footprint, higher throughput and lower capital and operating costs without moving parts compared to the conventional thickeners at the same capacity. The footprint of the multi-incline-tube thickener is about 50% of that occupied with the same throughput conventional thickener due to its double capacity compared to the conventional thickener's capacity. Most important characteristics of the multi-incline-tube thickeners is without any moving parts, which reduces operating cost due to its almost no maintenance involved during normal operation.

Large size thickener with large capacity is urgently required in China with the rapid increase of the number of >6 Mt/a large capacity coal preparation plants. There will be 2500 – 3000 m$^3$ of fine tailings slurry in its operating circuit in a 6 Mt/a coal preparation plant. Therefore, it is necessary to develop a new high efficiency thickener for China coal preparation industry to further reduce its footprint area, lower operating cost and capital investment.

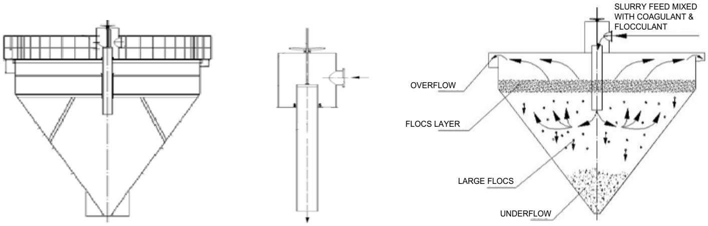
## DEVELOPMENT OF NEW HIGH EFFICIENCY THICKENER & ITS WORKING PRINCIPLE

### Consideration in the Development of the New High Efficiency Thickener

The development of new high efficiency thickener was initiated by BGTG in 2013. The development of the multi-incline-tube thickeners is based on the principle of the improvement of settling rate or throughput resulting from the increase of sedimentation surface area of the thickener without any increase of flocculant and coagulant dosages. The mechanical rotating rake system, equipped with the conventional thickener, was eliminated in the last generation of the multi-incline-tube thickeners. Many industrial practice demonstrated successful operations, higher (doubled) capacity and much less operation maintenance without any moving parts compared to the conventional thickeners. However, its throughput seems still lower than the high capacity thickeners developed and used in North America and overseas. Therefore, the target of development of new high efficiency thickeners is very clear, i.e., to further increase throughput capacity and keep the feature of without moving parts.

### Structure of the Developed High Efficiency Thickener

The high efficiency thickener is designed with cylindrical top and conical bottom steel structure (see Figure 1-(a)). The steel structure is much easy to fabricate and could reduce installation time significantly, especially in cold weather area. The cone pyramid is designed at 45$^o$ – 65$^o$. Compared to the conventional thickeners and the multi-incline-tube thickeners, the total volume or footprint of the high efficiency thickeners is significantly reduced in the design. Removal of the multi-incline-tube block also completely eliminates occasional plug of the inclined tubes due to accident overload or large impurities in the feed.




*(a)* *(b)* *(c)*

*Figure 1. Sketch of the structure and feed well of the high efficiency thickener*

There is a feed well device designed and installed in the middle of the cylindrical section of the thickener (see Figure 1-(b)). The feed well includes main feed well, thickener slurry feed inlet and feed outlet pipe. A thread rod-wheel system is designed to connect the top of the feed outlet pipe with the main body of the feed well. The feed outlet pipe can be adjusted up and down with the adjustment wheel. The bottom of the feed outlet pipe can be adjusted and inserted down to anywhere from 2 to 3 meters below the overflow surface of the thickener. The best operation performance can be obtained with the feed depth adjustment based on feed characteristics and feed throughput.

**Working Principle of the High Efficiency Thickener**

The slurry feed to the thickener is already pre-mixed well with coagulant and flocculant. The pre-flocculated slurry feed is fed to the center point of 2 to 3 meters below the overflow surface of thickener through the feed inlet, the main feed well and the feed outlet pipe.

Figure 1-(c) shows the settling process of particulate flocs in the designed thickener. The large flocs go downwards to the bottom of the thickener due to effect of gravity on the flocs whose settling velocity is higher than the water upward flow velocity. The small flocs will flow upwards since their settling velocity is lower than the water upward flow velocity. The horizontal cross section area of lower part of the thickener gradually increases upwards, which results in the gradually reduction of the water upward flow velocity. On a certain level in the thickener, the settling velocity of the flocs will equal to the water upward flow velocity. The flocs will stay and suspend on this level. More and more flocs will accumulate around this level to form a thick flocs layer.

With the above process continues, more and more large flocs sink to the bottom of the thickener and are then pumped out. The water goes through numerous water channels between the flocs upwards to the top surface of the liquid body in the thickener and flows out as overflow from the round edge of the thickener.

The thick flocs layer formed on the certain lever of thickener has two functions: 1) Prevent very small flocs from continuing to flow upwards, which is acting as a filter, and 2) More and more flocs are kept in the flocs layer and interact and become bigger flocs, which will accelerate settling of the flocs downwards to the bottom of the thickener.

**Development of Industrial High Efficiency Thickener**

***1) A Pilot Unit Test***

A pilot high efficiency thickener unit was developed before an industrial unit was designed and fabricated. The purpose of the pilot unit was to conduct a pilot test study based on the above working principle to observe the thickening process phenomenon to receive firsthand information and preliminary process design parameters. The small pilot unit was made with ϕ300 mm plexiglass material and constructed with 2000 mm of cylinder height and 220 mm of conical section height. The flocculation and settling processes could be observed through the transparent plexiglass material.

The pilot unit was tested in Tangshan Hongju Coal Preparation Plant in China. The slurry feed was from the plant flotation tailings. The results of one set of tests on the pilot unit are shown in Table 1. The tests were conducted at the throughput varying from 6.64 $m^3/h/m^2$ to 14.15 $m^3/h/m^2$ to see if our designed concept is working and verify its capability. The test results reached at our goal.

Table 1. Test results of the pilot scale thickener

| No. | Feed capacity $m^3/m^2/h$ | Feed solid content % | Under-flow solid content % | Overflow solid content % | Thickening efficiency % | Clarifi-cation coeff. | Underflow (Solid, liquid) yield % | Underflow solid recovery % |
|---|---|---|---|---|---|---|---|---|
| 1 | 6.64 | 2.54 | 24.51 | 0.15 | 87.07 | 0.94 | 9.81 | 94.67 |
| 2 | 8.22 | 2.54 | 17.21 | 0.28 | 79.11 | 0.89 | 13.35 | 90.45 |
| 3 | 10.42 | 2.5 | 15.79 | 0.16 | 81.63 | 0.94 | 14.97 | 94.56 |
| 4 | 10.42 | 2.58 | 17.42 | 0.12 | 83.96 | 0.95 | 14.22 | 96.01 |
| 5 | 11.39 | 2.54 | 18.96 | 0.11 | 85.51 | 0.96 | 12.89 | 96.23 |
| 6 | 14.15 | 2.54 | 19.12 | 0.19 | 83.15 | 0.93 | 12.41 | 93.45 |
| Avg | | 2.54 | 18.84 | 0.17 | 83.40 | 0.93 | 12.94 | 94.23 |

***2) Modification of A Multi-Incline-Tube Thickener***

The multi-incline-tube thickener developed by BGTG is equipped with multiple incline tubes on the upper section of the thickener and without any moving parts at the bottom. Its structure is very close to the high efficiency thickener which BGTG planned to develop. Therefore, BGTG modified one of the thickeners installed in Shanxi Jingfang Coal Preparation Plant in China primarily by removing all the multiple incline pipes and slightly changing some process parameters. It was found that the modified thickener unit was working very well and its performance was equivalent to or better than the unit before the modification when the other process parameters were kept the same.

***3) Development of An Industrial High Efficiency Thickeners and Plant Tests***

The two thickeners developed and manufactured by BGTG were installed in Panjiang Clean Coal Ltd. located in Guizhou Province of China. The throughput of raw coal feed to the plant was 3 Mt/year equipped with heavy medium cyclone and fine coal process circuit. The tested thickening process circuit is sketched in Figure 2, in which the 1st stage thickener is multi-incline-tube thickener while the 2nd stage thickener is the new developed high efficiency thickener to be tested.

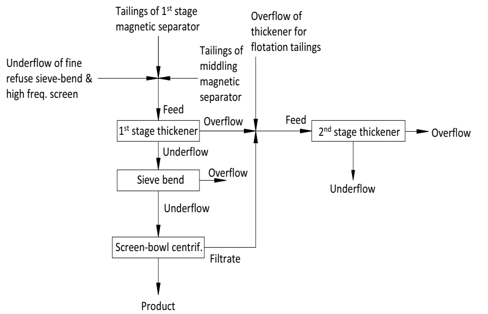



*Figure 2. Circuit diagram of the plant test of developed high efficiency thickener*

Table 2. Performance of the 2nd stage high efficiency thickener

| Item | 2nd Stage Thickener | | Model ST-15 |
|---|---|---|---|
| Feed | Solid content | % | 3.68 |
| | | g/L | 37.42 |
| | Ash content % | | 47.73 |
| | Throughput | Dry solid t/h | 60.24 |
| | | Flowrate $m^3/h$ | 1609.68 |
| | Unit capacity $m^3/m^2/h$ | | 9.11 |
| Overflow | Solid content | % | 0.14 |
| | | g/L | 1.4 |
| | Ash content % | | 61.7 |
| | Dry solid t/h | | 2.05 |
| | Flowrate $m^3/h$ | | 1469.39 |
| Underflow | Solid content | % | 34.73 |
| | | g/L | 414.75 |
| | Ash content % | | 46.79 |
| | Dry solid t/h | | 58.18 |
| | Flowrate $m^3/h$ | | 140.29 |
| Chemicals added | Flocculant | Name | Polyacrylamide |
| | | Dosage kg/t (in dry coal) | 0.13 ( including power plant & belt filter ) |
| Thickening efficiency η% | 89.66 | Clarification coeff. Δ | 0.96 |
| Underflow solid recovery % | | | 96.59 |

The feed capacity of raw coal to the plant was about 600 t/h (originally designed at 570 t/h) during the thickener tests. Please note that there was no flocculant and coagulant added into the 1st stage thickener. The flocculant and coagulant were only added to the 2nd stage thickener. Use of combination coagulant and flocculant in thickening process is extremely important (Jiang, M. and et al., 2009, Zhao, C. and et al., 2012).The 1st stage thickener is multi- incline-tube thickener with diameter size of ϕ7 meter.

The particle size distributions in the overflow and underflow were measured while the size distribution of the feed to the 1st stage thickener was calculated from the test data. Based on the test data, the partition curve of the particles to the overflow and underflow in the thickener can be drawn. The partition size $d_{50}$ or cut-point in the 1st stage thickener can be determined at 0.032 mm, which means the particle size to the overflow in the range of <0.032 mm. The purpose of the 1st stage thickener is to be used as a classifier at the cut-point of 0.032 mm.

The model and specifications of the 2nd stage high efficiency thickener is ST-15 with diameter size of 15 meter. Model 80ZJ-I-A36 pump was used as the underflow pump equipped with 18.5 kW motor. Flocculant and coagulant were added and pre-mixed with the feed slurry before fed to the 2nd stage thickener. Performance of the 2nd stage high efficiency thickener is shown in Table 2.

Table 2 shows the throughput capacity tested at 1609.68 $m^3/h$ for the φ15 m high efficiency thickener. The unit throughput capacity can be calculated at 9.11 $m^3/h/m^2$. The solid recovery rate in the underflow was tested accordingly at 96.59%. The clarification coefficient is calculated at 0.96 while the thickening efficiency is calculated at 89.66%. The clarification coefficient Δ is defined and calculated by following Equation-1 while the thickening efficiency $\eta_t$ is determined by Equation-2. The solid content in the overflow was tested at 1.4 g/L or 0.14% while the solid concentration in the underflow was 414.75 g/L.

$$\Delta = \frac{a-b}{a} \quad \text{(Equation-1)} \qquad \eta_t = \frac{100(a-b)(c-a)}{a(c-b)(100-a)} \times 100 \quad \text{(Equation-2)}$$

*a – feed solid content %, b – overflow solid content %, and c – underflow solid content %*

**Comparison with the Thickeners Currently Used in China Coal Industry**

Before 1980s most popular thickeners used in China coal industry were Former-Soviet-Union style conventional thickeners made in FSU or China. After 1990s large amount research and development was carried out in China on the various kinds of thickeners for coal industry. Of them the most popular thickeners in China were the traction-type thickeners with the rakes driven by trolley running on the thickener circular-edge rail and with auto-controlled rake-arm lift device supported on the center column. This type of thickener is called the side-driven and center-lift thickener in China. Another type of thickeners is hydraulic double central driven thickeners equipped with auto-rake-lift system, which are similar to American Envio-Clear and German Passvant thickeners (Li, X., 2008). There are also some of electrical motor driven thickeners equipped with auto-rake-lift system used in China, which are similar to Eimco's thickeners (Chen, Q., 2007).

Table 3 lists three typical kinds of thickeners used in China coal industry and their performance comparison. As described above, the thickeners predominantly used in China coal preparation plants are the conventional thickeners. The average throughput capacity of the conventional thickeners is about 2.5 $m^3/h/m^2$ while the average throughput capacity of the multi-incline-tube thickeners is about 4.5 – 5.0 $m^3/h/m^2$. The currently developed high efficiency thickeners significantly improved capacity without compromising the other technical and operational characteristics. The capacity of the high efficiency thickener doubled that of the multi-incline-tube thickeners and is almost 4 times of the capacity of the conventional thickeners. In our current plant design, use of the high efficiency thickeners could save about 50% thickening area compared to the multi-incline-tube thickeners and save over 75% thickening area compared to the conventional thickeners. Capital cost will be reduced significantly accordingly. Operation and maintenance costs will be reduced as well due to its simple structure without any moving parts.

Table 3. Performance comparison of three types of thickeners used in China coal industry

| Equipment | Actual uint capacity | Feed solid content | Underflow solid content | Overflow solid content | Thickening efficiency | Clarification coefficient | Underflow (solid, liquid) yield | Underflow solid recovery |
|---|---|---|---|---|---|---|---|---|
| | $m^3/(m^2 \cdot h)$ | % | % | % | % | | % | % |
| ITT1010 Multi-Incline-Tube Thickener | 4.32 | 1.41 | 24.98 | 0.02 | 94.44 | 0.99 | 5.57 | 98.69 |
| Φ30m Conventional Thickener | 1.4 | 1.48 | 32.9 | 0.03 | 95.06 | 0.98 | 4.41 | 98.06 |
| High Efficiency Thickener | 9.11 | 3.68 | 34.73 | 0.14 | 89.66 | 0.96 | 10.22 | 96.59 |

By the end of 2014, 45 units of ST series high efficiency thickeners developed by BGTG have been used in China coal preparation plants. It is deeply believed that more high efficiency thickeners will be widely used in China in the future new designed coal preparation plants and in the retrofitting projects as well.

## CONCLUSSION

The developed high efficiency thickeners filled up the gap of high efficiency thickeners in China coal industry and significantly improved throughput capacity by 3 – 4 times compared to the conventional thickeners. The footprint occupied and operating & capital investment costs can be considerably reduced due to their higher throughput and without any moving parts.

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# Gas heat carrier pyrolysis of low rank coal and associated heat transfer characteristics

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**Abstract**

A small scale gas heat carrier pyrolysis fixed-reactor (6–9 kg $h^{-1}$) was established. The pyrolysis temperature and residence time effect the dehydration process, pyrolysis products, caking property, and temperature profile characteristic. The dehydration process is completed below 300 °C. High temperature and short transfer distance shorten the pyrolysis time. With an increase in the final pyrolysis temperature, the yield of the volatiles and tar increases; however, that of the char decreases. The residence time significantly influences the char yield at 460 °C. The coal begins to agglomerate after 25 min at 460 °C for the first time. Agglomeration of the coal particles increases with an increase in the pyrolysis temperature and residence time. Based on the temperature profile, the temperature curve in the coking chamber can be divided into three stages: the rapid temperature-decline, coal temperature-increase stage, and coal temperature constant stage. The temperature of coal and gas achieves a short relative balance, which causes the temperature turning point between the first two stages. The exothermic heat makes the final temperature of coal higher than that of the inlet gas, thus the exothermic reaction cannot be ignored.

**Key words:** low rank coal; gas heat carrier; nitrogen gas; pyrolysis; heat transfer; temperature profile; agglomeration; exothermic heat;

## 1. Introduction

Low-rank coal (LRC) is high in moisture and ash content and low in calorific value. Moreover, the oxygen content is as high as 15–30%; therefore, it exhibits high tendency to combust spontaneously. In recent years, significant attention has been increasingly devoted to low temperature pyrolysis in order to efficiently utilize LRC. The pyrolysis process has been one of the preferred methods for the complete and efficient use of oil shale in industry. The pyrolysis with gas heat carrier exhibits several advantages including larger heat transfer area, rapid heating rate, and shorter pyrolysis time.

Early scholars have focused on the oil shale retorting and gas heat carrier LRC upgrading system and its characteristics. Such as The oil shale processing industry in Estonia, Brazil, and China have been investigating the retorting technology for many years using the gas heat carrier. For example, the oil shale processing industry in Estonia used Kiviter retorting technique, in Brazil used Petrosix retorting technique, and that in China used Fushun-type retorting technique [1]. Recently, the gas heat carrier retorting technology was applied for the lignite upgrade process, such as Lurgi–Spuelgas, SJ, and LFC [2–4]. The pyrolysis characteristic of coal was investigated using different carrier gases such as nitrogen ($N_2$) and carbon dioxide ($CO_2$) [5–9]. Lu et al. [5] studied the biomass pyrolysis under the $N_2$ atmosphere. This drying process provided the solid fuel with higher energy density. Under the $CO_2$ atmosphere at 480 °C, $CO_2$ formed carbonyl sulfide with sulfur-containing substance and resulted in an increase in the yield of volatiles [6, 7]. Zheng et al. [10] adopted the upgrade process involving high temperature $N_2$ gas. The dehydration process could also be promoted by increasing the initial $N_2$ temperature and decreasing the lignite particle size.

Nonetheless, the gas heat carrier pyrolysis and heat transfer process for LRC have not been



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extensively investigated. Therefore, based on the small pilot-scale gas heat carrier, pyrolysis reactor was developed by our research group, and the investigation adopted $N_2$ as the preferred carrier gas attributed to its easy availability from air separation unit and lower reactive characteristics [11]. The main scope focused on characteristics of pyrolysis products, agglomeration phenomenon, and heat transfer characteristics.

## 2. Experiment

### 2.1. Material

The coal was collected from Shen-Mu Shaanxi province and it was denoted as SM-LRC. The SM-LRC was crushed and ground to obtain the hybrid particle size from 5 to 10 mm. The results of proximate and ultimate analysis of the LRC are listed in Table 1. The values listed in Table 1 indicate that the SM coal has slight caking and its caking index was 14 based on the Chinese nation standard GB5445-87.

Table 1 The proximate analysis and ultimate analysis of SM coal

| Sample | proximate analysis/$w_{ad}$ % | | | | ultimate analysis/$w_{ad}$ % | | | | | Caking Index |
|---|---|---|---|---|---|---|---|---|---|---|
| | M | A | V | FC | C | H | O* | N | $S_t$ | |
| SM coal | 6.13 | 3.88 | 25.72 | 64.27 | 77.75 | 6.22 | 10.66 | 0.97 | 0.29 | 14 |

ad =air dried. *by difference

### 2.2 Experiment apparatus and method

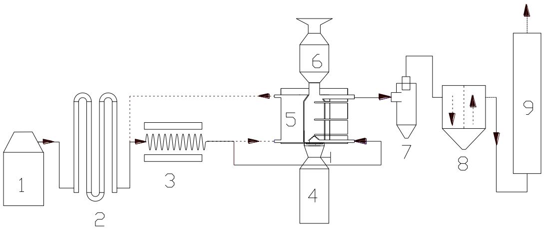


Fig.1 the schematic diagram of experiment equipment (1) Liquid nitrogen; (2) Vaporizer; (3) Gas heater;(4) coke quenching tank; (5) Reactor; (6) Coal hopper; (7) Cyclone; (8) Condenser; (9) Tar collector.

The schematic diagram of experiment equipment is shown in Fig. 1. The experimental set up primarily includes liquid $N_2$, vaporizer, gas heater, reactor, coal hopper, char receiver, and tar collector.

Fig. 1-(1) shows that the liquid $N_2$ is stored in liquid $N_2$ tank type DPL452-175-1.38I. Through a vaporizer (Fig. 1-(2)), the pure $N_2$ is evaporated with the maximum capacity 80 $Nm^3\ h^{-1}$. The $N_2$ gas is heated by the 30 KW gas heater (Fig. 1-(3)) to the required setting temperature. The small scale pyrolysis fixed-bed reactor as shown in Fig. 1-(5) is made of 316 stainless steel and includes two hollow cylindrical cavity structures. The inside cavity is the coking chamber, which is surrounded by a high temperature $N_2$ insulation cavity. The outer part of the reactor is covered by the cotton insulation. The external diameters of $N_2$ insulation cavity and cotton insulation are 180 and 400 mm, respectively. The hollow porous gas distributor, with four rows of eight gas flow holes, is located at the bottom of the reactor to support coal bed and it provides a uniform distribution of flow gas. The leak-tightness tests were performed on the reactor, which ensured no gas leakage. The required weighed SM coal was allowed to enter into the reactor when the desired temperature of coking chamber was reached and stabilized (400–640 °C). The char is received by the coke quenching tank as shown in Fig. 1-(4). The moisture is collected by condenser (Fig. 1-(8)) on time. The tar is obtained in the tar collector (Fig. 1-(9)). Along with the

experiment, the temperature, the pressure, and the gas flow of every system was recorded and saved in the computer software. The final pyrolysis temperature was set as 400, 460, 520, 580, and 640 °C and the residence time of 5, 10, 15, 20, and 25 min, respectively, were selected as the experimental parameters.

## 3 Results and discussion

### 3.1. Effect of temperature and residence time on pyrolysis characteristics

Fig. 2 shows the yield of the moisture, tar, and char based on the changes in the pyrolysis temperature and residence time. Fig. 2 shows that higher temperature leads to higher dehydration rate; however, the total moisture yield reaches its plateau value of 6.37 wt.% as residence time and temperature increase. With the change in the pyrolysis temperature from 440 to 640 °C, the total tar yield increases and the char yield decreases.

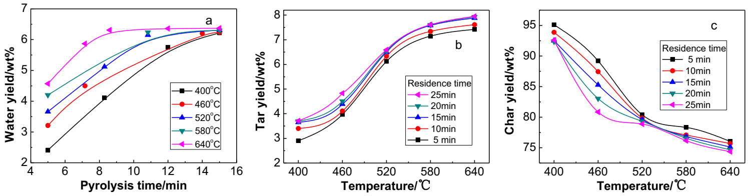


Fig. 2 The yield of the pyrolytic products (a) moisture; (b) tar; (c) char

Fig. 2-a clearly exhibits the dehydration of the coal below 300 °C during pyrolysis process. Based on the direct contact heat transfer process, the heat carried by carrier gas was absorbed rapidly by the coal. Thus, higher coal temperature leads to higher dehydration rate. The total moisture yield reaches its plateau value much earlier at higher temperature. With an increase in the pyrolysis temperature, more volatiles were released [12].

Fig. 2-b shows that the increase in volatiles yield at higher temperature is consistent with the tar yield and it is contrary to the char yield (Fig. 2-c). The yield of tar is 3.71, 4.52, 6.59, 7.61, and 7.95 wt.% and the char yield is 92.60, 80.84, 78.88, 76.16, and 74.36 wt.% at 400, 460, 520, 580, and 640 °C, respectively. The increase in volatiles yield leads to a decrease in the char yield. Therefore, an increase in temperature results in an increase in the tar and volatiles yield and a decrease in char mass. The tar yield increased and semi-coke yield decreased with an increase in the pyrolysis residence time. The minimum yield of char was obtained at 25 min; however, significant change was observed at 460 °C. There was enough residence time for the side chains and small molecules of coal to leave as volatiles in pyrolysis process, produced largely above 460 °C [13, 14]. The tar intermediates were also influenced by the residence time. The gas-coal temperature lagging behind coal particle temperature itself during long residence time and the secondary pyrolysis production was released. Therefore, the yield of char was sensitive to different residence time at 460 °C.

Fig. 3 shows that the coal begins to agglomerate at 460 °C and 25 min and the bonded particle mass increases with an increase in the temperature and residence time. The char agglomeration is performed without a binder substance to help, and the bonded coal size (>10 mm) is larger than the initial coal size 5–10 mm. It is generally argued that the bonding phenomenon, produced by bonded substances, leads to agglomeration of coal [15, 16]. The bonded substances include the smaller fragments called “metaplast” and their repolymerization products, and the tar intermediate includes the bonding bituminous and pitch.

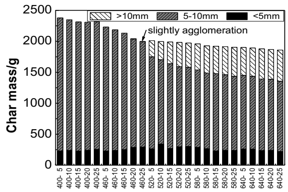


Fig.3 the coal's size distribution and mass change with pyrolysis temperature and residence time

The bonding component produced at 460 °C provided a weak caking property, which resulted in agglomeration of the coal particles. Therefore, small particles formed larger particles as pyrolysis temperature increased. Compared to the coal fixed-bed heat transfer process, the gas heat carrier process offered rapid heat transfer efficiency, and it led to rapid generation of the thermoplastic stage [15, 16]. Thus, with an increase in the temperature and residence time, the bonded particle mass increased.

**3.2 Heat transfer process**

Fig. 4 shows the temperature profile of the coking chamber from 400 to 640 °C. An increase in carrier gas temperature leads to higher final pyrolysis temperature. The coking chamber could be divided into three zones: bottom, middle, and top zones. The temperature in coking chamber curve could be divided into three stages based on the temperature distribution characteristics: the carrier gas temperature rapid decline stage, the coal temperature increase stage, and coal temperature constant stage.

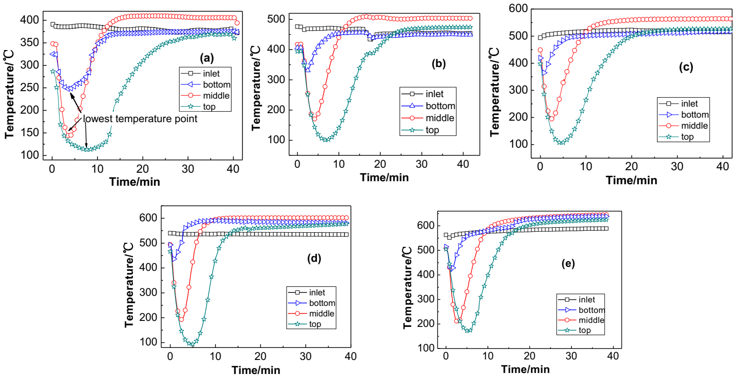


Fig.4 the temperature distribution at the pyrolysis temperature vs. residence time (a) 400°C; (b) 460°C; (c) 520°C; (d) 580°C; (e) 640°C

Fig. 4 shows that the gas heat carriers with different temperature influence the final pyrolysis temperature. They move upward from bottom to top zone in coking chamber and heat coal. The initial contact between hot gas and cold coal results in the rapid temperature-decline stage, and then the coal endothermic process leads to the coal temperature increase stage. The pyrolysis process, occurring in first two stages for LRC, was mainly the endothermic reaction, and the heat source for coal was only the carrier gas. The higher gas temperature yields higher. pyrolysis temperature, and the heat transfer distance increased from the bottom of the coking chamber to its top. The temperature of carrier gas decreased with an increase in the heat transfer distance. Therefore, the pyrolysis temperature decreased with the increase in heat transfer distance.

The temperatures of coal and carrier gas achieved a short relative balance and formed a turning temperature point (TTP) between the end of the carrier gas temperature rapid decline stage and the beginning of the coal temperature-increase stage. The temperature of carrier gas decreased at first, and it began to increase after TTP. Finally, the temperature of the carrier gas exhibited the increasing trend similar to that of the coal at the coal temperature-increase stage. In the pyrolysis process, the TTP at some height moved from the bottom to top zone in coking chamber. Finally, the temperatures of coal and carrier gas achieved a long relative heat balance for the second time at coal temperature constant stage, and its temperature stayed constant during residence time. The final pyrolysis temperature of the coal in the middle zone of coking chamber was higher than that of the inlet gas. It was not possible to heat high temperature char only by heat resource of carrier gas. Thus, the main endothermic pyrolysis process changed to exothermic process.

According to the literature, the exothermic heat is observed in the lignite and sub-bituminous coal in pyrolysis process [17], and the occurrence of exothermic reactions were associated with the production of volatile matter in or near the plastic region. The exothermic heat was released due to the devolatilization process [18] which shifted the heat to the coal. The exothermic effects were ignored because of the part counteracted the endothermic process [19] in the common 18 h fixed-bed pyrolysis process. However, the exothermic effects would not be ignored in the gas heat carrier pyrolysis process in short time.

## 4. Conclusion

Higher temperature of the carrier gas resulted in higher final pyrolysis temperature. High final temperature led to rapid dehydration rate for SM coal below 300 °C. The total yield of moisture was about 6.37 wt.%. The tar yield increased and char yield decreased as the pyrolysis temperature increased from 400 to 640 °C. The pyrolysis time was shortened attributed to higher temperature and shorter heat transfer distance. The yield of char changed significantly at 460 °C.The agglomeration phenomenon occurred after 25 min at 460 °C at first. With an increase in pyrolysis temperature and residence time, the mass of bonded particles increased.

According to the temperature distribution, the temperature curve of coking chamber could be divided into three stages. A temperature turning point was observed between the carrier gas rapid temperature-decline stage and the coal temperature-increase stage. The pyrolysis heat was released during devolatilization process, which resulted in higher center temperature of coking chamber compared to that of the inlet carrier gas.

## Acknowledge

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# Drying of coal fines assisted by ceramic sorbents

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**Abstract**

It is proposed to introduce contact sorption drying as a method to reduce the moisture content in coal fines. The aim of this study was to investigate the possibility of drying fine and ultra-fine coal using porous ceramic as the moisture sorbent. The main focus of this report is to define how the air temperature, particle size variations and mixing ratio would influence the contact between coal and ceramic for effective moisture adsorption. Drying of coal fines assisted by ceramic sorbents proved to be a viable concept as the ceramic was able to not only reduce the moisture content of fine coal, but of ultra-fine coal as well. The larger surface area of smaller ceramics allowed for efficient contact and consequently higher dewatering rates. The addition of more ceramic resulted in better contact with the wet coal and reduced the operating time quite significantly. Improved contact between the coal and ceramic therefore proved to be the main driving force during adsorption drying.

Keywords: Alumina; Ceramic; Coal; Drying; Fines; Moisture; Sorbent; Sorption

## Introduction

Mangena *et al*. (2003) estimated that about 12% of South Africa's mined coal can be classified as fine and ultra-fine coal. Due to its large surface area, the coal fine fraction retains the majority of the moisture in mined coal and can contain a moisture content of higher than 25% by weight (Le Roux, 2003). Compared to coarse coal, the fine and especially the ultra-fine fraction are far more difficult to dewater to an acceptable moisture content. As a result of the difficulty in dewatering, the fines are often dumped into discard dams or slurry ponds. Consequently these disposal methods leads to a series of environmental problems such as acid mine drainage, dust release, spontaneous combustion and it also occupies large areas of land. Aside from the environmental problems, the wasted coal fines have heating values comparable to the coarse coal fraction when dried (Reddick *et al*., 2007). It would then be sensible to investigate and implement cost effective and efficient drying methods to salvage the wasted fines and process the mined fine fraction. It is important to use coal wisely as it is the primary energy resource in South Africa (Fourie *et al*., 1980).

In fine coal processing the mechanical dewatering techniques are ineffective as these methods cannot reduce the moisture content to a desirable level. Dewatering of coal fines often relies strongly on thermal methods as coal fines are known to retain water (Le Roux & Campbell, 2003). There are several conventional thermal drying methods, however not all of these systems are ideal in terms of energy consumption and safety during operation (Kudra & Mujumdar, 2009).

It is proposed to introduce contact sorption drying as a method to reduce the moisture content in coal fines by creating a mass concentration gradient between the wet coal and dry sorbent. The aim of this study was to investigate the possibility to dry fine and ultra-fine coal using porous ceramic as moisture sorbent. The investigation focusses on defining how the temperature, particle size variations and mixing ratio will influence the contact between the coal and ceramic and consequently the moisture transfer. The efficiency of the contact drying technique would lay in the possibility of separating the coal and ceramic and furthermore regenerate the ceramic for re-use.



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**Background**

Ceramic is an inorganic solid that is relatively cheap as many manufactures produce it from waste material (Lin *et al*., 2012). Ceramic is used as an adsorbent due to its permeability and extensive pore size distribution. It is specifically favoured due to its retention ability and the permeate flux it causes (Li *et al*., 2006). According to Lin *et al*. (2012) ceramic is unique as it is porous, yet has great mechanical and thermal stability making it suitable for industrial purposes.

Contact sorption drying takes place in three main stages as indicated in Figure 1. During the first stage the wet coal comes in contact with the dry ceramic and a small amount of moisture transfer takes place. This phase relies mainly on effective contact between the material and sorbent. The moisture is adsorbed unto the surface of the sorbent through diffusion and then penetrates through the pores of the sorbent. This set in motion the macro scale diffusion due to the large moisture concentration gradient between the wet coal and dry ceramic. The majority of the contact sorption drying takes place during the second stage and reaches a clear equilibrium where no major moisture transfer takes place. At the completion of the drying process, minor penetration and diffusion will take place between the sorbent and material leading to a very slow drying rate. Moisture transfer between sorbent-sorbent particles and material-material particles will result in minor fluctuations in the final equilibrium moisture content (Kudra & Mujumdar, 2009).

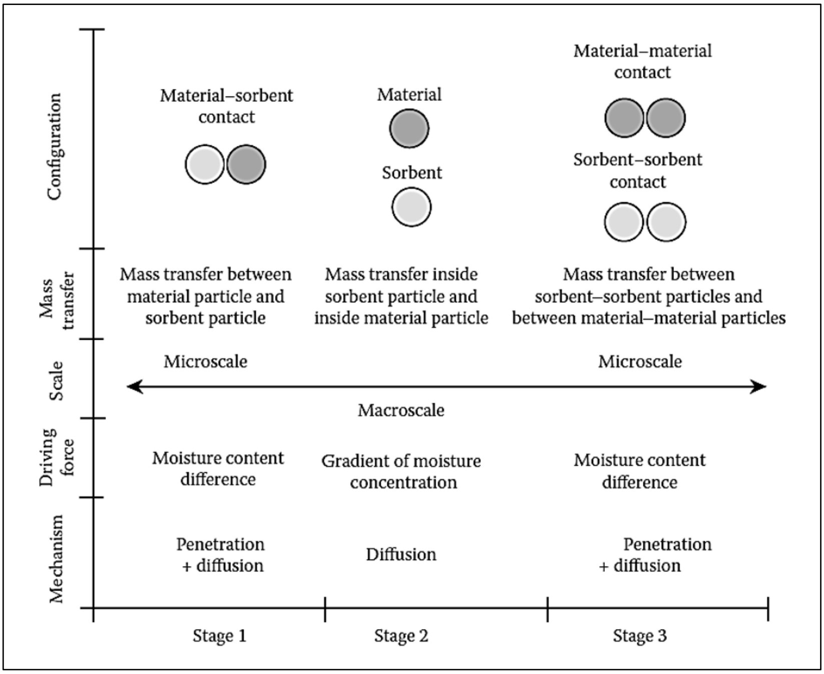


Figure 1. Drying mechanism for contact sorption drying (Kudra & Mujumdar, 2009)

## Experimental

This report aims to define how the temperature, size variations and ratio will influence the contact between coal and ceramic for effective adsorption. The efficiency of contact sorption drying was tested on fine coal as well as ultra-fine coal.

1. Equipment

The experimental work was based on laboratory scale units to test the proof of concept and Figure 2 gives a layout for this contact sorption drying process. Firstly the drying process took place within the rotating adsorber, after which the loaded ceramic and dry coal were separated by screening. Finally the saturated ceramic was dried within a packed bed to be re-used in the contact sorption drying. The adsorber was designed to rotate the enclosed cylinder with a diameter of 8cm at about three revolutions per minute to ensure sufficient contact between the coal and ceramic. The cylinder was specifically placed horizontally to prevent the coal and ceramic from separating due the large difference their particle size and density. Laboratory sieves were used to separate the saturated ceramic and dried coal before the ceramic was placed in a packed bed to be dried. Wet ceramic was easily separated from the dry coal and the packed bed dryer regenerated the ceramic to a point where it could be re-used. Note that only results from the rotary adsorber are discussed in this report.

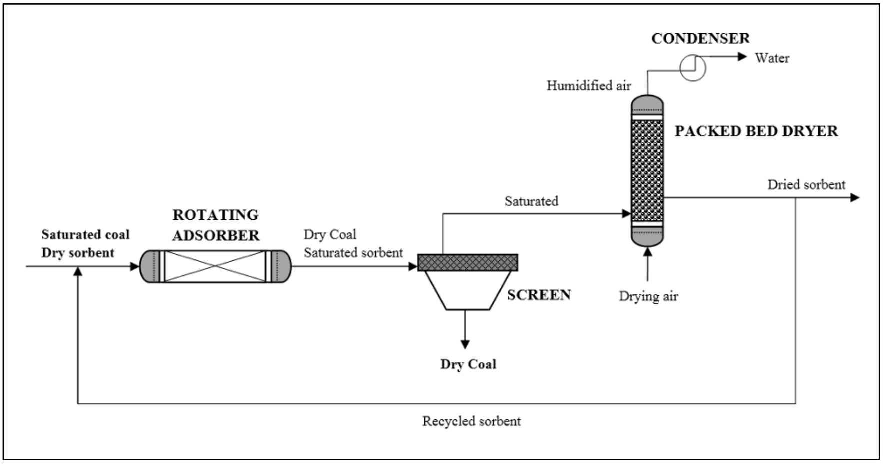


Figure 2. Diagram of contact sorption drying and regeneration process

2. Material

Coal from the Highveld coal field in South Africa was used in this study. The samples were initially left to dry in ambient air and thereafter crushed to a fine coal fraction of +1mm-2mm and an ultra-fine coal fraction of smaller than 0.5mm. A sample splitter was used to produce uniform samples of 100g each to obtain more repeatable experimental data. Afterwards the samples were drenched in water and filtered to a total moisture content of about 14-16%$_{wt}$ for the fine coal fraction and 18-25%$_{wt}$ for the ultra-fine coal fraction. Porous ceramic containing about 82-85%$_{wt}$ of Alumina oxide was used as an adsorbent to transfer water away from the coal. The test work was completed using ceramic spheres of between 6-10.5mm and 12-20mm.

**Results and discussion**

Fine coal was dried within the rotary adsorber and the moisture content of both the ceramic and coal was determined at various intervals. The experiment was conducted at 25°C and the material used included fine coal and ceramic with a particle size distribution of +1-2mm and +6-10.5mm respectively. Figure 3 shows that the moisture loss from the coal correlates with the moisture uptake in the ceramic spheres. The majority of the moisture transfer took place within the first 5 minutes indicating that the moisture within the bulk coal sample could rapidly be reduced from about 19$\%_{wt}$ to 9$\%_{wt}$. After the major moisture transfer, the drying rate slowed down significantly as the coal reached a level close to its final inherent moisture content. It was observed that towards the completion of the experiment that the coal adsorbed a small amount water again, but it was adsorbed by the ceramic again soon after. This phenomena occurred continually after the coal and ceramic reached equilibrium. Kudra & Mujumdar (2009) states that these moisture fluctuations are referred to as stage 3 during the drying process. During this stage variations in the moisture content are due to the moisture transfer between sorbent-sorbent, material-material as well as sorbent-material particles. Initially the contact sorption drying took place in an enclosed container at static conditions, but it was found that there was a lot of vigorous fluctuation in the data. A rotary adsorber was introduced to ensure maximum contact between the coal and ceramic and Figure 4 shows the improvement in the data thereafter.

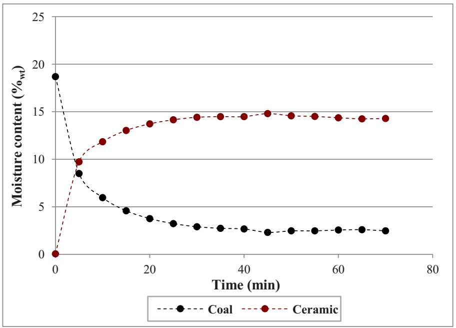

Figure 3. Moisture transfer during contact drying

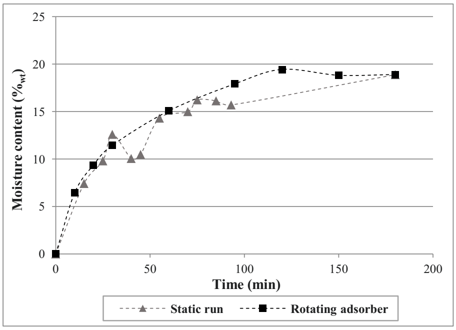

Figure 4. Static versus rotating adsorber

Ceramic spheres between size ranges of +6-10.5mm and +12-20mm were used to determine the difference in adsorption rate of moisture from ultra-fine coal samples at 25°C. The results obtained from these two experimental runs are given in Figure 5. It was found that the smaller ceramic spheres had the ability to adsorb more moisture from the coal at a much faster rate, when compared to the larger ceramic particles. Asmatulu & Yoon (2012) stated that finer particles attract and retain water mainly because they have a larger surface area compared to large particles. The larger surface area creates surface and capillary forces aiding in the adsorption process. It was therefore found that increasing the surface area by working with smaller ceramic, improved the contact during adsorption. However, it is important to keep the size ratio between the ceramic spheres and the fine coal particles large enough to aid in the separation thereof.

Figure 6 shows the difference in the sorption rate when fine and ultra-fine coal were dried. Ceramic with a size range of +6-10.5mm was used for adsorption at a temperature of 25°C. The ultra-fine coal fraction had a higher moisture content and took much longer to dry compared to the fine coal particles.

The fine coal showed a total of 8%$_{wt}$ moisture reduction in 14 minutes, while the ultra-fine particles took 87 minutes to reduce about 8%$_{wt}$. The larger surface area of the ultra-fine coal creates more surface and capillary forces that prevent the moisture from leaving the finer coal particles (Asmatulu & Yoon, 2012). Even though the ultra-fine particles took very long to dry, it must be noted that contact sorption drying was able to reduce the moisture content of the ultra-fine coal to its inherent moisture content. Moreover these results were achieved with just a rotary adsorber at atmospheric conditions without the need of expensive operating conditions. Bratton *et al*. (2012) also found that adsorbents were able to transfer water away from wet particles, irrespective of particle size.

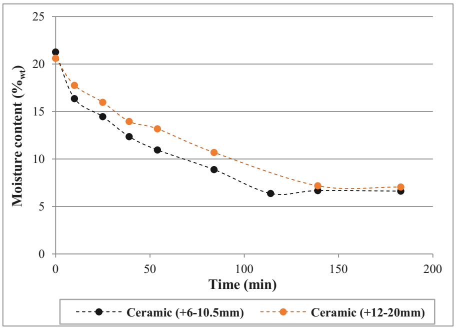


Figure 5. Variation in ceramic size

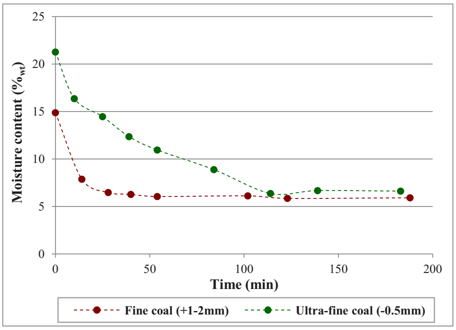


Figure 6. Drying rate of fine and ultra-fine coal

Fine coal samples of 100g each were placed within the rotary adsorber at temperatures of 25°C, 40°C and 55°C. Ceramic in a ratio of 1:1 were added to the one experiment, while the other experiment operated with more ceramic in a 1:3 ratio. The coal to ceramic ratio of 1:3 resulted in a faster dewatering rate compared to the feed ratio of 1:1. Increasing the amount of ceramic lead to more efficient contact between the coal and ceramic as more surface area of the ceramic was available for moisture transfer. Introducing elevated temperatures to the system didn't show a significant improvement in the drying rate. Figure 7 however illustrates that temperature had an effect on the final moisture content and a temperature of 55°C could reduce the moisture content by an additional 1%$_{wt}$.

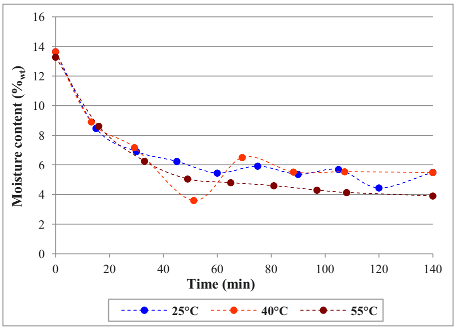


Figure 7. Ceramic to coal ratio of 1:1

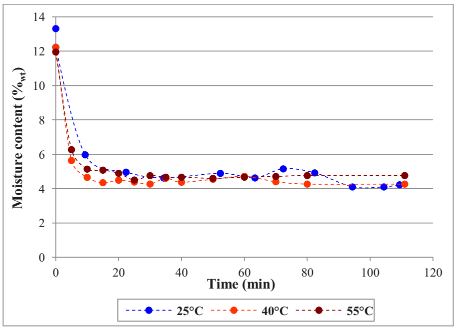


Figure 8. Ceramic to coal ratio of 1:3

It can be seen in Figure 7 that the temperature had a larger effect on the coal moisture content towards the completion of the experiment. As a 1:1 ratio was already packed and had already reached optimum contact, the temperature could improve the transfer rate. This however had no noticeable effect in the first few minutes of drying. Figure 8 shows that changes in temperature didn't have an effect when optimal contact ratio between coal and ceramic were in place.

## Conclusion

Ceramic was able to not only reduce the moisture content of fine coal, but of ultra-fine coal as well. The ceramic could reduce the moisture content irrespective of the size of coal dried. Working within a rotary adsorber ensured maximum contact between the coal and ceramic leading to more consistent results. Smaller ceramic had a larger surface area that allowed more efficient contact and consequently faster dewatering rates. The addition of more ceramic resulted in better contact with the wet coal and reduced the operating time quite significantly. In conclusion the results showed that improved contact between the coal and ceramic is the main driving force during adsorption drying.

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# Coal product moisture control using stockpiles

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**Abstract**

The moisture content of product coal is a major factor influencing the efficiency of downstream coal utilization processes. Product stockpiles are often used as a control measure to regulate the moisture content of the coal by gravity drainage and evaporation. An understanding of the mechanisms of water migration and retention in coal stockpiles are required to optimise the management of these stockpiles. Apart from the process water carried over into the product after beneficiation, additional water due to rainfall can add to the total moisture contained in a stockpile. When the rain falls on the stockpile, it either runs off the surface or infiltrates the stockpile. The infiltrated water can evaporate from the surface (down to a certain depth), drain through a saturated toe, or remain within the stockpile to add to the total final moisture content. To study these mechanisms, laboratory scale experiments were designed. A drainage column was used to simulate the percolation of water in a stockpile, and the data verified that particle size, especially the -0.5 mm fraction, had the most significant influence on both the drainage rate and the water retained in the bed. The ratio between run-off water and infiltration water during rainstorms were also quantified, and it was shown that compaction of the bed had a major influence on infiltration. Evaporation from a coal bed surface was tested by measuring the mass loss from coal beds exposed to the atmosphere, while measuring weather conditions like temperature, relative humidity and wind speed. The average evaporation loss was about 0.8 L per $m^2$ per day.

**Key words**: coal, stockpile, moisture, moisture control, rainfall, evaporation, drainage, percolation

## Introduction

Stockpiles are often used for product quality control, for example to reduce the total moisture content of a product. Because of more stringent quality requirements, an improvement in stockpile management is required (Keleher et al., 1998). Excess moisture in coal results in increased transportation costs, handling problems and a decreased calorific value (Rong, 1997; Leader et al., 1997). The need for moisture control has an influence on the method of the construction, maintenance and reclamation of stockpiles (Boyapati & Oates, 1994).

The free moisture content of a stockpile is increased by water carried over from the wet processing of the coal and by rainfall. During these events, water can either run off the surface of the stockpile, or it may ingress into the pile. On the other hand, the moisture content in a stockpile is reduced by natural gravity drainage or by evaporation from the surface (Brookman *et al*., 1981; Eckersley, 1999). To understand these processes, the phenomena of rainwater run-off versus ingress, gravity drainage and evaporation must be separately considered.

 
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**Table 1: Proximate and CV analysis**

| | Value | Standard |
|---|---|---|
| % Inherent moisture content (air-dried) | 2.5 | ISO 11722: 1999 |
| % Ash content (air-dried) | 35.6 | ISO 1171: 2010 |
| % Volatile matter (air-dried) | 19.0 | ISO 562: 2010 |
| % Fixed carbon (air-dried) | 42.9 | |
| Calorific value (MJ/kg) (air-dried) | 19.24 | ISO 1928: 2009 |

The relation between rainwater run-off and ingress depends on the paticle size, the rainfall intensity and duration, the stockpile slope and the degree of compaction of the stockpile surface (Curran *et al*., 2002; Wels *et al*., 2015). The proportion of rainfall that infiltrates a stockpile also depends on the infiltration capacity of the coal sample (Brookman *et al*., 1981). As the stockpile becomes more saturated, a steady final infiltration rate is reached, and any higher rainfall intensity will result in an increased amount of run-off water (Huang *et al*., 2012).

The process of gravity drainage is influenced by factors such as particle size (De Korte & Mangena, 2004), stockpile height (Curran *et al*., 2002), degree of compaction, and coal type (Eckersley, 1999). The bottom layer in the stockpile becomes saturated, allowing excess water to flow in the lower permeability outer "shell" to eventually seep through the stockpile toe. This seepage starts between 1 and 3 days following the construction of the stockpile, and it may continue for a number of weeks (Eckersley, 1999). It was found that coal with less than 10% fines (generally defined as smaller than 0.5 mm) is usually free draining. With fine coal, the voids between the particles are sufficiently small that they can act as capillary cavities that become filled with water. A study by the Fuel Research Institute of South Africa in 1964 found that the removal of the -0.5 mm particles had the most significant impact on the static drainage characteristics of a column of coal.

A number of studies have been done on evaporation from flat water surfaces (Finch & Calver, 2008). Many factors influence the rate of evaporation, especially those affecting the vapour pressure in the air, since evaporation is due to the difference in the vapour pressure at the surface of the evaporative medium and in the bulk atmosphere (CSEM-UAE, 2010; Headrick, 1967). The most important influences are wind speed, humidity, temperature and solar radiation (The University of Arizona, 2014; Headrick, 1967). When considering evaporation rates from moist coal particles rather than water surfaces, major differences can be expected, since attractive forces between the coal surface and the water tend to reduce evaporation. The fact that the coal particles are dark, and have uneven surfaces, can also affect the evaporation rate (TRC Environmental Consultants, 1983). It was also reported that evaporation rate from a bed of smaller particles is higher than from larger particles (CRA Limited, 2010).

## Experimental

Coal from the #4 seam of the Witbank coalfield in South Africa was used for all experiments. The proximate analysis is given in Table 1. To investigate the effect that particle size may have on the different mechanisms, three size classes of coal were considered: a coarser fraction where the 6.7 mm was screened out, a finer fraction consisting of only 6.7 mm material, and the un-sized sample covering the entire range of sizes. The top size was 50 mm.

For the drainage experiments, a 2 m long x 0.385 m diameter column was constructed (figure 1a). The column was fitted with a mesh at the bottom supporting the packed coal bed inside. A distributer enabling an even distribution of feed water was fitted to the top of the column. The drained water was collected at the bottom and continuously weighed by a scale connected to a data logger. The column was equipped with four sampling ports on the side of the column, where local samples could be taken for size and

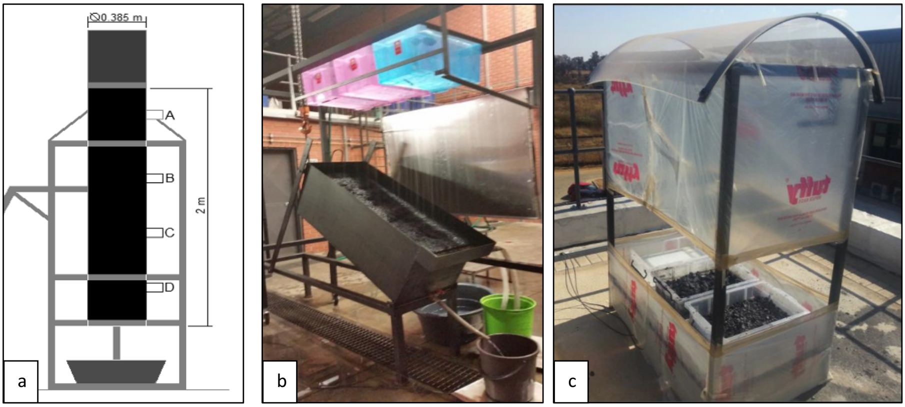


Figure 1: Equipment to determine the behavior of moisture in a coal stockpile: a) drainage column to determine drainage rates, b) inclined box to determine run-off versus infiltration, and c) open vessel evaporation determination.

moisture analysis. A measured amount of water was added in one step and allowed to drain through the column for a number of days, while the drained water mass was continually measured. Samples were taken through the ports at predetermined intervals, usually 2 to 3 days.

The run-off versus infiltration experimental setup consisted of a 1.24 m x 0.50 m x 0.55 m rectangular steel box filled to the brim with coal (figure 1b). The entire box could be tilted to simulate different stockpile angles. Rainfall could be simulated by adding water at controlled flow rates from overhead water reservoirs. This water either flowed across the inclined surface or infiltrated the bed. The run-off water overflowed via an overflow gutter, while the permeated water drained through a mesh-covered outlet at the bottom lower edge of the box. Both the run-off and permeated water was collected in containers that were continuously weighed. Coal loads with different particle size distributions, as well as at different degrees of compaction, were investigated.

Evaporation from a coal bed surface was measured by recording the mass loss from open containers exposed to the atmosphere (figure 1c), while simultaneously measuring weather conditions such a temperature, relative humidity and wind speed. Coal samples with known mass and particle size distributions were loaded into the containers to a depth of about 250 mm and saturated with water. The excess water was drained off, and a sample was taken to define the starting point moisture content. One container was filled with water only to act as a control. The experiment was allowed to run for two weeks while the mass and weather data was recorded, and the evaporation rates were determined from the mass loss information. After completion of the run, a final sample was taken for total and inherent moisture analysis. The actual moisture content over time was back calculated.

**Results and discussion**

An extensive set of results was generated during this project, of which only selected highlights are presented here.

From the drainage column test work, the drainage rate through beds of different particle sizes factor was firstly considered. Figure 2 illustrates how much of the added water was retained in the bed over time, and how the drainage rate through the coal bed was influenced by the presence of 6.7 mm material: the

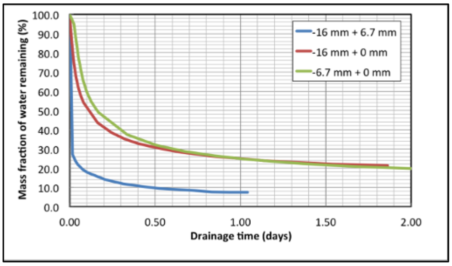


Figure 2: Drainage curves for differently sized coal samples obtained in a column over 2 days.

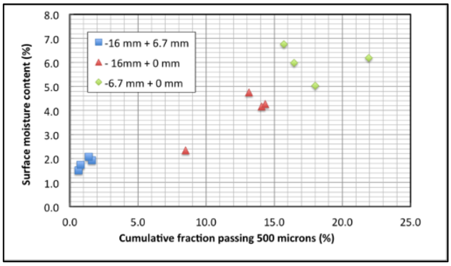


Figure 3: The relationship between the 500 µm content and surface moisture content at various positions in the column.

drainage rate was much faster for the cases where the 6.7 mm material was removed. There was no significant difference between beds containing this finer fraction, or whether it contained a full particle size distribution that included the fines. Thus any particle size above 6.7 mm did not influence the drainage rates significantly. Secondly, evidence of the importance of the ultrafine (-500 µm) material on the moisture retention is shown in figure 3, where the moisture content and ultra-fines sampled at the various sample ports are shown. This confirmed the finding in literature (Eckersley, 1999). This relationship is independent of the distribution of coarser particle sizes.

Coal with a size distribution of -19 mm + 0 mm was loaded into the run-off test box and was subjected to low, medium and high rainfall intensities. The ratio between run-off water and infiltration water was determined for a number of angles. Figure 4 confirms the concept of the infiltration capacity described by Brookman *et al*. (1981), where at a low slope angle any water addition higher than about 0.3 – 0.35 kg/s simply ran over the inclined surface. This value decreased for higher slope angles, while surprisingly the water infiltration deceased slightly with higher rainfall. This may be due to a higher kinetic energy of the falling drops that caused the water flow to deflect down the surface. The marked effect of bed compaction is illustrated by figure 5, where the infiltration capacity could be reduced from 0.38 kg/s to 0.12 kg/s by increased compaction, and even lower at high slope angles.

Figure 6 shows a typical mass loss trace for a period of 9 days for a -13 mm coal sample exposed to the atmosphere, compared to that of an open water surface. The first feature is the cyclical response of the coal mass illustrating the capacity of the coal to lose and re-adsorb moisture from the atmosphere. The

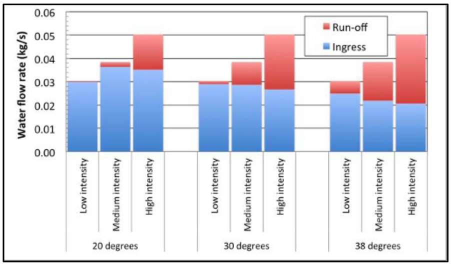


Figure 4: The run-off and infiltration rates for different rain flow rates at varying slope angles.

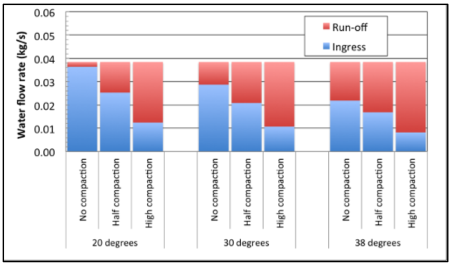


Figure 5: The run-off and infiltration rates for different degrees of compaction at varying slope angles.

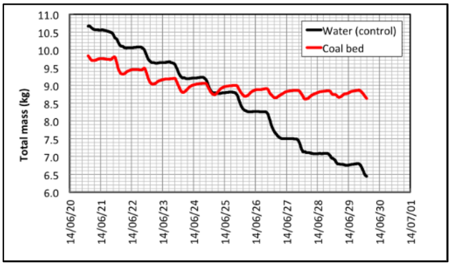


Figure 6: Mass loss of water due to evaporation for a coal bed and a control water surface.

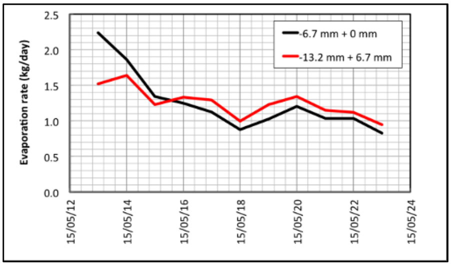


Figure 7: Daily evaporation rates for coal with and without the 6.7 mm fraction.

mass increased due to moisture adsorption in the early morning when the relative humidity of the atmosphere is the highest, and decreased during the late afternoon when evaporation was higher due to a lower relative humidity and higher temperature. With open water evaporation, the evaporation rate also varied with the time of day, with virtually no evaporation in the early morning. Under favourable conditions, the evaporation rate can initially be as high as 0.8 L per m$^2$ per hour, settling to equal absorption and desorption periods after 4 or 5 days. The effect of particle size on evaporation is shown in figure 7, where the coal that contained some 6.7 mm material showed a higher moisture loss for the first 1 – 2 days only, after which the evaporation rate became similar to that of the sample where the 6.7 mm material was screened out. It is assumed the higher initial moisture content in the finer fraction caused this.

**Conclusions**

This paper investigated the mechanisms involved in the water retention, drainage and evaporation in coal stockpiles. During the study the drainage through a stockpile could be simulated using a 3 m long column. The effect of the particle size distribution could be quantified. The effect of slope angle, rain flow rate, and degree of compaction on the water infiltration into stockpiles was determined. The evaporation rates of water from stockpile surfaces could be quantified. All of this information can be used to determine what the water drainage and retention properties of a large coal stockpile will be – this will be investigated in a follow-up study

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# ENHANCEMENT OF HIGH-FREQUENCY SCREENS PERFORMANCE AT COAL SLURRY DEWATERING

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**Abstract**

Process of coal slurry dewatering on the vibrating screen surface involves three stages: at the first stage, pulp is concentrated to high concentration due to the removal of free water by the action of hydrostatical pressure applied to its layer; at the second stage, high-concentration pulp is imparted with the viscoplastic properties due to removal of external superficial water by the action of inertia component of vibrations; at the third stage, viscoplastic material is converted into viscoelastoplastic one due to the partial removal of capillary water from the pore space of granular material by vibratory compaction.

Maximum dewatered material compaction rate is achieved at screen deck vibrations amplitude and frequency of 1.5-2.0mm and 1500-2000min$^{-1}$, respectively, depending on the layer thickness, and compaction time will be at least 35s.

A dewatering surface of high-frequency slurry screens must consist of three sections with different inclination with lead and tail sections having negative (up to -20 deg.) and positive (up to +15 deg.) inclination, respectively, in relation to the middle section, thus dewatering surface configuration can be described as asymmetrical S-shaped curve.

Application of modified high-frequency screen ГВЧ-41М at the Gukovskaya CPP for the spiral separator concentrate dewatering in place of the high-frequency screen ГВЧ-41 resulted in spiral concentrate moisture content reduction by 6% from 28.0% to 22.0% resulting in total plant concentrate moisture content reduction by 0.2% from 7.1% to 6.9% and confirmed correctness of theoretical assumptions.

**Keywords**

Dewatering, physical model, coal slurry, screen, enhancement, vibratory compaction, efficiency.

Slurry products are primary source of increased moisture content in commercial coal products of coal preparation plants. Elimination of the thermal drying from the dewatering processes at coal preparation plants resulted in exceeding of moisture content targets in the shipped concentrate.

Hence, development of the improved dewatering technologies for slurry products and primarily non-floating sizes is a vital task.

For the purposes of discussion, process of coal pulp dewatering on high-frequency vibration screen can be divided into three stages [1].

At the first stage, coal pulp is concentrated to high concentration due to the removal of free water.

At the second stage, high-concentration pulp is imparted with the viscoplastic properties due to removal of free superficial water.

At the third stage, the pulp dewatering is carried out in two stages: repacking and convergence of solid phase particles.



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At the repacking stage, vibration causes destruction and rearrangement of unsteady random structure of particles approaching the most energetically favourable position due to gravity [2, 3].

At the convergence stage, no any substantial particle structure rearrangement occurs. The mixture is compacted due to convergence of particles. Second stage duration is considerably longer than that of the first stage. At this stage, liquid phase is "squeezed out" from the mixture pores due to the relative displacement of particles [3, 4].

Based on the above-described principles of dewatering process, simplified dynamic model of pulp layer on the working surface of vibration screen can be represented as the viscoplastic rheologic body with the apparent mass equal to mass of unit section bar, of which height equals to the height of watered material layer and density equals to the pulp density [5]. Due to vibration action, material yielding point $\sigma o \to 0$, and the plastic member stress is proportional to strain: $\sigma_n = k_n \varepsilon_n$, where $k_n$ is the plasticity coefficient, and $\varepsilon_n$ is the plastic member strain. In addition, we assume the dewatered material layer height equal to $h_m$, so the plastic member stress at the layer compaction stage will be proportional to the $(h_1 - h) / (h_1 - h_m)$ value, i.e., the plastic resistance varies between zero and $k_n$.

We assume that the dewatered material layer height, $h = h$ (t), is a gradually varied parameter, which remains constant during one oscillation period of working surface. Then equation of dewatered material layer motion on the screen deck caused by pulsating load, $a_a \sin\omega t$, will be:

$$\rho h(t)\ddot{y} + K\dot{y} + \frac{h(t)}{h(t)-h_m}k_{\text{п}}y = k_{\text{п}} \cdot a_a \sin\omega t. \tag{1}$$

where $\rho = \rho_{ucm.}\dfrac{V_c}{S_c}\cdot\dfrac{l_c}{L_{\scriptscriptstyle M}}$ - is the dewatered material cross-section density, kg/m$^2$; $\rho_{ucm.}$ is the dewatered material density, kg/m$^3$; $V_c$ is the material on screen volume per second, m$^3$/s; $S_c$ is the screen area passed by material per 1s., m$^2$/s; $l_c$ is the length of material path per 1s, m; $L_{\scriptscriptstyle M} = 1$ m; $y$ is the length of material path on the screen deck, m; $K$ is the viscous resistance coefficient, N·s/m$^2$; $a_a$ and $\omega$ are the disturbance force amplitude (m) and frequency (s$^{-1}$) of the working screen surface forced vibration, respectively.

Since layer plastic strains are nonreversible and develop in the down-slope direction $h(t)$ only, during the semi-oscillation layer moves as a solid body having mass of $\rho h_i$, and during the second semi-oscillation it moves as an inertial viscoplastic Bingham body in accordance with equation (1).

Equation (1) will be solved in following form

$$y = A_i \sin\omega t + B_i \cos\omega t. \tag{2}$$

On substitution of the expression (2) in equation (1) and setting equal to zero the sum of coefficients at the $\sin\omega t$ and $\cos\omega t$ functions, we determine the values of $A_i$ and $B_i$ constants.

The expression (2) can be represented by as follows:

$$A_i = a_a \frac{\frac{h_1 - h_i}{h_1 - h_m}k_{\text{п}} - \rho h_i \omega^2}{\left(\frac{h_1 - h_i}{h_1 - h_m}k_{\text{п}} - \rho h_i \omega^2\right)^2 + K^2\omega^2}; \quad B_i = \frac{a_a K\omega}{\left(\frac{h_1 - h_i}{h_1 - h_m}k_{\text{п}} - \rho h_i \omega^2\right)^2 + K^2\omega^2}. \tag{3}$$

$$y = a_{a\,i} \sin(\omega t + \varphi_i), \tag{4}$$

where $a_{a\,i} = \sqrt{A_i^2 + B_i^2} = a_a \left[\left(\frac{h_1 - h_i}{h_1 - h_m}k_{\text{п}} - \rho h_i \omega^2\right)^2 + K^2\omega^2\right]^{-\frac{1}{2}}$, m;

$$\varphi_i = arctq\left(B_i / A_i\right) = arctq\frac{K\omega}{\frac{h_1 - h_i}{h_1 - h_m}k_{\text{п}} - \rho h_i\omega^2}, \text{ deg.}$$

Thus, the dewatered material layer surface vibration occurs with frequency of the forced vibration, and a phase displacement in the $i^{th}$ oscillation period is equal to $\varphi_i$ and depends on the layer height.

At the same time, since the concentrated pulp viscosity coefficient dependence on the vibration parameters is represented as [4]

$$\eta = \eta_o + \frac{k}{a_a\omega^3}, \tag{5}$$

where $k$ is the coefficient, N/m·s$^2$; $\eta_o$ is the apparent viscosity coefficient due to the vibration thixotropic destruction of dispersion medium, N·s/m$^2$, then rate of material layer compaction on the vibrating deck will be

$$\frac{dy}{dt} \approx -\frac{2h\tau_c}{\eta_o + \frac{k}{a_a\omega^3} + \frac{8h\tau_c}{\pi a_a\omega}}. \tag{6}$$

$\tau_c$ and $k$ parameters appearing in this equation are subject to the experimental determination.

As the expression (6) indicates, the layer compaction rate depends on the current layer height, $h$, layer material shear stress, $\tau_c$, apparent viscosity, $\eta$, and working surface vibration frequency, $\omega$, and amplitude, $a_a$.

The current values of apparent viscosity and shear stress can be well approximated by expressions [7]:

$$\eta = \eta_o \cdot \frac{h_1 - h_o^*}{h - h_o^*}, \tag{7}$$

$$\tau_c = \tau_o \cdot \frac{h_2 - h_o^*}{h - h_o^*}, \tag{8}$$

where $\eta_o$ and $\tau_o$ are apparent viscosity (N·s/m$^2$) and shear stress (N/m$^2$) at some initial solids content; $h_1$, $h_2$ and $h$ are initial, final and current layer heights, m; $h^*_o$ is the layer height at the closely packing of material mixture, m.

Difference $h$ - $h^*o$ appearing in the denominators of the expressions (7) and (8) is positive and always above 0.

Using relationships (7) and (8) for the apparent viscosity and shear stress, and one of $\eta(a_a, \omega)$ relationships, for example, $\eta = \eta_o + k / \omega^3$, we can rewrite equation (6)

$$\frac{dh}{dt} \approx -\frac{2\pi\tilde{h}_2\tau_o a_a\omega^3 h}{\pi\tilde{h}_1\left(\eta_o a_a\omega^3 + k\right) + 8\tilde{h}_2\tau_o\omega^2 h}, \tag{9}$$

where: $\tilde{h}_1 = h_1 - h_o^*,\ \tilde{h}_2 = h_2 - h_o^*$.

Integrating equation (9)

$$\frac{\tilde{h}_1\left(\eta_o a_a\omega^3 + k\right)}{2\tilde{h}_2\tau_o\omega^3 a_a}\int\frac{dh}{h} + \frac{4}{\pi a_a\omega}\int dh = -\int dt + C \tag{10}$$

at the initial condition

$$h = h_o \text{ at } t = 0, \tag{11}$$

we obtain $$t=\frac{4}{\pi a_a\omega}\left(h_1-h\right)-\frac{\tilde{h}_1\left(\eta_o a_a\omega^3+k\right)}{2\tilde{h}_2\tau_o\omega^3 a_a}\ell n\frac{h}{h_1}. \tag{12}$$

Expression (12) is kinetics equation for material layer compaction on the vibrating working surface during the dewatering process.

In order to plot relationships of working surface dynamic parameters effects on the dewatered material layer compaction performances, the experimental calculation was conducted with equations (9) and (12) coefficients preselected in accordance with the results of material tests described in studies [5-7]: $\tau_o$ = 10N/m$^2$, $k$ = 106N/m·s$^2$, $\eta_o$ = 103N·s/m$^2$, $h_1 = h_2$ = 0.1m. Based on the calculation results, the working surface vibration amplitude and frequency were plotted against the material compaction rate over the amplitude range from 0.25 to 4 mm and frequency range from 250 to 3500min$^{-1}$ (fig. 1). As plots indicate, within the traditional vibration frequency and amplitude range of existing screens increase in frequency affects the increase in layer compaction rate considerably more effectively than increase in vibration amplitude. For example, at the assumed material constants, increase in working surface vibration frequency from 250 to 2000min$^{-1}$ will result in the intensive increase in compaction rate for layer of 0.1m; however, further increase in vibration frequency causes sharp decrease in compaction rate. Reduction of material layer also causes decrease in compaction rate.

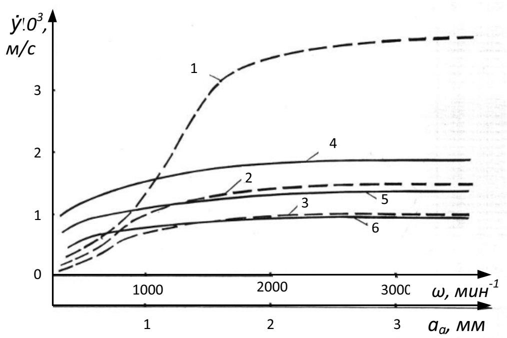


Fig. 1. Diagrams of the material compaction rate, $\dot{y}$, dependency on the working surface vibration amplitude $a_a$ and frequency $\omega$: 1, 2, 3 - $\dot{y}(\omega)$ at $a_a$ = 1mm; 4, 5, 6 - $\dot{y}(a_a)$ at ω = 750min$^{-1}$. Layer height: 1, 4 - $h$ = 0.1; 2, 5 - $h$ = 0.05; 3, 6 - $h$ = 0.025mm.

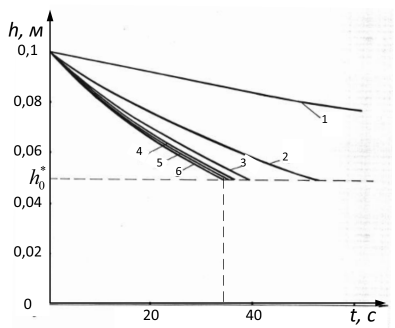

Fig. 2. Kinetics of simulated material layer compaction at working surface vibration amplitude of 2mm and frequency: 1 – 500min$^{-1}$; 2 – 1000min$^{-1}$; 3 – 1500min$^{-1}$; 4 – 2000min$^{-1}$; 5 – 2500min$^{-1}$; 6 – 3000min$^{-1}$.

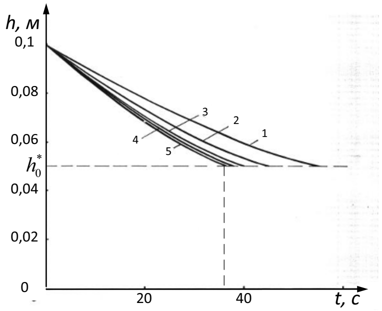


Fig. 3. Kinetics of simulated material layer compaction at working surface vibration frequency of 1500min$^{-1}$ and amplitudes: 1 – 0.5mm; 2 – 1.5mm; 3 – 2.5mm; 4 – 3.5mm; 5 – 4.0mm.

Increase in vibration amplitude for the simulated material is effective up to the value of 2mm, and further increase in amplitude causes insignificant increase in compaction rate.

Fig. 2, 3 shows that time of dewatered material transition to structured state makes at least 35s. After this time, compaction of structured material requires vibration pattern alteration, which could be achieved on the same screen by alteration of inclination angle of working surface tail section [8, 9].

In order to verify theoretical findings, the comparative production tests of coal slurry products dewatering technology were conducted at the Gukovskaya CPP with the use of high-frequency screens ГВЧ-41 and ГВЧ-41М [10, 11]. Screening surface of first screen consists of three uniform inclination sections, while that of the second screen consists of three different inclination sections forming the asymmetrical S-shaped curve.

*Table*

**Averaged grain-size distribution of dewatering products**

| Grade, mm | Grades to product yield, % | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Dewatering of WSS concentrate and ГЦ-75 thickened product | | | | | | Dewatering of WSS rejects | | | | | |
| | ГВЧ-41 screen | | | ГВЧ-41М screen | | | ГВЧ-41 screen | | | ГВЧ-41М screen | | |
| | Raw | Over size | Un-dersi ze | Raw | Over size | Un-dersi ze | Raw | Over size | Un-dersi ze | Raw | Over size | Un-dersi ze |
| 2.0-3.0 | 15.32 | 19.5 | - | 17.36 | 24.9 | - | 19.32 | 25.7 | - | 20.64 | 30.9 | - |
| 1.5-2.0 | 11.12 | 14.1 | - | 12.06 | 17.3 | - | 18.64 | 24.6 | - | 17.70 | 26.5 | - |
| 1-1.5 | 12.24 | 15.6 | - | 11.22 | 16.1 | - | 11.53 | 15.3 | - | 12.42 | 18.6 | - |
| 0.5-1 | 13.57 | 17.18 | - | 12.41 | 17.8 | - | 11.33 | 15.0 | - | 10.29 | 15.4 | - |
| -0.5 | 47.75 | 33.6 | 100.0 | 46.95 | 23.9 | 100.0 | 39.18 | 19.4 | 100.0 | 38.95 | 8.6 | 100.0 |
| **Total** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** | **100.0** |
| Moisture content, % | - | 28.0 | - | - | 22.0 | - | - | 21.3 | - | - | 16.0 | - |
| Solids con-tent, g/l | 425 | - | 40 | 425 | | 42 | 433 | | 160 | 433 | | 190 |

Screens are installed in the plant process line at the stage of dewatering of wet spiral separation concentrate and thickened product of ГЦ-75 hydrocyclones and at the stage of dewatering of wet spiral separation rejects.

Averaged performance indicators of ГВЧ-41 and ГВЧ-41М screens are resulted in the Table.
The best values of oversize product moisture content achieved at dewatering of WSS concentrate and thickened product of ГЦ-75 under otherwise equal conditions made 28.0% on the average at the ГВЧ-41 screen (at $\alpha_1 = \alpha_2 = \alpha_3 = 0°$) and 22.0% at the ГВЧ-41М screen (at $\alpha_1 = -20°$, $\alpha_2 = 0°$, $\alpha_3 = +15°$); at dewatering of WSS rejects – 21.3% on the average at the ГВЧ-41 screen (at $\alpha_1 = \alpha_2 = \alpha_3 = 0°$) and 16% at the ГВЧ-41М screen (at $\alpha_1 = -15°$, $\alpha_2 = 0°$, $\alpha_3 = +10°$).

**Conclusion.** Use of dewatering surface with the different inclination screen sections as a substitute for linear surface is reasonable from the technology viewpoint.

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# Statistical analysis of sedimentation process of mineral suspension with application of bioflocculation

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**Abstract**
On the basis of empirical data the approximation of settling curve $h=f(t)$ for various initial concentrations of the coal sludge flotation concentrate and coal tailings suspension was done in the paper, where $t$ – time of settling, $h$ –way passed by the suspension. The experiment was conducted for both cases with use of bioflocculant and without it. The bioflocculants were, in various doses, activated sludges originating from municipal sewage treatment plant. Such functions as exponential, logarithmic or rational ones were used to conduct the approximation. The evaluation of fitting was done by means of mean squared standard error. On the basis of obtained functions describing settling curve the settling velocity was determined as $v(t) = f'(t)$ and velocity was calculated as $v = \frac{1}{\Delta}\int_0^{\Delta} v(t)dt$, where Δ is maximum time of settling. The determined settling curves were the basis to investigate relation between settling velocity and initial concentration as well influence of applied bioflocculants on settling velocity. The evaluation of influence of bioflocculants sort on efficiency of mineral suspensions settling was done.

Key words: *settling velocity, bioflocculation, approximation, statistical analysis, active sludge, mineral suspension, sedimentation.*

## 1. Introduction

In processes of mechanical clearing of wash waters in technological systems of mineral processing mainly methods of thickening and filtration of suspensions are in use. In majority these suspensions characterize by fine or ultrafine granulation. The first stage of dewatering – thickening of mineral suspensions occur mainly in radial concentrators of Dorr type and then the thickened underflow is directed to filtration stage. It is worthy to notice that sedimentation velocity of ultrafine particles in the concentrator is low (Krawczykowska et al., 2005; Thomas et al., 2014).
Mineral suspension sedimentation velocity depends mainly on solid phase contents and particle size. Ultrafine particles of sizes close to colloidal dimensions sediment with low velocity because they move in various directions in suspension. To accelerate sedimentation process and raise efficiency of thickening the special chemical substances, so-called flocculants, are added to suspension causing creation of particles aggregates. Such constructed aggregates behave like particles of bigger sizes and sediment much easier. Thanks to application of flocculants is possible to efficiently accelerate time of overflow clarification and underflow thickening in concentrator. The flocculants can be organic origin compounds as well synthetic agents. The natural origin agents feature with high consumption being equal to from several even to several dozen grams per 1 Mg of substance. These are mainly polysaccharides and organic colloids. Polysaccharides are known also as starch agents produced from potato starch. Organic colloids created on the basis of glue and resins can be applied in acid environment, featured by high efficiency, but their solubility is limited and hard to perform. Synthetic agents feature with much lower consumption – from several grams per 1 Mg of solid phase – however, their price is high. To this group of agents the compounds produced from polyacrylamides, esters and fatty acids. These compounds have polar structure (anion, cation or anion-cation) or apolar structure (no ionic). Anion agents are being used mainly to

 
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clarify and thicken suspensions of alkaline or neutral reaction and cation ones in acid environment. No matter what type of flocculent is in use it is crucial to select its optimal dose because too much amount of flocculent can cause sudden growth of suspension viscosity what negatively influence on sedimentation process efficiency.

Introduction of flocculants to industrial technologies changes course of sedimentation process considering both quantity and quality. In short time is possible to completely separate solid phase from liquid. However, economic reasons (price of flocculant) causes searching for new methods and compounds which would give satisfying technological effects for lower price. More and more often the scientific researches with application of microorganisms to mineral suspensions flocculation (Blaschke et al. 1994; Kisielowska and Surowiak 2015; Smith and Miettinen 2006; Vijayalakshmi and Raichur 2002) are conducted. It was stated that during sludge cleaning processes such metabolic changes occur which cause microorganisms being present in activated sludge to produce slimy substance which chemically behaves like polymers (Szulicka 1980).

The paper presents the results of investigating sedimentation of concentrate nad waste suspension of coal slime with application of new means accelerating the process of mineral suspensions settling – bioflocculants in form of active sludge originating from the municipal tailings treatment plant.

## 2. Graphical presentation of suspension settling process

To evaluate the course of suspension sedimentation process the sedimentation curve can be used as well the determined value of sedimentation velocity. In purpose of plotting sedimentation curve the analysis of suspensions settling process must be done. This process is based on observation of phenomena occurring during periodical sedimentation conducted in laboratory scale. The investigation of suspension is based on observing changes occurring in the cylinder filled with suspension according to the time. With the time passing the individual zones are being created in the vessel, which are clarified water zone in the upper part, equivocal collective settling zone located below, medium zone and compression zone being observed at the bottom of the vessel. In all these zones various share of solid phase on appropriate vessel heights can be noticed till total transfer of solid phase particles to the sludge collected at the bottom and clarified water above the sludge.

## 3. Experimental part

The investigations of sedimentation of concentrate and tailings of coal slime were conducted by means of bioflocculants in form of active sludge originating from mechanical and biological municipal waste treatment plant. The materials to investigation were suspensions of concentrate and tailings originating from flotation of coking coal. The measurements of settling velocity of suspensions by initial concentrations being equal to 40 and 60 g/dm$^3$ without addition of bioflocculant – so-called samples “0”- were conducted. Then, the sedimentation tests were done for determined initial concentrations with addition of active sludge in amount of 20 or 50 cm$^3$/dm$^3$ for flotation concentrate and 20 cm$^3$/dm$^3$ for tailings of coal slime with share of active sludge.

The active sludge collected from municipal tailings treatment plant used as bioflocculant to sedimentation of flotation concentrate characterized with zoogleal ballooning, biocoenosis on the level 11.3 what indicates that their activity is good, quantity is above 10$^6$ units/dm$^3$ and occurrence of settled and crawling ciliates in trophic equilibrium is good too. Among settled ones there are numerous *Vorticelle* and colonies of *Epistyli*s which confirm the stability of the sludge. Also, numerous amebas *Arcelle* as well rotifers and some sysygambis *Tokophrya* could be observed in the sludge

Microscopic evaluation of activated sludge’s fluffs used in flotation concentrate experiments, indicated their irregular shape, compact structure, weak durability, sludge biotic index (SBI) of II class and domination of crawling and predatory ciliates, as well small diversity of specimens and small amount of bacteria what indicates that the quality of sludge is average. The dominant filament was *Microthrix parvicella* (0.6).

### 4. Elaboration of results

For each discussed case the approximation of the settling curve was done and mean settling velocity were calculated. To evaluate the quality of fitting of approximated curve to the real data the mean standard error value was used, calculated according to the formula (1) (Niedoba, 2013), with the equations of approximated functions presented in Table 1.

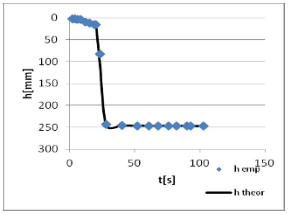
$$MSE = \sqrt{\frac{(h(t) - h(t)_{theor})^2}{n-2}} \tag{1}$$

Table 1. The functions *h*(*t*) and *v*(*t*) for investigated materials

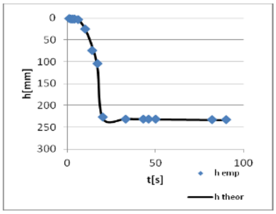
| Type of sample | Height in function of time *h*(*t*) | Velocity in function of time *V*(*t*) |
|---|---|---|
| Concentrate, sample "0" | | |
| 40 g/dm$^3$ | $h(t)=\begin{cases} 0.57t^{1.07} & for\ t\in[0,20] \\ 28,98t-565,71 & for\ t\in(20,28) \\ \frac{248}{1+0.01e^{-0.01t}} & for\ t\in[28,103] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 0,60t^{-0.07} & for\ t\in[0,20] \\ 28.98 & for\ t\in(20,28) \\ \frac{3.64e^{-0.01t}}{(1+0.01e^{-0.01t})^2} & for\ t\in[28,103] \end{cases}$ |
| 60 g/dm$^3$ | $h(t)=\begin{cases} t^{0.61} & for\ t\in[0,6] \\ 5.67t-31.00 & for\ t\in(6,10) \\ 0.04t^{2.81} & for\ t\in[10,17] \\ 38.5t-540.58 & for\ t\in(17,20) \\ \frac{234}{1+0.03e^{-0.02t}} & for\ t\in[20,90] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 0,60t^{-0.07} & for\ t\in[0,20] \\ 28.98 & for\ t\in(20,28) \\ \frac{3.64e^{-0.01t}}{(1+0.01e^{-0.01t})^2} & for\ t\in[28,103] \end{cases}$ |
| Concentrate, 50 cm$^3$/dm$^3$ | | |
| 40 g/dm$^3$ | $h(t)=\begin{cases} 22t^{1.88} & for\ t\in[0,3] \\ \frac{176}{1+1+0,05e^{-0.02}} & for\ t\in(3,120) \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 41.36t^{0.88} & for\ t\in[0,3] \\ \frac{3.50e^{-0.02t}}{(1+0.05e^{-0.02t})^2} & for\ t\in(3,120) \end{cases}$ |
| 60 g/dm$^3$ | $h(t)=\begin{cases} 23.89t^{0.90} & for\ t\in[0,8] \\ 47.36t-224.38 & for\ t\in(8,9) \\ \frac{212}{1+0.06e^{-0.02t}} & for\ t\in[9,121] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 21.44t^{-0.10} & for\ t\in[0,8] \\ 43.36 & for\ t\in(8,9) \\ \frac{4.90e^{-0.02t}}{(1+0.06e^{-0.02t})^2} & for\ t\in[9,121] \end{cases}$ |
| Concentrate, 20 cm$^3$/dm$^3$ | | |
| 60 g/dm$^3$ | $h(t)=\begin{cases} 12t^{2.63} & for\ t\in[0,2] \\ 43.58t^{0.77} & for\ t\in[2,7] \\ 26.98t+6.68 & for\ t\in(7,8) \\ \frac{234}{1+0.06e^{-0.04}} & for\ t\in[8,120] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 31.58t^{1.63} & for\ t\in[0,2] \\ 33.62t^{-0.23} & for\ t\in[2,7] \\ 26.98 & for\ t\in(7,8) \\ \frac{9.27}{(1+0.06e^{-0.04})^2} & for\ t\in[8,120] \end{cases}$ |
| Tailings, sample "0" | | |
| 40 g/dm$^3$ | $h(t)=\begin{cases} 9.35t^{0.96} & for\ t\in[0,20] \\ 10.69t-47.23 & for\ t\in(20,25) \\ \frac{234}{1+0.27e^{-0.06t}} & for\ t\in[25,70] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 8.99t^{-0.04} & for\ t\in[0,20] \\ 10.69 & for\ t\in(20,28) \\ \frac{13.73e^{-0.06t}}{(1+0.27e^{-0.06t})^2} & for\ t\in[28,103] \end{cases}$ |
| 60 g/dm$^3$ | $h(t)=\begin{cases} 8,99t^{0,82} & for\ t\in[0,12] \\ 16,65t-130,99 & for\ t\in(12,13) \\ \frac{220}{1+25e^{-0,09t}} & for\ t\in[13,90] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 7,35t^{-0,18} & for\ t\in[0,12] \\ 16,65 & for\ t\in(12,13) \\ \frac{106,88e^{-0,09t}}{(1+5,25e^{-0,09t})^2} & for\ t\in[13,90] \end{cases}$ |
| Tailings, 20 cm$^3$/dm$^3$ | | |
| 40 g/dm$^3$ | $h(t)=\begin{cases} 10t^{0,63} & for\ t\in[0,3] \\ 36,74t-90,22 & for\ t\in(3,5) \\ 26,86t^{0,7748} & for\ t\in[5,12] \\ 14,46t+10,69 & for\ t\in(12,15) \\ \frac{243}{1+0,10e^{-0,0288t}} & for\ t\in[15,90] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 6,31t^{-0,37} & for\ t\in[0,3] \\ 36,74 & for\ t\in(3,5) \\ 20,81t^{-0,22} & for\ t\in[5,12] \\ 14,46 & for\ t\in(12,15) \\ \frac{0,73e^{-0,03t}}{(1+0,10e^{-0,03})^2} & for\ t\in[15,90] \end{cases}$ |
| 60 g/dm$^3$ | $h(t)=\begin{cases} 9,86t^{1,02} & for\ t\in[0,10] \\ 15,7t-53,8 & for\ t\in(10,12) \\ 39,77\ln t+36,01 & for\ t\in[12,90] \end{cases}$ | $v(t)=h'^{(t)}=\begin{cases} 10,05t^{0,02} & for\ t\in[0,10] \\ 15,7 & for\ t\in(10,12) \\ \frac{39,77}{t} & for\ t\in[12,90] \end{cases}$ |

On the basis of approximated functions the graphs of functions were elaborated. The pictures with the value of MSE were positioned on Figs 1-4.

a)



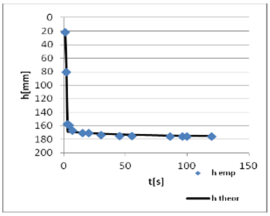

b)



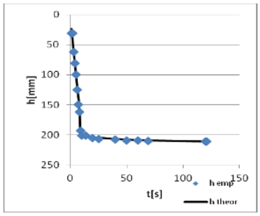

MSE = 4.79 MSE=3.61

Fig. 1. The function $h(t)$ for sample "0" – concentrate; a) concentration 40 g/dm$^3$; b) concentration 60 g/dm$^3$

c)



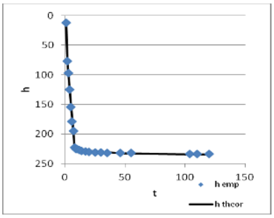

d)




MSE=4.51 MSE=5.91

Fig. 2. The function $h(t)$ for dose 50 cm$^3$/dm$^3$ – concentrate; c) concentration 40 g/dm$^3$; d) concentration 60 g/dm$^3$

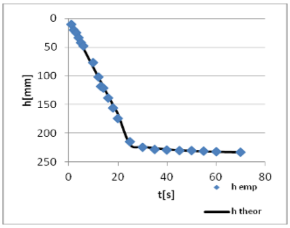
e)



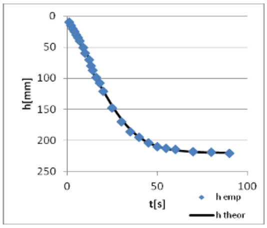

MSE=2.26

Fig. 3. The function $h(t)$ for dose 20 cm$^3$/dm$^3$ – concentrate; e) concentration 60 g/dm$^3$

f)




g)




MSE=3.90 MSE=2.70

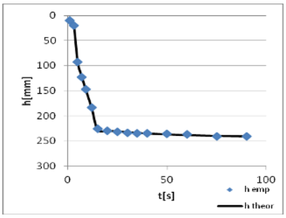
Fig. 4. The function $h(t)$ for sample "0" – tailings; f) concentration 40 g/dm$^3$; g) concentration 60 g/dm$^3$

On the basis of calculated functions the settling velocity was calculated according to the formula:

$$v = \frac{1}{\Delta}\int_0^{\Delta} v(t)dt \quad (2)$$

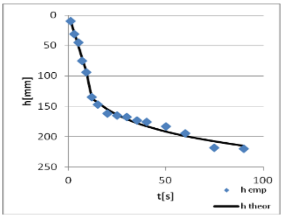
Where Δ is the value of time where the plot of function *h*(*t*) achieves maximum curvature radius.
The values of average settling velocity for each case were presented in Table 2.

h)




MSE=0.94

i)




MSE=3.68

Fig. 5. The function *h*(*t*) for dose 20 cm$^3$/dm$^3$ – tailings; h) concentration 40 g/dm$^3$; i) concentration 60 g/dm$^3$

Table 2. Settling velocity according to sort of sample

| Sample | Settling velocity $v$ [m/min] |
|---|---|
| "0" concentrate 40g/dm$^3$ | 0.53 |
| "0" concentrate 60 g/dm$^3$ | 0.69 |
| 50 cm$^3$/dm$^3$ concentrate 40 g/dm$^3$ | 1.46 |
| 50 cm$^3$/dm$^3$ concentrate 60 g/dm$^3$ | 1.35 |
| 20 cm$^3$/dm$^3$ concentrate 60 g/dm$^3$ | 1.68 |
| "0" tailings 40 g/dm$^3$ | 0.53 |
| "0" tailings 60 g/dm$^3$ | 0.34 |
| 20 cm$^3$/dm$^3$ tailings 40 g/dm$^3$1 | 0.91 |
| 20 cm$^3$/dm$^3$ tailings 60 g/dm$^3$ | 0.62 |

On the basis of conducted analyzes and calculated values of settling velocity of concentrate and coal slime tailings it was stated that settling velocity of suspensions of these materials without applying agents accelerating sedimentation is higher for lower initial concentrations in both cases. For concentrate it was stated that sedimentation velocity of sample without bioflocculant for higher initial concentration (60 g/dm$^3$) is higher than in case of lower concentration (40 g/dm$^3$). The first fragment of the curve has bizarre shape, which can be caused by the fact that the the autocoagulation of suspension occurred. That means that the finest particles of solid phase joined and created particles aggregates because of their mutual influences and started to sediment in this form after time t. For investigated mineral suspensions . For the investigated mineral suspensions two or even three times higher values of settling velocity were observed dependably on the dose of active sludge. By constant doses of bioflocculant, the settling velocity changes dependably on initial concentration and is lower by higher shares of solid phase in the volume unit.

Comparing the settling velocity for the suspension of concentrate and coal slime tailings by initial concentration being equal to 60 g/dm$^3$ and dose of bioflocculant 20 cm$^3$/dm$^3$ it is visible that the value of the velocity is equal to 1.68 m/min for concentrate and 0.62 m/min for tailings. Minimally higher value of solid phase density of tailings (1800 Mg/m$^3$) than concentrate solid phase density – 1500 Mg/m$^3$ should accelerate the settling velocity of coal slime tailings. However, the influence of the quality of bioflocculant on the sedimentation process effect is important here. The bioflocculants being used in the investigations in the form of active sludge differed between themselves by the presence of various sorts of microorganisms in their composition. That was caused by the fact that the active sludge was used in the summer period for concentrate samples and winter period for tailings samples. So, the flocculation abilities of the sludge depend on the surrounding temperature because it determines the development of the microorganisms.

## 5. Conclusions

On the basis of conducted experiments the following conclusions can be made: it is hard to correctly approximate the functions $h(t)$ – it is necessary to combine several functions (at least 2). It is caused by irregular shape of the graph; The obtained combinations were well fitted to the experimental data; Comparing the plots of $h(t)$ function it can be stated that the most bizarre shape of the function was obtained for the sample “0” of concentrate (with both concentrations 40 g/dm$^3$ and 60 g/dm$^3$). It can be caused by the autocoagulation effect; Adding of the doses of bioflocculants causes significant growth of the settling velocity for both cases of concentrate and tailings; In case of tailings the concentration 40 g/dm$^3$ causes the higher value of the settling velocity than in case of concentration 60 g/dm$^3$, the opposite relation can be observed for concentrate, sample “0”, but in case of adding bioflocculant the relation changes; The values of settling velocity in case of concentrate were higher than in case of tailings; Application of active sludge to sedimentation process of mineral suspensions accelerates the process of solid phase particles settling; The investigations of applying new agents accelerating sedimentation process indicated, that certain group of microorganisms exists which indicates flocculative action in suspensions of mineral particles; Application of microorganisms in processes of suspensions sedimentation allows to minimize costs significantly because such types of bioflocculants occur commonly and their cells can be collected from soil, water and air.

The paper was elaborated within the statutory works no. 11.11.100.196 and 11.11.100. 276

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# New control head design for hyperbaric disk filter gives better performance and longer life-time

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**Abstract**

For ultimate mechanical dewatering in coal beneficiation plants hyperbaric disk filtration (HBF) represents the best available technology to achieve low residual moisture. HBF is a continuous operation with high throughputs as needed in coal industry. Three zones are located on the respective disks and passed through consecutively, i.e., cake formation, drying and cake release. The disk is segmented by so called filter elements. Once such a filter element is entirely submerged in the slurry, access to the filtrate receiver is granted. This is also accompanied with a sudden pressure difference between the elevated pressure in the vessel and ambient air resulting in a two phase flow which causes abrasion. While the pressure difference across the filter increases, filter cake starts building up. When separating this venting stage from the cake formation, this vented air can be handled separately to reduce abrasion, as already proven in large scale HBF application in highly abrasive applications. Since most of the air is already released before the majority of cake formation starts, filtrate flow is at ease which improves cake formation and thus the throughput is increased. This is a quite simple measure for substantially improving performance also for existing installations.

**Key words**

hyperbaric disk filter, abrasion, cake filtration, control head, high pressure, air venting, filtrate flow, coal dewatering

## 1. Introduction of dewatering of ultrafine coal fraction

Coal beneficiation plants increase the valuable fraction of the mined material. Due to the general trend to mine lower grade sites and the desire to extract more of the mined material as valuable product and at the same time reduce the problematic waste fraction, beneficiations plants are increasingly challenged. Among the various product streams, it is particularly the increasing ultra-fine fraction that poses a problem. If this fraction does not meet product specifications in terms of residual moisture and coal content, it represents tailings thus a waste stream that needs special treatment [1].

If other product streams exceed the targeted overall product specification, part of this ultrafine fraction can be used to blend it into the final product though this attempt is generally limited [2]. Floatation is generally employed to remove part of the inert fraction and thus meet the specification in terms of carbon content. The product, present as froth, needs to be dewatered to a residual moisture level in the order of 10% at most. A combined mechanical dewatering step followed by thermal drying can cope with this task but today it is generally acknowledged that thermal drying is expensive and thus shall be phased out in favor of mere mechanical dewatering [2]. Dependent upon the remaining ash content and particle size distribution, different mechanical dewatering equipment can be employed. A preliminary decision, which type of equipment should be considered or can be ruled out, can be made based on a simple correlation [3]. Of course, laboratory tests are paramount to check whether this preliminary decision upon the type of

 
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equipment is correct and to arrive at substantiated decision in terms of specific throughput and residual moisture as a function of operating parameters [4].

For the ultrafine coal fraction, hyperbaric disk filters (HBF), chamber filter press or belt filter press represent feasible mechanical dewatering types of equipment [5]. HBF is particularly advantageous for very high throughputs where numerous industrial references are operating most satisfactorily worldwide [6]. The filtration is generally fast and the rotation speed is often governed by the drying step particularly for high ash containing slurries. Thus the cake filtration step is sometimes intentionally impeded and a thinner filter cake is accepted for the sake of a better dewatering situation [6]. Steam treatment of the filter cake during the filtration procedure is another measure to speed up the drying process and reach lower residual moisture [7].

## 2. Set-up and operation of large scale hyperbaric filter

The operation and design features of hyperbaric filters (HBF) are described in detail elsewhere (e.g. [9], [10]) with the main components shown schematically in Figure 1. The suspension is pumped to the filter trough. The level in the filter trough serves as correcting variable for regulating the flow from the feed pump. The gear motor drives the filter shaft with its one to two control heads and one to 12 discs, each with 20 segments. Agitators are used to homogenize the content of the filter trough and to avoid sedimentation. Figure 2 shows a picture of a large scale unit.

A disc filter operates continuously: Filter cake formation starts once the active area of one segment is fully submerged in the suspension of the filter trough (filtration stage) and the control head allows flow between the filtrate collection tube and the filtrate receiver. Once the filter cake is formed and leaves the suspension, it is dried. Generally an air drying step sets in immediately after cake formation at saturation

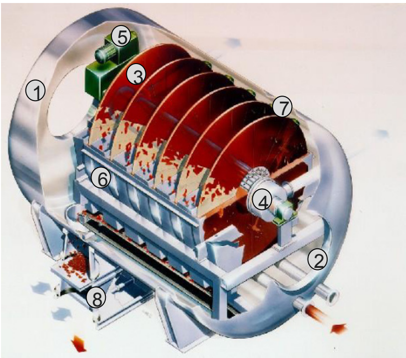


**Figure 1:** Hyperbaric disc filter: 1 pressure vessel, 2 man hole, 3 filter disc, 4 control head, 5 filter drive, 6 filter trough, 7 agitators, 8 discharger

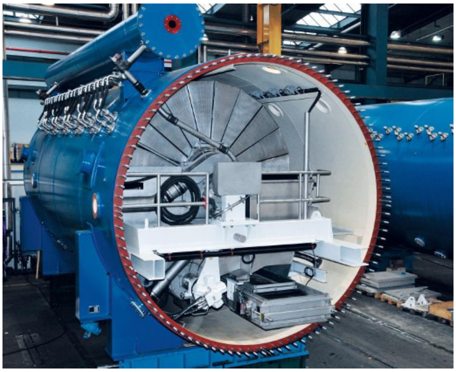

**Figure 2:** Hyperbaric disc filter located inside the pressure vessel

one. Afterwards the cake is removed from the disks by a snap blow device before the cycle starts again. In the snap blow zone the pressure level in the filter cell and filtrate collection tube is raised slightly above the pressure in the vessel to allow cake removal. The removed cake is transported via discharge chutes and trough chain or belt conveyers to the discharge device which is often designed as a sliding gate system to seal up to around 6 bars against ambient pressure. Through the discharge device the cake exits the pressure vessel.

The design of the control head ensures that the three zones are sealed from each other and it also sets the available areas for the respective zones depicted through cake formation angle, $\alpha_1$, and air drying angle, $\alpha_2$, respectively (see Figure 3 (left)). The zones are linked with the rotational speed of the filter discs n.

Clearly a larger drying angle and/or a longer drying time give lower cake moisture, and a larger formation angle and/or a longer filtration time increase the throughput. The area specific throughput $\dot{Q}_{HBF}$ is given by

$$\dot{Q}_{HBF} = \frac{m_{cake,D.S.}}{A_f \cdot (t_1 + t_2)} \cdot \frac{(\alpha_1 + \alpha_2)}{2 \cdot \pi} \tag{1}$$

where $m_{cake,D.S.}$ denotes the dry mass of the filter cake, $A_f$ is the filtration area, $\alpha_1 / \alpha_2$ and $t_1 / t_2$ relate to the angle and time for filtration and drying, respectively. Moreover the link between time, angle and rotational speed n is given by

$$n = \frac{1}{t_1} \cdot \frac{\alpha_1}{2 \cdot \pi} = \frac{1}{t_2} \cdot \frac{\alpha_2}{2 \cdot \pi} \tag{2}$$

Depending upon the material properties a rotational speed between 0.5 and 2 (1/min) is most common [10].

## 3. Situation at entrance to cake formation zone

In the cake formation zone the initial conditions for the filter cell and filtrate collection tube are set by the previous zone, i.e., the snap blow zone. After the snap blow zone the filtrate collection tube is disconnected from the snap blow tank by the control head. Consequently the elevated pressure is preserved and equilibrated with the pressure tank. Moreover filtrate and possibly dispersed fine particles may also be present in the filter cells and filtrate collection tube, respectively. This is particularly an issue if during the air drying zone not all of the filtrate is discharged and the snap blow caused some so called re-wetting of the filter cake.

The cake formation zone starts once the control head provides an opening that connects the filtrate collection tube with the downstream filtrate receiver generally operating at ambient pressure. In the cake formation zone one but often several filtrate collection tubes discharge their filtrate often mixed with remaining air through the control head opening into one common filtrate collector line which enters the exterior filtrate receiver where liquid and air are separated.

The filtrate receiver, more specifically the liquid level in the filtrate receiver is located below the lowest filtrate collection tube to utilize geodetic height and allow automatic filtrate discharge even in case of equilibrated pressure levels. Of course, also in normal operation where a large pressure difference is maintained between the pressure vessel and the pressure in the filtrate collection vessel, the geodetic height can be utilized to increase the driving force for cake formation as well as the flow for filtrate discharge. However, its full potential can only be harvested if the relevant vertical column is liquid only.

Any gas bubbles in the liquid column will reduce the relevant density and, even worse, if a connecting gaseous streamline exists, it is not the liquid height that matters but the gaseous one. Since the density of gas is three orders of magnitude smaller than that of liquid, this renders a negligible contribution.

At the initial stage the just entering filtrate collection tube is vented as the pressure is reduced and cake formation sets in simultaneously. Gas velocities are high and a situation of Mach one is approached. This highly turbulent flow causes re-entrainment of the filtrate together with its small fraction of fine dispersed particles. This basically three phase flow results in abrasion particularly at bends and the control head. Although venting will ultimately occur towards the filtrate receiver it will also affect the neighboring filtrate collection tubes and filter cells where cake formation occurs and that are accessible by air. Unless venting is hindered by e.g. a liquid column in the joint filtrate collector line downstream the control head, venting may last only for a short time in the order of less than seconds. (The actual venting duration depends upon the geometric situation and the rotational speed that is characteristic for the opening of the cross section between the filtrate collection tube and the downstream filtrate collection line.) Although the time might be short but the abrasion attack occurs every time a new filter cell enters the cake formation zone, i.e., 20 times per rotation.

This short disrupting venting stage will cause filtrate already present in the joint filtrate line to discharge faster but it may also slow-down the filtrate discharge from neighboring filter cells and filtrate collection tubes. Thus, in the neighboring cells cake formation may be affected negatively.

## 4. New concept of two cake formation zones

The new concept aims for a physical separation between a 1st cake formation zone, $\alpha_{venting}$, where cake formation starts and pressure release, i.e., air venting occurs and a 2nd cake formation zone, $\alpha_1$, where cake formation continues and discharge of filtrate is at ease since filter cells are not disturbed by the vented air [11]. In Figure 3 (right) this new set-up is depicted schematically. In this Figure D and d denote the external disk diameter and the internal onset dimeter of active filter area. The angle $\alpha_1$, and $\alpha_2$ denote

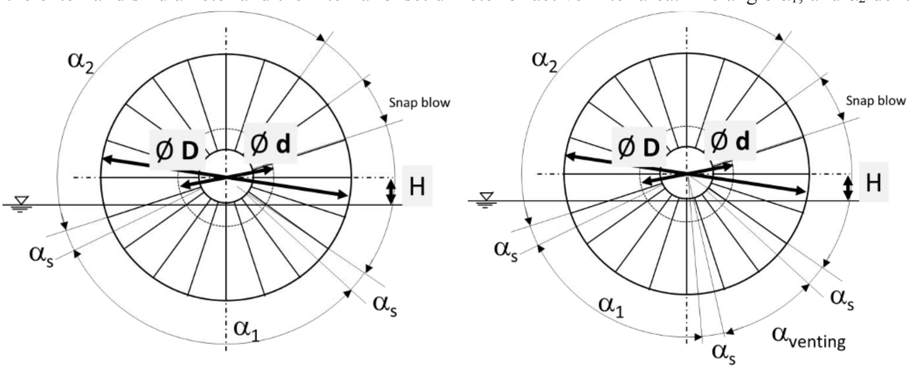


**Figure 3:** Disk side view with different zones: standard set-up (left), separate venting zone (right)

the cake formation and air drying stage, respectively. The segment with the angle $\alpha_s$ ensures proper sealing between different zones including a certain safety margin.

The vented air together with the filtrate from the 1st zone is guided to the filtrate receiver in a separate line. Abrasion will be confined to this line but it will be less since only a rather small quantity of filtrate is present. Moreover, the material selection and geometric design of the line account for this special flow situation. In case of a highly abrasive situation the rate of pressure release can be reduced by means of a special control valve. Of course, the design is more costly since the control head needs another physical barrier, and a separate line is needed from the control head to the filtrate receiver. But subsequent requirements for maintenance and service are reduced, cake formation is improved, down-time for repair is reduced.

The physical barrier between the 1st and the 2nd filtration zone reduces the available cake formation area (or angle) slightly. On mere geometric considerations the available filtration arc length is depicted for a typical set-up with and without separate venting zone in Figure 4. The submerge level *SL* is defined as the arc length of the filter that is submerged in the trough over the entire length, i.e., $2\cdot\pi$ in percent. This is valid for the drum filter as well as the disk filter though the relevant diameter for the disk filter is the inner diameter d where the active filtration area begins (see Figure 3). The link between the submerge level SL and submerge height H is given by

$$SL = \frac{1}{2} + \frac{asin\left(2 \cdot {H}/{d}\right)}{\pi} \qquad \text{in \%} \qquad (3)$$

From Figure 4 it becomes obvious that the loss of filtration area is almost negligible and increases slightly as the suspension level drops. This indicates that this concept is particularly advantageous for suspensions that require a large filtration area, i.e., poorly filterable materials which is often realized by allowing a large filtration arc. (A submerge level of around 15 to 20% represents the lowest practical limit which is dictated by the necessity of having at least one fully submerged filter cell available for cake formation.)

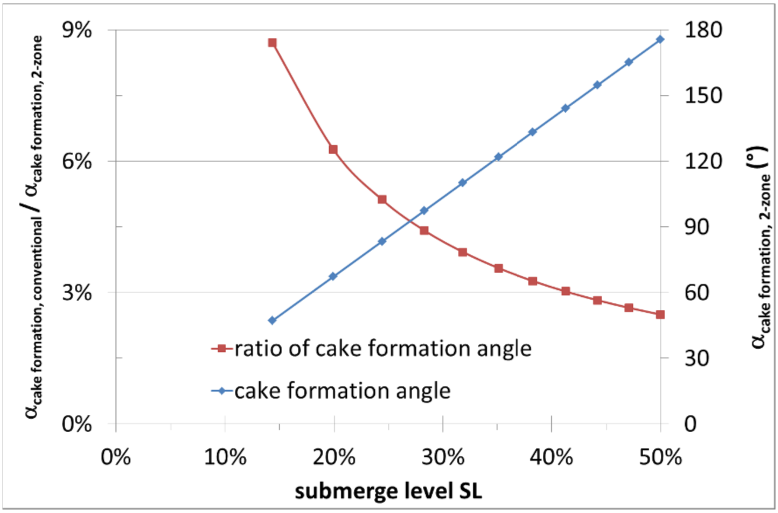


**Figure 4:** Filtration area over submerge level

## 5. Conclusions

In coal beneficiation plants hyperbaric disk filtration is widely used for ultimate dewatering of the ultra-fine fraction. High pressure is successfully utilized to accelerate cake formation and improve air drying to achieve very high throughputs on a small footprint. But high pressure differences may also have the adverse effect of high abrasion. Clever design and operation can, however, reduce abrasion or channel this problem to minimize maintenance requirements. At the start of the cake filtration zone the pressure release is critical. When separating this pressure release stage from the subsequent cake formation zone, this vented air can be handled separately and thereby decompression energy can be controlled effectively. Consequently abrasion is reduced or even avoided, at least confined, as already proven in large scale hyperbaric disk filter application in highly abrasive applications such as iron ore. The physical separation between the air venting and cake formation is again accomplished by inserting a partitioning element in the control head. Since most of the air is already released before the majority of cake formation starts, filtrate flow is at ease that allows taking full advantage of the potential of the barometric height of the filtrate in the system. As a consequence the throughput is increased for the very set-up, or the residual moisture of the cake is reduced for the same throughput, if the cake formation zone is reduced for the benefit of a larger dewatering zone. This is, in fact, a quite simple measure for substantially improving performance also for existing installations as a retrofit.

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# Part VIII
# Gravity Concentration Methods

# Experimental research on fine coal desulfurization by the efficient centrifugal gravity Separator

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**Abstract:** It is very difficult to remove sulfur and ash from fine coal. The froth flotation is not efficient in desulfurization because of the low rate of sulfide liberation from raw coal, the natural hydrophobicity caused by sulfur concentration on the surface of sulfide, and the presence of large amounts of middling feed. An efficient centrifugal gravity separator was employed to remove sulfur and ash for Xin-wen –0.5mm high sulphur coal experiment, the partition curve of centrifugal separator was obtained with conclusion that the Ecart probable error value of the separator is 0.095, the imperfection I value is 0.128. Finally, comparing with the coal flotation using progressive release method, the result shows that the effect of the desulfurization with centrifugal gravity force field based on density difference between coal and minerals are much better than that of the floatation which operates based on the difference in surface properties. It provides a new technical way for the desulfurization of high sulfur fine coal.
**Keywords**: centrifugal separator; fine coal; efficiency; desulfurization; deash; flotation; partition curve

It is important to remove sulfur and ash from fine coal for the coal industry. Although the froth flotation can achieve good results for –0.5mm particle size in some circumstances, especially with the application of new technology of flotation column which improved flotation efficiency, flotation has its own defects. It has a narrow separation size range and requires flotation agents with a high separation cost. It may cause environmental pollution; It is not effective for separating incompletely liberated particles, resulting in a loss of fine middling. In practice, the results of coal release flotation are always worse than that of the float-and-sink method, particularly when the content of particles whose density is similar to that of coal and the content of sulphur is high. It is difficult to achieve a satisfactory result of deashing and desulfurization by forth flotation. This is because flotation is a separation process based on the difference in surface properties of coal and minerals. It has been found that the mineral particle containing only five percent of hydrophobic surface can float easily during forth flotation. The coal middling and slightly hydrophobic pyrite are the source of poor flotation selectivity (Tao et al., 2006; Tao, et al., 2007).

The work by Perry and Aplan and so on showed that the gravity separation method according to the density difference of materials is the most effective method to clean fine coal with the large quantity of middling and pyrite. This is because coal's ash content is directly related to its density and the density of pyrite is 2.7g/cm$^3$ or above, which is significantly higher than that of clean coal. However, due to small inertia and low sedimentation rate of superfine particles, it is impossible to obtain good results by conventional gravity separation process. The use of centrifugal force is expected to improve gravity separation of materiel with different densities. Previous research results show that the enhanced gravity separator can effectively separate metal minerals and other mineral particles (Majumder et al., 2006; Liu et al., 2006). In addition, enhanced gravity separator, such as Falcon separator and Knelson separator, were also used for the beneficiation of fine coal, separation results showed that the desulfurization

 
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efficiency of inorganic sulfur was better than that of flotation (Honaker et al., 2005; Uslu et al., 2012; Ibrahim et al., 2014; Boylu et al., 2013; Zhang et al., 2011; Honaker et al., 1998).

**1 Separation equipment and working principle**

The coal separation tests were performed using the SB40 type Falcon separator. A 20L container was used for pulp conditioning and the feed flow rate was controlled by a peristaltic pump. Operating parameters examined include feed rate, centrifugal force, hydraulic pressure, and feed solids content.

All the present experiments are conducted with the Falcon concentrator made in Canada. Falcon centrifugal separator is considered as one of the most efficient fine coal gravity concentrator for fine coal (Figure. 1). It mainly consists of a center feed pipe, a revolving impeller at high speed, and a bowl shaped separation rotor. The material is fed through the center feed pipe to the revolving impeller, then thrown to the wall of separation rotor. The speeds of both impeller and rotor are quite high, resulting in a centrifugal force of up to 300g.The material deposits and stratifies on the wall of separation rotor and move up under the centrifugal force. There is a vertical pipe on the top of the separation rotor. The light products are expelled by overflowing, and the heavy products are expelled from the rejection hole in the lower part of the vertical pipe.

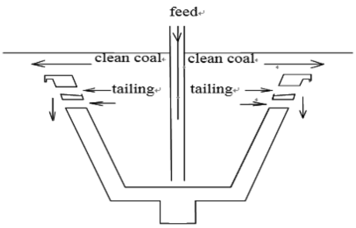



Figure 1 Structure of Falcon separator


Figure 2 Schematic diagram of separation system

Table 1 -0.5mm size fraction raw coal size distribution

| Size/mm | yield/% | Ash/% | Sulfur/% | Σyield/% | Ash/% | Sulfur/% |
|---|---|---|---|---|---|---|
| +0.25 | 18.42 | 41.87 | 5.35 | 18.42 | 41.87 | 5.35 |
| 0.25~0.12 | 26.44 | 40.22 | 4.25 | 44.86 | 40.90 | 4.70 |
| 0.125~0.074 | 13.27 | 39.29 | 4.32 | 58.13 | 40.53 | 4.62 |
| -0.074 | 41.87 | 26.95 | 3.82 | 100.00 | 34.84 | 4.28 |
| Total | 100.00 | 34.84 | 4.28 | | | |

**2 Separation tests**

The –0.5mm coal sample was obtained by screening the –3mm raw coal from Wennan mine of the Xinwen coal industry group. The results of size analysis and float-and sink experiment are shown in Table 1 and Table 2, respectively.

It can be seen from Table 1, the average sulphur content of raw coal is quite high, i.e., 4.28% of which 1.21% is organic sulphur. The ash content is 34.84%, so it is a high ash and high sulphur coal. It is interesting that the –0.074mm slime accounts for 41.87% with an ash content of 26.95% and a sulphur content of 3.82%, which are lower than the average of the feed. The ash content and the sulphur content of the +0.25mm raw coal amount to 41.87% and 5.35% respectively, which are the highest. These data indicate that the ash and sulphur liberation is not good. So it is difficult to effectively remove sulphur by means of conventional forth floatation.

Table 2 –0.5mm raw coal density distribution

| Density/ | yield/% | Ash/% | Sulfur/% | Σyield/% | Ash/% | Sulfur/% |
|---|---|---|---|---|---|---|
| -1.3 | 37.72 | 1.86 | 2.37 | 37.72 | 1.86 | 2.37 |
| 1.3-1.4 | 15.30 | 6.23 | 2.28 | 53.03 | 3.12 | 2.35 |
| 1.4-1.5 | 4.98 | 13.31 | 2.56 | 58.01 | 4.00 | 2.37 |
| 1.5-1.6 | 2.31 | 23.35 | 3.54 | 60.32 | 4.74 | 2.41 |
| 1.6-1.8 | 2.49 | 43.33 | 6.24 | 62.81 | 6.27 | 2.56 |
| +1.8 | 37.19 | 83.10 | 7.19 | 100.00 | 34.84 | 4.28 |
| Total | 100.00 | 34.84 | 4.28 | | | |

The float-and-sink results show that the yield of the +1.8 g/cm$^3$ fraction amounts to 37.19%, with the ash content and the sulphur content being 83.10% and 7.19%, respectively. The yield of the middle density fraction between 1.4 g/cm$^3$ and 1.8 g/cm$^3$ is only 9.78%, indicating that this coal may be readily cleaned by gravity separation. Therefore, it is advantageous to use the Falcon centrifugal separator for de-ashing and desulfurization.

**3 Results and discussion**

**3.1 Performance of Falcon separator**

The partition curve of Falcon centrifugal separator shown in Figure 3 is obtained according to the feed and products from the float-and-sink analysis of Wennan's –0.5mm high sulphur coal. From the partition curve , we can determine that the Ecart probable error of the separator is 0.095, the imperfection is 0.128, and the separation density is 1.74kg/l. These data show that the separation precision of Falcon centrifugal separator is better than that of the jig used for lump coal. For –0.5mm fine particle size coal, the precision of this separator is very good.

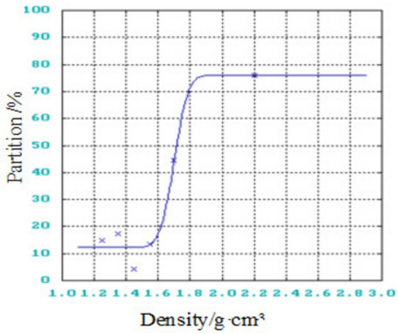



Figure 3 The partition curve of Falcon test

**3.2 Desulfurization and de-ashing results by Falcon separator**

To study the desulfurization and de-ashing performance for –0.5mm particle size coal by the Falcon separator, the release analysis tests of flotation were carried out and the results are shown in Table 3. The Falcon centrifugal separation results are shown in Table 4, which can be compared with the release analysis results (Figure .4).

Table 3 Release analysis results

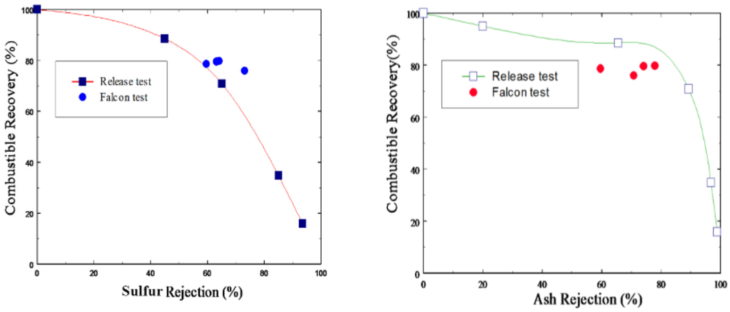
| | Products | | | Cumulative Clean | | | Cumulative reject | | |
|---|---|---|---|---|---|---|---|---|---|
| | y/% | Ad/% | St/% | y/% | Ad/% | St/% | y/% | Ad/% | St/% |
| 1 | 11.25 | 3.57 | 2.47 | 11.25 | 3.57 | 2.47 | 100.00 | 34.84 | 4.28 |
| 2 | 13.60 | 5.14 | 2.68 | 24.85 | 4.43 | 2.58 | 88.75 | 38.81 | 4.51 |
| 3 | 27.27 | 9.68 | 3.14 | 52.12 | 7.18 | 2.87 | 75.15 | 44.90 | 4.84 |
| 4 | 20.37 | 40.64 | 4.24 | 72.49 | 16.58 | 3.26 | 47.88 | 64.96 | 5.82 |
| 5 | 27.51 | 82.96 | 6.99 | 100.00 | 34.84 | 4.28 | 27.51 | 82.96 | 6.99 |
| Total | 100.00 | 34.84 | 4.28 | | | | | | |

The release analysis results show that when the ash content of the clean coal is 14% and the sulphur content is 3.28%, the yield is 56.53%; when the ash content of the clean coal is 21% and the sulphur is 3.63%, the productivity is 67.92%.

Comparison of the test results show that at the same circumstance of the combustible recovery, the total sulphur of the clean coal by the Falcon test is lower than that by release analysis test. In addition, the test results are very consistent. The desulfurization results for fine raw coal by Falcon centrifugal gravity separation method is better than that of the forth flotation even under an ideal condition. This indicates that the results of the total desulfurization with intensified centrifugal gravity separator based on density difference between coal and minerals are much better than that of the floatation which operates based on the difference in surface properties; However, the Falcon separator is not as good in de-ashing as the flotation, because the –0.5mm feed contains a high slime which may overflow into the clean coal of the Falcon separator.

Table 4 Falcon centrifugal gravity separation results

| Number | Feed | | | Clean Coal | | | Reject | | |
|---|---|---|---|---|---|---|---|---|---|
| | y/% | Ad/% | St/% | y/% | Ad/% | St/% | y/% | Ad/% | St/% |
| 1 | 100 | 41.67 | 4.72 | 56.45 | 21.54 | 2.25 | 42.55 | 67.78 | 7.93 |
| 2 | 100 | 37.47 | 3.76 | 58.14 | 14.22 | 2.32 | 41.86 | 69.77 | 5.75 |
| 3 | 100 | 33.36 | 3.80 | 61.61 | 13.99 | 2.27 | 38.39 | 64.44 | 6.27 |
| 4 | 100 | 33.20 | 3.56 | 65.89 | 20.33 | 2.18 | 34.11 | 58.07 | 6.22 |




(a) Combustible recovery vs sulfur rejection (b) Combustible recovery vs ash rejection

Figure 4 Results comparison of Falcon test with release test

**4 Conclusions**

(1) Because Falcon centrifugal gravity separator enhances fine particles separation results by using centrifugal force, its performance is better than regular floatation results. It is particularly useful when dealing with poorly liberated and high sulfur coal. However the de-ashing results of Falcon separator are not as good as flotation when feed contains a high content of slime. Use of Cyclowash superfine coal cyclone will remove slime prior to gravity separation, improving Falcon separator performance.

(2) Flotation column is able to produce super clean coal from super fine feed due to its high selectivity. So when the requirement for the ash content of fine particle products is strict, one may consider to combine the Falcon separator desulfurization with floatation column to enhance both ash and sulfur rejection. The work in this area is under way.

(3) The test sample was from Wennan mine of Xinwen coal industry group. The total sulphur content is 4.28%, and organic sulphur content is 1.21%. From the release analysis results and Falcon separation tests, we can see that the total sulphur content of the clean coal is about 3.20% from the release test, while it is about 2.20% in Falcon separation test for the clean coal with an ash content of about 16%. The centrifugal gravity separation method is better for pyritic sulfur removal than flotation. However, it is a physical separation method and can't remove organic sulfur.

In conclusion, Falcon concentrator is a new effective gravity separation technology for fine coal desulfurization. Its desulfurization results are much better than that of the regular forth floatation. Its separation cost is lower than that of floatation, too. This technology provides a better alternative in flotation to remove sulfur from fine coal, which is very important for the coal industry.

**Acknowledgements**

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# Process Features of Panji Coal Preparation Plant

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**ABSTRACT**
Panji coking coal preparation plant is currently being under construction and located in the more economically developed eastern region of China. It is a super-large mine site coking coal preparation plant with an annual production of 12.0 million metric tons per year (2272.7 metric tons per hour). The raw coal processed by Panji plant is characterized by significantly difficult washability and severe degradation effect. Panji coal preparation plant applies advanced and reliable separation circuit and equipment with China's independent intellectual property rights such as gravity-fed non-deslimed non-classified three product heavy medium cyclone, fine coal heavy medium cyclone, jet flotation machine, inclined pipe thickener, and etc. This paper introduces major characteristics of Panji coal preparation plant in detail in the following aspects: application of the world's largest gravity-fed three-product H.M. cyclone; fine coal H.M. cyclone without the preparation&storage system for ultrafine magnetite; application of deslimed rougher flotation-cleaner flotation process; and deeply clarified circulating water.

**Keywords**
Three-Product H.M. Cyclone, Fine Coal H.M. Cyclone, Deslimed Rougher Flotation-Cleaner Flotation, Deeply Clarified Circulating Water, Super Large, Coking Coal Preparation Plant, Process Feature.

## 1 Introduction

Panji coal preparation plant, built by Huainan Mining Industry (Group) Co., Ltd in 2015, is a super large coal preparation plant with a production of 12 MTPA (2272.2 TPH). Based on the experience of another two large plants constructed in recent years, Panji coal preparation plant applied advanced circuit (shown in Figure 1) and equipment with independent intellectual property rights of China to process difficult-to-wash and brittle Huainan coal.

The raw coal is fed to the gravity-fed three-product cyclone without being deslimed and classified. The circuit majorly includes the following three sections: three-product H.M. cyclone and fine coal H.M. cyclone; Flotation; Thickning.

This paper introduces the following four features of Panji coal preparation plant.

## 2 Application of gravity-fed three product dense medium cyclone

Panji coal preparation plant is comprised of four production systems. In each system, a 3GHMC1500/1100 three-product cyclone with the world's largest dimension and capacity is employed (shown in Figure 2). It consists of a first-stage cylindrical unit with an inner diameter of 1500mm and a second-stage cylindro-conical unit, which are connected in series. Total 41 units have been employed in 34 coal preparation plants in China with a total annual capacity of 123 million tonnes. Table 1 shows the operation performance of Panyi plant also in Huainan mine field and other four plants located in outside of Huainan mine fields.

$E_p$ value, as an important index, is used to evaluate the performance of the H.M. cyclones. $E_p$ of the first-stage cyclone in Panyi plant is 0.020kg/L while $E_p$ of the second-stage is outstanding at 0.029kg/L.

 
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0.029kg/L.

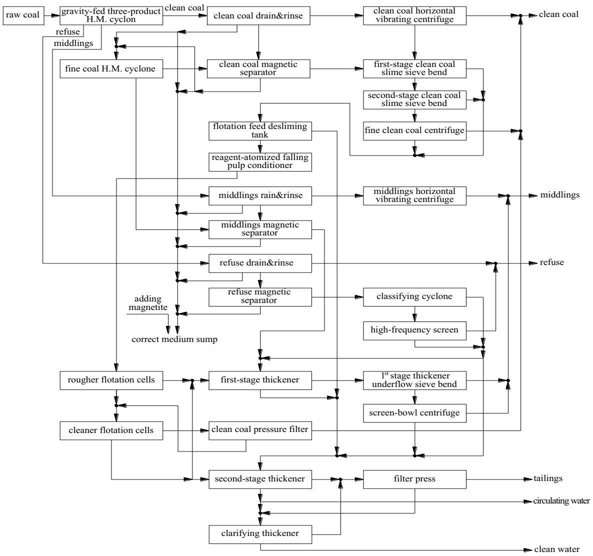



Figure 1. Process flowsheet of Panji coal prepration plant

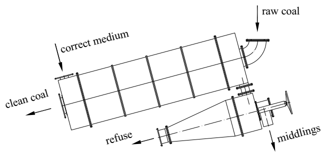



Figure 2. Schematic diagram of gravity-fed three-product heavy medium cyclone

Table 1 Operating performance of 3GHMC1500/1100 three-product HM cyclone

| Plant | Working condition | | Performance index | | | Separation density, kg/L | | |
|---|---|---|---|---|---|---|---|---|
| | Capacity t/h | Near-gravity material δ±0.1, % | $E_p$, kg/L | | Organic efficiency $\eta_e$, % | Theoretical | Actual | |
| | | | First stage | Second stage | | | First stage | Second stage |
| Panyi | 580 | 60.03 | 0.020 | 0.029 | 93.20 | 1.408 | 1.397 | 1.851 |
| Hexi | 540 | 9.69 | 0.023 | 0.030 | 99.12 | 1.605 | 1.585 | 1.899 |
| Mengxi | 588 | 47.06 | 0.027 | 0.036 | 91.68 | 1.419 | 1.410 | 1.678 |
| Xinyu | 615 | 13.93 | 0.022 | 0.034 | 98.47 | 1.520 | 1.505 | 1.898 |
| Gengyang | 603 | 4.98 | 0.033 | 0.020 | 99.34 | 1.648 | 1.630 | 2.018 |
| Average | 585 | | 0.025 | 0.030 | | | | |

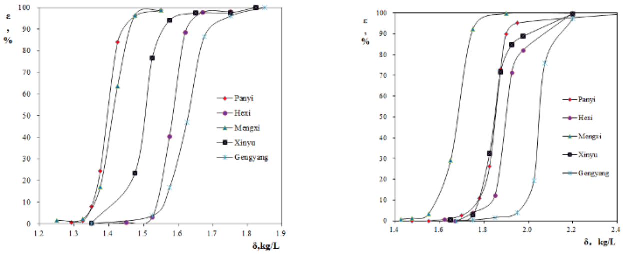
Partition curves of the H.M. cyclones in the five coal preparation plants are shown in figures 3(a) and (b). The steeper the curve is, the lower the $E_p$ value will be. As the partition curves in figure 3 are nearly vertical, an excellent separation accuracy was achieved with these H.M. cyclones.

The theoretical separation density of the first-stage cyclone in Panyi plant is 1.408 kg/L while the actual separation density is 1.397 kg/L. The negligible differential between them is directly resulted from low $E_p$. The second stage is deshaling at a remarkably high actual separation density of 1.851kg/L, which produces pure reject at an ash content as high as 84.90%.

From near-gravity material content $\delta \pm 0.1$ and organic efficiency $\eta_e$ in Table 1, the following regression model was derived:

$$\eta_e = 100.16 - 0.14(\delta \pm 0.1) \quad (1)$$

The reliability (coefficient of determination) of this model was calculated to be 95%. Based on this model, the separation performance (organic efficiency) can be effectively predicted for raw coal feeds with different washabilities. It indicated that every 1% decrease in $\delta \pm 0.1$ will lead to 0.14% increase in organic efficiency.




(a) first-stage partition curve (b) second-stage partition curve

Figure 3 Partition curves of 5 coal preparation plants

**3 Application of fine coal H.M. cyclone**

Panji coal preparation plant applied patented fine coal H.M. separation circuit. The most important characteristics of the patent is to ingeniously utilize the cyclone under centrifugal force. Finer magnetite particles always automatically report to the clean coal stream. Both of the fine coal and finer fraction magnetite, accumulated in the underflow of sieve bend of clean coal side of 3-product cyclone, are fed to the fine coal H.M. cyclone for further processing. A portion of concentrate from clean coal magnetic separator can be added to the fine coal H.M. cyclone when needed. The light and heavy products of the fine coal cyclone are reported to the clean coal & middlings magnetic separators separately. The circuit was thus significantly simplified by eliminating the preparation and recovery systems of ultrafine heavy medium for fine coal H.M. cyclone.

Industrial optimization tests were conducted in Xinzhuangzi coal preparation plant, a member of Huainan Mining Group to optimize the operating parameters (feeding pressure, heavy medium density, magnetite grade) of the fine coal H.M. cyclone. Test results are shown in Table 2.

Table 2. Test results of the separation of 0.25-0.1mm coal with FHMC350 fine coal H.M. cyclone

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Optimized parameters | Pressure, MPa | 0.213 | Capacity $m^3/h$ | 133 | Performance index | $E_p$ | 0.081 |
| | Medium density, kg/L | 1.16 | | | | | |
| | Magnetite grade %<325 mesh | ≥79 | Actual separation density, kg/L | 1.455 | | Organic efficiency $\eta_e$, % | 90.0 |

The density distribution and partition number of the feed and products of the fine coal H.M. cyclone are shown in Table 3. The partition curve is shown in Figure 4 based on the data in Table 3.

As shown in Figure 4, $E_p$=0.081kg/L when processing 0.25～0.1mm coal, which indicates that the FHMC fine coal H.M. cyclone can achieve a lower size limit as low as 0.1mm. As a result, the top size limit of the downstream flotation process will be lowered and so will be the operating cost.

Additionally, the feed to the fine coal H.M. cyclone mainly comes from a part of the underflow of the clean coal sieve bend, which will reduce the slime content in the recirculating correct medium to the three-product cyclone and thus facilitate the de-H.M. process and magnetite recovery, and make it possible to keep the magnetite loss at 1.0kg/tonne raw coal.

Table 3. Density distribution and partition number of the feed and products of the fine coal H.M. cyclone (standard deviation 0.623)

| Density kg/L | Feed, % | | Clean, % | | | Reject, % | | | Calculated feed, % | | Partition number % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Yield | Ash | Yield | | Ash | Yield | | Ash | Yield | Ash | |
| | | | % of Stream | % of Feed | | % of Stream | % of Feed | | | | |
| ＜1.3 | 37.80 | 3.46 | 47.40 | 36.47 | 3.27 | 8.37 | 1.93 | 4.54 | 38.40 | 3.33 | 5.03 |
| 1.3-1.4 | 38.66 | 9.91 | 39.70 | 30.55 | 9.23 | 30.33 | 6.99 | 10.56 | 37.54 | 9.48 | 18.62 |
| 1.4-1.5 | 15.99 | 18.83 | 11.16 | 8.58 | 18.29 | 34.06 | 7.85 | 18.34 | 16.44 | 18.31 | 47.77 |
| 1.5-1.6 | 4.45 | 28.00 | 1.44 | 1.11 | 27.75 | 15.81 | 3.64 | 27.65 | 4.75 | 27.67 | 76.73 |
| 1.6-1.8 | 2.42 | 38.29 | 0.31 | 0.24 | 36.23 | 8.70 | 2.01 | 38.69 | 2.25 | 38.43 | 89.30 |
| ＞1.8 | 0.69 | 62.68 | 0.00 | 0.00 | - | 2.74 | 0.63 | 62.36 | 0.63 | 62.36 | 100.00 |
| 计 | 100.00 | 10.75 | 100.00 | 76.95 | 7.77 | 100.00 | 23.05 | 19.27 | 100.00 | 10.42 | |

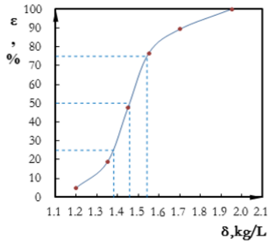



Figure 4. Partition curve on the separation of 0.25~0.1mm coal with fine coal H.M. cyclone

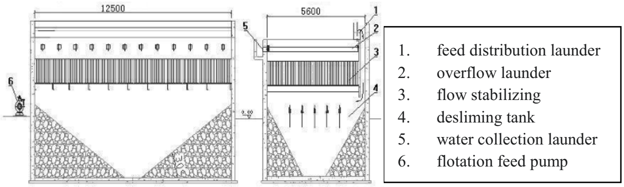



Figure 5. Schematic diagram of flotation feed desliming tank structure cyclone

## 4 Flotation feed desliming and cleaning flotation

### 4.1 Slime size distribution

As shown in the representative size distribution of the original fines in Table 4, the ash content of the <0.03mm (500 mesh) fraction increased rapidly to 57.45% with a yield of nearly 20%. Correspondingly, the <0.03mm fines take up to 2/3 of the total fines reporting to the flotation system due to the friable characteristics of the coal. The overall ash content of the fines reporting to flotation system is thus increased to 42.76%. As a result, producing high quality flotation clean coal has become unrealistic.

Table 4. Size distribution of original fines in the feed and flotation fines

| Size, mm | | ＞0.50 | 0.5～0.25 | 0.25～0.125 | 0.125～0.075 | 0.075～0.045 | 0.045～0.030 | ＜0.030 | Total |
|---|---|---|---|---|---|---|---|---|---|
| Primary fines | Yield, % | 0.07 | 26.86 | 37.62 | 8.45 | 6.51 | 1.18 | 19.31 | 100.00 |
| | Ash, % | 23.42 | 23.70 | 27.66 | 30.81 | 34.13 | 33.03 | 57.45 | 33.10 |
| Flotation fines | Yield, % | 0.02 | 2.60 | 9.67 | 13.15 | 8.65 | 0.73 | 65.18 | 100.00 |
| | Ash, % | 9.76 | 8.72 | 9.74 | 11.79 | 18.51 | 28.64 | 58.65 | 42.76 |

### 4.2 Flotation feed desliming

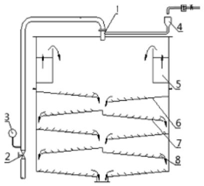
Due to large amount of slime in the flotation feed, a patented flotation feed desliming tank (China patent number FFDP70, Figure 5) is employed in each production system in Panji coal preparation plant to remove the ultrafine clay. The flotation feed desliming tank is structured with reinforced concrete. Its role is to control the flowrate besides desliming the flotation feed.

### 4.3 Cleaning flotation

The jet flotation cell combined with patented reagent-atomized falling pulp conditioner has been successfully commissioned in other coal preparation plants in Huainan Mining Group. Scale-up larger equipment is applied on the basis of previous successful experience.

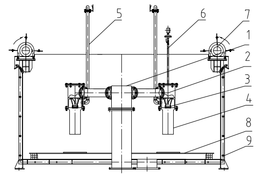
Each rougher flotation cell in Panji coal preparation plant was equipped with one FCA2800 reagent-atomized falling pulp conditioner (shown in Figure 6). The reagent-atomized falling pulp conditioner is equipped with atomizing jet spray nozzles to improve the dispersion effect of chemical reagent. Particles with different sizes move along the sliding plates at different velocity which makes longer contact time and more uniform attachment of particles with reagents as a function of particle size. Fines with high ash are discharged immediately while coarse particles have sufficient time to form a stable oil film. Thus, the consumption of chemical reagent will be reduced and the selectivity of the flotation will be improved.

FJCA36 jet-type coal flotation machines (shown in Figure 7) were used in both rougher and cleaner flotation system in Panji coal preparation plant. The FJCA jet flotation machine has advantages such as simple structure & long service life with good selectivity, high capacity, less consumption of flotation agents, low power consumption, and etc.



1-air-drive jet spray；2-valve；3-pressure gauge；4-reagent addition hopper；5-overflow cell；6-upper sliding plate；7-lower sliding plate；8-barrier

Figure 6. Spray cascading pulp conditioner



1-distribution chamber；2-jet cell；3-nozzle；4-throat tube；5-aspirating tube；6-pressure gauge；7-froth scraper；8-false bottom；9-cell body

Figure 7. Schematic diagram of the FJCA jet flotation cell

The deslimed rougher flotation combined with cleaner flotation process was first time applied in Panyi coal preparation plant in 2013, Table 5 shows the typical test results from numbers of tests.

Table 5. Test results of the evaluation of flotation system in Panyi coal preparation plant

| | 0.5-0mm | <0.03mm |
|---|---|---|
| Ash content of the feed to desliming tank, % | 37.73 | 53.55 |
| Finial clean coal(filter cake)ash content, % | 10.66 | 12.66 |
| Tailings ash content, % | 66.41 | 72.41 |
| Final clean coal yield, % of fraction | 51.57 | 35.04 |
| Final clean coal combustible recovery, % | 73.30 | 65.88 |
| Final clean coal non-combustible recovery, % | 14.53 | 8.28 |
| Flotation perfect index, % | 58.77 | 57.60 |

As shown in Table 5:

① The clean coal ash content can reach 10.66% with the flotation perfect index as high as 58.77% by using desliming tank feed with 61.61% slime content (<0.03mm) and 37.73% ash content.

② The ash content of the <0.030mm slime was as high as 53.55%. Over 1/3 of this fraction was recovered as high quality product with the flotation perfect index up to 57.60%.

**5 circulating water deep clarification**

Inorganic electrolytes coagulants combined with high polymer flocculants has been used in the $2^{nd}$-stage thickener in Panji coal preparation plant. In addition, a clarifying thickener was innovatively employed in Panji plant to deeply clarify filtrate of the filter press & a portion of overflow from the $2^{nd}$-stage thickener by adding alum solution. By using clarifying thickener the finest particles with the highest ash content will be effectively removed and clarified water can be reused at where it’s needed such as dilute water for cleaner flotation cells.

Table 6 shows the performance index of the thickeners in Panyi coal preparation plant. Deep clarification of the circulating water was achieved as indicated by a very low solids concentration in the clarifying thickener overflow. Since Panji coal preparation plant is only 10km from Panyi coal preparation plant and has similar geological conditions, it is estimated that equivalent performance regarding circulating water deep clarification and washing water closed circuit should be achievable, which will eliminate the discharge of any plant effluent to nearby rivers. Moreover, less water will be used as the water consumption is only $0.055m^3$/tonne raw coal.

Table 6. Performance index of thickeners in Panyi coal preparation plant

| | | Secondary thickener | Clarifying thickener |
|---|---|---|---|
| Feed solids concentration, g/L | | 22 | 0.44 |
| Overflow solids concentration, g/L | | 0.35 | 0.045 |
| Underflow | Solids concentration, g/L | 267 | 203.0 |
| | Solids recovery, % | 98.52 | 87.6 |

## 6 Conclusion

Process technologies used in Panji coal preparation plant are introduced in this paper. It shows advancement of China's coal processing industry from enlargement & quantity oriented to technology & quality oriented. The technology and equipment employed in Panji plant are endowed with China proprietary intellectual property rights and have been tested & proved in industrial practice.

It is without a doubt that in 2017, a modernized super-large coking coal preparation plant with its own characteristics will be successfully constructed, commissioned and operated in the eastern area of China.

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# Application of 3-Product Heavy Medium Cyclone in the Retrofit Project at Umlalazi Coal Preparation Plant in South Africa

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**Abstract:** In order to solve the production of refuse with higher calorific value using 2-product heavy medium cyclone in Umlalazi Coal Preparation Plant, South Africa, 3-product heavy medium cyclone was utilized in the retrofit and the separation performance was discussed in this paper. The results demonstrate that 3-product heavy medium cyclone has good separation performance, with organic efficiency 98.0% and Ep(écart probable moyen) 0.0199 $g{\cdot}cm^{-3}$ in 1st stage and organic efficiency 91.8% and Ep 0.0417 $g{\cdot}cm^{-3}$ in 2nd stage; Compared with clean coal with calorific value 27.5MJ/kg and refuse with calorific value 15MJ/kg in the original design, there are production of clean coal, middlings and refuse in retrofit with calorific value 27.5 MJ/kg, 21 MJ/kg and 11 MJ/kg. The treatment of refuse becomes easier as it is justified to be abandoned in terms of calorific value. Additionally, quality of magnetite powder, liquid-solid ratio and slime content in suspension should be given a priority consideration in the adjustment of 3-product Heavy Medium cyclone to ensure the separation performance.

**Key words:** South Africa; Umlalazi; Coal Preparation Plant; 2-product; 3-product; Heavy Medium Cyclone; retrofit; middlings; calorific value

## 1 Introduction

According to BP Statistical Review of World Energy 2013, South Africa is rich in coal, with proved coal reserves 30.1 billion which accounts for 3.5% of total world resources and reserve/production ratio (R/P) 116 years by the end of 2012. In South Africa, coal consumption accounts for more than 70% in primary energy consumption structure, and there are approximately 60 coal preparation plants and separation proportion of raw coal is about 85%, and the main separation technique is adapted that heavy medium separators separate lump coal and fine coal is separated using spiral.

Umlalazi Coal Preparation Plant with handling capacity 200t/h was originally designed to produce steam coal by adapting 2-product heavy medium cyclone. Initially, 2-product cyclone produced clean coal with desired calorific value 27.5 MJ/kg and refuse with calorific value 15 MJ/kg accordingly. However, refuse with 15MJ/kg seems to have no attraction for user, and it also will be a waste to discard them. In order to solve this problem, five 3-product HM cyclones with pump feeding manufactured by Weihai Haiwang Hydrocyclone Co., Ltd were adapted to treat lump and/or coarse coal (50-1mm) for retrofitting. This paper discusses application of 3-product HM cyclone for retrofitting in Umlalazi coal preparation plant in brief.

## 2 Principle of 3-product HM cyclone with pump feeding

Although 3-product heavy medium cyclone wasn't originated in China, it has been being widely used as a main separator through continuous renovation. Till now, there are many problems and it still has attraction to many researchers. Figure 1 and figure 2 are structure principle sketch of 2-product HM cyclone with pump feeding in original design and 3-product HM cyclone with pump feeding in retrofit in Umlalazi coal preparation plant, respectively.



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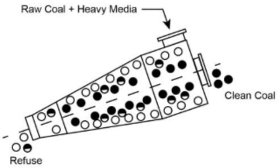


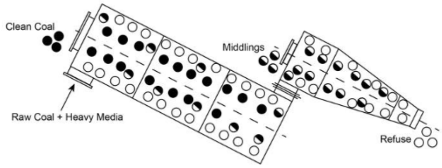




Figure 1 2-product HM cyclone with pump feeding Figure 2 3-product HM cyclone with pump feeding

Taking 2-product HM cyclone as an example, mixture of raw coal and heavy suspension was fed via a pump into tangential entry under pressure which imparts a swirling motion to the pulp, generating a powerful rotation flow which was divided into an outer region of downward flow, moving towards underflow along the wall, and an inner region of upward flow, moving towards overflow along the axis. Under centrifugal force and acting drag, particles are separated into low density fraction and high density fraction based on their density, that's, high density particles will be discharged through underflow in an outer region as tailings while low density particles will be discharged through overflow in an inner region as concentrates. In some sense, 3-product HM cyclone could be simply regarded as two 2-product HM cyclones in series, and has the same separation principle.

It could be seen easily that from above figure 1 and figure 2, 2-product HM cyclone separates coal into two fractions, that's, clean coal and refuse, in other words, middlings goes either into clean coal or refuse. Under the circumstance, refuse will have comparatively low ash content when clean coal has low ash content which meets user's requirement, which leads to misplace and a waste of coal. 3-product HM cyclone produces three products in a suspension system, clean coal of low ash content, refuse of high ash content and salable middlings of certain calorific value, which provides a way to solve the refuse with higher calorific value in Umlalazi coal preparation plant. Supposing 2-product heavy medium is used to produce three products, two suspension systems are required, which will lead to complicated process and high operation and maintenance cost.

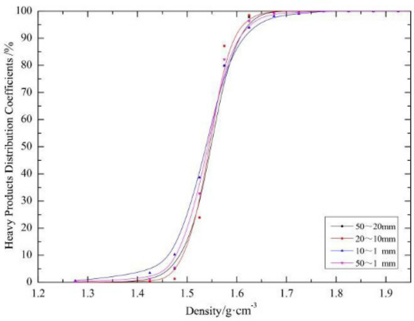
**3 Separation performance of 3-product HM cyclone**

After installation of 3-product HM cyclone in Umlalazi coal preparation plant, commissioning was carried out by adjusting many parameters, such as, quality of heavy medium, liquid-solid ratio, slime content in suspension and so on, to produce qualified clean coal continuously and steadily. Separation performance analysis, including ash of feed, concentrates and tailing, yield, cut density, probable error (Ep), organic efficiency and near-density content and so on, of 3-product HM cyclone with pump feeding was carried out in April 3$^{rd}$, 2013 based on different size fractions samples, 50-20, 20-10, 10-1 and 50-1mm, respectively. Consider the length, this paper simply gives the separation performance results of the different size fractions samples, neglecting the calculating progress, shown in table 1 and 2, and figure 3 and 4 are partition curves of 1$^{st}$ and 2$^{nd}$ stage of 3-product HM cyclone, accordingly. It should be pointed out that underflow of primary cyclone is regarded as feeding of the secondary one, and the ash is calculated based on mass balance equation and the yield is obtained using Grignard's method based on principle of least square method.

Table 1 Separation performance results of different size fractions in 1$^{st}$ cyclone

| Size fraction, mm | 50～20 | 20～10 | 10～1 | 50～1 |
|---|---|---|---|---|
| Ash of feed ,% | 33.6 | 26.3 | 26.6 | 29.0 |

| | | | | |
|---|---|---|---|---|
| Ash of overflow ,% | 11.5 | 11.1 | 10.7 | 11.1 |
| Ash of underflow,% | 57.1 | 45.3 | 44.0 | 48.9 |
| Yield of overflow ,% | 51.41 | 55.42 | 52.40 | 52.69 |
| Sp gr of separation | 1.545 | 1.540 | 1.536 | 1.538 |
| Probable Error,sp gr | 0.0181 | 0.0136 | 0.0254 | 0.0199 |
| Organic efficiency,% | 99.5 | 99.2 | 96.3 | 98.0 |
| δ±0.1,% of feed | 17.8 | 30.5 | 36.1 | 28.7 |



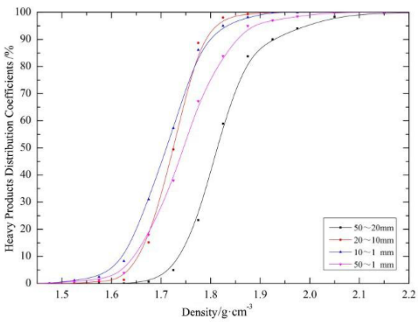




Figure3 Partition curves of different particle sizes in 1st cyclone Figure 4 Partition curves of different particle size in 2nd cyclone

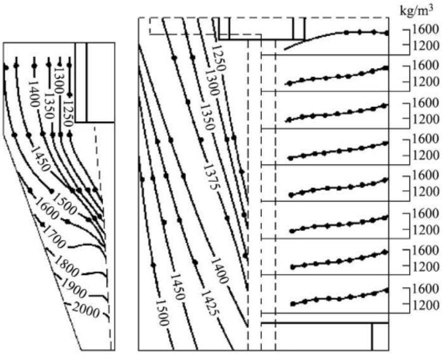
Table 2 Separation performance results of different particle size fractions in 2nd cyclone

| Size fraction, mm | 50～20 | 20～10 | 10～1 | 50～1 |
|---|---|---|---|---|
| Ash of feed, % | 57.1 | 45.3 | 44.0 | 48.9 |
| Ash of overflow, % | 33.1 | 29.0 | 26.3 | 29.1 |
| Ash of underflow, % | 74.5 | 60.4 | 60.8 | 65.9 |
| Yield of overflow, % | 42.04 | 48.24 | 48.63 | 45.94 |
| Sp gr of separation | 1.816 | 1.726 | 1.701 | 1.747 |
| Probable Error,sp gr | 0.0324 | 0.0203 | 0.0421 | 0.0417 |
| Organic efficiency,% | 99.5 | 96.6 | 91.6 | 91.8 |
| δ±0.1,% of feed | 17.7 | 24.7 | 28.4 | 20.7 |

As seen from Table 1, in primary stage, washability grade of 50~20 mm, 20~10 mm and 10~1 mm are classified to be middle-clean, hard-clean and hard-clean, respectively, while probable Error (Ep) of those fractions are 0.0181 $g\cdot cm^{-3}$, 0.0136 $g\cdot cm^{-3}$, 0.0254 $g\cdot cm^{-3}$, respectively. The overall Ep and organic efficiency in primary cyclone are 0.0199$g\cdot cm^{-3}$ and 98.0%. Similarly, as seen from table 2, in secondary cyclone, washability grade of 50~20 mm, 20~10 mm and 10~1 mm are regarded to be middle-clean, relative hard-clean and relative hard-clean, respectively, while probable Error (Ep) of those fractions are 0.0324 $g\cdot cm^{-3}$, 0.0203 $g\cdot cm^{-3}$, and 0.0421 $g\cdot cm^{-3}$, respectively. The overall Ep and organic efficiency for secondary stage are 0.0417 $g\cdot cm^{-3}$ and 91.8%. As discussed above, separation performance of the

3-Product HM cyclone has achieved good performance ($Ep_1 < 0.03$ g·cm$^{-3}$, $Ep_2 < 0.05$ g·cm$^{-3}$) and met the requirements of plant.

Compared with original 2-product HM cyclone, 3-product HM cyclone produces clean coal more stably. Just as shown in figure 5 that demonstrates the distribution of density field when feed pressure is 0.1MPa and suspension density is 1400 kg·m$^{-3}$, cylindrical cyclone, i.e. 1$^{st}$ stage of the 3-product HM cyclone, has more uniform density distribution than cylindrical-conical cyclone, that's, density gradient is smaller and suspension density of each section axially is almost the same in cylindrical cyclone, leading more stable separation process occurring in cylindrical cyclone.




(a) (b)

Note: (a) Cylindrical-conical cyclone, (b)-Cylindrical cyclone

Figure5 Comparison of internal density field between cylindrical-conical cyclone and cylindrical cyclone

After retrofit in Umlalazi, 3-product HM cyclone produces clean coal with calorific value 27.5 MJ/kg, middlings with 21MJ/kg and refuse with 11MJ/kg. With refuse discarded and middlings salable, it brings maximum benefit to Umlalazi.

**4 Some suggestions in 3-product HM cyclone adjustment**

It could be seen easily that some parameters, quality of magnetic powder, liquid-solid ratio and slime content in suspension, play important roles in the adjustment of 3-product HM cyclone.

**1）Quality of magnetite powder**

The density of medium solid should meet requirement of coal separation density with HM cyclone, which keeps suspension solid concentration by volume in the range 20~30%. Research shows that viscosity of the suspension decreases with an increase in density of medium solid. Additionally, separation density, medium recovery and other factors should also be taken into consideration when choosing proper medium solid. Usually, magnetite powder is chosen as medium solid in coal preparation .

Suspension's stability could be expressed using settling velocity of magnetite particle in gravitational field. The same magnetite particle falls much faster in centrifugal field than in gravitational field. Therefore, magnetite powder should be fine enough to ensure the suspension stable in cyclone. Quality of magnetite powder in Umlalazi is shown in table 3.

Table 3 Magnetite powder's quality

| Content of -200 mesh % | Content of -325 mesh % | Magnetic content % |
|---|---|---|
| 97.3% | 76.3% | 95.6% |

**2）Liquid-solid Ratio**

Liquid-solid ratio reflects concentration of coal slurry. As we all know, particle's separation in HM cyclone also follows principle of Archimedes in centrifugal field. The higher concentration is, the worse separation efficiency becomes. Therefore, liquid-solid ratio is a key parameter to affect separation efficiency. Also, feed rate is a key factor affecting liquid-solid ratio.

Nominally, three-product HM cyclone, YTMC 900/650, in Umlalazi could treat140~200 tons coal per hour. After several trials, separation efficiency is satisfactory when actual feeding rate is 180t/h.

**3）Slime Content in the suspension**

Slime content affects volume concentration and viscosity of the suspension. If the slime content is low, the suspension is mainly comprised of magnetite powder and the suspension will not be stable. Otherwise, viscosity will be higher, which will exert negative effect on separation performance.

Slime content cannot be monitored directly online in actual activities; however, there exists some relationship between magnetic content and slime content. Therefore, slime content is calculated based on magnetic content which could be monitored online. Table 4 shows the relationship between magnetic material concentration and slime content when solid concentration by volume is 25~30%, assuming that density of magnetite powder is 4.5g·cm$^{-3}$ and that of slime is 1.5g·cm$^{-3}$ .

Table 4 Relationship between separation density and magnetic material concentration and slime content

| Separation density/g·cm$^{-3}$ | 1.50 | 1.55 | 1.60 |
|---|---|---|---|
| Magnetic material concentration/g·L$^{-1}$ | 563~525 | 638~600 | 713~675 |
| Slime content/% | 25.0~34.4 | 20.3~29.4 | 16.2~25.0 |

Good separation performance could be achieved when the magnetic material concentration in Umlalazi is maintained at 550~650 g·L$^{-1}$.

## 5 Conclusions

**1)** The separation performance analysis of 3-product heavy medium cyclone shows that YTMC 900/650 has good separation performance in Umlalazi coal preparation plant, with organic efficiency 98.0% and Ep 0.0199 g·cm$^{-3}$ in 1$^{st}$ stage and 91.8% and 0.0417 g·cm$^{-3}$ in 2$^{nd}$ stage;

**2)** Compared with refuse with calorific value 15MJ/kg in the original design, there is production of clean coal, middlings and refuse in retrofit with calorific value 27.5 MJ/kg, 21 MJ/kg and 11 MJ/kg, respectively, which will bring benefit to Umlalazi by recovering coal from the refuse with high calorific value;

**3)** Three parameters should be taken into consideration to make 3-product HM cyclone achieve good separation performance, such as quality of magnetite powder, liquid-solid ratio, and slime content in suspension.

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# Application of density monitoring system upgrading in SuanCiGou Coal Preparation Plant

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**Abstract**

Through the SuanCiGou coal preparation plant density monitoring system and separation system of research and analysis, according to the site installation environment in SuanCiGou coal preparation plant and system principle, the author proposed density monitoring system technical revamp and process management improvement measures, to improve the centralized control of the density of the separation system in automatic control. After the transformation of density monitoring system can better reflect the actual density of coal dressing system, the density automatic control system can adjust the separation density washing system in time. The process management has improved the professional attitude of the staff, and improved the production efficiency of the coal preparation plant. Through the above technological transformation and management improvement, improve the separation efficiency of the SuanCiGou coal preparation plant, it was obtained significant economic benefit by modification, which made separation efficiency improved, the product yield enhanced and the gangue yield reduced.

**Key Words:** Density Monitoring System, Density Gauge Installation Position, Automatic Density Control, Density Curve, Overflow of H.M Vessel, Technical Modification, Management Improvement

## 1. Introduction

SuanCiGou CPP is a pithead steam coal preparation plant subordinate to Inner Mongolia YiTai Coal Co., Ltd, Jing Yue SuanCiGou Mining Co., Ltd. The CPP was designed and operated by Beijing YZH Engineering Technology Co., Ltd. The CPP's design capacity is 12 Mt/a, whose production process including 200~13mm lump coal adopt heavy medium vessel separator(H.M vessel), 13~1mm slack coal adopt two-product dense-medium cyclone, and slime recovery from dewatering.

## 2. Exiting Problems

### 2.1. Density Monitoring System

1）In the beginning, SuanCiGou CPP used ray density meter, whose receiver instrument damages frequently for receiver instrument and radioactive source purchased from different manufacturers. After that, there was added a set of pressure-differential density meter in each coal preparation system. The two kinds of density meters were used together and those two would verify each other. The accuracy and stability of density measurement data from pressure-differential density meter easily influenced by the environmental conditions, such as the velocity of the fluid, gas bubbles in fluid, and etc.

2）It was difficult to adjust density in the production caused by the large fluctuations of density measurement data, which has great impact on magnetic adding and spilt flow distribution of circulating medium. Therefore, automatic controlling of density was hardly achieved and density regulation can be obtained by manual but delayed.

3）The centralized control system only show the real-time separation density, the earlier separation



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density could be recorded by manual only, which cannot be accurately evaluated and checked in daily management.

### 2. 2. Fluctuant Quality of Raw Coal

The CPP is service for SuanCiGou mine whose provide all of the CPP's raw coal feed. The main excavating coal seams are NO.4 coal seam and NO.6 coal up seam. NO.4 coal seam is low-sulfur and high-ash coal (Ad 50%). NO.6 coal up seam is low-sulfur and middle-ash coal (Ad 37%). The two seam coals cannot achieve underground loading independently, and they are transported to ground raw coal bunker by a belt conveyer. There were lots of troubles of CPP production for the large different quality and wash-ability of the two seam coal and its fluctuation and non-uniformity.

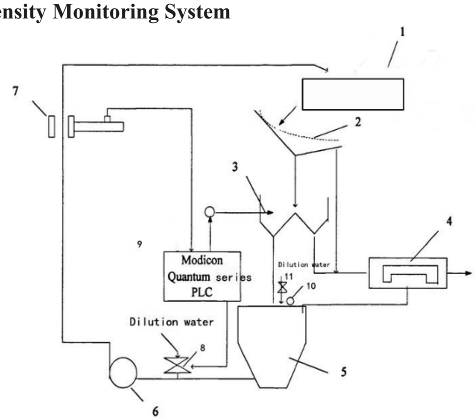
### 2. 3. Coal misplaced in Reject of H.M vessel

There are spilt flow distribution plates at the overflow weir of H.M vessel in beginning design for clean lump coal enter dewatering and medium draining screen evenly to avoid screen offset load. The plates were installed upon the correct position when construction so that clean coal from H.M vessel cannot discharge smoothly and coals stay in the vessel or discharge with rejects.

## 3. Improvement Measures

### 3.1. Density Monitoring System Improvement

#### 3.1.1. Introduction of Density Monitoring System




**1-Main separator; 2-sieve-bend; 3-flow-spitting box; 4-magnetic separator; 5-circulating medium bucket; 6-pump; 7-density meter; 8-electronic valve (water supply valve for fine adjustment); 9-electric actuator; 10-liquid monitor; 11-electronic valve (water supply valve for coarse adjustment);**

**Figure1. Brief density monitoring system diagram of SuanCiGou CPP**

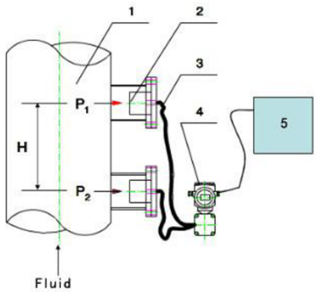
Density monitoring system of SuanCiGou CPP is seen in figure1. Coal and circulating medium enter together into main separator, then products enter dewatering and medium draining screen. A part of screen underflow circulating medium flow into its bucket, the others into flow-spitting box which can distribute them into circulating medium bucket and dilute medium bucket. Magnetic separator recycling dilute medium, and its concentrate return circulating medium bucket.

#### 3.1.2. Modification of Density Meter

1) Replacement of radioactive source density meter' receiver instrument

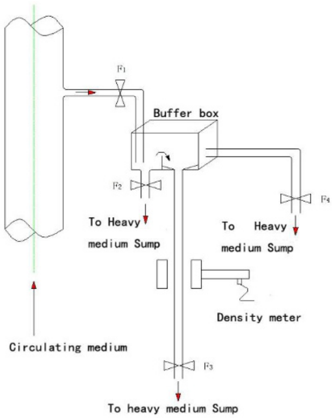
Original receiver instrument cannot work at all because it is damaged frequently, which does not match the radioactive source. SuanCiGou CPP contacts other radioactive source density meter' manufacturer and inspects the strength of on-site radioactive source and customizes receiver instrument which matches for on-site requirements.

2) Installation position replacement of radioactive source density meter




**1- pipe; 2- diaphragm seal; 3- capillary; 4- transmitter; 5-control box**

**(1) Installation diagram of pressure-differential density meter before modification**




**(2) Installation diagram of bypass radioactive source density meter after modification**

**Figure2. Installation diagram of density meter before and after modification**

Density meter is directly installed on the main pipe of circulating medium originally. Because the large diameter of main pipeline and the fast and unstable medium flow in main pipeline, it has effect on the accuracy of density detection. After technician's discussion, circulating medium can be educed and inspected by opening the hole in main pipe and installing bypass pipe.

Bypass pipe of circulating medium main pipeline is connected with buffer box, on the bottom of which installing Φ100mm pipe. There is a density meter on the pipe as well as installing manual valve on the bottom of it to control the speed of discharging, details seen in figure 2. Installing 200mm high baffle plate in the buffer box. The feed coal firstly goes into box and overpass baffle plate into another half of the buffer box and finally goes into pipe with the density meter. In addition, installing wear resisting steel plate with slope 10°on the bottom plate of buffer box, it can make medium flow in pipe more stable as well as media and lump coal hardly accumulated in the box bottom. Also, it could reduce the attack caused by the deviation and prevent pipe and buffer box from blocking. Finally, density detection became more stable and accurate.

3) The real time data of on-site test and density meter measurement after modification are shown in Table1.

**Table1**. **Densities comparison of on-site test and measured by density meter**

| **Times** | 20:00 | 21:00 | 22:00 | 23:00 | 0:00 | 1:00 | 2:00 | 3:00 | 4:00 | 5:00 | 6:00 | 7:00 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **On-site test(g/cm$^3$)** | 1.93 | 1.92 | 1.92 | 1.92 | 1.91 | 1.91 | 1.92 | 1.93 | 1.91 | 1.92 | 1.90 | 1.89 |
| **Density meter measurement(g/cm$^3$)** | 1.91 | 1.90 | 1.90 | 1.91 | 1.91 | 1.91 | 1.90 | 1.91 | 1.90 | 1.90 | 1.90 | 1.90 |

### 3.1.3. Perfecting Density Automatic Control of Separation System

Since density automatic control of separation system in SuanCiGou CPP has been further improved by cooperating with centralized control system design manufacture, density automatic control system not only online monitors in real time and adjusts the density of separation medium, and solves the problem of delayed regulation of medium density in general dense-medium separation process, but also achieve the automatic adjustment of the whole system with a simple control system and less investment. The automatic level of CPP has a great improvement.

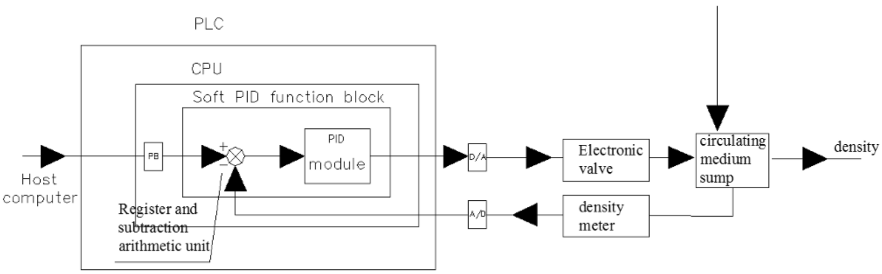



**Figure3**. **Principle of density automatic control in CPP**

PID pre-set parameters and density pre-set parameters are given by host computer, which are stored in memory and register. Density meter continuously inspects density of heavy medium suspension transported by circulating medium pump and take 4-20mA standard signal as process variables of inspected density value. After A/D transforming, it was stored in the input register by the corresponding channel. Writing collected process variables (density value of heavy medium suspension) into another register by a subtraction arithmetic unit.

The data from the two registers is operated by PID. The operation and processing results are stored in a register, which are restored in an output register by a subtraction arithmetic unit. The value of data in the output register is transformed through D/A conversion into 4-20mA regulatory signals transmitted to electric actuator, which drive make-up water valve of correcting unit change opening degree so that the amount of water supplement could varied regularly. It made density value within the range of pre-set parameters through overcoming interference and eliminating bias in time.

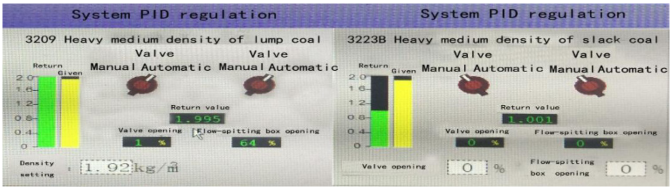



**Figure4. PID density control page of circulating medium in SuanCiGou CPP's centralized controlling room.**

When the difference of pre-set parameters and inspection signal becomes zero, actuator does not work and this system becomes stable temporarily. When new disturbance appear, the system will automatic control followed the above procedure. Besides, the PID module also has manual working model. In this working model, it needs to manipulate electric actuator directly to drive make-up water valve of correcting unit works based on the output variable value set by host computer, with little manual operation work during the process.

### 3.1.4. Adding Density Trend Curve, Enhancing Density Fluctuation Assessment

Adding history curve of density into automatic control system, it is convenient to manage and check out daily density variations. Real-time variety trend of density data during a time period is shown in history density curve. From observing the fluctuation of the curve, in which it can be obtained the most problems period in order to control density much better by checking and analyzing the reasons. In addition, it could provide guidance and reference for current production based on density history curve combined with the quality change of raw coal.

Density history curve has been obtained from CPP's centralized controlling system. Density history curve before modification is shown in figure 5 (red curve). Within 5 hours the range of density fluctuation is 0.6g/cm$^3$, which is comparatively large. However, within 5 hours the range of density fluctuation is 0.005g/cm$^3$ after modification, which is more stabile. Density history curve after modification is shown in figure 6 (red curve). Compare the two curves, the stability for density meter is improved obviously after modification.

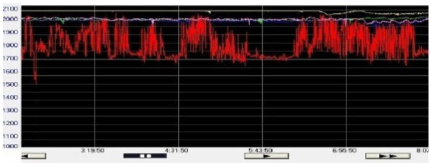


**Figure5. Density history curve before modification**

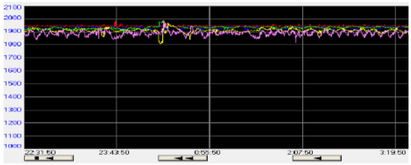


**Figure6. Density history curve after modification**

## 3.2. Improvement of H.M Vessel

The overflow weir of H.M vessel has been modified. Original design easily leads to liquid surface retention in chute between H.M vessel and dewatering and medium draining screen. By making the inlet of chute as the same size of the outlet of vessel, and reducing the height of inlet of chute to keep a certain height difference for ensuring the speed of overflow, it could prevent liquid surface retention, and reduce the possibility of middling-density layer, improve separation efficiency and decrease misplaced ratio.

## 3.3 Upgrading Production Management

1）Grasp the quality of raw coal feed and adjust process parameters

2）Grasp market demands and ensure the ratio of washing coal and the quality and quantity of high and low calorific value of commercial coal. Washing process is adjusted basis on the feedback information of market demands given by the transportation & marketing department.

3）Making Washing process comply with special geologic situation. The corresponding emergency production plans are established under special geologic and different commercial coal consist situations. It should obey emergency production regulation to organize production in special time.

4）Enhancing assessment and management of coal quality

（1）Density assessment

Once the fluctuation value is out of the pre-set range by checking separation density trend curve every day, it should assess the on duty centralized control staff.

（2）Adding gangue float-and-sink experiments and enhancing assessment ratio of misplaced coal in rejects.

## 4. Conclusions

The stability and accurate of density, separation efficiency of separation equipments and the recovery rate of clean coal are improved in SuanCiGou CPP by density monitor system modification and production management improving. Also the discharge of gangue and environmental pollution are decreased.

After adopting above measures, the yield of gangue is reduced 3.31% compared with the last year and the recovery rate of clean coal is obvious increased in SuanCiGou CPP. Calculating all the fees, the income is increased more than ￥23 million and economic benefit is remarkable.

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# Study on the Model System of Jig with Flexible Air Chamber and Pulsating Current Characteristics

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**Abstract:** A laboratory model system of jig with flexible air chamber is designed and established. The influences of jigging parameters on motion of pulsating current and bed layer are studied via model jig stratification test. The experimental results show that jigging cycle has significant effects on pulsating height of water flow and bed layer. With jigging cycle increasing, pulsating height of water flow and bed layer increases and porosity of bed layer is improved. With air pressure increasing within a certain range, not only acceleration, velocity and amplitude of water flow but also pulsating height and porosity of bed layer increase. Meanwhile, the collaborators obtain the motion parameter curves of one point on water surface and several bed layer particles based on high-speed dynamic video test. By analyzing these curves, the collaborators get following conclusions: Bed layer stratifies at last stage of particles ascending motion and early stage of particles descending motion during jigging. Differences in comprehensive force acting on particles at early stage of particles ascending motion are the motive for bed layer stratification.

**Key words:** jig with flexible air chamber; model jig stratification test; jigging cycle; pulsating current; high-speed dynamic video test; bed layer stratification; jigging parameter curves

Jigging has a wide range of applications, with low costs and good adaptability of different coal. However, control of jigging process is mostly based on experience rather than systemic theoretical guidance. Therefore, further exploration in bed layer stratification mechanism and influences of jigging parameters on pulsating current and bed layer motion can provide a significant impetus for improving jigging process controlling and bed layer stratification effect.

The collaborators propose that a bladder with a certain shape that is made of flexible waterproof material can be used as a jig's air chamber. They establish a model system of jig with flexible air chamber, equipped with parameter detecting and control system. Through experiment on the model system, the collaborators study the influences of jigging parameters on pulsating current and bed layer motion and bed layer stratification mechanism.

## 1 Model System of Jig with Flexible Air Chamber

A cylinder made of polyurethane has a diameter of 300mm and height of 900mm. With one end closed and the other end connected to the air duct, the cylinder becomes an impermeable flexible bladder. This bladder is used as the air chamber of the model jig. To achieve jigging process, water in the jig pulsates because of the agitation caused by the bladder's periodically expanding and flatting.

Figure 1 is the model system of jig with flexible air chamber. All parts such as pneumatic cover plate valves, air compressor and roots blower are automatically controlled. Parameter detecting and control system detects amplitude of water flow pulsation, porosity of the bed layer, wind pressure and wind velocity, etc. All of these important jigging parameters are stored and displayed in real-time.

## 2 Testing Process and Parameter Settings

### 2.1 Stratification Test on Model Jig

In test, rubber equilateral cylinder particles are used as jigging materials. Table 1 shows the properties of these particles. Suitable operating conditions matching the model jig and jigging materials are selected based on trial tests. Table 2 shows operating parameters of the stratification test on model jig.

 
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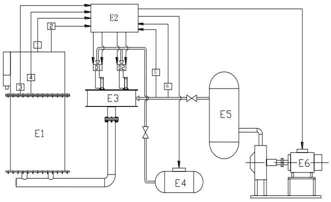



E1—Model jig E2—Control cabinet
E3—Damper system E4—High-pressure blower
E5—Wind package E6—Roots blower
D1, D2—Solenoid valve
1, 2—Ultrasonic displacement sensor
3, 4—Pressure sensor
5—Air pressure sensor
6—Wind velocity sensor

**Figure 1** Schematic diagram of model system of jig with flexible air chamber

### 2.2 High-Speed Dynamic Video Test

The collaborators shoot movements of pulsating current and bed layer during stratifying process using high-speed dynamic video. The frame rate and duration of each shot are 100 frame/s and 16s respectively. Several jigging processes periodically occur in 16s. Therefore, movements of pulsating current and bed layer are recorded completely in each test to show detailed process of bed layer stratification. The same particle in each of 1600 pictures obtained from one shot is drawn in sequence, forming 1600 points. Then the 1600 points are connected orderly and processed with software. This provides the motion parameter curves of points on water surface and particles in bed layer, such as the displacement curve, the velocity curve, the acceleration curve, the force curve and the energy curve. Table 3 shows operating parameters of high-speed dynamic video test.

**Table 1** Properties of particles

| size/mm | density / g.cm$^{-3}$ | color | mass 1 / Kg | mass 2 / Kg | mass 3 / Kg |
|---|---|---|---|---|---|
| 25 | 1.7 | green | 13 | 30 | |
| 25 | 1.3 | red | 13 | 30 | |
| 13 | 1.7 | green | 17 | | 30 |
| 13 | 1.3 | red | 17 | | 30 |
| total | | | 60 | 60 | 60 |

**Table 2** Operating parameters of stratification test on model jig (mass 1)

| No. | cycle / s | air-inlet time/% | air-expanding time/% | air-exhausting time/% | air pressure / MPa |
|---|---|---|---|---|---|
| 1 | 1.1,1.2,1.3,1.4,1.5 | 27 | 35 | 27 | 0.028 |
| 2 | 1.1,1.2,1.3,1.4,1.5 | 27 | 35 | 27 | 0.030 |
| 3 | 1.3 | 27 | 33 | 24 | 0.028,0.03,0.033,0.035,0.038 |
| 4 | 1.4 | 27 | 33 | 24 | 0.028,0.03,0.033,0.035,0.038 |

**Table 3** Operating parameters of high-speed dynamic video test

| No. | cycle/s | air-inlet time/% | air-expanding time/% | air-exhausting time/% | air pressure/MPa | material |
|---|---|---|---|---|---|---|
| 1 | 1.1 | 24 | 35 | 20 | 0.026 | mass 2 |
| 2 | 1.2 | 24 | 35 | 20 | 0.026 | |
| 3 | 1.3 | 27 | 35 | 27 | 0.028 | mass 3 |
| 4 | 1.4 | 27 | 30 | 20 | 0.026 | mass 1 |

## 3 Influence of Jigging Parameters on Motion of Pulsating Current and Bed Layer

During stratification test on model jig, the collaborators calculate pulsating current high level, pulsating current low level, pulsating height and biggest porosity of the bed layer under different jigging cycles and air pressure respectively. These are four typical parameters reflecting motion characteristics of pulsating current and bed layer. Figure 2 and Figure 3 show the curves drawn according to test results.

### 3.1 Influence of Jigging Cycle

Figure 2 shows that under air pressure of 0.028MPa and 0.030MPa, both pulsating current high level and bed layer's pulsating height increase obviously with extension of jigging cycle, provided that the air-inlet time, air-expanding time and air-exhausting time are the same. Under air pressure of 0.028MPa, we can see that with jigging cycle increasing from 1.1 s to 1.5 s, the difference between pulsating current high level and low level, also known as amplitude of water flow pulsation, increases from 107 mm to about 149 mm, and bed layer's pulsating height increases similarly, from 107 mm to 151 mm. Biggest porosity of bed layer increases to some extent, too. In summary, jigging cycle has significant effects on pulsating height of water flow and bed layer.

### 3.2 Influence of Air Pressure

Curves of pulsating current high level, pulsating current low level, bed layer's pulsating height and biggest porosity in Figure 3 show similar tendency when materials properties, jigging cycle, air-inlet time, air-expanding time and air-exhausting time are the same. With air pressure increasing, all curves slope upward obviously, which means that water and bed layer's pulsating height increases markedly. Because velocity is once differential of displacement to time and acceleration is twice differential of displacement to time, it can be inferred that displacement, velocity and acceleration of water all increase with increased air pressure. On the whole, air pressure influences motion of water and bed layer significantly.

With jigging cycle 1.3 s, bed layer stratifies significantly after 7, 7, 5 and 4 jigging cycles corresponding to air pressure of 0.028MPa, 0.030MPa, 0.033MPa and 0.035MPa respectively. Under higher air pressure of 0.038MPa, bed layer jumps up and falls down steadily, but does not stratify significantly throughout regardless of jigging cycles. Similar experimental phenomena are observed with jigging cycle 1.4 s. Therefore, all these suggest that better stratification effect of bed layer is not necessarily related to higher air pressure. The amplitudes of water flow and bed layer pulsation and stratification effect need to be considered when regulating air pressure. For jig with flexible air chamber, limit of air chamber's expansive degree should be taken into consideration.

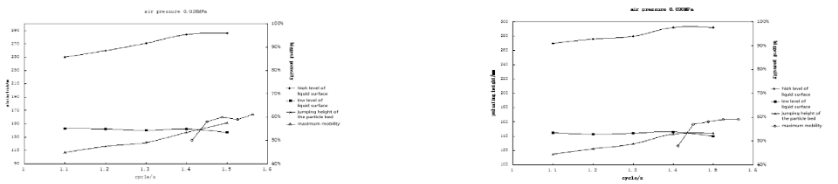


**Figure 2** Influence of jigging cycle

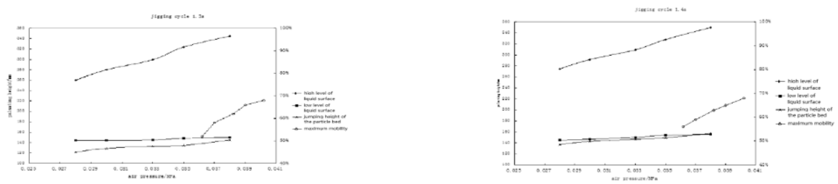


**Figure 3** Influence of air pressure

## 4 Movement of Current and Particles

### 4.1 Movement of Current and Particles

Pulsating current is the motivation of bed layer's stratification. Being pushed by pulsating current, particles begin to move. But particles lag behind current obviously in movement because of dead weight and some other factors [1-6]. Figure 4 demonstrates the motion parameter curves of one point on water surface and three particles (with curve (1) about point on water surface and the other three about particles).

Some information obtained from these curves are listed as follows:

(1) Peak of water velocity curve is higher than that of particle velocity curve, while trough of water velocity curve is lower than that of particle velocity curve. It shows that change range of water velocity is larger than that of particle velocity during jigging. When the movement starts, water acceleration is at the maximum value and towards positive direction, but particle acceleration is nearly zero. With water acceleration decreasing, particle acceleration firstly increases to the maximum and then decreases. Acceleration curves continue to drop below zero after water and particle acceleration both reducing to zero, indicating that acceleration direction has shift from positive to negative at this time and the value increases. And then particle and water acceleration curves reach trough with the former earlier than the latter. Acceleration curve indicates that particle acceleration reaches the negative maximum earlier than water acceleration, which means that particle stops ascending earlier than water, according to velocity curves.

(2) Force can be reflected by acceleration. Therefore, acceleration curves of water and particles are similar to force curves. In the beginning of movement, force acting on water is much greater than force acting on particles. At early stage of ascending motion, force acting on particles is mainly generated by water. Influence of gravity on particles motion increases drastically at last stage of ascending motion.

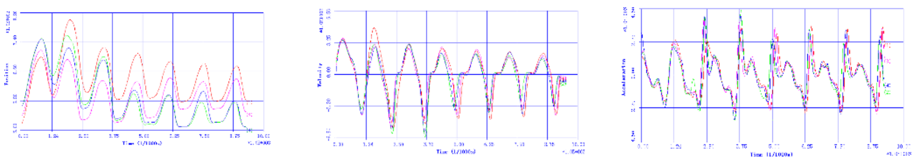
(3) Energy curves present changes in kinetic energy of water and particles. The moment when energy curves reach peak corresponds to the time when velocity curves reach peak or trough. The time of kinetic energy being zero is equal to the time of water velocity or particles velocity being zero. Peak of water energy curve is higher than that of particle energy curve. Particle energy at descending motion stage is greater than that at ascending motion, indicating gravity is not the resistance but the driving force for particle descending.

(4) In the first 1～3 jigging cycles, bed layer particles of different properties are in disorder, so the force mainly generated by pulsating current acts on particles most significantly and the differences in influence of pulsating current on different particles are the most obvious. As a consequence, the maximum value of particle acceleration, velocity, displacement and energy appears during this period. Peak of motion parameter curves corresponding to these jigging cycles is higher than that in subsequent cycles.

**4.2 Procedure of Bed Layer Stratification**

"Particles of different densities exchanging positions" is the mark of bed layer stratification. "Displacement curves of particles of different densities crossing" displays the mark graphically. In Figure 5, the green displacement curve of high density particle lies above the red displacement curve of low density particle at the beginning. After several jigging cycles, the two curves cross at a certain moment. This means that the two particles of different densities exchange positions, namely, bed layer stratifies.

All of the three groups of curves in Figure 5 show that under different test conditions, the green and red curves cross for the first time (i.e. bed layer stratifies) at last stage of particles ascending motion and early stage of particles descending motion. The reason for the second crossing of the curves is that some particles with smaller size rise through gaps among particles.



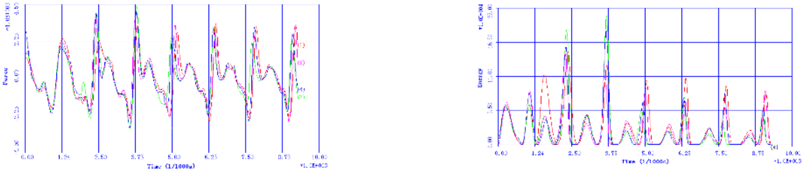


**Figure 4** Motion parameter curves ( Table 3 No.4 )

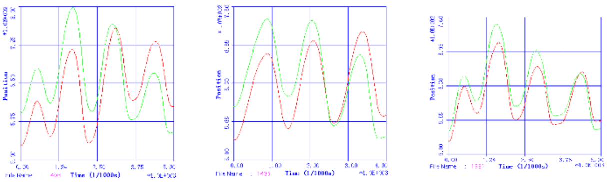


Table 3 No.1 Table 3 No.2 Table 3 No.3

**Figure 5** Particle displacement curves

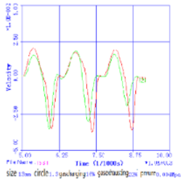


Table 3 No.1

**Figure 6** Particle velocity curves

**Table 4** Terminal velocity and jigging velocity

| size/mm | density/g·cm$^{-3}$ | terminal velocity/cm·s$^{-1}$ | No.1/cm·s$^{-1}$ |
|---|---|---|---|
| 25 | 1.3 | 50.19 | -35～20 |
| 25 | 1.7 | 76.66 | -35～18 |

The 4th column of table 4 shows velocity range of particles during jigging, which corresponds to the data from particle velocity curves in Figure 6. The 3rd column of table 4 shows terminal velocity of particles, which are data from theoretical calculation. Clearly, velocity of particles during jigging is much less than terminal velocity. Besides, particles of different densities stratify at last stage of ascending motion and early stage of descending motion. The two points indicate that the difference in terminal velocity is not the entire motive for particles of different densities to stratify. Difference in terminal velocity is only to keep the relative positions of particles layered consistent with status of them when they begin to descend [4-10].

### 4.3 Bed Layer Stratification Mechanism

During jigging, force generated by pulsating current and the remaining force compose a comprehensive force that acts on particles. Curves in Figure 4 show that at the beginning of a jigging cycle, there is hardly any change in displacement, velocity and energy of particles (displacement, velocity and energy is zero), but differences in the comprehensive force and acceleration among particles of different properties emerge obviously. At early stage of particles ascending, direction of the comprehensive force acting on them is upward. The greater the comprehensive force is, the greater kinetic energy particles get. And the later particles stop ascending, the further particles move upward. On the contrary, the smaller the comprehensive force acting on particles is, the shorter particles move upward at ascending motion stage. During descending motion, the relative positions of particles remain unchanged. Therefore, after several similar procedures, particles getting greater comprehensive force at early stage of ascending motion will finally move upward above particles getting smaller comprehensive force. That is to say, bed layer stratifies.

Based on the above analysis, it can be concluded that the properties of particle such as density, size and initial position determine the comprehensive force acting on it at early stage of ascending motion.

Differences in comprehensive force among particles bring differences in motion parameters like velocity, displacement and energy, and lead to bed layer stratification eventually.

**5 Conclusions**

Through stratification test and high-speed dynamic video test on model system of jig with flexible air chamber, the collaborators study the influences of jigging parameters on motion of pulsating current and bed layer, analyze motion characteristics of water and particles, and explore procedure of bed layer stratification as well as bed layer stratification mechanism. Conclusions are drawn as follows:

(1) Jigging cycle has significant effects on pulsating height of water flow and bed layer. With jigging cycle increasing, pulsating height of water flow and bed layer increases and porosity of bed layer is improved.

(2) With air pressure increasing, not only acceleration, velocity and amplitude of water flow but also pulsating height and porosity of bed layer increase. Higher air pressure does not necessarily bring better stratification effect of bed layer. Regulating air pressure considers amplitude of water and bed layer and stratification effect. For jig with flexible air chamber, limit of air chamber's expansive degree should be taken into consideration when regulating air pressure.

(3) During jigging, bed layer stratifies at last stage of particles ascending motion and early stage of particles descending motion. Difference in terminal velocity is not the entire motive for particles of different densities to stratify. Difference in terminal velocity is just to keep the particles layered at relatively same positions during descending.

(4) Properties of particle such as density and size are the key determining factors of the comprehensive force acting on it at early stage of ascending motion. Differences in comprehensive force among particles are the motive for bed layer stratification.

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# Amenability of representative Greek lignite deposits to beneficiation. An overview

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**Abstract**

Lignite is the most abundant and important raw material of Greece. It has played a very important role in the energy policy of Greece, as it rendered the country sufficient to energy resources and considerably contributed to its development. The most important coal basins of Greece are located in the western part of Macedonia (Florina, Ptolemais, Kozani), Megalopolis (central Peloponessos) and Elasson (Thessaly). The deposits are characterized as low-rank coals, with their quality exhibiting great fluctuations.

Despite the magnitude and the importance of the deposits, there is limited research on the improvement of their quality. The objective of the present work is to examine the amenability of Greek lignite deposits, originated from the most important locations of the western Macedonia coal basin, to beneficiation by gravity methods (heavy media separation, jigging and tabling).

The experimental results clearly denote that gravity separation is efficient; the best results are obtained for xylite deposits while the results for lignite are slightly inferior. Among the methods examined, heavy media separation provides the best results while tabling the worst. The experimental results show that gravity separation could be efficiently applied to deposits from the coal basin examined as well as from other basins of Greece, because of their similar characteristics.

**Keywords**

Coal heavy medium separation; Coal jigging concentration; Coal tabling concentration; Coal washing; Gravity concentration; Greek coal deposits; Lignite beneficiation; Low-rank coal beneficiation

## INTRODUCTION

Despite the continuous rising of renewable energy sources, the use of coal as energy source remains crucial to the economies worldwide, because of its reliability and comparatively lower cost. Coal has been blamed for the adverse environmental impacts because of $CO_2$ emissions during burning in power stations. To maintain its share in power production and economic significance in the development of the countries around the world, coal must be competitive through quality improvement, and, consequently, environmental impacts reduction. Apart from the operational problems, the cost of passing inorganic materials through the combustion process is also a matter of serious concern. Moreover, according to the protocol of Kyoto, $CO_2$ emissions coming from the use of fossil fuels must be reduced. Also, further reduction of sulfur and nitrogen oxides and particulates is required. These issues clearly indicate the necessity for coal raw materials with low ash and high combustible matter grade.

Processing aims at improving coal quality, thermal efficiency of conventional and advanced power stations, and reducing environmental impacts during combustion and gasification. The processing methods used for coal beneficiation are mainly gravity ones (Alexis, 1980; Burt, 1984; Deurbrouck and Palowitch, 1979; Govindarajan and Rao, 1994; Lovell, 1979; Palowitch and Deurbrouck, 1979; Tiernon, 1980; Zimmerman, 1975); flotation and oil-agglomeration have also been tested on bituminous and low-rank coal (Ignasiak et al., 1996; Laskowski and Yu, 2000; Mohanty et al., 1997; Mohanty et al., 1998).

### Lignite in Greece

Lignite is the most important raw material of Greece. Almost all mined lignite is consumed for power generation but its share in energy mix is getting continuously reduced throughout the last years. In 2012, its share was approximately 55% but decreased to 46% in 2013.

The term "lignite" is used to describe a variety of coal types, from peat to sub-bituminous. The Greek deposits are characterized as low-rank coals. The lignite deposits of Greece are composed of successive

 
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lignite seams (with the thickness of seams ranging from some centimeters to some meters) alternating with inorganic material (argillaceous, carbonaceous, and siliceous) of varying thickness. The quality of coal deposits varies significantly. Moisture content varies from 8% (Eocene age deposits) to 62% (Pleistocene age); $C_{fix}$ ranges from about 40% (Eocene age) to only 10% (Pleistocene age), with the content of the majority (Pliocene and Miocene age) being between 14 and 30%; volatile matter content ranges from 40% (Eocene age) to about 17% (Pleistocene age), with the majority (Pliocene and Miocene age) ranging from 18 to 35%. Finally, the sulfur content of almost all the deposits is low, ranging from 0.8 to 1.2%.

Although the annual mining of lignite in Greece amounts to millions of tonnes (47.4 Mt in 2011), the beneficiation studies are limited to lab scale (Anastassakis, 2004; Chammas et al., 1999; Chammas et al., 2000; Dolgyras et al., 2005). This paper is an overview on the amenability of the Greek lignite deposits to beneficiation, applying mainly gravity separation methods (heavy media, jigging and tabling); it presents the results of coal processing from three different sites of the most important coal basin in Greece, which is that of Florina – Ptolemais - Kozani basin, located in the south-western Macedonia.

## EXPERIMENTAL

### Materials and Methods

The examined samples originated from open-pit mines, all in operation, of the following regions: Servia (Kozani), Kardia (Ptolemais) and Achlada (Florina). The lignite from Servia coal mine is used as fuel and reducing agent in the pyrometallurgical processing of nickeliferous laterites; the material from the other two coal mines is used to feed two thermal power plants located at Ptolemais and Florina area, correspondingly. As with the majority of coal deposits of Greece, the material from Servia (Kozani), Achada (Florina), and Kardia (Ptolemais) mines is low-rank coal.

The material from Servia was obtained, correspondingly, from the lower xylite and lower lignite horizon, which have relatively higher ash content compared to the upper two; the material from Achlada is xylite while that from Kardia is low-rank lignite.

Each representative sample was crushed, screened and subjected to heavy liquid testing. Subsequently, selected size fractions were submitted to the following gravity separation methods: a) heavy media separation (cone, cyclone), jigging, and tabling were tested on the sample from Servia mine, and b) jigging on the sample from Achlada and Kardia mines. The products were dried, weighted and their content in certain substances or elements was determined, on dry basis, by chemical analysis.

## RESULTS AND DISCUSSION

### Servia Coal Mine

Heavy media separation

Based on the results of float-sink tests, heavy media separation was further studied. The +5 mm size fraction of the lower xylite sample was separated in a cone separator while the -5 mm fraction in dense media cyclone. The corresponding results at sp. gr. 1.35 are presented in Table 1.

The results show that heavy media separation seems to be feasible for xylite. A merged product with increased yield (70%), low ash content (14.8%) and satisfactory ash rejection to the tailing (approx. 58%) is obtained, in case that separation at specific gravity 1.35 is adopted.

The same trends were generally observed for lignite (Table 2). In this case, the ash content of the float was significantly reduced but not below 18%.

Jigging

The results of jigging (Figs 1 and 2) denote that separation is satisfactory for the two coarser fractions; the ash content of the float (concentrate) ranges between 13 and 17% in most cases, and the corresponding yield between 35 and 56%. In some cases, the ash content was even lower but the yield of the float was low. The corresponding ash removal in the tailings ranges between 55 and 85%.

For the coarser fractions, the ash content and ash distribution in the float are getting lower with pulsion stroke decrease. For the –2+1 mm sieve fraction, the reduction is not significant. The same conclusion is reached with the decrease of water flow-rate.

Table 1. Results of dense media separation (sp. gr. 1.35) of lower xylite for particle size +1 mm.

| Particle Size (mm) | Product | Yield % | Yield % of initial feed | Moisture % | Ash % | Ash Distribution % |
|---|---|---|---|---|---|---|
| +5 | Float | 74.79 | 62.81 | 16.35 | 15.01 | 47.49 |
| | Sink | 25.21 | 19.34 | 8.43 | 49.23 | 52.51 |
| | Feed | 100.00 | 82.15 | 14.49 | 23.63 | 100.00 |
| -5+2.8 | Float | 55.16 | 4.44 | 24.41 | 13.82 | 30.47 |
| | Sink | 44.84 | 3.17 | 13.93 | 38.81 | 69.53 |
| | Feed | 100.00 | 7.61 | 20.05 | 25.03 | 100.00 |
| -2.8+1 | Float | 44.88 | 2.79 | 27.49 | 11.24 | 19.22 |
| | Sink | 55.12 | 2.94 | 15.61 | 38.48 | 80.78 |
| | Feed | 100.00 | 5.73 | 21.39 | 26.26 | 100.00 |
| -1 | | 4.71 | 4.51 | 24.91 | 37.87 | |
| Merged Product | Float | 68.77 | 70.04 | 17.30 | 14.81 | 41.66 |
| | Sink and –1 mm | 31.23 | 29.96 | 12.19 | 45.68 | 58.34 |
| | Total Feed | 100.00 | 100.00 | 15.77 | 24.45 | 100.00 |

Table 2. Results of dense media (cone) separation for +5 mm particle size of lower lignite

| Sp. gravity | Product | Yield % | Moisture % | Ash % | Ash Distr. % |
|---|---|---|---|---|---|
| 1.25 | Float | 38.92 | 20.51 | 18.10 | 20.61 |
| | Sink | 61.08 | 18.04 | 44.44 | 79.39 |
| | Feed | 100.00 | 19.02 | 34.19 | 100.00 |
| 1.30 | Float | 54.04 | 18.37 | 20.20 | 31.94 |
| | Sink | 45.96 | 15.77 | 50.62 | 68.06 |
| | Feed | 100.00 | 17.20 | 34.18 | 100.00 |
| 1.35 | Float | 71.12 | 13.12 | 24.08 | 50.40 |
| | Sink | 28.88 | 10.80 | 58.35 | 49.60 |
| | Feed | 100.00 | 12.46 | 33.97 | 100.00 |

Tabling

The results of tabling (Table 3) show that separation is less efficient than that of heavy media and jigging.

**Florina-Ptolemais Coal Mine**

Float-sink analysis

The results of float-sink tests on the samples from Achlada (Florina area) and Kardia mine (Ptolemais area) are presented in Tables 4 and 5 (Dolgyras et al., 2005).

From Table 4, it derives that the float product at sp. gr. 1.25 has satisfactory qualitative characteristics; its yield is about 55% and ash content 11.3%, while the corresponding ash removal to the sink is 77.5%. The comparison with xylite from Servia (Kozani area) reveals that the results are similar.

From Table 5, it is concluded that the lignite sample from Ptolemais area presents differences, when compared with the lower lignite from Servia. At sp. gr. 1.25, the quality of the float is slightly better than that of the feed. This could be attributed to the very low ash content of Ptolemais lignite compared with that from Servia, possibly because of mistaken sampling. These results are not encouraging for successful gravity separation.

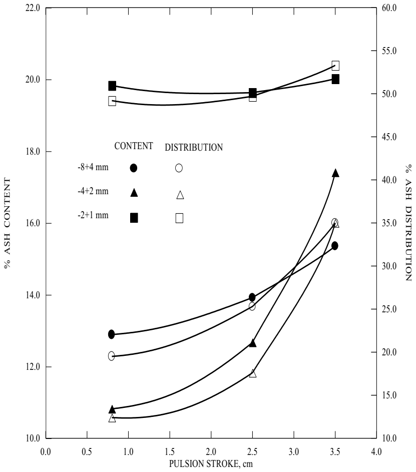


Figure 1. Effect of pulsion stroke on float ash content and distribution. Water flow-rate 10 l/min

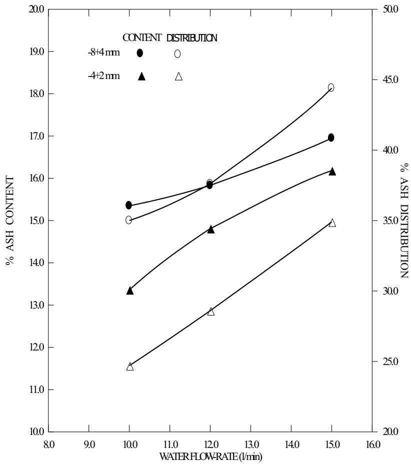


Figure 2. Effect of water flow-rate on float ash content and distribution. Pulsion stroke 3.5 cm

Table 3. Selected (best) tabling results for xylite

| **Size mm** | **Speed rpm** | **Stroke mm** | **Tilt degrees** | **Product** | **Yield %** | **Moisture %** | **Ash %** | **Ash Distr. %** |
|---|---|---|---|---|---|---|---|---|
| -5+2 | 220 | 18 | 10 | Light | 48.93 | 11.81 | 19.15 | 37.7 |
| | | | | Mid. 1 | 21.18 | 8.72 | 21.96 | 18.7 |
| | | | | Mid. 2 | 15.80 | 7.90 | 29.03 | 18.4 |
| | | | | Heavy | 14.09 | 6.81 | 44.56 | 25.2 |
| | | | | Feed | 100.00 | 9.88 | 24.89 | 100.0 |
| -2+1 | 250 | 18 | 10 | Light | 50.20 | 23.34 | 20.95 | 40.6 |
| | | | | Mid. 1 | 17.89 | 19.90 | 23.22 | 16.0 |
| | | | | Mid. 2 | 17.00 | 16.29 | 29.02 | 19.0 |
| | | | | Heavy | 14.91 | 13.56 | 42.46 | 24.4 |
| | | | | Feed | 100.00 | 20.24 | 25.93 | 100.0 |
| -1+0.5 | 230 | 16 | 10 | Light | 46.02 | 23.74 | 21.42 | 37.2 |
| | | | | Mid. 1 | 30.13 | 19.78 | 24.62 | 28.0 |
| | | | | Mid. 2 | 13.47 | 16.71 | 33.46 | 17.0 |
| | | | | Heavy | 10.38 | 14.54 | 45.41 | 17.8 |
| | | | | Feed | 100.00 | 20.78 | 26.50 | 100.0 |

Jigging
In both cases, jigging tests simply confirmed the results of float-sink analysis. The results on xylite from Achlada (Table 6) show that it is possible to obtain float products with very low ash content but of very low yield (Dolgyras et al., 2005). As concerns Ptolemais lignite, in the best case, the ash content of the light product was no lower than 10.7% (feed 12.6% ash) while the yield was only 7%.

Table 4. Float-sink analysis of Achlada mine xylite (Florina area, western Macedonia)

| Product | Yield % | Chemical Analysis % | | | Distribution % | | |
|---|---|---|---|---|---|---|---|
| | | $C_{fix}$ | S | Ash | $C_{fix}$ | S | Ash |
| -1.10 | 0.83 | 55.18 | 1.50 | 9.29 | 1.1 | 1.0 | 0.3 |
| +1.10-1.25 | 54.44 | 52.19 | 1.43 | 11.33 | 70.8 | 62.3 | 22.2 |
| +1.25-1.30 | 11.54 | 40.39 | 1.38 | 27.71 | 11.5 | 12.8 | 11.5 |
| +1.30-1.50 | 18.89 | 26.22 | 0.65 | 45.85 | 12.2 | 9.9 | 31.1 |
| +1.50-1.70 | 8.27 | 15.14 | 0.35 | 63.24 | 3.1 | 2.4 | 18.8 |
| +1.70 | 6.03 | 8.44 | 2.40 | 74.34 | 1.3 | 11.6 | 16.1 |
| Feed | 100.00 | 35.02 | 1.00 | 30.66 | 100.0 | 100.0 | 100.0 |

Table 5. Float-sink analysis of Kardia mine lignite (Ptolemais area, western Macedonia)

| Product | Yield % | Chemical Analysis % | | | Distribution % | | |
|---|---|---|---|---|---|---|---|
| | | $C_{fix}$ | S | Ash | $C_{fix}$ | S | Ash |
| -1.25 | 89.70 | 49.81 | 0.45 | 12.98 | 93.5 | 94.4 | 71.2 |
| +1.25-1.50 | 8.68 | 32.89 | 0.25 | 43.90 | 5.9 | 5.1 | 23.3 |
| +1.50-1.70 | 1.12 | 20.16 | 0.15 | 49.00 | 0.5 | 0.4 | 3.4 |
| +1.70 | 0.50 | 8.67 | 0.05 | 70.41 | 0.1 | 0.1 | 2.1 |
| Feed | 100.00 | 47.45 | 0.36 | 16.35 | 100.0 | 100.0 | 100.0 |

## CONCLUSIONS

Lignite is one of the most important primary raw materials in Greece. The huge reserves, the relatively loose specifications of the power plants feed and the need for low-cost power production are some of the reasons that prevented the Greek coal deposits from processing. The various deposits show more or less similar characteristics. This study is an overview on the amenability of low-rank Greek coal deposits to gravity separation. The samples originated from three different operating mines (Servia, Ptolemais, and Florina) of western Macedonia, where the major power plants of Greece are installed.

This study shows that Greek coal deposits are amenable to beneficiation by gravity methods. Among the various methods tested, the most efficient proved to be heavy media separation and jigging, which are suitable for relatively coarse particles. The results are better for xylite than for lignite deposits. Xylite deposits show, more or less, similar response to gravity separation; the lignite deposits showed different behavior but this could be attributed to the significant difference in ash content of the samples. This further denotes that the high-ash content lignite deposits are more amenable to gravity separation.

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Table 6. Jigging results of Achlada mine (Florina area) xylite

| **Particle Size (mm)** | **Water Flow-rate (l/min)** | **Product** | **Yield %** | **Ash %** | **Ash Distr. %** |
|---|---|---|---|---|---|
| -12+6 | 3.0 | Light | 13.9 | 5.20 | 3.1 |
| | | Heavy | 86.1 | 26.13 | 96.9 |
| | | Feed | 100.0 | 23.22 | 100.0 |
| -12+6 | 7.0 | Light | 27.9 | 5.14 | 8.0 |
| | | Heavy | 72.1 | 22.78 | 92.0 |
| | | Feed | 100.0 | 17.86 | 100.0 |
| -6+2 | 3.0 | Light | 31.1 | 9.98 | 16.1 |
| | | Heavy | 68.9 | 23.5 | 83.9 |
| | | Feed | 100.0 | 19.3 | 100.0 |
| -6+2 | 7.0 | Light | 36.3 | 10.23 | 18.7 |
| | | Heavy | 63.7 | 25.39 | 81.3 |
| | | Feed | 100.0 | 19.89 | 100.0 |

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# TECHNOLOGICAL EVALUATION OF WORK OF COAL PREPARATION PLANTS

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**Abstract**

The performed technological analysis of qualitative and quantitative operation performance of coal preparation plants of Ukraine for 23 years as compared with the reference analysis of 1990, which had shown that the power complex operates with concentrate yield loss, which tends to decrease, though. The coke complex operates with positive balance. In general, the sub-industry suffered the loss of 12.9 million tons of concentrate for 23 years due to losses of combustible mass with production waste. The analysis of operation performance of certain plants (Duvanskaya CCPP – coke, Kurakhovskaya CCPP – power, Sverdlovskaya CCPP – anthracite) proves the stated trends in general. Hence, the gain in concentrate production makes 215 thousand tons in Duvanskaya CCPP and about 2 million tons in Kurakhovskaya CCPP, while concentrate losses account for about 2.4 million tons in Sverdlovskaya CCPP.

They refer to necessity of scientific approach to improvement of process diagrams applied in coal preparation plants.

**Key words:** Coal, preparation, plant, concentrate, yield, losses, analysis.

Under conditions of continuous deterioration of quality of mine coals the preparation is a mandatory stage in production of fuel, which would satisfy the consumers' demands in quality.

Coal preparation becomes particularly important with a view to development of market relations, when not only the consumption value of coal concentrates, but also the recovery ratio of commercial components into marketable products at minimum material costs for their production are expressed in direct economic terms. In this regard the technological evaluation of operation of coal preparation plants is the most important indicator.

Technological evaluation of operation of coal preparation plant is performed through comparison of annual rates of reference and report periods. Likewise, the technological evaluation may be held for the group of plants or plant preparation industry in general.

The year, when the plant reached its highest qualitative and quantitative characteristics, is taken as a reference year. For instance, it is the year 1990 for Ukraine, when plants operated at their maximum load, complied with strict requirements to quality of marketable products for deliveries, offered the satisfactory technical condition of their basic and auxiliary facilities and had no down times caused by the lack of railway or motor transport and materials, agents and flocculants were supplied in accordance with regulations.

Increase or decrease in yield of marketable coal products in the year under review as compared to the reference one is considered the criterion for technological evaluation of operation of coal preparation plant.

Variation of qualitative characteristics of raw material, which does not depend upon coal preparation plants, may be eliminated through reduction of indicated characteristics of raw material and concentrate, inherent to the reference year, to characteristics of the year under review.

Re-calculation of qualitative characteristics is different for two and three preparation products derived and shall be held in accordance with.



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**Adjustment of concentrate yield at production of three end preparation products**

The concentrate yield shall be adjusted by the formulas:

a) Adjustment with reference to changes in ash contents of run coal.

The consumption index ($B_1$) is determined by the formula:

$$B_1 = \frac{100 - \gamma_{кб}}{A^d_{рб} - A^d_{кб}}, \tag{1}$$

where $\gamma_{кб}$ – concentrate yield in the reference year, %;

$A^d_{рб}$ and $A^d_{кб}$ – ash content, respectively, of run coal and concentrate in the reference year, %.

Difference in ash content of run coal in the reference and new cases:

$$\Delta A^d_р = A^d_{рб} - A^d_{рн}, \tag{2}$$

where $A^d_{рн}$ – ash content of run coal in a new case, %.

Change in concentrate yield in % as adjusted with reference to changes in ash contents of run coal:

$$\Delta\gamma'_к = B_1 \cdot \Delta A^d_р; \tag{3}$$

б) Adjustment with reference to change in ash content of concentrate.

The consumption index ($B_2$) is determined by the formula:

$$B_2 = \frac{\gamma_{кб}}{A^d_{ппб} - A^d_{кб}}, \tag{4}$$

where $A^d_{ппб}$ – ash content of mid-coal in the reference year, %.

Difference in ash content of concentrate in the reference and new cases:

$$\Delta A^d_к = A^d_{кн} - A^d_{кб}, \tag{5}$$

where $A^d_{кн}$ – ash content of concentrate in new case, %.

Change in concentrate yield in % as adjusted with reference to changes in ash contents of concentrate,

$$\Delta\gamma''_к = B_2 \cdot \Delta A^d_к; \tag{6}$$

в) Adjusted concentrate yield in the reference case

$$\gamma_{кск} = \gamma_{кб} + \Delta\gamma'_к + \Delta\gamma''_к; \tag{7}$$

г) Change in concentrate yield in the new case caused by implementation of continuous process flow

$$\Delta\gamma'''_к = \gamma_{кн} - \gamma_{ксн}. \tag{8}$$

**Adjustment of concentrate yield at production of two end preparation products**

a) Adjustment of concentrate yield with reference to change in ash content of run coal.

Adjusted concentrate yield in % with reference to change in ash content of run coal

$$\gamma'_{кск} = \frac{A^d_{отх.б} - A^d_{рн}}{A^d_{отх.б} - A^d_{кб}} \cdot 100\% \tag{9}$$

where $A^d_{отх.б}$ and $A^d_{кб}$ - ash content, respectively, of waste and concentrate in the reference case, %; $A^d_{рн}$ - ash content of run coal in the new case, %.

Change in concentrate yield in % with reference to change in ash content of run coal

$$\Delta\gamma' = \gamma'_{кск} - \gamma_{кб}, \tag{10}$$

where $\gamma_{кб}$ - concentrate yield in the reference case, %;

б) Adjustment of concentrate yield with reference to change in ash content of concentrate.

Adjusted concentrate yield in % caused by the change in ash content of concentrate

$$\gamma''_{кск} = \frac{A^d_{отхб} - A^d_{рб}}{A^d_{отхб} - A^d_{кн}} \cdot 100\%, \quad (11)$$

where $A^d_{рб}$ and $A^d_{кн}$ - ash content, respectively, of run coal in the reference case and of concentrate in the new case, %.

Change in concentrate yield in % with reference to change in its ash content

$$\Delta\gamma''_{к} = \gamma''_{кск} - \gamma_{кб}; \quad (12)$$

в) Adjusted concentrate yield in % in the reference year caused by the change in ash content of ash content of run coal and of concentrate

$$\gamma'''_{кск} = \gamma'_{кб} + \Delta\gamma'_{к} + \Delta\gamma''_{к}; \quad (13)$$

г) Change in concentrate yield in % in the new case

$$\Delta\gamma'''_{к} = \gamma_{кн} - \gamma'''_{кск}. \quad (14)$$

In case the change in concentrate yield in the new case gets “+”, the plant operation in the period under review improved, in case it gets “-” – the plant operation deteriorated.

*Table*

**Change in concentrate yield (Δγ, %) and production (ΔP, thousand tones)**

| Year | Plants | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Power | | Coke | | Power and coke | | Duvanskaya (coke) | | Kurakhovskaya (power) | | Sverdlovskaya (anthracite) | |
| | Δγ, % | ΔP, thous. tons | Δγ, % | ΔP, thous. tons | Δγ, % | ΔP, thous. tons | Δγ, % | ΔP, thous. tons | Δγ, % | ΔP, thous. tons | Δγ, % | ΔP, thous. tons |
| 1990 | - | - | - | - | - | - | - | - | - | - | - | - |
| 1991 | -3.3 | -2,162.3 | -2.9 | -1,276.7 | -3,0 | -3,286.5 | +0.7 | +11.8 | -1.8 | -26.6 | -1.7 | -64.2 |
| 1992 | -3.3 | -2,012.2 | -4.2 | -1,806.0 | -3.6 | -3,743.1 | +0.8 | +13.5 | -1.4 | -22.4 | -2.2 | -69.0 |
| 1993 | -3.2 | -1,718.9 | -2.6 | -839.0 | -2.8 | -2,407.5 | +0.4 | +4.2 | -0.6 | -8.5 | -2.9 | -85.8 |
| 1994 | -3.5 | -1,476.4 | -2.4 | -527.0 | -4.7 | -3,014.6 | -5.9 | -26.9 | -0.9 | -8.5 | -2.5 | -51.2 |
| 1995 | -4.4 | -1,532.4 | -1.5 | -289.0 | -4.9 | -2,650.7 | -3.7 | -14.4 | -0.7 | -5.1 | -1.7 | -28.3 |
| 1996 | -4.2 | -1,134.6 | -1.9 | -352.5 | -3.9 | -1,777.1 | -5.5 | -10.0 | +2.1 | +11.1 | -1.9 | -14.8 |
| 1997 | -3.0 | -799.7 | +0.4 | +106.9 | +0.2 | +106.7 | -3.1 | -11.5 | +8.0 | +65.1 | -2.5 | -21.0 |
| 1998 | -3.0 | -800.3 | +1.6 | +415.1 | +0.2 | +105.2 | -0.6 | -1.5 | +4.9 | +32.6 | -2.0 | -27.4 |
| 1999 | -3.1 | -856.0 | +0.6 | +129.4 | -1.5 | -737.7 | -4.8 | -18.8 | +3.2 | +23.5 | -2.2 | -35.7 |
| 2000 | -3.1 | -867.4 | +0.7 | +174.3 | +0.1 | +47.9 | -2.9 | -14.5 | +4.3 | +29.0 | -2.5 | -39.8 |
| 2001 | -2.5 | -662.1 | +0.4 | +103.7 | -0.6 | -314.3 | -3.3 | -16.6 | +2.5 | +35.8 | -3.1 | -75.4 |
| 2002 | -2.2 | -556.0 | +0.3 | +52.6 | -1.6 | -685.0 | -7.3 | -20.6 | +4.0 | +62.5 | -2.5 | -63.8 |
| 2003 | -2.4 | -579.0 | +2.3 | +388.4 | -0.7 | -287.1 | +0.1 | +0.6 | +5.4 | +91.5 | -3.0 | -83.7 |
| 2004 | -1.4 | -550.0 | +2.9 | +1,201.0 | +1.4 | +1,129.7 | +1.6 | +15.8 | +6.4 | +114.9 | -2.1 | -69.2 |
| 2005 | -1.1 | 431.2 | +2.3 | +804.7 | +0.8 | +593.5 | -1.9 | -19.9 | +5.9 | +65.6 | -2.8 | -91.3 |
| 2006 | -1.5 | -645.1 | +1.9 | +597.1 | +0.3 | +223.9 | -1.2 | -13.5 | +6.5 | +99.3 | -2.7 | -94.1 |
| 2007 | -1.9 | -775.6 | +2.7 | +788.4 | -0.2 | -140.0 | +3.9 | +40.1 | +6.7 | +121.1 | -3.5 | -118.0 |
| 2008 | -2.4 | -992.5 | +2.5 | +755.1 | 0 | 0 | +3.7 | +42.8 | +7.4 | +163.3 | -6.1 | -193.7 |
| 2009 | -2.2 | -847.9 | +2.7 | +704.8 | +0.2 | +129.3 | +4.5 | +48.0 | +7.6 | +147.9 | -5.2 | -197.9 |
| 2010 | -2.0 | -801.3 | +3.0 | +871.3 | +1.1 | +760.2 | +3.9 | +46.2 | +10.1 | +223.9 | -4.4 | -198.1 |
| 2011 | -1.3 | -577.8 | +3.2 | +952.7 | +0.8 | +593.7 | +3.6 | +43.5 | +11.5 | +278.6 | -6.3 | -235.8 |
| 2012 | -1.2 | -533.8 | +4.1 | +1,167.8 | +1.4 | +1,021.5 | +4.3 | +54.4 | +7.0 | +162.6 | -6.0 | -225.4 |
| 2013 | -1.2 | -551.2 | +3.8 | +1,149.3 | +1.8 | +1,371.2 | +6.4 | +70.7 | +12.3 | +276.6 | -7.3 | -285.1 |
| **Total** | | **-21,863.7** | | **+5,272.4** | | **-12,961.4** | | **+215.0** | | **+1,933.8** | | **-2,369.3** |

This study deals with two issues: operation of coal preparation as of a sub-industry and operation of certain coal preparation plants. The first issue comprises preparation of coke and power coal and coal preparation in general, while the second issue comprises operation of coke (Duvanskaya CCPP), power (Kurakhovskaya CCPP) and anthracite (Sverdlovskaya CCPP) plants.

Reference data for analysis were taken from the reference books “Technical and Economic Analysis of Operation of Coal Preparation Plants of Ukraine” for 1990-2013, which are annually published by Ukrniiugleobogashcheniye.

The Table and Fig. 1, 2 present the results of analysis of process diagrams of coal preparation plants of Ukraine for 1990-201.

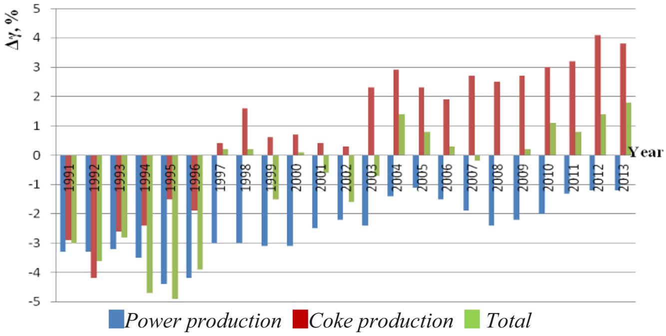


Fig. 1. Concentrate yield balance of power and coke plants and of industry in general as compared to the reference year 1990

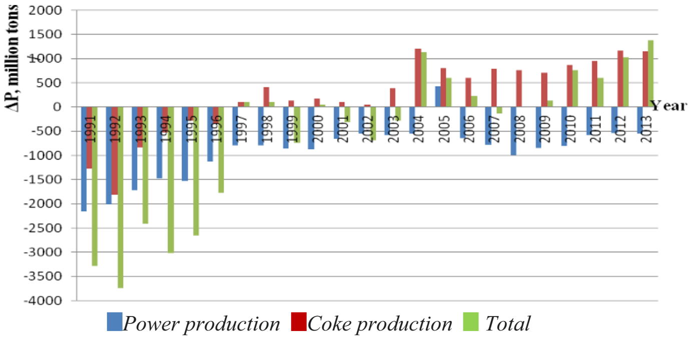


Fig. 2. Concentrate yield balance of power and coke plants and of industry in general as compared to the reference year 1990

The analysis of these data shows that the power complex of coal preparation plants of Ukraine have operated with the yield loss for the last 23 years (as compared to 1990). The losses reached their maximum value in 1995, when they made 4.4%.

Reduction in losses tends to be inherent during the last years. Average concentrate yield losses made -3.4% in 1991-2000, -2.0% in 2001-2010, -1.2% in 2011-2013. Total losses of marketable power concentrate made 21.8 million tons, including over 1.5 million tons or 0.5 million tons annually for the last 3 years. The data obtained evidence of an opportunity to get additional quantity of marketable power concentrate due to reduction in losses of combustible mass with coal preparation waste.

With a view to results of analysis of operation of coke preparation plants it can be concluded that their operation if more effective. The yield loss in these plants could be observed from 1991 till 1996. And the balance of these plants has been positive from 1997 on.

Average concentrate yield losses made -1.2% in 1991-2000, +2.1% in 2001-2010, +3,7% in 2011-2013. The data given prove that the modernization of coke coal preparation plants allowed to improve the coal preparation technology, which caused the increase of coke concentrate yield due to reduction in its losses with coal preparation waste. Additional production of coke concentrate made over 5 million tones, including over 3.2 million tons or 1 million tons per year for the last 3 years.

The analysis of operation of all coal preparation plants of Ukraine demonstrates that the industry tends to generally decrease the losses of combustible mass. If total concentrate yield losses made -2.4% in 1991-2000, they were almost absent in 2001-2010 ($\Delta\gamma_к$ = +0,1%), and concentrate yield increased for +1.3% in 2011-2013. The industry lost 12.9 million tons of concentrate in general, but its gain made about 3 million tons or about 1 million tons per year for the last 3 years.

The analysis of qualitative and quantitative performance characteristics of certain coal preparation plants (Duvanskaya CCPP – coke, Kurakhovskaya CPPP – power, Sverdlovskaya CCPP – anthracite), given in Table 1 and Fig. 3, 4, allows to make the following conclusions.

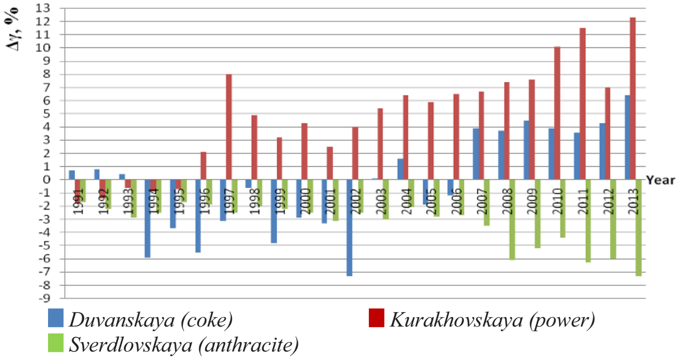


Fig. 3. Concentrate yield balance of Duvanskaya, Kurakhovskaya and Sverdlovskaya coal preparation plants as compared with the reference year 1990

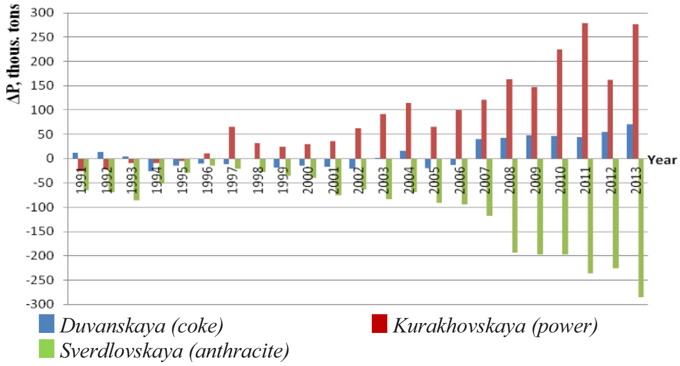


Fig. 4. Concentrate yield balance of Duvanskaya, Kurakhovskaya and Sverdlovskaya coal preparation plants as compared with the reference year 1990

Duvanskaya CCPP, which prepares coals of К and Ж ranks, constantly improves its process diagram. In case in 1991-2000 the concentrate loss made 2.5% in average, the next decade accounted for no coal losses ($\Delta\gamma_{к}$ = +0.4%). Concentrate yield made +4.8% in 2011-2013. Concentrate production yield for 23 years made 215 thousand tons, including 168.6 thousand tons or 56.2 tons per year for the last 3 years.

Kurakhovskaya CCPP, which prepares coals of ДГ and Г ranks, constantly improves its process diagram and increases the concentrate yield due to reduction in losses of combustible mass with production waste. Average concentrate yield gain made +1.7% in 1991-2000, +6.3% in 2011-2010, +10.3% in 2011-2013. The plant gained about 2 million tons of concentrate additionally for 23 years, including 717.8 thousand tons or about 240 thousand tons per year for the last 3 years.

Sverdlovskaya CCPP, which prepares coals of A rank, constantly increases the losses of combustible mass with coal preparation waste and reduces concentrate yield and production. Hence, if the average concentrate yield loss made -2.2% in 1991-2000, it made -3.5% in 2001-2010 and -6.5% in 2011-2013. Total concentrate yield losses for 23 years made about 2.4 million tons, including 746.3 thousand tons or about 250 thousand tons per year for the last 3 years.

The above-stated facts cause the necessity of technical improvement and modernization of operating enterprises based on scientific rationale, search and application of new progressive processes, high-tech, high-performance, effective equipment with minimum energy consumption, which allows to reduce the losses of combustible mass with production waste.

Only in this case there will appear an opportunity to eliminate negative effects of run coal treatment, which exist in coal preparation plants of Ukraine, and to increase the yield of marketable product and its realization value.

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5. Technical and economic analysis of operation of coal preparation plants of Ukraine for 1993. – Lugansk: Ukrniiugleobogashcheniye. – 1994 (p. 143).

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10. Technical and economic analysis of operation of coal preparation plants of Ukraine for 1998. – Lugansk: Ukrniiugleobogashcheniye. – 1999 (p. 122).

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# ТВЗП-50 TECHNOLOGY OF MIDDLING PRODUCT REWASHING

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**Abstract**

The coke concentrate yield could be increased due to reduction of combustible matter loss with coal preparation rejects and elimination of produced commercial middling products.

The solution of this problem involves middling product separation in every concentrating process and rewashing with the use of the ТВЗП-50 technology developed by the Ukrniiugleobogashchenie Institute, which is based on application of three-product hydrocyclone with magnetite medium.

Feature of the ТВЗП-50 technology is separation of true middling product from initial middling product with its release and return to the feed of three-product hydrocyclone.

ТВЗП-50 technology components:

- middling product collection, deslurrying and preparation (on the stationary СД-2,0 sieve bend and ГВЧ-61М vibration screen);
- middling product concentration in the three-product heavy-medium hydrocyclone ГТ-630/500;
- magnetite washing and concentrated products dewatering on the ГВЧ-61М and ГИСТ-61М screens;
- two-stage magnetite medium regeneration in the СБМ-0,9/2,5Р and СБМ-0,9/1,5Р separators;
- secondary middlings preparation by size degradation to the -3mm size in the СМ-43 crusher and return to the ГТ-630/500 heavy-medium hydrocyclone feed.

Main factors of the ТВЗП-50 technology applicability are as follows:

| | |
|---|---|
| - initial product output rate, max. | - 50 t/h; |
| - initial middling product size | - 0-13 mm; |
| - initial middling product ash content | - 30-45%; |
| - coke concentrate (final) ash content, max. | - 12.5%; |
| - rejects ash content, min. | - 70.0%; |
| - concentrate moisture content, max. | - 10.0%; |
| - rejects moisture content, max. | - 18.0%. |

The recommendations are developed for the ТВЗП-50 technology implementation at the Krasnolimanskaya, Chumakovskaya, Uzlovskaya and Oktyabrskaya CPPs.

**Keywords**

Primary middling product, three-product hydrocyclone, magnetite medium, secondary middlings, size degradation, concentration.

Shortage and low quality characteristics of coke concentrates adversely affect coke production development in the Ukraine [1]. One of the coke concentrate shortage reduction solutions is increasing the concentrate yield due to the reduction of above-standard coal losses with coal preparation rejects.

The above-standard coal losses with coal preparation rejects at the coal preparation plants in the Ukraine made 440.6 thousand tons in 2012 and 371.5 thousand tons in 2013 [2].

Principal reasons of above-standard losses of combustible matter are due to the use of concentration technologies without the middling product separation to the commercial product, which predetermines

 
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middling product distribution into concentrate and into rejects causing the ash content increase in the former case and combustible mass losses increase in the latter case.

Other coke concentrate shortage reduction solution is increasing the concentrate yield due to elimination of production of middlings as a commercial product.

It is known that since 2012, coal preparation plants in the Ukraine produce about 250 thousand tons of commercial middling products with ash content of 35-45%. It is found that commercial middling products consist (on the average) of 30% of the concentrate fractions, 40% of middlings and 30% of rocks. Therefore, even the simple density separation of this product could result in concentrate recovery from it, particularly, with the use of additional middling product release.

Hence, solution of this problem should be middling product separation in every concentrating process and development of the special-purpose technology for middling product rewashing.

It is reported [3-6] that concentration of middling product involves two major operations. First operation being known as preparatory operation is mineral grains release, and the second major operation is mineral grains density separation. It stands to reason that achievement of efficient separation of minerals requires proper release of grains. So, should the first operation is not quite done, second main operation could not be thoroughly completed. This statement could be applied to coal concentration technology to a certain degree only, because separation could be to the some extent carried out at presence of aggregates. However, above-mentioned statement is fully applied to concentration of middling product containing considerable amount of aggregates. Thus, middling product concentration problem could only be correctly solved after the relative middling product preparation for concentration, that is, sufficient release of aggregate-forming component grains.

Current coal concentration practice usually involves only size degradation of coarse jig middling product in order to bring its size nearer to that of fine jig middlings (formation of the graded mixture of middlings for concentration in the scavenger jig or in the fine jig as a circulating load.) At such size degradation quality, minimum grain release occurred, and middling product concentration process was inefficient.

Presently, most common coal preparation practice methods are wet gravitation concentration techniques. These methods differ widely and offer considerable potential for a various combinations and process layout alternatives [7-9].

Based on the analysis of fine coal machine grade and fine middling product concentration technology and data of experimental study of middling product concentration in various gravity machines, heavy-medium hydrocyclones with magnetite medium are the most simple and efficient equipment providing the most clean-cut separation at set density [10].

Use of three-product hydrocyclones for middling product concentration as compared with two-product ones is more efficient due to prevention of significant middling product amount rejecting and guaranteed separation of three standard quality products – concentrate, rejects and middling product. After the middling product concentration in three-product hydrocyclones, amount of separated commercial middling product is considerably less than amount of initial middling product. The problem of commercial middling product yield reduction is solved, but not to the fullest extent. Therefore, it is necessary the release grains of separated secondary (“true”) middling product by means of size degradation and partially recover combustible matter into concentrate. Although size degradation of middling product not necessarily guarantees the complete grain release, but reasonable degree of material degradation is only determined on a case-by-case basis after the carrying out applicable tests. It is imperative that the middling product concentration process flow-chart includes three-product hydrocyclone middling product degradation operations for further concentration also with the use of heavy-medium hydrocyclone separation method.

Based on the investigation and analysis of international practices of middling product rewashing, the Ukrniiugleobogashchenie Institute developed technology of additional production of commercial products by means of middling product concentration, which is known as ТВЗП-50.

ТВЗП-50 technology components:

- - middling product collection, deslurrying and preparation (at the stationary СД-2,0 sieve bend and ГВЧ-61М vibration screen);

- middling product concentration in three-product heavy-medium hydrocyclone ГТ-630/500;
- magnetite washing and concentrated products dewatering on the ГВЧ-61М and ГИСТ-61М screens;
- two-stage magnetite medium regeneration in the СБМ-0,9/2,5Р and СБМ-0,9/1,5Р separators;
- secondary middlings preparation by size degradation to the -3mm size in the СМ-43 crusher and return to the ГТ-630/500 heavy-medium hydrocyclone feed.

Main factors of the ТВЗП-50 technology applicability are as follows:

- initial product output rate, max. - 50 t/h;
- initial middling product size - 0-13 mm;
- initial middling product ash content - 30-45%;
- coke concentrate (final) ash content, max. - 12.5%;
- rejects ash content, min. - 70.0%;
- concentrate moisture content, max. - 10.0%;
- rejects moisture content, max. - 18.0%.

Feature of the ТВЗП-50 technology is separation of true middling product from initial middling product with its release and return to the feed of three-product hydrocyclone, which considerably reduces crushed material weight and amount of generated slurry.

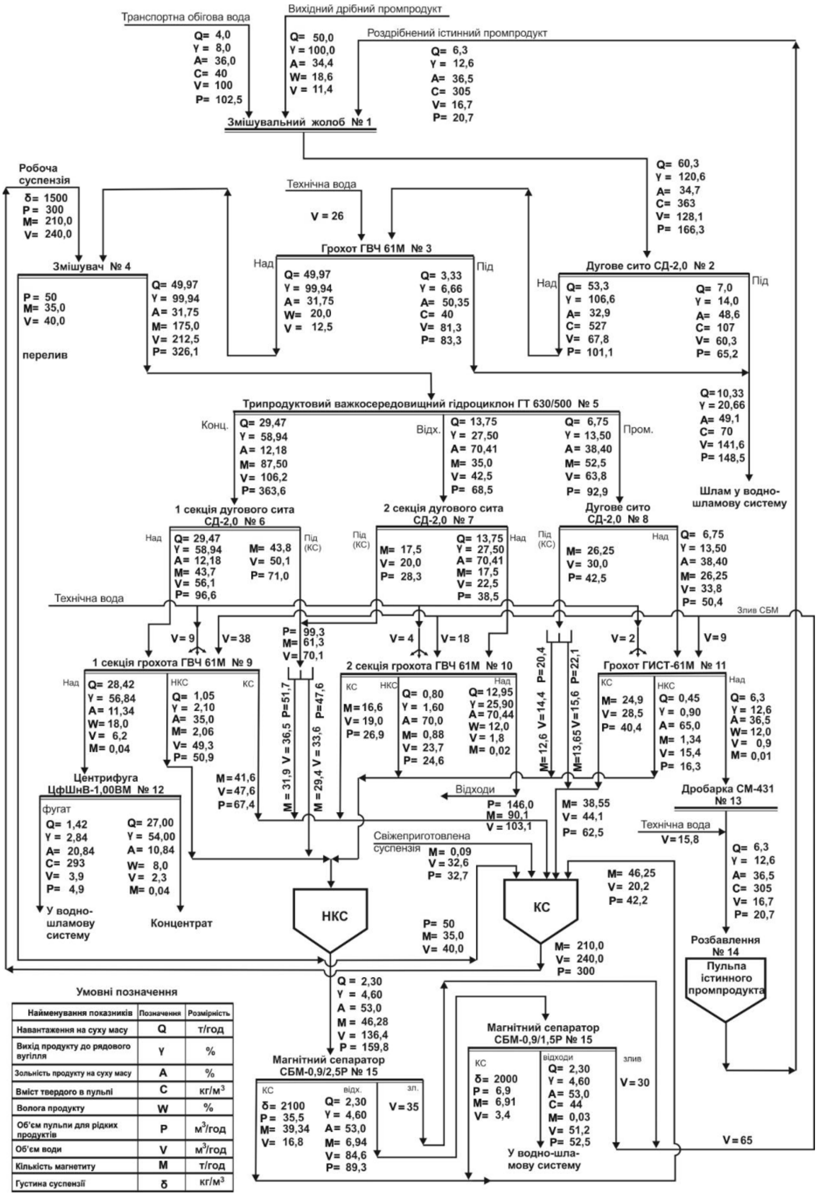
The ТВЗП-50 qualitative-and-quantitative and water-slurry flow sheet is shown in fig. 1; coal-cleaning balance and water balance are presented in the Table 1 and Table 2, respectively.

Table 1 ТВЗП-50 technology coal-cleaning balance

| No. | Product | Indices | | | |
|---|---|---|---|---|---|
| | | Feed, Q, t/h (on a dry weight basis) | Yield, γ, % | Ash content, $A^d$, % | Total moisture, $W^r$, % |
| 1 | Initial feed:<br>initial middling product<br>carrying circulated water | <br>50.0<br>4.0 | <br>100.0<br>8.0 | <br>34.4<br>36.0 | <br>18.6<br>40 кг/м$^3$ |
| | Total feed: | 54.0 | 108.0 | 34.52 | - |
| 2 | Concentrate | 27.0 | 54.0 | 10.84 | 8.0 |
| 3 | СД-2,0 sieve bend and screen ГВЧ 61М slurry (undersize) | 10.33 | 20.66 | 49.1 | 70 кг/м$^3$ |
| 4 | ЦфШнВ-1,00ВМ centrifuge effluent | 1.42 | 2.84 | 20.84 | 293 кг/м$^3$ |
| 5 | Rejects | 12.95 | 25.9 | 70.44 | 12.0 |
| 6 | Regeneration rejects | 2.3 | 4.6 | 53.0 | 44 кг/м$^3$ |
| | Total | 54.0 | 108.0 | 34.52 | - |

Table 2 ТВЗП-50 technology water balance

| No. | Loss with products | Water volume, V, m$^3$/h | System intake | Water volume, V, m$^3$/h |
|---|---|---|---|---|
| 1 | With concentrate | 2.3 | With the initial product | 11.4 |
| 2 | With slurry | 141.6 | Carrying circulated water | 100.0 |
| 3 | With centrifuge effluent | 3.9 | With freshly prepared medium | 32.6 |
| 4 | With rejects | 1.8 | Process water for washing | 41.0 |
| 5 | With regeneration rejects | 51.2 | Process water for dilution | 15.8 |
| | Total | 200.8 | Total | 200.8 |




| Найменування показників | Позначення | Розмірність |
|---|---|---|
| Навантаження на суху масу | Q | т/год |
| Вихід продукту до рядового вугілля | γ | % |
| Зольність продукту на суху масу | A | % |
| Вміст твердого в пульпі | C | кг/м³ |
| Волога продукту | W | % |
| Об'єм пульпи для рідких продуктів | P | м³/год |
| Об'єм води | V | м³/год |
| Кількість магнетиту | M | т/год |
| Густина суспензії | δ | кг/м³ |

| | |
|---|---|
| **Транспортна обігова вода** | Carrying circulating water |
| **Вихідний дрібний промпродукт** | Initial fine middling product |
| **Роздрібнений істинний промпродукт** | Crushed true middling product |
| **Змішувальний жолоб** | Mixing launder |
| **Робоча суспензія** | Work pulp |
| **Технічна вода** | Process water |
| **Змішувач** | Mixer |
| **Грохот** | Screen |
| **Над** | Oversize |
| **Під** | Undersize |
| **Дугове сито** | Sieve bend |
| **перелив** | Overflow |
| **Трипродуктовий важкосередовищний гідроциклон** | Three-product heavy-medium hydrocyclone |
| **Конц.** | Concentrate |
| **Пром.** | Middling product |
| **Відх.** | Rejects |
| **секція дугового сита** | Sieve bend section |
| **Шлам у водно-шламову систему** | Slurry to water-slurry circuit |
| **Під (КС)** | Undersize (KS) |
| **Злив СБМ** | SBM drain |
| **секція грохота** | Screen section |
| **НКС** | NKS |
| **КС** | KS |
| **Центрифуга** | Centrifuge |
| **Відходи** | Rejects |
| **Дробарка** | Crusher |
| **Свіжеприготовлена суспензія** | Freshly prepared pulp |
| **Фугат** | Centrifuge effluent |
| **У водно-шламову систему** | to water-slurry circuit |
| **Концентрат** | Concentrate |
| **Розбавлення** | Dilution |
| **Пульпа істинного промпродукта** | True middling product pulp |
| **Магнітний сепаратор** | Magnetic separator |
| **Злив** | Drain |

**Legend**

| Parameter | Denotation | Measurement unit |
|---|---|---|
| Load on a dry weight basis | Q | t/h |
| Product output against ROM coal | Y | % |
| Product ash content on a dry weight basis | A | % |
| Solids content in pulp | C | kg/m$^3$ |
| Product moisture content | W | % |
| Pulp volume for the liquid products | P | m$^3$/h |
| Water volume | V | m$^3$/h |
| Magnetite quantity | M | t/h |
| Pulp density | δ | kg/m$^3$ |

Diagram - Qualitative and quantitative flow-sheet of additional production of commercial products by middling product concentration

The Table 1 shows that yield of concentrate with ash content of 10.84% and moisture content of 8.0% makes 54.0% at initial middling product ash content of 34.4%. Rejects ash content makes 70.44%. Circuit water amount makes 200.8m$^3$/h.

Annual cost advantages of this technology implementation make over 15.9 million UAH. Capital payback period (82 million UAH) makes 0.51 year [3].

The ТВЗП-50 technology implementation at the specific plant requires use of existing equipment to the maximum extent practicable.

The Ukrniiugleobogashchenie institute developed recommendations for the ТВЗП-50 technology implementation in the number of coke plants, implementation performance expectations are resulted in the Table 3.

*Table 3*

| Index | Plant name | | | |
|---|---|---|---|---|
| | Krasnolimanskaya CPP | Chumakovskaya CPP | Uzlovskaya CPP | Oktyabrskaya CPP |
| Concentrate yield increase, % | 6.6 | 4.5 | 4.92 | 1.73 |
| Rejects ash content increase | 2.73 | 2.37 | - | 7.2 |
| New process implementation | WSS | WSS | - | GT |
| Additional equipment cost, million UAH | 8.4 | 8.8 | 7.3 | 21.4 |
| Cost advantages, million UAH | 81.59 | 299.8 | 118.82 | 132.76 |
| Capital payback period, month | 2 | 1 | 1 | 2 |

Thus, the ТВЖП-50 technology implementation at these coke coal-concentrating plants is technologically and economically sound.

Prior to the technology implementation, it is necessary to obtain more detailed baseline information as to ROM coal quality, potential re-engineering, equipment cost, commercial coal products and process materials prices, etc.

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# HEAVY-MEDIUM HYDROCYCLONES AS A SUBSTITUTE FOR JIGGING MACHINES

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**Abstract**

The use of heavy-medium hydrocyclones for small machine grade concentration as a substitute for hydraulic jigging machines will allow reduction of combustible matter losses with production rejects thereby increasing the yield of concentrate. This replacement is implemented primarily at plants where coarse machine grade is concentrated in heavy-medium separators. The replacement of jigging machines with heavy-medium hydrocyclones at the Pavlogradskaya CCPP and Oktyabrskaya CCPP showed the increase in ash content in rejects of 1-13mm size by 7.4% and 2.0%, respectively, which proves practicality of this direction of coal preparation development.

**Keywords**

Coal, grade, concentration, jigging machine, heavy-medium hydrocyclone, concentrate, rock, contamination, efficiency.

Review of process layouts of newly built coal preparation plants both in the Ukraine [1] and worldwide, as well as proceedings of XVI and XVII Coal Preparation Congresses [4, 5] demonstrate that heavy-medium hydrocyclones with magnetite medium are only in common use for concentration of small coal machine grades. Practicality of their use as a substitute for hydraulic jigging machines is supported by the known $E_{pm}$ values data for this equipment. For example, according to the data in [6], at the low separation density of 1600kg/m$^3$, difference in the $E_{pm}$ values between the Batak jigging machine and heavy-medium hydrocyclone makes 54 – 16 = 38kg/m$^3$ for 35-100mm size, 60 – 22 = 38kg/m$^3$ for 18-35mm size, 66 – 22 = 44kg/m$^3$ for 5-18mm size, and 96 – 40 = 56kg/m$^3$ for 0.5-5mm size. The same at the high separation density of 1800kg/m$^3$ makes 72 – 20 = 52; 80 – 26 = 54; 84 – 24 = 60; 120 – 46 = 74kg/m$^3$, respectively.

Thus, at the low separation density of 1600kg/m$^3$, combustible matter losses with production rejects in case of coal machine grade concentration in heavy-medium hydrocyclones as a substitute for hydraulic jigging machines are potentially reduced 3.37 times for 35-100mm size, 2.73 times for 18-35mm size, 3 times for 5-18mm size, and 2.4 times for 0.5-5mm size. The same at the high separation density of 1800kg/m$^3$ will be 3.6, 3.2, 3.5 and 2.6 times respectively.

Based on the above calculations, virtually 2-fold combustible matter loss reduction could be expected for small machine grade concentration at any separation density. Hence, replacement of the hydraulic jigging with heavy-medium hydrocyclones for the small coal concentration is reasonable from the technology viewpoint. Of course, this replacement must be conducted primarily at the coal preparation plants adopting heavy-medium separators for coarse machine grade concentration.



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Heavy-medium cyclone separation process involves separation density adjustment by alteration of minimum parameters number: initial pulp density, drain sleeve and lower orifice diameters. The other factors affecting the process (coal mixture and pulp pressure at the hydrocyclone inlet, feed solid to liquid ratio, etc.), remain constant [6, 7]. Separation process control could be readily implemented by means of automation facilities.

There are two common heavy-medium hydrocyclone types: two-product cyclone where the separation occurs at single density into two finished products, concentrate and rejects; three-product cyclone where the single initial pulp stream is separated at two densities into three finished products, concentrate, middling product and rejects [8, 9].

The small coal concentration in heavy-medium hydrocyclone finds ever-widening applications and offers following advantages: capability of exact density separation of difficult and very difficult coals with the high rock content at the minimum concentration products contamination with the foreign fractions; wide effectively washed coal size range from 0.15-0.2 to 40-50mm, capability of required separation density selection over the range from 1300-1350 to 2000-2200kg/m$^3$ and its adjustment by means of automation facilities; robustness as to variation of load and quantitative composition of concentrated raw material [10, 11].

The hydrocyclone unit implementation is characterized by the cascade equipment layout with gravity streams. Unit operation requires minimum operating personnel.

General-purpose heavy-medium hydrocyclones are suitable for concentration of coal, anthracite and shale. These are mainly used for concentration of small coking coal (separation into three products), small power coal and anthracite (separation into two products), small coal and anthracite jig middlings rewashing (separation into three and two products).

Heavy-medium hydrocyclones take advantages also at the separation of materials for production of high quality concentrates, for example, anthracitic concentrates for the electrode industry, low-sulphur concentrates, shale concentrate with the increased organic content.

In recent years, in the Ukraine the DTEK OOO specialists replace the hydraulic jigging machines with heavy-medium hydrocyclones at the small machine grade concentration stage as a part of improvement of process layouts of the coal preparation plants washing coarse machine grade in the heavy-medium separators [12].

In this regard, the experience is interesting of small machine grade concentration process switchover from the hydraulic jigging machines to heavy-medium hydrocyclones at the Pavlogradskaya CCPP and Oktyabrskaya CCPP.

For example, the Pavlogradskaya CCPP adopted heavy-medium hydrocyclones ТГЦ-1070 as a substitute for hydraulic jigging machines ОМ-24 for the small machine grade concentration, thus one jigging machine was replaced with one heavy-medium hydrocyclone.

The Pavlogradskaya CCPP small machine grade concentration technology performance data are presented in the Table 1.

The Table 1 shows that the use of heavy-medium hydrocyclones resulted in reduction of rejects contamination with -1800kg/m$^3$ fraction from 2.73% to 0.73%, and $E_{pm}$ from 70 to 30kg/m$^3$. At the same time, increase in load per equipment unit on the small machine grade basis made 60 t/h. After the modernization, reduction of combustible matter loss with coal preparation rejects resulted in the 1-13mm grade rejects ash content increase by 7.4% from 80.8% to 88.2%.

At the Oktyabrskaya CCPP, hydraulic jigging machines БОММ-16М were replaced with heavy-medium hydrocyclones В-33 of 840mm in diameter, thus one jigging machine was replaced with one heavy-medium hydrocyclone. Results of small machine grade concentration at the Oktyabrskaya CCPP are presented in the Table 2.

The Table 2 shows that the use of the heavy-medium hydrocyclones resulted in reduction of rejects contamination with -1900kg/m$^3$ fraction from 0.94% to 0.12%, and $E_{pm}$ from 82 to 25kg/m$^3$. After the modernization, reduction of combustible matter loss with coal preparation rejects resulted in the 1-13mm grade rejects ash content increase by 2% from 84.6% to 86.6%.

The Table 3 presents availability of coarse, small and fine machine grades concentration technologies at the Ukrainian coal preparation plants (T – heavy-medium concentration, O – jigging, F – flotation, N – grade concentration is not available).

Table 1 Pavlogradskaya CCPP small machine grade concentration data

| Density, kg/m$^3$ | Equipment | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | OM-24 jigging machine | | | | | | ТГЦ-1070 heavy-medium hydrocyclone | | | | | |
| | Products | | | | | | | | | | | |
| | Raw | | Concentrate | | Rejects | | Raw | | Concentrate | | Rejects | |
| | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % |
| -1500 | 44.93 | 11.5 | 78.03 | 11.5 | 0.6 | 11.8 | 59.14 | 6.3 | 98.32 | 6.3 | — | — |
| 1500-1600 | 5.40 | 27.1 | 9.37 | 27.2 | 0.1 | 27.2 | 0.59 | 27.8 | 1.0 | 27.8 | — | — |
| 1600-1700 | 1.61 | 30.4 | 2.59 | 30.3 | 0.3 | 31.4 | 0.86 | 35.0 | 0.36 | 33.1 | 0.05 | 28.4 |
| 1700-1800 | 1.64 | 77.5 | 1.58 | 77.3 | 1.73 | 78.4 | 1.17 | 83.7 | 0.22 | 86.0 | 0.68 | 84.6 |
| +1800 | 46.42 | 81.4 | 8.39 | 80.5 | 97.25 | 81.4 | 38.24 | 86.7 | 0.1 | 88.0 | 99.27 | 88.2 |
| **Total** | 100.0 | 46.2 | 100.0 | 20.3 | 100.0 | 80.8 | 100.0 | 38.3 | 100.0 | 6.4 | 100.0 | 88.2 |
| Yield of products, % | 100.0 | | 57.2 | | 42.8 | | 100.0 | | 60.1 | | 39.9 | |
| Load, t/h | 160-180 | | | | | | 220-240 | | | | | |
| $E_{pm}$, kg/m$^3$ | 70 | | | | | | 30 | | | | | |

Table 2 Oktyabrskaya CCPP small machine grade concentration data

| Density, kg/m$^3$ | Equipment | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | БОММ-16М jigging machine | | | | | | В-33 heavy-medium hydrocyclone, d=840mm | | | | | |
| | Products | | | | | | | | | | | |
| | Raw | | Concentrate | | Rejects | | Raw | | Concentrate | | Rejects | |
| | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % |
| -1400 | 42.5 | 6.3 | 88.5 | 6.3 | 0.00 | 0.0 | 57.5 | 6.1 | 98.14 | 6.1 | - | - |
| 1400-1500 | 1.6 | 23.9 | 1.7 | 25.2 | 0.23 | 10.9 | 0.4 | 29.4 | 1.06 | 29.4 | - | - |
| 1500-1600 | 0.4 | 33.2 | 0.4 | 33.5 | 0.14 | 13.9 | 0.1 | 35.5 | 0.32 | 35.5 | - | - |
| 1600-1700 | 0.7 | 41.1 | 1.3 | 41.3 | 0.19 | 25.9 | 0.1 | 43.8 | 0.24 | 43.8 | - | - |
| 1700-1800 | 0.5 | 43.3 | 7.0 | 43.5 | 0.16 | 42.2 | 0.1 | 47.9 | 0.22 | 48.1 | 0.02 | 45.1 |
| 1800-1900 | 0.3 | 52.2 | 1.1 | 52.8 | 0.22 | 51.6 | 0.1 | 62.2 | 0.03 | 61.6 | 0.10 | 62.2 |
| +1900 | 53.9 | 85.1 | 0.0 | 0.0 | 99.06 | 85.1 | 41.7 | 86.6 | 0.00 | 0.0 | 99.88 | 86.6 |
| **Total** | **100.0** | **49.8** | **100.0** | **10.3** | **100.0** | **84.6** | **100.0** | **39.9** | **100.0** | **6.6** | **100.0** | **86.6** |
| Yield of products, % | 100.0 | | 46.9 | | 53.1 | | 100 | | 58.4 | | 41.6 | |
| Load, t/h | 70 | | | | | | 80 | | | | | |
| $E_{pm}$, kg/m$^3$ | 82 | | | | | | 25 | | | | | |

Table 3 Characteristics of coal preparation plants

| No. | Plant name | Plant type | Main coal brands | Coal concentration processes | Limiting size, mm |
|---|---|---|---|---|---|
| 1 | Trudovskaya | CPP | Д | O-N-N | 13 |
| 2 | n.a. Chelyuskintsy | CPP | ДГ | O-N-N | 13 |
| 3 | Shakhterskaya | MPM | А | Jigging | 6 |
| 4 | Krasnolimanskaya | MPM | Ж, Г | T-O-F | 0 |
| 5 | Slavyanoserbskaya | MPM | ДГ, Г | O-O-Ts | 0.5 |
| 6 | Novopavlovskaya | MPM | А | Jigging | 6 |
| 7 | Partisanskaya | MPM | А | T-O-N | 6 |
| 8 | Privolnyanskaya | MPM | Д | O-N-N | 13 |
| 9 | Gorskaya | MPM | Г | O-N-N | 13 |
| 10 | Svyato-Varvarinskaya | CPP | К | T-T-F | 0 |
| 11 | Kievskaya | MPM | Ж | O-O-F | 0 |
| 12 | Uzlovskaya | CCPP | ОС, К, Т | T-O-F | 0 |
| 13 | Kalininskaya | CCPP | К, Ж, ОС | T-O-F | 0 |
| 14 | Komsomolskaya | MPM | Г, ДГ | T-O-F | 0 |
| 15 | Russia | MPM | Г | T-O-N | 0.5 |
| 16 | Krasnaya Zvezda | MPM | А | T-O-N | 0 |
| 17 | Dzerzhinskaya | MPM | Ж, К | O-O-F | 0 |
| 18 | Proletarskaya | MPM | К | O-O-F | 0 |
| 19 | Kondratevskaya | MPM | Т | T-O-F | 0 |
| 20 | Anthracite | CCPP | Т, А | O-O-N | 0 |
| 21 | Torezskaya | CCPP | А | O-O-N | 6 |
| 22 | Selidovskaya | MPM | Г | T-O-F | 0 |
| 23 | Chumakovskaya | CCPP | К | T-O-F | 0 |
| 24 | Stakhanovskaya | MPM |  | T-O-F | 0 |
| 25 | Gorlovskaya | MPM |  | O-O-F | 0 |
| 26 | Kolosnikovskaya | CCPP | ОС | O-O-F | 0 |
| 27 | Dobropolskaya | CCPP | Г | O-O-F | 0 |
| 28 | Oktyabrskaya | CCPP | Г | T-T-F | 0 |
| 29 | Mospinskaya | CCPP | Г | T-O-F | 0 |
| 30 | Kurakhovskaya | CCPP | ДГ, Г | T-O-N | 0.5 |
| 31 | Pavlogradskaya | CCPP | ДГ, Г | T-T-N | 0.5 |
| 32 | Pioner | CPP | ДГ | O-O-N | 0.5 |
| 33 | Rovenkovskaya | MPM | А | O-O-N | 6 |
| 34 | Vakhrushevskaya | MPM | А | T-O-N | 6 |
| 35 | Comendantskaya | CCPP | А | T-O-N | 0 |
| 36 | Sverdlovskaya | MPM | А | T-O-F | 0 |
| 37 | Krasnopartizanskaya | MPM | А | T-O-N | 6 |
| 38 | Tsentrosoyuz | MPM | А | T-O-N | 6 |
| 39 | Mikhaylovskaya | MPM | ДГ, Г | O-O-F | 0 |
| 40 | Belorechenskaya | CCPP | ДГ, Г | O-O-F | 0 |
| 41 | Krivorozhskaya | MPM | К | O-O-F | 0 |
| 42 | Luganskaya | MPM | ДГ | T-O-N | 0 |
| 43 | Samsonovskaya | MPM | Ж | O-O-F | 0 |
| 44 | Duvanskaya | CCPP | Ж | O-O-F | 0 |

Continuation of the Table 3

| No. | Plant name | Plant type | Main coal brands | Coal concentration processes | Limiting size, mm |
|---|---|---|---|---|---|
| 45 | Uglegorskaya | MPM | T | O-N-N | 6 |
| 46 | Donetskaya | CCPP | A | O-N-N | 6 |
| 47 | No. 105 | CPP | ДГ | O-N-N | 13 |
| 48 | Postnikovskaya | MPM | A | O-O-N | 6 |
| 49 | Krasnoluchskaya | MPM | A | O-N-N | 6 |
| 50 | Donugol-Alliance OOO | MPM | A | O-N-N | 6 |
| 51 | Intercom PNK OOO | MPM | A | O-N-N | 6 |
| 52 | Nagolchanskaya | MPM | A | T-O-F | 0 |
| 53 | Yanovskaya | CCPP | A, Г | O-N-N | 0 |
| 54 | Chervonogradskaya | CCPP | Г, Ж | T-O-N | 0 |
| 55 | AKHZ CPP | CPP | charge | O-O-F | 0 |
| 56 | Komsomolets Donbassa Mine CPP | CPP | T | T-N-N | 6(13) |
| 57 | DUT Mine CPP | CPP | Г | T-T-N | 0 |
| 58 | Sav-Plast OOO | CPP | coal-bearing material | N-T-N | 0 |

The above Table 3 shows that four coal preparation plants in the Ukraine are only meet the offered technology: Oktyabrskaya CCPP, Pavlogradskaya CCPP, Svyato-Varvarinskaya CPP and Donetsk Coal Fuel CPP. Concentration process at 23 coal preparation plants allows implementation of the offered technology.

It should be kept in mind that today’s coal heavy-medium concentration process could be carried out in two ways: with coal deslurrying or without it (in the dry state). Each method has its advantages and disadvantages. See [7, 8] for detailed information.

**Conclusions**. Introduction of heavy-medium hydrocyclones as a substitute for hydraulic jigging machines for the operation of small machine grade concentration is a principal direction of process layout improvement at coal preparation plants in the Ukraine allowing reduction of combustible matter loss with production rejects thereby increasing the yield of commercial coal products.

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## Application of Heavy Medium Cyclone in the SILESIA Coal Preparation Plant

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*Abstract*

Przedsiębiorstwo Górnicze SILESIA Sp. z o.o. (Polish Limited Liability Company) is a company, located in southern Poland, which main activity is excavation of hard coal. Since 2012 the Company has been producing hard coal intended for energy sector but also specific types of coal for retail sales. In December 2010 PG SILESIA took over 110-year-old Silesia mine from Polish coal giant - Kompania Węglowa S.A. Before that Silesia mine was supposed to be closed due to its unprofitability. During last four years investor has invested in mine's development about 250 min €. The existing Treatment Plant was also renovated, in which 10 mil € was invested. Currently PG SILESIA is one of the few mines in Poland which are effective, modern and safe.
In Poland in general the jigs are used for fine coal enrichment. The heavy media cyclones are not so commonly used. There were some trials for their implementation at some Polish mines in '70s of last century. Then they could not manage to run the industrial process. The main reason was changing amount of material delivered to treatment plant from several exploited coal seams.
Currently practically only PG SILESIA uses heavy media cyclones for enrichment. This is an example in which direction modernizations of many technologically old treatment plants should go to.

*Key words:* coal fine, heavy media cyclone, beneficiation

### Introduction

**PG SILESIA (Przedsiębiorstwo Górnicze "SILESIA" sp. z o.o.)** is a company, located in southern Poland, which main activity is excavation of hard coal. Since 2012 the Company has been producing hard coal intended for energy sector but also specific types of coal for retail sales. Except from hard coal, PG SILESIA also sales mine gas (methane), stone for building works and provides solutions referring to ecological storage of energy minerals (ash and ash mixes). Total balance resources exceed 500 million tons of low-sulphur coal, and balance resources of methane exceed 1,1 billion m3. Exploiting hard coal since 110 years “Silesia” mine can boast a rich mining tradition. Mine’s underground includes 2 production levels and 2 ventilation levels with the network of 39,7 kilometers of excavations.
Mine’s surface includes treatment plant, salt water reservoir in Kaniów and mechanical, electrical and shaft infrastructure. Underground is connected with surface by 5 shafts: 3 shafts for exploitation and transport purposes and 2 ventilation shafts with the maximum depth of 556 meters.

 
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Since mine's take over, it has been the priority to create a company investing in innovative solutions, while retaining the highest standards of safety.
PG SILESIA is a private company with foreign capital. The company is proud of the fact that a minority stake of shares in PG SILESIA belongs directly to the employees. PG SILESIA operates in accordance with the law and with respect for the principles of sustainable development. As a part of the local community, PG SILESIA supports the development of society, takes an active part in providing vocational and educational opportunities and in generally interpreted development of the region.
Coal seam is deposited in mining area called "Czechowice II", with an area of 21,36 km2 and it was accepted by decision of Ministry of Environmental Protection, Natural Resources and Forestry, dated 26.08.1994. Borders of mining area "Czechowice II" are the same as borders specified in concession number 162/94, dated 26.08.1994, issued by Ministry of Environmental Protection, Natural Resources and Forestry. The concession refers to excavation of coal and methane from the seam of Silesia Mine. Abovementioned concession, applying to excavations of coal and methane – as accompanying mineral - from mining area called "Czechowice II", was changed by Ministry of Environmental Protection, Natural Resources and Forestry on 23.07.1999. This change was connected with enlarging the mining area in two seams: 312 and 315 up to 21,85 km2 and enlarging the mining area up to 30,81 km2.
When it comes to geographical location of "Czechowice II" mining area, which includes coal seams "Silesia". In geological structure of „Silesia" seam, to a documentation depth of 1000 meters, there are involved Quaternary and Neogene deposits and productive carboniferous including following layers: "łaziskie" (group 200), "orzeskie" (group 300), "rudzkie" (group 400), "siodłowe" (group 500) i "brzeżne" (group 600). Balance seams, in amount of 45, occur in all stratigraphic layers and commercial seams, in amount of 20, occur only in „łaziskie" layers (group 200) and „orzeskie" layers (group 300). Thickness of balance seams is within 1,00-7,20 meters and commercial seams is within 1,20-3,50 meters. As of 31.12.2012 balance reserves of the seam amount to 505,175 mln Mg, where128,926 mln Mg is classified as commercial reserves. Additional mineral, which accompanies the hard coal in Silesia seam, is coal seam's methane. Balance resources of the methane are estimated to gain approximately 1,116 billion $m^3$. Output of the methane from surface holes and output from underground demethanization process was approximately 1,48 million $m^3$ in the year 2012 . It is fully managed by selling the methane to other entities. According to the data from 2012 absolute methane-bearing capacity of the mine amounts to approximately 35,21 $m^3$ per minute. Silesia Mine excavates the seams which are classified to 4th category of methane hazard, to the class B when it comes to coal dust explosion hazard and to 1st category of water hazard.

## 1. Coal beneficiation

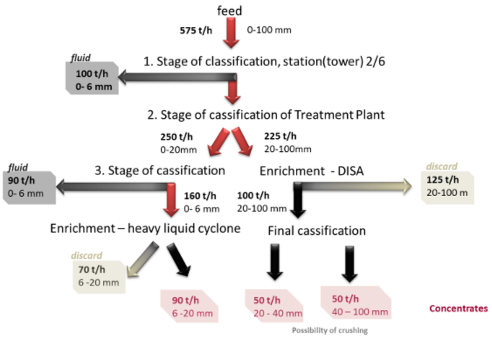
PG SILESIA aspires to produce the coal of high parameters. In years 2010-2011 Coal Processing Plant of the mine was modernized in area of enrichment of big coal sizing. The main aim was to use heavy media cyclones for fine coal enrichment instead of jig. PS Silesia produces power fine coal of high quality parameters.
The coal, offered by PG SILESIA is of high and stable quality parameters. PG Silsia gained it thanks to full automatisation of production process. The process is based on the most modern devices and systems of production parameters' stabilisation. It is also based on modern systems of quality constant control conducted on loading conveyors (with measurement scanning the whole plough).
High and stable quality of the coal in the seam, highly qualified personnel of Processing Plant, knowledge and competences, used technologies of also production regime and constant control of quality, guarantee high quality offered products.
Starting data for processing plant:

- Planned production - 575 t/h
- Annual production 3 000 000 t
- Granulometric analysis of raw output
- Densimetry analysis of raw output

- Analysis of crushing capability of coal and stone

Basic technological scheme of coal processing plant at PG Silesia is shown at Figure 1.

Figure 1. PG Silesia – technological scheme of coal processing




## 1. Dense-medium cyclone

**Assumptions for heavy liquid cyclones enrichment**

- Dry separation of raw fine coal class 0-6 mm
- Enrichment of fine coal class 6-20(30) and achiving following quality parapeters assuming maximum outcome not less than 90 t/h
  - Ash max 10%
  - Moisture max 9%
- Accuracy of enrichment not higher than Ep=0.06
- Production of stone class 6-20(30)
- Magnetite consumption not higher than 0.8 kg/t

Feed granular composition is shown at Fig.2, densitometric composition at Fig. 3.

Figure 2.

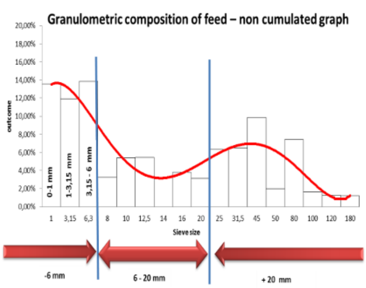
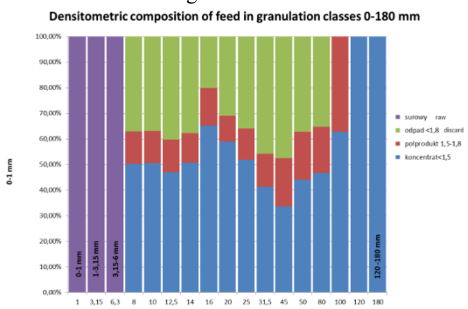



Figure 3.




PG Silesia uses heavy liquid cyclone type : MAX 660 – 20 - 0/B – A/200 VEX (Fig.1) with the following parameters:

- Diameter of cyclone: 660 mm
- Amount of cyclones in process: 2
- Graine size of feed 6 – 20 mm
- Performance of 1 cyclone. 80 t/h
- The slope of conical part: 20°
- Estimated density of heavy liquid: 1,35 to 1,55 t/m3
- Inner lining of the cyclone ceramic styles

HM (Fig.4) cyclones are powered using pomps with automatic pressure control system (Fig.5)

Fig.4. Heavy media cyclone in PG Silesia

Fig.5. Feed of HMC

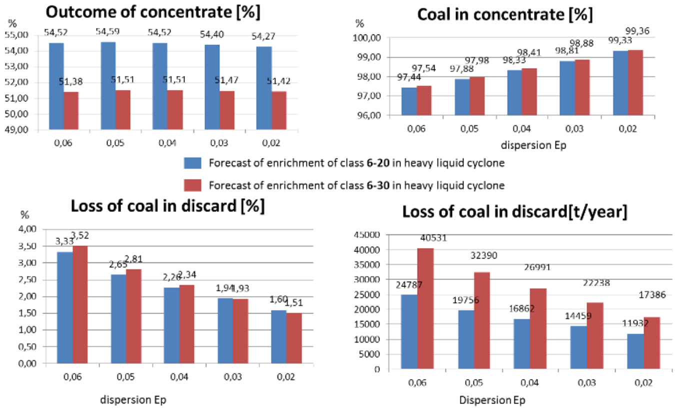
**Characteristic of heavy media cyclone separation - accuracy of separation**

Values of imperfections' indicators for heavy liquid cyclones vary within **0.04 – 0.12** min. assumption For comparison – in jiggers imperfection varies within **0.18-0.22**, even 0.31 – in older machines – which means worse accuracy in separation – bigger loss of coal. As it is summarized in the literature, dense-medium gravity/centrifugal separators are widely used in the mineral processing industry for classifying particles by density and size.

**Technical analysis and technical suggestions**

The results of HMC separation are shown at Fig. 6.

Fig. 6. The results of HMC separation




From all the accounts of previous experience in the cleaning of fine coal by heavy medium, the magnetite medium emerges as the main parameter. For efficient operation of the separating cyclone, the medium must be kept in a clean, uncontaminated state. A build-up of fine coal or fine clay in the circulating medium increases the viscosity of the medium and negatively influences the separation in the cyclone.

It has also been found that fine pyrite may accumulate in the media. Very fine pyrite, being non-magnetic, reports to the cyclone overflow and contaminates the product coal, increasing the ash content. The coarser pyrite, by virtue of it's high density, reports to the cyclone spigots and becomes 'part' of the circulating medium.

It appears that for low-density separations in small diameter cyclones, very fine medium is required. One reason for this may be the fact that in order to achieve a low relative desity cut point, a low differential is needed in the cyclone. Coarser medium classifies according to size in the cyclone and this results in a high density of separation being achieved.
The finer the magnetite, the smaller this ‘density shift’ becomes.

The washability data from a 20mm x 0.5mm size fraction of raw coal were evaluated and show at Table 1. The data show that the ash content of this coal can be reduced from 34.0 percent to 9.0 percent by washing in a heavy-media cyclone at 1.725 sp.gr.; the actual yield of clean coal would be 63.1 percent. This product would meet compliance limitations of 500 g of $SO_2$/GJ

Table 1. Comparison of the results of jig and dense medium cyclone. Size fraction 20mm-0,5 mm

| | Raw Coal | Jig | | | Dense Medium Cyclone | | |
|---|---|---|---|---|---|---|---|
| Partition sp gr | - | 1,375 | 1,425 | 1,550 | 1,525 | 1,625 | 1,725 |
| Yield, % | 100,00 | 33,4 | 37,9 | 47,5 | 57,4 | 60,7 | 63,1 |
| Ash, % | 34,0 | 7,5 | 8,0 | 9,1 | 7,0 | 8,0 | 9,0 |
| Sulphur % | 0,65 | 0,63 | 0,64 | 0,65 | 0,63 | 0,65 | 0,66 |
| Calorific Value MJ/kg | 17,0 | 27,0 | 26,5 | 47,5 | 27,0 | 26,5 | 26,0 |
| Emission $gSO_2$/GJ | 765 | 470 | 480 | 500 | 470 | 480 | 500 |

Cost comparison of Jig vs Dense-Medium Cyclone for 20 x 0.5 mm size fraction show clean coal prodution cost 75,9 USD/ton HMC vs 57,7 USD/tonne
Based on the assumption that the new preparation plant will use heavy media cyclones as the primary washer, sources of domestic magnetite were sought. One potential source of inexpensive heavy-media was located at the metallurgical plant near the town of Jaworzno where a waste product (mill scale) appears to have the physical and magnetic properties required for coal washing in cyclones [9].

**Remarks on dense medium cyclone**

Heavy-medium cyclone beneficiation of fine coal offers the following benefits:
- sharp, efficient, separation of fine coal is possible,
- low relative density cutpoints can be achieved,
- the density of separation can be controlled,
- oxidised or weathered coal can be processed,
- oversize particles can be handled,
- the process is flexible and suited to almost any type of coal,
- low (at PG Silesia) about 200 g/tonne) magnetite consumption,
- simply magnetite recovery,
- the medium density is easy to control,

**Quality parameters of HMC control**

PG SILESIA owns a laboratory where daily analysis of ongoing production are conducted. At the moment PG SILESIA offers measuring the samples (while loading and from production) by independent institutions as J.S. Hamilton Poland Ltd and SGS Eko-Projekt Sp.z o.o.
System for dispatch supervision and production control in a coal preparation plant was implemented in 2012. The supervision system to monitoring and control mechanical coal preparation processes, as well as

to prepare and deliver quality-classified power blends, encompasses the following functions: — complex (or partial) automation of coal preparation lines and a water- and slurry-cycle devices, — automation of the process of preparation and delivery of quality-classified power blends, — advanced monitoring and control systems for heavy-industry technological processes and for devices working within the technological line of a plant.

## Conclusions

There were many discussions on how to solve the technological and machinery systems for coal enrichment. The conceptions of replacing existing point for fine coal enrichment in jigs, implementation of air enrichment on the concentration table type FGX and enrichment in heavy media cyclones were considered. As a result of discussions and achieved results the heavy media cyclones were introduced to the existing technological system in PG Silesia. In Poland in general the jigs are used for fine coal enrichment. The heavy media cyclones are not so commonly used. There were some trials for their implementation at some Polish mines in '70s of last century. Then they could not manage to run the industrial process. The main reason was changing amount of material delivered to treatment plant from several exploited coal seams. Currently practically only PG SILESIA uses heavy media cyclones for enrichment. This is an example in which direction modernizations of many technologically old treatment plants should go to.

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# INTENSIFICATION THE SEDIMENTATION PROCESS OF COAL SUSPENSION

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## ABSTRACT

This paper presents the research on the sedimentation process of coal suspension. The main purpose of the research presented in this article is to investigate and understand the intensification of the sedimentation process of a coal suspension. It summarizes the impact of particular techniques for intensifying the sedimentation process, shows the possibility and high efficiency of the use of intensification techniques of sedimentation process: multiflux fillings conjugated with the process of flocculation. The paper presents an influence of concentration of the coal suspension on degree of compression, velocity of intensification process, and basically on the effectiveness of the sedimentation process. The experiments were done in the static measurements laboratory stand equipped in mechanical system controlling a digital camera that registers the sedimentation process, and in software analyzing the image in order to plot sedimentation curves. Experiments were done for two set ups of measuring cylinder: vertical and inclined, imitating (simulating) respectively the conventional sedimentation process and the multiflux sedimentation. Performed measurements indicate the possibility and justification of using the multiflux package together with flocculation process.

## KEYWORDS

Sedimentation process, multiflux, coals suspension, gravity concentration methods, flocculation, intensification techniques, image processing

## INTRODUCTION

Sedimentation is the separation of solids from liquids by use of the phenomenon of sinking solid particles in a liquid under the action of gravity. Thus, sedimentation process is used in various production sectors, such like: chemistry industry, food processing industry, as well as processing of natural mineral resources in mining. It's well known that the sedimentation process is beneficial from energetic point of view, the low need for an energy is caused by necessity of aiding low flow of the suspension, as well as to subdue the resistance movement during scrapping of the sludge [Kowalski, 2004]. The biggest drawback of this process is its very low speed, therefore there is a need to use a various techniques of intensification of the sedimentation process. One of the technique of supporting the sedimentation processes is so-called 'shallow sedimentary', which is realized in multiflux fillings [Kowalski, 2004]. This processes can be applied in conventional settling tanks by filling them, either partially or entirely, with multiflux fillings.

The other intensification techniques of sedimentation process of coal suspension is flocculation [Bürger, 2005]. The flocculation process is used mainly in the case of suspensions with a relatively high concentration. The purification process of suspension supported with the addition of flocculant allows the use of devices with much less sediment surface. The main purpose of the research presented



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in this article is to investigate an influence of concentration of coal suspension on the sedimentation rate of thickening and the use of automatic test sediment in order to eliminate time-consuming of plotting a sedimentation test in a classical way, the subjectivity of reading the height level of the liquid surface. . In this paper, we present the influence of some of intensification techniques in order to improve the speed of the process, and to increase the effectiveness of the coal suspension sedimentation process.

## PHYSICAL PROPERTIES OF RESEARCH MATERIAL

The suspension used in this research came from coal processing factory. Samples of the investigated suspension were collected from the water-mad system of coal enrichment plant, before flocculant dosing system. For each collected sample the standard procedure was to mark the concentration of the suspension. Content of the solid state fraction in the suspension was marked with the drain method. In order to eliminate measurement errors were made a series of measurements of concentrations of the suspension. The first part of the study was collected a suspension of three different concentrations to determine the effect of concentrations of the suspension at the rate of sedimentation process.

Table 1. Concentrations of coal suspension

| Suspension | S1 | S2 | S3 |
|---|---|---|---|
| Concentration [kg/m$^3$] | 65,87 | 41,73 | 34,94 |

## PARTICLE SIZE OF SUSPENSION

Measurements of particle size distribution of the coal suspension were performed using a laser diffractometer Mastersizer 2000, Malvern. Method of measurements using a laser diffractometer was discussed in detail in the literature [Kowalski, 2012]. A detailed description of the analyzer and tests methodology of its use is given in manual [Malvern]. [Kowalski, 2012] i [Banaś, 2012].

Analyzing the results of determinations of feed particles size [Fig. 1] (especially graph of frequency) can be explicitly stated that the analyzed suspension is a suspension fine-grained, in which half the particles have a size below 11 microns, which means that the sedimentary section is necessary to clumping small particles into larger particles, such as through the use of flocculation techniques [Kowalski 2013].

Designated suspension particle size parameters are as follows: $d_{10}$ = 1.512 µm, $d_{50}$ = 10.838 µm, $d_{90}$=65.153 µm.

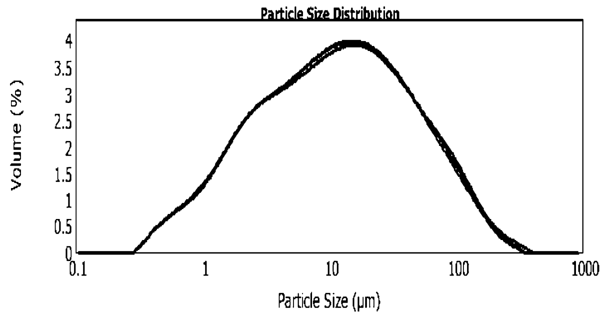


Fig. 1. The distribution of particle size in suspension (feed)

## AUTOMATED STATIC TESTS OF COAL SUSPENSION

The research of static sedimentation are based on performed sedimentation test of suspension, which we obtain by plotting the sedimentation curve of suspension [Kynch, 1952]. Implementation of the static test is possible for the suspensions with a relatively high concentration, suspensions for which there is a constrained sedimentation. [Orzechowski, 1990]. For suspension used in the study of the solid volume fraction is 1.85% (for tests with differential doses of flocculant and for shallow-sedimentary process).

During the constrained sedimentation process, due to the occurrence of zonal sedimentation, in suspension are formed zones of different concentrations, in particular case is formed the clear liquid zone, that is clearly separated from the suspension layer. Execution of the suspension sediment test depend on the height of the boundary separation between pure liquid and a first zone of thickened sludge, depending on the time. On the basis of the sedimentation curve we can determine for example the speed of sedimentation of the suspension concentration, maximum attainable level of compression, as well as we can determine the size of the settler needed to obtain a pure overflow and underflow with the assumed concentration. [Bandrowski, 2001]. Implementation of the static tests took place on a laboratory designed to study the process of static sedimentation.

The biggest problems of classical sedimentation tests are their time-consuming and subjectivity of reading the liquid surface level [Kowalski, 2015].

In order to overcome these problems, proposed execution of laboratory equipped with a digital camera with a system of automatic conversion of images to determine the level of the liquid surface. The scheme of the automatic sedimentation test and conception of automated test are shown below (Fig. 2).

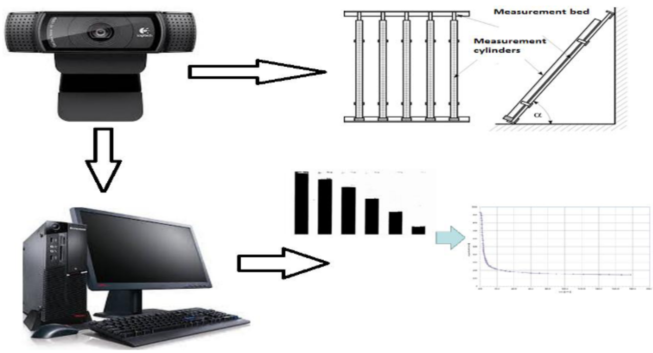


Fig. 2. Scheme of automated sedimentation test.

## STATIC TESTS WITH DIFFERENT CONCENTRATION OF COAL SUSPENSION

Sedimentation test for suspension with three different concentrations: S1, S2 and S3, were carried on the automated sedimentation test workplace. Sedimentation curves for test with increasing concentration of coal suspension are shown on Fig. 3. Sedimentation curves obtained clearly show that with increasing concentration of the suspension increases the slope of curves sedimentation and thus the rate of sedimentation process coal suspension.

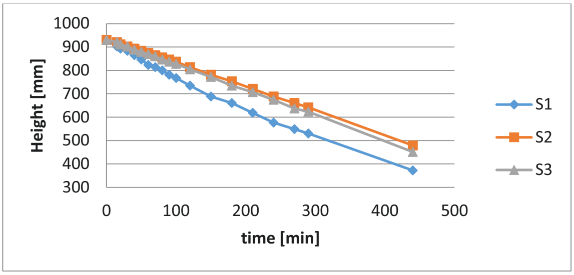


Fig. 3. Sedimentation curves for test with increasing concentration of coal suspension

STATIC TESTS OF COAL SUSPENSION WITH INTENSIFICATION TECHNIQUES OF SEDIMENTATION PROCESS

The next stage of the study was to determine the effect of intensifying the process of sedimentation to increase the rate of sedimentation process. They were used two main techniques intensification of the process: the use of multiflux fillings and the addition of flocculant.

The research was conducted for two settings of the measuring cylinder - vertical and inclined. A vertical cylinder was implemented as a process of sedimentation occurring in the device without multiflux filling, while the inclinded cylinder was carried out as a sedimentation process in the settling tank with multiflux filling. In this research setting of the inclined cylinder is adopted at an angle of 60° relative to the ground - as a typical angle of the mulitflux filling duct. Measurements of raw suspension are a reference point for subsequent measurements with different doses of flocculant (Fig. 4).

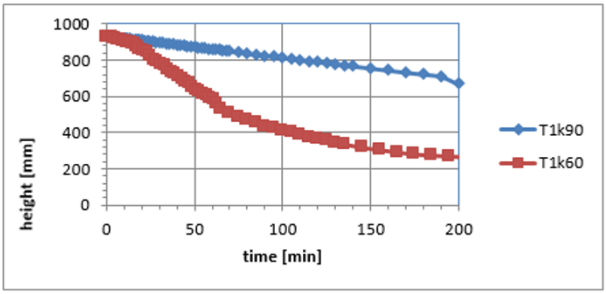


Fig. 4. Sedimentation curves for conventional sedimentation (T1k90) and with multiflux fillings (T1k60)

The purpose of the performed sedimentation tests was to estimate the influence of flocculant dose on the velocity of the investigated suspension was well as to estimate the maximum level of compression during sedimentation process in the shallow sedimentation system and in the classical system (without multiflux filling). Tests were performed for two settings of the measuring cylinder: vertical and inclined. In the vertical cylinder the sedimentation process was performed like in the device without the multiflux filling, whereas in the inclined cylinder the sedimentation process was performed like in cylinder system is faster than for the cylinder in a vertical arrangement, both in systems with and without flocculant.

## SUMMARY

Results from the experimental data clearly show the influence of each intensification technique on the improvement of the effectiveness and the increase in the speed of the process. The use of the multiflux fillings caused an increase in the speed of the sedimentation. The fluctuation process has also a big influence on the intensification of the sedimentation. The use of the flocculant increased the speed of the sedimentation; the sedimentation speed increased repeatably with the increase in addition of the flocculant. According to theory, an increase of flocculent doze (in range of performed tests) causes an increase in sedimentation rate. This relationship is valid in the investigated range of the floccualnt addition, with increasing concentration of coal suspension increases the speed of the sedimentation process. Also compression level is much higher with the use of flocculation and multiflux fillings then in classical sedimentation process.

The speed of the sedimentation achieved by the simultaneous use of both techniques is several hundred times higher than in case of the classical sedimentation without applying the intensification techniques. Performed measurements indicate the possibility and justification of using the multiflux package together with flocculation process.

The use of automatic test sediment allowed to exclude measurement errors caused by the difficulty in determining the precise position of the interfacial and led to a very large saving time during execution of sedimentation tests.

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# PERFORMANCE EVALUATION PRACTICES AT DENSE MEDIUM SEPARATION CIRCUITS

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**ABSTRACT**

In this paper, the application of density tracers for performance prediction of dense medium circuits and the strategies for identification of potential problems in operating plants are outlined, and the results obtained at various performance prediction and evaluation studies performed in various coal washing plants are summarized. Based on the performance data and their interpretation, the potential problems in existing operations are identified, and strategies and comments are given for optimum operation. The performance prediction tests using density tracers enabled the identification of inadequacies in operating dense medium drums, the quantification of retention phenomena in dense medium cyclones and indicating the importance of equally distribution of flows. Additionally, the results of the density tracer tests down to 2mm are given which is valuable being one of the very few studies ever conducted at such fine sizes.

**Keywords:** dense medium separation, coal, density tracers, performance, optimization, dense medium cyclones, dense medium drums

## 1. INTRODUCTION

Dense medium separation being the most widely applied coal beneficiation method, incorporates low or high inefficiencies depending on various design and operating conditions similar to any other separation technique (Khoury, 1981; Burt, 1984). An efficient coal washing operation requires periodic performance evaluation studies in order to investigate the effects of any change in any input parameters such as run-of-mine coal washability, throughput, medium quality (grade and particle size), medium flowrate, screen apertures etc. As a result, the effects of these changes on coal recovery, coal loss, the performance of individual separation equipment and overall plant performance could be quantified and hence, techno-economical optimum operating conditions could be identified (Lynch and Napier-Munn, 1986).

 
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The conventional performance prediction technique requires heavy liquid tests conducted on various size fractions of samples taken from feed and product streams of each equipment. However, the completion of the tests takes at least weeks depending on the number of equipment in the plant and therefore this method requires too much manpower and time. Unfortunately, the performance data becomes old as soon as it is obtained.

The use of density tracers, on the other hand, for performance prediction in dense medium circuits offers advantages over conventional method. The performance curve of a dense medium separation equipment could be obtained in approximately 1-3 hours. Hence, the density tracers offer very reliable results in a very short time and is much less costly than conventional performance prediction method (Wood et. al, 1989, Wood, 1990). Although initial density tracer practices date back to 1950s by Visman (Wood, 1990), the detailed and systematic approach was developed in late 1980s (Davis, 1987; Scott, 1988; Wood et. al, 1989, Wood, 1990, Clarkson and Wood, 1993).

Another advantage of density tracers is that they provide some additional information which cannot be obtained with conventional sink-float tests such as the "retention phenomena" in dense medium cyclones. "Retention" is the accumulation of relatively coarse near-density particles due to the medium density differential between underflow and overflow streams. Depending on the amount of these particles in the feed, as the accumulation continues, there comes a point when the hydrodynamics inside the cyclone is deteriorated and all the slurry inside the cyclone whether heavy or light is surged via the apex. Depending on the frequency of surging, the clean coal loss to tailings may reach 5%. The effect is more pronounced in cyclones with smaller apexes [Wood, 1990].

In this paper, the performance data of dense medium vessels, drums and cyclones obtained during plant performance evaluation studies at dense medium separation circuits are outlined. Comments and suggestions towards optimization are put forward for possible problems that were identified. The plant audits using density tracers were conducted at three coal washing plants of Turkish Coal Enterprises (TKI) which is the largest coal mining company in Turkey.

## 2. PLANT PRACTICES

During the plant performance prediction and evaluation studies, three coal washing plants (Ömerler, Tunçbilek and Dereköy plants) of TKI were audited and the performance data were obtained density tracers (Gülsoy et.al, 2013; Orhan et.al, 2014). The flowsheet of the three plants (Figure 1) were similar in terms of production targets and the feed size ranges subjected to coarse and fine dense medium separation circuits. The coarse clean coal (-150+18mm) is produced for house-hold heating purposes, the fine clean coal (-18+0.5mm) is utilized in mainly sugar and cement production plants and the middlings of both coarse and fine circuits are sent to thermal power plant for electricity generation. The main difference of the circuits is in the coarse coal beneficiation circuit where Ömerler plant has two stages of dense medium vessels (Peters type), Tunçbilek plant has two stages of dense medium drums and Dereköy plant has a two-compartment Wemco drum separator.

Density tracers were used in plant performance prediction studies at each coal washing plant (Figure 2). The density tracers were supplied at various densities in the range 1.30 to 2.15 g/cm$^3$ and at sizes of 64, 32, 16, 8, 4 and 2mm from Partition Enterprises Ltd. In a typical application of density tracers for performance prediction of a dense medium separator, pre-determined number of each density/size class of tracers are added to the feed stream of the separator and collected from all product streams. Generally, the drain and rinse (D&R) screens are the most suitable collection points where the tracers are collected from the end of the screens before they leave the screen (Figure 3).

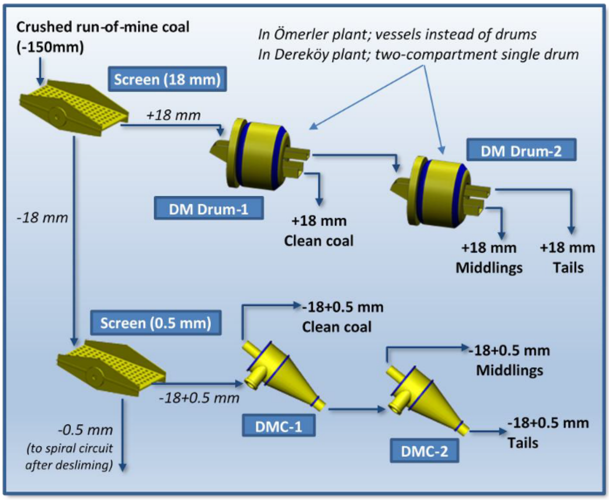



Figure 1. The dense medium separation circuit flowsheets of Ömerler, Tunçbilek and Dereköy plants

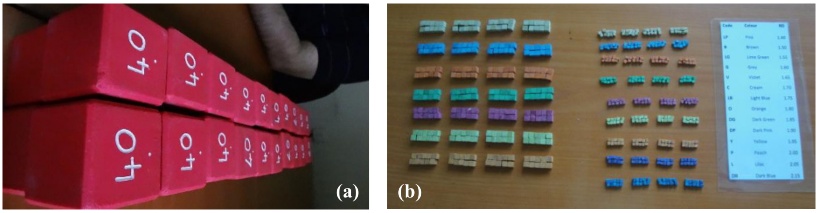



Figure 2. (a) 64mm and (b) 4 and 2mm, density tracers

Figure 3. The collection of density tracers on drain/rinse screens

After the tracers are collected, they are grouped and the number of tracers at each size/density class is recorded. The determination of the partition numbers are straight forward and obtained as the number of tracers of a size/density class in the sinks product divided by that in the feed. The partition numbers are then used to plot the partition curve of the separator. The performance of a separator can commonly be indicated by Ep (Ecart probable) which is given as $Ep = \frac{\rho_{75}-\rho_{25}}{2}$ where, Ep is Ecart probable, $\rho_{75}$ and $\rho_{25}$

are specific gravity values where the partition coefficients are 75% and 25% respectively. The Ep and $\rho_{50}$ values can also be estimated using a suitable model such as JKMRC model which is given as;

$$Y_i = \frac{1}{1 + exp\left[\frac{ln3(\rho_{50} - \rho_i)}{Ep}\right]}$$

where, $Y_i$ is the partition number indicating the fraction with density $\rho_i$ in the feed reporting to sinks after separation and $\rho_{50}$ is the density of separation (cut-density). The JKMRC model has been used successfully in numerous studies on modelling and simulation aided design and optimization of dense medium separation circuits (Napier-Munn, 1991). In the partition curves throughout this paper, the data points represent actual partition coefficient values obtained with density tracers whereas the solid curves represent the fit of data to JKMRC model.

### 2.1 Dense Medium Vessels and Drums

All three coal washing plants of concern are operating in order to obtain a coarse (-150+18mm) clean coal and dense medium vessels (in Ömerler plant) and dense medium drums (in Tunçbilek and Dereköy plants) are applied for the coarse coal cleaning.

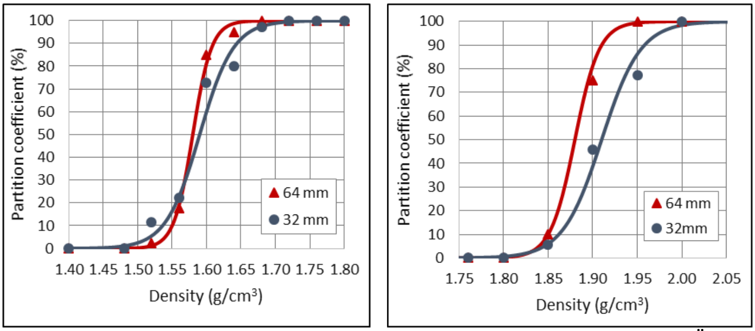


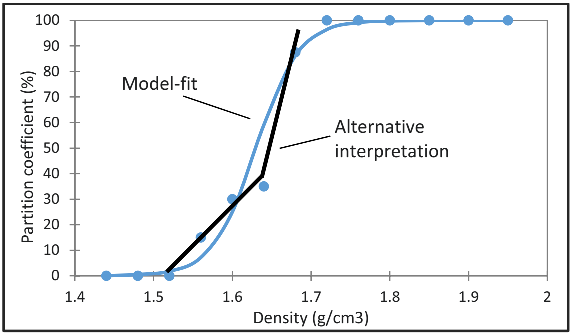

Figure 4. The partition curves for (a) primary and, (b) secondary dense medium vessel at Ömerler plant.

Table 1. Performance data for dense medium vessels at Ömerler plant.

| | Primary | | Secondary | |
|---|---|---|---|---|
| **Parameter** | **64 mm** | **32 mm** | **64 mm** | **32 mm** |
| Medium density, $\rho_m$ | 1.543 g/cm$^3$ | | 1.859 g/cm$^3$ | |
| Cut density, $\rho_{50}$ | 1.580 g/cm$^3$ | 1.590 g/cm$^3$ | 1.880 g/cm$^3$ | 1.910 g/cm$^3$ |
| Ep | 0.015 | 0.027 | 0.015 | 0.024 |
| Cut-point shift, ($\rho_{50}$ - $\rho_m$) | 0.037 g/cm$^3$ | 0.047 g/cm$^3$ | 0.021 g/cm$^3$ | 0.051 g/cm$^3$ |

As can be seen in Figure 4 and Table 1, the Ep values of separation show that a good separation is achieved at Ömerler coal washing plant. The cut-point shift values (the difference between cut-density and medium density) are determined to be between 0.02 to 0.05 g/cm$^3$ (King, 2001).

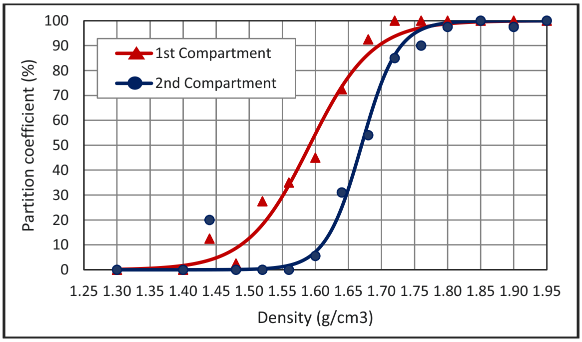
In Figure 5, the performance curve of the primary drum separator at Tunçbilek plant is given. The data indicates an Ep value of 0.030 which is slightly high for 64mm size. Another issue is that although actual data reveals a curve with two distinct slopes, the partition curve model fails to present this. Actually, the separation performance of the equipment is considerably higher for high density particles and lower for the low density (clean coal) particles. Although, Ep phenomena and the models based on it give an idea about the performance of a separator, they fail to represent the independently moving tails of a partition curve.




| Parameter | Value |
|---|---|
| Tracer size | 64 mm |
| Cut density, $\rho_{50}$ | 1.630 g/cm$^3$ |
| Ep | 0.030 |

Figure 5. The partition curve of primary drum at Tunçbilek plant

The low performance at low density range arises when the feed material contains high amount of low density material and the drum fails to effectively remove them. A dense medium drum has lifter plates to remove the sinking high density particles, however does not have a built-in mechanism for the floating particles. They have to let themselves out with the aid of the carrier medium and of the pushing of other floating particles in the drum. This mechanism obviously is not as effective as the lifter plates for the sinking particles. As the floating particles build up within the drum, some of them are easily caught by the lifter plates and taken out with the high density material. This occurs when the amount of floating material exceeds the amount that the drum lip length will allow. A similar case is observed at the two-compartment drum at Dereköy plant (Figure 6). The performance curve of the 1$^{st}$ compartment shows that the high density tail of the partition curve is significantly steeper than the low density tail which results in loss of clean coal particles to sinking product of the 1$^{st}$ compartment. These particles could easily be recovered in the 2$^{nd}$ compartment. The existence of the issue could also be identified by comparing the Ep values which is higher for the 1$^{st}$ compartment, although the opposite would be anticipated due to medium density (hence viscosity) increase in the 2$^{nd}$ compartment.




| Parameter | Value |
|---|---|
| Tracer size | 64 mm |
| 1$^{st}$ Compartment | |
| Cut density, $\rho_{50}$ | 1.592 g/cm$^3$ |
| Ep | 0.052 |
| 2$^{nd}$ Compartment | |
| Cut density, $\rho_{50}$ | 1.670 g/cm$^3$ |
| Ep | 0.030 |

Figure 6. The partition curves of two-compartment drum at Dereköy plant.

### 2.2 Dense Medium Cyclones

Detailed performance prediction tests were carried out at Ömerler plant which contains three 900 mm cyclones at the primary stage and two 700 mm cyclones at the secondary stage. The combined partition curves of the dense medium cyclones (DMCs) are given in Figure 7.

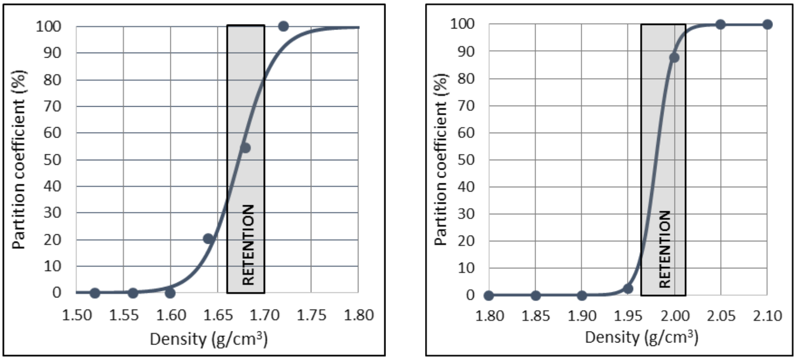



Figure 7. The partition curves of the primary and secondary DMCs with 32mm density tracers

Table 2. Performance data for primary and secondary DMCs with 32mm tracers

| | **Primary DMCs** | **Secondary DMCs** |
|---|---|---|
| Feed medium dens. ($\rho_m$) | 1.579 g/cm$^3$ | 1.908 g/cm$^3$ |
| Overflow med. dens. ($\rho_o$) | 1.513 g/cm$^3$ | 1.905 g/cm$^3$ |
| Underflow med. dens. ($\rho_u$) | 1.645 g/cm$^3$ | 2.033 g/cm$^3$ |
| Cut density, $\rho_{50}$ | 1.673 g/cm$^3$ | 1.980 g/cm$^3$ |
| Ep | 0.021 | 0.010 |
| Retained density range | 1.66-1.70 g/cm$^3$ | 1.96-2.01 g/cm$^3$ |
| Cut-point shift, ($\rho_{50} - \rho_m$) | 0.094 g/cm$^3$ | 0.072 g/cm$^3$ |

The Ep value of the primary stage being 0.021 is significantly higher than that of the secondary stage being 0.010. The lower efficiency of the primary cyclones is due to the failure of the feed distributor in equally distributing the feed to each of the three cyclones. When the feed and medium are pumped from the feed tank to the cyclones, the slurry passes through a distributor which is supposed to equally divide the flow to three parallel cyclones. However, the number of tracers obtained from each cyclone shows that approximate distribution to the cyclones is as 57% to DMC-1, 14% to DMC-2 and 29% to DMC-3. The lower efficiency of the primary cyclones is due to the failure of the feed distributor in equally distributing the feed to each of the three cyclones which results in obvious decrease in efficiency.

In the determination of the retention in cyclones the following procedure was followed. The tracers continued to be retrieved for another 5 minutes after the introduction of the last tracer to the feed. Considering that the residence time of the tracers being 1.5-2.0 minutes in the system, the tracers subjected to separation would have enough time to report to either product of the equipment. After 5 minutes the cyclone pumps were turned off and the material inside the cyclones are taken from the D&R

screen of the underflow product. The tracers obtained as such are collected separately. The shaded area in Figure 7 shows the densities of the retained tracers.

Another tracer test was conducted to achieve the performance data at finer sizes, namely, 16, 8, 4 and 2mm. As it is not possible to collect very fine tracers (such as 2mm) between the coal particles, this test was conducted at feed-off conditions and the coal feed to the cyclones was turned off, letting only the medium flow through the cyclones. As soon as the tracers are observed on the D&R screen, the medium pumps and the screen were turned off so that the tracers would stay on either product (overflow or underflow) drain and rinse screens. On top of each screen, 16 and 8mm tracers were collected by hand and as 4 and 2mm tracers were magnetic, they were collected using bar magnets. The partition curves obtained during the test are given in Figure 8.

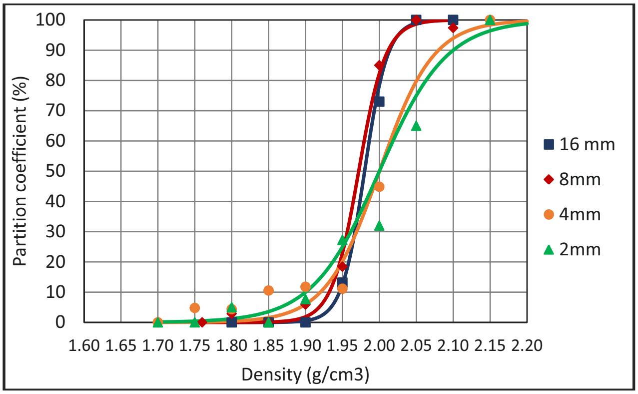



Figure 8. Partition curves of secondary DMCs with 16, 8, 4 and 2mm density tracers

Table 3. Performance data for secondary cyclones with 16, 8, 4 and 2mm density tracers

| **Parameter** | **16mm** | **8mm** | **4mm** | **2mm** |
|---|---|---|---|---|
| Medium density, $\rho_m$ | 1.882 g/cm$^3$ | | | |
| Cut density, $\rho_{50}$ | 1.980 g/cm$^3$ | 1.972 g/cm$^3$ | 2.000 g/cm$^3$ | 2.000 g/cm$^3$ |
| Ep | 0.017 | 0.020 | 0.040 | 0.050 |
| Cut-point shift, ($\rho_{50} - \rho_m$) | 0.10 g/cm$^3$ | 0.09 g/cm$^3$ | 0.12 g/cm$^3$ | 0.12 g/cm$^3$ |

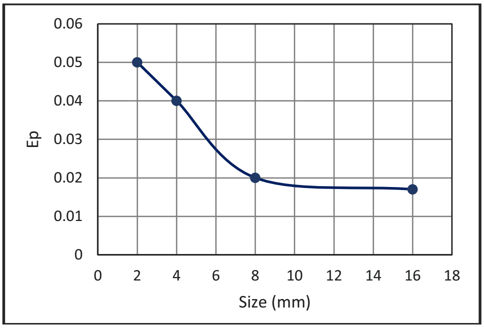



Figure 9. The dependence of Ep on size at secondary cyclones

It can be seen in Figure 8 and Table 3 that the Ep value gradually increase from 0.017 to 0.050 as the size decrease from 16mm to 2mm. The cut-point shift also slightly increases to 0.12 g/cm$^3$ at 2mm size. The variation of Ep with the change in size is given in Figure 9. As the test was conducted at "feed-off" condition, the results should be evaluated as an ideal (the best) case that could be achieved at these operating conditions.

## 3. CONCLUSIONS

The performance prediction and optimization is crucial in dense medium separation circuits. The density tracers being easy, cost-effective, reliable and fast in performance prediction, offers advantages over conventional float-sink analysis method.

The performance prediction studies conducted at three coal washing plants results show that the main issue at Ömerler plant is the distributor at the primary DMC circuit. The failure in equally dividing the flow to three cyclones causes a significant decrease in separation efficiency to less than the efficiency of the secondary cyclones.

The dense medium drums at Tunçbilek and Dereköy plants exhibit problems in the primary stage. The separation efficiency of low-density (clean coal) particles is less than that of the high density material which results in the selective loss of low density particles to sinks. This is mainly the consequence of the presence of high amount of low-density fraction which exceeds the floating limit of the drums determined by the lip length.

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# In-Plant Testing of Teetered Bed Separator in Omerler Washing Plant

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## Abstract:

In this study, the performance of pilot scale teetered bed separator (TBS) was evaluated using the data obtained in Omerler coal washing plant in Kutahya region. One of the individual spiral feed was used as feed for TBS. Pressure set point and the amount of teetered water were used as operating variables. At steady state conditions, the samples were collected from feed, underflow and overflow. Then, the samples were sieved into size fractions and the ash contents were determined on size basis.

The results showed that both cut density and Ep decreased with increasing particle size verifying earlier observations published in the literature. If clean coal is targeted, single stage separation suffers from high coal loss. However, it was found that significantly higher ash content was obtained with TBS comparing with the spiral tailings.

**Keywords:** Fine coal beneficiation, Teetered bed separator, Coal washing, Lignite, Coal preparation, Partition curve, Performance evaluation, Classification.

### Introduction

The common coal preparation flowsheet in Turkey consists of dense medium vessel for coarse coal (+18mm), dense medium cyclone for -18+0.5mm coal and spiral concentrators for -0.5mm. In many of the operating plants, screen apertures are increased up to 2mm due to the inefficiencies in fine screening.

The amounts of -0.5 mm and -2mm fractions are 15-25% and 20-40%, respectively. Therefore, proper design of fine coal beneficiation circuit is very important. On the other hand, the spiral circuits are usually poorly operated. The main reasons are the fluctuation in size distribution of ROM coal which causes change in the solid and water tonnage to the fine coal circuit, difficulties of the uniform split of flow to the parallel spiral units under fluctuating conditions, their low unit capacity, and high Ep value.

The spiral tailing usually contains coal and clean coal product which is not suitable for directly mixing with DMS products.

TBS's have been found applications in coal preparation of fine coal circuits in different countries (Bethell, 1988; Drummond et. al., 1998; 2002; Kohmuench, 2006; Newling, 1998; Ratlou, 2006; Reed, 1995; Sarkar, 2008; Tao, 2011). Turkish Coal Enterprises has set an experimental program and tested TBS for different coals.

In this paper, the results of the experimental studies performed at Omerler coal washing plant located in mid-west of Turkey are presented.

### Experimental Studies and Results

Omerler coal preparation plant has a nominal capacity of 600 tph. The coarse and fine coal are washed using DMS baths and DMS cyclones, respectively. The fine coal is cleaned by spirals after desliming with hydrocylones. The pilot scale TBS was installed in spiral section of the plant. The feed of the individual spiral unit was diverted to TBS. During testwork, it was waited until the system is stabilized. Then, the samples were taken from underflow (tailing) and overflow (coal).

The size distribution of the two samples taken from spiral feed are given in Figure 1.

 
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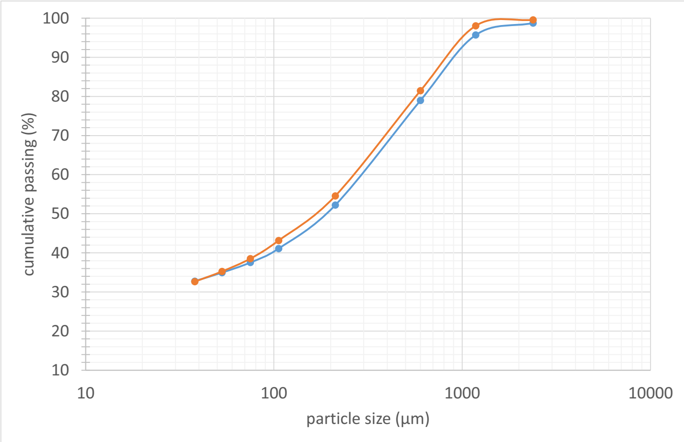


Figure 1. Size distribution of spiral feed at Omerler washing plant during the study.

As can be seen from Figure 1, the spiral feed contains 23-26% +0.5mm material and 32% -0.038mm material revealing that the feed was not in a narrow range for optimum TBS application.

The washability of spiral feed is given at Table 1.

Table 1. Washability of size fractions of spiral feed.

| | 250-600 (µm) | | 600-1000(µm) | | +1000(µm) | |
|---|---|---|---|---|---|---|
| ρ(g/cm$^3$) | Weight (%) | Ash (%) | Weight (%) | Ash (%) | Weight (%) | Ash (%) |
| Float 1.3 | 41.7 | 16 | 44.2 | 10.78 | 40.6 | 9.93 |
| 1.30×1.40 | 13.3 | 25.92 | 9.4 | 16.04 | 9.9 | 17.29 |
| 1.40×1.50 | 6.5 | 41.75 | 5.4 | 30.53 | 5.5 | 28.93 |
| 1.50×1.60 | 4.8 | 50.64 | 5.7 | 42.23 | 5.4 | 40.12 |
| 1.60×1.70 | 4.2 | 58.09 | 5.4 | 51.42 | 4.9 | 47.73 |
| 1.70x1.80 | 4.1 | 64.63 | 3.9 | 58.57 | 4.3 | 54.29 |
| 1.80x1.90 | 2.6 | 68.13 | 4.7 | 64.73 | 5.4 | 64.68 |
| Sink 1.9 | 22.8 | 76.46 | 21.2 | 76.76 | 24 | 74.62 |

*Experimental Studies*

228.6×406.4mm pilot-scale TBS was used in the experimental studies. The underflow discharge is adjusted by using an automatic control system. The pressure in the bed is measured and when the pressure is reached to set value, the valve opens until it drops below the set value. This keeps separation density constant.

Several test campaigns were carried out. Teeter water flow rate (TW) was 17-41 liters/min (lpm) equivalent to superficial velocity of 2-4.6 mm/s, and set point (SP, relative density) varied as 530-620 (Kohmuench et. al., 2002).

Since the feed contains significant amount of fines and they report to overflow (clean coal product), the results are presented for overall and for +0.1mm separately (Table 2).

Table 2. The results of the TBS testwork.

| | | Ash(%) | LHV (kCal/kg) | Ash(%) | LHV (kCal/kg) | Ash(%) | LHV (kCal/kg) |
|---|---|---|---|---|---|---|---|
| Condition | | Feed | | Overflow | | Underflow | |
| SP: 530, TW: 23.88 lpm | Overall | 46.6 | 3252 | 44.28 | 3574 | 70.02 | 1379 |
| | +0.1mm | 33.51 | 4687 | 27.71 | 5200 | 71.22 | 1356 |
| SP: 530, TW: 29.8 lpm | Overall | 43.76 | 3460 | 47.29 | 3087 | 75.29 | 820 |
| | +0.1mm | 34.09 | 4656 | 33.74 | 4523 | 76.55 | 864 |
| SP: 530, TW: 37.9 lpm | Overall | 47.11 | 3289 | 45.1 | 3643 | 74.16 | 1065 |
| | +0.1mm | 36.29 | 4281 | 32.26 | 4561 | 75.1 | 1097 |
| SP: 550, TW: 23.9 lpm | Overall | 46.30 | 3446 | 44.45 | 3601 | 71.25 | 1351 |
| | +0.1mm | 32.43 | 4611 | 29.78 | 4833 | 69.79 | 1473 |
| SP: 550, TW: 38.9 lpm | Overall | 45.76 | 3491 | 42.44 | 3770 | 71.86 | 1299 |
| | +0.1mm | 33.92 | 4486 | 31.61 | 4680 | 70.44 | 1418 |
| SP: 575, TW: 23.9 lpm | Overall | 47.94 | 3308 | 46.72 | 3411 | 74.62 | 1067 |
| | +0.1mm | 34.61 | 4428 | 31.96 | 4650 | 74.01 | 1118 |
| SP: 575, TW: 37.9 lpm | Overall | 45.72 | 3370 | 45.22 | 3375 | 76.13 | 706 |
| | +0.1mm | 30.98 | 4749 | 30.89 | 5017 | 76.38 | 808 |
| SP: 600, TW: 16.9 lpm | Overall | 53.97 | 2801 | 51.90 | 2975 | 78.29 | 759 |
| | +0.1mm | 39.21 | 4041 | 34.48 | 4439 | 77.57 | 819 |
| SP: 620, TW: 29.8 lpm | Overall | 46.1 | 3374 | 41.69 | 3833 | 76.77 | 887 |
| | +0.1mm | 33.68 | 4595 | 27.81 | 4999 | 76.45 | 913 |
| SP: 620, TW: 38.9 lpm | Overall | 52.8 | 2714 | 49.3 | 2950 | 76.32 | 736 |
| | +0.1mm | 46.2 | 3589 | 38.76 | 4181 | 77.63 | 739 |

**all the results reported are in dry basis.*

The underflow ash contents were significantly higher than spiral tailings (59-63% ash) indicating higher recovery of coal. However, the lowest ash content achieved for +0.1 mm clean coal is 27.71% which is substantially higher for the clean coal produced in the area (18-20% ash).

**Discussion**

To evaluate the size by size performance of the TBS, its feed and products were screened into +1mm, -1+0.600 mm and 0.6+0.25 mm size fractions. Heavy liquid analyses were performed on these size fractions. Then, partition coefficient for size fractions were calculated. The partition curves of different size fractions are given in Figure 2.

As can be seen from Figure 2, coarser fractions were separated at lower densities. The same trend was observed at different operating conditions. Similar behaviour was also reported in the literature (Kohmuench et. al., 2002; Young and Klima; 2000; Das et. al., 2009; Li et.al., 2014; Luttrell et.al. 2006). This also explains the importance of feeding closely sized material to TBS. Therefore, the lower quality of coal product obtained in the testwork was mainly due to the wide size range of the feed. The partition curves for different TW and SP are given in Figure 3 and 4, respectively. Although the partition curves a trend, the relationship is not as powerful as particle size. SP has more pronounced effect than TW on the performance.

Regarding the implementation of TBS in Omerler washing plant, the flowsheet given in Figure 5 is proposed. It was observed that significant bypass occurs both in underflow and overflow of hydrocyclones. Therefore, hydrocyclones are removed from the circuit. -18mm coal will be screened at 2mm aperture. -2mm will be the fine coal circuit feed. If clean coal is targeted, the application should be arranged in two stages. Alternatively, power plant coal can be produced with a minimum coal loss in a single stage. In either case, the products should be classified. This circuit will be less sensitive to the fluctuations in solid and water flowrates. The final decision can only be made after a detailed feasibility study.

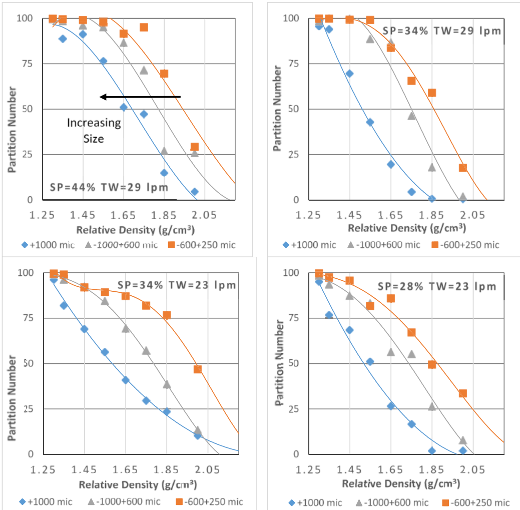


Figure 2. Partition curves for different size fractions.

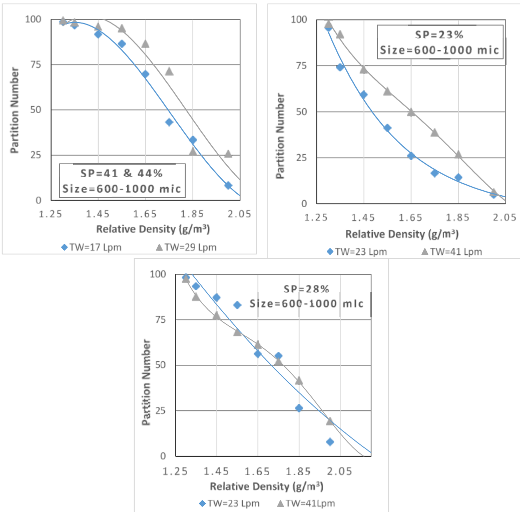

Figure 3. Partition curves for 1000-600 µm size fraction for varying TW.

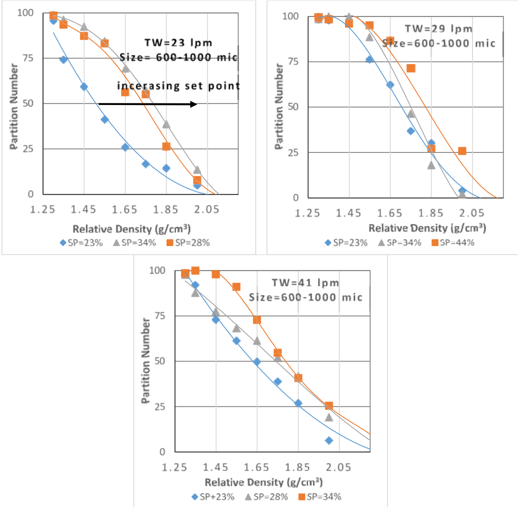


Figure 4. Partition curves for 1000-600 µm size fraction for varying SP.

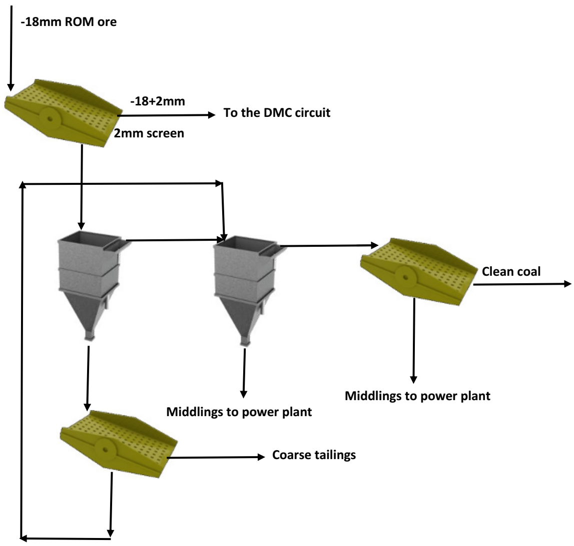


Figure 5. Proposed flowsheet for fine coal beneficiation

## Conclusions

The pilot scale test results showed that there is a potential to improve fine coal washing performance in Omerler washing plant by using TBS separator.

As the particle size increases the separation density decreases regardless other operating variables, i.e., TW and SP.

SP has more pronounced effect than TW, although both have an effect on performance.

Unless the classication is very efficient, the coal product should be classified to get a desired product.

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# Commercial Coal Preparation Plants Capability for the Removal of Trace Elements

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**Abstract.** Işıklar and Dereköy coal preparation plants clean lignite coals produced in Soma district. Run-of-mine coals contain trace elements and major elements that are associated with coal and inorganic matter. Trace elements are potentially harmful for human health and ecosystem. Physical coal cleaning is effective in reducing the concentration of many trace elements. In this study, the samples were collected from run-of-mine lignite (feed), coal preparation plant products, namely coarse (+18 mm) clean coal, fine (10-18mm, 0.5-10mm) clean coals, middlings, coarse and fine refuses and slimes from the Dereköy and Işıklar plants systematically and chemical analyses of ash, trace and major elements were performed on each sample. Trace elements contents and distributions in each product were determined. During the cleaning of run-of-mine lignites in Işıklar and Dereköy coal preparation plants, 54.69% and 72.04% ash have been removed from the plants, respectively. Around 1/3 of Ga and Rb contents of Işıklar feed and more than 1/3 of Ga, Rb, Sr, contents of Dereköy feed could be removed by ash. Referring to major elements, rejection of 60% Ca and 53% Mg from Işıklar, and 69% Ca and around 80% Mg from Dereköy plants have been achieved.

**Keywords:** Lignite, trace elements, major elements, physical coal cleaning, coal preparation plant, removal of trace elements, dense medium separation

## 1 Introduction

The energy projections showed that coal will maintain its importance in the energy sector in future. The main use of coal is in the electricity generation by combustion which results in ash production and emissions of gases and some trace elements. Trace elements are potentially hazardous to human health and ecosystems.

Trace elements are partly organically bound in coal while some are associated with mineral matter that refers to all elements which are generally considered to be inorganic. Most of the trace elements in coal are associated with silicon-rich minerals (quartz, sheet silicates, clay minerals), sulfides, carbonates, phosphates and as discrete minerals. Arsenic is commonly associated with pyrite, Cd with sphalerite. Most of the Sb, Co, Hg, Mo, Cu, Pb, Fe, Zn, Se, Ni and Tl are found in sulfide forms. Cr is mostly found with clay minerals. Be, Ge, P, Ga, W, U and V are more closely associated with the elements having strong organic affinities.

In many cases, a trace element occurs in more than one form. B, S and V are mainly organically bound, but B and V are also found in clays and S in sulfides.

Coal cleaning, when applied before combustion to remove ash in coal, is considered to be an economical and effective technique in minimizing environmental problems. During the removal of ash, trace elements associated with inorganic matter is partitioning and be rejected by autogenously (Conzemius, et al.1988, Wang et al, 2009, Tang, et al, 2009, Özbayoğlu, 2010, 2011).

The effects of beneficiation on trace elements removal in Australian washeries and in USA for commercially cleaned coals were extensively investigated (Swaine, 1990). Besides, the influence of

 
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flowsheet configurations on the removal of trace elements was examined (Swaine,1998). However, very limited work was done on the trace elements removal from the commercial coal preparation plants in Turkey (Özbayoğlu, 2013). All these works showed that the degree of removal of trace elements is dependent on the coal, the degree of liberation of the trace elements bearing mineral matters, and the cleaning method (Vito et al,1996).

The objective of this study is to determine the extent of removal of some of the trace elements from two commercial coal preparation plants in Turkey.

### 1.1 Coal Preparation Plants

Two commercial coal preparation plants (CPP), namely Dereköy and Işıklar from Soma district- situated at the western part of Turkey, 121 km away from İzmir - were chosen for this investigation. These plants are washing lignite coals produced from various mines at Soma district. Dereköy CPP cleans coal coming from Işıklar and Eynez open pit mines as well as Eynez underground mine while Işıklar CPP treats lignite coals coming from underground mines of Merkez-Mumya and Işıklar.

The capacity of Dereköy and Işıklar CPP are 450 tph and 800 tph, respectively. The cleaning methods in both coal preparation plants are based on dense medium separation. Dereköy CPP consists of two units, 400 tph each, involving two-stage dense medium separation and spirals. The coal is screened at 18mm through multislope banana screens. The fines and slimes are separated by vibrating screens and sieve bends. +18mm is cleaned in double compartment dense medium drum at a density of 1.55g/cm$^3$ and 1.80 g/cm$^3$ and -18+0.5mm size is cleaned in two-step dense medium cyclones at the same densities while -0.5+0.1mm size is cleaned in spirals. -0.1 mm slime is sent to thickener and *thickened p*ulp is pumped to tailing pond.

The main difference between Dereköy and Işıklar CPP is the use of two separate drums in Işıklar for the cleaning of +18mm coarse coal. The densities in the first and second drum are 1.45 g/cm$^3$ and 1.75 g/cm$^3$ respectively. For the cleaning of -18+0.5 mm fine size coal, the density in the first cyclone is 1.35 g/cm$^3$ and in the second cyclone is 1.80 g/cm$^3$. Magnetite is used in the preparation of dense medium in both plants. The other application in Işıklar CPP is that, the thickened pulp is passed through filter press before sending to the tailing pond.

## 2 Materials and Method

Coal preparation plant producs, namely coarse (+18 mm) clean coal, fine (10-18mm, 0.5-10mm) clean coals, middlings, coarse and fine refuses and slimes were collected from the Dereköy and Işıklar plants systematically. Besides, run-of-mine (r.o.m) coal samples were obtained . All the samples were analysed for ash, total sulfur, major and trace elements on dried representative samples by using XRF and ICP-OES techniques. Trace elements contents and distributions in each product were determined. The results of clean coals of +18mm, 10-18mm and 0.5-10mm sizes were combined as clean coals and coarse and fine refuses were combined as refuses in order to see how much trace elements would be removed from the wastes and how much would be retained in the clean coals.

## 3 Results and Discussions

The characteristics of r.o.m coals of Işıklar and Dereköy are shown in Tables 1 and 2.

**Table 1 Chemical analyses of run-of-mine coals of Işıklar and Dereköy Coal Preparation Plant**

| Analyses | Işıklar | Dereköy |
|---|---|---|
| Moisture %, (as received) | 14.46 | 11.37 |
| Ash %, (dried basis) | 47.70 | 46.84 |
| Low heat value (kcal/kg), (as received) | 1822 | 1999 |

As seen from Table 1, the r.o.m coals have high ash contents, however their moisture content is low.

Mineral composition of ash of Işıklar r.o.m coal showed that quartz and silicates are the dominant minerals; it is also consisted of calcite, siderite, kaolinite and small amounts of pyrite, mica and smectite. Dereköy r.o.m coal contains mainly calcite and silicates with siderite. Clay occurences in both coals create problems in the plants.

**Table 2 Trace and major elements grades of run-of-mine coals of Işıklar and Dereköy CPP**

| IŞIKLAR | | | | DEREKÖY | | | |
|---|---|---|---|---|---|---|---|
| **Trace elements** | ***Trace elements grade, ppm*** | ***Major elements*** | ***Major elements grade, %*** | **Trace elements** | ***Trace elements grade, ppm*** | ***Major Elements*** | ***Major elements grade, %*** |
| Cr | <0.00051 | Si | 18.46* | Cr | <0.00051 | Si | 13.76* |
| Co | <0.00071 | Al | 9.55* | Co | <0.00071 | Al | 6.75* |
| Zn | 91.29* | Fe | 3.20* | Zn | 56.40* | Fe | 3.33* |
| As | 93.30 | Ca | 20.17* | As | 129.30 | Ca | 30.53* |
| Y | 36.31* | Mg | 0.796* | Y | 26.85* | Mg | 0.584* |
| Ta | <0.00010 | S | 1.44* | Ta | <0.00010 | S | 1.45* |
| Th | <0.00010 | K | 0.827 | Th | <0.0030 | K | 0.565 |
| Ti | 4011.11* | P | 0.0167 | Ti | 3126.49 | P | 0.037 |
| Ni | <0.00051 | Mn | 0.0056 | Ni | <0.00051 | Mn | 0.0104 |
| Ga | 35.93 | | | Ga | 33.35 | | |
| Rb | 64.12* | | | Rb | 53.58* | | |
| Zr | <0.00051 | | | Zr | <0.00051 | | |
| W | <0.00051 | | | W | <0.00051 | | |
| U | 0.00059 | | | U | 0.00036 | | |
| V | 585.65* | | | V | 541.26* | | |
| Cu | <0.00040 | | | Cu | <0.00040 | | |
| Ge | <0.00020 | | | Ge | <0.00020 | | |
| Sr | 391.29* | | | Sr | 314.33* | | |
| Hf | <0.00010 | | | Hf | <0.00010 | | |
| Pb | <0.00522 | | | Pb | 0.0006 | | |

- By Calculation = $\frac{\sum(\text{Products Weight \% x Products Grade})}{100}$

Most of the trace elements grades of r.o.m coals of Işıklar and Dereköy are much lower than world and

Turkish averages, like lead, uranium, thorium, manganese, chromium, wheras arsenic and vanadium grades are high (Tuncali, et al, 2002) Sulfur grade of the coals is also low which supports the low content of sulfide minerals.

Referring to the major elements in Table 2, the Si, Al and Mg contents of Işıklar r.o.m. coal are higher than Dereköy r.o.m. coal. These elements explain the reason of higher clay contents of Işıklar. Ca and Fe contents of Dereköy r.o.m. coal are higher than Işıklar r.o.m. coal which approves the higher content of calcite.

Partition of trace and major elements of Işıklar and Dereköy r.o.m. coals in the plant products were investigated and their results are summarized in Tables 3 and 4. In these tables, the average grades and distributions of trace and major elements are tabulated for combined clean coals and combined refuses, as stated in section 2.

**Table 3 Trace and major elements grades and distributions of clean coals of Işıklar and Dereköy CPP**

| | **IŞIKLAR** | | **DEREKÖY** | |
|---|---|---|---|---|
| **TRACE ELEMENTS** | ***Clean coal Weight %: 24.88 Grade, ppm*** | ***Distribution of trace elements, %*** | ***Clean coal, Weight%: 32.09 Grade , ppm*** | ***Distribution of trace elements %*** |
| Ti | 5488.82 | 34.04 | 5244.57 | 53.88 |
| Sr | 558.57 | 35.52 | 370.74 | 37.85 |
| Ga | 40.05 | 27.74 | 36.28 | 34.91 |
| Y | 61.13 | 41.88 | 49.88 | 59.62 |
| Rb | 64.47 | 25.01 | 69.35 | 41.53 |
| V | 1563.5 | 66.42 | 1435.63 | 85.11 |
| Zn | 156.86 | 45.32 | 100.01 | 56.91 |
| **MAJOR ELEMENTS** | ***Clean coal, Weight %: 24.88 Grade of major elements ,%*** | ***Distribution of major elements , %*** | ***Clean coal, Weight %: 32.09 Grade of major elements ,%*** | ***Distribution of major elements, %*** |
| Al | 11.64 | 30.32 | 10.35 | 49.14 |
| Fe | 4.94 | 38.39 | 4.87 | 46.82 |
| Ca | 8.97 | 11.06 | 12.57 | 13.21 |
| Mg | 0.592 | 18.49 | 0.052 | 2.88 |
| S | 2.95 | 51.05 | 3.26 | 71.92 |
| K | 0.788 | 23.69 | 0.788 | 44.72 |
| Si | 18.42 | 24.83 | 17.32 | 40.38 |
| Average ash % of clean coals | 15.13 | 7.74 | 12.54 | 9.33 |

As seen in Table 3, V and S in Işıklar and V,Y, Zn,Ti and S elements in Dereköy plants mostly retain in clean coals.

The S contents of r.o.m. coals of Işıklar and Dereköy were 1.44% and 1.45%, respectively. They were enriched in clean coals of Işıklar and Dereköy to 2.95% and 3.26%, respectively. It shows that the sulfur forms in both coals might be due to organic sulfur and/or frambodial sulfur which can not be removed by physical cleaning methods; therefore they were retained in the clean coals.

**Table 4 Trace and major elements grades and distributions in refuses of Işıklar and Dereköy CPP**

| | IŞIKLAR | | DEREKÖY | |
|---|---|---|---|---|
| **TRACE ELEMENTS** | ***Refuses Weight %: 37.76 Grade of trace elements, ppm*** | ***Distribution of trace elements, %*** | ***Refuses Weight %: 46.17 Grade of trace elements , ppm*** | ***Distribution of trace elements, %*** |
| Ti | 2827.0 | 26.61 | 1509.00 | 22.28 |
| Sr | 267.70 | 25.83 | 260.09 | 38.32 |
| Ga | 31.50 | 33.09 | 28.06 | 39.59 |
| Y | 19.00 | 19.76 | 11.05 | 19.77 |
| Rb | 57.70 | 33.97 | 40.00 | 34.47 |
| V | 84.50 | 5.45 | 36.00 | 3.07 |
| Zn | 50.80 | 21.01 | 21.50 | 17.6 |
| **MAJOR ELEMENTS** | ***Refuses Weight %: 37.76 Grade of major elements ,%*** | ***Distribution of major elements, %*** | ***Refuses Weight %: 46.17 Grade of major elements ,%*** | ***Distribution of major elements, %*** |
| Al | 7.24 | 28.61 | 3.74 | 25.55 |
| Fe | 2.30 | 27.11 | 2.23 | 30.84 |
| Ca | 32.26 | 60.38 | 45.64 | 69.01 |
| Mg | 1.11 | 52.58 | 1.01 | 79.86 |
| S | 0.55 | 14.41 | 0.26 | 8.28 |
| K | 0.76 | 34.73 | 0.33 | 27.03 |
| Si | 16.70 | 34.15 | 10.19 | 34.18 |
| Average ash % of refuses | 67.78 | 54.69 | 67.31 | 72.04 |

In the refuses, as seen in Table 4, Ca, Mg contents in both plants could be removed. However, no appreciable amounts of removals of trace elements were obtained.

Sr, Ga, Rb, and K are relatively consistent with regard to their distribution in all products. They showed no preference to any product stream.

## 4 Conclusions

1. Most of the trace elements grades of r.o.m. coals (feeds) of Işıklar and Dereköy CPP are much lower

than world and Turkish averages.

2. Both of the feeds are consisted of clay, calcite, quartz, mica, siderite, smectite and pyrite in various amounts. The dominant minerals in Işıklar are quartz and silicates while they are calcite and silicates for Dereköy.

3. The ash removal from Işıklar CPP is 54.69% while it is 72.04% in Dereköy CPP.

4. Compared to the feed of Işıklar, cleaned coals have higher grades of Ti, Sr, Ga, S, Y, V, and Zn whereas the major elements of Ca, and Mg are lower in grades than the feed. In case of Dereköy cleaned coals, Zn,V,Y,Al,S,Si and Ti grades are higher than the feed, whereas Ca and Mg are much lower than the feed.

5. During ash rejection from coal preparation plants, 60% Ca and 53% Mg from Işıklar, and 69% Ca and around 80% Mg from Dereköy are removed. The removal of trace elements are limited. Around 1/3 of Ga, Rb contents of Işıklar feed and more than 1/3 of Ga, Rb, Sr, contents of Dereköy feed could be removed by ash. It shows that trace elements removal is not proportional to ash removal.

6. Sulfur contents of both Işıklar and Dereköy clean coals have higher grades than their feeds. This might be explained due to the forms of sulfur which are organic and frambodial forms. These can not be removed by physical methods.

7. Although the removal of trace elements from Işıklar and Dereköy plants are limited, coal preparation is an effective technique to minimize the harmful effect of trace elements.

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# The Development of a new Low Cut Point Spiral for Fine Coal Processing

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**Abstract**

Records show that Spirals, for the separation of heavy minerals, existed over 100 years ago and have continued to undergo development ever since. Applications of spirals to fine coal (between 1.00 mm and 100 micron) are relatively more recent dating back approximately 30 years. The advantages of spirals are well recognised within the coal industry and include simplicity of operation, reliability, a high tolerance to variation in operating conditions along with low capital and operating costs.

A limiting feature of conventional coal spirals is an inability to achieve d50 cut points at specific gravities (sgs) lower than 1.65. With technological advances in coarse coal treatment enabling lower cut points, this limitation of coal spirals in fine coal treatment has become increasingly more significant in the context of overall plant performance.

Mineral Technologies has addressed this limitation and recently developed the LC3, a new coal spiral capable of separation cut points lower than those of other available models. Extensive testing occurred during the design and development stages and the LC3 readily achieved cut points in the specific gravity (sg) range 1.40 to 1.55.

The design of the LC3 is a clear departure from other current coal spirals. It entails continuously variable profiles and pitches and is based around reducing turbulence to enhance subtle separating mechanisms. Initially developed as an 8-turn unit, it is however no taller than other industry-accepted 7 turn models. A 4-turn format LC3 has also now been developed.

As well as achieving lower cut points at higher than expected unit loadings, preliminary testing also indicates comparable or better separation efficiencies at conventional cut points and, additionally, a potential for separating particles at sizes previously deemed too fine for spiral applications.

This paper presents test data and findings from the development program and subsequent, post-development investigations such as a 2-stage (middling retreat) circuit.

Keywords: LC3 spiral, middlings retreatment, gravity separation, spirals, low cut point spiral, fine coal processing, spiral circuit

**Background**

In 2013 Mineral Technologies presented to the industry a new low cut point coal spiral, the LC3. The new spiral featured a unique profile design that provided better opportunities for clean coal particles to spread across the trough surface, enabling improved discrimination at low sgs.

The new spiral was performance benchmarked against an existing range of coal spiral models: LD4E, LD7 and the taller LD7RC compound spiral. Results confirmed that lower cut points could readily be achieved. In fact it was revealed that the actual d50 cut point was no longer a limiting factor and that an operator had to take care not to target too low a cut point as it could result in a lower than desired recovery to product. Data from a subsequent test program (discussed below) also indicated that, operating at a conventional cut point of around 1.7 sg, the LC3 achieved higher metallurgical performance than the leading spiral in the conventional range, the LD7RC.

 
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It was further found that whilst a target for capacity was set by leading industry entities at 1.5 t/h per trough, the LC3 could tolerate increases in feed rate up to 2.5 t/h and beyond without the metallurgical performance suffering significantly. These results suggest that the LC3 may have a role to play in both traditional and low cut point applications.

## Results and Discussion

### *Application at Conventional Cut Point*

Test work was conducted on a sample of spiral feed from Queensland Australia. The LC3 model was compared with the LD7RC. Both models demonstrated encouraging metallurgical performances and achieved low product ash grades. The LC3 achieved lower ash grades in both the product and the middling streams. The LC3 displays superior separation at both the low cut point and the higher cut point.

The charts in figure 1 below present the fitted partition curves for each spiral.

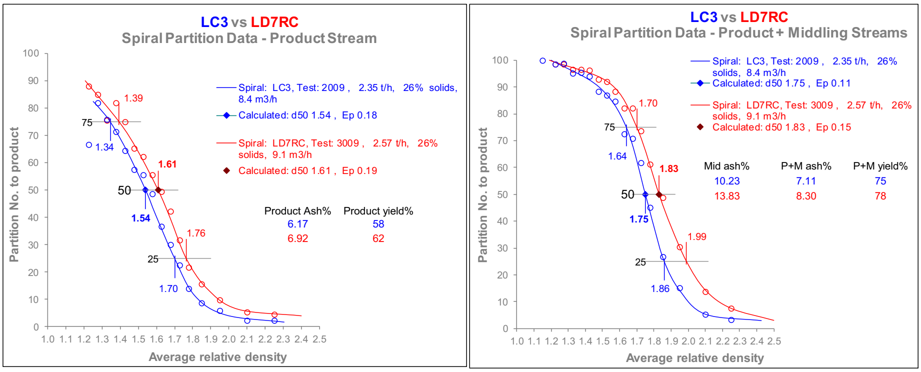


**Figure 1 - LC3 8T versus LD7RC**

The data plotted in figure 1 show that the LC3 achieved a lower product ash grade of 6.2% compared with the LD7RC which achieved a product of 6.9% ash.

A feature of the plotted data above is in the “tails” of the curves; revealing that the LC3 was superior in rejecting high sg material.

Note that during operation of the LD7RC spiral, particular effort was made to set up the product splitter for as low a cut-point as possible and was deemed to be around the LD7RC’s limit whereas the LC3 could go lower if required. The LC3 also achieved an improved separation efficiency on the higher cut with an Ep of 0.11 compared with the LD7RC which attained an Ep of 0.15. Again, the “tails” of the curves pertaining to the higher cut show that the LC3 is more capable of rejecting high sg material into the refuse stream. This enabled the LC3 to generate a mid-stream with a substantially lower ash content compared with the mid-stream of the LD7RC. The ash grades were determined to be 10.2 and 13.8% respectively. This was an important point of distinction for the owners of this operation as they commercially produce a primary and a secondary product.

It is also notable that the combined product plus mids generated by the LC3 has an ash content of only 7.1% (at a yield of 75%) which is very close to the ash content of the LD7RC's product stream alone at 6.9% ash and a yield of only 62%. This represents a yield differential of 13 percentage points over 62; or on a relative basis, this could equate to around a 20% production improvement to the LC3.

*Application in Ultrafine Feed*

The opportunity arose to test the LC3 on ultrafine feed. The sample was from a washery tailings dump in Africa and initial size analysis indicated 87% passing 0.15mm and 71% passing 0.075mm.

An 8-turn LC3 was operated at 1.3 t/h producing a separation described by the data in table 1 below:

**Table 1 - LC3 Test data on Ultra-fine Feed (Ash basis)**

Before wet screening (38um)

| Stream | mass yield% | Ash% |
|---|---|---|
| Product | 63.4 | 26.8 |
| Middling | 14.3 | 26.4 |
| Reject | 22.3 | 40.8 |
| Calc' Feed | 100.0 | 29.9 |

After wet screening

| Stream | mass yield% | Ash% |
|---|---|---|
| Product | 53.6 | 14.6 |
| Middling | 14.3 | 16.7 |
| Reject | 32.1 | 41.5 |
| Calc' Feed | 100.0 | 23.5 |

Some degree of separation occurred with the reject upgraded in ash content to 40.8% from a feed grade of 29.9%. The product and middling ash grades were only modestly downgraded but it was suspected that the presence of slimes (-38 micron) was masking the separation. To better understand the effect of particle size, the test fractions were deslimed by wet screening at 38 micron. The -38 micron wet screen undersize fractions from all three streams, product, middling and reject, were determined to have very similar ash contents at 37%, 37% and 39% respectively indicating practically no separation in this range (not surprisingly). With the bulk of the -38 micron material removed, the ash content of the product and middling streams dropped considerably from 26 and 27% respectively to 15 and 17% ash.

An ash by size analysis was carried out on the wet screened test fractions. Table 2 below indicates significant discrimination even on the -53+38 micron fraction in which the product ash grade was determined to be 25.0% compared with that of the reject at 46.1%.

**Table 2 – Ash by size Test data on +38 micron Fraction**

| | Fractional | | | | Cumulative | | | |
|---|---|---|---|---|---|---|---|---|
| | Product plus mid | | Reject | | Product plus mid | | Reject | |
| Seive size | mass% retained | Ash grade % | mass% retained | Ash grade % | mass% retained | Ash grade % | mass% retained | Ash grade % |
| 1000 | 2.22 | 23.5 | 2.31 | 62.5 | 2.22 | 23.5 | 2.31 | 62.5 |
| 500 | 7.00 | 9.3 | 19.67 | 41.6 | 9.22 | 12.7 | 21.98 | 43.8 |
| 250 | 10.82 | 8.0 | 26.56 | 38.6 | 20.04 | 10.2 | 48.54 | 41.0 |
| 150 | 15.98 | 9.0 | 16.85 | 40.3 | 36.02 | 9.7 | 65.39 | 40.8 |
| 106 | 13.66 | 10.8 | 9.08 | 40.2 | 49.68 | 10.0 | 74.47 | 40.7 |
| 75 | 18.60 | 14.3 | 10.32 | 42.8 | 68.28 | 11.1 | 84.79 | 41.0 |
| 53 | 14.87 | 19.9 | 7.55 | 45.3 | 83.15 | 12.7 | 92.34 | 41.3 |
| 38 | 14.00 | 25.0 | 6.03 | 46.1 | 97.15 | 14.5 | 98.37 | 41.6 |
| -38 | 2.85 | 36.8 | 1.63 | 38.7 | 100.00 | 15.1 | 100.00 | 41.6 |
| sub-total | 100.00 | 15.1 | 100.00 | 41.6 | | | | |

For the size fractions between 53 and 150 microns the collective product ash grade is 15.1% compared with the reject at 44.4%.

These results were considered somewhat of a revelation given the very low specific gravity of the particles involved compared to other mineral types. The ability of the LC3 to separate coal at very fine particle sizes is attributed to its unique trough design, the objective of which was to reduce turbulence.

*The Introduction of a 4-Turn LC3*

The original version of the LC3 is an 8-turn unit. Other “tall” coal spirals like the SX7 and LD7RC comprise 7 turns; however due to its reduced pitch the 8-turn LC3 occupies the same height and has a similar or marginally smaller diameter.

Why 8 turns? Achieving a lower cut point on a coal spiral was deemed to be a very challenging goal since previous attempts had met with limited success. At the outset therefore, the designers availed themselves of the maximum height set by industry precedent.

Separation efficiency on a spiral is a function of several factors, one of which is residence time. However, like many natural phenomena, the relationship obeys the law of diminishing returns. Results of recent fundamental spiral research on mineral sands bear this out. Figure 2 below contains a plot of heavy mineral separation efficiency (Shultz Efficiency) versus number of turns. Two curves pertaining to different feed rates are shown. The shape of the curves indicates the diminishing performance gains with subsequent turns (residence time).

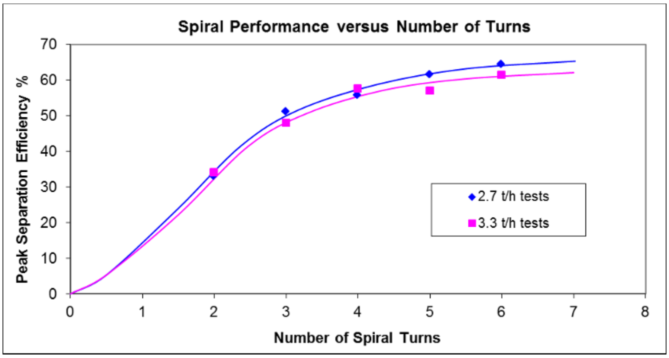


**Figure 2 -Relationship between performance and number of turns**

It is worth noting that between 5 and 6 turns the slope is still trending positive. Unfortunately the testing didn’t extend to 8 turns but one would expect the gradient, by that point, to be relatively flat for practical purposes.

It should be emphasised that while this “Performance-versus-Turns Profile” is likely to always form the same general shape, the profile will be unique for each different feed material, on each different spiral type, with each different set of feed conditions. More research is required but it’s generally accepted that an easy to separate feed type will reach an optimal level at fewer turns while a more difficult material will require more turns (residence time) to reach its practical separation potential.

The majority of coal spirals in service in Australia are of the shorter, 4-turn variety. Given a general need for replacements, it was a logical step to develop a 4-turn version LC3. Based on the above discussion on performance versus number of turns, it's not surprising that testing of the 4-turn version revealed its performance to be not greatly less than that of the 8-turn. Particularly given that many Australian coals have relatively low near-gravity material.

*Two-stage Circuit with Mids Re-treat on the 4-Turn LC3*

A test program on a new feed sample from a different location in Queensland Australia was recently conducted in which an LC3 4-turn spiral was set up to generate a bulk quantity of product, middlings and refuse. The middling fraction from this rougher (primary) stage was then repassed over the 4-turn LC3, generating a further 3 streams: secondary product, secondary middling and secondary reject. Note that the top size of the feed was approximately 2mm and the results pertain to the full size range from 0.106 to 2.00mm. The diagram in figure 3 below depicts the 2-stage circuit and shows the mass balance along with measured ash values.

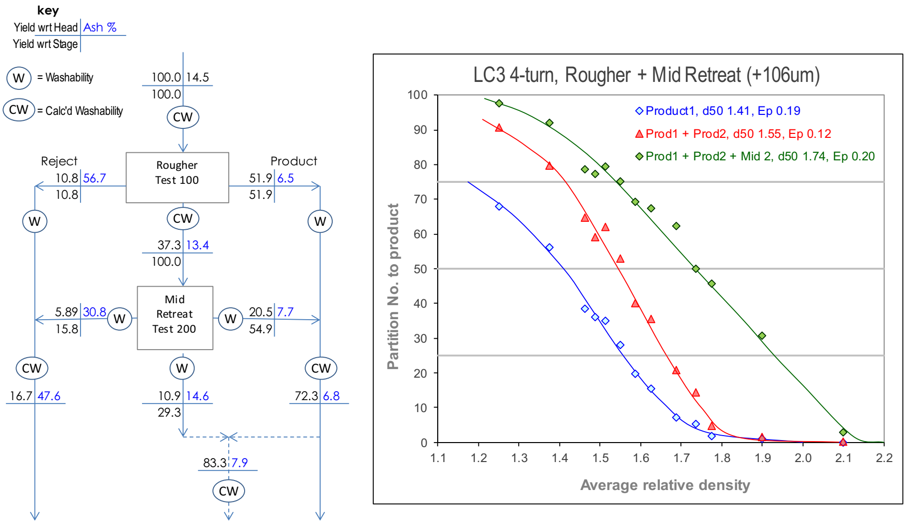


**Figure 3 - 2-stage circuit using 4-Turn LC3**

The test samples were wet screened at 0.106mm and subjected to washability analysis. The results from the 3 secondary streams allowed the washability of the primary middlings to be back-calculated. This provided the data necessary to calculate and plot sets of partition points for both individual and combined stages.

The chart in figure 3 presents 3 sets of partition data: for the primary product, combined products, and combined products plus secondary middling.

The blue curve shown above in figure 3 pertains to the primary product only and the red curve pertains to the combination of both primary and secondary products. The overall circuit d50 cut point is shown by the red partition curve to be 1.55 which is lower than generally achievable on conventional spirals. Due to the additive nature of partition data, the overall d50 cut point necessarily moved to the right (from the position of the primary stage cut point of 1.41) indicating the need to aim for lower cut points in the individual stages than the target required for the overall circuit.

The results indicate that adding a middling retreat stage can result in higher recoveries of low sg particles and higher separation efficiency (0.12 Ep compared with 0.19 Ep). A 20% increase in mass yield to product occurred with only a marginal increase in product ash grade, from 6.5 to 6.8%. The data presented in figure 3 above pertain to a wide size distribution from 0.106 to 2000mm. However, at the time of writing, further analyses are being conducted to generate washability by size data to better define partition by size.

The secondary middling represents approximately 11% of the mass balance in the overall circuit and it is evident from the partition curves that it contained some valuable low sg (1.3 to 1.4) material. In an operating plant this stream, could be reintroduced to either the rougher or middling retreat stage which would result in some further increase in the recovery of low ash coal.

## Conclusion

- A new coal spiral model, the 8-turn LC3 was introduced to the industry in 2013 and test results showed that it was capable of cut points significantly lower than conventional spiral models (1.40 to 1.60).
- Further work on the 8-turn LC3 in 2014 confirmed its ability to achieve low cut points and also indicated that it could be used to separate at conventional cut points (around 1.7 sg) with performance parameters that matched or exceeded those of the LD7RC model.
- Preliminary work on ultrafine feed showed that the LC3 could achieve a significant degree of separation on particles as fine as -53+38 micron.
- A 4-turn version of the LC3 was developed in 2015.
- Test work on the LC3 4-turn spiral showed that by repassing the primary middling and combining a secondary product with the primary product, a higher recovery of low sg particles could be achieved while still attaining a low cut point (1.55 sg) and an improved efficiency (0.12) as measured by Ep.
- Partition data attained on LC3 spirals to date has generally pertained to a wide particle size distribution. It is recommended that further work be undertaken to define cut point and Ep at narrower size ranges.

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# Circuits Incorporating New Low Cut Point Spirals

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**Abstract**

Recent research and development work by Mineral Technologies has resulted in a commercially available low cut-point spiral separator (LC3). Comparative test results show that the LC3 is capable of cut points significantly lower than those achievable with conventional coal spirals. The ability to target a cut point in the 1.45 to 1.60 range with spirals opens up new possibilities for coal processors who prefer the operational simplicity of spirals.

Spirals have been a standard fine coal beneficiation device in the coal industry for decades, but typically produce RD50 cut points in the range of 1.7 to 1.8. Where lower fine coal cut-points are desired hindered bed separators are typically used.

This paper discusses the relative merits of various fines coal circuit options incorporating LC3 spirals. Desktop LIMN simulations have been used to model "full plant" outcomes for dense medium cyclone processing of coarse coal in conjunction with one or two stage fine processing. Both Hunter Valley thermal coal and Queensland coking coal have been considered.

Keywords: coal preparation, fine coal treatment, gravity separation, spirals, low cut point spiral, LC3 spiral, middlings retreatment

**Background**

Spirals typically produce cut-points in the range of 1.7 to 1.9 RD, while hindered bed separators have demonstrated an ability to produce lower cut points at reasonable efficiencies (Swanson and Atkinson, 2007). Spirals utilise gravitational and centrifugal force to separate fine particles based on density, with the denser particles gravitating towards the centre and the less dense particles moving outward. The (adjustable) splitters located at the bottom of the column control cut-point, and density cut-points in the range of 1.65 to 2.00 have been achieved in operating plants, noting that cut-point depends on feed loading, washability, and whether middlings are recycled or sent to product or reject.

The LC3 spiral has a unique trough design that delivers a superior flow regime in which a specific gravity profile develops across the particle bed. It is this change in density profile which allows lower cut points to be achieved in comparison to traditional spiral designs (Palmer and Weldon, 2014). This superior flow regime is reflected in the clearly visible concentric wave pattern as shown in Figure 1 (cf. radial wave pattern developed by traditional spirals).

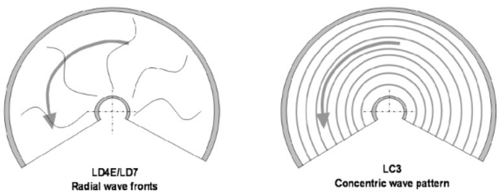


**Figure 1 - Wave patterns on different model spirals**

 
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The LC3 design dampens the flow energy and reduces the turbulence to create a new gentler flow regime. This approach presented a challenge because the separating mechanism also requires that the pulp bed maintains a level of continuous mobility and fluidity. To meet this challenge, the LC3 has been designed with a constantly changing set of profiles and pitches that progressively compensate for the changing nature of the slurry on its journey down the trough. The standard LC3 is an eight turn spiral, although a shorter four turn version is also available.

**LC3 Process Model Development**

QCC use LIMN for simulating practical yield-ash outcomes from coal processing circuits. LIMN applies practical inefficiencies to theoretical washability data to simulate the misplacement of material in sizing and density based separations, and also allows for recycled streams within a process.

As LIMN models are built around a size by density matrix, it was necessary to develop a LIMN process model specifically for the LC3 spirals based on the available testing and analysis data.

Testwork data have been sourced from a single start LC3 test rig in the Mineral Technologies workshop (these data have been supported by other plant testing, eg Thornton, et al, 2016). These results had the most complete set of performance data and were the most useful in generating a LIMN process unit model. The approach taken consisted of the following steps:

- Partition analysis was completed using the laboratory float-sink test,
- A curve-fit was used to calculate the standard form of a density based separation model used in LIMN, which relates partition number at each density fraction to Ep and cut-point. The form of this model (Whiten model) is shown in equation (1):

$$\text{Partition Number} = T_0 + \left( \frac{T_1 - T_0}{1 + \exp\left(1.098\left(\frac{D50 - RD}{Ep}\right)\right)} \right) \quad (1)$$

  - Note that low and high density bypass terms (the “T” terms), which change the form of the equation, are typically not included in spirals modelling (i.e. $T_0$=1 and $T_1$=0 typically).
- Curve fit parameters were determined for two separate cases for the LC3 test data, namely:
  - product partition and
  - product + middlings partition
- The existing QCC LD7 process unit LIMN model D50 and Ep by size data was adjusted to suit the overall partition performance observed for the LC3 spirals. It should be noted that ongoing testwork will enable a more definitive LC3 LIMN model to be developed.
- With developing a spiral performance model, it is important to note that different feed conditions or different splitter positions will result in different partition curves. Hence, it is important for continuing the testing of many coal types at different operating conditions.

**Partition Comparison Between LD7 and LC3 Spirals**

From the analysis of the 2009 Rangals coal test work data, LC3 partition curves were generated and these are compared to typical spiral (LD7) partition curves in Figure 2. Figure 2 clearly demonstrates that the LC3 can achieve a significant cut-point reduction compared to the LD7 for both the product only and the combined product and middlings separations (from 1.70 to 1.53 RD and 1.92 to 1.78 RD respectively). Testing over a range of feed rates and splitter positions has confirmed the LD7 and similar generation spirals are not capable of achieving this cut-point level. For the product only split, the LC3 displays a similar Ep to the LD7 (Ep of 0.19 for both spirals), while the product plus middlings partition for the LC3 shows better Ep compared to the LD7 (0.13 to 0.23 respectively).

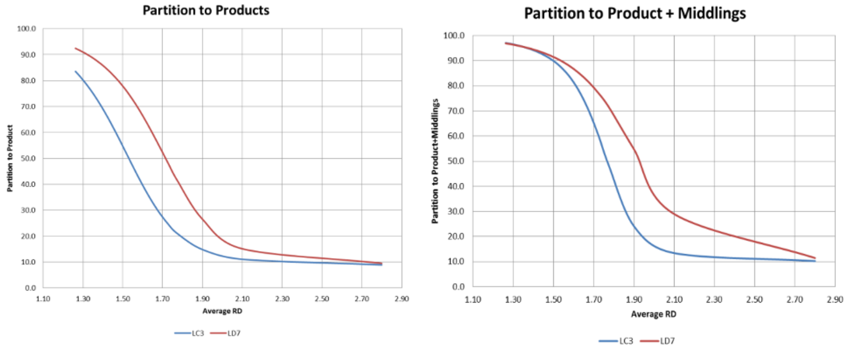


**Figure 2 - Partition comparison of LC3 and LD7**

At the 1.53 RD product only cut, the LC3 shows a higher bypass of low RD material into the middling fraction relative to the LD7 cutting at 1.70 RD, with approximately 10% more Cumulative Float 1.30 material being bypassed into the LC3's middlings fraction. While this level of bypass is not optimal, this F1.30 material is recovered in the LC3's middlings stream. The combined product and middlings partition curves show the LC3 compares very well with the LD7 at low density but offers the benefit of a lower overall cut-point and better efficiency.

**Single Stage Process Yield-Ash Analysis**

To evaluate relative benefits of alternative fines beneficiation options, Limn simulations were used to develop practical full plant yield-ash curves over a practical range of cut-points, for a Hunter Valley thermal coal and a Queensland Rangals coal. The process options all considered dense medium cyclone processing of the coarse coal (+2 mm), whereas three fines options were considered, including standard LD7 spirals, low cut-point LC3 spirals and hindered bed separators. Ultrafine flotation (-0.25+0 mm) was included as part of the processing configuration for the Queensland Rangals coal. The process size splits were:

- Hunter Valley coals (+2 mm to DMC, -2 +0.125 mm to fines, discard -0.125 mm)
- QLD coals (+2 mm to DMC, -2 +0.25 mm to fines, -0.25 +0 mm to flotation)

The "product only" cut-points (D50) of the LD7 and LC3 spirals were kept constant at 1.70 and 1.54 respectively utilising the LIMN sized based models described (Nicol, 2001). For the spirals circuits, options for middlings to product, middlings to reject and middlings recycle were included. The cut-point for the HBS was varied across a range of cut-points from 1.45 to 1.70 with a sized based LIMN model also being applied (Drummond, 2002). Simulation outputs (practical yield-ash plots) for the various options considered are provided in Figure 3 to Figure 5.

For the Hunter Valley coal, it can be seen in Figure 3 that if the site is targeting maximum yield the LD7 with middlings to product offers a slight benefit over the LC3, albeit at increased product ash. However if a more selective ash is required, then the optimum yield-ash curve results from using LC3 spirals with the middlings recycled. For processing Rangals coals, Figure 4 shows that, at lower target product ash levels, there is a small, but distinct, advantage in using the LC3 spirals. Both Figure 3 and 4 show that it is preferable not to send LC3 middlings to reject unless targeting a minimum product ash.

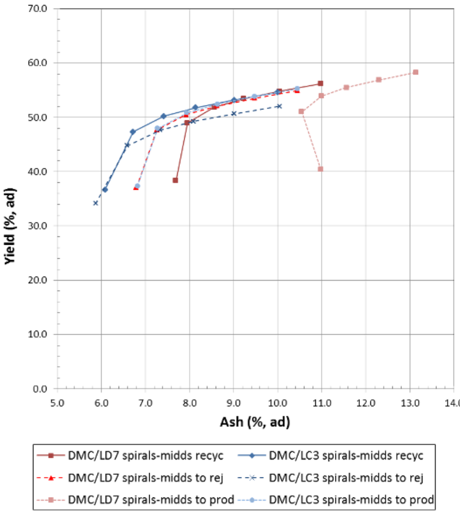


**Figure 3 - DMC/Spirals options yield-ash for Hunter Valley coals**

When comparing LC3 spirals (with middlings recycle) to a TBS, it can be seen in Figure 5, there is little difference in overall plant yield-ash across the majority of the operating range of densities. For very low cut-point requirements (nominally less than 1.4 RD) the modelling suggests the TBS offers a better yield result for equivalent ash.

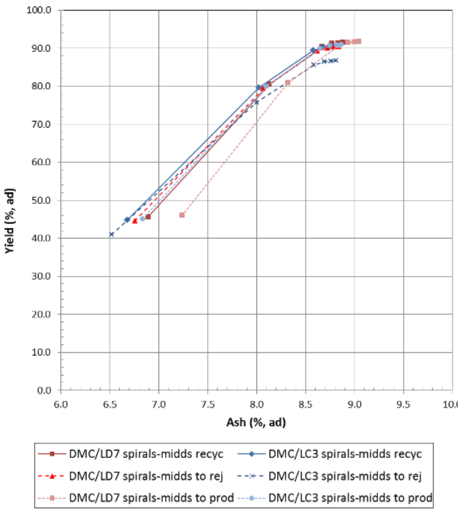


**Figure 4 - DMC/Spirals options yield-ash for Queensland Rangals coals**

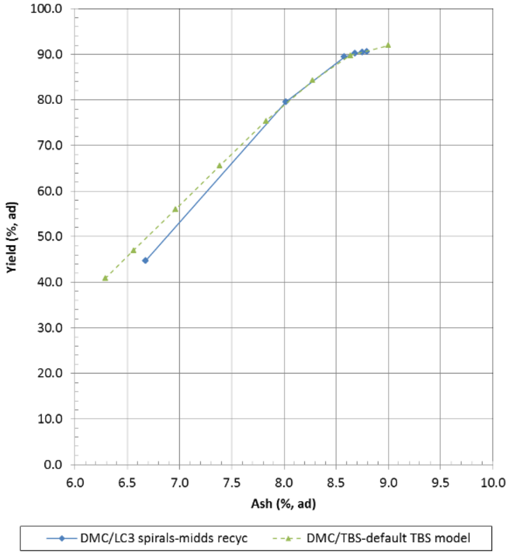


**Figure 5 - DMC/LC3 vs DMC/TBS yield-ash for Queensland Rangals coals**

### Two Stage Process Yield-Ash Analysis

If very low primary cut-point (minimum ash) is required from the LC3 spirals, there needs to be an allowance for recovering or retreating the middlings stream which contains the bypassed CF1.30 material. Apart from the middlings recycle and middlings to product options considered above, a secondary bank of spirals could be utilised. Simulations were carried out to consider secondary spiral treatment options. Secondary spirals were used to retreat the primary spiral middlings, and potential spirals circuit yield-ash results are shown in Table 1.

**Table 1 – Yield-Ash Allowing for Secondary Spirals Treatment of Primary Middlings**

| Circuit/Stream | HV Nominal | | Qld Rangals | |
|---|---|---|---|---|
| | Yield (% CPP Feed) | Ash (%) | Yield (% CPP Feed) | Ash (%) |
| *Primary Spirals* | | | | |
| Feed | 17.7 | 30.3 | 21.3 | 15.2 |
| Prod | 9.8 | 14.8 | 13.8 | 8.0 |
| Midds | 2.8 | 17.5 | 4.3 | 10.5 |
| Rej | 5.2 | 66.4 | 3.2 | 53.1 |
| *Secondary Spirals (Prim Midds Rewash)* | | | | |
| Feed | 2.8 | 17.5 | 4.3 | 10.5 |
| Prod | 1.8 | 11.9 | 2.9 | 8.4 |
| Midds | 0.5 | 16.5 | 0.9 | 10.5 |
| Rej | 0.5 | 37.5 | 0.6 | 21.1 |
| *Overall 2 Stage Spiral Circuit Yields* | | | | |
| Prim + Sec Prod | 11.5 | 14.4 | 16.7 | 8.1 |
| Prim Prod + Sec Prod & Sec Midds | 12.0 | 14.5 | 17.5 | 8.2 |
| Single stage w/midds recyc | 12.1 | 14.3 | 17.6 | 8.2 |

From the modelling completed, it can be seen that the LC3 two stage circuit options don't appear to offer any obvious yield-ash benefit over the single stage with middlings recycle option described above. However, it should be noted that there has been no optimisation of the two stage simulations in terms of developing a separate set of partition curves for a secondary duty in comparison to a primary duty, nor have there been any changes considered in the relative primary and secondary spirals product and middlings cut-points to potentially improve overall circuit outcomes. Ongoing testwork is being undertaken in effort to better define these differences, but it is expected that there may be opportunity to achieve a better overall spirals circuit result with optimised two stage processing.

## Conclusion

The LC3 is a relatively new spiral that offers the ability to achieve low cut-points not normally achieved with spirals. The ability to target a cut-point in the 1.45 to 1.60 RD range with spirals opens up new possibilities for coal processors who prefer the operational and maintenance simplicity of spirals. Depending on the washability of the coal and the cut-point being targeted, there may need for an allowance for middlings recycle or retreatment.

The simulation work completed shows that the LC3 spirals offer a metallurgical benefit over standard spirals where a more selective separation is desired. Generally it can be concluded that the optimal spiral yield-ash performance is achieved via single stage LC3 processing with middlings recycle. Although LC3 data shows loss of low density coal due to the low cut-point, middlings recycle provides a workable solution. Based on the available simulation model developed for the LC3s, single stage treatment with middlings recycle actually provides an equivalent, or slightly better, outcome compared to retreating middlings in a second stage of spirals. More test work is planned to confirm this finding over the next year.

Site test work to date has highlighted that application of the LC3s can produce results similar to the laboratory tests in both cut-point and Ep which allows the process designer to have a high degree of confidence in scale up from laboratory test work. Additional test work is planned on a wide range of coals operating across a range of conditions to quantify the full potential benefits of the LC3s.

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# Reviving Antiquated BATAC® Jigs in India through Technological Upgrades

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**Abstract: -**
Deteriorating quality of run off mine (ROM) coal, meeting environmental regulations and quality stipulation of end users, made coal washing in India to assume unprecedented importance. In order to increase its washing capacity, Coal India Limited (CIL) proposed setting up of many coal washeries, some of which are already under construction. Many private companies are also mulling to set up their own washeries. However, these new washeries shall take time to become fully operational. The existing washeries of CIL have aged significantly and therefore run below their installed capacity. Until the new coal washeries come up, these existing washeries may serve as the life line of washed coal, provided that they are upgraded suitably. Many existing washeries have BATAC® jigs as the main washing equipment and this paper details the modifications carried out in them for reviving and improving their efficiency resulting in improvement of capacity utilization of such washeries which made CIL look into such renovations more seriously. These upgrades breathed fresh life into those antiquated machines making them capable to handle the changed feed coal more efficiently and produce clean coal of required quality.

**Key Words: - Run Off Mine (ROM), Coal India Limited (CIL), Washeries, BATAC®, Technological Upgrades, Efficiency, Antiquated**

## Introduction

India is one of the largest producers[1] of non- coking coal in the world and also one of the largest user[2] of the same. The coking coal reserves of India are not impressive hence it has to import huge quantity of hard coking coal from other countries. However, it might be surprising to note that even though the reserves of non- coking coal in India is remarkable, still it imports a huge quantity of non-coking coal too from other countries. The reasons for this perplexing reality maybe several, which include costs, quality, availability etc. As the demand for power in India is increasing with each passing day and in order to make electricity available to every citizen, supply of non- coking coal has to increase which is going to remain as the primary source of energy for India even with the renewed focus on renewable energy sources. This fact has been realized in many forums that India should become self- dependent for non- coking coal which will help it save billions of dollars of foreign exchange and also enable it reduce its current account deficit (CAD). With this in view the government has set a stiff but achievable target of 1[3] billion tons coal production by Coal India Limited (CIL), India's coal mining behemoth which is also its largest producer, by the year 2019[3]. It was also realized that just by increasing production won't serve the purpose as quality is also another major criteria for coal usage. In addition to that environmental concerns play a significant role in coal usage and hence in order to handle both the issues the total coal washing capacity of India was to be increased. With this in view several coal washeries have been planned by CIL and Ministry of Coal in order to cater to the increased demand of washed coal. Though this is a good start but such washeries will take time to become fully functional due to several reasons which are outside the scope of this paper. In order to resolve these issues CIL came up with another novel way i.e. to refurbish the old washeries, many of which have outlived their lives but are being still operated as they are indispensable. Such washeries are currently running on partial capacity, reasons of which shall be explained in later parts of this paper and such refurbishment were aimed to increase their capacity utilization and increase the productivity of the plant. The BATAC®[4] jig, which is a trademark equipment of MBE Coal and Mineral Technology GmbH, is a pneumatically operated under- pulsated jigging machine which has been successfully used in beneficiation of different types of coal. This paper discusses in detail such refurbishments which were taken



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up in BATAC® Jig washeries and the effect they had on the productivity and quality of the washed coal from such antiquated coal washeries.

**Present Scenario**

The last fully operational greenfield washery to come up in CIL was at Bina in the year 1999(5). This itself is more than fifteen years ago and after that no new washery of CIL have started production. The new washeries for which different parties have received orders are under execution stage. Others are still under planning stage or yet to get certain approvals. As a result the older washeries are still being operated which are well past their prime and life but are indispensable. The following table, though not comprehensive, gives a general idea about how old the currently working washeries of CIL/ SAIL are and is still being operated as the planned new washeries have not yet started operation and still has a long waiting period before all the proposed washeries having cumulative washing capacity of 100(7) million tons actually start operating.

Table1: Major Operating Jig Washeries of CIL and Steel Authority of India Ltd (SAIL)

| Company Name | Location | Jig Type | Type of Coal | Year of supply/ commissioning of Jigs(6) |
|---|---|---|---|---|
| Central Coalfields Ltd | Sawang, Jharkhand | BATAC® Jig | Coking | 1985 |
| Central Coalfields Ltd | Rajrappa, Jharkhand | BATAC® Jig | Coking | 1984 |
| Central Coalfields Ltd | Kedla, Jharkhand | BATAC® Jig | Coking | 1984 |
| IISCO (now SAIL) | Chasnalla, Jharkhand | BATAC® Jig | Coking | 1987 |
| Bharat Coking Coal Ltd | Bhojudih, W. Bengal | BATAC® Jig | Coking | 1988 |
| Bharat Coking Coal Ltd | Madhuband,Jharkhand | BATAC® Jig | Coking | 1987 |
| Central Coalfields Ltd | Piparwar, Jharkhand | BATAC® Jig | Non-Coking | 1998 |
| Northern Coalfields Ltd | Bina, Uttar Pradesh | ROMJIG® | Non-Coking | 1997 |
| Central Coalfields Ltd | Kargali, Jharkhand | ROMJIG® | Non-Coking | 1997 |

Most of the above listed washeries are currently operating in under- capacity(8) conditions as the coal characteristics on which they were designed have changed drastically. The washing equipment which is BATAC® Jig in this case also had antiquated controls as much technological advancement have been carried out in the past decade which has tremendously improved the performance of the jig. MBE Coal and Mineral Technology Pvt. Ltd. (formerly Humboldt Wedag- Coal and Mineral Division), took up the task of reviving these antiquated BATAC® jigs by refurbishing the controls, changing the mechanical parts etc. in order to make them ready for the present coal characteristics. This improved the efficiency of the jig and also reduced the downtime which eventually increased the production of the whole plant. In addition, due to these refurbishments product quality control became easier and better. Few equipment subsequent to the jig were also refurbished so that they did not prove to be a bottleneck anymore to the increased production from the jigs. This proved to be a cost effective way to revive the BATAC® jigs to their earlier conditions and help them function to their best possible level till the new washeries come up and take their place.

**Major modifications carried out**

Many BATAC® jigs supplied to CIL and SAIL as per table 1 were upgraded and renovated. These renovations included not only mechanical renovations but also change of controls and other instrumentation and/ or electrical components. The most common changes that were brought upon these antiquated BATAC® jigs in order to upgrade them are listed in table 2.

Table 2: Common(9) renovations carried out in the antiquated BATAC® jigs

| Sl. No. | Description | Type |
|---|---|---|
| 1 | Renovation of eroded Jig hutch walls | Mechanical |
| 2 | Dismantling and supply of new discharge gate system | Mechanical |
| 3 | Supply of float/ scanning device | Mechanical |
| 4 | Supply of hydraulic power packs | Mechanical |

| | | |
|---|---|---|
| 5 | Modification of existing blower pressure control system | Instrumentation |
| 6 | Supply of new PLC based control panel | Instrumentation |
| 7 | Supply of ultrasonic float measurement and gate measurement system | Instrumentation |
| 8 | Renovation and maintenance of existing air chamber | Instrumentation |

Apart from the major renovations which are listed in table 2, other maintenance works which were necessary for the proper functioning of BATAC® jigs were also carried out. These included activities like maintenance of fixed screen underhopper, addition of some important feedback like compressor air pressure etc. to the control panel which were absent in the earlier system, change of old control cables, supply of Local Push Button Station (LPBS), maintenance of the bucket elevator etc. These renovations were carried out in order to ensure that the jig operates more efficiently with less break- down in a hassle free manner.

**Comparative Study of Characteristics of the old and new BATAC® Jig system**

The principles of jigging remained the same in both the older jigs and the new jigs. However, the ways in which these principles were affected were changed over the due course of time with better mechanical design and better controls. It is noteworthy that whatever systems were used in the older BATAC® jigs were up-to-date and modern for that time. With improvements in electronics and with continuous research and development, the control systems became more efficient and less cumbersome and these improvements were incorporated into the BATAC® jigs at appropriate times. As most of the BATAC® jigs supplied to CIL and SAIL were during the 80's and 90's, ageing of the machines was normal. Also, due to changes in design as well as the controls, availability of spare parts became one of the major bottlenecks of the older machines which resulted in further deterioration. As a result it was decided that in order to run the machine effectively it was necessary to update and upgrade both the machine and its controls. Following table compares the most significant differences between the old and the new system.

Table 3: Comparison between Old and New BATAC® Jig System

| Sl. No. | Old System | New System |
|---|---|---|
| 1 | Card based control panel was used in the older BATAC® jigs which consisted of protonic controller and relay logic, hence interlocking was only possible through hardware changes | New PLC based control panel is being provided which made the control system more efficient and less cumbersome and any changes in logic could be made through simple changes in the software only and no hardware changes were required |
| 2 | Number of bed height measurement mechanisms (scanning device) and discharge gates across the width of old BATAC® jigs were fewer, at a minimum width of 2m | Number of bed height measurement mechanisms (scanning device) and discharge gates across the width of the new system for BATAC® jigs have been increased, at a minimum width of 1.5m |
| 3 | Measurement of bed depth and discharge gates' level were done by Linear Variable Differential Transformer (LVDT) which was based on the principle of conversion of mechanical displacement into electrical signals | Measurement of bed depth and discharge gates' level in the new system are being done by using new improved Ultrasonic sensors, which is more sensitive and require less maintenance |
| 4 | Level of discharge gates was controlled by a centralized hydraulic system which meant that the level of discharge gates at all times remained similar to each other | Each discharge gate is being supplied with an independent control and dedicated hydraulic system which meant that gates could operate independent of each other and was only dependent on the feedback of its associated scanning device and control panel logic |

| | | |
|---|---|---|
| 5 | Working air control unit was of Askania type | New working air control unit is being provided with AUMA actuators which have been further improved upon in India by using pneumatic actuators which are more sensitive |
| 6 | Air bypass system of the air control unit, which is the heart of BATAC® jig, was carried out through motorized valves | Air bypass system of the air control unit is being carried out through electro- pneumatic valves which is more sensitive and has less response time |
| 7 | Hutch section of the old BATAC® jigs consisted of Boat and H section | New improved hutch section design is being provided which replaced the old Boat and H section which made pulsations more smooth and was maintenance friendly |
| 8 | Local push button station (LPBS) was not provided for the discharge gates | New LPBS is being provided at each discharge gate for ease of doing maintenance |

It is interesting to note that not many changes were carried out in the mechanical design of the BATAC® jig apart from the change in hutch and air chamber and the regular maintenance that was carried out. This feature of the BATAC® made it more suitable for modernization as the changes to be carried out were mostly in the controls and instrumentation part. The original mechanical design of BATAC® is still relevant in today's mineral processing industry and works with superior efficiency. Once these changes were carried out in the antiquated BATAC® jigs, they were re- commissioned and the controls and logic were set to suit the present coal characteristics and product quality requirements. The results obtained by the client were satisfactory and as a result repeat orders were generated which proved that this exercise of modernizing the antiquated BATAC® jigs was indeed novel, fruitful and economic and produced the desired results as required by the client.

**Advantages of the new system**

The most important and also the most visible advantage of carrying out such a renovation work was that thorough maintenance of the antiquated BATAC® jigs was carried out. Such maintenance could only be carried out by experts which improved the working of the jigs to a large extent. Unwanted leakages were stopped, proper cleaning of the instruments and other parts could be done etc. Apart from the very much evident advantage there were many others which are worth mentioning. They are :

(1) Increase in performance of the plant with substantial reduction in downtime(10)

(2) As the control panel was PLC based, changes in logic could be made very easily by just changing the software program. No hardware links were required in order to change the logic as was required in the old system. This made the control system less cumbersome and more resilient. It could now be easily adapted with the changing raw coal feed characteristics

(3) The PLC system reduced the amount of control wires requirement by about 25% which proved to be very economic

(4) Increase in number of discharge gates made the discharge system more sensitive

(5) As the discharge gates are now made independent of each other, the quality control became more efficient. The activity of each gate was now solely controlled by the depth of bed which the floats measured and not dependent on other gates reducing the misplacement of cleans into rejects

(6) Pneumatic air flaps used were much more efficient and also reduced the number of motors required. Also option was provided in order to operate the pneumatic valves in manual mode, which enabled the maintenance of such valves in online condition without stopping the whole jig

(7) Ultrasonic level sensors used are more accurate and sensitive. Hence they helped in adequate quality control and proper working of the discharge gates reducing misplacement.

**Conclusion**

There are many BATAC® jigs working throughout the world. Many of them are decades old which may result in inefficient working which is very normal. Also, spare parts availability of such jigs is a problem as many of the controls have become redundant and not available anymore. Due to these reasons many plant owners might think of replacing the machine. However, in some machine suitable changes could be made in order to upgrade the same old machine into a newer model. Although, it should always be kept in mind that such renovations and refurbishments are not always economically possible in all the old machines as they depend on their actual condition and many other factors. The antiquated BATAC® jigs of SAIL and CIL provided a unique opportunity to MBE Coal and Mineral Technology Pvt. Ltd. and after several deliberations and careful analysis, such a job of renovation and refurbishment was accepted and carried out successfully. Due to their success CIL has given repeat orders(9) for their other jigs so that similar advantages could be realized in those machines too. This has allowed the older machines to work more efficiently and reduced the downtime to a great extent which increased the overall productivity of the whole plant. Until the time the new washeries as planned by CIL come up, these refurbished jig washeries shall continue to act indispensable and produce clean coal of required quality with increased productivity when compared to the production few years back when such modifications and renovations were not carried out. This could serve as an excellent example to other clients as well who have old BATAC® jig washeries and who want to explore such an option of renovation.

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# Effect of some operating variables on the performance of a 100 mm Heavy Medium Cyclone treating high ash Indian Coking Coal

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## ABSTRACT

Out of the total reserves of Indian coals about 83% constitute non-coking coal, 14% coking coal and the rest are others. Coking coal is an essential input for production of Iron & Steel through blast furnace route. To save steel industry facing acute dependence on imported coking coal, domestic availability of coking coal in desired quality has become imperative. The good quality coking coals of the upper seams are fast depleting leaving behind the inferior quality lower seam coal.

Tests were carried out on a 100 mm heavy medium cyclone test rig treating coal in the size range - 6 +0.5 mm. A total of twenty seven experiments were carried out by varying the parameters like vortex diameter, spigot diameter and feed pressure. The effect of these variables on the performance of the 100 mm Heavy Medium Cyclone (HMC or HM Cyclone) was investigated. The results indicate that it is possible to achieve about 24% clean coal at an ash content of 17.3% at a feed pressure of 5 lb/in$^2$, vortex diameter of 42.5mm and spigot diameter of 28mm. The tromp curves obtained from operating the cyclone gave a probable error of 0.04, indicating good separation efficiency.

*Key words:* coal cleaning, heavy medium cyclone, magnetite, specific gravity of separation

## INTRODUCTION

Coking coal is an essential input for production of Iron & Steel through blast furnace route. To save steel industry facing acute dependence on imported coking coal, domestic availability of coking coal in desired quality has become imperative. The good quality coking coals of the upper seams are fast depleting leaving behind the inferior quality lower seam coal or LVC (Low Volatile Coking) coals. The cleaning of the Indian coals requires crushing to a reasonable size

 
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for liberation of ash forming minerals and suitable technology adopted for washing this size fraction is processing them through Heavy Medium Cyclone. Further for processing of the LVC (Low Volatile Coking) coals or lower seam coal, the coal is to be first deshaled and crushed to 6 mm and for processing the coarser fractions (- 6 +0.5 mm),HM Cyclones are the only efficient washers. It has been established worldwide that coal washing of intermediate sizes (- 6 +0.5 mm), HM cyclone is the best separator. (1-4)

Around seventy percent of the existing Indian coal washeries use heavy medium cyclones (HMC) as the main unit operation in the washing circuit. Most of these washeries are old and feed characteristics had drastically changed over the period, as a result it has become difficult to optimize the process which ultimately reduced efficiency. With this view, tests were carried out to study the performance of 100 mm heavy medium cyclone test rig treating LVC coal in the size range -6+0.5 mm.

**EXPERIMENTATION**

The LVC coal from Muradih was taken for the study. For beneficiation by HMC scheme, the "as received" sample was crushed to below 75 mm in a double roll crusher and a representative portion of crushed coal was screened at 6 mm.The screened fraction of -75+6 mm was deshaled at density of 1.80 .The deshaled product of -75+6 mm size fraction was crushed to below 6 mm and crushed product was then mixed with untreated minus 6 mm fraction to form the sample of -6+0 mm. The -6+0 mm size fraction was screened at 0.5 mm. Thereafter, Tests were carried out on a 100 mm heavy medium cyclone test rig treating coal in the size range -6+0.5 mm. Detailed washability studies were carried out for size - 6 +0.5 mm .The experimental set up of 100 mm diameter H M Cyclone test rig is shown in Figure 1.

Figure-1: H. M. Cyclone Test Rig

The Mayers curve is depicted in Figure 2. The medium of desired specific gravity is prepared in the slurry tank. Finely ground magnetite (95 percent passing through 44 micron) was used to prepare the heavy medium. The medium is fed to the cylindrical vessel and Coal is fed from the top at the rate of desired quantity to maintain pulp density and suitable media to coal ratio. The media and the coal particles are mixed in the slurry tank and the mixture is fed to cyclone. There is a by-pass arrangement also to find out the specific gravity as well as ratio of media and coal fed to the cyclone. The by-pass line in the slurry tank is adjusted to feed the cyclone at definite pressure. The products of the cyclone are passed over launder divided into two parts for cleans and sinks. The coal particles (cleans and sinks) coated with magnetite are water sprayed, cleaned and collected. The dilute media is collected in a separate tank where the media settles and is reused.

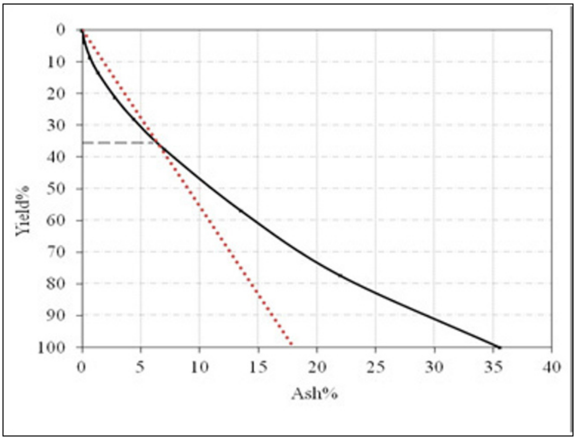


Figure 2: Mayers curve for coal size - 6 +0.5 mm mm

Twenty seven experiments for each diameter cyclone were carried out to study the effects of Vortex Finder Diameter (VFD), Spigot diameter (SPD) and Feed Pressure (P) on the quality and quantity of the cyclone products (5). The level of the parameters and efficiency data for 100 mm diameter HM cyclone treating coal of size - 6 +0.5 mm is shown in Table 1.

## RESULTS AND DISCUSSION

From the washability data of coal size 6-0.5 mother theoretical yield at 18 % ash level is 35.6%, where the corresponding rejects being 64.4 % at ash of 45.2 %.The cut density is 1.59 and the NGM at this gravity is 36 %, which categorize the coal as difficult-to-wash coal. Effect of the operating parameters on 100 mm diameter HM Cyclone is shown in Table 1.

Table 1: Efficiency data for 100 mm diameter H.M. Cyclone treating coal of size - 6 +0.5 mm

| Test No. | Feed Pressure lb/in2 | Vortex Finder Diameter mm | Spigot Diameter mm | Organic Efficiency | d50 (density of Separation) | Ep (Ecart Probable Moyen) |
|---|---|---|---|---|---|---|
| 1 | 5 | 35 | 21 | 88.60 | 1.96 | 0.010 |
| 2 | 7 | 35 | 21 | 95.20 | 1.96 | 0.030 |
| 3 | 10 | 35 | 21 | 95.10 | 1.97 | 0.010 |
| 4 | 5 | 35 | 28 | 57.60 | 1.41 | 0.050 |
| 5 | 7 | 35 | 28 | 53.40 | 1.39 | 0.040 |
| 6 | 10 | 35 | 28 | 41.60 | 1.36 | 0.030 |
| 7 | 5 | 35 | 35 | 46.70 | 1.39 | 0.110 |
| 8 | 7 | 35 | 35 | 41.50 | 1.36 | 0.070 |
| 9 | 10 | 35 | 35 | 35.50 | 1.35 | 0.050 |
| 10 | 5 | 42.5 | 21 | 94.60 | 1.96 | 0.012 |
| 11 | 7 | 42.5 | 21 | 95.30 | 1.97 | 0.018 |
| 12 | 10 | 42.5 | 21 | 65.20 | 1.70 | 0.250 |
| **13** | **5** | **42.5** | **28** | **72.70** | **1.51** | **0.040** |
| 14 | 7 | 42.5 | 28 | 69.30 | 1.48 | 0.050 |
| 15 | 10 | 42.5 | 28 | 72.30 | 1.50 | 0.060 |
| 16 | 5 | 42.5 | 35 | 57.30 | 1.43 | 0.060 |
| 17 | 7 | 42.5 | 35 | 49.30 | 1.41 | 0.100 |
| 18 | 10 | 42.5 | 35 | 49.00 | 1.41 | 0.060 |
| 19 | 5 | 50 | 21 | 67.00 | 1.57 | 0.090 |
| 20 | 7 | 50 | 21 | 63.20 | 1.53 | 0.090 |
| 21 | 10 | 50 | 21 | 68.20 | 1.60 | 0.110 |
| 22 | 5 | 50 | 28 | 28.80 | 1.36 | 0.060 |
| 23 | 7 | 50 | 28 | 38.60 | 1.40 | 0.040 |
| 24 | 10 | 50 | 28 | 37.30 | 1.39 | 0.060 |
| 25 | 5 | 50 | 35 | 39.80 | 1.42 | 0.060 |
| 26 | 7 | 50 | 35 | 27.10 | 1.35 | 0.150 |
| 27 | 10 | 50 | 35 | 26.50 | 1.35 | 0.050 |

Based on the above findings the cyclone parameters in terms of yield and ash content of the cyclone cleans and sinks at different operating conditions and also considering the efficiency parameters the H. M. Cyclone was standardized and the final parameters which meet the required quality is shown in Table 2.

Table 2: Standardization of H. M. Cyclone Parameters

| Cyclone diameter: | Pressure, $lb/in^2(g)$ | Vortex diameter, mm | Spigot diameter, mm | Cleans wt.% | Cleans ash% | Rejects wt.% | Rejects ash% | Feed ash% |
|---|---|---|---|---|---|---|---|---|
| 100 mm | 5 | 42.5 | 28 | 24.0 | 17.3 | 76.0 | 40.1 | 34.6 |

## CONCLUSIONS

Hence, it may be concluded that for recovery of clean coal at desired ash level, crushing of lower seam coal should be done to below 6 mm after deshaling, for optimum recovery. The effect of these variables on the performance of the 100 mm Heavy Medium Cyclone (HMC or HM Cyclone) was investigated. The results indicate that it is possible to achieve about 24% clean coal at an ash content of 17.3% at a feed pressure of 5 $lb/in^2$, vortex diameter of 42.5mm and spigot diameter of 28mm. The tromp curves obtained from operating the cyclone gave a probable error of 0.04, indicating good separation efficiency. Since, the NGM of the lower seam coal is very high, it is always beneficial to wash the coal in Heavy medium cyclone. The rejects which is as high as 76% at 40% ash level can be used for power generation. This is because when the coal was crushed to 6 mm more and more coal was liberated. The yield can be further enhanced by flotation of -0.5mm coal fines.

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# Part IX
# Crushing, Grinding, Screening and Sizing Specification

MECHANIZED CRUSHING & SCREENING PLANT

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*Abstract. This article presents a method and a mechanized crushing and screening plant designed to sort loading of raw coal into gondola cars, in which the ROM is separated by size, thus allowing for a mechanical waling of rock and separate loading of coal, first small and then large grades to prevent floating the coal away from gondola cars during movement, and protecting the environment from pollution, it also presents quality design characteristics serving to calculate the performance of the complex providing the given performance and the crusher energy saving.*
*Keywords: method, complex, crushing, sorting, loading, coal, parameters*

One of the problems of the coal industry is sorting of the run-of-mine coal by sizes for treating and shipping to consumers.

In [1-4] the authors present a variety of ways and technical means of sorting and mechanized picking up the rock.

This paper, based on the patented method of mechanical coal preparation [5] presents the crushing and screening mechanized complex of high performance for crushing and classification of rock mass, which can be used for mechanical rock waling, which is under research in LLC "Sibniiugleobogaschenie."

The mechanized crushing and screening complex is designed for separate loading of coal grades into gondola cars and for preventing dust and small grades of coal from floating away during transportation, which helps to prevent environmental pollution.

The method of crushing, screening and separate loading of the rock mass into gondola cars and the complex for its implementation include: crushing, sorting, transportation, accumulation and loading of the rock mass, and is characterized by the following: the crushing of the ROM is done to pieces in a roll crusher by point impacts of the teeth and the back of their holders; sorting of pieces by size is done through slotted screens gaps and spacing between slotted screens and the roll due to its rotation and installation of teeth in a spiral; small and large grades of the ROM coal are bypassed separately to the conveyors moving at different speeds, and some fines got through the roll spacing and screens are separated from the large grades on the slotted screens of the upper scraper conveyor; accumulation of small and large grades is into separate bins and the loading of the rock mass into gondola cars is done in turn – first small and then large grades.

The Complex realizing the method of crushing, sorting and separate rock mass loading into gondola cars comprises: a receiver, a roll crusher, conveyors, chutes and bins; and it differs by the fact that a receiving device is made of arc slotted screens installed at a depositional gradient for POM coal at 35-45º, covering 2/3 of the roll diameter from the bottom, having spacing with the crusher roll larger than the width of the slotted screen gaping, and the roll teeth are set along the spiral and directed opposite the roll rotation, while the back of the rolls holders is wedge-shaped with 45-55º gradient to prevent the clinch of a tooth in spacing by the lumps of ROM coal; the conveyors for transporting large and small grades are arranged one above the other and have different speeds - for large grades it is slower, for small ones it is faster; the upper conveyor for large grades has a slotted screen in the bottom to bypass the small grades got through the spacing and formed during transportation; bins are equipped with chutes for ROM coal to feed and discharge it separately into a gondola car, chutes from conveyors to the bins are

 
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directed to different bins, and the chutes for discharge – to the one, through which it passes consistently first small grades and then large ones of ROM coal into a gondola car.

A method of crushing, sorting and the complex of separate loading of the rock mass into gondola cars are illustrated on Figures 1-3 with the following notation: 1 – a roll crusher; 2 - slotted screen gap; 3, 4 - roll crusher chutes barriers, 5 – tooth holder, 6 - tooth; 7 – the roll of the crusher; 8 – a chute for large grade bypass; 9 – a conveyor for the large grade; 10 – a conveyor slotted screen for large grade; 11 - a conveyor for small grade; 12 – coal track; 13 – gondola car; 14 - large grade bin; 15 - small class bin; 16 – large grade conveyor chute; 17 - small grade conveyor chute.

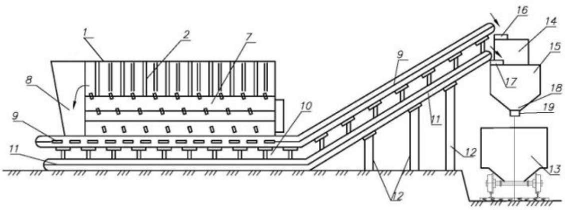

Figure 1-Mechanized grinding-sorting complex (side view)

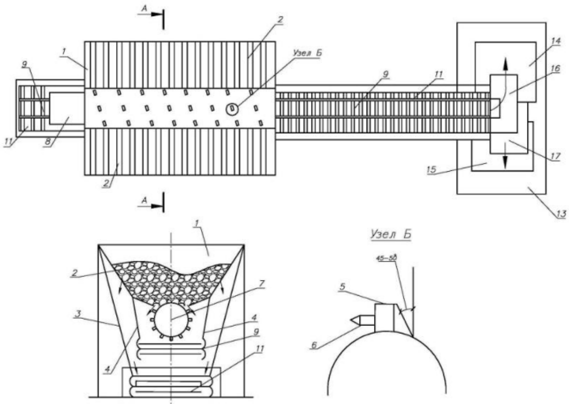


Figure 2-grinding-sorting complex (top view, section a-a, node b)

ROM is feeding into the roll crusher 1, in which small grades of rock mass are separated through the slotted screen gappings 2 and via sloping chutes formed by barriers 3 and 4 inside the crusher bypass to the lower small grade conveyor 11.

Large lumps of ROM are crushed by point impacts of the teeth and the back of their holders 5 and by the tooth itself 6 in contact with the edge.

During rotation of the crusher roll 7 the holders 5 and the teeth 6 turn, crush and move lumps of ROM along the length of the roll 7 due to the installation of the holders 5 and the teeth 6 along the helix.

Non-crushed lumps are moved through the chute 8, located at the end of a roll crusher 1, to the upper large grade conveyor 9, having a slotted screen at the bottom 10 with the size of the gap equal to the size of the gap of the slotted screen of the crusher 2, through which fines coming from the gaps of the slotted screens of the crusher 2 and the roll 7 bypass to the lower conveyor for small grade 11.

Conveyors 9 and 11 are installed on the coal track 12 up to the point of ROM loading into the gondola car 13, where the bins for large grade 14 and small grade 15 are located. Conveyors for large grade 9 and small one 11, are equipped with chutes 16 and 17, and have different directions, because the track is located at the junction of the bins, which are arranged in turn along the ROM loading into the gondola cars, first the large grade bin, and then the small grade one, and the chutes for ROM discharge from the bins 18 and 19 are directed oppositely. This arrangement of the bins 14 and 15 and the chutes 18 and 19 makes it possible to load into the gondola car 13 first small grade, and on top of it the large grade to prevent blowing out the small grades during transportation and environment pollution.

As the amount of small grade is always greater than the large one, the speed of the small grade conveyor should be higher than the speed of the conveyor with a large grade in multiple numbers. The grade is specified by the slotted screen size of the roll crusher 1.

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# Technology of electromagnetic dynamic crushing and milling

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**Summary**

The technology of electromagnetic dynamic breakdown allows to effectively carry out crushing and milling of metallic and non-metallic materials, including rocks, to a given size with simultaneous removal of part of the total moisture and foreign objects. The reduction ratio of crushing and milling per stage can exceed 3,000 times, which is many times higher than the best performance of mechanical crushers and mills. The minimum attained size of particles during milling is 0.5 microns.

A pilot version of a crushing and milling unit for metallic and non-metallic materials with simultaneous removal of part of the moisture and foreign objects with the capacity of 2.0 tonnes per hour was used for milling the iron ore concentrate (IOC) of Kovdorskiy GOK, which contains more than 65.0% of +40 micron particle size and less than 14.0% of 20-40 micron particle size, we used. The dimensions of the crushing and milling unit are 4,500x2,300x3,200 mm and the total weight is 25.0 tonnes. No special base plates or foundations for the operation of the unit are required.

Measurements of the particle size of magnetite before and after milling were taken using the technology of applying diffraction of laser radiation in a convergent laser beam patented by FRITSCH, Germany. At present, this technology is the international standard for quick and reliable determination of the particle size of any substance. The measurements were taken on a multi-purpose ANALYSETTE 22 MicroTec plus laser device with a range of measurements of 0.08-2000 microns.

Measurement results for each sample were automatically printed as files with appropriate names. Each file contained information on the size range of the sample in a graphic and tabular form. The integral values of d20, d40, d80 and d99.9 for each sample mean that d20 is the maximum particle size expressed in sample microns, whose total content does not exceed 20% of the total quantity of all particles in a sample. Thus, d40, d80 and d99.9 are the maximum particle sizes (in microns) of a sample whose total content does not exceed 40%, 80% and 99.9%, respectively, of the total quantity of all particles in a measured sample.

In terms of particle size distribution, the non-milled magnetite in the initial sample whose data represents the weighted average values of several tests, contained less than 35% of particles of less than 40 microns and 15% of particles of more than 150 microns in size at the weighted average particle size of 79.8 microns. In terms of particle size distribution in the test magnetite sample which had been milled once, whose data represents the weighted average values of several tests, the content of particles of less than 40 microns was 78.8%, while the content of particles of 40-100 microns was up to 22% at the weighted average size of particles of 42.4 microns.

Thus, the IOC of Kovdorskiy GOK was milled in a single stage to the standard sizes using a pilot crushing and milling unit for metallic and non-metallic materials with a capacity of 2.0 tonnes per hour. The lack of mechanical impact mills reduces the wear of steel structures of the unit manifold. The specific power consumption is up to 1.0 kW*hour per tonne. Safe operation of the unit is ensured in the range of temperatures from -50˚C to +50˚C. The lack of any foundations and feasibility of installing the unit outside makes it indispensable for breaking down metallic and non-metallic materials to the required particle size distribution under any natural and climatic conditions.

**Keywords**

Electromagnetic dynamic breakdown of solid bodies, crushing and milling unit for metallic and non-metallic materials.



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The technology of electromagnetic dynamic breakdown allows to effectively carry out crushing and milling of metallic and non-metal materials, including rocks, to a given size with simultaneous removal of part of the total moisture and foreign objects. The reduction ratio of crushing and milling per stage can exceed 3,000 times, which is many times higher than the best performance of mechanical crushers and mills. The minimum attained size of particles during milling is 0.5 microns.

The task of finding an effective way of crushing iron ore concentrate (IOC) of Kovdorskiy GOK to the sizes which allow use of magnetite for benefication of coals at the company's processing plants was set for specialists of JSC SUEK. The limits of changes of the size distribution of IOC before milling and after electromagnetic dynamic breakdown following a series of tests are presented in Table 1.

**Table 1: Quality characteristics of magnetite based on results of several tests**

| Condition of magnetite | Quality characteristics of magnetite based on results of several tests |
|---|---|
| Initial magnetite before milling | 1. Magnetite size distribution (%)<br>• 150-500 micron ……………………………….25.0 – 30.0<br>• 40-150 micron ………………………………...45.0 – 50.0<br>• 20-40 micron …………………………………10.0 – 15.0<br>• 0-20 micron ……………………………….……..15.0 – 20.0<br>2. Content of magnetic fractions (%) ……………………..…..96.0 – 97.5<br>3. Hardness on Mohs scale ………………………………...5.5 – 6.0<br>4. Bulk density ($t/m^3$) …………………………..…..……4.5 – 5.0<br>5. Apparent density ($t/m^3$) ………………………..…..….5.0 – 5.5 |
| Magnetite after milling in the crushing and milling unit | 1. Magnetite size distribution (%)<br>• 150-500 micron ………………………………..0.0 – 5.0<br>• 40-150 micron ……………………………….....15.0 – 20.0<br>• 20-40 micron …………………….……………55.0 – 60.0<br>• 0-20 micron ………………….……….……..20.0 – 25.0<br>2. Content of magnetic fractions (%) ………….…….……..97.0 – 98.5<br>3. Hardness on Mohs scale ……………………………..…...5.5 – 6.0<br>4. Bulk density ($t/m^3$) ………………………………………4.0 – 4.5<br>5. Apparent density ($t/m^3$) …………………….…….…….5.0 – 5.5 |

A pilot crushing and milling unit for metallic and non-metallic materials with simultaneous removal of part of the moisture and foreign objects with a capacity of 2.0 tonnes per hour was used for milling the IOC of Kovdorskiy GOK. The dimensions of the crushing and milling unit are 4,500x2,300x3,200 mm and the total weight is 25.0 tonnes. No special base plates or foundations for the operation of the unit are required.

Samples of the IOC from Kovdorsky GOK for milling were obtained at the magnetite warehouse of Polysaevskaya CPP of JSC SUEK Kuzbass. The minimum weight of an increment sample when sampling manually and the number of samples required for obtaining magnetite were determined in accordance with GOST 17495-80.

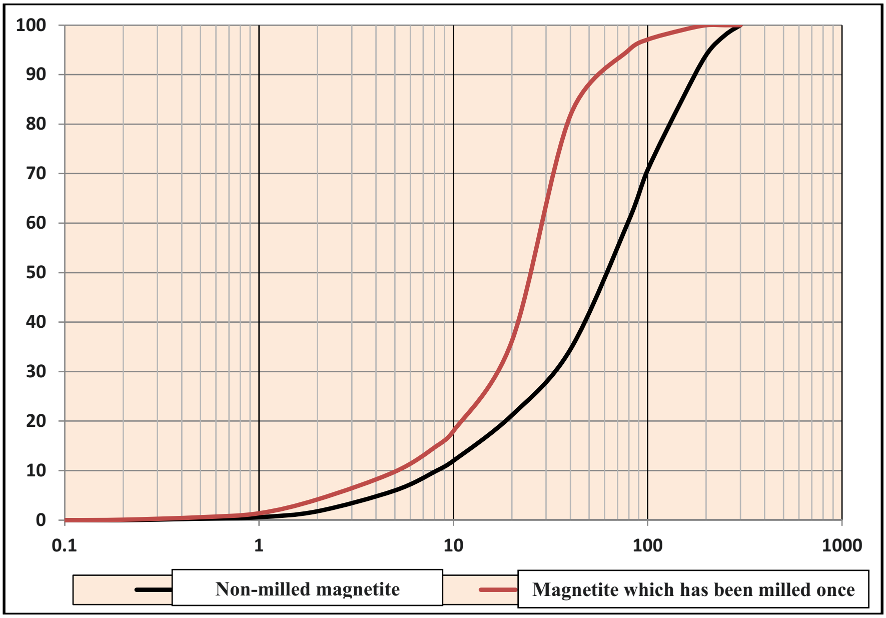
Measurements of the particle size of magnetite before and after milling were taken using the technology of applying diffraction of laser radiation in a convergent laser beam patented by FRITSCH, Germany. At present, this technology is the international standard for quick and reliable determination of the particle size of any substance. The measurements were taken on a multi-purpose ANALYSETTE 22 MicroTec plus laser device with a range of measurements of 0.08-2000 microns.

Dispersal in liquid medium, which is an optimum method for receiving reliable results was used in all of the tests. Magnetite samples were introduced into a closed circuit with circulating liquid. The built-in programmable ultrasonic radiator with pre-selected mode for each sample ensured quick and efficient separation of the material.

Measurement results for each sample were automatically printed as files with appropriate names. Each file contained information on the size range of the sample in a graphic and tabular form. The integral values of d20, d40, d80 and d99.9 for each sample mean that d20 is the maximum particle size expressed in sample microns, whose total content does not exceed 20% of the total quantity of all particles in a sample. Thus, d40, d80 and d99.9 are the maximum particle sizes (in microns) of a sample whose total content does not exceed 40%, 80% and 99.9%, respectively, of the total quantity of all particles in a measured sample.

The weighted average results of the measurements of the size distribution of non-milled magnetite, which were obtained in a series of tests, and particle size distribution in the test magnetite sample which had been milled once, whose data represents the weighted average values of several tests, are presented in Figure 1.

**Figure 1: Results of measuring size distribution of non-milled magnetite and magnetite which has been milled once**




The graphs are integral curves which represent the results of the measurements of the size distribution of non-milled magnetite and of the magnetite which has been milled once as percentages. The size range data for the samples is shown in Table 2.

**Table 2: Weighted average particle size in microns at various contents (%)**

| Item | Weighted average particle size in microns at various contents (%) | | | | |
|---|---|---|---|---|---|
| 1. | Non-milled magnetite | | | | |
| 1.1. | Particle size distribution (microns) | 0-20 | 0-40 | 0-80 | 0-100 |
| 1.2. | Particle content (%) | 18.2 | 34.5 | 60.6 | 70.5 |
| 1.3. | Weighted average particle size (microns) | 13.8 | 25.6 | 50.4 | 68.8 |
| 1.4. | Variation factor (%) | 1.2 | 1.9 | 1.1 | 1.4 |
| | Magnetite which has been milled once | | | | |
| 2.1. | Particle size distribution (microns) | 0-20 | 0-40 | 0-80 | 0-100 |
| 2.2. | Particle content (%) | 28.2 | 78.8 | 97.4 | 99.2 |
| 2.3. | Weighted average particle size (microns) | 10.5 | 19.8 | 34.2 | 42.4 |
| 2.4. | Variation factor (%) | 1.1 | 1.4 | 1.1 | 1.2 |

In terms of particle size distribution, the non-milled magnetite in the initial sample whose data represents the weighted average values of several tests, contained less than 35% of particles of less than 40 microns and 15% of particles of more than 150 microns in size at the weighted average particle size of 79.8 microns. In terms of particle size distribution in the test magnetite sample which had been milled once, whose data represents the weighted average values of several tests, the content of particles of less than 40 microns was 78.8%, while the content of particles of 40-100 microns was up to 22% at the weighted average size of particles of 42.4 microns.

Additional milling of magnetite did not significantly affect the particle size distribution because of the design setup of the crushing and milling unit. In terms of particle size distribution, the characteristics of the magnetite concentrate which had been milled twice fully corresponded to the fine type (T) of magnetite weighting agent (see Table 3), according to the Temporary technological design standards for CPPs (ВНТП-3-92).

**Table 3: Requirements of the Temporary technological design standards (ВНТП-3-92) for magnetite size distribution**

| Magnetite type as per ВНТП-3-92 | Magnetite yield based on size distribution (%) | | |
|---|---|---|---|
| | 40-300 microns | 20-40 microns | 0-20 microns |
| Coarse (К) | 2-10 | 40-50 | 3-10 |
| Small (М) | 2-10 | 50-60 | 10-25 |
| Fine (Т) | 0-5 | 60-75 | 25-35 |

Thus, the IOC of Kovdorskiy GOK was milled in a single stage to the standard sizes using a pilot crushing and milling unit for metallic and non-metallic materials with a capacity of 2.0 tonnes per hour. The lack of mechanical impact mills reduces the wear of steel structures of the unit manifold. The specific power consumption is up to 1.0 kW*hour per tonne. Safe operation of the unit is ensured in the range of temperatures from -50˚C to +50˚C. The lack of any foundations and feasibility of installing the unit outside makes it indispensable for breaking down metallic and non-metallic materials to the required particle size distribution under any natural and climatic conditions.

Based on the conducted study a specification was prepared for manufacturing of a pilot crushing and milling unit for metallic and non-metallic materials with a capacity of 10.0 tonnes per hour. The key technical specifications of the magnetite mill are shown in Table 4.

**Table 4: Key technical specifications for a magnetite mill**

| Item | Description of the parameter of the mill | Value |
|---|---|---|
| 1. | Capacity (tonnes per hour) | 10.0 |
| 2. | Single stage particle reduction factor (no less than) | 3 000 |
| 3. | Size distribution adjustment mode | Automatic |
| 4. | Mill vibration frequency ($min^{-1}$) | 0.0 |
| 5. | Mill vibration amplitude (mm) | 0.15 |
| 6. | Dimensions (length x width x height) (m) | 4.5x2.8x2.5 |
| 7. | Foundations | None |
| 8. | Location of the unit | Open air |
| 10. | Max total weight of the unit (tonnes) | 30.0 |
| 11. | Supply voltage (V) | 380 |
| 12. | Installed power (kW) | 200 |
| 13. | Specific power consumption (kW*hour per tonne) | Less than 2.0 |
| 14. | Operating frequency (Hz) | 50-60 |
| 15. | Intrinsically safe and explosion proof design of the electrical equipment | Yes |
| 16. | Ambient temperature variation range (˚C) | +/-60.0 |

# ACTIVATION GRINDING OF COAL IN PRODUCTION OF LOW-ASH HIGH-PURITY CONCENTRATES

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**Abstract-** The present investigation is focused on feasibility of activation grinding to expose mineral components and to improve reactivity of surface layers of a particle. Particular attention is given to grindability, demineralization, modification of reactive capacity of organic components, metamorphism of coal substance, type and level of energy effects. It is demonstrated that disintegrating grinding with a free destructive impact is the most efficient process to open intergrown aggregates and to higher the energy state of surface coal layers in combustion and ignition processes. The activating dispersion of long-flame coal at the particle collision velocity V = 100 m/s yields – 20 µm products showing the greatest dispersability. The most favorable for processing product of – 0.25 + 0.02 mm is produced from long-flame and lean coals at V = 40 m/s. Long-flame coal fraction of – 20 µm exhibits the highest ash-content, $A^c$ being as high as 30 % and the removal of this fraction can be considered as a coal enrichment operation.

Significant differences established in the thermal-oxidative destruction kinetics of initial and activated coals and the reduction in activation energy permit to higher oxidation velocity by 20% and to intensify fuel combustion.

It is found experimentally that high demineralization of coals with 1% and lower ash content is feasible by the chemical treatment of mechanically activated coals in an aqueous medium at a centrifugal-planetary mill, operating at shear-and-crushing mode. The process is accompanied with defect formation and amorphisation of mineral component structures: quartz, feldspars, sheet silicates, clays along with improving their activity.

***Key words***: coal, minerals, activation, grinding, metamorphism, ash content, impact crushing

## INTRODUCTION

The interest to low-ash and high-purity coal concentrates as fuel for power stations, a promising energy source for gas turbines, internal-combustion and jet engines, in production of carbonaceous functional materials is ever increasing on the part of science and industry.

The coal beneficiation aims at production of concentrates with less than 5% ash content for the electric power industry. The reduction of ash content to 2% allows considering this coal product as a fuel capable to replace mazut [1-3]. Coal concentrates with ash content less than 1% are also used in other fields [4].

The production of high-purity products is based on selective breaking of organomineral aggregates by physical and chemical processes for coal beneficiation. In the present research two high power-intensive processes for coal grinding are studied in terms of demineralization. These processes exert different specific effects on opening of mineral components and destruction of an organic matter in coals. However any mechanical effects are accompanied with certain changes in properties of organic and mineral components in coal, and the term “activation grinding” is based on this factor.

The first of the grinding processes deals with the selective breaking of organomineral components and liberation of mineral components by size, suitable for the follow-up efficient physical and flotation processing of coal. This goal can be achieved at free-impact disintegrators. The other process is oriented to more sophisticated transformations, viz., defect formation and amorphization of mineral structure with enhancing their chemical activity to dissolution in an aggressive media. To realize the second process the researchers used a centrifugal-planetary mill producing an attrition-crushing effect [5].

In grinding at a disintegrator the surface layers of coal particles undergo destruction and conversion into a higher reactivity state, thus intensifying the conbustion in ignition and inflammation stages, in particular. In the centrifugal mechanical activation the structural transformations concern volume of

 
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particles, herewith the bulk of coal passes into an active reactive state, being of prime importance for organic component extraction and chemical demineralization processes.

As for ecological aspects, the combustion of fuel activated by grinding provides the reduced emission of $CO_2$, $SO_3$, NO, and solid particles [6].

## EXPERIMENT

The study objects were of D- and T-metamorphism stage coals originated form Kuznetsky Coal Basin. The activation grinding was performed at IA35 disintegrator with a free fracturing blow, Estonia, an impact microgrinder Culatti, Keinfeld Co., Germany, and M-3 centrifugal-planetary mill, designed at Institute of Geology and Mineralogy, SB RAS, Novosibirsk, Russia [5].

Two experimental mechanical activation modes were selected: 40 m/s and 100 m/s collision velocity for rods and particles in the disintegrator, 20 m/s and 40 m/s velocity in Culatti grinder; and mechanical treatment of 40 g batches in M-3 mill.

The performance of different activation grinding techniques was considered in terms of feasibility to beneficiate coals by physical methods with the use of heavy media of 1.3 g/cm$^3$ and 1.4 g/cm$^3$ in density and by chemical demineralization processes. The chemical treatment was realized in medium of 0.6% sodium silicofluorite in 3% sulfuric acid at 80$^0$ C for 2 h. [7]. Experimental flowsheet and results are given in Fig. 1.

### Variations in dispersion ability, mineralization and activation energy of coals under activation grinding

The dispersion ability of coals was evaluated by the yield of -0.25 + 0.02 mm and – 0.02 +0 mm classes [8]. The first fraction is more favorable for separation and beneficiation processes as the fine product in a greater degree meets requirements for flaring combustion in heat power industry and appears promising in chemical production.

Coal metamorphism and velocity *V* of particles - breaking elements collision are crucial factors for comminutability of the material (Fig. 2). The maximum yield of -0.25 + 0.02 mm fraction is at approximately 40 m/s velocity at first and second impact disintegrators.

It is essential to focus on behavior of minerals in disintegration process. The X-ray phase analysis of long-flame coals indicates the presence of quarts (d/n 4.27; 3.24 A$^o$), kaolinite (d/n 7.1;3.57; 3.03 A$^o$), plagioclase (d/n 4.04; 3.79; 3.20 A$^o$), and mica (d/n 10.0 A$^o$). Disintegrating grinding both opens minerals and causes changes in mineral structure. It is observed essential reduction in quarts reflex and its content in the light product, plagioclase is not fixed, whilst kaolinite reflexes remain unchangeable.

Increase in collision velocity up to 100 m/s sharply intensifies grinding, and the yield of - 20 μm lean coal fraction reaches 81.6%, the similar parameters for long-flame coal are appreciably lower and amount to 21.8%.

The differentiation of fine products by ash content is performed (Fig. 3). The maximum value of this parameter is obtained for – 20 μm fraction of Long-flame coal, its ash content is 30% at 6 % yield, velocity being V = 20 m/s. The increase in this parameter leads to reduction in ash content in the product, may be, due to aggregation of organic and mineral particles. Thus, low-speed impact grinding of long-flame coal with follow-on removal of fine fraction is considered as an efficient method to lower mineralization of coal and to improve coal concentrate grade.

In the activation grinding of coal the organic components undergo deep transformations because the rupture of chemical bonds and destruction of macromolecules cause sharp increase in chemical activity of coal along with alteration of most properties of its organic components [9, 10]. When considering coal as a fuel, the effect of mechanical activation on coal activation energy *E* was studied as a function of thermal-oxidative process. The thermal analysis was performed with the use of TG 209 FI thermoscales, Netzsch, Germany, and the software of the same company [11, 12].

The experimental data analysis shows great opportunities of the activation grinding as a promising instrument to transform organic components and to manage the activation energy. It is demonstrated on long-flame coal that mechanical treatment appreciably lowers the activation energy (Fig. 4).

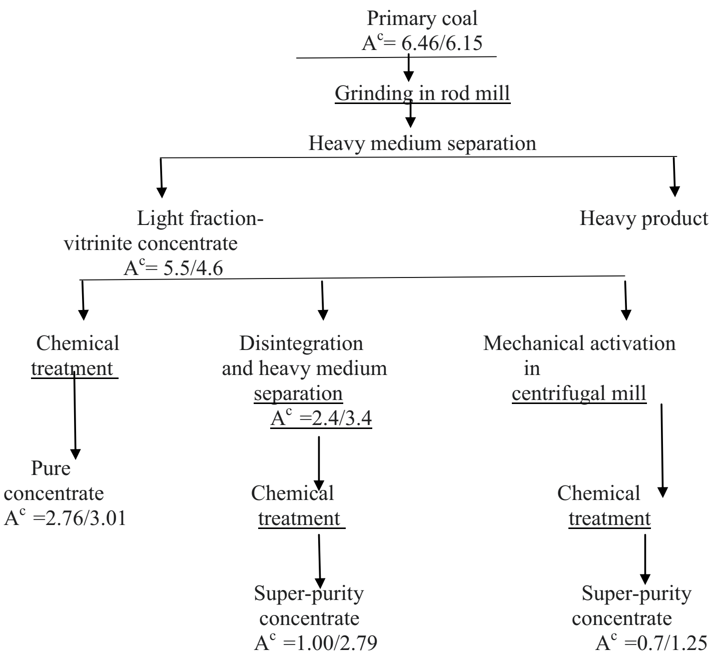


**Fig. 1**. Deep coal concentration flowsheet
$A^c$ – ash content in coal, %; Ash content in long-flame coal in numerator; Ash content in lean coal in denomiantor T

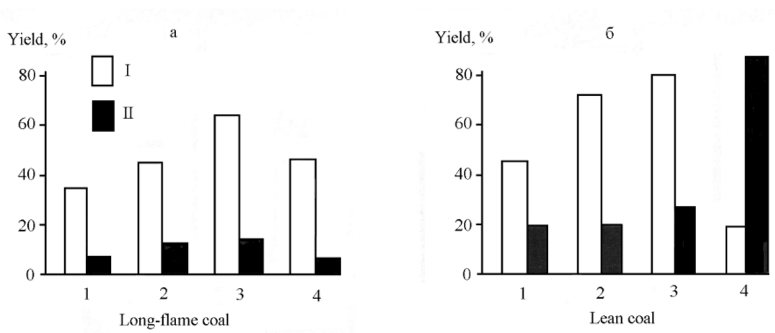


**Fig. 2.** (a), (b) – Comminutability of different coals: 1 – grinding in Culatti microgrinder at V = 20 m/s; 2 – the same at 40 m/s, 3 – grinding in the disintegrator at V = 40 m/s, 4 – the same at V = 100 m/s. I – -0.25 +0.02 mm fraction, II - –0.02 mm fraction.

At the organic matter transformation degree α = 0.2 the disintegration treatment lowers *E* magnitude from 120 kJ/mol down to 80 kJ/mol. Thereto it is important to note that *E* values of activated coal and its light fraction are close and amount to d < 1.4 g/cm$^3$, in other words it indicates the significance of fuel beneficiation before its utilization in power production.

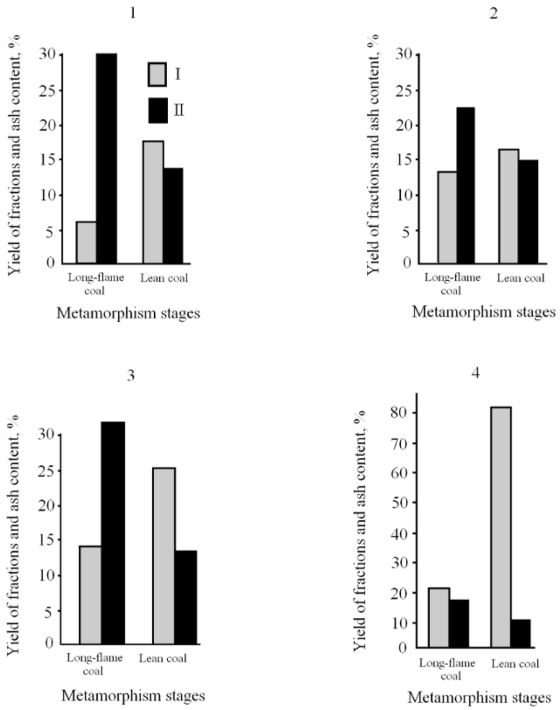


Fig. 3. Yield and ash-capacity (I, II) of -20 μm fractions depending on type and energy-intensity of grinding facilities: 1, 2 – Culatti at V = 20 and 40 m/s; 3, 4 – disintegrator at V = 40 and 100 m/s.

The activation of lean coal gives less reduction in *E* (Fig. 5), perhaps, due to increased share of aromatic carbon form, higher density and strength of organic-mineral associations. The high energy-intensive grinding should be applied to lower *E* of high metamorphized coal. The transformation of coal into a high-dispersed state is directly related to its combustion in a fine coal flame. In view that the pulverized-coal fired boiler is operated with the use of gas- and fuel-oil flame, the utilization of mechanically activated coal with enhanced reactive capacity to this effect can be considered as an innovative process.

Mechanical activation increases the ignition and combustion velocity, but kinetic characteristics of these processes are mainly stipulated by a kind and mode of the destructive effect. The free-blow disintegration results in more essential weakening of intra- and intermolecular bonds of coal and reduces the energy barrier for oxidizing reactions as compared to grinding at a centrifugal-planetary mill. Destruction of organic components of coal in mechanical activation at a centrifugal-planetary mill at the mode of restricted effects is also accompanied with aggregation along with compaction of the matter, deionization of radicals and other active centers, thus aggravating the kinetic parameters of coal combustion. To conclude, it is reasonable to state that the most effective activation process is a free blow,

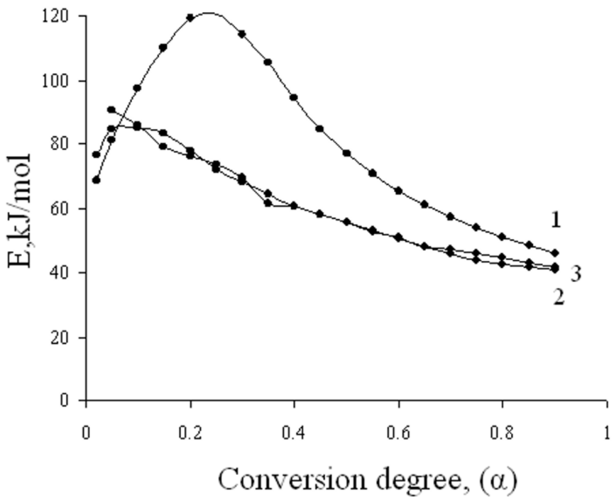


Fig. 4. Variation in activation energy in thermal-oxidative degradation of long-flame coal: 1- initial specimen; 2 – specimen treated in the disintegrator; 3 – light fraction (d < 1.4 g/cm$^3$).

realized in a disintegrator, prone to reduce activation energy in different reactions, including the thermal-oxidative destruction of coal substance in our case. The destruction of this kind intensifies combustion velocity by approximately 20%.

It is demonstrated below that treatment of coal at M-3 mill results in destruction of mineral components and their better solubility in aggressive media in chemical treatment stages.

It is essential that activated coal can be produced at available high energy-intensive disintegrating facilities of 0.1 – 1.0 t/h capacity [12].

**Chemical demineralization of coal**

The chemical demineralization was investigated on the same long-flame and lean coals (Fig. 1). The gravity medium separation of coals, ground in a rod mill, permits to bring down ash content in the concentrates by 0.95% and 1.55 %. The similar parameters after the chemical treatment are 2.76 and 3.01 %, respectively. Thereto, it is apparent that syngenetic clayey substances are not recoverable.

Disintegrating grinding of a light product enabled to gain deeper disintegration in both secondary gravity medium separation and in chemical treatment in solution $Na_2SiF_6$ and to produce concentrates with $A^c$ = 1.00 % (long-flame coal) and $A^c$ = 2.79 % (lean coal).

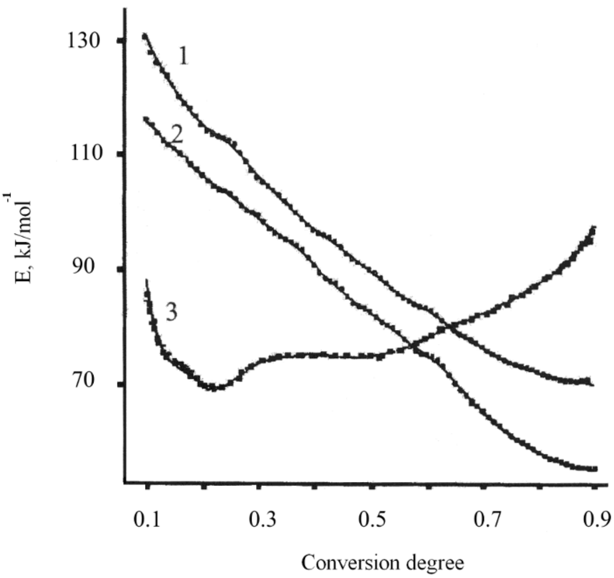


Fig. 5. Variation in activation energy in thermal-oxidative destruction of hard-ash coal: 1- primary specimen; 2 – specimen treated in a disintegrator; 3 – specimen activated in centrifugal mill.

The lower-ash concentrates were obtained in chemical treatment of coals with activation of its light fraction at a centrifugal-planetary mill. The ash content in the final concentrates was 0.70 % and 1.25 %.

As for higher metamorphized coals, it is to a less extent possible to recover mineral components, though by X-ray fluorescent analysis data quarts and kaolinite become X-ray-amorphous by 40 – 50 % in the stage of activation [11].

Therefore, coals of lower carbonization stages are preferable for production of low-ash and extra-purity coal concentrates.

## Conclusions

1. In heat power industry and chemical demineralization the grinding capacity of coals down to beneficiation size -0.25 + 0.02 mm and superfine size strongly depends on a metamorphism stage and a type of fuel destruction

2. From the viewpoint of beneficiation the disintegrating activation gives an advantageous opportunity to gain mineral components concentrated in -20 μm fine product, as the removal of these components improves the grade of coal concentrate. This process runs most efficiently under mechanical treatment of long-flame coal.

3. Coals of higher metamorphism stages are most preferable in production of pulverized fuel at a disintegrator.

4. The investigations into thermal-oxidative destruction revealed that the mechanical activation lowers activation energy thus providing an opportunity to adjust kinetic parameters of ignition and combustion of fuel.5. In production of pure and super-purity coal products the most efficient performance of chemical demineralization is accomplished with activation grinding of coal at a centrifugal-planetary mill operating

at the shear-crushing mode, thus the produced concentrates were: one with $A^c$ = 0.70 % for long-flame coal and the other with $A^c$ = 1.25 % for lean coal.

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# Prospects of electric pulse technology for production coal-water fuel (CWF) and extracted from coal deposits of rare elements

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**Abstract**

Growing fuel consumption of ferrous, nonferrous and rare metals, non-metallic ore materials that are the basis for the development of the economy and modern industry requires an increase in production and processing of minerals. In recent years, be activated a new trend in the coal energy. There are technological solutions for the gasification of coal, produced water-coal fuel (WCF) and special equipment for its combustion. All these new technologies require a higher quality of coal fuel and sufficiently pulverizing coal - is not greater than 50 microns. Requirements to increase the number of processed ores in improving quality indicators processing (increasing the degree of extraction) put very important problem to streamline processes and reduce the cost of crushing and grinding. A fundamental solution to the problems of complex use of raw coal crushing and grinding it with minimal content ash, narrow particle size distribution, as well as enhance the complete extraction of the minerals it can be achieved on the basis of the process electric pulse destruction materials (EPD).

**Keywords:** electric pulse technology; electric pulse disintegration; selective disintegration ores; electric pulse grinding; fragmentation; extracted rare elements; production coal-water fuel; enrichment ores

## Introduction

Russian scientists have proposed and investigated a method of electric pulse method destruction materials (EPD) [1-3]. At present time method to receive recognition and development abroad [4-6]. EPD is characterized by high energy efficiency and unique technological features that allow its universal use for drilling holes with different diameters and destination, crushing and milling ores and industrial materials, cutting and surface treatment array and block stone. The physical basis of the EPD-method is that creates conditions for introduction of the discharge in the surface layer of the solid body, which acts on the rock is analogous to the explosive placed inside. EPD- method allows to destroy all the rocks and materials, except ore with a solid conductive inclusion [7]. One of the promising areas of technological application of the EPD-method in the coal industry is a special grinding materials, which guarantees a minimum of hardware metal brings in product of grinding. In combination with a high selectivity of destruction, the EPD- method provides a high purity extraction of deposits from the ore. It is possible to adjust the particle size distribution of the product, including the product to obtain a narrow size fraction, which is an important factor in the production of CWF. EPD-method promotes enrichment a comprehensive ores. Increase of recovery in the concentrate depending on the kind ore can reach several percent, and even tens of percent. In addition, EPD-technology provides additional opportunities to address environmental problems by increasing the complexity of the use of mineral raw materials and waste products.

 
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**Selective disintegration ores for extracted rare elements**

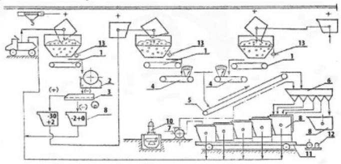
EPD in the cycle ore preparation provides better disclosure of mineral grains with high safety of their natural shape and less grinding material. In the next step of ore concentration possible to increase extraction of minerals from a few percent (for disseminated ore) to 1.5-6 times (output conditioned raw materials) with the best group composition. When the destruction of mono crystalline formations provided the maximum yield of mono crystal defect-free. Technological efficiency of the method is confirmed by the grinding of non-ferrous and rare metals, mica and asbestos ore in the allocation of various cutting and polishing crystal raw material from productive ores.

**Special electric pulse grinding materials how prospect for production CWF**

EPD ensures minimal hardware brings the metal in the product regardless of its grinding abrasiveness. In combination with a high selectivity fracture provides a high purity monomineral differences. It is possible to adjust the particle size distribution of the product, including the product to obtain a narrow size grade. Currently, disclosure of mineral grains of coal are widely used mechanical means of crushing and grinding. Typically, coal concentrate remain rocks intergrowths with coal, which impairs its quality. Middling product of approximately 25-27% of the total weight of the feedstock is used as the low-grade energy fuel. Application of EPD for recrushing during deposition middling product able to increase the yield of the concentrate and improve its quality. According to [8] in the enrichment process dropping out Karaganda coal at electric pulse splitting middlings ash content decreased by 0.8%, concentrate output increased by 3.7%.

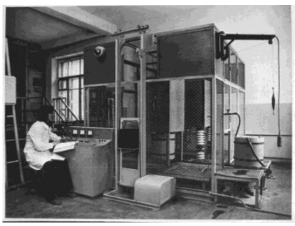
**Constraints industrial development of electric pulse technology and it improvement**

To date the development of manufacturing technology, despite the wide range of applications and technological efficiency, constrained by insufficient a number of technical issues stipulating operational and technical-economic characteristics of the equipment and technology in general. Previously used electrical equipment has low energy and weight and size characteristics (Fig.1).



**Fug.1** The scheme of continuous apparatus testing

Projects plants for preparation processes at the concentrators for ordinary ores were disproportionately large in size compared to conventional means.



**Fug.2** Crushing and grinding system for selective disintegration of geological samples CGS (30sqm,5m)

To solve the problem of industrial development electric pulse destruction method proposed development of technology by improving the electric pulse equipment, due to the transition to the new (modern) element base with improved mass-dimensional and power characteristics. To date, there are technical prerequisites for the creation of compact and energy-efficient technical means for generating high-voltage pulses of electro-technology material destruction. This opens the way for practical implementation of high energy efficiency and unique technological advantages of the method electric pulse destruction geomaterials a wide range of technologies for the extraction and processing of mineral raw materials. Recent advances in high-technology and electronics (Fig.3) to allow the procedure and further improve all the characteristics of electrical equipment, which can be used to implement EP technologies, including for crushing raw coal with subsequent extraction of a rare element production and CWF.

**Fig 3.** Appearance modern charging device according to scheme high-frequency voltage transformation (HFVT) Spellman firm.

Based on the experience of using pulse transformers in a pulsed technique [9-11] in Centre for Physical and Technological Problems of Energy in Northern Areas of Kola Science Centre of the Russian Academy of Sciences (CPTPENA KSC RAS) have been suggested possible schematics for electric pulse transform technology (Fig.4).

Due to improvement of the generating equipment dimensions of the electrical installation become by an order less than in similar installations and stands electric pulse destruction (CGS and others) and comparable to the size of technological devices.

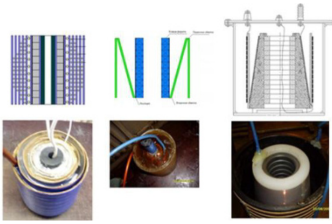


**Fig.4** Construction pulse transformers to various embodiments of coil geometry

Developed in CPTPENA KSC RAS model pulse transformers (Fig.4) experimentally confirmed receipt of the voltage pulses with the required parameters for efficient breakdown of rocks, including when an electrically conductive working environment (industrial water). Also, this type pulse transformers expand opportunities to maximize the efficiency of energy transfer in the discharge channel in the rock to the desired mode of energy to improve the energy efficiency of destruction [12-16]. Using the proposed solutions can improve the compactness and energy efficiency electric pulse technological installations, as well as to create conditions for improving the technical unit power installations by increasing the number of parallel units.

**Conclusion**

The prospect of industrial development electric pulse processes associated with the ability to solve the main problem - the need to radically improve the specific energy and mass-dimensional characteristics of the electrical unit and electro-technological installations in general. Develops and offers in this article of new methods to the development of providing electrical electric pulse processes can give a powerful impetus to improve electric pulse technologies, including the production of coal-water fuel, as well as to increase the extraction of rare elements from coal.

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# Toothed screw crushers in the processes of coal beneficiation

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**Abstract**

Scales of mining operations are continuously increasing, their quality, however, is becoming worse due to transition from rich ore mining to non-commercial ore mining, to extraction of raw materials, having high moisture, with the high content of viscoplastic clay materials etc..Change of raw material composition affects negatively the set enrichment process, in particular, the process of source material crushing.

Recently the roller toothed screw crushers, their application range and objective performance are of great interest during modernization of existing beneficiation plants.

Based on long-term experience of designing; production; starting-up and adjustment; and such equipment operation, the paper studies the peculiarities of the design of this type crushers, their practical possibilities, examples of usage at the existing beneficiation plants, and also some individual points, taking into account of which is necessary during designing of crushing processes, using toothed screw-type crushers.

**Key words:** coal, a toothed screw crusher, a roller crusher, ore dressing, crushing, crushing size, crushing scheme

The analysis of the world experience dealing with crushing shows presently a wide spread of toothed roller crushers of various design for crushing of wide variety of materials. The most common explanation is that the toothed roller crushers are the most adaptable to conversion of materials prone to sticking or containing clay inclusions. Application of another types of crushers for such materials conversion is either impossible at all or causes frequent downtime, connected with continuous and time-consuming work on cleaning of crushing chamber or crushing elements.

The toothed screw crushers (Figure 1) can be used for crushing of material having hardness ratio up to 16 units by M.M. Protodjakonov [1, 2], can be also used for soft materials (coefficient of hardness is less than 5), in particular, for coal, marlstone, limestone.

The advantages of the crushers of such types are:

-the possibility to obtain the greater degree of reduction of piece being crushed : from 2:1 to 10:1 per one reduction stage,

-crusher small dimensions in comparison with the other crusher types,

-low level of product overgrinding,

-the possibility to convert the sticking and plastic materials,

- the possibility to convert the adfreezed materials,

-the absence of necessity of check screening after crushing,

-the possibility to construct the crushers for arctic conditions at ambient temperature up to -50 $^{o}$C,

- the possibility to construct the crushers for solution of a wide class of problems depending on performance, initial material size, material solidity, output material size, clay content, temperature, etc..

Crushing using of the toothed screw crushers (TSC) can be carried out by two basically different schemes [3]:

 
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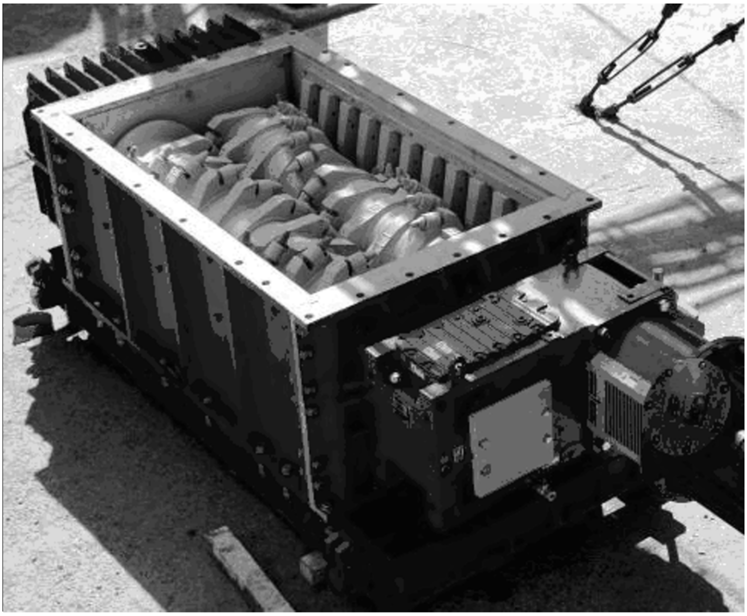

Figure 1 —A toothed screw crusher (TSC)

**-** Central crushing. Material crushing takes place as a result of pressure of the material being crushed between the crushing teeth, mounted on two parallel contra-rotating rollers, Figure 2. Maximum size of a piece, fed to crushing, is defined by inter-axle distance, output piece size is defined by radial and axial pitch of installation of crushing teeth. Fine fractions pass through gaps between crushing teeth and leave the crusher being nondestructive.

**-** Peripheral crushing. Material crushing takes place between screw teeth, rotating in the direction of crusher walls at small peripheral speed and teeth installed on the housing side wall. This crushing scheme is used at a fine crushing stage and allows increase twice the crushing capacity with unchanged dimensions of the working area.

One of the basic processing characteristics of the crushers is particle size distribution of material being crushed or absence of initial material overgrinding [4]. Table 1 below shows data as a percentage on particle size distribution of various materials after crushing using the toothed screw crushers (TSC) in comparison with initial grain-size category more the set limit of crushing D.

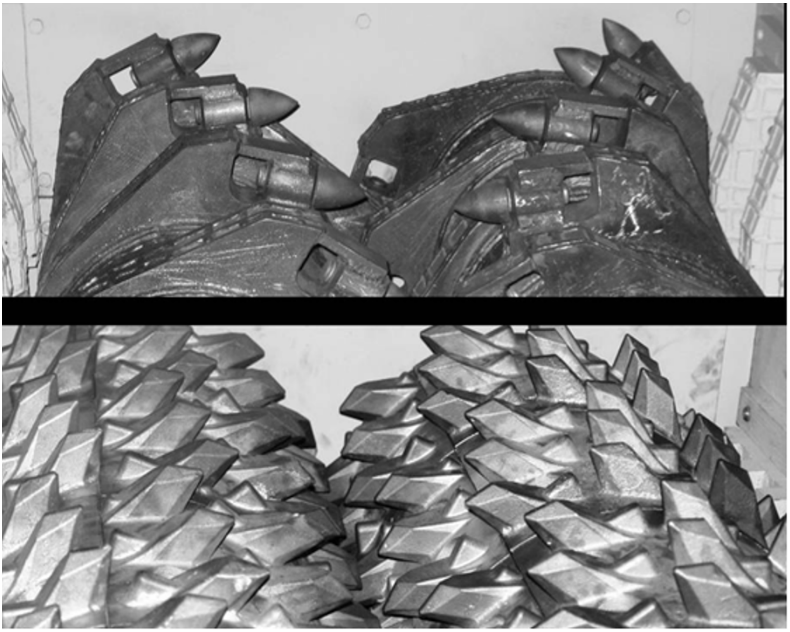

Figure 2 —Toothed rollers of toothed screw type crushers (TSC)

Table 1 - Particle size distribution of crushed materials

| Processed product | D, mm | output % from initial grain-size category +D | | | | |
|---|---|---|---|---|---|---|
| | | +D | D/2-D | D/4-D/2 | D/4-D/8 | 0- D/8 |
| Fluorite ore | 250 | 0 | 72 | 14 | 9 | 5 |
| Fluorite ore | 50 | 0 | 76 | 15 | 6 | 3 |
| Barytic ore | 50 | 0 | 68 | 17 | 11 | 4 |
| DG-grade coal | 150 | 0 | 69 | 19 | 10 | 2 |
| T-grade coal | 150 | 0 | 52 | 26 | 13 | 9 |
| Slate coal | 100 | 0 | 66 | 18 | 10 | 6 |

Figure 3 demonstrates derived and integral curves for crushed material size, obtained during slate coal crushing with getting grain-size category 0-50 mm using a toothed screw crusher (TSC)-500-50-75 and jaw crusher (JC) -0,6x0,6 [5].

As can be seen from diagrams, when using the toothed screw crushers in ore-preparation cycles, a considerable reduction of categories below the specified size takes place. Thus, upon replacement a hammer crusher for a toothed screw crusher (TSC) DSHZ-500 at works «Tamme Auto», Estonia, specific output of grain-size category 25-50mm during slate coal crushing was increased 2,2 times. 20% reduction of electric power consumed by the crusher took place due to decrease in overgrinding of initial material. Crusher use factor was increased from 0,3 to 0,86 due to avoidance of material sticking on working surfaces. Actually the crusher shutdowns for two years of operation have been occurred just to carry out the scheduled-preventive maintenance, oil change in reducers and working elements of roller teeth.

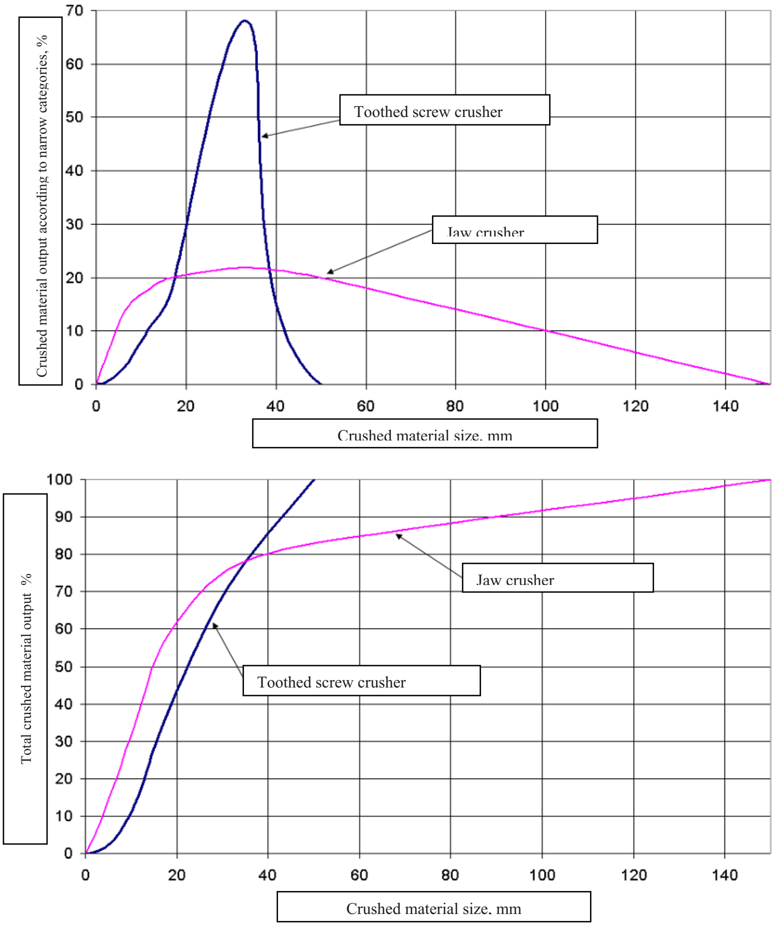


Figure 3—Crushed material size curves

One of the important operating characteristics of crushers is coefficient of wear of crushing elements and their lifetime. In the general case the wear and tear is proportional to crushing capacity and depends on solidity of crushable material, its abrasive ability and etc.. Drive power and capacity is defined in each particular case depending on type and size of feed material and desired size of final fractions. Easiness in service is reached due to quick replace of screw individual segments. During crushing of material having hardness ratio up to 8 units by M.M. Protodjakonov, the standardized rock cutters are used, during crushing of hard rocks and also at stages of fine grinding, cast teeth of steel 110Г13Л are used, Figure 2.

Nowadays more than thirty toothed screw type crushers (TSC), crushing various material, are operating for several years in Russia and abroad, it has allowed to obtain the definite statistical data on wear and tear. Mean lifetime of rock cutters is from two to six months, mean lifetime of cast teeth is from five months to four years. Over a period of five-year operation no wear of internal surfaces of crusher housing is noticed. Replaceable inserts are industrially manufactured in the RF, that is of no small importance, because it eliminates the dependence on consumables in the operation process [6].

In most cases the crushing is the main and often the most power consuming operation, intended for stuff breaking to the required dimensions, and also for breaking of mutually coalescent aggregations and formation particles of separate minerals. Use of the toothed screw crushers makes it possible to solve the problems of wet and clay-containing stuff processing.

The main advantage of the crushers of the given type is the exclusive productivity while employing a compact design, having low specific quantity of metal; crushing surfaces are able to self-clarification, that allows to crush steadily materials having high moisture and viscoplastic sludge, having also an automatic response system on entering non-crushable materials into the device [7, 8].

One of the advantage of the crushers of the given design is the opportunity to construct such machine configuration, which can ensure the optimal parameters of a crusher depending on the current task: capacity requirement, initial and crushed material size, rock solidity and conditions of service [9,10].

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# Application of flip flow screen in Sihe Coal Preparation Plant

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**Abstract:** Compared with the traditional vibration sieve,great advantages have been showed in flip flow screen in coal preparation industry,including the enhancement of vibration strength,the function of automatic cleaning,hard blockage of the mesh,etc.In this paper,the structure and working principle of flip flow screen are elaborated,and the development process of flip flow screen in Chinese coal preparation industry are introduced.Flip flow screen,which is in the forefront of the domestic counterparts,has been comprehensive application in Sihe Coal Preparation Plant.The application practice and effect of flip flow screen on raw coal screening, desliming before lump and slack coal separation, dehydration and sculping of slack coal are comprehensively introduced.The undersize fraction and the oversize fraction are simultaneously decreased because of the application of flip flow screen on raw coal screening.Due to application of flip flow screen on the desliming before lump and slack coal separation,the amount of coal slime in lump coal system is reduced by 75%,and the slack coal recovery was increased from 60%~65% to 70%~75% meanwhile the reagent dosage is reduced nearly 70%.The application has brought huge economic benefits and reduced the labor intensity.

**Key words:** flip flow screen; working principle; application; screening efficiency; transformation results; desliming;undersize fraction;coal preparation.

## Introduction

Due to the level of coal mining mechanization increasing, the content of fine coal increases in coal mining with higher moisture content. For the ordinary vibrating screen which is limited by its structure, performance and design characteristics, the screening efficiency of fine coal is generally not high, especially in screening of viscous blocking fine coal [1].In this context, flip flow screen as a new screening equipment in recent years is introduced and applied in coal preparation plant in China. Sihe Coal Preparation Plant (SHCPP) of Jincheng Coal Industry Group in Shanxi has the most comprehensive and thorough use of flip flow screen by far, the application of flip flow screen is in the forefront of the domestic counterparts.

SHCPP began the transformation for the process of flip flow screen in 2012, including raw coal screening; lump and slack coal dry desliming before the separation, and medium draining of slack coal. According to the practical application, the difficult problem of sieve pore blocking was effectively solved, the screening efficiency was improved, the process was simplified, and the economic efficiency of the enterprise was improved [2].

## 1 Origin and development of flip flow screen [3]

Flip flow screen technology originated from Germany in 1950s. In 1951, the German A. Wehner applied the patent of DUO (double mass) vibrating feeder which is the embryonic form of flip flow screen with the design of inner and outer screen frame, . The subsequent development of Umbra vibrating screen reached the level of utility under the guidance of him, this screen is a real sense of the first flip flow screen. But Umbra sieve ended in failure in practical application because of the high cost and low yield of

 
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sieve plate.

With the development of metal and plastic material plate technology, Umbraex impact screen (Impact Screen) developed from bidirectional Umbra screen, forms semi-circumference motion using the impact device and drives the second screening system. The vibration strength of this screen reaches 2.5~4.5g while the particle acceleration can reach 40g. Although this screen has certain advantages in screening of fine materials in particular in screening of difficult to screen fine materials, it failed in the end due to the high loss of drive device.

With the development of high elastic rubber and polyurethane sieve plate, flip flow screen technology had seen great progress, which makes the material not only has a sliding movement along the screen surface, but also makes the material spring back on the screen through the ductility and elastic of the screen surface. This definitely facilitates the material spreading out and screening. The industrialization level of flip flow screen had been greatly improved; flip flow screen development had entered the period of third dynasties.

## 2 Application of flip flow screen in China coal preparation

The application of flip flow screen in China coal preparation industry is relatively late. At first flip flow screens from Austria's Binder, Germany's Liwell and other brands were utilized. With the localization of the flip flow screen, products of companies including EuroCMA, Birtley, Aury and other independent brands are widely adopted in coal preparation plants. At the beginning the flip flow screen has higher requirements on the coal type and particle size. Fortunately, with enormous application in coal preparation plant and continuous improvement of technology,the grading size is further extended, flip flow screen can be used in screening up to 150mm and down to 3mm; the applicable coal type is more and more broad, including lignite, non-caking coal, gas-fat coal, coking coal and anthracite; the application region spreads from the areas of coal enterprises concentrated, such as Shanxi, Inner Mongolia to the areas of Ningxia, Xinjiang, Henan, Beijing and other regions.

According to the differences of coal type and process, the application of flip flow screen is also diversified, including single, double flip flow screen, three layers of flip flow screen etc. The multilayer structure of flip flow screen also includes two forms: all flip flow screen plates, a layer or multilayer of flip flow screen plates and a fixed screen plate.

The number of applications of flip flow screen developed rapidly, since the technology was introduced in China coal preparation industry in 2011. After three years development until the end of 2014, there was about 200 screens in the practical application stage according to incomplete statistics, its application in the different process of washing system is more comprehensive, including raw coal screening, further screening (6mm/3mm), dry desliming, dehydration and medium draining, which provides a lot of experiences for the further promotion.

## 3 Working principle and characteristics of flip flow screen

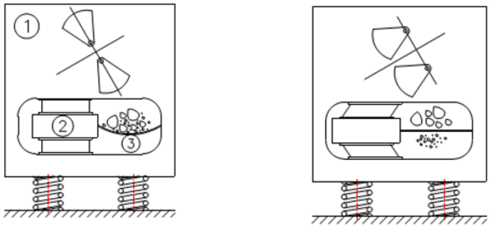
The working principle of flip flow screen is shown in figure 1. The fixed frame of Flip flow screen are in reciprocating motion driven by the exciting force, resulting in the relative motion between the movable frame and the fixed frame due to the dual vibration principle[4]. One of the screen plate ends is fixed on the fixed frame beam, the other end is fixed on the movable box beam, any two adjacent beams have a relative motion, elastic screen surface is tension and slack periodicity, namely screen surface not only has a translational motion along with the screen box, but also a relative motion relative to the screen box at the same time. When the screen plate extends, the material can be transferred up to 50*g* acceleration by the screen surface (traditional vibration acceleration is only about 5g[5], which makes difficult to screen material is easy to be penetrated, reaching a high screening efficiency [6]. The tension and relaxation motion not only makes the screen pores deformation constantly, but also increases the vibration intensity of sieve surface, so as to effectively overcome the clogging phenomenon, significantly improve the processing ability of the machine[7].

The advantages of flip flow screen compared with the traditional vibration sieve are as followings [8]:

1)The flexible polyurethane sieve plate with elastic deformation increases the vibration intensity and has the function of automatic cleaning;

2) Less screen rotation parts, less maintenance, reliable operation;

3) The vibration intensity of movable plate can be adjusted, the mesh is not easy to be blocked, and the screen is suitable for dry screening for the difficult to screen material.




1-fixed frame；2-movable frame；3-screen plate

Figure 1 the principle diagram of the flip flow screen

## 4 Comprehensive application of flip flow screen in SHCPP

### 4.1 An overview of SHCPP

SHCPP is a large modern mine type coal preparation plant, processing coal is anthracite, with a processing capacity of 16 million tons per year. Its production process is: raw coal is screened by 13mm, +13mm coal is separated by the heavy medium shallow slot sorting machine after desliming, -13mm coal is separated (13mm-1.5mm) by heavy medium cyclone after desliming, coarse slime (1.5mm-0.1mm) is separated by spiral separator, -0.1mm coal slime is recovered by the plate and frame filter press. The major products are washing middling block, washing small block, washing dust coal, standard dust coal, SHCPP was awarded the honorary title of "national top ten coal preparation plant" in 2005.

### 4.2 **Application of flip flow screen in screening of raw coal[9]**

The raw coal in Sihe coal mine, with high water content, is easy to argillization. With the rapid increase of coal mining mechanization degree, the water for dust suppression increases, the external moisture of raw coal is 7%~11%. When screening by 13mm, there are common problems that screen surface is easy clogging and the efficiency is not high. The poor screening efficiency results in a large number of -13mm materials mislead into the shallow slot separation machine, causing a series of problems such as lump coal products exceeding the undersize fraction, the effect of dewatering and medium draining for lump coal is poor, slime content increases. Medium consumption, drug consumption and other cost increase. In order to ensure the quality of commercial coal, lower the undersize fraction of lump coal, SHCPP has to extend the sieve pore of 13mm into 15mm.

Although the low sieving efficiency of raw coal is alleviated to some extent, the problem is not solved fundamentally, and that would lead to the increase of the oversize fraction of -13mm dust coal, as well as part of the small lump products mislead into the undersize product, which causes great economic losses. In order to solve the problem, SHCPP took on the transformation of the former seven screening equipment in 2012 October, the original banana vibrating screen was replaced by the linear motion of the banana shaped flip flop screen with the application of EuroCMA technology .

In order to verify the screening effect of flip flow screen, we sample for the oversize product and screen underflow of raw coal classifying screen with the different feeding quantity of 300-500t/h, the oversize fraction and undersize fraction are calculated with 13mm as the benchmark, as the following table:

| Sampling times | Handling capacity （t/h） | Lump sample weight （kg） | Undersize weight （kg） | Undersize fraction （%） | Mean value | Slack coal weight （kg） | Oversize weight （kg） | Oversize fraction （%） | Mean value |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 300 | 41.8 | 1.9 | 4.5 | 3.3 | 25.3 | 1.1 | 4.3 | 4.6 |
| 2 | | 25.85 | 1.1 | 4.3 | | 23 | 0.8 | 3.5 | |
| 3 | | 23.5 | 0.9 | 3.8 | | 23.2 | 1.4 | 6 | |
| 1 | 400 | 23.6 | 0.8 | 3.4 | 3.5 | 26.7 | 1 | 3.7 | 4.6 |
| 2 | | 24.7 | 0.9 | 3.6 | | 23.7 | 1.2 | 5.1 | |
| 3 | | 23.7 | 0.7 | 3 | | 21.4 | 1.1 | 5.1 | |
| 1 | 500 | 21.5 | 0.9 | 4.2 | 4.2 | 23.4 | 1.4 | 6 | 6.3 |
| 2 | | 24.5 | 0.6 | 2.5 | | 24.7 | 1.6 | 6.5 | |
| 3 | | 22.1 | 0.7 | 3.2 | | 24.6 | 1.55 | 6.3 | |
| Overall mean | | Undersize fraction：3.7% | | | | Oversize fraction：5.2% | | | |

Compared with the dates before the transformation the undersize fraction descends from the original 10.9% to 3.7% now; similarly the oversize fraction decreases from 12% before the transformation to 5.2% according to the tests. The screening effect has been significantly improved after the modification that the undersize fraction of lump coal is reduced; the amount of slime coal in the lump coal system is also significantly reduced, while the effect of medium draining for lump coal is enhanced. Consequently, the medium loss of lump coal is obviously reduced. Moreover, the decline of the oversize fraction of slack coal leads large quantity of slack coal being separated, which produces enormous economic benefits.

With annual processing capacity of 14.4 million tons, the application of flip flow screen leads to 620,000 tons more output of small lump coal compared with the traditional classifying screen. The price difference between slack coal and small lump coal is 400 Yuan, the additional profit per year is 248 million Yuan. At the same time the self-cleaning function of flip flop screen with less problem of screen blocking greatly reduces the workload and working frequency of workers. In sum, flip flow screen is reliable operation, with little daily maintenance and with low requirement of workers labor intensity.

### 4.3 Application of flip flow screen in the process of lump/slack coal desliming prior to separation

Originally the production process was lump/slack coal wet desliming prior to separation in SHCPP, although the process has the advantages of high screening efficiency and less amount of slime entering the medium system, the wet process leads to a large quantity of coal slime, the coal slurry system load increases, and the adverse consequences of cost increase of drug consumption, electricity consumption, water consumption and operation cost emerged.

In the former process, the undersize product of lump coal desliming screen, which was further dehydrated by dewatering sieve, was mixed with slack coal products which increased the moisture content of slack coal products. When the moisture content of raw coal was high, it was easy to cause the moisture content of slack coal product exceeding the requirement. In order to reduce the amount of slime and the moisture content of products and improve product recovery. Taking into account that the flip flow screen was not easy to be blocked and the characteristic of higher dry sieving efficiency, SHCPP breakthrough transformed the process of lump coal wet desliming technology into the process of 13mm dry desliming by flip flow screen and transformed the process of slack coal wet desliming into the process of 3mm dry desliming by flip flow screen, The application feasibility of dry desliming by flip flow screen was tested in this paper.

#### 4.3.1 Application of lump coal desliming process prior to separation

In terms of lump coal dry desliming, the screening effect was contrasted before and after the transformation. Although the screening efficiency of dry desliming process by flip flow screen increased only 0.59% compared to the former wet desliming process before the transformation, the amount of coal slime in lump coal system was reduced by 75%, which greatly reduced the production costs of subsequent processing, and effectively reduced the moisture content of slack coal product.

However, the application of flip flow screen in dry desliming process brought the adverse consequences of large dust on screening site which means certain bad influence to the workers' physical and mental health. Aiming at the defect, SHCPP installed the dust removal system and solved the problem. As of now, the system had been well operated for about a year after the transformation, which showed that the process of lump coal dry desliming by flip flow screen was successful.

#### 4.3.2 Application of slack coal dry desliming process prior to separation

In terms of slack coal dry desliming, the processing ability of a single flip flow screen was 300t/h after the transformation, the screening efficiency was up to 63.5%; the washing coal slime decreased by nearly 60%, which greatly reduced the processing pressure of coal slurry system. The reduced coal slime was translated into undersize product with higher prices, which increased economic income; the slack coal recovery was increased from 60%~65% to 70%~75%; the usage amount of polyacrylamide was reduced nearly 70%, the consumption of polymerization aluminum chloride was reduced nearly 75%. In other words, the cost was significantly saved [11].

For the problems in the transformation process,pressure medium pipeline position inside the barrel was risen, the pressure of pipeline medium was increased, so that the material and the medium were mixed completely, and the problem was solved. As of now, the system had been good running for about two years after the transformation; it indicates that the application of 3mm dry desliming by flip flow screen was successful.

## 5 Exploration of Application of the process of slack coal dehydration and medium draining by flip flow screen

The traditional 1.4mm dewatering screen and medium draining screen have the problems that sieve pore is likely to be blocked, medium running is extremely easy to happen, and the dewatering effect is poor. Furthermore the mesh is difficult to clean up once blocked, the separating result can be ensured only by replacing the sieve plate, thereby causing a great waste of materials. Considering the stretching characteristics of flip flow screen, great self-cleaning ability, and mesh is not easy to be blocked, SHCPP replaced the ordinary vibration sieve with the monolayer flip flow screen in the process of slack coal medium draining , the application feasibility of the process of dehydration and medium draining is tested.

The medium consumption of slack coal descends from the original 1.98kg/t to 1.57kg/t one month later after the transformation. The reduced medium consumption indicated that the application of flip flow screen in medium draining process is feasible. Because the most vibration frequency of flip flow screen is 800r/min lower than the original vibration sieve 900r/min, the material in the sieve surface of flip flow screen stay longer, resulting in a thick layer of material under the same conditions; and the elastic characteristics of flip flow screen results in more water and medium on sieve surface. The effect of dehydration and medium draining has not reached the desired results; there is still much room for improvement. SHCPP is planning to modify the fixed sieve structure on collecting box in the next step in order to increase the medium recovery from fixed screen, thereby reducing the amount of medium draining by flip flow screen. The effects of dewatering and medium draining may be further improved and the medium consumption will be reduced again.

## Conclusion

The successful transformation and application of flip flow screen in SHCPP fully embody advantages in process, technology and economic of flip flow screen over the traditional banana screen, providing valuable experience for other coal preparation plant of flip flow screen transformation. It might be a promotion for the widespread use of flip flow screen in the washing industry. Under the background that the current coal price continues to decline and the downward pressure in the coal industry continues to increase, the extensive application of the flip flow screen plays a very positive role on simplifying the process, improving the coal recovery and increasing the economic efficiency for enterprises and stimulating the vitality of the coal industry.

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# Research on difficult-to-separate coal grinding-dissociation and flotation process

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**Abstract:**Coal with high content (with higher content of undissociated mineral) of middles is difficult-to-separate, which is one of the key factors to influence coal flotation result. This paper analysis properties of two kinds of different feeds of coal slime and the coarse particles, and determine the dissociation degrees of slime through the the density composition. At the meantime, flotation tests have been carried out with better dissociation degree coal slime. The experiment results show that coal slime directly grinding time with 15min, -0.074mm fineness taking 86.85% is appropriate to float, and graded coarse coal grinding time with 10min, -0.074mm fineness taking 46.19% is appropriate to float. Flotation effects are obvious, and clean coal recovery rate increased significantly. The feasibility of coal grinding to dissociation was verified through the grindability index. The grindability index of coal slime is 74.09%, and that of coarse particles is 76.01%. These will inevitably cause difference of energy consumption in grinding process.

**Keywords:** coal slime, classification, sizing, fine grinding, dissociate, flotation, density, grindingability

## Introduction

Crushing and grinding are mainly the process of comminution which the main objective is the liberation of the valuable minerals from the gangue, and minerals have an economic optimum particle size, which will depend on many factors, including the extent to which the values are dispersed in the gangue, and the subsequent separation process to be used[1]. Coal separation system has the same problems of intergrowth, and dissociation is essential for improving the efficiency of flotation. Many research[2-6] shows that regulated factors of the flotation process pull the intergrowth coal into the concentrate intensely, and flotation recovery rate increase too. If the intergrowth press into the tailings intensely and caused the coal loss, recovery rate is low. Because of the non homogeneous phase of coal composition, the dissociation of coal and the formation of the ash content is only a relative concept[7]. Therefore, grinding of coal should not only make the maximizing mineral dissociation, but also prevent produce over-fineness slime.This paper mainly introduces the relationship between the degree of fineness and the degree of dissociation of different materials, and discusses the most efficient and economical way of the dissociation products.

## 1 Test Section

This paper analysis coal industrial analysis, density of composition, particle size distribution and petrographic constituent, and provides reliable basic data for fine slime dissociation. The coal slime grinding test is carrying on in the XMQ240×90 type ball mill (ball ratio is: Φ30: Φ25: Φ20 = 40.04:33.45:26.51, filling rate is 34.82%) , and using float and sink test[8] to assess dissociation degree of different grinding times. Take copies of mixed homogeneous slime sample 1kg, and copies of +0.074mm grain size samples 1kg, and respectively carrying out the grinding test with different grinding times. Finally dissociation is further verified by flotation tests.

### 1.1 Sample Aanalysis

Coal ash content is high, reached 23.24%, so separation process should consider two aspects. One is the

 
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gangue (clay) and coal get mixed up, and the gangue content is higher. The second is that minerals(gangue) and clean coals compose the particles. In the first case, the gangue and clean coal can be separated by conventional methods, but in the second case, conventional methods is difficult to achieve the separation, namely, it need the liberation of coal and gangue minerals in order to achieve the separation.

**1.2 Particle Size Analysis**

The particle size analysis is take with different sieve pore of 0.28mm, 0.15mm, 0.45mm 0.074mm and 0.050mm standard set screen.

Tab. 1 Screening Analysis of Coal Slime

| Grain size（mm） | rate | ash | Positive accumulation | | Negative accumulation | |
|---|---|---|---|---|---|---|
| | | | rate | ash | rate | ash |
| +0.45 | 6.69 | 23.36 | 6.69 | 23.36 | 100.00 | 23.62 |
| +0.28-0.45 | 13.79 | 18.46 | 20.48 | 20.06 | 93.31 | 23.64 |
| +0.15-0.28 | 17.95 | 19.80 | 38.40 | 19.94 | 79.52 | 24.54 |
| +0.074-0.15 | 15.21 | 19.94 | 53.64 | 19.94 | 61.57 | 25.92 |
| +0.050-0.074 | 9.74 | 22.43 | 63.38 | 20.32 | 46.36 | 27.89 |
| -0.050 | 36.62 | 29.34 | 100.00 | 23.62 | 36.62 | 29.34 |
| subtotal | 100.00 | 23.62 | - | - | - | - |

Coal sample grain size distribution is very uneven(Tab.1), particularly fine particles is high content, and ash content is decreasing with particle size increasing. Compared with the whole grade the content of +0.45mm particles is less, and the content of 0.28-0.050mm is higher.

**1.3 Coal Petrographic Constituent Analysis**

Petrographic constituent analysis use Zeiss company M1m microscope photometer meter, with the reflected light, and scan step from the row spacing of 0.6 mm on the entire film. The coal organic composition has great relationship with the samples of density( See Tab.2).

Tab. 2 Coal petrography components of coal with different density（%）

| density（g·cm-3） | Organic component | | | Inorganic component | | | |
|---|---|---|---|---|---|---|---|
| | Vitrinite | Inertinite | Exinite | Clay | Pyrite | Calcite | Quartz |
| -1.3 | 87.77 | 8.93 | 2.33 | 0.39 | 0.58 | — | — |
| -1.4+1.3 | 74.71 | 20.23 | 1.95 | 2.33 | 0.78 | — | — |
| -1.5+1.4 | 52.22 | 41.59 | 2.13 | 3.48 | 0.58 | — | — |
| -1.6+1.5 | 43.55 | 50.17 | 2.36 | 2.56 | 1.36 | — | — |
| -2.0+1.6 | 22.96 | 60.12 | 0.97 | 11.87 | 2.53 | — | 1.56 |
| +2.0 | 9.54 | 63.26 | 0.93 | 15.58 | 10.23 | — | 0.47 |

Coal samples of vitrinite content decreased gradually with the density increased, and content of inertinite and gangue minerals include pyrite and clay gradually increased. The slime density in a certain extent also reflects the floatability[9]. Low density coal slime is rich in vitrinite, which is hydrophobic and low ash. Coal and mineral impurity density is about 1.45-1.80g·cm-3. The higher the content of the intermediate density, the poorer is the degree of dissociation of coal and mineral impurities, and the lower is the flotation efficiency. The higher the density, the mineral impurities content is the higher .

## 2 The Characteristics of Different Grinding Coal and the Dissociation Behavior

### 2.1 Grinding Test of Coal Slime

According to the analysis of coal slime characteristics, grinding test conditions of the coal further determines. The process is wet grinding, and concentration is 40%, and the grain size is -0.50mm. The grinding fineness is gradually increasing with the extension of the grinding time (see Fig.1), and the -0.074mm content of 15min is 86.85%. Grinding time longer than 15min, the increase of the fineness of grinding coal is slow. Float and sink analysis of grinding fineness carry out, compared with the density analysis of coal slime show in Figure 2.

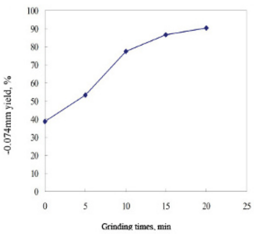



Fig. 1 Curve of grinding fineness

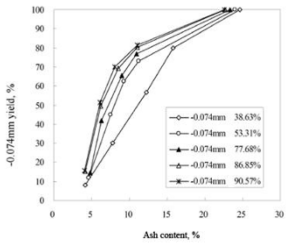



Fig. 2 Grinding coal yield - ash curve of density analysis corresponding

Relative distance between two adjacent curves can be consider as dissociation degree of the relative increasing value (Fig. 2). It is not difficult to find that the relative distance between the adjacent curves decreases, that means, the increase of the degree of dissociation gradually reduce.

Theoretically speaking, the finer grinding size, the mineral liberate better. But the finer of material grinding product, will inevitably lead to physical chemical properties changes of coal in the surface, and produce high ash slime at the same time. therefore, the reasonable grinding time and suitable grinding fineness is choose synthetical consider subsequent processing of the fine grinding of coal slime.

### 2.2 Flotation Test with Grinding Fineness

Grinding-flotation process optimum separation conditions are that speed is 8000r • min-1 with XHF-D high-speed dispersion machine, pretreatment pulp time is 20min, dosage of sec-octyl alcohol is 100g • t-1. dosage of sodium hexametaphosphate is 1500 g • t-1.When kerosene amount greater than 800 g • t-1, increasing trend of clean coal combustible recovery began to slow down, continue to increase the dosage of kerosene, coal combustible recovery increased slightly, but the coal ash begin to rise. The best separation result is clean coal yield about 58.72%, and coal ash content 10.69%, and coal combustible recovery 68.42%.

### 2.3 Property of Classified Coarse Particles

There is a little difference between +0.074mm coarse particles and coal slime according to the size composition and properties. The properties of +0.074mm particles analyzes in Tab.3 and Tab.4.

Tab.3 Size analysis of +0.074mm (%)

| Grain size（mm） | Rate,% | Ash,% | Ash distribution% |
|---|---|---|---|
| +0.5 | 8.18 | 19.23 | 7.72 |
| -0.5+0.25 | 14.86 | 19.10 | 13.91 |

| | | | |
|---|---|---|---|
| -0.25+0.125 | 41.57 | 19.87 | 40.50 |
| -0.125+0.074 | 35.39 | 21.82 | 37.87 |
| Subtotal | 100.00 | 20.40 | 100.00 |

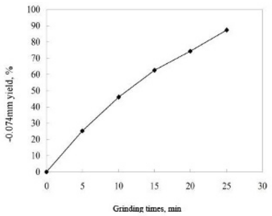
Tab.4 Density analysis of +0.074mm (%)

| Rate,% | Ash,% | Rate,% | Floating Cumulative Rate，% | Floating Cumulative Ash,% |
|---|---|---|---|---|
| -1.30 | 22.08 | 4.03 | 22.08 | 4.03 |
| -1.40+1.30 | 20.71 | 7.88 | 42.79 | 5.89 |
| -1.50+1.40 | 18.32 | 17.85 | 61.10 | 9.47 |
| -1.60+1.50 | 18.88 | 23.17 | 79.98 | 12.71 |
| +1.60 | 20.02 | 52.53 | 100.00 | 20.68 |
| Subtotal | 100.00 | 20.68 | — | — |

Coal slim ash change a little with particle size decrease, and still show an increasing trend(see Tab.3). Compared with the density composition of coal slime and the coarse, each density fraction yield is close, and low density content is higher, namely, that the organic matter fractions in the coal increase(see Tab.4). Based on the properties of coarse particles can be found that the grinding of coarse particles achieve the liberation of coal and gangue, and reduce the fine mud effecting on grinding environment.

**2.4 Fine Grinding Test of Coarse Particles After Classification**

The coarse grinding test is still used the XMQ240×90 type ball mill. The process is wet grinding, and concentration is 40%, the grain size of feed is -0.5+0.074mm. Results of the test shown in Fig. 3.



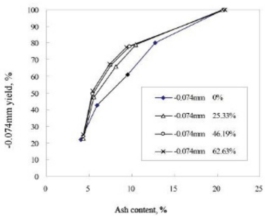



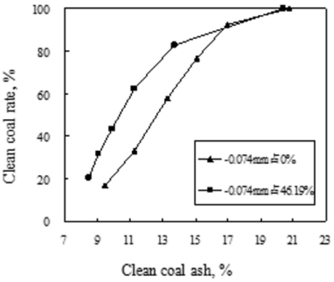

Fig. 3 Curve of grinding fineness of coars corresponding

Fig. 4 Grinding coal yield - ash curve of density analysis different finesse

-0.074mm fineness yield increase from 46.19% to 62.63% with the increase of the grinding times(see Fig. 3), and the relative distance between the adjacent curves of the low density product is reduced(see Fig. 4), that is, the degree of dissociation decreases gradually. Considering that the over-fineness materials is prone to mud and other factors, the suitable grinding fineness of coarse particles determine to be more appropriate as -0.074mm for 46.19%.

**2.5 Flotation Test with Grinding Fineness**

The dissociation degree of +0.074mm coarse particles increase, and the grinding fineness and -0.074mm slime of raw materials float separately, and then the two clean coal products mixed. The fineness of coarse particles select as -0.074mm for 46.19%. There is a big difference between the results of coarse grinding fineness flotation and direct flotation( See Fig. 5). After grinding flotation clean coal comprehensive cumulative curves show in Figure 7,that shows calculation of the regressive release flotation test results of the products of different size. Curve a is -0.074mm taking for 46.19%. Curve b is -0.074mm fractions after grading directly into the regressive release flotation test data calculated.



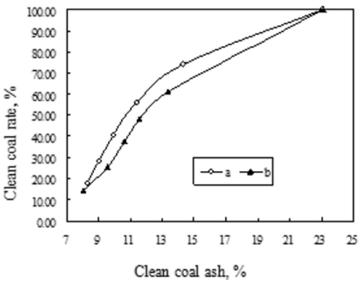




Fig. 5 Concentrate accumulation curves with different fineness of coarse particles

Fig. 6 Concentrate accumulation curves a Flotation of +0.074mm coal grinding 10mins, b Flotation of raw coal

The separation results of the coarse particles is better than that of the whole stage(see Fig. 6). From the line a can find that when the clean coal ash is about 11%, the yield of clean coal is about 55%. When the coal ash is 12%, the yield of clean coal is about 60%. By the regressive release flotation data values for the separation theory, that is, under normal circumstances, the flotation tests is only close to regressive release flotation analysis data, and it is hard to exceed. Therefore, the ash content of 12% and the cleaned coal yield of 60% can be considered as the optimization results of grinding-flotation flowsheet. Compared with the flotation production of this process and the plant field, the index has a certain increase.

**3 Discussion and Perspective**

This paper mainly studies the feasibility of coal grinding, and carries on the comparative analysis of different feeding by disassociation effects, and further investigates on coal grinding efficiency.

Through the analysis of the structure of clay and pyrite in the coal slime, coal and gangue minerals has not completely dissociated yet, and directly machining processing is easy to cause waste and pollution. Considering the grinding process of different feeds of coal slime, the coal quality of the matrix is the same. Therefore, there is a little change of the grindability. Researchers[10] use coal float and sink test data estimating the grindability, and establish the grindability index with the density change of the linear regression equation. It can consider as qualitative judgment of different material slurry grinding regularity. The paper verify the difference of grindability index of calculation and test results. The index equation is [11,12]: $HGI = -27.6325 SG_{cum} + 116.6265$ .SGcum is coal sample average density.

Tab.5 Grindability index of different grinding feed

| Grinding feed | Average density, % | HGI |
|---|---|---|
| Coal slime | 1.51 | 74.90 |
| +0.074mm coarse size | 1.47 | 76.01 |

Grindability index of coal slime and that of +0.074mm particles are separately 74.90 and 76.01(See Tab.5), which is close to each other, and still have a certain difference. Grindability index of coarse particles is obvious larger, and the grindability is relatively small, and grinding test results shows that the fine-grained level increased less consistent of coarse. It can be further shown that the calculation of the two kinds of grinding coal is basically reasonable. However, considering the amount of the grinding feed, that of the coarse obviously reduce, and the over-grinding reduce too. Therefore, finding the appropriate coal grinding technology should consider the various indicators.

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# Investigation of Energy Consumed Characterization of Mixture Grinding in the Ball-and-Race Mill

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**Abstract:** In this paper, grinding experiments of mixtures of super clean anthracite coal and pyrite or calcite in different volumetric ratios are conducted on a Hardgrove machine, with a power meter. Mass weighted Moh's Scale of hardness of mixture is incorporated into the classical coal breakage model of $t_{10}$ vs grinding energy, and this modified model can describe the energy-size reduction of mixture grinding. Interaction between components is embodied with energy and energy split phenomenon is discussed based on this mind. Based on the energy balance calculation of heterogeneous grinding, energy split factors (ESFs) of components in the ball-and-race mill at different grinding time are firstly computed. These two energies are calculated by the modified model with parameters in the mixture and single breakage, respectively. Interaction between components changes energy efficiencies of components in mixture grinding. ESFs of coals in mixtures are smaller than 1, which means less energies are required to yield the same $t_{10}$. Only ESF of calcite is bigger than 1. On the other hand, ESFs of components change with not only grinding time, but also volumetric proportion of components in mixture.

**Key words:** Pure mineral Mixed proportion Mixture grinding Energy consumed characteristic Energy-size reduction model Energy split factor Ball-and-race mill

**Introduction**

Comminution is an important technology in mineral processing to realize the size reduction and liberation of associated minerals. As metallic minerals and raw coal consist of multiple components, breakage phenomenon would be heterogeneous and grinding behavior of component is different with that in the single-component breakage. Note that heterogeneous grinding of mixture has been studied by researchers since 1960s, but these grinding experiments are mainly conducted on ball mill. Fuerstenau and coworkers have employed mixture feeds consisting of different pure minerals or the same mineral in different sizes (Fuerstenau and Abouzeid, 1991; Fuerstenau and Sullivan, 1962; Fuerstenau et al, 1992; Fuerstenau et al, 2010; Fuerstenau et al, 2011; Kapur and Fuerstenau, 1988). Based on differences in density, particle size and Moh's Scale of hardness, grinding behaviors of components in mixture are studied. On the other hand, energy split functions in terms of specific breakage rate of initial feed and generating rate of fines are established for analyzing different aspects of heterogeneous grinding (Kapur and Fuerstenau, 1988). Also, effect of fine particles on the breakage of coarse ones is revealed by ESF (Fuerstenau and Abouzeid, 1991; Fuerstenau et al, 2011). Mixtures grinding of ternary mineral systems of ceramic raw materials were also conducted to study the distribution of applied energy among components and offered optimization of energy utilization in the preparation of ceramic raw materials (Ipek, et al, 2005). Note that types of grinding devices are various and grinding mechanism are also different. Grinding behavior and energy consumed characterization are different among grinding equipment. Energy split function developed by Kapur and Fuerstenau cannot be used directly for mixture grinding in Ball-and-Race mill. For ball mill, impact and abrasion forces are mainly used to break minerals. But ball-and-race mill utilizes the extrusion force. Resistance strengths of particles to different forces are different and finally result in differences in grinding kinetics.

Hence, studies of energy characteristic of heterogeneous grinding in ball-and-race mill should be done based on experimental researches. In order to prevent the effect of associated minerals on breakage, super



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clean coal is selected. Two main associated minerals in coal are chosen, namely pyrite and calcite. Energy-size reduction of binary mineral system of coal with pyrite or calcite in different volumetric proportions are conducted on a Hardgrove mill. The classical coal breakage model is modified to describe the relation between product $t_{10}$ and specific energy of all mixture grinding. Interaction between components during mixture grinding is reflected on energy. Calculation of ESFs of components are done based on modified model. Changes of ESFs of components in mixture grinding are discussed.

## Materials and Methods

### *Experimental Materials*

Super clean coal was sampled from the Taixi Coal preparation plant and ash content is only 2.95%. Pyrite and calcite were bought from market. These materials were firstly broken by a jaw crusher separately and particles in the size fraction of -2.8+2 mm were sieved out for subsequent experiments. Moh's scale hardness of pyrite, coal and calcite are 7, 4 and 3. On the other hand, three components show big differences in density, with 1.25 $g.cm^{-3}$ of coal, 2.8 $g.cm^{-3}$ of calcite and 5 $g.cm^{-3}$ of pyrite.

### *Experimental Methods*

As size reduction of particles is the volumetric breakage, volume of materials for each test was fixed at 40 ml. For mixture A, coal and pyrite were in volumetric ratios of 7:1, 3:1, 2:1 and 1:1. Mixture B consisted of coal and calcite in these four proportions. Grinding tests were conducted on a Hardgrove mill, with a power meter. Grinding time was 0.5, 1, 1.5 and 2 min, and power signals were recorded. Powers were converted to energies and energies for particle breakage were calculated by subtracting non-load energy from total. Size distribution of ground products was analyzed by dry sieving, with sieves in meshes of 2, 1.25, 0.71, 0.355, 0.2 and 0.09 mm. Rosin-Rammler function was used to fit size data and value of $t_{10}$ was recalculated. Components of sized ground products were separated. Components of mixture in +0.09 mm were separated by float-sink tests and density of media was 1.5 $g.cm^{-3}$. Contents of components in mixture of -0.09 mm were calculated by subtracting content of +0.09 mm component from the total. $T_{10}$ of component in mixture was computed following the method mentioned above.

## Results and Discussion

### *Description of Energy-Size Reduction of Mixture*

Experimental data of $t_{10}$ vs specific grinding energy of this study are plotted in Figure 1. All in all, experimental figures are relatively scattered. Mixtures in different mass weighted Moh's scale hardness would have different grinding behaviors. Due to the relatively big difference in particle hardness of coal and pyrite, mass weighted hardness index of mixture A obviously increases with the increase of volumetric proportion of pyrite. As harder particles are difficult to be broken, energy efficiency, namely the value of product fineness $t_{10}$ at the same energy input level, sees a decrease trend with more pyrite in mixture. Because of relatively similar Moh's scale hardness of coal and calcite, differences in energy efficiency for mixture B in different mixed conditions are small.

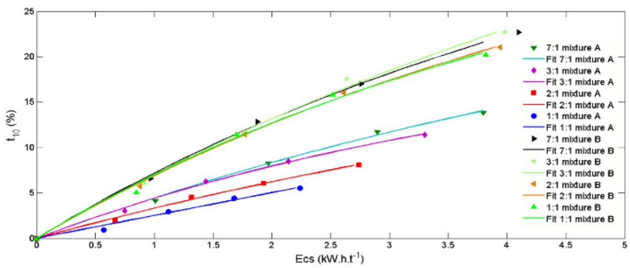


**Figure 1** Experimental data and fitting line for each multi-component grinding test

As mixture with each volumetric proportion of components also shows the property of narrow size and density, the original breakage model is utilized to fit the data of each mixture grinding. Namely,

$$t_{10} = A * (1 - e^{-b*E_{cs}}) \quad (1)$$

Where $t_{10}$ is percent of ground products in size of less than 1/10th of initial geometrical mean size of mixture (%), $E_{cs}$ is specific energy ($kWh.t^{-1}$). A and b are impact breakage parameters of that mixture.

Fitting results of equation 1 to experimental data are also shown in Figure 1. For each condition, fitting result is quite good. But it cannot fit data of all experiments. Hence, centralization of experimental data is very important. For our study, mass weighted hardness of mixture has an effect on grinding energy efficiency and results in different values of $t_{10}$ at same specific energy input. Obviously, as mass weighted hardness increases, the product fineness $t_{10}$ decreases. If specific energies are divided by corresponding mass weighted hardness, data points in Figure 1 will move towards left and scattered degree also decreases. Based on this mind, the hardness effect is modelled in the exponential term of equation 1. Modified equation shows the following form.

$$t_{10} = A * \left(1 - e^{-b*(E_{cs}/H_w)}\right) \quad (2)$$

Where $H_w$ is the mass weighted hardness index of mixture.

Fitting results of equation 2 are shown in Figure 2, with $R^2$>0.97. Hence, model incorporating hardness of mixture can describe energy-size reduction of mixtures in different conditions. Beyond these experimental results, supplemental tests were done. Namely, mixtures of coal-to-pyrite and coal-to-calcite in ratio of 1:2. These data points distribute around fitting line in Figure 2, which demonstrates modified model can reflect effect of weighted hardness on relation between $t_{10}$ and specific energy.

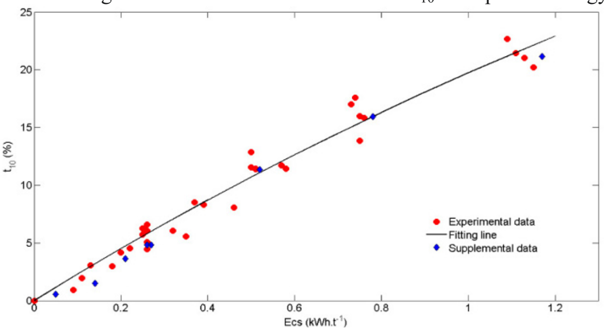


**Figure 2** Fitting results of modified breakage model to experimental and supplemental data

***Energy Split Factors of Components in Mixture Grinding***

In the above section, effect of hardness index of mixture on energy-size reduction is modelled in the modified breakage model. But interaction between components still cannot be revealed to explain the change of grinding behavior of component in mixture if compared with single-component breakage. Considering the establishment of modified model, attempt can be made to explicate the change of breakage behavior from the perspective of energy. Based on this method, following works are conducted.

Note that the total energy input during mixture grinding will be consumed by components. Namely,

$$E = E_1 + E_2 \quad (3)$$

Where E, $E_1$ and $E_2$ are grinding energies (kW.h) consumed by mixture, component 1 and 2, respectively. Grinding energies are converted to specific energy by dividing the mass of mixture.

$$E_{cs} = E/m = E_1/m + E_2/m \quad (4)$$

Where $E_{cs}$ is the specific energy of mixture (kW.h.t$^{-1}$), $m$ is the mass of mixture (t).

For component 1

$$E_1/m = (E_1/m_1)/(m/m_1) = w_1 * E_{cs1m} \quad (5)$$

Where $m_1$ and $w_1$ are mass and mass percentage of component 1 in mixture, respectively. $E_{cs1m}$ is the specific energy of component 1 in mixture grinding (kW.h.t$^{-1}$).

Substitution of equation 4 and 5 in equation 3 yields

$$E_{cs} = w_1 * E_{cs1m} + w_2 * E_{cs2m} \quad (6)$$

Where $w_2$ is the mass percentage of component in mixture, $E_{cs2m}$ is the specific energy of component 2 in mixture grinding (kW.h.t$^{-1}$).

Next, ESF of component is defined as the same meaning with Kapur and Fuerstenau's study.

$$S_1 = E_{cs1m}/E_{cs1a} \quad (7)$$

Where $S_1$ is ESF of component 1 in mixture grinding, $E_{cs1m}$ is the specific energy consumed by component 1 in mixture grinding (kW.h.t$^{-1}$), $E_{cs1a}$ is the specific energy consumed by component 1 to produce the same $t_{10}$ with component 1 in mixture (kW.h.t$^{-1}$).

Substitution of equation 7 in equation 6 yields

$$E_{cs} = w_1 * S_1 * E_{cs1a} + w_2 * S_2 * E_{cs2a} \quad (8)$$

Energy balance function is standard to evaluate the accuracy of calculated ESF. But energy consumed by component in mixture grinding cannot be measured directly. As modified breakage model incorporates hardness index of material, specific energy consumed by component in mixture can be calculated based on variant form of equation 2 by measured $t_{10}$ of component in mixture grinding. Equation 2 is converted to following form to compute required specific energy for $t_{10}$ of component in heterogeneous breakage.

$$E_{cs1m} = -ln((A - t_{10})/A) * H_1/b \quad (9)$$

Where $H_1$ is the Moh's scale hardness of component 1.

For homogeneous grinding, specific energy for $t_{10}$ is calculated by the variant form of equation 1.

$$E_{cs1a} = -ln((A - t_{10})/A)/b \quad (10)$$

According to equation 6, comparisons of measured and calculated specific energies of mixture A and B are made and results are illustrated in Table 1. Analyses of these date point out that differences between measured and calculated energies are relatively small, with the deviation of 15% of measured ones. Hence, ESFs of components can be computed by equation 7 and results of components in mixture A and B are displayed in Table 2. For mixture A, ESFs of components during all the grinding time are smaller than 1. Reason of this experimental phenomenon may be the difference between theory model and test. If equation 2 is only developed from data of mixture A or B, this modified model would be more suitable and accuracy for the energy-size reduction of binary system. As mass weighted hardness indexes of mixture A and B are modelled in equation 2, this model actually reflects the effect of all components on mixture grinding and can be applied for the ternary mineral system. But for mixture A or B, there are only two components and effect of the third component on breakage, which is described in modified model, cannot be investigated. ESFs of coals in mixtures are smaller than 1. For calcite of mixture B, ESF is bigger than 1. According to definition of ESF, value of it above 1 means more specific energy is consumed to yield the same $t_{10}$ of products in mixture grinding if compared with single-component breakage. Hence, breakage of calcite in mixture B is slowed. On the other hand, ESFs of components change not only with grinding time, but also with volumetric proportion of components in mixture. For mixture A, ESF of coal decreases with the decrease of volumetric ratio of it, and pyrite shows the same trend with the decrease of pyrite in mixture.

Presence of hard mineral pyrite in mixture promotes breakage of coal and this effect increases with more pyrite added in mixture. ESFs of calcite change slightly with different mixed conditions, but for coal in mixture B, ESFs increase with the decrease of volumetric ratio of coal. As time passing by, ESFs of coals in mixture increase gradually, which indicates the decrease of energy efficiency. But for the other component in mixture, ESF decreases with time.

**Table 1** Measured and calculated specific energies of mixture A and B

| Mixed proportions | 7:1 | | | 3:1 | | | 2:1 | | | 1:1 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Grinding time/min | $t_{10}$/% | Measured | Calculated | $t_{10}$/% | Measured | Calculated | $t_{10}$/% | Measured | Calculated | $t_{10}$/% | Measured | Calculated |
| | | | | | | Mixture A | | | | | | |
| 0.5 | 4.18 | 0.92 | 1.01 | 3.04 | 0.73 | 0.75 | 1.96 | 0.62 | 0.67 | 0.92 | 0.39 | 0.42 |
| 1 | 8.30 | 1.79 | 1.97 | 6.27 | 1.40 | 1.44 | 4.52 | 1.01 | 1.17 | 2.96 | 0.86 | 0.97 |
| 1.5 | 11.72 | 2.46 | 2.90 | 8.51 | 1.97 | 2.14 | 6.07 | 1.76 | 1.78 | 4.44 | 1.53 | 1.53 |
| 2 | 13.86 | 3.24 | 3.80 | 11.44 | 2.44 | 2.80 | 8.08 | 2.18 | 2.54 | 5.56 | 1.84 | 2.09 |
| | | | | | | Mixture B | | | | | | |
| 0.5 | 6.58 | 0.97 | 1.04 | 6.04 | 0.92 | 1.02 | 5.74 | 0.88 | 0.79 | 5.08 | 0.85 | 0.79 |
| 1 | 12.86 | 1.88 | 2.03 | 11.57 | 1.79 | 1.94 | 11.45 | 1.77 | 1.86 | 11.41 | 1.70 | 1.73 |
| 1.5 | 17.01 | 2.76 | 2.69 | 17.57 | 2.64 | 2.97 | 16.00 | 2.61 | 2.62 | 16.56 | 2.52 | 2.57 |
| 2 | 22.70 | 3.60 | 3.58 | 22.69 | 3.48 | 3.83 | 21.05 | 3.44 | 3.47 | 22.21 | 3.32 | 3.42 |

**Table 2** Energy split factors of components in mixture A and B

| Mixed proportions | 7:1 | 3:1 | 2:1 | 1:1 | 7:1 | 3:1 | 2:1 | 1:1 |
|---|---|---|---|---|---|---|---|---|
| Grinding time/min | | Coal in mixture A | | | | Pyrite in mixture A | | |
| 0.5 | 0.76 | 0.75 | 0.67 | 0.70 | 0.87 | 0.80 | 0.95 | 0.82 |
| 1 | 0.84 | 0.84 | 0.79 | 0.78 | 0.53 | 0.85 | 0.98 | 0.86 |
| 1.5 | 0.90 | 0.87 | 0.82 | 0.80 | 0.40 | 0.47 | 0.66 | 0.94 |
| 2 | 0.94 | 0.93 | 0.88 | 0.84 | 0.34 | 0.37 | 0.58 | 0.77 |
| | | Coal in mixture B | | | | Calcite in mixture B | | |
| 0.5 | 0.69 | 0.77 | 0.58 | 0.77 | 1.59 | 1.58 | 1.59 | 1.59 |
| 1 | 0.77 | 0.86 | 0.86 | 0.85 | 1.57 | 1.56 | 1.56 | 1.56 |
| 1.5 | 0.81 | 0.94 | 0.93 | 0.91 | 1.54 | 1.53 | 1.54 | 1.53 |
| 2 | 0.88 | 1.02 | 1.01 | 1.00 | 1.51 | 1.51 | 1.52 | 1.50 |

## Conclusions

Based on homogeneous and heterogeneous grinding in the Hardgrove mill, the mass weighted hardness of mixture was modelled into breakage model. The model can well fit the data of experimental and supplemental data. Energy split phenomenon is discussed based on modified model. ESFs are defined as the ratio of calculated specific energies, which is required to yield same $t_{10}$ of component in mixture, during the homogeneous and heterogeneous grinding. Analyses of ESF point out that presence of component has an effect on the breakage of the other one if compared with the single component breakage. Due to difference between theory model and experiments, only ESF of calcite is bigger than 1, which indicates more energy would be consumed for the breakage of calcite. ESFs of harder minerals in mixture decrease with the decrease of volumetric proportion of coal, and soft minerals in mixture show the contrary trend. On the other hand, ESF does not keep constant in experiments. ESFs of coal in mixture A and B decrease with time and the other component shows the opposite trend.

**Acknowledgement**

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# Advanced vibration technology innovations for coal screening

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**Abstract:**

With more than 95 years of experience is JÖST one of the worldwide leading companies in the field of vibration technology and bulk handling with subsidiaries and agencies all over the world.

JÖST will present a new screening solution for sticky materials like coal. The JÖST Non-Blinding screens offer a well approved solution for the fine separation of ROM coal without water and the fine separation of clean coal with a higher content of moisture. Conventional screens as well as flip flow screens cannot handle such critical product and have much shorter cleaning and maintenance intervals. With the JÖST Non-Blinding screens, screen cuts down to 10 mm are possible even if the feed material has a high content of clay or other impurities.

The presentation will provide technical details and results based on several years of experience in different installations.

In the long company history, JÖST delivered a lot of very interesting and impressive Non-Blinding-screens. From France over Mongolia to China. The JÖST Non-Blinding screens are successfully used around the world.

**Key Words:**
Vibration technology, Non-Blinding high capacity Screen, fine separation of ROM coal, coal handling, Vibrating Screen

The port in the province of Tianjin, 200 km from Beijing, serves as the central distribution point for the supply of coal to the country's power plants and steel mills. To meet the users' high demand, the coal is graded at the port in a screening plant before shipment. The enormously high flow of material - up to 10 000 m3/h - requires special technology and comprehensive knowhow.

**Case Study 1**

Working with a local plant engineering company, Jöst Germany was awarded the contract to supply two high-capacity mechanical screens and feed chutes for the Tianjin project.

Task:

- Material flow: 8000 tph coal (up to 10 000 m3/h).
- Screen cut: 60, 120 or 160 mm.
- Coal moisture content up to 14%.
- Material feed via a 2.6 m wide belt conveyor at a speed of 5.9 m/s.
- Division of the flow of material into no more than two subflows.
- Subsequent pooling of screen overflows.

A difficult challenge on this project was that the coal feed from a single conveyor had to be split uniformly to two screens.

 
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A special substructure diverted the material flow from the belt to a temporary storage bunker. The bunkers´ fill level was continuously monitored. The rate of material extraction was regulated by a hydraulically operated knife valve and the downstream feed chutes, which featured variable delivery rates and also performed the bunker extraction function simultaneously.
The two 3.8 x 4.0 m vibrating feeders in front of the screens were and designed in such a way that the material was evenly distributed for best efficiency and maximum utilization of the screen area. Key to the success of the project were two linear ‑motion vibrating banana screens. These high-capacity mechanical screens each had a screening area of 4.0 x 8.3 m (33.2 m3). A unique slotted screen deck was designed by Jöst to handle the moist and very sticky coal product. The slotted grid system's segmental design assures high throughput rates and can be replaced quickly. Different screen deck angles make it possible to achieve different conveying rates on the screen. This way the product layer in the feed zone remains so thin that bridging and blinding as a result of material pressure is virtually impossible. The conveying speed on the discharge end is reduced sufficiently to ensure that a very high efficiency of screening particles that are close to the screen cut size can be separated efficiently. These high-capacity vibrating screens, with an oscillating mass of 33t, were optimized by Jöst's designers using finite element analysis (FEA). Only two Jöst Type JR exciters are required for the drive system. The stroke can be continuously adjusted and frequency can be matched to the various feed particle sizes and distribution.

**Case Study 2**

Before coal can be treated at the power plant, it must be preclassified to prevent coarse material from impeding further processing, which must take place irrespective of weather conditions. Even after days of rainfall, a hard to handle coal slurry must be classified.

Jöst has solved this problem on behalf of a German equipment manufacturer for a French coal plant.
For the first step of classification, the operators used a roller screen which was fed by a feeding belt conveyor. The maximum possible grain size could not exceed 500 mm due to a upstream layer height limit.

In severe weather conditions, not only loose single grains but big lumps were transported over the screen deck under the assumption they could not be classified. But this material can be classified: after it was treated using Jöst vibrating technology, a test run was done with a coarse separator with positive results.

The solution for this application was a specially designed, shaft driven 1200 x 4800 Jöst grizzly screen. The shaft drive was placed off-centre to handle the extremely high load, especially in the feeding area. The Grizzly reaches acceleration values at a maximum stroke over 5 g and can therefore break up the large, contiguous chunks of material.

Special care was also taken to minimize potential for build up inside the machine. Inside the very steep side and end walls Jöst utilized a particularly smooth plastic liner. For this application, no inlet bottom was used in order that dead spots could be avoided. While the grizzly was designed for a feed capacity of 300 tph coal slurry, the actual performance has almost doubled at a screen cut of approximately 100 mm. Additional special applications for Jöst screens include fine coal screenings with moisture content up to 12%. This is vital for power plants which burn both domestic and imported coal. The quality of the latter does not always meet the required specifications. Here, linear vibrating screens with special screen decks are utilized, which can handle screen cuts up to 7 mm and remain almost clog-free. For even smaller screen cuts of products with high moisture content, the flip-flow OSCILLA screen is required. Extremely sticky materials may require the clog-free Grecco screen. For conventional screening of coal and lignite, time-tested and reliable circular and linear motion screens are available, as well as the pulverized coal injection control screens that can be designed to accommodate a particular application.

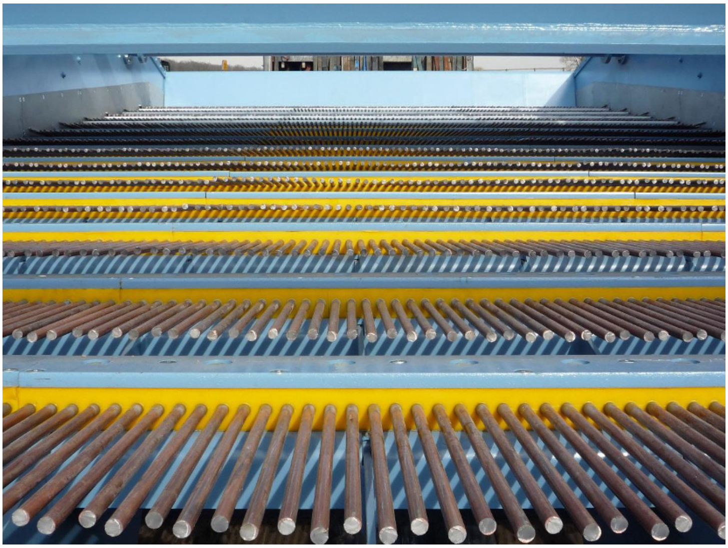
**Case Study 3**

For a raw coal processing plant in Baotou in Inner Mongolia, Jöst supplied non-blinding screens for a dry preclassification with a total capacity of 1200 tph. Because water supply in Mongolia is very limited, wet classification is very difficult, complex, and expensive. At the same time the raw coal in Baotou has a significant content of clay and is sticky and difficult to screen consistently. For such material, wet sorting screens are normally the machine of choice.

The solution: a water saving, dry pre-classification with high acceleration. The first test runs of the customer utilizing extremely high acceleration values failed. Machines with screen areas of more than 30 m3 experienced cracking and drive failures. Substituting flip-flow screens did not deliver success. The solution proved to be Jöst's special bar screens which concentrate high acceleration values while protecting the drive and screen box. The proper screening result is guaranteed as demonstrated by intensive test runs. The screen body is designed in best practice via FEA and is powered by a robust Jöst exciter. Special cross members take the over the segmental clamping of the bar screen system. Each screen can be fed with raw coal up to 600 tph. The smallest screen cut is 13 mm. The machines have an effective screen width of 4000 mm and a length of 8000 mm. Without isolation frame the screens weight 32 t per piece. Jöst supplied these screens as part of a major order which also included several large screens for pre-and fines-classification as well as the dewatering of the treated coal and mountain material.

Caption Figure 1: Non Blinding Screen Type SRGN 4000 x 8000

Caption Figure 2: Rod Finger System: STABROFLEX



# Improved screening of coal and other difficult to screen bulk materials by means of LIWELL® screens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**Abstract:**

Typically bulk materials such as coal and others like ore, coke, sand and gravel show significant worse screening properties in case of higher moistures and/or smaller required cut sizes. The surface moisture causes adhesive powers resulting in massive chokings of installed screen media. Consequences are loss of capacity / efficiency, fluctuation of product qualities and negative influences on other connected process steps.
To enable a sufficient classification even with smaller cut sizes and/or higher surface moistures the screening machine must provide higher accelerations to avoid choking of perforation up to the physical limits.
The special design of LIWELL® screens provide much higher amplitudes and accelerations compared to standard machines. To achieve higher amplitudes/accelerations of the installed screen panels the machine consists of 2 screen cases powered by an eccentric shaft. The eccentric shaft allows a 100 % defined movement independent of influences like load, granulometry, speed and others.
Typical applications are classification of coal for improved next processing (jig), production of fine coal for fluidized bed boiler, removal of fines for coal gasification processes.
Furthermore many other applications exist, for example, improvement of ore qualities by removal of fines, recycling applications and production of sinter coke.



**Key words: Flip-Flow, screen, LIWELL®, dry screening, fine screening, sizing, efficient screening, self-cleaning panel.**





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**Introduction**

First Flip-Flow machines type LIWELL® were introduced into the market about 50 years ago. Their most distinctive feature compared to conventional machines known before is a dynamic moving screen media. Conventional machines generally use screen panels fixed to structural elements of the machine. Therefore the screen panels typically provide the same amplitude and frequency as the machine itself. Achieved amplitudes and frequencies are limited to values the structural elements of machine(s) can resist. Standard machines run with accelerations around 4 – 5 g. Consequently the same acceleration is provided by the screen media itself to screen the bulk material.

Especially in case of small cut sizes and/or higher moisture levels of feed material (surface moisture) the mentioned acceleration is not sufficient to avoid clogging of meshes and perforations [Meinel, 1998]. The clogged perforation requires an interruption of production to clean and remove clogged particles.

Clogging and plugging problems can be reduced to a certain level by using specific screening media such like harp screens and other executions providing self-cleaning features [Westerkamp, 1997]. But, significant disadvantages such as short life time of screen media and increased percentages of misplaced particles limit their range of application.

The other common solution is the use of spray water (wet screening application) to avoid any adhesion forces of fine particles. Very good screen results opposed to the negative aspects of water use (economical and environmental impacts) request other solutions which allow classification without water.

Flip-Flow machines type LIWELL® are operated with dynamically moving screen panels. The movement is different to the movement of the structural machine components. Consequently much higher amplitudes and accelerations can be achieved [Schmidt, 1977]. Clogging and plugging of perforations can be avoided to a maximum possible extend without use of spray water.

In the meanwhile 50 years after introduction Flip-Flow machines became the state of art technology for most screening applications with small cut sizes and moist feed materials such as coal, coke, ore and others.

**Scheme of construction and function**

To achieve a significantly higher acceleration Flip-Flow machine always consist of flexible perforated screen mats made of Polyurethane. Each side of the mat is fixed on an individual cross beam. Cross beams. The complete number of cross bars is divided into 2 groups – system 1 and system 2 – which are moving during operation in the opposite directions. The result is a permanently changing distance which relaxes and stretches the screen mats. To generate the contrarious motion the cross beams of system 1 and system 2 are alternately attached to an inner and outer screen frame. An eccentric drive unit at the feed end of the machine actuates both screen frames.

While the screen frames and cross bars are subject to limited accelerations only (about 2 g [Hirsch, 1992]) the screen mats instead create accelerations of much more than 50 g. This extremely high acceleration is a result of the unique movement of screen mat: after relaxation the mats get tensed while the amplitude between relaxed and tensioned position is significantly high. Just before coming into the tensioned position the mat reaches its fastest speed [Schmidt, 1977]. The fast vertical speed of the screen mat is abrupt reduced to zero when the mats is tensioned resulting in highest negative accelerations releasing clogged and plugged particles.

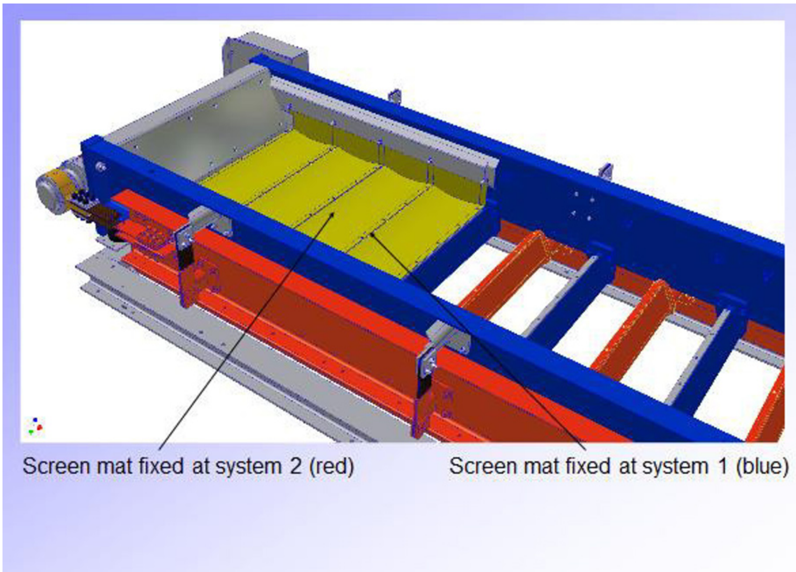


Fig. 1: screen system 1 (blue) and 2 (red) with some fixed screen mats (yellow)

For a sufficient flow of material the machine is installed in an inclined position. The inclination depends on the volume of feed material and its properties.

**Typical coal process applications**

Common requirements for coal classification processes are:

- Removal of fines for further improved processing
- Removal of fines for coal gasification purposes [Blesl, 2010]
- Removal of coarse particles for fluidized bed boiler applications
- Sizing of coal for final products [Gerdes, 1992]

Typically for the first 3 groups of applications the cut size is in the range between 3 – 10 mm [Kurtz, 1992]. The sizing of coal for the use as a final product can also range in bigger cut sizes. Due to the characteristic moisture up to 10 % and more the smaller cut sizes cannot be realized with common vibrating machines and the use of Flip-Flow machines becomes more and more compulsory [Snoby, 2003].

To improve further processing depending on fine coal properties the customer has several options of use:

- Fines can be sold as a final product
- Fines can be mixed with coarse coal concentrate or intermediate products if the ash content of coal mixture meets the customer requirements

In case of fluctuating coal properties screen arrangements with bypass offer a flexible operation mode with more or less removal of fines depending on the requirements.

**Example 1:**

In year 2015 the biggest Russian coal producer - SUEK group installed a Flip-Flow machine type LIWELL® LF 3.0 – 8.82/28 DD in double deck execution (screen area each deck 26.5 m²) in their coal processing plant at “Taldinskaya Zapadnaya 1”. The machine is designed for a small cut size of 6 mm. The additional cut size of 30 mm is used as a protection removal of lump sizes at a nominal capacity of about 500 tph.

After commissioning the results confirmed, that high efficiencies can be achieved even with higher moisture levels of coals:

Table 1: Typical results of coal grade “D” with an analyzed moisture of total 13.1 %

| Coal size | Feed | Coarse Product | Efficiency |
|---|---|---|---|
| **6 - 130 mm** | 42,0 % | 94,0 % | 95,4 % |
| **0 - 6 mm** | 58,0 % | 6,0 % | |

**Example 2:**

Since 2007 SDS Coal operates 2 parallel installed machines type LIWELL® LF 3.0 – 8.82/28 ED (screen area 26.5 m²) to remove fines to increase the efficiency of the installed jigs. The capacity of each machine amounts to 500 tph.

Table 1: Typical results with an moisture of 10.8 – 11.3 % of coal grade “D”

| Coal size | Feed | Coarse Product | Efficiency |
|---|---|---|---|
| **+ 50 mm** | 16,0 % | 26,0 % | 88,4 % |
| **13 - 50 mm** | 34,2 % | 54,2 % | |
| **6 – 13 mm** | 16,0 % | 14,2 % | |
| **0 – 6 mm** | 33,9 % | 5,6 % | |

Also the installation of LIWELL® screens at the coal preparation plant of Vorkutinskaya mine (Severstal Resources) confirms the positive effect of fines removal to improve the coal preparation efficiency at the already existing jig. After installation of Flip-Flow screens with size 2000 x 7560 mm, which enable a cut size of 4 mm (feed 220 tph, surface moisture 6,0 %, efficiency > 90 %), the yield of coal concentrate at the plant itself increased about 3.0 % resulting in an increased production volume of 30.000 tons per year.

**Example 3:**

The biggest coal producer in China SHENHUA at his preparation plant in Baotou (Lijiahao CPP) operated 4 circular motion machines for a total capacity of more than 2000 tph, but smallest cut size was about 13 mm only. To reduce the cut size down to 6 mm and to increase the production of coarse coal the circular motion machines were exchanged against 4 LIWELL® screens type LF 3.0 – 8.82/28 ED.

After commissioning the results of the reduced cut size of 6 mm were:

Table 2: Operational results at Shenhua Baotou Lijiahao CPP

| Analyze No. | + 6 mm in feed | - 6 mm in feed | + 6 mm in overflow | - 6 mm in overflow | Capacity [tph] | Efficiency [%] |
|---|---|---|---|---|---|---|
| 1 | 70,2 % | 29,8 % | 96,4 % | 3,6 % | 600 | 92,2 % |
| 2 | 69,8 % | 30,2 % | 97,5 % | 2,5 % | 600 | 94,7 % |
| 3 | 72,1 % | 27,9 % | 97,3 % | 2,7 % | 600 | 93,7 % |
| 4 | 66,5 % | 33,5 % | 96,8 % | 3,2 % | 600 | 94,2 % |

Even with capacities up to 750 tph per machine the screen(s) still similar efficiencies are achieved.

Picture 1: installed LIWELL® screen

Picture 2: Shenhua Lijiahao CPP installation opening

**Example 4:**

A new application in the Republic of South Africa confirms the above mentioned results. Here, the Stuart Coal Group commissioned two LIWELL® screens LF 3.0 – 8.82/28ED at the Stuart Coal Mine:

- The fines content in the feed material amounts to 40.6 % minus 5.8 mm.
- The achieved efficiency in terms of fines removal was 93,6 % while the percentage of coarse particles bigger 6 mm in the underflow product was less than 4,2 %

While taking the samples a surface moisture of 7,4 % in the underflow was analyzed – also this relatively low moisture partially provided the remarkably good results while higher ash contents generally reduce the efficiency.

**Further applications**

Other bulk material applications are usually applications with smaller cut sizes and/or higher moisture levels. Important examples are coke, iron ore, crushed hard rock, different kinds of slags [Dangeleit, 1998] and natural sand [Kortekamp, 1980].

Coke produced for sinter purposes requests a cut size around 3 mm. Typically coke has a high moisture level up to 15 % and very sticky properties. Also for such applications Flip-Flow screens offer sufficient solutions due to the self-cleaning properties of screen panels. Even with the mentioned significant difficult properties it is still possible to achieve feed capacities around 4 tph/hm².

**Conclusion**

Flip-Flow screens type LIWELL® provide reliable results in the coal industry and many other industrial applications worldwide.

With the use of the Flip-Flow technology it is possible to achieve cut sizes in the range of 3 – 6 mm or bigger with exceptional efficiencies for dry screening of moist materials

At the same time operational cost are in a usual range and allow a return of investment after a short period of time.

Environmental aspects are fulfilled due to additional advantages/consequences like operation of the screening machine(s) without water and the reduced amount of flocculants required for the operation of processing plant itself.

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# A new approach for processing and agglomeration of low-rank coals for material usage

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Abstract

The material usage of low-rank coal offers a high potential regarding the increasing in added value. The production of coke from a wide span of low-rank coals may be a promising perspective. Since low-rank coals do not exhibit any baking properties they need to be briquetted before pyrolysis to get a lump coke. Using two brown coals the influence of processing parameters and briquetting parameters were investigated. The comprehensive investigations showed that, besides the well-known relations between briquetting parameters, the changing of processing from conventional comminution of the coal to new processing leads to increasing strength of briquettes and coke. Due to the intense novel comminution in the flat die press as well as in the mixer the binding between the particles may be increased and the shrinkage behaviour of the coals was improved during pyrolysis so that coke strength could even exceed the briquette strength. The new developed pyrolysis briquettes and the lump coke show high mechanical strength and only small amounts of fines due to the mechanical stress, leading to high yield in the coarse coke fraction.

Key words
Processing of low-rank coal, material usage of coal, briquetting, granulation, mixer agglomeration, press agglomeration, lump coke

## 1 Introduction

Today low-rank coals like brown coal, lignite and sub-bituminous coals are mainly used for power and heat production. Considering the opening of new resources, the material utilisation of low-rank coals will be intensified. Besides extraction of coal ingredients like montan wax (Fehse et al. 2013, Fehse et al. 2014a) or gasification of coal the pyrolysis of coal enables the production of new materials for various applications (Schmalfeld 2008). The three main products of coal pyrolysis are coke, tar and oil as well as gas. Since so called low-rank coals exhibit higher volatile matter they offer comparatively high amounts of tar/oil and gas.

Gases, tars and oils may be processed to chemical raw materials. The quality of the coke is highly dominated by the operation parameters of the pyrolysis as well as the previous steps of refinement and the raw material itself. Thus every coal needs specially matched processing and pyrolysis parameters.

In contrast to so called coking coals, the low-rank coals do not exhibit any baking capacity. Thus if lump coke shall be produced from low-rank coals, e. g. brown coal this is only possible by the gentle pyrolysis process up to 1000 °C of high quality briquettes. The processing of lump coke from Lusatian brown coal was developed by Erich Rammler and Georg Bilkenroth in 1952 (DD4630, 1952). This coke was a milestone in brown coal conversion and could substitute the coke of bituminous coals in several applications. But if the raw material base is extended to coals of other deposits or ranks, further developments on the oven and the operation as well as the pre- and posttreatment are needed.

 
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For this reason the briquetting and coking behaviour of selected low-rank coals was investigated first in laboratory scale in fundamental investigations to analyse the potential of lump coke production.

## 2 Material and Methods

### 2.1 Materials

For the following investigations a Lusatian brown coal from Germany (LBC) with a moisture content of 56.2 % and an ash content of 2.98 %(d) as well as an Indonesian brown coal (IBC) with a moisture content of 58.7 % and an ash content of 3.55 %(d) were used.

### 2.2 Processing, agglomeration and coking of the coal

To investigate the influence of coal processing three different approaches were used (figure 1). For the investigations the coal was pre-comminuted in a hammer mill to $\Delta d \approx 6/0$ mm.

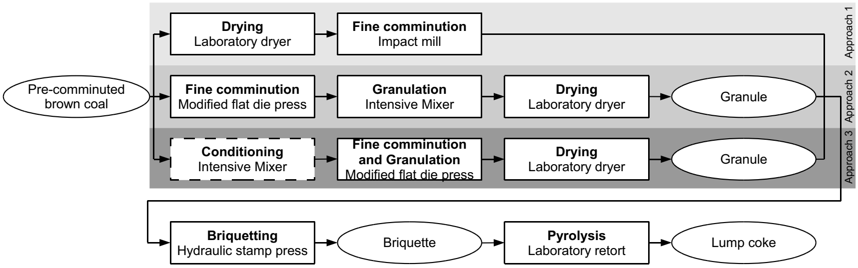


*Figure 1: Approaches for coal processing for lump coke production from low-rank coals*

Using approach 1 the pre-comminuted coal was dried in a laboratory drying cabinet with air of 105 °C. Afterwards the dry coal was fine comminuted in an impact mill with beater disk rotor to $\Delta d \approx 1/0$ mm. Briquetting was done using a hydraulic stamp press forming briquettes of 50 mm diameter and 20 mm height. The formed briquettes are subsequently coked in a laboratory retort using a gentle heating regime according to Vollmaier (Vollmaier 1954) up to 1000 °C.

In approach 2 the fine comminution is done of the wet coal in a modified flat die press (A. Kahl, Type 14-175) and granulated in an intensive mixer (Eirich, Type R02) according to Fehse et al. (Fehse et al. 2014a, Fehse et al. 2014b). Due to shear and compressive stress the coal is highly comminuted in the flat die press. The modified die causes only a slight agglomeration in the press channel since the tight part ($d_1 = 4$ mm) is only very short ($h_1 = 2$ mm) and widens up to 6 mm. This enables a high re-expansion of the coal after compaction. In the mixer the granules get their final form. By higher comminution especially of the wet coal the binding potential may be increased (Straßburger 2004). If the lignite is formed to wet granules e. g. by mixer agglomeration the new binding potential could be comprehensively used (Schubert et al. 2003). As in approach 1 drying is done in a laboratory drying cabinet at 105 °C. The dried granules are subsequently briquetted and coked like in approach 1.

A third approach considers the comminution and granulation in one step using the modified flat die press. If necessary the coal can be conditioned in the intensive mixer using steam, water or other additives. The flat die press can be equipped with dies of various press channel diameters of $d_1 = 2 \ldots 4$ mm using the geometry discussed in Fehse et al. 2014a. The formed granules are dried, briquetted and coked just like in approach 2.

### 2.3 Scope of experiments and parameters of evaluation

In the first step, the influence of the processing parameters (figure 1) was determined for constant briquetting parameters ($w = 11$ %, $p = 140$ MPa, $\vartheta_P = 70$ °C, $t_P = 3$ s). In the second step briquetting parameters ($w$, $p$) were varied using Lusatian brown coal processed according to approach 1. Under constant briquetting pressure ($p = 140$ MPa) the moisture content was varied between 5 % and 21 %

($\vartheta_P = 70$ °C, $t_P = 3$ s). Afterwards the influence of the briquetting pressure was determined by pressure variation between 100 MPa and 200 MPa. The moisture content remained constant at 11 % ($\vartheta_P = 70$ °C, $t_P = 3$ s).

To characterise the coal for briquetting the *particle size distribution* of the coal was determined by sieving analysis. The quality of the briquettes and cokes was determined by the following parameters: *Compressive strength* (according to former TGL 9491): The briquettes and cokes are loaded with a force between two stamps of 30 mm diameter until breakage. The maximum pressure is defined as compressive strength. *Raw density*: After briquetting, respectively coking, the agglomerates were measured with a calliper gauge and weighted to calculate the raw density.

## 3 Results

### 3.1 Influence of processing parameters

#### 3.1.1 Particle size distribution of the charging material

At first the influence of processing parameters should be discussed. Regarding figure 2 the particle size distribution of Lusatian brown coal shows high differences against the processing parameters.

The impact comminuted brown coal shows 90 % of the particles in the grain size of $\Delta d = 1/0$ mm. This particle size distribution complies with the specification of the approach developed by Rammler and Bilkenroth. If the coal is comminuted in a modified flat die press and granulated by mixer agglomeration, the particle size distribution changes to coarser grain size. The here exemplarily shown size distribution was achieved by granulation of the comminuted coal with 20 % of steam for 180 s at impeller speed of 11.1 m s$^{-1}$ in the intensive mixer. Due to agglomeration processes the amount of fines ($d < 0.1$ mm) is highly reduced to 1.2 %. The highest amount of granules shows the grain size fraction of $\Delta d = 1.6/1.0$ mm. By comminution and granulation of the coal in the modified flat die press with $d_1 = 2$ mm an even tighter particle size distribution may be achieved. Due to the press channel and the height of the blade the maximum size of the granules is fixed. Since the agglomeration is only very smooth a small amount of fines may also accrue due to breakage and abrasion $Q_3(d = 0.1\text{ mm}) = 1.89$ %.

In general wider particle size distributions with a small amount of oversize particles are favourable for press agglomeration. Since the volume of the gaps between the particles is smaller, rearrangement processes are reduced and no comminution of coarse granules is affordable. The granules do offer the opposite: The particle size distribution is narrow and shows a highly reduced amount of fines and oversize particles. But the original particle structure of the coal was comminuted in the flat die press and the formed granules exhibit new properties.

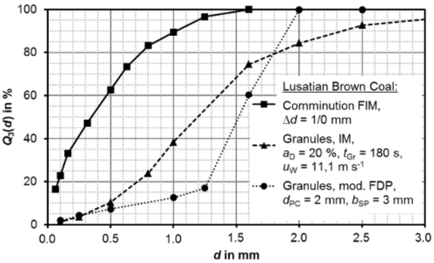


*Figure 2: Particle size distribution of Lusatian brown coal against varying processing approaches, w = 11 %*

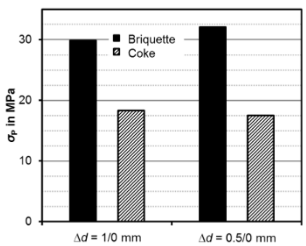


*Figure 3: Compressive strength of briquettes and cokes of Lusatian brown coal with varying grain size (w = 11 %, p = 140 MPa, $\vartheta_P = 70$ °C, $t_P = 3$ s)*

#### 3.2.2 Briquette and coke quality

Using comminuted dry coal of $\Delta d = 1/0$ mm the briquettes achieve a compressive strength of 27 MPa (figure 3). If the coal is comminuted to $\Delta d = 0.5/0$ mm the compressive strength slightly increases to circa

33 MPa, which means a 22 % increasing of the compressive strength. Due to finer particle sizes the specific surface of the particles and thus the binding potential increases, which leads to higher briquette strength. For the cokes no clear tendency could be observed. The coke compressive strength only reaches 18 MPa. It could have been assumed, that due to the finer particles in the briquette the agglomerates offer better shrinkage behaviour of the cokes. But for the Lusatian brown coal this effect is not visible. In contrary the average compressive strength decreases slightly.

Comparing briquette and coke strength, the briquette strength is higher than the coke strength, which means due to the degassing of the briquettes during pyrolysis the structure is weak and could not be raised in the same quantity by the following shrinkage process.

If approach 2 is used oppositional correlations can be observed. Figure 4 shows the briquette and coke strength using granules of mixer agglomeration for briquetting. During mixer agglomeration the steam ratio was varied. The experiments were done for Lusatian brown coal from Germany and an Indonesian brown coal. As covered in Fehse et al. 2015 for Lusatian brown coal the steam ratio is the most influencing factor regarding the variation of steam ratio, granulation time and impeller speed. New experiments with Indonesian coal confirmed this trend. Therefore the influence of steam ratio should be discussed at this point.

For Lusatian brown coal the briquette compressive strength reaches up to 39.6 MPa with maximum steam ratio of 20 %. The compressive strength of the coke shows the same trend. For a steam ratio over 10 % the coke compressive strength exceeds the briquette strength. When the amount of steam in the mixer increases the temperature rises as well, leading to intensified stress of the particles and higher comminution. During the pyrolysis those granules offer better shrinkage behaviour; and due to the granular structure in the briquette, defined ways for degassing lead to less weakening of the briquettes.

Comparing the achieved compressive strength of the cokes from approach 2 to approach 1 the compressive strength could be more than doubled by new processing regime. For Indonesian brown coal a similar trend is visible. As discussed for the Lusatian brown coal the briquette and coke compressive strength increase with rising steam ratio. The average briquetting compressive strength reaches the maximum at a steam ratio of 25 %. The coke compressive strength reaches the average maximum at the steam ratio of 20 %. But since the fluctuation rate of the coke compressive strength is definitely high due to a non-uniform temperature distribution during pyrolysis in the laboratory retort only the trend may be discussed. Comparing the data with the Lusatian coal the briquettes from the Indonesian brown coal offer comparable compressive strengths. But the coke of the Indonesian coal does exceed the briquette compressive strength independent of the steam ratio. It even reaches up to 110 MPa compressive strength at $a_{St} = 20$ %. Probably the shrinkage behaviour of this coal is much better than for the Lusatian brown coal.

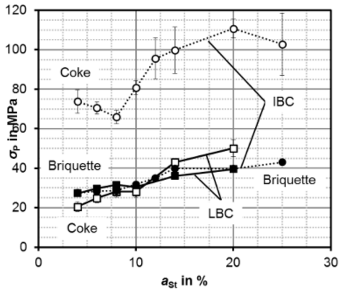


*Figure 4: Compressive strength of briquettes and cokes from granules of approach 2 varying the steam ratio ($t_{Gr}$ = 180 s, $u_I$ = 11.1 m $s^{-1}$, w = 11 %, p = 140 MPa, $\vartheta_P$ = 70 °C, $t_P$ = 3 s)*

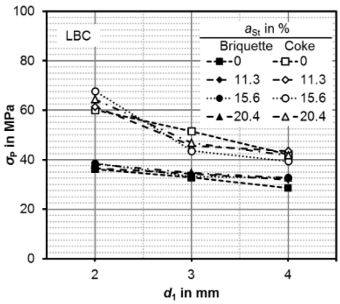


*Figure 5: Compressive strength of the briquettes and cokes using approach 3 under variation of the steam ratio and the press channel diameter (w = 11 %, p = 140 MPa, $\vartheta_P$ = 70 °C, $t_P$ = 3 s)*

In approach 3 Lusatian brown coal was conditioned with steam in an intensive mixer and afterwards comminuted and granulated in the modified flat die press. During the investigations the steam ratio of conditioning and the press channel diameter in the die were varied. As known finer press channel diameters lead to stronger granules due to the higher compaction. The porous structure of the granules reduces; and as the area of the holes in the die decreases with decreasing press channel diameter, the dwell time in the comminution area increases, leading to higher comminution before granulation. Figure 5 shows the achieved results of the compressive strength. The changes of particle structure effects the increasing compressive strength of the briquettes with decreasing press channel diameter. Even if the granules get stronger the briquettes get stronger too. This means that even if more energy is needed for rearrangement and breakage of the particles in the stamp press before the agglomeration the briquettes show higher compressive strengths than the ones from lower pre-agglomerated material.

The steam does have a slight influence on the briquette compressive strength. But it is dominated by the press channel diameter of the die. For the coke a clear trend of the steam ratio could not be determined. But due to the higher comminution of the coal at smaller press channel diameters the cokes offer higher compressive strength potentially caused by the better shrinkage behaviour.

Comparing all three approaches of processing, the briquetting and following coking of granules from the modified flat die press offers the highest mechanical strength. The approach also allows the combination of comminution and granulation in one single step. If scale-up is done to a larger flat die press the conditioning with steam can take place in the press as well since the dwell time of material in the machine increases.

### 3.2 Influence of briquetting parameters

In the second step moisture content and briquetting pressure were systematically investigated using Lusatian brown coal processed according to approach 1 with a grain size of $\Delta d = 1/0$ mm.

At first the moisture content was varied between 5 % and 21 % using a briquetting pressure of 140 MPa, briquetting temperature of 70 °C and a briquetting time of 3 s. Figure 6 shows the compressive strength and raw density of the briquettes.

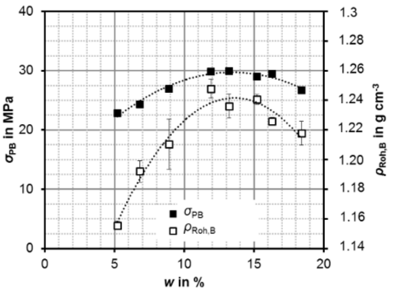


*Figure 6: Compressive strength and raw density of the briquettes of Lusatian brown coal for different moisture contents ($p$ = 140 MPa, $\vartheta_P$ = 70 °C, $t_P$ = 3 s)*

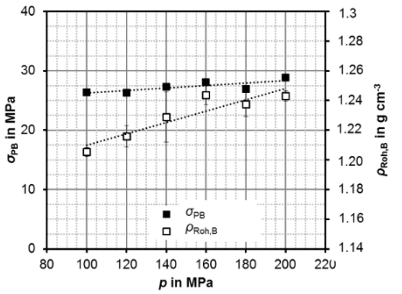


*Figure 7: Compressive strength and raw density of briquettes of Lusatian brown coal for different briquetting pressures ($p$ = 140 MPa, $\vartheta_P$ = 70 °C, $t_P$ = 3 s)*

Figure 6 shows the highest compressive strength at the moisture content of 12 % to 13 % with circa 30 MPa. The compressive strength follows a parabolic curve. Regarding the raw density, the maximum is in the same range of moisture content and achieves a maximum value of 1.248 g cm$^{-3}$. This means if the moisture content is fewer than 12 % the bonding between the coal particles is on a lower level since the lower moisture content leads to more brittle behaviour of the coal under stress. At higher moisture content the coal shows a more plastic behaviour and more hydrogen bonds can be created between the particles.

The higher moisture content also leads to higher raw density as the embedded amount of water increases. But over the moisture content of 13 %, respectively 15 %, the compressive strength and raw density increase as the water avoids a higher compaction.

As numerous investigations showed (Krug/Naundorf 1984) for the production of pyrolysis briquettes a so called "semi optimum" moisture content should be used to reduce the water which needs to be removed in the following pyrolysis causing the weakening of the briquette structure. Therefore a moisture content of 11 % was chosen for the following variation of the briquetting pressure.

The briquetting pressure was varied between 100 MPa and 200 MPa at briquetting temperature of 70 °C and a briquetting time of 3 s. As shown in figure 7 only a slight increasing in compressive strength could be determined for increasing briquetting pressure. The maximum compressive strength may be achieved at 200 MPa. The raw density shows the same trend. Due to higher pressure the compaction can be intensified and more Van-der-Waals forces can be created between the coal particles leading to higher strength. Besides this other investigations show, that a further increasing of briquetting pressure leads to decreasing briquette strength. Because at the higher pressure new defects in the briquette structure are created, which lead to less briquette strength.

Although the investigations were done for one approach the influence of the briquetting parameters can be applied to the charging materials from the other processing approaches. But due to the different comminution and the pre-agglomeration the position and level of the maxima will differ.

**4 Summary**

Using a Lusatian brown coal and an Indonesian brown coal the influence of the preparation process on the briquette and coke quality was determined. The compaction of pre-agglomerated material, especially from the modified flat die press (approach 3), significantly increases the briquette and coke strength due to the higher comminution in the wet state by different stress causing a better compaction behaviour of the granular coal. The variation of moisture content and briquetting pressure affects the briquette quality as well. On the one hand a minimum moisture content and briquetting pressure is required for agglomeration. On the other hand the exceeding of a coal specific maximum level of both parameters lead to less compaction. That means an intense processing of the coal is required for the creation of high strength briquettes and only in combination with optimum briquetting parameters pyrolysis briquettes can be produced to create a high strength lump coke.

In the next step the influence of the pyrolysis regime needs to be determined with varying raw material base in consideration of the optimum processing and briquetting parameter sets. In combination of all three steps the parameter sets for the production of a new lump coke from low-rank coals with custom-designed parameters can be identified.

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# HYDRAULIC SCREENING AS METHOD OF ROM COAL MACHINE GRADES PREPARATION

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**Abstract**

Considered are aspects of application of hydraulic screens with the fixed and combined (fixed + moving) screening surface of various geometrical shapes (rectangular, conical, conical-and-rectangular) for preparation of machine grades at preparatory wet screening of ROM coals by separation size of 6-20mm.

It is shown that efficiency of preparatory wet screening depends on energy intensity of process, but is determined by the value of so-called screening work consumed directly for screening the grades being under the separation size to the undersize product. Based on cost-effectiveness balance, optimum is application of the combined shape of screening surface providing the factory section productive capacity of 500-1000t/h in case of single-flow version, with the resulting quality of large machine grade meeting the requirements of heavy-media separation in magnetite medium.

**Keywords**

Coal, hydraulic screening, hydraulic screen, separation size, machine grade, efficiency, energy intensity.

In recent years, given the dramatically increase of fines and moisture content in the ROM coals and lowering of near-mesh separation size of preparatory screening, preparation of machine grades becomes one of major process operations at the coal preparation plants with the separation depth of 0(0.5)mm. Such significance level of this operation is due to the fact that the hydraulic jigging and heavy-media separation (as the main separation processes) offer the highest quality and quantitative indexes when handling the raw material of specific grain-size distribution where content of substandard grains in this material impairs separation efficiency of concentrating machines [1, 2].

As coal preparation experience in the Ukraine has shown, the most perspective method of machine grades preparation is hydraulic screening being a size classification of particulate matter on the screening surfaces, generally, by utilizing hydrodynamic forces, and involving elements of water saturation, hydraulic transport, washing and filtration. Scientific basis of hydraulic screening was developed by the Ukrniiugleobogashchenie Institute [3-7]. Machines utilizing hydraulic screening method are known as hydraulic screens.

The Ukrniiugleobogashchenie Institute developed and introduced hydraulic screens with the fixed screening surface (NPP) of flat (ГГН type, fig. 1*a*) and conical (ГНК type, fig. 1*b*) shapes and with the combined (fixed + moving) screening surface (KPP) of conical-rectangular (УМГ type, fig. 1*c*) and conical (ГГКИ-К type, fig. 1*d*) shapes.

Hydraulic screen operating principle involves hydraulic preparation of raw material, its pulpifying and forming of two-phase flow (liquid + solid) having the specific hydrodynamic parameters; washing of oversize product on the fixed screening surface with the high-pressure hydrodynamic jets; and check screening and dewatering on moving screening surface. There are no any hazardous and harmful chemical, biological and human operational factors.



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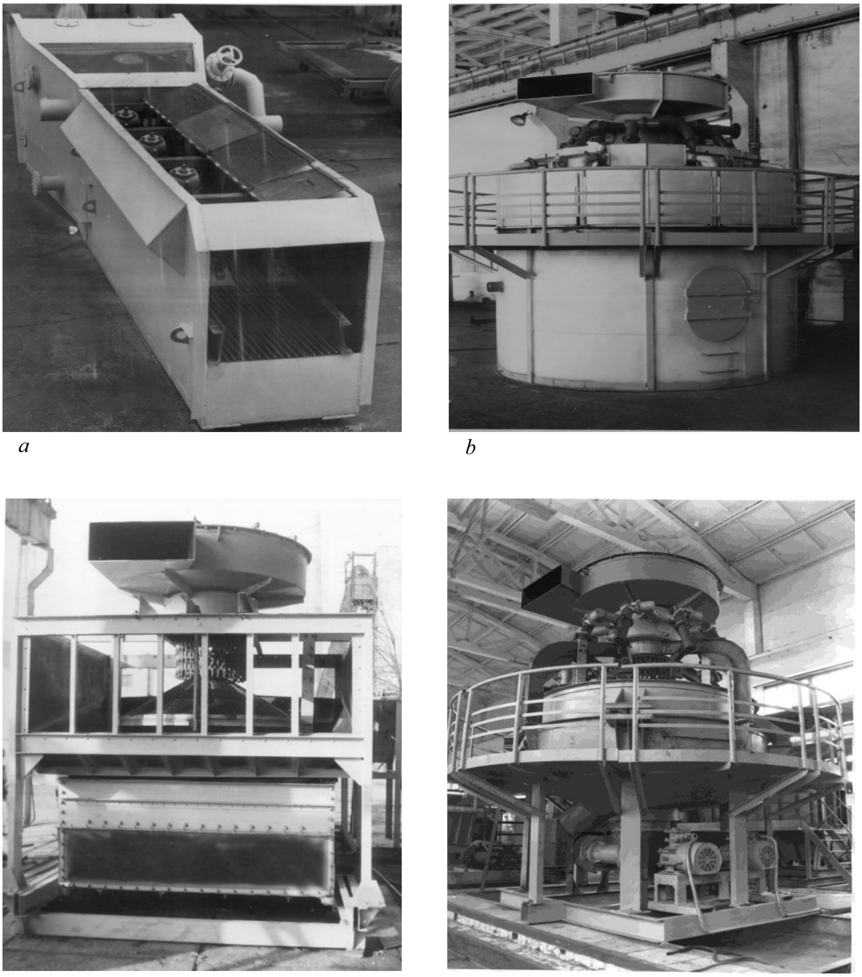
screening and dewatering on moving screening surface. There are no any hazardous and harmful chemical, biological and human operational factors.



*a* *b* *c* *d*

Fig. 1. Hydraulic screen types: *a* – ГГН; *b* – ГНК; *c* – УМГ; *d* – ГГИК-К

The hydraulic screens of ГГН type (fig. 1*a*) are intended for the preparatory wet screening of ROM coal into two machine grades of granular material with density of over 1.5t/m$^3$ and the near-mesh separation size of 6-18mm for the hydraulic jigging separation process.

The hydraulic screens of ГГН type offer ROM coal classification capacity of 400-850 t/h per machine at the load per unit surface of 105-160 $t/m^2 \cdot h$ and oversize and undersize products contamination with the substandard grains of 12-18 and 1.2-1.8%, respectively, at oversize product moisture content of over 16%. Hydraulic screens of ГГН type are also used for the modular (serial) installation of ГГН-2,7 and ГГН-4,2 hydraulic screens with ГИСЛ-62У and ГИСТ-72А unbalanced throw screens.

The hydraulic screens of ГНК type (fig. 1б) are intended for the preparatory wet screening of ROM coal into two machine grades with the near-mesh separation size of 10-20mm for the large machine grade classification by the method of heavy-media separation in magnetite medium. The ROM coal must be readily or moderately screenable; otherwise, the oversize product moisture content exceeds required value of 10%.

The tests showed that at the ГНК-600 and ГНК-1000 screens capacity of 525-600 and 650-970 t/h, respectively, averaged contamination of oversize product at the separation size of 10(13)mm makes 7,7%, including content of 0-1mm grade in the oversize product of at most 1.5%. Screening efficiency as for the lower grade reaches 97.6%. Distinctive feature of ГНК hydraulic screens is that they offer high productive capacity of 600-1000 t/h at minimum water consumption of 0.5-0.7$m^3$/t for preparation of machine grades over the wide range of near-mesh separation size.

Based on reasonable aggregation principles, hydraulic screens of the УМГ-2,5 and ГГКИ-К type with the combined screening surface were created on the base of ГНК hydraulic screen [8].

The УМГ-2,5 installation (fig. 1*в*) includes the conical NPP (right cone with distributor and pan) equipped with the cyclone-type feeder and located inside the special-purpose bin above the moving screening surface (PPP) of the ГИСТ-72А unbalanced throw screen at its main frame. Outlet funnel of feeder is also mounted on crosspiece in line with the distributor of conical screen. Discharge sleeve from pan of fixed screen could pass through the screens of unbalanced throw screen between two tie-beams or through the back wall of screen.

Acceptance tests of the УМГ-2,5 installation showed that at capacity of 650-800 t/h, specific water consumption of 0.6-0.8$m^3$/t and separation size of 13mm, screen sizing results meet the requirements as to the separator feed (content of 0-13mm grade in the oversize product – 6.4-10.1%, 0-1mm grade – 0.41-1%, over 13mm grade in the undersize product – 0.8-1.7%; average screening efficiency makes 93.9% efficiency as for the lower grade makes 96.8%.)

The ГГКИ-К hydraulic screen (fig. 1*г*) is intended for the wet classification of ROM coal into the machine grades with the near-mesh separation size of 6-20mm and is suitable for the dewatering and deslurring operations.

The ГГКИ-К hydraulic screen consists of feeder with the metering device and conical NPP and PPP mounted on fixed funnel-type pan and moving frame installed inside the fixed bowl, respectively. Hydraulic screen is equipped with the anti-vibration mounts, water supply devices and electric-driven vibrators, and is installed on the supporting legs.

ГГКИ-К-600 hydraulic screen in-process testing results showed that at the coal separation into the machine grades by separation size of 10mm, content of substandard grains in the oversize product made 6.3-8.5% including 0-1mm grade content of 0.1-1.3%. Contamination of undersize product with +10mm grains made 1.4-2.0%. Moisture content of oversize product did not exceed 6.4%, average screening efficiency made 90.9%, screening efficiency as for lower grade made 98.2%. It is found that at the hydraulic screen specific productive capacity of 42-80$t/h \cdot m^2$ corresponding to productive capacity of 420-800t/h and specific water consumption up to 0.6$m^3$/t, separation products quality completely meets the requirements as to quality of heavy-media separator feed.

Technical specifications of hydraulic screens are resulted in the Table below.

Fig. 2 shows results of the investigation of kinetics and energy intensity of preparatory coals wet screening on the fixed (NPP), moving (PPP) and combined (KPP) screening surfaces [9,10].

According to the hydraulic screening kinetics analysis, in case of serial installation of hydraulic screens and moving screens (fig. 2x) over the total length of about 10m, sustained recovery is observed of the undersize product of grade under the near-mesh separation size within the 90-99%. Similar recovery at the УМГ-2,5 installation is achieved over the length of about 6m.

**Technical specifications of hydraulic screens**

| Type | ГГН-2,7 | ГГН-4,2 | ГГН-5,5 | ГНК-600 | ГНК-1000 | УМГ-2,5 | ГГКИ-К-600 | ГГКИ-К-1000 |
|---|---|---|---|---|---|---|---|---|
| Productive capacity, t/h | 500 | 750 | 850 | 600 | 1000 | 830 | 600 | 1000 |
| Total area of screens, $m^2$ | 2.7 | 4.2 | 5.5 | 9.2 | 12.0 | 17.2 | 10.0 | 16.4 |
| fixed | 2.7 | 4.2 | 5.5 | 9.2 | 12.0 | 3.6 | 4.0 | 6.4 |
| moving | - | - | - | - | - | 13.6 | 6.0 | 10.0 |
| Specific water consumption, $m^3$/t | 1.2 | 1.2 | 1.2 | 0.7 | 0.7 | 0.8 | 1.0 | 0.7 |
| Oscillation frequency, $s^{-1}$ | - | - | - | - | 12.25 | 12.25 | 12.25 | 12.25 |
| Oscillation amplitude, mm | - | - | - | - | 6 | 6 | 6 | 6 |
| Power of electric motors, kW | - | - | - | - | - | 44 | 26 | 44 |
| Overall dimensions, mm: | | | | | | | | |
| length | 5550 | 7616 | 8035 | 4600 | 5100 | 7300 | 3850 | 4800 |
| width | 1670 | 1990 | 1600 | 4600 | 5100 | 4620 | 3850 | 4800 |
| height* | 4700 | 6900 | 7280 | 4300 | 4950 | 5360 | 4860 | 5800 |
| Weight, kg | 6400 | 6600 | 6800 | 12500 | 14000 | 18435 | 11890 | 19800 |

* - Working position height

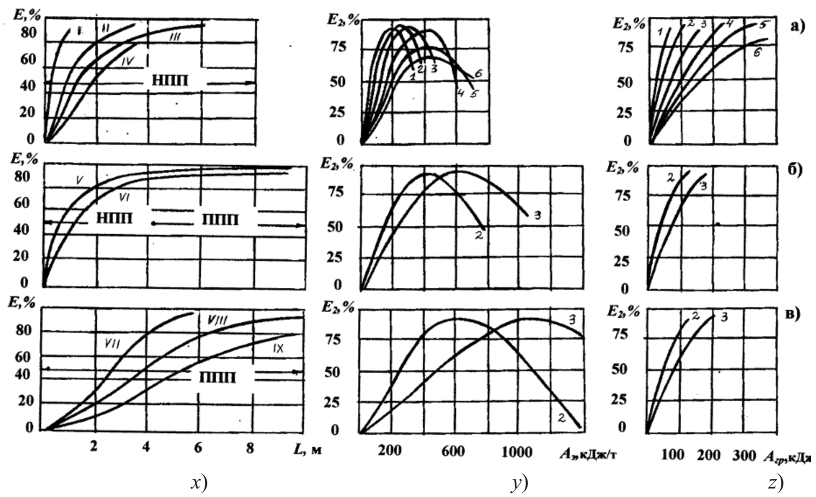



Fig. 2. Results of the investigation of kinetics and energy intensity of preparatory coals wet screening: 1, 2, 3, 4, 5, 6 – separation size: 1, 3, 6, 10, 13, 18mm, respectively

Fig. 2*y* shows the screening efficiency $E_2$ variations with preparatory wet screening process energy intensity implying existence of certain screening energy consumption limit, behind which the increase in process energy intensity causes reduction rather than increase in screening efficiency. The larger near-mesh separation size, the more quickly this limit occurs. For example, in case of the near-mesh separation size of 18 and 6mm, optimum energy intensity of screening process on the fixed screening surface makes 150 and 350 kJ/t (fig. $2y_a$), respectively.

Fig. 2*y* also implies that achievement of optimum value of energy amount input for hydraulic screening at the same near-mesh separation size will be different for different types of screening surfaces.

For instance, at the near-mesh separation size of 10mm, optimum hydraulic screening process energy intensity makes about 300 kJ/t for the fixed screening surface, about 600 kJ/t for the combined surface and about 1200 kJ/t for the moving surface.

Separation process and its efficiency depend on the screening work $A_{ГР}$. Fig. 2*z* shows screening efficiency variation with *AGR* at preparatory wet screening on NPP, PPP and KPP. For example, fig. 2 $z_a$ implies that the smaller near-mesh separation size, the greater specific screening work shall be executed by the system in order to achieve the same efficiency. Thus, in order to achieve, for example, screening efficiency of 80% at the separation on NPP by the near-mesh size of 1, 3, 6, 10, 13, 18mm, system should consume 500, 250, 180, 120, 70 and 50 kJ/t directly for screening.

**Conclusion.** The ROM coals hydraulic screening on the fixed and combined screening surfaces of various geometrical shapes is a reliable method of machine grades size classification prior to separation using the heavy-media separation and hydraulic jigging methods with productive capacity of 500-1000 t/h.

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# Emprical Relationships of HGI in terms of proximate analysis of coal

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**Abstract**
In this study empirical relationship of HGI (Hard Groove Grindability Index) was established in terms of proximate analysis of coal. HGI is important parameter at mining applications such as excavation, crushing, grinding, dust generation. HGI has long been researched and empirical equations evaluating coal proximate analysis data has been proposed. Grindability of coal should be in consideration for power plant operators as well since milling requires high energy and no frequent maintenance is desired for mills. Determination of HGI experimentally takes time and studies address on the estimation of them in an easier way. Proximate analysis of coal is easily evaluated and coals are classified with respect to their proximate analysis results. That is why, with the help of proximate analysis data, method can be developed to estimate HGI. .In this study user friendly methods were proposed to predict HGI and this could be helpful tool for the initial estimates. This user friendly method was proposed and applied on the data taken from previous studies. Proposed estimation method of HGI are in good agreement with the previous studies' results.

**Key words**: Grindability, HGI, Proximate analyis

## Introduction

Estimation of HGI should include parameters which could be easily obtained by field engineers. Establishment of relation between HGI and proximate analysis data of coal would help to field engineers, since evaluation of proximate analysis data takes not much time. Regression analyses techniques have used to assess various coal properties with HGI. HGI has been tried to be correlated with proximate analysis (Ozbayoglu et al., 2008, Chelgani et al., 2008), petrographic composition (Ozbayoglu et al., 2008; Bagherieh et al., 2008; Chelgani et al., 2008; Jorjani et al., 2008); ultimate analysis (Chelgani et al., 2008) and mineral matter and ash content (Ural and Akyıldız, 2004; Jorjani et al., 2007). In the study of Ural and Akyildiz (2004) suggested a method employing with mineral matter and ash content to estimate HGI. According to Chelgani et al., (2008) ultimate analysis of coal was related to coal HGI, and in same study it was found out that coal proximate analysis data gives higher regression coefficient than that of ultimate analysis. Peisheng et al., (2005) proposed method by relating the proximate analysis data of Chinese coals and HGI and high precision was obtained. Ruberia et al., (1999) studied the effect of blending on grinding and combustion behavior of 3 types of coals. Park and Kim (2012) has carried out a study in order to determine the affecting parameters on the ratio of product loss from a mill and they evaluated the low rank coals' (China and Indonesia lignites) proximate analysis and HGI.

HGI of coal could be related to many parameter including proximate analysis. Having obtained as many parameters of coal as possible would ease the correlation between HGI and these parameters. Approach should be employing as less parameters as possible and it should show good agreement with experimentally obtained HGI



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values. Furthermore the new approach should also be user friendly and field engineers easily can estimate HGI. In this study, new approach for HGI estimation by using proximate analysis is proposed. HGI was related to proximate analysis data of previous studies. Proposed new approach was compared with the previous studies experimental findings.

**Experimental Procedure of HGI**

The experimental procedure for the determination of HGI is as followed according to the ASTM standard. The 50 g sample of prescribed size -1.18+0.6 mm is taken into the ball mill of HGI machine along with 8 iron balls. Mouth of the ball mill is closed and it is set to rotate for exactly 60 revolutions. After the requisite rotation, the machine stops automatically. The sample left in the ball mill is then collected along with any powdered substance sticking to the surface of the machine with the help of a brush. This sample is then put in a sieve of 75µm size and is shaken for about 10 minutes. The sample which passed through 75µm size and retained on 300 µm size is weighed on the balance. The hardgroove grindability index of coal is than calculated using the following formula.

HGI = 13 + 6.93 w

Where, w = weight of the test sample passing through 75µm sieve and retained on 300 µm sieve after grinding in the HGI machine.

**Result and Discussion**

From previous studies, proximate analysis result of coals and corresponding HGI values are given in Table 1. Previous studies' proximate analysis data was employed in order to set up a user friendly method to estimate HGI. .Equation (1) is the proposed method for HGI prediction and valid among 124 coal proximate analysis data and HGI.

$$HGI= -105+A+11*VM+FC+0.03*A^2-0.42*VM^2+0.0024*FC^2-0.0005*A^3+0.005*VM^3-0.00001*FC^3 \quad (1)$$

Prediction of HGI was succesfull with provided formulas given (Eqn 1). Test results obtained by using the prediction methods are presented in Table 2.

The methods proposed are applied on the previous studies data [1,2,3,4,5]. Results of the proposed methods and experimental results obtained by the researchers are compared in Figure 1, predicted and experimental HGI values are plotted.

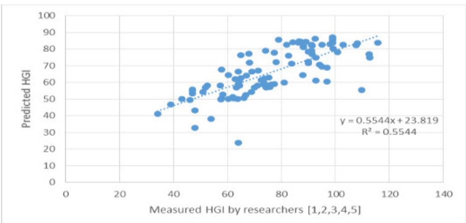


Figure 1. Predicted HGI vs experimental HGI by each researcher

**Conclusion**

In this study, prediction of HGI in terms of proximate analysis of coal was realized regarding previous studies. Ash content, moisture, volatile matter or fixed carbon could be effective more or less individually or together. Proximate analysis of coal samples is is more or less related to measured HGI values. Determination of which proximate analysis element is more crucial on HGI and how these proximate analysis parameters do effect HGI was remarked deeply. Prediction method resulted as the ash content is more significant on HGI characteristics. Engineering applications need a quick answer of HGI values and proximate analysis of the coal is always available. Regarding the proximate analysis, user friendly method is suggested in this study. In order to have a quick estimate of HGI is provided. In this study data from previous studies was evaluated in terms of the estimation of HGI. Although limited number of coal samples and the variety of coal type studied by the researchers, very good agreement is achieved between measured and predicted HGI values respectively by proposed method.

Table 1. Previously proposed methods to predict HGI

| Reference | A | VM | FC | HGI | Reference | A | VM | FC | HGI |
|---|---|---|---|---|---|---|---|---|---|
| Rubiera et al. 1999 | 6.70 | 34.10 | 59.20 | 47.00 | Peisheng et al 2005 | 27.3 | 16 | 55.7 | 86.4 |
| | 8.00 | 17.40 | 74.60 | 91.00 | | 34.3 | 37.1 | 24.5 | 109.6 |
| | 9.30 | 3.80 | 86.90 | 34.00 | | 18.9 | 16.7 | 63.6 | 88.2 |
| Rattanakawin and Tara 2012 | 44.50 | 22.19 | 23.75 | 68.00 | | 25.2 | 11.2 | 61.4 | 112.4 |
| | 25.27 | 29.53 | 34.02 | 47.00 | | 29 | 12.2 | 57.4 | 88.1 |
| | 14.65 | 28.29 | 43.60 | 43.00 | | 29.4 | 36.6 | 32.3 | 75.6 |
| | 15.25 | 32.62 | 42.41 | 39.00 | | 27.2 | 46 | 24.8 | 60.2 |
| | 13.11 | 30.24 | 40.93 | 48.00 | | 19.2 | 35.2 | 40.1 | 57.5 |
| | 18.61 | 29.56 | 38.45 | 46.00 | | 33.7 | 21.2 | 44.2 | 115.6 |
| Park and Kim 2012 | 6.43 | 33.68 | 33.68 | 64.00 | | 41.5 | 30.8 | 26.6 | 71.3 |
| | 11.87 | 28.02 | 31.88 | 48.00 | | 22.7 | 37.6 | 35.2 | 66.1 |
| Su et al 2010 | 26.22 | 23.99 | 49.79 | 89.9 | | 23 | 11.4 | 64 | 67.8 |
| | 19.71 | 24.54 | 55.75 | 92.7 | | 38.4 | 42.8 | 14.8 | 57.3 |
| | 14.71 | 26.74 | 58.55 | 96.9 | | 32.3 | 15.4 | 50.9 | 99 |
| | 8.05 | 26.85 | 65.10 | 69.10 | | 30.4 | 8.7 | 58.3 | 94.1 |
| | 9.51 | 30.91 | 59.58 | 77.40 | | 29.6 | 12.9 | 56.1 | 102.9 |
| | 13.96 | 25.24 | 60.80 | 83.70 | | 18.8 | 36 | 41.7 | 62 |
| | 8.45 | 30.41 | 61.14 | 80.90 | | 21.6 | 40.4 | 33.9 | 62.3 |
| | 54.41 | 18.08 | 27.51 | 77.40 | | 41.3 | 25.9 | 31.7 | 82.5 |
| | 11.48 | 28.47 | 60.05 | 87.80 | | 27.8 | 33.4 | 37.4 | 64.2 |
| | 71.05 | 13.32 | 15.63 | 53.90 | | 12.2 | 32.1 | 52.6 | 50.8 |
| | 30.96 | 20.02 | 49.02 | 78.80 | | 28.6 | 19.5 | 50.8 | 84.2 |
| | 18.91 | 23.05 | 58.04 | 91.30 | | 22.3 | 37.8 | 37.4 | 58.3 |
| | 32.42 | 21.18 | 46.40 | 87.2 | | 29.8 | 39 | 30.1 | 63.3 |
| | 15.7 | 25.5 | 58.80 | 89.9 | | 27.6 | 38.6 | 30.9 | 68.7 |
| | 4.67 | 31.08 | 64.25 | 52.5 | | 33.2 | 13.2 | 52.1 | 86.3 |
| | 26.09 | 38.12 | 35.79 | 51.8 | | 34.7 | 36.4 | 27.2 | 76.1 |
| | 21.37 | 36.58 | 42.05 | 47 | | 27.8 | 33.2 | 38.1 | 72.4 |
| Peisheng et al 2005 | 35.2 | 17.7 | 46.4 | 99 | | 21.4 | 40.9 | 33.1 | 63.8 |
| | 40.3 | 18.6 | 40.3 | 92.4 | | 24.3 | 38.3 | 34.2 | 66.7 |
| | 50 | 24.3 | 24.9 | 112.6 | | 16.4 | 34.6 | 44.4 | 57.8 |
| | 31.9 | 32.2 | 33.7 | 75.2 | | 21.7 | 22.2 | 54.9 | 100.8 |
| | 28.3 | 35.6 | 34.8 | 71 | | 30.4 | 23.6 | 44.6 | 74 |
| | 28.5 | 34.7 | 35.5 | 73.6 | | 31.1 | 20.5 | 47.4 | 99 |
| | 27.7 | 35.6 | 35.6 | 64.4 | | 30 | 21.3 | 47.6 | 95 |
| | 24.3 | 35.3 | 39.2 | 74.3 | | 19.2 | 25 | 54.6 | 90 |
| | 26.2 | 36.4 | 36.5 | 70 | | 31 | 21 | 46.8 | 98 |
| | 16.5 | 40.3 | 40.9 | 60.4 | | 45 | 24.1 | 28.2 | 65 |
| | 19.9 | 20.1 | 59.1 | 99.1 | | 24.1 | 29.9 | 42.4 | 65 |
| | 35 | 22.6 | 41.7 | 107.6 | | 25.6 | 28.8 | 43 | 64 |
| | 20.2 | 15.6 | 63.2 | 91.4 | | 27.4 | 25.2 | 445 | 67 |
| | 25.1 | 20 | 54 | 91.2 | | 22.1 | 31.4 | 42.8 | 64 |
| | 44.3 | 33.4 | 21.3 | 62.9 | | 28.4 | 27.2 | 40.9 | 95.1 |
| | 53.1 | 31.4 | 14.4 | 73.5 | | 21.5 | 25.2 | 51.4 | 78 |
| | 37.8 | 33.7 | 26.9 | 73.6 | | 27.1 | 19.2 | 52.5 | 108 |
| | 27.8 | 33.1 | 37.6 | 92.7 | | 31.4 | 21.3 | 46.1 | 82 |
| | 43.6 | 34.9 | 20.5 | 96.9 | | 45.7 | 15.8 | 37.2 | 89 |
| | 16.4 | 8.8 | 73.7 | 57.8 | | | | | |

A (Ash %), VM (Volatile Matter %), FC (Fixed Carbon %), M (Moisture %), HGI (Hard Groove Index)

Table 2. Test results obtained by the prediction methods proposed.

| Reference | Predicted HGI | Experimental HGI | Reference | Predicted HGI | Experimental HGI |
|---|---|---|---|---|---|
| Rubiera et al. 1999 | 53.41 | 47.00 | Peisheng et al 2005 | 84.86 | 86.40 |
| | 79.05 | 91.00 | | 55.54 | 109.60 |
| | 40.96 | 34.00 | | 81.83 | 88.20 |
| Rattanakawin and Tara 2012 | 71.73 | 68.00 | | 76.92 | 112.40 |
| | 55.10 | 47.00 | | 81.22 | 88.10 |
| | 50.11 | 43.00 | | 57.22 | 75.60 |
| | 46.88 | 39.00 | | 64.42 | 60.20 |
| | 43.24 | 48.00 | | 49.91 | 57.50 |
| | 49.52 | 46.00 | | 83.74 | 115.60 |
| Park and Kim 2012 | 23.64 | 64.00 | | 67.00 | 71.30 |
| | 32.72 | 48.00 | | 50.66 | 66.10 |
| Su et al 2010 | 78.54 | 89.90 | | 77.22 | 67.80 |
| | 74.92 | 92.70 | | 58.06 | 57.30 |
| | 68.81 | 96.90 | | 85.60 | 99.00 |
| | 66.59 | 69.10 | | 70.76 | 94.10 |
| | 59.17 | 77.40 | | 82.55 | 102.90 |
| | 71.34 | 83.70 | | 51.19 | 62.00 |
| | 59.83 | 80.90 | | 50.42 | 62.30 |
| | 77.94 | 77.40 | | 76.07 | 82.50 |
| | 64.34 | 87.80 | | 60.64 | 64.20 |
| | 38.16 | 53.90 | | 54.25 | 50.80 |
| | 85.49 | 78.80 | | 83.99 | 84.20 |
| | 77.06 | 91.30 | | 52.65 | 58.30 |
| | 84.56 | 87.20 | | 56.99 | 63.30 |
| | 71.53 | 89.90 | | 54.22 | 68.70 |
| | 58.06 | 52.50 | | 83.69 | 86.30 |
| | 57.01 | 51.80 | | 58.77 | 76.10 |
| | 55.86 | 47.00 | | 61.50 | 72.40 |
| Peisheng et al 2005 | 86.98 | 99.00 | | 50.01 | 63.80 |
| | 86.31 | 92.40 | | 52.56 | 66.70 |
| | 74.77 | 112.60 | | 50.42 | 57.800 |
| | 62.90 | 75.20 | | 78.11 | 100.80 |
| | 58.18 | 71.00 | | 78.96 | 74.00 |
| | 59.26 | 73.60 | | 83.87 | 99.00 |
| | 58.18 | 64.40 | | 82.53 | 95.00 |
| | 57.00 | 74.30 | | 72.47 | 90.00 |
| | 57.07 | 70.00 | | 83.05 | 98.00 |
| | 50.09 | 60.40 | | 76.22 | 65.00 |
| | 80.28 | 99.10 | | 62.55 | 65.00 |
| | 82.26 | 107.60 | | 66.39 | 64.00 |
| | 81.95 | 91.40 | | 64.18 | 67.00 |
| | 82.52 | 91.20 | | 58.86 | 64.00 |
| | 62.16 | 62.90 | | 69.46 | 95.10 |
| | 58.79 | 73.50 | | 72.28 | 78.00 |
| | 62.18 | 73.60 | | 83.61 | 108.00 |
| | 60.97 | 92.70 | | 82.79 | 82.00 |
| | 60.49 | 96.90 | | 84.31 | 89.00 |
| | 67.68 | 57.80 | | 84.86 | |

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# Sub 38 µm Wet Sizing with Sieves

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**Abstract**

The development of methods to accurately wet-sieve at sub 38 micron size fractions have allowed important information to be gathered on a range of materials in the coal and minerals processing industries. Methods have been developed to sieve at 2, 5, 10 and 20 micron. Using sub 38 µm sieves gives the ability to actually study (on a bulk scale) the material within these size fractions. For example, the actual material in the 0 to 10 µm or 10 to 20 µm size range can be studied. As well as producing the particle size distribution, sufficient quantities of each size range can be collected to allow further test work and laboratory analysis to be performed. Other techniques do not allow for this. Laser sizing gives only a size distribution. Cyclosizers operate on a very small sample size (20 g total) and have numerous other problems that are discussed in this paper. A range of data from samples that have been gathered and processed at the sub 38 µm sieve sizes is presented as well as direct comparisons of the various sizing techniques. Materials so far studied are: magnetite, coal flotation feed, product and reject, coal thickener underflow and mineral sulfides.

Key words; sizing, particle size distribution, sieves, cyclosizer, laser sizing, magnetite, partition

## Introduction

Currently, three laboratory methods of sub 38 µm sizing are used in the coal industry in Australia. They are; laser sizing techniques, sedimentation/elutriation techniques and the fine sieving techniques that are the subject of this paper. All three methods have advantages and disadvantages and it is important to understand these when comparing the three techniques.

Laser sizing works on the principle of diffraction of light by the particles and the accuracy is limited by the refractive index and the shape of the particles (Eshel et al. 2004 and Campbell 2003). If all the particles in the sample to be studied have the same refractive index and are perfect spheres, then the technique can be highly accurate. Laser sizing techniques give only a size distribution, they do not produce samples of different size ranges for further analysis.

Cyclosizing is the most common sedimentation/elutriation technique used in Australian coal laboratories. Essentially, Cyclosizing separates the samples into different fractions that have the settling velocity within a set range. This settling rate is dependent upon particle shape, particle density and particle size, as well as water temperature, density and viscosity. For materials that have a constant particle density across all size fractions in question and a constant particle shape, then the Cyclosizer may produce accurate results. As stated in Austin et al. 1992, "for materials of different density there is obviously a further complication since the Cyclosizer cut sizes are smaller for higher density materials. It is the necessary to be able to split the collected fractions into the specific gravity fractions, calculate cut sizes for each component, apply the shape factor(s), and recombine the data an as equivalent cumulative screen size distribution". Particle density would then have to be calculated using the density bottle method (AS1038.21.1.1) or the density volumetric method (AS1038.21.1.2), where the required mass of sample may not be present for each density fraction collected from each cyclone.

 
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Cyclosizers do produce samples of different settling rate ranges for further analysis, but the mass of sample in each range is very limited. The starting mass for a Cyclosizer run is limited to 20 g and this is then split into six size fractions. Thus, each size fraction contains limited mass. The finest size fraction can, in theory, be collected, but in practice is not. If the finest size is collected, then the collected sample is likely to be erroneous as it may be only a few grams in perhaps 400 L of water.

The need to develop the -38 µm sieving techniques came from the desire to understand what is happening within the smallest size fractions of material especially around flotation circuits, classifying cyclones and dense medium systems.

The use of the sub 38 µm sieving method in the sizing of magnetite samples was also seen as of great benefit. The sub 38 µm sieving method allows for samples of the individual sized fractions to be analysed separately for a range of parameters such as percent magnetics, particle density, etc. as well as examination by an optical microscope.

The techniques used for completion of this test work were developed by Clean Process Technologies and are proprietary methods and equipment. Processing of samples is a time consuming exercise and is expensive, due to a high labour requirement and high equipment costs. As with all laboratory analysis, operator training and knowledge is very important to ensure accurate results are obtained. This is especially the case with these very fine sieve sizes.

**Yields of sub 38 µm material in coal flotation**

In Table 1 below are the results of a flotation plant audit. Only the data from the sub 38 µm fractions are presented and these are given as the total 38 to 0 µm fraction and also by sizing the fraction at 10 and 20 µm. The 38 to 20 µm fraction in the feed is only 15% ash (d), yet the 10 to 0 µm is 30% ash (d). These size fractions will occur in the froth product due to simple entrainment, but the ashes of the product at these sizes show that they are not only recovered to the product by entrainment, but by true flotation as well. The ashes for all three size fractions are greatly reduced in the product and greatly increased in the tailings, there is not simply entrainment occurring. If it were simple entrainment, then the ashes of all three (feed, product and tailings) would be similar. That they are so different shows that true flotation is occurring.

**Table 1 Coal quality changes in the sub 38 µm size fractions**

| Size Fraction | Feed | | Product | | Tails | | Mass Yield (ash) | Combustible Yield (ash) |
|---|---|---|---|---|---|---|---|---|
| microns | Ash | Mass | Ash | Mass | Ash | Mass | | |
| | d | d | d | d | d | d | d/d | d/d |
| | %,m/m | %, m/m | %,m/m | %, m/m | %,m/m | %, m/m | %, m/m | %, m/m |
| | | | | | | | | |
| 38 – 20 | 42.3 | 6.4 | 8.5 | 8.4 | 71.8 | 7.2 | 47 | 74 |
| 20 – 10 | 45.5 | 6.7 | 13.2 | 9.5 | 72.1 | 10.8 | 45 | 72 |
| 10 – 0 | 74.3 | 45.2 | 32.9 | 7.9 | 84.0 | 41.9 | 19 | 50 |
| | | | | | | | | |
| 38 – 0 | 67.4 | 58.4 | 17.7 | 25.8 | 80.4 | 59.9 | 21 | 52 |

**Magnetite Particle Size Distributions**

A comparison of laser sizing, fine sieving, and a Cyclosizer was undertaken upon a sample of both Superfine and Ultrafine magnetite. These samples were collected using a spear type sampler from multiple locations within a magnetite stockpile at the distribution centre. Samples were then processed using a laboratory rotary sample divider to ensure that each sub-sample was representative of the initial lot.

The S50 values for each method are shown in Table 2.

**Table 2 Comparison of the S50 values from the size analysis of both Superfine and Ultrafine magnetite samples.**

| **Superfine Magnetite** | | **Ultrafine Magnetite** | |
|---|---|---|---|
| **Sizing Method** | **S50** | **Sizing Method** | **S50** |
| | (μm) | | (μm) |
| Sieve | 19.3 | Sieve | 16.4 |
| Cyclosizer | 32.5 | Cyclosizer | 30.5 |
| Laser Sizer | 28.5 | Laser Sizer | 24.0 |

As stated in Australian Standard 4156.3-2008, laser sizing cannot be used to analyse the particle size distribution of magnetite, with sieving being the only suitable method, but this was thought to be limited to 38 μm and above.

Currently in Australia Cyclosizers are used on the ‘sub-sieve’ (sub 38 μm) material. Cyclosizers though, are very susceptible to particle density. As the new sub 38 μm sieving methods can produce 50 or more grams per size fraction, it is possible to find the particle density in each size fraction.

**Using Sub 38 μm sizing data for Partition Curves**

Sizing partition curves, especially on classifying cyclones, can be greatly improved by using sub 38 μm wet-sieving. In fact, some partition curves have been shown to be quite erroneous when the finest sieve is 38 μm.

With sizing partition curves, the “average” size is given by the geometric mean size.

$S = \sqrt{(S_1 \times S_2)}$

Where: $S_2$ = smaller sieve size

$S_1$ = larger sieve size

As “0” cannot be used in this calculation, an assumption must be made for $S_2$. A survey of coal preparation engineers in Australia found that a range of assumptions were made for $S_2$ or for S of the finest size. These are given in the table below. It is interesting in that the relevant Australian Standard (AS 4156.7- 1999) gives no assistance as to what should be assumed for $S_2$.

**Table 3: Comparison of the data obtained from Figure 1**

| **Upper Size** | **Lower Size (assumed)** | **Lower Size (fixed)** | **Mean Type** | **Plotted Value S** | **S25** | **S50** | **S75** | **Ep** |
|---|---|---|---|---|---|---|---|---|
| **μm ($S_1$)** | **μm ($S_2$)** | **μm** | | **μm** | **μm** | **μm** | **μm** | **μm** |
| 38 | 1 | 0 | Geometric | 6.2 | 5.0 | 12.0 | 26.0 | 10.5 |
| 38 | 0.1 | 0 | Geometric | 2.0 | 1.3 | 5.0 | 19.0 | 8.9 |
| 38 | 0.001 | 0 | Geometric | 0.2 | 0.1 | 0.9 | 9.5 | 4.7 |
| 38 | 0.0001 | 0 | Geometric | 0.1 | 0.0 | 0.5 | 7.0 | 3.5 |
| 38 | 0 | 0 | Arithmetic | 19.0 | 17.0 | 23.0 | 35.0 | 9.0 |
| 38 | n/a | 0 | Assumed | 10.0 | 8.0 | 15.0 | 30.0 | 11.0 |
| 38 | n/a | 0 | Assumed | 5.0 | 4.0 | 9.5 | 24.0 | 10.0 |

The first four rows of Table 3 show the effect of different assumptions for $S_2$ and the last three columns show the effect of using an arithmetic average instead of a geometric average and also assumed values for S. All seven variations were found to be used by Australian coal preparation engineers. This gives a wide variation that is simply due to the value of $S_2$ that is assumed.

Even if only 10 and 20 μm sieves are used, much of this problem is reduced as the 10 and 20 μm sieves give defined (not assumed) boundaries to much of the material in the sub 38 μm fraction.

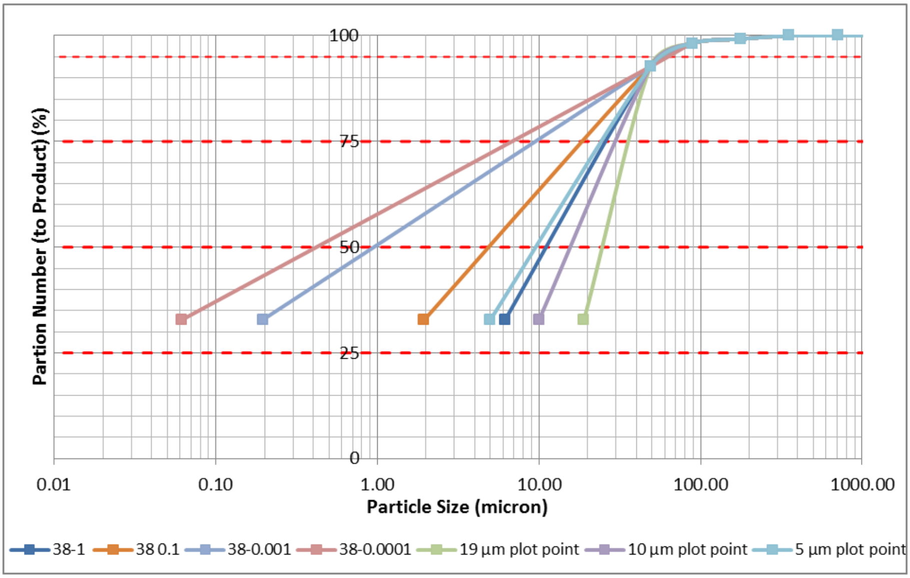


*Figure 1: Partition curves comparison for fine cut classifying cyclone by altering assumptions of the lower S2 value and set plot point values for the bottom size fraction for 38 μm as the finest sieve size.*

Table 4 below shows how the assumed $S_2$ effects the lowest plotted value and then the effect of this on S25, S50, S75 and Ep, With 10 μm as the finest sieve size, the effect of the assumed $S_2$ is greatly reduced. Only the S25 is affected, there is no affect at all on the S50 or S75 and very little on the Ep. If the 10 and 20 μm sieves were not used, then the finest plotted point would be above the S50, in this case.

**Table 4: Comparison of the data obtained from Figure 2**

| Upper Size | Lower Size (assumed) | Lower Size (fixed) | Mean Type | Plotted Value S | S25 | S50 | S75 | Ep |
|---|---|---|---|---|---|---|---|---|
| μm ($S_1$) | μm ($S_2$) | μm | | μm | μm | μm | μm | μm |
| 10 | 1 | 0 | Geometric | 3.2 | 11.0 | 45.0 | 125.0 | 57.0 |
| 10 | 0.1 | 0 | Geometric | 1.0 | 9.0 | 45.0 | 125.0 | 58.0 |
| 10 | 0.001 | 0 | Geometric | 0.1 | 6.0 | 45.0 | 125.0 | 59.5 |
| 10 | 0.0001 | 0 | Geometric | 0.0 | 5.0 | 45.0 | 125.0 | 60.0 |

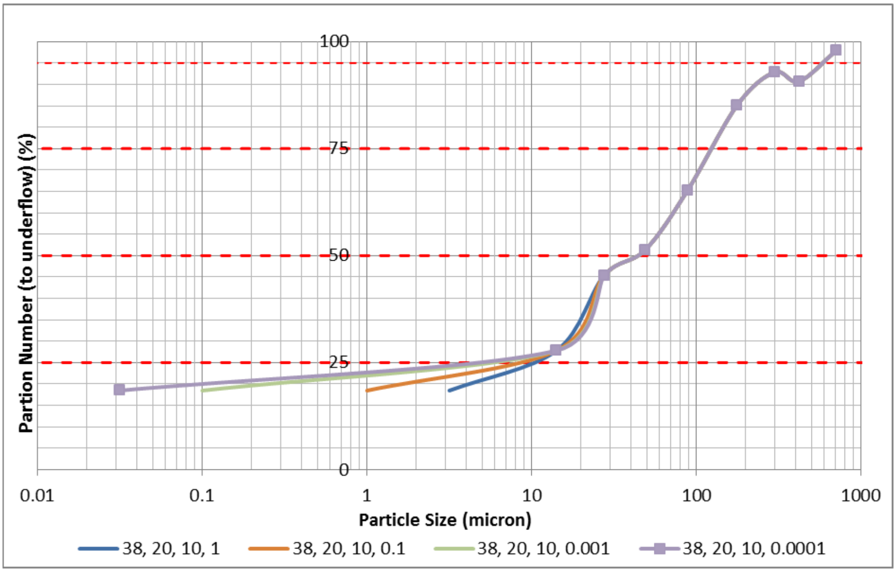


*Figure 2: Partition curves comparison for a classifying cyclone by altering assumptions of the lower S2 value for the bottom size fraction for 10 µm as the finest sieve size.*

## Conclusions

The sub 38 µm wet sieving method developed and successfully used offers significant advantages over existing analysis methods. The sieve method has been successful in sizing both coal and mineral samples and producing samples that can be used for further analysis of the discrete size fractions.

The generation of the discrete size fractions gives a greater understanding of what is occurring in the smallest sized fractions. This is evident in the samples taken from around a flotation circuit that show that there is true flotation taking place of the finest size fractions, not just entrainment of the material within the froth.

The accuracy of magnetite analysis can be improved using these methods as compared to fundamentally erroneous laboratory methods currently used, such as Cyclosizing and laser sizing. Discrete size fractions can be collected for further analysis, which may include particle density or percent magnetics. This is especially important on samples collected from within a CHPP's dense medium circuit.

The use of the fine sizing in the evaluation of fine cut classifying cyclone data is especially interesting in that it allows for greater definition of the bottom size fraction of the material and can increase the accuracy of the true Ep calculation as by having more true plot points on the charts reduces the effect of assumptions as to where the lowest size fraction plot point is plotted.

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# Sulcis Coal Water Jet Assisted Comminution

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**Abstract**

This study has sought to verify the effectiveness of high-speed water jets on solid minerals grinding. To this end, an innovative gyratory crusher called HERA (High Energy Reduction Apparatus) has been designed and employed.
The experiment has been carried out on samples of sub-bituminous coal extracted from the Sulcis coal basin (Sardinia, Italy). The experiment has aimed to examine and evaluate the effects of high-speed water jets on the milled product. These grinding tests have returned carbon suspensions in water, which could be used to power fluidized bed combustors.
The parameters used to evaluate the adequacy of the assisted water jet comminution are: jet mill pressure, nozzles size, water flow rate, jets hydraulic power and specific energy consumption. The effects of preliminary water imbibition of coal samples have also been considered. The results have shown that the synergic combination of hydraulic and mechanical energy can have positive effects on grinding.

**Key Words**: **Comminution, Gyratory Crusher, Water Jets, Coal Water, Sulcis Coal, Coal Imbibition, Water Jet Assisted Comminution**

## 1. Introduction

The main objective of the study is to verify the effectiveness of high-speed water jets on solid minerals grinding. The experiment was carried out on samples of sub-bituminous coal extracted from the Sulcis coal basin (Sardinia, Italy). The laboratory tests aimed to analyse and evaluate the effects of high-speed water jets on the milled product. The machine specifically designed and used to this end is an innovative gyratory crusher called HERA (High Energy Reduction Apparatus). This crusher was installed in the laboratory of the Department of Civil-Environment Engineering and Architecture at the University of Cagliari (Italy) to prepare carbon suspensions in water, which could be used to power fluidized bed combustors. The experiments were carried out using two different top particle size feeds at -20 mm and -10 mm respectively. The parameters used to evaluate the adequacy of the assisted water jet comminution are jet mill pressure, nozzles size, water flow rate, jets hydraulic power and specific energy consumption. The experiments involved both dry milling (i.e. no water jets were applied) and wet milling procedures (i.e. water jets were applied). The comparison of these two sets of experiments has highlighted the effects of each parameter mentioned above as well as the relevance of preliminary coal preparation (i.e. grinding with or without water imbibition of the feed samples).

 
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**2. Characteristics of coal-water slurry**

There is currently a growing necessity to develop alternatives to fuel that can be both technically and economically viable. Therefore, creating new coal-water slurries has become a central research topic.
The particle size for coal-water slurries should be within the 150-500 μm range while the mean particle size should be between 10 and 20 μm. Generally, more than 80% of these particles should have a diameter below 74 μm whereas no more than 10% may have the finest particle size.
In order to obtain stable coal content in coal-water slurry, the coal should be grinded and this also guarantees a bimodal particle size composition (Boylu, 2004).
Nonetheless, if coal stability is a secondary requirement, a suspension made of coarse grain size can be employed. This is as an alternative to the particle size conventionally used in coal-water slurries, which can also ensure more fluidity and reduced grinding costs.

**3. General characteristics of the water jet technology**

Water jet technology is based on high-pressure and high-speed streams of water, which are ejected from a nozzle made of extremely hard materials. The nozzle diameter should usually be in the range of 0.1 mm and 0.6 mm. Water pressure may vary between 200 MPa and 400 MPa, and ejecting speed should be between 500 m/s and 900 m/s (Ciccu, 1996), (Mazurkiewicz, 1993), (Ciccu, 1993), (Bortolussi, 1993).

**4. Water jet assisted comminution and the HERA gyratory crusher**

Mineral comminution is often a costly process which also requires high levels of energy, especially when dealing with fine sizes (Austin, 1979).
The desegregating process may induce tensile stress in the manipulated materials. Minerals having permeable porosity usually oppose resistance to such stress (Griffith, 1920). Hence, a viable solution to this problem can be combining in a synergic way both mechanical energy (which in usually employed in traditional grinding machines) and hydraulic energy that is transferred via water jets, as done by the HERA gyratory crusher (Ciccu, 1998). The main components of HERA are reported in Figure 4.1.

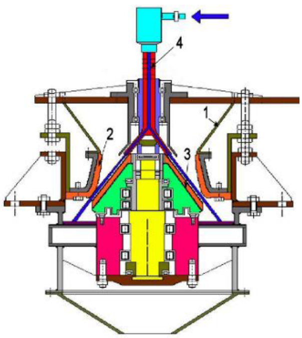


Figure 4.1 Schema of the water jet gyratory crusher (HERA)

(1) loading cone; (2) concave; (3) mantle; (4) rotating lance with two nozzles

Each nozzle is placed at the bottom end of the lance via a Y shaped reverted joint. The nozzles are firmly tightened to the mantle and rotate at the same speed. The nozzles are placed on a vertical plate that is perpendicular to the plate that covers the diameter that links the maximum and minimum regulation points between the concave and mantle. The acceptable $D_{max.}$ for feeding is approximately 40 mm. This crusher also has a regulating system that can provide different product sizes.

## 5. Materials and methods

The experiments were carried out on a manually selected sample of coal and classified as "long-flame sub-bituminous coal", in accordance with the American ASTM D 338 specifications. Product sizes could vary from 5 to 10 cm, although some pieces could be larger than 15 cm.
The coal was crushed using a single toggle jaw crusher in a closed circuit with a squared mesh control screen of 20 mm. The $d_{50}$ and $d_{80}$ values for particle size are respectively 5.7 and 14.1 mm.
The material was divided into samples of an approximate weight of 3 kg each.
The $d_{50}$ and $d_{80}$ values of the crushed coal having a 10 mm top particle size are respectively 3.2 and 7.3 mm.
Two series of experiments were carried out on the -20 mm samples. The first series of tests were based on a dry milling procedure with a varying feed rate of the machine. For the second series, water jets were applied but the same varying feed rate of the crusher was retained. This aimed to verify the relevance of using hydraulic energy during the milling process. The parameters considered during the wet crushing procedure were: feed rate, water pressure applied (from 50 to 180 MPa), nozzle size (0.8mm and 0.5mm) and flow rate of water jets.
During each experiment, the treatment capacity was verified by measuring the outlet flow rate.
The following stage of the experiment aimed to verify the possibility of improving the system performance by saturating the coal with water via imbibition.
These experiments were also carried out on samples of smaller top particle size (-10 mm).
The crusher settings applied were: 1.00 mm eccentricity; 1.24 mm travel; 1.00-2.07 mm outlet opening.
These settings sought to limit to approximately 2 mm the maximum size of the crushed coal (Cau, 2009).

## 6. Results and discussion

### 6.1. Dry comminution

All experiments returned an outlet time (around 70s) greater than the mill feed time (approximately three times longer for the 3 kg samples)
In order to verify whether the particle size at feeding influenced the coal grinding process, more experiments on -10mm top particle size samples were carried out. These experiments did not return any significant differences in terms of product granulometry, but the outlet rate was doubled.
When eccentricity and rotation speed are equal, the outlet rate (i.e. treatment capacity) mainly depends on the mill outlet opening size.

### 6.2. Assisted water jet comminution

In order to verify the real treatment capacity, each experiment was carried out by measuring both the feed and outlet time of the crusher.
Figures 6.2.1. and 6.2.2 below show a comparative particle size analysis respectively for -20 mm and -10 mm feed samples, which were comminuted by means of dry and water-jet assisted procedures.
During both dry and water-jet assisted comminution tests, the same water pressure was applied. All tests have demonstrated that feed rate has not a significant effect over the outlet rate of solid material. Yet, it should be noted that during the water-jet assisted comminution, the flow rate of the material was higher that during dry milling.
When the employed hydraulic power was set at 14 kW (1WJ test), the treatment capacity (obtained dividing the weight of each sample by its release time after comminution) was equal to 5.4 kg/min.

This means that it was approximately twice as high as the capacity detected during dry comminution. The production of fine particles was proportionally higher than what was obtained during dry milling, due to the action of the water jets.

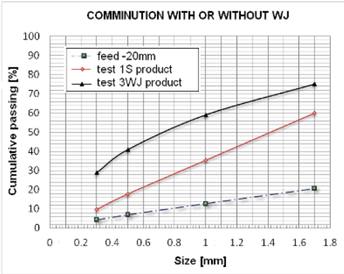


Figure 6.2.1 0.8mm nozzle diameter

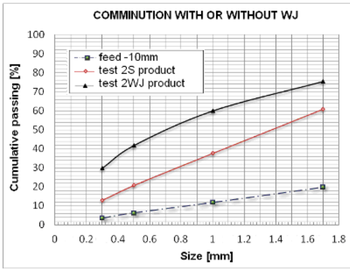


Figure 6.2.2 0.8mm nozzle diameter

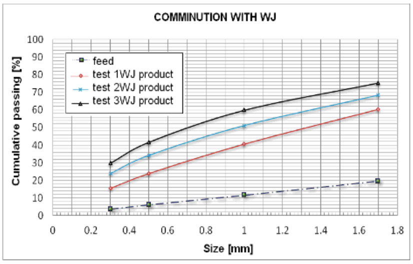


Figure 6.2.3 Effects of pressure.
0.8mm nozzle diameter. Feed <20mm

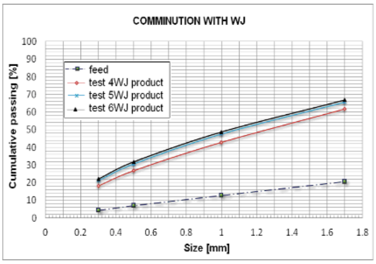


Figure 6.2.4 Effects of pressure.
0.5mm nozzle diameter. Feed <20mm

When higher water pressure is applied (cf. 2WJ test with 40 kW jet output), the treatment capacity becomes 7.3 kg/ min (2.8 times higher than during dry comminution). Again, the production of fine particles is proportionally higher due to the higher water pressure applied, which continues to increase accordingly (cf. 3JW test with 74 kW jet output). Although less significantly, the treatment capacity also increases.

It should be noted that both tests 1WJ (0.8 mm nozzles diameter) and 4WJ (0.5 mm nozzles diameter) have been carried out by applying a similar jet output (approx. 15 kW), thus resulting in similar products in terms of particle size. Similarly, tests 2WJ and 6WJ use water pressure of 39 kW and obtain approximately the same results.

Consequently, it can be said that the water pressure for nozzle diameters of 0.5mm results in a treatment capacity that is lower than when the nozzle diameter is 0.8mm. This is due to the higher water jet rate, which pushes the material down into the comminution chamber.

Again, using 0.5 mm diameter nozzles returned an increased treatment capacity.

### 6.3 Energy consumption

Specific energy consumption during water jet comminution in relation to treatment capacity, shown that specific energy consumption almost proportionally increases according to pressure. In contrast, when the pressure is the same, specific energy consumption increases according to the nozzle diameter, even if this means an increased treatment capacity. If nozzle diameters of 0.8mm are used during water-jet comminution, the particle size of the feed material does not significantly affect energy consumption. Decreases are instead detected using nozzle diameters of 0.5mm and -10 mm feed. Besides hydraulic power consumption, the consumption of mechanical power should also be taken into account.

## 7. Influence of coal imbibitions with water

During the course of the test series, the cutting action of the water jet on rocks revealed that when a rock is saturated by the water, it disaggregates more quickly than when a rock is dry. In addition, this different disaggregating speed tends to reduce when high pressure is applied.

Therefore a series of experiments were carried out by feeding water-saturated coal into the crusher.

In order to compare the results obtained during the two different comminution procedures (i.e. with water-saturated and dry grinded coal with particles below 10 mm), an additional test on a “dry” sample (labelled “Test 9WJ”) was carried out. The same operational conditions used during the test with water-saturated coal (labelled “Test 8WJ9) were undertaken. The corresponding particle size curves are shown in Figure 7.1. and 7.2.

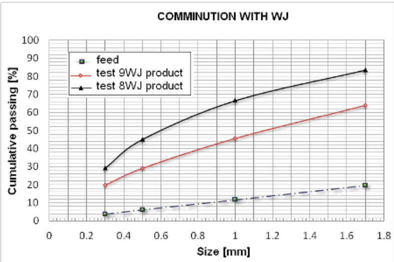


Figure 7.1. Particle size <10mm. Pressure 100MPa. 0.5mm nozzle

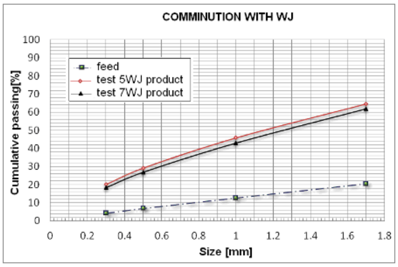


Figure 7.2. Particle size <20mm. Pressure 100MPa. 0.5mm nozzle

The coal saturation returned marked effects on the analysed crushed sample of -10mm particle size. Conversely, water saturation in the sample of -20mm particles size produced less significant results.

The positive effects of the imbibition process over coal grinding for finer material (-10 mm) can be explained as follows. Every single coal grain upon which a water jet was applied has its specific porosity, which mainly features planar discontinuity (i.e. microcracks and cleavages). These empty spaces are filled with water via imbibition.

When the water jet hits the water-saturated area, it applies a strong pressure to the liquid already filling the gaps, thus affecting the discontinuity walls. Consequently, the grain is broken more effectively than when the gaps are air filled (Ciccu, 1998).

When grains of different size are immersed in water, their level of saturation also differs in relation to their total volume. If the saturation process is limited to the external layer of each grain, the portion of volume that becomes saturated will be smaller in case of bigger particles than in the portion of finer grains. This means that the effect of the water jet pressure will be more effective on finer water-saturated grains.

The advantages offered by the preliminary imbibition could be further enhanced by means of other actions aimed at favouring water saturation, by reducing the time needed for this process and/or increasing the quantity of water that each grain can absorb so as to ensure a complete saturation of coal porosity. These actions could include thermal treatments that can increase the gaps within each coal grain.

Also, vacuum gases could be eliminated and total immersion of the coal in pressurised water could be sought.

## 8. Potentials of the HERA technology

The particle size of the product crushed by HERA (which is of 1 mm) satisfactorily corresponds to that needed to feed a boiler system based on a fluidized bed (which is a water suspension type). This boiler system requires well shaped solid particle dimensions, which are of a finer class. This also implies a limited presence of ultra-fine particles (the average particle size is approximately 500μm and its maximum size is below 2mm) (Cau, 2009).

Even when considering advanced systems for the production of electrical and thermal energy based on coal gasification, the coal-water slurry obtained by means of the HERA gyratory crusher could be extremely effective (e.g. for the production of syngas in a fluidized bed reactor) (Cau, 2009), (Calabrò, 2009).

## 9. Conclusions

The water jet assisted comminution process discussed here demonstrated to be more effective than dry milling since the former returned better levels of fine particle crushed material. The particle size dispersion degree was also significantly lower than in case of dry milling. Better results are obtained if increasing water pressure is applied.

Using high pressure water jets increases the treatment capacity, thus significantly reducing managing costs. However, the energy consumption to generate water jets and the relative reduction of specific mechanical energy should be taken into account.

Finally, it should be noted that the best results were obtained using a preliminary water-saturated material via imbibition. In this light, further studies on the grinding of saturated materials are certainly needed.

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# Optimisation of operating costs due to the implementation of innovative technologies in coal preparation

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**Abstract**

The screening and drying of coal are very important steps for enrichment of the row material, substantially raising its quality and making it possible to coal companies to enter new markets and achieve higher market prices for the top product.
Depending on the required cut points, initial moisture, ash and volatiles content, coal can be a very challenging material to screen and thermally enrich. The green field projects offer wider opportunities regarding machine size and design as compared to plant alterations, where the space is often limited due to the existing constructions and foundations. Nevertheless, tailor made solutions for process optimization are needed in both cases.
This article describes the innovative screening and fluid bed drying equipment, developed and produced by Binder+Co, an Austrian specialist for bulk goods processing, with the focus on optimisation of operating costs for plant operators and thus achieving sustainable success.

**Key words:** screening, BIVITEC, fluid bed drying, DRYON, optimisation, operating costs

**Inroduction**

Binder+Co is an internationally successful Austrian specialist for machines and overall systems for screening, drying and sorting of any kind of bulk solids. In screening technology the company is seen as the global market leader, it's machines are used not only in the coal industry, but also for the processing of industrial minerals, ores, salt and chemical fertilizers.

Binder+Co machines effectively deal with difficult to process materials and special requirements of the customers. Regarding overall cost-effectiveness of a processing plant, Binder+Co focuses on development, implementation and integration of innovative screening systems.

**Importance of tailored solutions for coal processing**

In recent years, processing of different types of coal has undergone enormous innovative development. Not to process your coal and thus not to improve its quality means to lose at least the export market, because the requirements for product quality are constantly increasing.

As the current coal price remains historically low and reliable forecasts on the matter are difficult to derive, plant operators demand a decrease of investment and operating costs, which becomes a real challenge for plant and equipment constructors.



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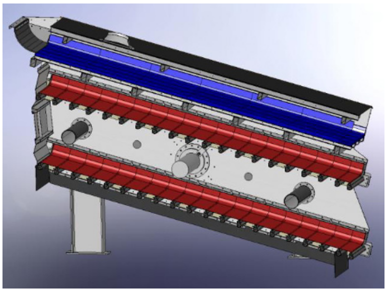
Considering plant alterations and modernisation, the existing constructions and available space play often a crucial role in choosing the equipment, so tailored solutions both for process and engineering are required.

To offer optimum solutions, the equipment producers have to implement continuous advancement and invest in research and new product development.

**Screening technology and BIVITEC-screening effect**

As a specialist for the **SCREENING technology**, we at Binder+Co experience an increasing demand for solutions that can deal perfectly with material that is difficult to screen, such as coal and coal intermediates with commensurate surface moisture.

The special screening technology BIVITEC, developed by Binder+Co, is used to screen material, when other conventional screening technologies fail (e.g. because of plugging). Currently single-, double- and triple-deck (fig. 1), as well as 2 ½- deck machines are in operation, depending on the feed material and needed fractions.



(fig. 1) Bivitec KRL/DDS 1900x6m with 2 flip flop decks and a protective deck

The screening machine Bivitec with punching mats (fig. 2) has proven its effectiveness for processing of difficult to screen coals at challenging cut points like 3 mm, 5 mm and higher.

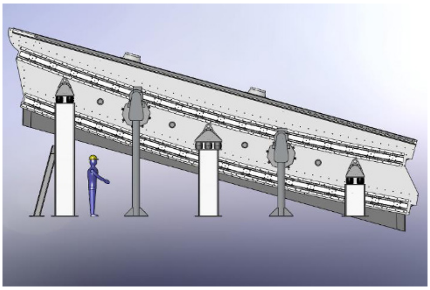
(fig. 2) Bivitec effect on the screening mats

The function itself is based that a second mass is added into a standard screen. Those two masses are connected with rubber blocks. The result is an oscillated vibration system with accelerations up to 50g.

Thus the screen has 10 times more material accelerations compared to conventional screening technology. The screenbox itself is subjected to only very minor acceleration values.

The swing characteristic can be set separately for each screen deck, or changed retroactively, if needed.

Even when dealing with difficult-to screen-materials, the trend goes to bigger machines (fig. 3): our latest BIVITEC screens have a screening surface of up to 36 m2 per screen deck, as well as to separation of the finest fines (0-3 mm, 0-6 mm) from the moist coal in order to reduce the amount of the feed for the downstream equipment.



(fig. 3) Double-deck Bivitec KRL/DD 3000x12 m – R60 “banana”

Using “conventional” screening machines, e. g. circular or linear vibrating screens, resonant screens, often encourages the formation of clogging grains and leads to blinded screen cloth. In such cases, the separation cut falls within a very short time to an unacceptable level in terms of screened product quality, and sometimes screening is no longer possible and the line has to be shut down for cleaning. This results in additional, often non expected operating costs and losses of profit.

**Example of a solution offered by Binder+Co to a client in China:**

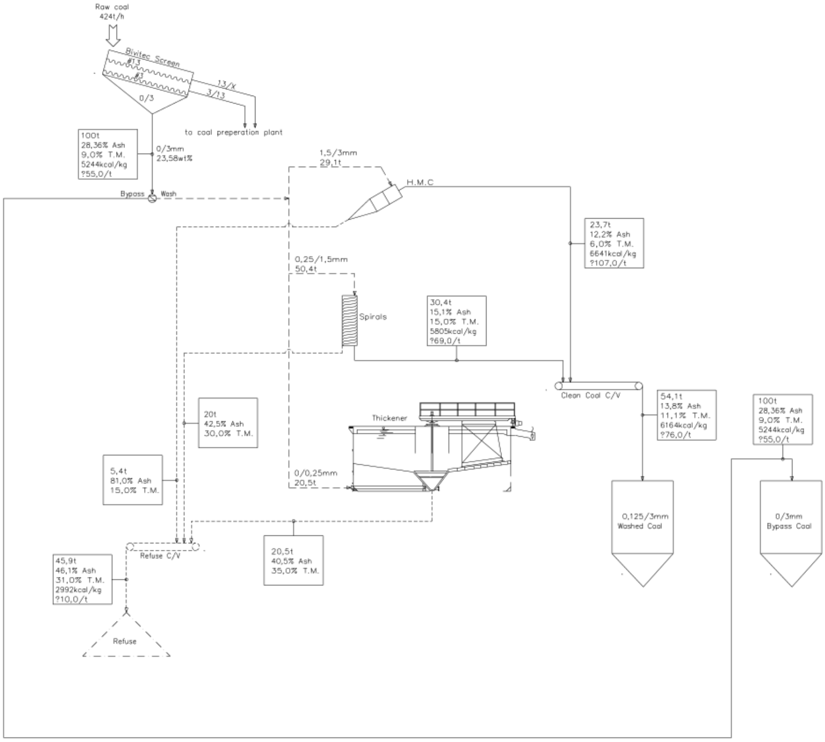
The BIVITEC is applied to screen the ROM coal at approx. 2 or 3 mm. The oversize is fed to the Coal Handling and Preparation Plant, the fine material is passed by of the CHPP. (fig. 4)

The benefits beside the economical improvements are:

- Higher throughput of the plant
- Water savings (approx. 106.000 m³/year)
- Energy savings

According to a project in China we can summarize the following facts:

| | |
|---|---|
| Feed of raw coal per year: | 4.000.000 t/a |
| Fine material “0/3” mm: | 943.000 t/a |
| Revenue for the bypassed “0/3” mm per year: | 51.875.000 €/a |
| Revenue for the washed “0/3” mm per year: | 42.875.000 €/a |
| Benefit if the product “0/3” mm is bypassed per year: | 9.000.000 €/a |

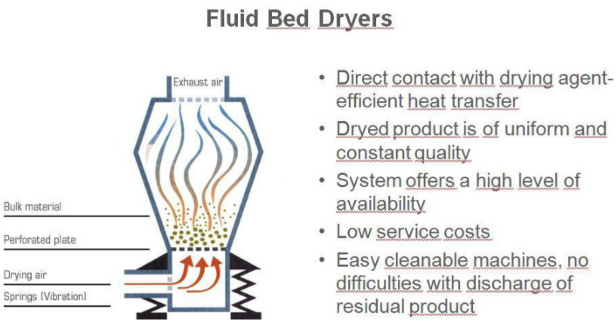



(fig.4) Optimization of a coal preparation plant through bypass of 100 t/h

**Optimising drying costs**

As a specialist for the **FLUID BED DRYING technology**, we have early recognized that the drying equipment should be cost-effective with a high operational availability, enabling the plant operators to get the product in a constant quality.

Compared to other steps or ways of material processing, the thermal treatment is more energy consuming, thus making the issue of energy efficiency one of the central for choosing the drying equipment. The benefits of fluid bed dryers are: gentle drying without damaging the particles, less wear, high accessibility of the parts for easy maintenance, and adjustability of the drying process (fig. 5).




(fig. 5) Schematic representation of drying in a fluidized bed

Flued bed machines are reliable and require practically no maintenance. Since the dried material for the most time does not touch the grid, and as there are no mechanical devices in direct touch with the dried material, the wear of such machines is minimal. The side walls have additional wear protection.ffe

**Job story**

For an ambitious coal enrichment project in Russia, we developed a solution with our DRYON-system applied for drying and cooling energy coal and coking coal as well as middling coal at average feed moisture of 14% up to max. 22%.The yearly capacity amounts 6,5-8,6 Mio.t/year coal to be dried.

The main benefits beside energy savings and reliability are transportation cost savings (export coal) as well as revenue by increased caloric value (increase 500-1000 kcal/kg).
Additionally, we see the trend in the development of drying and cooling solutions for ignitable and explosive bulk materials. Our new technology enables the monitoring of the drying process in terms of $O_2$-content, and if necessary, reduction of it by means of steam inerting.

The initial summarizing statement is:

These two processes, screening and drying, and especially their combination with innovative technology enable the optimisation of operating costs and the careful and economic use of natural resources, combined with increased efficiency and profits for the plant operators.

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# Part X
# Flotation

# IMPROVEMENT OF THE EFFICIENCY OF FINE COAL SLIMES FLOTATION

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**Abstract:** The problems of selective flotation of fine slimes have been analyzed. Different technical solutions aimed at the improvement of the efficiency of flotation of coal containing fine slimes have been considered. As a criterion of the efficiency of separation results in flotation the index of maximum flotation ability has been suggested, which is determined by the method of flotation fractionation as the maximum possible yield of concentrate with a specified ash content. The effect of the degree of fineness of slimes and the influence of interaction of flotation feed grain sizes on the flotation results have been investigated. It has been demonstrated that the maximum possible yield of concentrate with a specified ash content strongly depends on the content of ash units in the less than – 50 μm fraction of the flotation feed that allows for forecasting of the optimal indices of separation by flotation. The indices of maximum flotation ability under conditions of one stage flotation are very difficult to achieve in case of higher content of ash units of – 50 μm size in the flotation feed. The examples have been provided of the optimization of flotation results (obtaining of flotation separation indices close to those for the maximum flotation ability) by selection of reagent regimes (type and consumption of reagents and time of flotation), using of flocculation-flotation method; using of complex flotation flowsheets (rougher, scavenger and cleaner flotation); regulation of aerohydrodynamic parameters in flotation cells.

**Keywords:** flotation, fine coal slimes, criterion of efficiency of flotation ability, optimization of flotation results, flotation flowsheets, flocculation-flotation method, fly-ash, multi sectional columns

**1 Background**

Fine coal slimes (size under 0.2 mm) are very difficult to clean. Today the main method of fine coal slimes separation is flotation.

The results of flotation depend on particle size and physico-chemical characteristics of particles. For instance, coal rank, fines content (less than 50 μm) and their ash content have a significant effect on the selectivity of the process. Flotation properties of ultrafine slimes also depend on their nature, that is whether they are primary slimes in the run-of-mine coal or they are formed as a result of processing of coarse and fine sizes of coal [1, 2]. Small size of particles determines their significant specific surface and high adsorption capacity.

The presence of fine slimes in the slurry causes the following technical phenomena [1, 2]:

- pollution of froth product with fine fractions of waste rock;
- worsening of coarser grains flotation and deterioration of tailings quality;
- higher reagents consumption;
- lower flotation rate;

To enhance the efficiency of separation of coals containing fine slimes the following approaches are used:

- selection of optimal reagent regimes (type of reagents and their consumption, flotation time, stage addition of reagents);
- special conditioning regimes (time and intensity of mixing, slurry density during its treatment with the reagents);
- using of appropriate flotation cells allowing for variation of aerodynamic conditions in flotation;
- using of special processes (e.g., flocculation-flotation method, etc.)

 
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The suitability of one or another method may be determined based on the experimental study of fine slimes behavior in flotation taking into account the requirements to the separation indices.

In the coal supplied for cleaning the percentage of slimes including so-called secondary slimes varies from 10 to 35% and their ash content is about 20-40%. The percentage of fine particles of less than 50 μm can vary from 40 to 70%.

The optimization of flotation conditions shall be based on the criteria allowing to determine the optimal separation indices.

**2 The criteria of floatability efficiency**

The indices of maximum flotation ability determined by the method of flotation fractionation can be used as a criterion of flotation ability efficiency. The results of investigations are presented in the form of Mayer curves, which link the yield and the content of ash units in the flotation products. They characterize the distribution of materials by flotation ability and make it possible to evaluate the maximum recovery of concentrate with a specified ash content and to determine ash content of tailings produced [2].

The results of fractionation by flotation do not depend on flotation time, reagents used and their consumption. Under real-life-conditions achieving the results close to the maximum flotation indices will mean achieving the optimum. The deviation of flotation performance from maximum values depends on a number of factors, and in accordance with the analysis of a series of investigations it depends on the percentage and ash content of fine particles of less than 50 μm.

**3 The effect of the degree of fineness of slime and the interaction of grain sizes on the flotation results**

The investigation has been carried out with the samples of actual flotation feed of coal preparation plant «Nerungrinskaya», coals of rank K (coking) from Nerungrinskoye deposit, South Yakutia, and central coal preparation plant «Pechorskaya», coal rank Zh (fat) from Pechora basin. In accordance with the particle-size analysis of representative quantity of samples the content of particles of less than 50 μm was found to be 60-75%. At that, ash content of over 50 μm size fraction was 6-8%, that is, it was virtually a concentrate, and ash content of over 50 μm size varied in the range of 20 - 40%.

Consider the results of the investigations of flotation ability of slimes with different content of fines through the example of coal slimes of the central coal preparation plant «Pechorskaya». These slimes are artificial mixtures containing in a different ratios particles of less than 50 μm with ash content of 39.8% and particles over 50 μm with ash content of 5.8%. The slime mixtures investigated have different initial ash content and therefore have different potential for recovery of concentrate of a specified grade [3, 4].

It should be noted that ash content of feed was as follows: 5.8% at 100% of > 50 μm size; 13.4, 22.5 and 39.8% at<50 μm size content of 25, 50 75 and 100%, respectively.

The flotation kinetics indices for artificial mixtures of slimes from the central coal preparation plant «Pechorskaya» obtained at the identical reagent regimes and flotation conditions for the mixtures containing fines < 50 μm (0; 25, 50 and 75%) have shown that during 1 minute of flotation of 100% of >50 μm size it is possible to recover 97.4% of concentrate with A=4.7% and produce tailings with ash content of 40% . It seems to be impossible to recover by flotation a concentrate with A=8.5 % from mixtures containing 75 and 100% of <50 μm particles. The yield of concentrate with ash content of 8.5% for the mixtures containing 25, 50 and 75% of <50 μm particles will be 90, 68 and 20%, respectively.

In accordance with the maximum flotation ability curves the yield of concentrate with ash content of 8.5% can be 41, 63, 75 and 92% for the mixtures containing 100, 75, 50 and 25% of <50 μm particles, respectively. As one can see, the reagent regime selected is virtually optimal for the flotation feed containing 25% of <50 μm particles, it is close to optimal at 50% <50 μm particles and is essentially different for the mixtures containing 75 and 100 % of <50 μm particles.

The comparison of the values of maximum possible yield of concentrate with ash content A=8.5% as a function of the quantity of ash units of <50 μm size in actual slimes of flotation feed of coal preparation plants «Pechorskaya» and «Nerungrinskaya» and the values obtained during flotation of artificial mixtures has demonstrated a well-defined dependence of maximum possible yield of concentrate with a specified ash content on the percentage of ash units of less than 50 μm size in the flotation feed. The accuracy of the approximation (taking into account overlapping of indices for different coals) was of the order of $R^2$=0.99) (Figure 1).

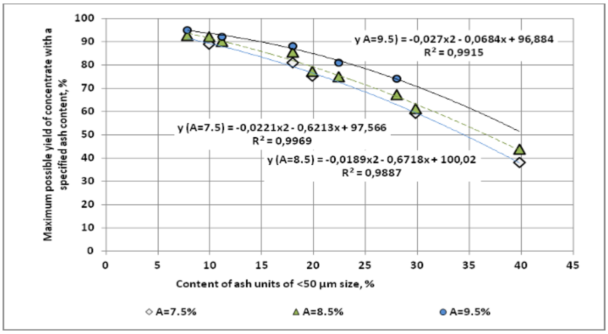



Figure 1 – The effect of ash units of <50 μm size on the indices of maximum possible yield of concentrate with ash content A=7.5; A=8.5 and A=9.5%

The separation indices obtained under laboratory conditions in flotation with different reagent regimes and different flotation time have been plotted on the Mayer curves obtained by the method of fractionation by flotation, and the limits of the deviation of the concentrate yield in ordinary flotation from the maximum possible value with a specified ash content (A=8.5%) in fractionation by flotation of slimes from the central coal preparation plant «Pechorskaya» have been determined (Figure 2).

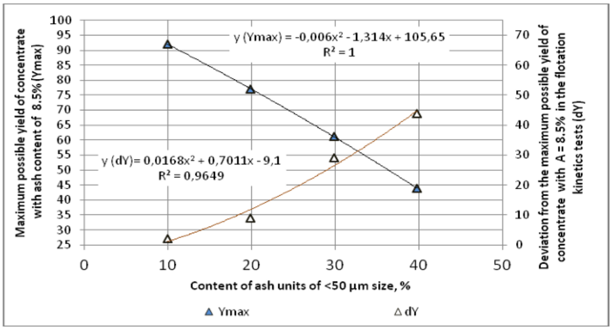



Figure 2 - The effect of the percentage and ash content of <50 μm size particles on the flotation performance (slimes from the central coal preparation plant «Pechorskaya»).

$Y_{max}$ – maximum possible yield of concentrate with ash content of 8.5% based on fractionation by flotation curves; dY – loss of concentrate yield with ash content of 8.5% in ordinary flotation under optimal conditions as compared with the maximum possible value

It has been demonstrated that the presence of more than 15% of ash units in less than 50 μm size fraction has a negative effect on the achievement of optimal results of separation by flotation; the deviation of concentrate yield with a specified ash content (A=7.5% for «Pechorskaya» plant) from the maximum values (in accordance with the flotation ability curve) is more than 10%.

**4 The examples of optimization of results of separation by flotation in order to bring them closer to maximum possible values**

It is possible to come close to the optimum indices by the selection of reagent regime (type, consumption and method of adding the reagents), the optimization of flotation time, using of flocculation-flotation method, application of complex flotation circuits, regulation of aerodynamic parameters in the cells of flotation machines [5-11].

Consider some of the above mentioned options.

1) Optimization of reagent regime [6, 7, 8, 9]. Different reagent options have been studied on the flotation of slimes of the central coal preparation plant « Pechorskaya» where flotation feed content in ash units is <50 μm =10. Different reagents varieties have been used: CATGOL – distillation residues of 2-ethylhexanol production, M-800 and M-802 – derivatives of higher aliphatic alcohol on the basis of methylisobutylcarbinol – both individually and combined with fuel oil (FO) as a collector. The consumption of selected reagents and time of flotation have been variable parameters of the process.

Relative efficiency has been determined in comparison with the results of fractionation by flotation using the following formula [3].

$$E_0 = \frac{\gamma_{кф}\left(A_{пит} - A^{ф}{}_{к}\right)}{\gamma_{кт}\left(A_{пит} - A^{т}{}_{к}\right)}; \quad (1)$$

where $\gamma_{кф}$ – actual concentrate yield; $A_{кф}$ – actual ash content of concentrate; $\gamma_{кт}$ – theoretical concentrate yield based on the maximum flotation ability curve at ash content of $A= A_{кф}$; $A_{кт}$- theoretical ash content which means ash content based on the maximum flotation ability curve at the concentrate yield of $\gamma_{к}=\gamma_{кф}$.

For a specified ash content of 7.5% the maximum efficiency is 89.5% and corresponds to the combination of FO+M-802 at a consumption of 1.5 kg/t FO and 200 g/t M-802 and flotation time of 3 min. Under these conditions the concentrate yield will be 87% with the maximum possible yield being 89%. The deviation from the maximum possible yield is 2% abs. (Figure 3).

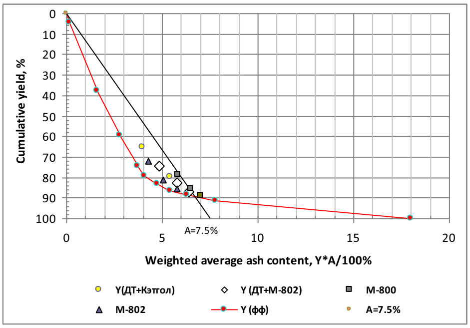



Figure 3 – Comparative indices of separation by flotation of slime from «Pechorskaya» plant (content of ash units of <50 μm size in the feed =18%)

2) Implementation of flotation circuits using scavenger and recleaner operations in ordinary flotation cells involves additional capital and operating costs. Using of multi-section columns allows for the

implementation of ramified flotation circuits in one apparatus having sections with different aerodynamic parameters [4, 9, 10].

As an example can be cited production of pure mineral part with coal particles content (LOI) less than 5% from fly-ash of power plants by recovery of carbon-containing particles by flotation (Figure 4).

Multisection columns make it possible to implement in one apparatus the flowsheets including the rougher, scavenger and recleaner operations, to use countercurrent and forward pulp and air supply in different sections of the column to obtain optimal results [4, 8, 9]. The flotation results obtained with this apparatus for fly-ash from «Novochekasskaya» GRES power plant are presented in the form of a table.

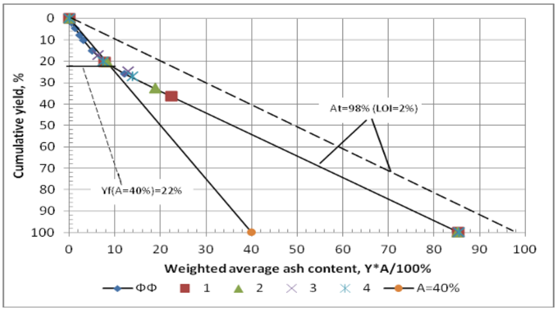
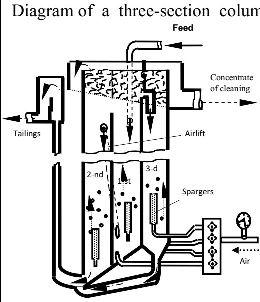



Figure 4 – Indices of fly-ash separation by flotation at Novocherkasskaya GRES power plant (Rostov region, coal rank A). ΦΦ – fractionation by flotation; 1,2,3,4 – flotation tests in a multi-section column

Flotation flowsheet including a rougher and a scavenger flotation stages makes it possible to obtain a mineral part with LOI less than 5%. Ash content of coal concentrate is slightly more than 50%. It is possible to obtain coal concentrate with ash content up to 40% when a three-stage column is used.

Table. Optimum indices of fly-ash separation by flotation

| Operation | Concentrate | | Tailings | | Feed | Diagram of a three-section column |
|---|---|---|---|---|---|---|
| | Yield, % | Ash content, % | Yield, % | Ash content, % | Ash content, % | |
| Rougher and scavenger flotation | 27.2 | 51.1 | 72.8 | 98.4 | 85.5 |  |
| Cleaning of concentrate | 20.1 | 36.8 | 79,9 | 97.8 | 85.5 | |
| $Q_S/Q_L$=200g/l; V=1.4 cm/s; $Q_p/Q_a$ =0.6 (in all sections of column cell); intensity of mixing with reagents prior to flotation for n=1800 rpm | | | | | | |
| Consumption of agents: rougher flotation - kerosene – 1400 g/t; MIBC-350 g/t; scavenging flotation - kerosene –1400 g/t; MIBC – 250 g/t; Concentrate cleaning– kerosene – 700 g/t; MIBC – 250 g/t; | | | | | | |

3) Using of flocculation-flotation method has been considered through the example of recovery of ultra pure concentrates in a flotation circuit with several recleaner stages [4, 9, 11, 12]. Figure 5 shows a curve of maximum flotation ability of 0-100 µm coal slime and the results of flotation using a

flocculation-flotation method with addition of sodium hexametaphosphate as a dispersing agent and latex BF30 as a flocculant.

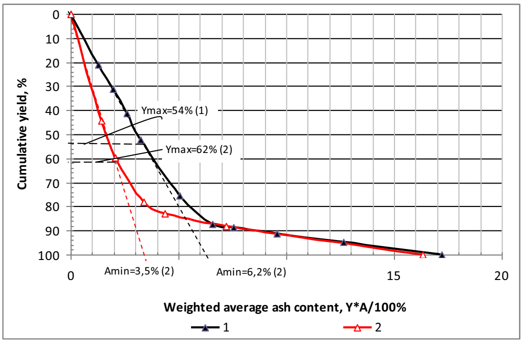



Figure 5 – Comparative evaluation of flotation ability of slimes in flotation-flocculation separation process: the curve of maximum flotation ability of 0-100 μm slimes(1); flotation ability of 0-100 μm slimes using flocculation-flotation method with recleaning stages (2).

The results obtained are indicative of the possibility to improve selectivity of separation by flocculation of fine coal particles and to obtain a concentrate with 3.5% ash content from slimes for which a minimum ash content of 6% can only be obtained in ordinary flotation.

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# Efficient selection of flocculants - the key to success

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Annotation

The article is devoted to the actual problem of return water clarification without dumping sludge into external slurry ponds. In this article the author analyzes the main factors leading to higher solids content in the return water of water-slurry circuits at coal preparation plants as well as determines the main indicators characterizing slurry sedimentation process at CPPs of LLC "SUEK". Besides a coal deposit is of great importance. The paper presents different ways of feeding flocculants, which serve to satisfactory performance and requirements of return water quality. The paper describes the main factors that affect the speed and efficiency of the sedimentation process. The right choice of a flocculant and its consumption allows for maximum reduction of water flow from additional sources that will prevent the discharge of waste water outside the factory. The study of this problem proves that the effective selection of flocculants can solve many production problems of CCPs - to reduce the amount and wet of the produced cake, as well as to decrease the amount of flocculants, thereby reducing production costs.

**Flocculant, consumption, dewatering, clarifying, return water, solid content.**

The increase in automation of coal mining and its subsequent transportation to the site results in the rise of small classes and ash content in the extracted ROM. In addition, at washing plants some coal fines are formed during coal washing in water and suspended mixtures, and some – during transporting coal along the mill.
Modern coal preparation plants operate in a closed water-slurry circuit, resulting in the accumulation of slurry in the process flow, namely the increase of solids content in circulating water.
The above described factors lead to an increase of solid content in the return water of water-slurry flows at CPPs.
The results of coal preparation depend significantly on the degree of return water clarification. High solids content in the circulating water increases its viscosity; technological indicators of preparation are worsening respectively, the efficiency of slurry thickening and dewatering decreases, as well as water clarification.
The speed and efficiency of the sedimentation process are significantly influenced by many factors. Such as coal mark, concentration and size of the particles, their surface properties, dissolved in water impurities, medium acidity change, introduction of coagulants and flocculants.

Water-soluble synthetic flocculants are widely used around the world in the process of coal preparation. They are used in most systems for differentiation of solid / liquid phases, increasing sedimentation speed, improving the transparency of the clarified water, increasing the equipment efficiency, filtration efficiency and reducing costs significantly.
In this regard, there was a need to study the effect of cation and anion flocculants on the slurry thickening process to get a clean discharge of a radial thickener, a denser sludge underflow and its further dewatering on the filter presses.
Selection of flocculants at CPPs is determined by the technological purpose of flocculants:

 
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- to achieve a high sedimentation speed the flocculants with a high molecular weight are usually used;
- if for suspended mixture sedimentation we need a high degree of water clarification, we use medium or low molecular weight polymers with high ionic activity.

During the study of the effect of flocculants on the sedimentation process we conducted a series of laboratory tests to identify the most effective flocculants at SUEK coal preparation plants: CPP “Kirov mine” “SUEK Khakasia”, “Tugnuiskaja”, “Chegdomyn”, “Komsomolets mine” LLC “SUEK-Kuzbass” with flocculants produced by CJSC “BASF”, “Himtorgservis”, “Kemira Company”, “Santec Company”, “Ashland Industries”, LLC “ SNF Vostok”.
The main indicators of the sedimentation process are:

- Sedimentation speed;
- Purity of discharge;
- Solids content in thick layer;
- Flocculants consumption and concentration

And as one of the main estimates of the flocculants effectiveness the author used a quantitative evaluation of flocculation ability α, taking into account the rate of slurry sedimentation without additives and with a flocculant, as well as its consumption.

$$\alpha=\frac{(\upsilon-\upsilon_0)}{(\upsilon_0 \times g \times 100)} \quad , (7)$$

where υ – sedimentation speed of flocculated slurry mm / min;
$\upsilon_0$ - sedimentation speed of slurry without a flocculant mm / min;
$g$ - flocculant consumption, g / t.

Due to the fact that the effect of flocculants on sedimentation process for each particular mill was shown in different values, the values have been determined empirically.
The studies have been conducted on the feed of radial thickener. Size distribution of the solid sediment of radial thickener feed for each mill is presented in Table 1

Table 1

| Grade, mm | CPP “Kirov mine” | | CPP “SUEK Khakasia” | | CPP “Tugnuiskaja” | | CPP “Chegdomyn” | | “Komsomolets mine” LLC “SUEK-Kuzbass” | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Solids in the feed – 65,0 kg/m$^3$ | | Solids in the feed – 32,6 kg/m3 | | Solids in the feed – 56,0 kg/m3 | | Solids in the feed – 34,0 kg/m3 | | Solids in the feed – 47,8 kg/m3 | |
| | γ, % | $A^d$,% | γ, % | $A^d$,% | γ, % | $A^d$,% | γ, % | $A^d$,% | γ, % | $A^d$,% |
| +1,0 | - | - | - | - | 0,7 | 39,6 | - | - | - | - |

| | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| 0,5-1,0 | 0,2 | 3,8 | - | - | 4,6 | 35,5 | - | - | 0,4 | 13,1 |
| 0,3-0,5 | 0,7 | 6,3 | - | - | 10,8 | 30,7 | 0,4 | 14,4 | 1,8 | 13,1 |
| 0,2-0,3 | 1,0 | 13,0 | 1,7 | 9,5 | 12,3 | 29,0 | 0,5 | 9,6 | 2,5 | 13,6 |
| 0,1-0,2 | 1,8 | 23,3 | 5,4 | 5,0 | 8,9 | 33,7 | 3,8 | 9,7 | 6,1 | 16,5 |
| 0,05-0,1 | 9,6 | 13,1 | 17,3 | 7,7 | 4,1 | 33,6 | 9,5 | 13,6 | 1,3 | 17,2 |
| -0,05 | 86,7 | 56,1 | 75,6 | 35,5 | 58,6 | 37,9 | 85,8 | 57,4 | 87,9 | 45,2 |
| Total: | 100,0 | 50,5 | 100,0 | 28,6 | 100,0 | 35,4 | 100,0 | 51,0 | 100,0 | 41,6 |

CPP Komsomolets mine” LLC “SUEK-Kuzbass” "and CPP “Kirov mine” wash coals of “G” and “GZh” marks of Leninsky district with a high content of clay fine particles. During the research at these mills we could not get clean enough discharge and solids in the compressed sediment. Tables 1 and 2 show the results of a flocculated slurry suspension.

The results of flocculated slurry suspension of CPP “Kirov mine”.

Table 2

| Flocculant name | Sedimentation speed, m/h | Solids content, kg/m $^3$ | | α |
|---|---|---|---|---|
| | | In clarified layer just after the test | In an hour in a condensed sediment | |
| Tehnoflok TN 336 | 8,2 | 5,21 | 230,3 | $0{,}29\times10^{-2}$ |
| Superfloc A 110 HMW | 25,7 | 2,27 | 383,2 | $1{,}17\times10^{-2}$ |
| Praestol 2640 | 18,0 | 3,46 | 331,1 | $0{,}65\times10^{-2}$ |
| AN 934 VHM | 17,7 | 4,27 | 351,9 | $0{,}82\times10^{-2}$ |

The results of flocculated slurry suspension of “Komsomolets mine” LLC “SUEK-Kuzbass”

Table 3

| Flocculant name | Sedimentation speed, m/h | Solids content, kg/m 3 | | α |
|---|---|---|---|---|
| | | In clarified layer just after the test | In an hour in a condensed sediment | |
| Superfloc A 100 HMW | 33,3 | 1,72 | 387,9 | $0{,}35\times10^{-2}$ |
| AN 905 MPM | 30,0 | 1,16 | 369,0 | $0{,}31\times10^{-2}$ |
| Praestol 2514 | 30,0 | 2,88 | 360,8 | $0{,}31\times10^{-2}$ |

Adding cation active flocculants before and after anion active ones did not have a visible effect. In order to achieve satisfactory results of sedimentation process it required considerable consumption of flocculants as compared with the other mills of “SUEK”. For the CPP “Kirov mine” the following flocculants have been used: Superfloc A-110 HMW, AN 934 VHM, Praestol 2640; for CPP “Komsomolets mine” of LLC “SUEK-Kuzbass” – the flocculants Superfloc A-100 HMW, AN 905 MRM, Praestol 2514.
CPP “Chegdomyn” washes coals of “G” mark from Verhnebureinski area of Urgalsk deposit. Nevertheless, slurry thickening is effective with the use of one anionic flocculant. Finally we got satisfactory results with the use of the following flocculants: Magnaflok-155, Accofloc A-110 and AN 923 MPM. Table 4 shows the results of flocculated slurry.

The results of flocculated slurry suspension of CPP “Chegdomyn”

Table 4

| Flocculant name | Sedimentation speed, m/h | Solids content, kg/m 3 | | α |
|---|---|---|---|---|
| | | In clarified layer just after the test | In an hour in a condensed sediment | |
| Magnaflok 155 | 25,7 | 0,58 | 371,0 | $0{,}49\times10^{-2}$ |
| Accofloc A 110 | 26,7 | 0,44 | 375,7 | $0{,}51\times10^{-2}$ |
| AN 923 MPM | 25,7 | 0,55 | 376,5 | $0{,}49\times10^{-2}$ |

Above listed mills wash coals of “G” mark. Satisfactory sedimentation of coals from various deposits required the use of flocculants of different firms.
As a rule, slurry sedimentation takes place in a radial thickener using one anionic flocculant. But during the research the use of just anionic flocculants at CPP “SUEK Khakassia”, washing coals of “D” mark from Chernogorsky deposit, did not show any positive effect.
In this regard, the tests were carried out with additional supply of cation flocculants after two patterns:
The first pattern – feeding a cationic flocculant, and then at an interval - anionic;
The second pattern – feeding anionic flocculant, and then at an interval - cationic.
Satisfactory slurry sedimentation has been reached in the first pattern with the flocculants of the following series: Magnaflok 380 + Magnaflok 10, 4565 + FO AN 910, Superfloc C 498+ Superfloc A130 HMW, Accofloc N 100 and Tehnoflok TO 68 + Tehnoflok 10 M.
The results are shown in Table 5.

The results of flocculated slurry suspension of CPP “SUEK Khakassia”

Table 5

| Flocculant name | Sedimentation speed, m/h | Solids content, kg/m 3 | | α |
|---|---|---|---|---|
| | | In clarified layer just after the test | In an hour in a condensed sediment | |
| Magnaflok 380 + Magnaflok 10 | 32,7 | 0,15 | 464,4 | $0{,}35\times10^{-2}$ |
| Tehnoflok TO 68+ Tehnoflok TN 10M | 21,8 | 0,21 | 451,1 | $0{,}23\times10^{-2}$ |

| | | | | |
|---|---|---|---|---|
| Superfloc A130 HMW+Superfloc C 498 | 28,8 | 0,33 | 438,3 | $0{,}30\times10^{-2}$ |
| Accofloc N 100 | 24,5 | 0,26 | 450,4 | $0{,}26\times10^{-2}$ |
| FO 4565 SUEK +AN 905MPM | 22,5 | 0,13 | 434,2 | $0{,}24\times10^{-2}$ |

The results of flocculated slurry suspension of the mill

Table 6

| Flocculant name | Sedimentation speed, m/h | Solids content, kg/m 3 | | α |
|---|---|---|---|---|
| | | In clarified layer just after the test | In an hour in a condensed sediment | |
| Magnaflok 336 | 43,4 | 0,26 | 399,0 | $3{,}09\times10^{-2}$ |
| Superfloc A110HMW | 30,0 | 0,24 | 398,8 | $2{,}13\times10^{-2}$ |
| Praestol 2540 | 30,3 | 0,25 | 328,7 | $2{,}16\times10^{-2}$ |
| Tehnoflok 62020 | 28,8 | 0,25 | 372,6 | $2{,}05\times10^{-2}$ |
| AN 913 | 31,3 | 0,18 | 372,5 | $2{,}23\times10^{-2}$ |

CPPs “SUEK Khakassia” and “Tugnuiskaya” wash coals of “D” mark. Satisfactory results have been achieved with the use of the same series of flocculants, but at CPP “SUEK Khakassia” there was a need to add a cationic flocculant.

Efficient choice of flocculants can solve many problems in production. Thickening of waste slurry plays a fundamental role in maintaining production at a certain level, as well as in providing the possibility to re-use the return water. In addition, the flocculants have a significant impact on the process of slurry dewatering, allowing to reduce the volume and wet of the produced cake, and to prolong the life-time of the filter cloth. As well as the correct choice of flocculants ensures stable operation and productivity gains of the equipment and consequently reduces the flocculant consumption, thus decreasing the production costs.

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# Economic Benefits of a Deslime Flotation Circuit Using StackCell Technology at Patriot's Kanawha Eagle Coal Preparation Plant

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## ABSTRACT

In 2011, a fine coal flotation circuit was installed at the Kanawha Eagle coal preparation plant using the Eriez StackCell™ flotation technology. Laboratory testing indicated that this system would successfully recover a significant amount of fine coal currently lost as waste. However, after installing the by-zero flotation circuit, it was found that sufficient frother could not be added on a continuous basis to maximize combustible recovery. As frother was increased, the plant became unstable due to froth handling issues manifesting in the thickener and magnetic separator circuits. Based on a detailed engineering analysis of the plant, it was determined that significantly more product tons would be recovered by desliming the feed slurry to remove ultrafine slimes smaller than about 0.045 mm. This circuit change, instituted in 2014, resulted in a higher concentration of frother being utilized in the 150x45 micron flotation circuit, which in turn improved the froth characteristics of the StackCell™ flotation system. This approach also complemented the existing dewatering technology and ultimately resulted in an improvement in global combustible recovery by adding about 10 t/hr of high-quality clean coal to the saleable product.

**Keywords:** Fine Coal, Column Flotation, StackCell™ , Deslime Cyclones, Froth Handling.

## 1. INTRODUCTION

The Kanawha Eagle mining complex is one of several coal production facilities owned and operated by Blackhawk Mining, LLC. The company mines and sells metallurgical, thermal, PCI, stoker and specialty coals from the Central Appalachian Basin (Kentucky and West Virginia) and thermal coals from the Illinois Basin. The Kanawha Eagle complex is located in Kanawha County in southern West Virginia. Kanawha Eagle is home to a 600 t/hr coal preparation plant (Figure 1) that is supplied run-of-mine feed by three underground mines operating in the Peerless and Eagle seams. The cleaned products generated by this facility are primarily sold into the metallurgical (coking) market. The cleaned coals are shipped via CSX rail directly to customers or by private railroad lines serving barge loadouts along the Kanawha River.

Figure 1. Kanawha Eagle preparation plant.

A simplified flowsheet for the historical design of the Kanawha Eagle plant is provided in Figure 2. The plant was designed to receive coarse run-of-mine coal that was sized and crushed to below 50 mm using a raw coal screen in combination with a roll crusher. The material passing the raw coal screen was fed to a deslime screen configured with 1 mm openings. The 50x1 mm oversize material from the deslime screen was combined with the minus 50 mm crusher product and fed to a dense medium cyclone (DMC) circuit. The DMC products were passed across drain-and-rinse screens to recover the dense medium and to dewater coarse particles. Smaller particles from the lower deck of the clean coal drain-and-rinse screen were passed to a centrifugal dryer to reduce the surface moisture of the clean coarse coal product. The fine minus 1 mm particles passing the deslime screen were pumped to raw coal classifying cyclones to generate a nominal 1x0.15 mm underflow as the feed material for a bank of spiral concentrators. The minus 0.15 mm particles in the cyclone overflow were passed to the refuse thickener and discarded as waste. The clean coal product from the spiral was dewatered/deslimed using sequential stages of clean coal cyclones and static sieves. The spiral refuse was dewatered using a high frequency



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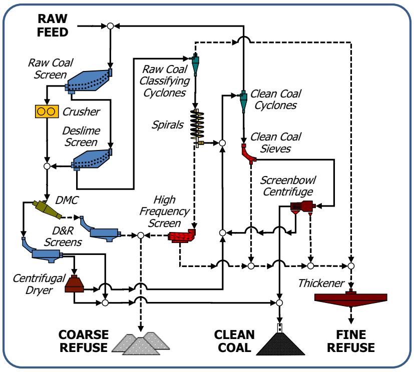


Figure 2. Original plant flowsheet incorporating only dense medium cyclones and spirals.

screen and back-blended with the coarse refuse from the DMC circuit. Final dewatering of the spiral/cyclone/sieve clean coal product (1x0.15 mm) was accomplished using a screenbowl centrifuge. The effluent from the screenbowl, which consisted of a dilute slurry of ultrafine (minus 0.045 mm) particles, was passed to the thickener and discarded as waste. The clean coal product from the screenbowl was back-blended with the coarse clean coal product generated by the DMC circuit prior to shipment to customers.

## 2. PLANT UPGRADES

### StackCell™ Installation

It had long been recognized that a major shortcoming of the original Kanawha Eagle plant was the loss of fine coal in the minus 0.15 mm raw coal cyclone overflow. To eliminate this problem, plant management investigated several options for recovering clean coal from this stream. While froth flotation seemed to be an obvious choice, there were several issues that complicated any retrofit to the existing plant circuitry. The first issue was the large amount of ultrafine clay slimes present in the plant feed. As shown in Table 1, the minus 0.045 mm (325 mesh) fraction of the plant feed represented over 8% of the total feed mass and contained more than 65% ash (dry basis). The ash content of the minus 0.045 mm material was much greater than that present in the other size fractions between 1 and 0.045 mm, which ranged from about 31% to 37% ash. The presence of high-ash ultrafine slimes in the feed forced the plant to adopt column flotation technology for recovering lost coal in the fine (minus 0.15 mm) waste stream. Column machines utilize froth washing systems to eliminate the hydraulic carryover of high-ash clay slimes into the clean coal product (Finch and Dobby, 1990; Rubinsetin, 1995). To evaluate the potential of column technology, a series of flotation release analysis tests were conducted on slurry samples from the fine waste stream at the plant. This experimental method provides a series of froth concentrates that are essentially free of entrained material (Dell, 1963; Dell et al., 1972). As such, a release analysis test mimics the separation performance expected from a column cell equipped with a well-designed and properly operated froth washing system (Luttrell et al., 2001). The release analysis data, which are summarized in Table 2, indicate that the waste fines fed to the Kanawha Eagle plant could be upgraded from a feed ash content of about 56% ash to a clean coal product of less than 8% ash at a combustible

Table 1. Size and quality analysis of plant feed.

| Size (mm) | Mass (%, dry) | Ash (%, dry) | Sulfur (%, dry) |
|---|---|---|---|
| Plus 100 | 0 | 0 | 0 |
| 100 x 50 | 5.01 | 85.74 | 0.49 |
| 50 x 12.7 | 23.83 | 74.37 | 0.27 |
| 12.7 x 9.5 | 6.44 | 58.45 | 0.5 |
| 9.5 x 1 | 42.62 | 43.09 | 0.72 |
| 1 x 0.5 | 5.21 | 35.14 | 0.79 |
| 0.5 x 0.15 | 5.66 | 31.83 | 0.79 |
| 0.15 x 0.045 | 3.16 | 37.38 | 0.78 |
| Minus 0.045 | 8.07 | 65.16 | 0.22 |
| Total | 100.00 | 54.22 | 0.56 |

Table 2. Flotation release analysis conducted on fine (minus 0.15 mm) feed coal.

| Product Stream | Individual | | | | Cumulative | | | |
|---|---|---|---|---|---|---|---|---|
| | Mass (%, dry) | Ash (%, dry) | Sulfur (%, dry) | Recovery (%) | Mass (%, dry) | Ash (%, dry) | Sulfur (%, dry) | Recovery (%) |
| Froth 1 | 25.66 | 6.31 | 0.88 | 55.30 | 25.66 | 6.31 | 0.88 | 55.30 |
| Froth 2 | 5.21 | 6.83 | 0.67 | 11.16 | 30.87 | 6.40 | 0.84 | 66.46 |
| Froth 3 | 2.98 | 7.12 | 0.76 | 6.37 | 33.85 | 6.46 | 0.84 | 72.83 |
| Froth 4 | 8.26 | 13.89 | 0.68 | 16.36 | 42.11 | 7.92 | 0.81 | 89.19 |
| Froth 5 | 1.08 | 24.98 | 0.74 | 1.86 | 43.19 | 8.35 | 0.80 | 91.05 |
| Tail 1 | 1.02 | 71.31 | 1.56 | 0.67 | 44.21 | 9.80 | 0.82 | 91.72 |
| Tail 2 | 55.79 | 93.55 | 0.08 | 8.28 | 100.00 | 56.52 | 0.41 | 100.00 |
| Total | 100.00 | 56.52 | 0.41 | 100.00 | -- | -- | -- | -- |

recovery approaching 90% and mass yield approaching 43%. The primary waste stream (Tail 2) rejected by the release analysis method contained a very high ash content of 93.5% ash and represented nearly 55.8% of the mass present in fine flotation feed sample.

While the release analysis data were very promising, the decision to utilize column flotation technology was difficult to accommodate at the Kanawha Eagle plant due to the large volumetric footprint and high foundation loads that would be required for this application. In light of this challenge, plant management identified an alternative flotation system known as the StackCell™ technology for this retrofit application. As shown in Figure 3, the StackCell™ technology is a compact low-profile flotation machine that was developed by the Eriez Flotation Division


Figure 3. Schematic of the Eriez StackCell™.

Figure 4. Preferred layout of StackCell™ units.

for retrofit applications that require washing of froth concentrates (Kohmuench et al., 2007). The low-profile design allows multiple units to be easily stacked in-series (Figure 4) so that high tonnage rates of froth product can be produced within a volumetric space less than that typically occupied by a single column flotation machine. Moreover, the live volume of slurry within the machine is only a fraction of that held within a column cell, greatly reducing the structural demands of a retrofit application.

The StackCell™ technology makes use of a pre-aerated high-shear feed canister that provides for highly efficient bubble-particle contacting, thereby substantially shortening the residence time required for coal collection and virtually eliminating most of the column height (Kiser et al., 2013). During operation, feed slurry enters the StackCell™ through either a bottom or side mounted feed nozzle where low pressure air is introduced using a blower. The slurry and gas mixture is forced into an enclosed high-shear mixing chamber that simultaneously disperses the injected gas into very small bubbles and provides intense high-energy contacting between bubbles and particles. After a few seconds of contacting, the gas-slurry mixture is radially discharged into the primary flotation tank. The tank serves as a quiescent chamber for phase separation of the tails pulp and froth concentrate. Slurry level is maintained inside the tank so as to provide a deep froth that can be washed, thereby providing a high-quality low-ash froth product.

The StackCell™ technology was installed at the Kanawha Eagle plant in 2010. Figure 5 shows a simplified flowsheet of the retrofit installation. The three-stage cells were designed to intercept and recover lost fine (minus 0.15 mm) coal present in the waste overflow slurry generated by the raw coal classifying cyclones. Unfortunately, several problems were encountered that limited the operation of the new flotation circuit. The first and most serious problem was the unexpected and unwanted stability of the froth product that made passage of the clean coal product to

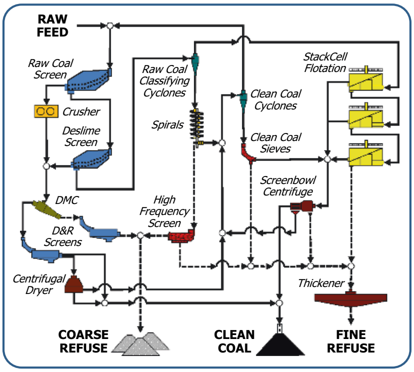


Figure 5. Retrofit flowsheet showing the installation of new StackCell™ units to recover fine (minus 0.15 mm) coal from a previously discarded waste fines stream.

downstream dewatering units very challenging. This problem is not uncommon in fine particle flotation circuits (Wills, 2006). Excess frothing and foaming problems also occurred in many of the plant sumps in the flotation, classification, dense medium and magnetic separator circuits. Persistent froth also began to form on the plant thickener, creating black water problems in the clarified water system. To avoid downstream froth handling problems, the dosage of frothing agent was significantly reduced to about 2 PPM, which was too low for effective operation of the StackCell™ or any other flotation system. Therefore, the initial retrofit installation of the new flotation circuit failed to meet expectations of plant management in terms of coal recovery and overall profitability.

## Deslime Cyclone Installation

After the installation of the new flotation circuit, Cardno, Inc. was contracted to identify opportunities for plant-wide yield and quality improvements at the Kanawha Eagle plant. Site visits and a thorough review of efficiency data confirmed that the new flotation cells installed at the plant were well designed and mechanically sound, but had failed to perform to expectations due for two main reasons. First, the dosage of frother added to the flotation feed was too low to generate small gas bubbles and maintain a stable froth phase that could withstand the introduction of wash water. Since the Kanawha Eagle circuit uses screenbowl centrifuges to dewater the froth product, only about half of the ultrafine (minus 0.045 mm) solids fed to the circuit was recovered. The remainder of the solids and a substantial amount of the residual frother was discharged as screenbowl effluent to the thickener. As such, the clarified process water that circulated back to the plant caused massive frothing problems in many of the other processing circuits. Second, the feed to the StackCell™ was found to contain a substantial amount of ultrafine hydrophobic slimes (i.e., >65% minus 0.045 mm coal). The ultrafines quickly overloaded the available bubble surface area, resulting in lost high-quality coal in the 0.15 x 0.045 mm size range. Increasing the bubble surface area via the use of more frother (which would reduce bubble size and maintain a deep froth for washing) was not possible due to excessive froth handling problems throughout the plant.

As a consequence of the problems described above, the plant management determined that the currently configured StackCell™ system would not be capable of attaining the target coal production. Instead, an alternative circuit design called a deslime column flotation circuit was recommended by Cardno, Inc. This type of flotation circuit, which has been described in detail elsewhere (Bethell and Luttrell, 2005; Bethell et al., 2006), involves the installation of small diameter (15 cm) classifying cyclones ahead of flotation. In this layout, the overflow stream from deslime cyclones is typically discarded as waste, while the underflow stream is passed to the flotation circuit for upgrading. This type of circuit offered several notable advantages for the Kanawha Eagle plant. First, the use of a deslime circuit would greatly reduce the total volume of slurry that would need to be dosed with frother, thereby minimizing downstream froth handling problems. Second, the rejection of ultrafine coal particles from the flotation feed would increase coal recovery in the 0.15 x 0.045 mm size range by eliminating the hydrophobic slimes that overloaded the available bubble surface area. Third, the rejection of ultrafine clays from the flotation feed would improve the quality of the froth product by substantially reducing the amount of high-ash solids contained in the process water carried into the froth launders. Finally, the use of a deslime circuit would lower the total plant moisture by reducing the amount of ultrafines recovered in the dewatered products from the screenbowl centrifuges. The only downside of the deslime circuit would be the loss of potentially marketable ultrafine coal, which was determined to be minimal at the Kanawha Eagle plant due to the high-ash content of the minus 0.045 mm size fraction.

In light of the many benefits offered by the proposed deslime circuit, plant management decided to move ahead with the installation of the deslime cyclone circuitry. Unfortunately, a major hurdle was encountered with the utilization of StackCell™ technology in a deslime circuit configuration. Because of the unique design of the compact high-capacity StackCells™, the volumetric flow of slurry fed to the flotation bank had to be maintained at a level similar to the original design of 690 $m^3$/hr (3,000 GPM). Since the underflow from the deslime cyclones was estimated to be only 112 $m^3$/hr (488 GPM), other process streams had to be utilized to make up the difference. After a careful analysis of the plant circuitry, a unique approach was conceived in which all of the sieve screen underflow and portions of the flotation tails and screenbowl effluent would be passed back to the flotation feed.

A simplified schematic of the final plant flowsheet incorporating deslime cyclones together with the compact three-stage StackCell™ flotation bank is provided in Figure 6. As shown, the feed to the flotation circuit consisted of:

- 112 $m^3$/hr (488 GPM) of fresh circuit feed from the deslime cyclone underflow,
- 115 $m^3$/hr (500 GPM) of underflow from the clean coal sieve screens,
- 242 $m^3$/hr (1050 GPM) of main effluent from the screenbowl centrifuges,
- 173 $m^3$/hr (750 GPM) of recycled tailings from the end of the froth flotation bank. and
- 49 $m^3$/hr (212 GPM) of clarified make-up process water from the plant thickener overflow.

The new feed flow from the underflows of the deslime cyclones and sieve screens accounted for about one-third (16+17=33%) of the flow to the flotation circuit, while the remaining two-thirds (7+35+25=67%) of the flow was comprised of diverted portions of clarified process water, screenbowl effluent, and flotation tails. Another important benefit of the new circuit layout was the recycle of process streams (i.e., screenbowl effluent and flotation tails) that contained high concentrations of frother. The recycle/reuse of this surfactant lowered the consumption of frother in addition to reducing downstream froth handling problems.

In early 2014, the design, procurement and installation of equipment associated with the new deslime cyclones was initiated. The construction project involved considerable

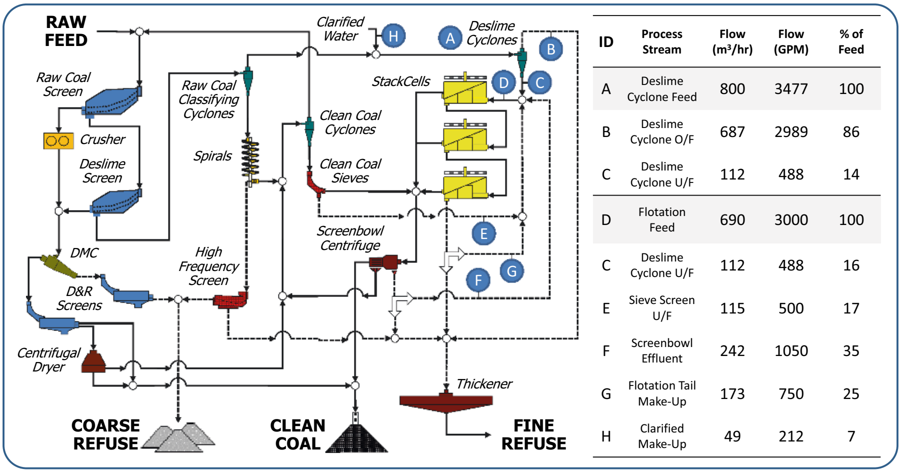


Figure 6. Retrofit flowsheet showing the installation of new deslime cyclones and reconfiguration of slurry flows.

work to install piping, valves, and control systems necessary to facilitate the redistribution of slurry flows. The construction project was contracted to Powell Construction Company of Beckley, West Virginia. The newly configured circuit commenced production in December 2014.

## 3. CIRCUIT COMPARISON

A detailed analysis of the production rates for the newly installed deslime flotation circuit indicated that the upgraded layout efficiently captured the high-quality 0.15 x 0.045 mm coal lost in the originally installed StackCell™ circuit. Initial test data at the plant site confirmed that with adequate frother the StackCells™ were able to achieve combustible recoveries of up to 85-90% (Graley et al., 2013). However, due to low frother constraints, the day-to-day performance of the flotation bank was relatively poor. As shown in Table 3, the pre-upgrade flotation circuit achieved a tailings ash of only 59.8% ash from a feed ash of 51.5%. Likewise, the mass yield of clean coal reporting to the froth was only 15.5%, which corresponded to a recovery of combustible material of less than 30%. Consequently, only 8.4 t/hr (dry basis) of clean coal was recovered from the pre-upgrade flotation circuitry that had

Table 3. Comparison of fine coal circuit performance before and after the deslime cyclone upgrade.

| Circuit | Dry Solids (t/hr) | | | Ash (%, dry) | | | Yield (%) | | Recovery (%) | |
|---|---|---|---|---|---|---|---|---|---|---|
| Layout | Feed | Froth | Tails | Feed | Froth | Tails | Froth | Tails | Froth | Tails |
| Pre-Upgrade | 54.5 | 8.4 | 46.1 | 51.50 | 6.20 | 59.80 | 15.5 | 84.5 | 29.9 | 70.1 |
| After Upgrade (Eagle Seam) | | | | | | | | | | |
| Test 1 | 28.9 | 16.7 | 12.2 | 38.50 | 6.39 | 82.21 | 57.6 | 42.4 | 87.7 | 12.3 |
| Test 2 | 27.0 | 16.1 | 10.9 | 40.54 | 7.53 | 89.57 | 59.8 | 40.2 | 92.9 | 7.1 |
| Test 3 | 36.4 | 17.7 | 18.7 | 43.75 | 5.58 | 79.96 | 48.7 | 51.3 | 81.7 | 18.3 |
| Average | 30.8 | 16.8 | 13.9 | 40.93 | 6.50 | 83.91 | 55.4 | 44.6 | 87.5 | 12.5 |
| After Upgrade (Peerless Seam) | | | | | | | | | | |
| Test 4 | 34.7 | 18.4 | 16.3 | 39.23 | 6.56 | 76.10 | 53.0 | 47.0 | 81.5 | 18.5 |
| Test 5 | 38.9 | 20.1 | 18.8 | 48.29 | 9.07 | 90.35 | 51.7 | 48.3 | 91.0 | 9.0 |
| Test 6 | 33.7 | 19.2 | 14.5 | 43.57 | 11.05 | 86.85 | 57.1 | 42.9 | 90.0 | 10.0 |
| Test 7 | 38.8 | 20.6 | 18.2 | 41.10 | 9.30 | 77.19 | 53.2 | 46.8 | 81.9 | 18.1 |
| Average | 36.5 | 19.6 | 16.9 | 43.05 | 9.00 | 82.62 | 53.8 | 46.2 | 86.1 | 13.9 |

been installed to treat 54.5 t/hr of the by-zero feed slurry.

After upgrading the circuit with deslime cyclones, the StackCell™ performance improved dramatically. The slimes removal and effluent stream circulation enabled the flotation cells to be "pulled hard" without excessive froth handling problems in the rest of the plant. After the upgrade, the average recoveries improved to 87.5% and 86.1%, respectively, for the Eagle and Peerless seams (Table 3). The average tails ash values increased to 83.9% for the Eagle seam and to 82.6% for the Peerless seam, which were much better than the 59.8% tails ash obtained in the pre-upgrade testing program.

Based on the values presented in Table 3, management determined that the new deslime cyclone circuit and improved slurry redistribution generated 16-20 t/hr of fine coal product from the StackCell™ circuit. This level of production represented an increase of more than 8 t/hr of saleable coal for the Eagle seam and more than 11 t/hr of saleable coal for the Peerless seam. The gains for 5,500 operating hours per year represented in excess of 75,000 additional clean marketable tons per year for the Kanawha Eagle plant. For the market price at that point in time, the new tonnage provided ~US$4 million in additional annual revenue for the capital investment of US$0.85 million. Moreover, the recirculation of the main effluent streams from the screenbowl centrifuges retained much of the frothing chemicals within the flotation circuit. This change reduced the consumption of frother at the plant by about 40% compared to pre-upgrade dosing levels.

## 4. SUMMARY

A new fine coal flotation system called the StackCell™ technology was installed at the Kanawha Eagle plant to recover fine (minus 0.15 mm) coal from a previously discarded waste stream. The unique design of this compact cell made it ideally suited for retrofit applications where column-like separation performance via froth washing was required but footprint space and capital funds were limited. Unfortunately, the initial installation of this new flotation system failed to meet expectations due to downstream froth handling problems created by the large amount of ultrafine (0.045 mm x 0) slimes present in the coal feed coupled with deficiencies in the design of the existing plant flowsheet. In light of these constraints, engineers from Cardno, Inc. developed and recommended an improved plant flowsheet that utilized deslime classifying cyclones to remove slimes prior to flotation. The new circuitry reduced the volume of flotation slurry that had to be reagentized with frothing agent by nearly two-thirds, thereby eliminating froth handling problems throughout the plant. Unfortunately, this modification also eliminated much of the volumetric slurry flow required by the StackCell™ machines. To overcome this problem, the flows of several effluent streams within the fine coal circuit were intentionally redirected back to the flotation feed in order to provide sufficient volume flow for proper operation of the StackCell™ technology. The circulated streams included all of the clean coal sieve underflow and portions of effluent from the screenbowl centrifuges and flotation tails. The circulation of the main effluent from the screenbowl units also had the desirable side effect of retaining much of the frothing agent internally within the flotation circuit, thereby reducing the consumption of frother by nearly 40%. This approach also complemented the existing dewatering technology and ultimately resulted in substantial improvements in global combustible recovery and coal quality. Once fully implemented at the Kanawha Eagle plant, experimental testing showed that the new deslime flotation circuit increased the capture of fine coal by 8-11 t/hr. The undiscounted payback period for the capital investment associated with the deslime circuit installation was estimated to be less than 3 months.

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# Overview of Typical Coal Flotation Flowsheets in China's Coal Preparation Plants

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**Abstract:** The entrainment of high-ash fine slime severely contaminates clean coal product of traditional flotation cell, leading to serious technical problems such as high ash in clean coal, low recovery, and the sacrifice of the yield of gravity separation to lower the ash content of final product. In order to solve those issues, the performance of FCMC series cyclonic microbubble flotation column and traditional flotation cell were compared. Several typical flowsheets for the separation of coal flotation in China's coal preparation plants (CPPs) were extensively reviewed and deeply explored. Results show that FCMC flotation column is especially suitable for the beneficiation of fine and non-selective coal slime downstream the heavy medium separation. The mixed (one-stage) flotation process utilizing the combination of flotation column and clean coal quick-operating pressure filter is recommended in newly built and retrofitted CPPs. As for those utilizing one-stage flotation flowsheet equipped with traditional flotation cell and pressurized filter, either classified flotation process or secondary flotation for the filtrate of pressurized filter using flotation column and clean coal quick-operating pressure filter is recommended in their technical modifications.

**Keywords:** Flotation column; Flotation cell; Fine coal slime; Coal flotation process; Coal preparation plant; Pressurized filter; Clean coal quick-operating pressure filter

## 1 Introduction

China is the largest producer and consumer of coal resources in the world. Coal provides 65–70% of primary energy in China (Zhou et al. 2006, Chen 2012). The quality of the run of mine coal and coal slime increasingly deteriorate with the improvement of mechanization in mining and the increase in exploitation depth. Currently, more than 450 million metric tons of fine coal (the fine fraction with size smaller than 0.5 mm) presents in the raw coal produced annually in China. The fine fraction of coal is difficult to process, and its utilization efficiency is low. The recovery of -0.5 mm fine coal slime, which accounts for 15–25% of raw material, allows us to effectively utilize coal slime resources. It is of great strategic significance and realistic importance to the sustainable development of China's coal industry and environmental protection (Xie et al., 1999, Xie et al., 2004, Xie et al., 2005).

Nowadays, customers in Chinese market require strictly on the quality of clean coal product, and demands lower and lower ash in clean coal. The popularization of dense medium separation technology realizes precise beneficiation of coarse coal particles; however, coal slime entering flotation section is smaller, and its floatability becomes inferior. The entrainment of high ash fine slime in flotation significantly contaminates clean coal, making the clean coal ash of flotation 1-2% higher than that of gravity separation. It also results in low ash in tailing and low recovery of clean coal. The quality of overall clean coal product fluctuates dramatically, which affects the quality of clean coal product and economic benefit of enterprises (Ding 2010, Guo 2013). On one hand, the coal flotation section is the largest barrier for a coal preparation plant (CPP) to enhance its recovery of coal product. On the other hand, the coal flotation section has the greatest potentiality to be improved. In other words, the enhancement of coal flotation performance could be the easiest way to improve the overall recovery of the plant. The major solution to this technical dilemma is seeking a breakthrough in flotation equipment and flowsheet.

 
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## 2 Selection of and Performance Comparison between Flotation Devices in China's CPPs

Currently, the approaches for the beneficiation of fine and ultrafine coal particles primarily include froth flotation, bulk-oil flotation and selective flocculation. Among them froth flotation is the most mature and well commercialized technology. The dominant flotation devices in China's CPPs are mechanical flotation cell and flotation column. In China, the mechanical flotation cells were traditionally used as flotation equipment in CPPs. However, those devices had several disadvantages including low separation efficiency, high ash in flotation clean coal, and difficulty in regulating ash. They could not meet end users' requirements on clean coal quality (Wang et al., 2007, Liu, 2012). Therefore, they are gradually replaced by large flotation columns in China's newly built CPPs and in the technical retrofitting of existing plants. (Wang et al., 1999).

As shown in Table 1, the ash of clean coal obtained from floatation cell is 1-2% greater than the required clean coal ash. However, the ash of clean coal obtained from the flotation column can meet the requirements on clean coal quality in the CPPs. Furthermore, flotation column performs a better performance in the separation of -0.045 mm size fraction than flotation cell. Under the same conditions, the ash of the -0.045 mm fraction of clean coal of flotation column is 1-4.5% lower than the counterpart of flotation cell. Therefore, flotation column produces an overall clean coal product that has ash content 1-2% lower in comparison with flotation cell. The high ash fine slime is easier misreported to clean coal in the traditional mechanical flotation cell (equipped with shallow vessel) than in the flotation column (with thick froth layer).

Table 1 Comparison of Clean Coal Ash between Flotation Cell and Column

| CPP | Required Clean Coal in Market | Clean Coal of Flotation cell | | Clean Coal of Flotation Cloumn | |
|---|---|---|---|---|---|
| | Ash /% | Ash /% | -0.045mm Ash /% | Ash /% | -0.045mm Ash /% |
| Xuzhou Sanjia River CPP | 9.00 | 10.96 | 13.87 | 8.45 | 11.43 |
| Xuzhou Zhangxiaolou CPP | 9.00 | 11.09 | 14.65 | 8.97 | 10.89 |
| Hebei Dongpang Mine CPP | 9.00 | 11.17 | 14.61 | 8.25 | 10.75 |
| Xinwen Panxi Mine CPP | 10.00 | 12.88 | 16.73 | 10.02 | 13.97 |
| Linyi Gucheng CPP | 9.00 | 10.92 | 14.98 | 9.09 | 10.84 |
| Xinwen Longgu CPP | 8.50 | 8.70 | 13.35 | 8.42 | 12.54 |
| Linyi Wanglou Mine CPP | 9.00 | 11.24 | 14.81 | 8.91 | 11.09 |
| Panjiang Jinjia Mine CPP | 10.50 | 12.95 | 13.88 | 10.47 | 12.83 |
| Shanxi Lingshi CPP | 9.50 | 11.22 | 14.98 | 9.43 | 13.99 |

## 3 Review on Typical Coal Flotation Flowsheets in China's CPPs

In present, CPPs in China employ primarily flotation cell and flotation column in coal flotation. Because traditional flotation cells are encountering problems including low separation efficiency and high ash in clean coal caused by serious entrainment of high ash fine slime, researchers have been seeking a solution to this technical issue through breakthrough in the design of flotation process. Three representative flotation flowsheets in China's CPPs will be deeply explored in the following section.

### 3.1 Mixed (one-stage) Flotation Flowsheet

Mixed (one-stage) flotation process is the most commonly used flowsheet in CPPs in China. In a mixed (one-stage) flotation process, the 0.5-0 mm size fraction is separated directly in a flotation cell or a flotation column without classification to produce clean coal and tailing. Flotation feed is prepared in a slurry preparation device or a slurry pre-processor upstream the flotation equipment. Flotation clean coal is dewatered by a pressurized filter or a quick-operating pressure filter, while tailing is directed to a thickener for settling and clarification. The underflow of thickener is dewatered by a quick-operating pressure filter, and the overflow is recycled. Figures 1 and 2 present such a process equipped with flotation cell and flotation column, respectively.

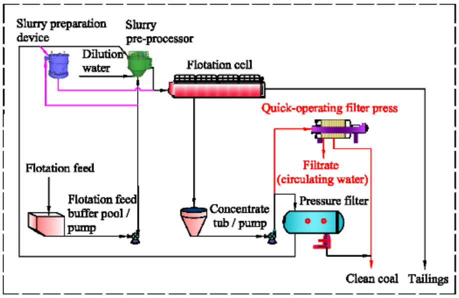


Figure 1. Mixed (single-stage) flotation flowsheet equipped with flotation cell and pressurized filter (or quick-operating pressure filter).

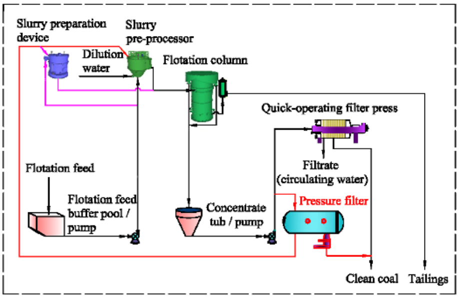


Figure 2. Mixed (single-stage) flotation flowsheet equipped with flotation column and quick-operating pressure filter (or pressurized filter).

Figures 1 and 2 show that the flotation device used in mixed (one-stage) flotation process is either flotation cell or flotation column. As mentioned above, it is concluded that cyclonic microbubble flotation column has superior performance to traditional flotation cell. The flotation column is a preferred option for coal flotation.

As for the dewatering of flotation clean coal, pressurized filter and quick-operating pressure filter are two dominant technologies used in CPPs in China.

Nowadays, clean coal quick-operating pressure filter is gradually employed in China's CPPs. Therefore, mixed (one-stage) flotation flowsheet using the combination of flotation column and quick-operating pressure filter is recommended in newly built and retrofitted CPPs. Figure 3 illustrates a typical flowsheet of the recommended process.

### 3.2 Classified Flotation Flowsheet

In a classified flotation process, the 0.5-0 mm wide size fraction is first divided into coarse and fine subcategory by a hydrocyclone prior to flotation. Corresponding reagent regimes, and flotation and dewatering equipment are selected for those two subcategories according to their floatability and filtration characteristics. In this manner, the advantages of separation and dewatering in each subcategory are maximized, thus, realizing the precise separation and efficient dewatering of coarse and fine coal slime, respectively. A representative classified flotation process is presented in Figure 4.

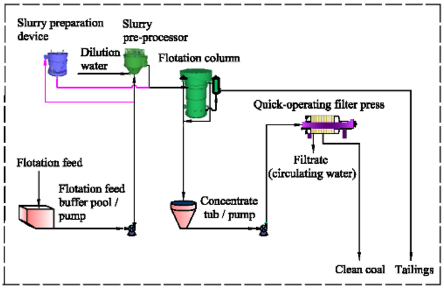


Figure 3. Optimum mixed (one-stage) flotation flowsheet using the combination of flotation column and quick-operating pressure filter.

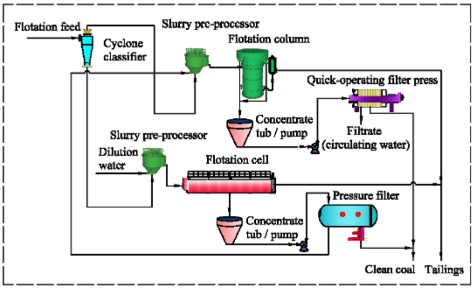


Figure 4. A representative classified flotation flowsheet.

In China, many CPPs employ one-stage flotation process equipped with traditional flotation cell and pressurized filter. The ash content and moisture of clean coal are high. Retrofitting is needed to solve this technical problem. It is preferred to take advantage of existing mechanical flotation cell and pressurized filter to reduce capital investment. Therefore, classified flotation process is adopted. The traditional flotation cell cannot effectively separate -0.074 mm fraction, particularly the -0.045 mm part, and pressurized filter suffers from inferior dewatering performance. The -0.074 mm fraction can be beneficiated by flotation column, and dewatered by quick-operating pressure filter. The coarse subcategory is still processed and dewatered by existing flotation cell and pressurized filter, respectively. Because different reagent regimes and flotation and dewatering devices are used for coarse and fine coal slime, respectively, the performance of the selected devices for flotation and dewatering in each subcategory are maximized. As a result, coarse and fine particles are precisely processed and efficiently dewatered, respectively. This classified flotation process has been successfully applied in Handan Coal Preparation Plant, Hebei, China, and has been proven suitable for the retrofitting of old CPPs.

**3.3 Two-stage Flotation Flowsheet**

The entrainment of high-ash fine slime seriously contaminates clean coal in traditional flotation cell. Therefore, the ash content of flotation clean coal is high and plants usually sacrifice the yield of gravity separation to lower the ash of final product. In order to resolve this issue, Guohua Technology Group Ltd. proposed the two-stage flotation process as shown in Figure 5.

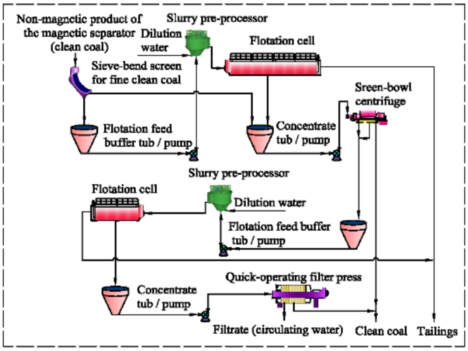

Figure 5. A representative two-stage flotation flowsheet.

Because primary flotation uses traditional flotation cell, flotation clean coal has a high ash content. The ash of the non-magnetic product of magnetic separator is also high. Therefore, the ash of dewatered clean product from the centrifuge is high. The particles in the secondary flotation cell are finer as the feed is the filtrate of centrifuge. As a result, the clean coal of secondary flotation, which will be mixed with final clean coal product, contains even higher ash. In conclusion, this process cannot effectively lower the ash content in flotation clean coal. Instead, it complicates the flotation system by using more devices. The operation and management of this flowsheet is difficult, and the capital investment is high.

However, if the flotation cell in the secondary flotation circuit is replaced by flotation column, the ash of secondary flotation clean coal might be effectively reduced. It is recommended to use flotation column in secondary flotation circuit, while remain primary flotation circuit unchanged. The modified two-stage flotation flowsheet is depicted in Figure 6.

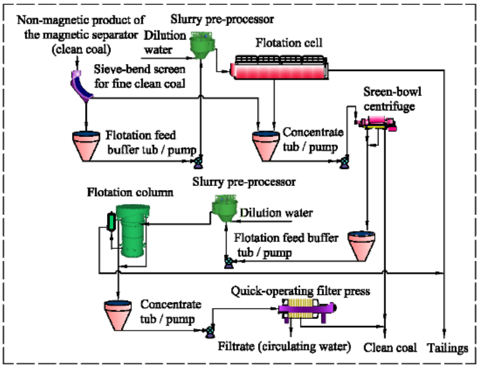


Figure 6 Variant (I) of two-stage flotation flowsheet.

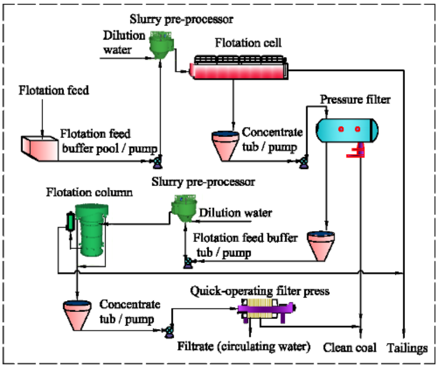


Figure 7. Variant (II) of two-stage flotation flowsheet.

In China, a number of CPPs utilize one-stage flotation circuit equipped with traditional flotation cell and pressurized filter. In addition to the methods described above, another variant of two-stage flotation flowsheet as shown in Figure 7 can be used to reject the ash in flotation clean coal. In this variant, plants use existing flotation cell and pressurized filter for primary flotation and dewatering. Filtration mesh with larger pores is preferred in pressurized filter to allow fine particles in primary flotation clean coal to penetrate to filtrate. The primary flotation circuit focuses on the recovery of coarse particles in flotation clean coal, and the improvement of dewatering performance, thereby mitigating the ash and moisture of filtration cake. The filtrate is then introduced to secondary flotation circuit, and its clean coal is dewatered by quick-operating pressure filter. The advantages of flotation column and quick-operating pressure filter in the treatment of fine particles are well utilized in this improved two-stage flotation flowsheet. Therefore, the ash and moisture in flotation clean coal are effectively reduced. This variant of two-stage flotation flowsheet has been successfully applied in the coal preparation plants of Bayi Mine within Shandong Zaozhuang Mining Group.

In summary, two-stage flotation scheme and its Variant I are not recommended. Instead, as for those utilizing mixed (one-stage) flotation flowsheet equipped with traditional flotation cell and pressurized filter, the Variant II of two-stage flotation process is highly recommended in their technical modifications. It will effectively reject ash and moisture in flotation clean coal.

## 4 Conclusions

(1) The efficient flotation and recovery of coal slime is an important part in coal preparation. Enhancing the performance of flotation circuit has the greatest potentiality to improve the recovery of clean coal in a CPP.

(2) Under the same conditions, the ash of the -0.0045 mm fraction in the clean coal of FCMC series cyclonic microbubble flotation column is 1-4.5% lower than the counterpart of traditional flotation cell. Flotation column produces an overall clean coal product that has ash content 1-2% lower in comparison with flotation cell. FCMC series column has obvious advantages over traditional flotation cell in reducing ash for the -0.045mm size fraction. It is especially suitable for the beneficiation of fine and non-selective coal slime downstream the heavy medium separation. It is a promising and efficient flotation device to reject ash in flotation clean coal.

(3) The mixed (one-stage) flotation process utilizing the combination of flotation column and clean coal quick-operating pressure filter is recommended in newly built and retrofitted CPPs. As for those using one-stage flotation flowsheet equipped with traditional flotation cell and pressurized filter, either classified flotation process or secondary flotation for the filtrate of pressurized filter using flotation column and clean coal quick-operating pressure filter is recommended in their technical modifications. These recommended solutions can effectively reduce ash and moisture in flotation clean coal product.

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# Coal flotation improvement through hydrophobic flocculation induced by polyethylene oxide

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**Abstract:** Polyethylene oxide (PEO) is an efficient flocculant for various minerals including coal, quartz and phyllosilicates. However, it is rarely used in coal flotation process. In this research, the flotation of coal with the addition of PEO was investigated. It was found that PEO could worsen the flotation performance if it was used solely. In the presence of sodium hexametaphosphate, however, PEO could improve flotation performance significantly. The combustible matter recovery was increased while the ash content was reduced when the PEO dosage was lower than 5 g/t. The tube settlement and floc observation experiments of coal and hydrophilic minerals indicated that PEO flocculated coal mainly through hydrophobic interaction whereas it flocculated kaolinite mainly through hydrogen bonding. The presence of $(NaPO_3)_6$ hindered the flocculation of hydrophilic minerals due to the enhancement of electrical double layer repulsion so coal was selectively flocculated. As a result, the particle size of coal was enlarged and its collision efficiency with bubbles was increased.

**Keywords:** coal; flotation; floc; flocculation; polyethylene oxide; settlement experiment; observation

## 1. Introduction

Flotation is the most widely used method for fine mineral separation. The concentration of valuable minerals is achieved via their attachment to bubbles. So the collision efficiency between minerals and bubbles to a large extent determines the flotation effect. As studied for a long time, the collision efficiency is assumed to be proportional to a power function of the ratio of particle and bubble size (Miettinen et al., 2010). Within certain limits, the collision efficiency is higher for the coarser particles or finer bubbles. If the particle size is very fine, the flotation performs badly due to the low collision efficiency (Chipfunhua et al., 2012). This problem has emerged for a long time in fine coal flotation.

As the collision efficiency increases as particle size, researchers tried methods to enlarge the particle size to improve fine mineral flotation. Oil agglomeration induced by the addition of non-polar oil could selectively aggregate hydrophobic minerals so as to improve flotation effect (Ozkan and Duzyol, 2014). However, this method is not very economic due to the large consumption of oil. Hydrophobic flocculation is also an efficient method to improve fine coal flotation.

Song has conducted extensive research in floc flotation of coal and sulphide ores (Song et al., 2000; Song et al., 2012). Fine coal flocculation can also be achieved by polymer flocculant (Sabah and Erkan, 2006; Brostow et al., 2007). Polyethylene oxide (PEO) is an efficient flocculant for various minerals. It shows better flocculation effect than the conventional polyacrylamide in the settlement of kaolinite and smectite due to the less electrostatic repulsion between mineral and polymer molecules (Mpofu et al., 2003; Mpofu et al., 2004). In the flotation of sulphide ores, PEO showed surprising effect in concentrate upgrading via the selective flocculation to quartz (Gong et al., 2010). PEO could also reduce the kaolinite entrainment due to the size enlargement (Liu and Peng, 2014). However, the influence of PEO addition on coal flotation has rarely been studied. As PEO can flocculate both coal and gangue minerals such as kaolinite, smectite and quartz, it is doubtable whether selective flocculation could realize. It is also unknown whether the flocculation mechanism of hydrophobic coal and hydrophilic gangue minerals is different and whether the difference could be used. In this research, coal flotation experiment with PEO was conducted. Through the tube settlement experiment and observation, the reason why PEO could improve coal flotation in the presence of $(NaPO_3)_6$ was also explained.



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## 2. Materials and methods

*2.1 Materials*

The bituminous coal sample was from a coal preparation plant in Zaozhuang, China. The particles of +2.0 g·cm$^{-3}$ density fraction were ground to minus 45 μm to conduct X-ray diffraction measurement (D8 Advance, Bruker, Germany). The results show that the main minerals in this sample are kaolinite, quartz and calcite. The fractions of these three minerals are around 49.7%, 33.4% and 16.9%. Due to the highest fraction, kaolinite was chosen to conduct the tube settlement and floc observation experiments.

The coal particles finer than 250 μm were screened out to conduct flotation experiments. And the particles of -1.4 g·cm$^{-3}$ density fraction were ground to finer than 45 μm to conduct the tube settlement and floc observation experiments. The laser particle size analysis (S3500, Microtrac, the US) showed that the $d_{80}$ of the -1.4 g·cm$^{-3}$ coal after ground is 36.4 μm.

Analytically pure grade kaolinite, PEO and $(NaPO_3)_6$ were purchased from Sinopharm Chemical Reagent Co. Ltd. The laser particle size analysis (S3500, Microtrac, America) showed that the $d_{80}$ of kaolinite is 4.8 μm. The PEO was dissolved in distilled water at 0.1 wt% and agitated on a magnetic agitator at 40 °C for 5 hours. Then the PEO solution was diluted to 0.01 wt% with distilled water. The $(NaPO_3)_6$ was dissolved in tap water at 5 g/L and then diluted to required concentration. Kerosene and 2-octanol were used as collector and frother in flotation experiments.

*2.2 Flotation experiment*

Flotation experiments were conducted in a 1.5 L XFD flotation cell using 50 g of coal. The impeller speed was fixed to 1900 r/min and the aeration rate was 0.69 cm/s. In each experiment, coal was agitated in tap water or $(NaPO_3)_6$ solution in the flotation cell for 2 min. Then required amount of 0.01 wt% PEO solution was added. Two minutes later, collector was added and a continued 2 min of conditioning was kept. Subsequently, frother was added and 0.5 min later the aeration started. All concentrates and tailings were filtered and dried for further analysis.

*2.3 Tube settlement experiment*

Settlement experiments for -1.4 g·cm$^{-3}$ density fraction of <45 μm fine coal and kaolinite were conducted in tube for quick and visual contrast. In each run of experiments, four kinds of slurry were prepared. Two kinds of medium were selected, namely tap water and $(NaPO_3)_6$ solution. Within each medium, PEO was added to one kind of slurry but not to the other one. In each experiment, 0.5 g of coal or kaolinite was introduced into 15 mL of the medium. The slurry was agitated in a beaker on a magnetic agitator at 500 r/min for 2 min. For the slurry with PEO addition, three drops of 0.01 wt% PEO solution (around 0.10 mL) were added with a dropper at this time. The PEO dosage was about 20 g/t. No matter whether PEO was added, a subsequent 2 min period of agitation was kept. Then the preparation of slurry was finished, four kinds of slurry were gently dumped into four separate tubes at the same time. The four tubes were inverted for three times and then put in a tube holder. During the settling process, photos of the four tubes were taken at specific time.

*2.4 Floc observation experiment*

Coal and kaolinite flocs in slurry were observed via the optical microscope (XSP-13CC, Batuo, China). The preparation of slurry was the same as that in tube settlement experiment. Small amount of agitated slurry was taken with a dropper from three different depths in the beaker. The slurry from different depth was respectively dropped on a glass slide and the cover glass was put on it. Then the glass slide was put on the object stage and five pictures at different positions were taken by the built-in CCD camera. The images were further analyzed by Images to obtain quantified data.

## 3. Results and discussion

*3.1 Flotation result*

Fig. 1 shows the flotation results. The flotation was significantly worsened with the addition of PEO when coal was floated in tap water. The combustible matter recovery experienced a steady decrease from 57.64% to 53.44% when the PEO dosage increased to 20 g/t and the concentrate ash content increased from 8.54% to 12.18%. This terrible result is assumed to be related to the nonselective flocculation of coal and hydrophilic minerals.

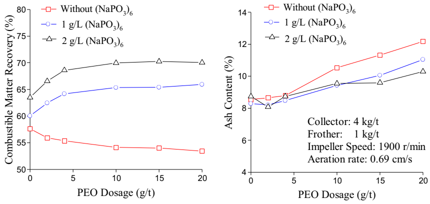


Fig. 1 Flotation result of coal as a function of PEO dosage in the absence and presence of $(NaPO_3)_6$

However, when the flotation was performed in $(NaPO_3)_6$ solutions, the flotation effect was greatly improved by PEO. When 20 g/t PEO was added in 2 g/L $(NaPO_3)_6$ solutions, the combustible matter recovery was increased by 6.54% with a slight increase of ash content of 1.54%. Therefore, it is considered that the presence of $(NaPO_3)_6$ helped to achieve the selective flocculation of fine coal particles by PEO. Whether this hypothesis is correct will be discussed later in the following sections.

*3.2 Tube settlement experiment*

The tube pictures of different coal slurry in the beginning and end of settling are shown in Fig. 2. The left picture taken at 2 min after settling indicated that the coal in No. 2 tube settled very fast with 20 g/t PEO addition. This is attributed to the fast formation of coal flocs induced by PEO. The slurry in No. 4 tube also seemed to be clearer than that in No. 1 and No. 3 tube. After settling for 60 min, the supernatant in No. 2 and No. 4 tube became clearer. And the supernatant in No. 1 and No. 3 tube kept turbid. This means that PEO could still flocculated coal particles in $(NaPO_3)_6$ solution, although the flocculation is slowed down or the floc size is decreased. As Gochin (1985) studied, the flocculation of coal by PEO is mainly caused by hydrophobic interaction while hydrogen bonding only plays an unimportant role. In $(NaPO_3)_6$ solution, the hydrogen bonding on coal interface is occupied by $(PO_3)_6^-$ and the hydration layer around coal interface was thickened. Hence, the decrease of flocculation efficiency in $(NaPO_3)_6$ solution is possibly caused by the longer attachment time between PEO molecules and coal interface due to the thicker hydration shell.

Fig. 3 shows the pictures of different kaolinite slurry at different settling time. What is quite different from the pictures of coal slurry is that the slurry in No. 4 tube settled very slowly. It kept turbid even after 60 min of settling, just the same as that in No. 3 tube. The slurry in No. 2 tube still settled the fastest, followed by No. 1. Therefore, the kaolinite in tap water all settled faster than that in $(NaPO_3)_6$ solutions, no matter whether PEO was added. This indicates that $(NaPO_3)_6$ has strong hindering effect on kaolinite flocculaiton, which has been shown by other researchers (Castellini et al., 2012). It is proved that $(NaPO_3)_6$ had a strong effect in increasing the absolute value of Zeta potential and the steric hindrance of kaolinite. Many researchers promoted that PEO flocculates phyllosilicate minerals mainly through hydrogen bonding (Mpofu et al., 2003; Mpofu et al., 2004) and the flocculation does not change at different pH. Fig. 3 agrees well with this assumption. The presence of $(PO_3)_6^-$ greatly weakened the hydrogen bonding around kaolinite interface. So PEO could hardly flocculate kaolinite particles. It is also promoted that the flocculation of quartz by PEO is due to the hydrogen bonding (Gong et al., 2010). Therefore, it is assumed that the flocculation of quartz was also prevented by $(NaPO_3)_6$.

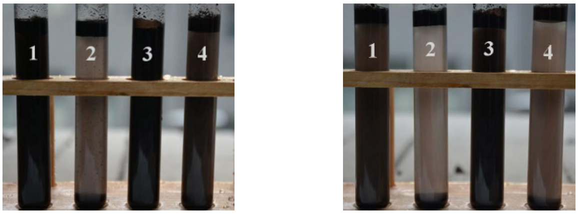

Fig. 2 Pictures of coal slurry tubes after 2 (left) and 60 (right) min of settling, the numbers on tube represent different slurry (1: coal in tap water without PEO; 2: coal in tap water with 20 g/t PEO added; 3: coal in 2 g/L $(NaPO_3)_6$ solution without PEO; 4: coal in 2 g/L $(NaPO_3)_6$ solution with 20 g/t PEO added)

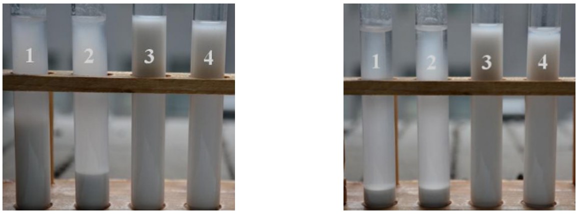

Fig. 3 Pictures of kaolinite slurry tubes after 2 (left) and 60 (right) min of settling, the numbers on tube represent different slurry (1: kaolinite in tap water without PEO; 2: kaolinite in tap water with PEO added; 3: kaolinite in 2 g/L $(NaPO_3)6$ solution without PEO; 4: kaolinite in 2 g/L $(NaPO_3)6$ solution with PEO added)

The settlement result throws light on the flotation result. In 2 g/L $(NaPO_3)_6$ solution, the same concentration as that in flotation, coal particles could still be flocculated by PEO whereas kaolinite could hardly be flocculated. Therefore, the particle size of coal was enlarged but kaolinite particles were highly dispersed almost at their inherent size. So the collision efficiency between coal particles and bubbles was increased, leading to the higher combustible recovery. However, the dispersed kaolinite particles could hardly collide with bubbles. It is important to note that the concentrate ash content also went up as PEO dosage increased. This may be attributed to two mechanisms. First, the water entrainment was enhanced due to the highly dispersion of hydrophilic minerals. Second, the larger flocs induced by higher PEO dosage caused the increase of mechanical entrapment of hydrophilic minerals.

*3.3 Floc observation experiment*

The images of coal particles in slurry are shown in Fig. 4. It illustrates that small flocs also existed in tap water due to hydrophobic interaction. With the addition of PEO, lots of large and loose flocs formed in the slurry and ultra-fine particles almost disappeared as Fig. 4 (B) shows. This is consistent with the clear supernatant in No. 2 tube in Fig. 2. It is worth noting that the similar but a little smaller flocs also existed in 2 g/L $(NaPO_3)_6$ solution if PEO was added, as shown in Fig. 4 (C). This is because the distance of hydrophobic interaction is larger than the hydration shell around the particles. The weakening of hydrogen bonding by $(PO_3)_6^-$ could not stop the flocculation of coal by PEO. Therefore, the flocs got smaller but did not disappear. A total number of 15 pictures were analyzed via the Analyze Particles plugin of Images. The result is shown in Table 1.

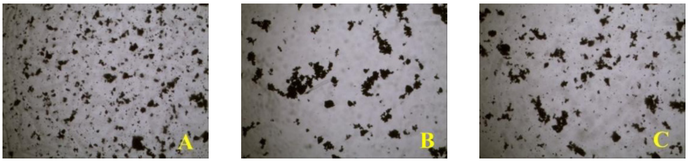


Fig. 4 Images of coal particles at 40 times magnification (A: coal in tap water without PEO; B: coal in tap water with PEO added; C: coal in 2 g/L $(NaPO_3)_6$ solution with PEO added)

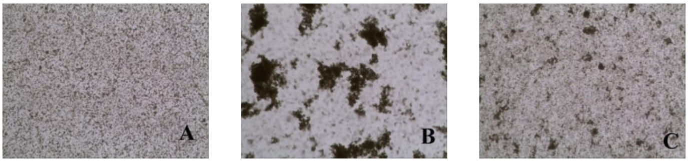


Fig. 5 Images of kaolinite particles at 100 times magnification (A: kaolinite in tap water without PEO; B: kaolinite in tap water with PEO added; C: kaolinite in 2 g/L $(NaPO_3)_6$ solution with PEO added)

Table 1 Particle size analysis through Images

| Mineral | | Coal | | | Kaolinite | |
|---|---|---|---|---|---|---|
| Medium | Tap water | Tap water | $(NaPO_3)_6$ solution | Tap water | Tap water | $(NaPO_3)_6$ solution |
| PEO addition | No | Yes | Yes | No | Yes | Yes |
| Number of particles | 7355 | 1940 | 4674 | 52749 | 4030 | 47661 |
| $D_{ave}$ [a] (μm) | 5.01 | 6.05 | 4.77 | 1.76 | 5.73 | 1.69 |
| $D_{max}$ [b] (μm) | 76.41 | 133.22 | 94.19 | 30.51 | 158.81 | 61.98 |
| $D_{t10}$ [c] (μm) | 44.23 | 82.87 | 59.77 | 11.00 | 98.45 | 33.56 |
| $D_{t50}$ [d] (μm) | 25.54 | 41.59 | 29.66 | 7.35 | 44.69 | 18.85 |

[a] The average diameter of all particles;
[b] The maximum diameter of all particles;
[c] The average diameter of 10 top large particles;
[d] The average diameter of 50 top large particles.

Fig. 5 shows the images of kaolinite in slurry at 100 times magnification. Kaolinite particles were very fine and highly dispersed in tap water as Fig. 5 (A) illustrates. When PEO was added, large and compact flocs emerged. However, when kaolinite particles were dispersed in $(NaPO_3)_6$ solution, the addition of PEO could hardly flocculate them. This is because the flocculation of kaolinite and other phyllosilicate minerals by PEO is through hydrogen bonding. Large amount of $(PO_3)_6^-$ ions around the particle interface greatly prevented the interaction between mineral and PEO molecules.

Table 1 shows the statistical result of a large amount of particles, which is more persuasive. In the case of coal, the average particle size almost did not change in different slurry due to the large proportion of ultra-fine dispersed particles. But the diameter of largest, top 10 largest and top 50 largest particles increased dramatically with PEO addition in tap water. The particles in $(NaPO_3)_6$ solution with PEO addition were also enlarged but to a smaller extent. The data for kaolinite are completely different. As PEO added in tap water, the average diameter of top 10 largest particles increased from 11.00 μm to 98.45 μm, a larger increase extent than that of coal. However, in $(NaPO_3)_6$ solution, the top 10 and 50 largest particles average diameters also increased by a very large extent. But it should be noted that the numbers of particles in kaolinite slurries were much larger than those in coal slurries, so the average diameters of top 500 largest kaolinite particles in tap water without PEO and in the $(NaPO_3)_6$ solution with PEO were also calculated. They were 4.62 μm and 6.12 μm respectively. It indicates that PEO can hardly flocculate kaolinite in the $(NaPO_3)_6$ solution.

## 4. Conclusions

The flotation of bituminous coal indicated that PEO could effectively improve the combustible matter recovery in the presence of $(NaPO_3)_6$. The improvement is attributed to the size enlargement of ultra-fine coal particles through hydrophobic interaction, resulting in higher collision efficiency with bubbles. However, the concentrate ash content also increased as PEO dosage, this may be due to the aggravation of gangue mineral entrainment and entrapment. The tube settlement and floc observation experiments proved that PEO flocculates coal mainly through hydrophobic interaction and the presence of $(NaPO_3)_6$ can only prevent the flocculation to a small extent. But the flocculation of kaolinite is mainly through hydrogen bonding, which can be greatly prevented by $(NaPO_3)_6$.

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# A classification device capable of being integrated to flotation columns and its classification performance

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**Abstract:** In order to discharge fine slime from the flotation tailings and make preparations for the possible individual recovery of coarse coal losses in flotation tailings, a classification device that can be integrated to flotation column was developed. It has no moving parts or extra occupations, and needs no electrical driving units as well. The experiments were carried out to study the influence of the operation parameters involving slurry volume $Q$, split ratio $R_f$ of the underflow, and the design parameters of the device on its classification performance. The results show that: the cutting size $d_{50}$ and comprehensive efficiency $\eta$ both decrease with the increase of $R_f$ while increase with the increase of $Q$; the sediment cone helps to increase the $d_{50}$ and improve the $\eta$; bigger inner tank has a coarser $d_{50}$; under proper conditions, the $\eta$ can be as high as more than 75%; for a certain structure, the $\eta$ increases almost linearly with the $d_{50}$.

**Keywords:** fine coal, classification, flotation tailing, flotation column, sediment cone, cutting size, comprehensive efficiency

## 1 Introduction

It is a usual practice for modern coking coal preparation plants to clean the full size-range of raw coal to realize the efficient utilization of coal resources. Flotation is the most widely accepted method for cleaning fine coal. But flotation presents a relatively poor collection on coarse particles of >0.25mm. Although classification equipment such as hydrocyclone is normally installed before the flotation equipment, some coarse particles still come into flotation process owing to various operation or maintenance problems. In fact, as is known, it is impossible to prevent >0.25mm particles to go into the overflow and then into flotation when the cutting size of the previous classification is 0.25mm or even 0.125mm. So, the recovery of coarse particles is always a problem must be faced with.

Many researchers have attempted to improve the recovery of coarse coal by various methods, including setting coarse coal recovery equipment such as TBS and RC, combining flotation columns with flotation machines (Xie 2009), adopting multi-stage flotation (Honaker and Monanty 1996, Tao et al. 2000, Kohmuench 2007, Gui 2012), adjusting operation or design parameters (Harris et al. 1992, Dashti and Nasab 2013), developing new flotation equipment (Yang and Wang 2012), selecting advanced reagent regime (Moxon and Jones 1986, Moxon et al. 1988, ), providing sufficient air flow and dense bubbles (Jameson 2005), and so on. But most of them are limited and not universally practical.

Coal preparation plants are increasingly realizing the necessity of recovering the coarse losses in flotation tailings to acquire maximum economic benefit. It is normal for some coal preparation plants in China to recovery coarse particles in flotation tailing employing high-frequency screen or settlement filtering centrifuge without selectivity and then mix them with either clean coal or medium coal according to their ash content. Fu et al. industrially separated >0.25mm fractions from flotation tailing and obtained clean coal of 8.29~8.87% ash with a yield of more than 60% by superfine grinding and flocculation flotation (Fu et al. 2005). Duan reported a multi-stage dense medium cyclone process for >0.067mm fraction in the flotation tailing after classification and desliming, recovered 50,000 tons of clean coal from 120,000 tons of flotation tailing in a coal preparation plant every year (Duan 2010).

Some scholars have also attempted to find an economic way to treat flotation tailings. Mao et al. developed a three-product separating and classifying cyclone to treat the flotation tailing experimentally and the ash content of >0.25mm fraction in its product is less than 10% (Mao et al. 2013).

 
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No matter which method is to be used to recover the coarse particles in flotation tailing, classification or desliming is the first step. Not all the flotation tailings is necessary to recover and the treating equipment is not always applicable for the initial concentration or original size distribution of the flotation tailings. Classification of the coarse and the fine or desliming is necessary. High frequency screen and hydrocyclone are the most commonly used pretreatment equipments for flotation tailing, but they occupy extra area, have the disadvantage of power consumption, and result in complex flowsheet. A classification device with inside overflow discharge and without moving parts or extra auxiliary facilities has been reported aiming at the classification of flotation tailings (Yang and Fan 2012). But the coarse losses into the fine products and the fine entrainment into the coarse products were both high.

This article is to develop a new classification device with periphery overflow discharge, aiming to conduct reasonable classification and make preparation for possible individual recovery of coarse coal losses in flotation tailings. It is capable of integrated to the bottom of flotation equipment. The design and operation parameters are to be optimized to acquire satisfactory classification on coal flotation tailings.

## 2 Methods

### 2.1 Design of the classification device with periphery overflow

A schematic of flotation column integrated with a classification device and the design of the experimental classification device are given in Fig.1. The basic design is to put a tank with conical bottom outside the lower part of flotation equipment (take flotation column as an example).

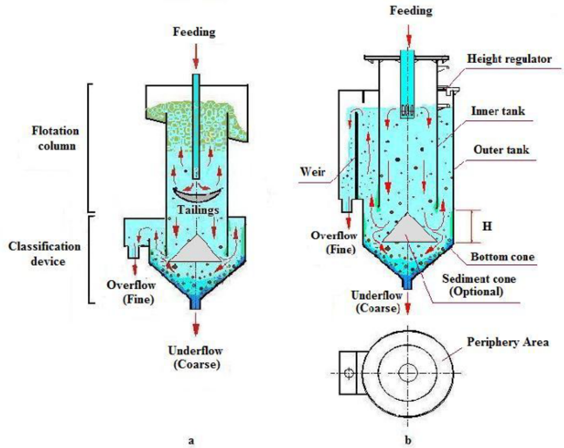


Fig.1 Grouped schematic of a flotation column and the classification device (a), and the design of experimental classification device (b)

After conducted flotation in the upper part of flotation column, the tailing slurry that does not go into the froth flows downward under its gravity and reaches the classification section. In this section, some slurry containing coarse particles is discharged through the underflow apex at the bottom of the cone, while more slurry turns upward and is ultimately discharged from the periphery area between the outer tank and the column. The coarse particles have high settling velocity, quickly settle down and are discharged into the underflow. The fine particles, on the other hand, have low settling velocity and tend to be discharged with the overflow. Even if some fine particles go down near the cone bottom, they still have more chance to turn upward because of the condensing effect of the cone.

To facilitate the set up of the experimental device, an individual classification device with an outer tank of $\Phi$400mm was designed, and an inner tank (a column) was put inside it. Two alternative inner tanks were prepared with different diameter: $\Phi$260mm and $\Phi$300mm, respectively. The ratio of the periphery area to the cross sectional area of the inner tank (i.e. flotation column) is 1.4 and 0.8, respectively. In order to fully understand the classification characteristics, the classification device was designed to be much taller than practically needed to find the proper height.

The relative height $H$ between the bottom edge of the inner tank and the outer tank as shown in Fig.1 is adjustable among 270mm, 185mm, and 85mm. To shorten the sediment distance of particles and prevent direct entrainment of fine particles into the underflow, a sediment cone can be assembled in addition. According to the previous literature, the cone with different angle was tested: 60°, 90°, and 120°, the length of the cone generator remaining constant (Yang and Fan 2013).

### 2.2 Experimental system and conditions

The volume $Q$ of the feeding slurry into the classification device is an important parameter deciding the cutting size of the classification. To ensure the settlement of coarse coal particles (for example 0.25mm, 1.35 g/cm$^3$) into the underflow, the maximum feeding volume was determined to be 2.5m$^3$/h. Another important operating condition is the split ratio of the underflow $R_f$ (or the split ratio of the overflow 1-$R_f$ ). Previous study has proved that the proper $R_f$ should be between 0.2 and 0.4 to obtain necessary classification (Yang and Fan 2012). The feeding solid concentration was fixed to be 30g/L by referring to the practical concentration of the flotation tailings.

The experimental system is composed of the classification device, an agitation vessel, and a feeding pump. The pump motor was controlled by a frequency converter to provide variable feeding slurry volume. During the experiments, the overflow and the underflow products were collected simultaneously. The split ratio $R_f$ of the underflow, i.e. the coarse products, was adjusted by the valve at the outlet of the cone bottom and calculated by comparing the underflow volume with that of both products. All products were dried, weighed, and screened.

The coal sample for the experiments was collected from the flotation tailing in a coal preparation plant, Shanxi Coking Coal Group, Co. Ltd. The coal samples were dried, mixed, subsampled and then packaged for use. A sieving analysis was carried out and the results are listed in Table 1.

Table 1 Sieving results of coal sample

| Size, mm | Weight, % | Accum. Weight, % | Ash, % | Accum. Ash, % |
|---|---|---|---|---|
| >0.5 | 9.93 | 9.93 | 14.81 | 14.81 |
| 0.5-0.25 | 14.38 | 24.32 | 32.91 | 27.66 |
| 0.25-0.125 | 15.66 | 39.98 | 27.7 | 27.68 |
| 0.125-0.074 | 29.24 | 69.22 | 23.32 | 25.58 |
| <0.074 | 30.78 | 100.00 | 25.63 | 25.60 |

## 3 Results and discussion

### 3.1 The effect of the operation conditions

On the classification device with $\Phi$260*mm* inner tank and $H$=185*mm*, the influence of $Q$ and $R_f$ on the classification was tested and the results are shown in Fig.2. Fig.2 shows that with the increase of $Q$, the partition curve skews right and downward. The entrainment of -0.074*mm* is as high as about 35% when $Q$ is 1.2$m^3/h$. This means that too little feeding may result in high entrainment. The $R_f$ has no influence on the right end of the partition curves, but smaller $R_f$ contributes to less hydraulic entrainment of -0.074*mm* materials in the coarse products (see the left end of the curves).

Compared with the classifying device with inside overflow discharge reported previously, this device with periphery overflow discharge shows much better classification performance by giving less fine entrainment in the underflow as well as zero coarse losses in the overflow.

### 3.2 The effect of the structure adjustments

Fig.3 shows the partition curves of the device with different inner tank diameter. The medium part of the partition curve moves right when the inner tank is smaller.

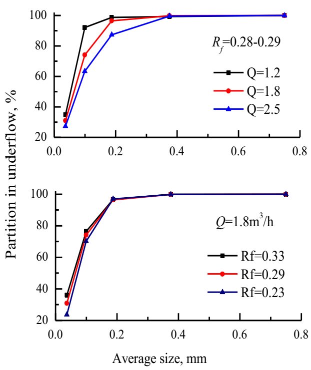


Fig.2 Effect of $Q$ and $R_f$ on the distribution curves

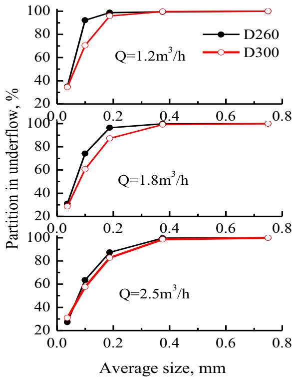


Fig. 3 Effect of the inner tank diameter on the distribution curves

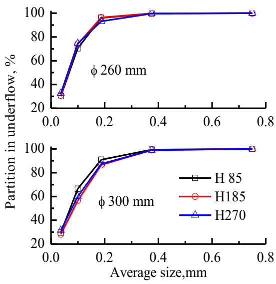


Fig.4 Effect of H on the partition curves at $Q$=1.8$m^3/h$ and $R_f$=0.28-0.29

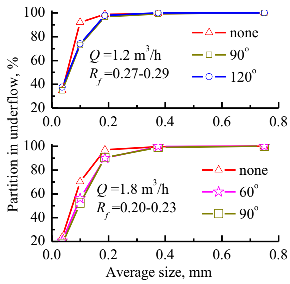


Fig.5 Effect of the sediment cone angle on the distribution curves

Fig.4 is the partition curves at different $H$. It indicates that provided a sufficient upward periphery area, $H$ has negligible influence on the classification. But if the periphery area is limited, for example for the inner tank of $\Phi$300*mm*, while $H$ is as high as 270*mm* or 185*mm,* particles of medium size might get a chance to come into the ascending flow by short pass before settling to the bottom cone. To decrease the total height cost of the device and ensure a relatively good classification, the inner tank was suggested to be around $H$=85mm.

Fig.5 is the partition curves with the presence and the absence of the sediment cone. Because the sediment cone hinders the fine particles to settle and entrain directly into the underflow, the fine entrainment is decreased, especially at 1.8$m^3/h$.

But it should be noted that the sediment cone could cause the upward flow in the periphery area to be less uniform, resulting in a high local velocity of the upward flow near the wall even several times higher than the average velocity (Yang and Fan 2013). It may cause the particles of medium size to get more chance to be carried into the overflow, and thus increase $d_{50}$.

**3.3 The cutting size and the comprehensive efficiency**

Cutting size $d_{50}$ and comprehensive efficiency $\eta$ were used to assess the classification performance of the device. The $\eta$ was obtained from the correctly-placed efficiency of oversize materials $E_c$ and the correctly-placed efficiency of undersize materials $E_f$, as shown in Equ.(1) (Xie et al. 2001).

$$\eta = E_c + E_f - 100 = \frac{\gamma_u U_c}{F_c} + \frac{100F_f - \gamma_u U_f}{F_f} - 100 \quad , \tag{1}$$

where $\gamma_u$ is the yield of the underflow, $U_c$ and $F_c$ are the weight percentage of the oversize material in the underflow and in the feedings, respectively. $U_f$ and $F_f$ are the weight percentage of the undersize material in the underflow and in the feedings, respectively.

Fig.6 to Fig.8 show the $d_{50}$ and the $\eta$ at different conditions. Fig 6 indicates that for a device with certain structure, $d_{50}$ and $\eta$ both decrease with the increase of $R_f$ while increase with the increase of $Q$. To acquire high efficiency, $R_f$ should be controlled to be less than 0.3.

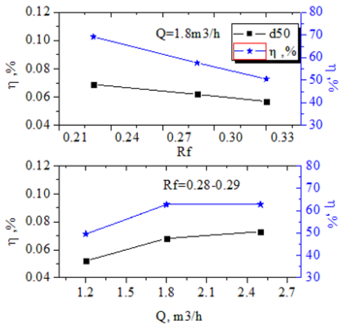


Fig.6 Effect of $Q$ and $R_f$ on $d_{50}$ and $\eta$

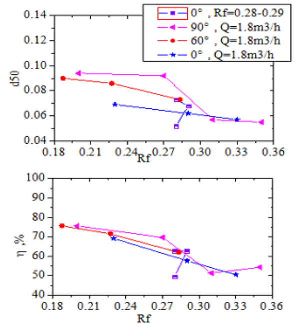


Fig.7 Effect of sediment cone on $d_{50}$ and $\eta$

Fig.7 indicates that the sediment cone helps to improve the $\eta$ of the device. With the presence of the sediment cone, the $d_{50}$ is coarser. This is because the sediment cone enables some particles of medium size to go upward into the overflow. At the same $R_f$, larger cone angle gives bigger $d_{50}$ and slightly higher $\eta$ as well. Under proper conditions, the $\eta$ can be as high as more than 75%.

Fig.8 shows that bigger inner tank has a coarser $d_{50}$. This is because it provides a broader periphery area and thus smaller velocity of the upward flow.

In summary, under proper conditions, the classification device can realize relatively satisfactory classification for flotation tailing.

Taking all of the classification results into account with 90° sediment cone, plot $\eta$ as a function of $d_{50}$ in Fig.9 and fit it with a linear relationship. It can be seen that for the certain structure, the $\eta$ increases almost linearly with the $d_{50}$.

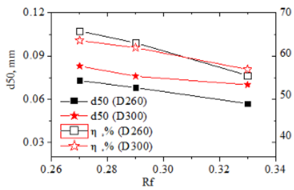


Fig.8 Effect of inner tank diameter on $d_{50}$ and $\eta$

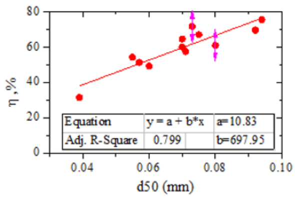


Fig.9 The relationship between $d_{50}$ and $\eta$

## 4 Conclusions

A classification device with periphery overflow was developed to classify the coal flotation tailings. Under proper conditions, the classification device can realize relatively satisfactory classification for flotation tailing by giving less fine entrainment in the underflow as well as zero coarse losses in the overflow. This makes it possible for coal preparation plants to treat the flotation tailing more readily, easily and flexibly.

The main conclusions are: the $d_{50}$ and $\eta$ both decrease with increasing $R_f$ while increase with the increasing $Q$; the sediment cone helps to increase the $d_{50}$ and improve the $\eta$; under proper conditions, the $\eta$ can be as high as more than 75%; for a certain structure, the $\eta$ increases almost linearly with the $d_{50}$.

**Acknowledgment**

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# Latest Development in the Study of the XJM-S Flotation Machine

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**Abstract**: Based on a study of the characteristic of the flow field in the XJM-S mechanically-agitated sub-aeration flotation machine by using computational fluid dynamics method, a description is made of the state of motion of fluid in the cell. Through CFD numeric simulation, the plots of fluid flow patterns, distributions of velocities at different cell profiles are obtained. In addition, the state of distribution and the laws governing the variation of the cell's characteristic parameters such as turbulence kinetic energy, intensity and dissipation have also been brought to light. Based on analysis of the methods applied in China and overseas countries for the design of large-sized flotation machines, the principles for the design of the XJM-S flotation machine are proposed as follows: maintaining a high turbulence intensity in the agitation zone; adoption of self-suction aeration mode; and use of shallow trough. The scale-up of the machine is made by observing the similarity criteria of maintaining geometric similarity of impeller-stator by keeping constant impeller peripheral speed, constant power and constant air streams. The analog and scale-up design of the XJM-S machine has met with satisfactory result. Lastly, the author points out the future research direction of the flotation machine.

**Keywords**: Coal cleaning Sub-aeration flotation machine Structure characteristic Numerical simulation Turbulent motion Similarity scale-up Research progress

Froth flotation is the major fine ore treating method. Among the metalliferous ore flotation machines used worldwide, the mechanically-agitated type accounts for 75% followed by flotation column (15%) and other types (10%) [1]. The majority of the flotation machines used in China are the mechanically-agitated type which shares about 80% followed by flotation column (15%) and jet cell (5%) [2]. Most of the machines currently in operation in China's coal cleaning plants are XJM-S mechanically-agitated type which accounts for approximately 70%. Such cells vary in more than 10 different volumes ranging form 4 $m^3$ up to 90 $m^3$.

Over the recent years, along with the increasing demands on the development of large, high-efficiency and energy-saving coal cleaning plants, the XJM-S cell tends to become still larger in size, and the way for the definition of its crucial parameters is shifting from previously empirical design to similarity scale-up approach. The efforts have met so far noticeable success.

## 1. Progress of Theoretical Research Work

Extensive study has been made by numerous research workers on flotation micro-process, kinetics and flotation machine designing method, in order to bring to light the objective laws of flotation process, improve flotation performance and master the way for flotation machine design. The research-derived achievements are of theoretical guiding significance for the optimization of parameters and enlargement of flotation cells.

The work so far conducted involve: the study by Zeng Kewen and Yu Yongfu on particle-bubble collision, adhesion and de-attachment processes in a bid to make known the effect on flotation performance produced by intensity of turbulent flow [3]; and the study by Cheng Hongzhi et al on the effect of hydrodynamic parameters on particle-bubble collision, adhesion and de-attachment rates, based on turbulent flow diffusion and collision mechanisms. Through the study by Cheng Hongzhi, a description is made using a mathematical model of the coefficients of particle-bubble collision and de-attachment rates in relation to hydrodynamic parameters like number of air streams and agitation Reynolds number [4].

An analysis is made recently by Liu Chunyan et al of the kinetic characteristics of flow field in XJM-S flotation cell by using the computational fluid dynamics method (CFD). Through numerical simulation, the pressure distribution pattern in the impeller-stator region, the fluid-line in cell's different sections and velocity distribution pattern have been obtained. In addition, the distribution and variation patterns of the characteristic parameters such as turbulent kinetic energy, intensity and dissipation, etc have been brought to light [5]. Through flow-line analysis, it is observed that: the fluid in XJM-S cell assumes a W-formed

 
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stereo motion pattern with a lateral eddy flow at the bottom and a steady flow at the top. The accords with the demands for flotation process (See Fig. 1); after passing through the stator, the fluid is uniformly distributed across the horizontal cross section of the cell under the guiding effect of the stabilizing plate. This indicates that the XJM-S cell is rational in structural design (See Fig. 2); analysis of the characteristic parameters of turbulent flow shows the turbulent kinetic energy, intensity and dissipation assume a symmetrical distribution around the center of the stator (See Fig. 3); the agitating zone of the XJM-S8 version is spaced less than 0.7-m from the cell bottom and above this zone is a relatively stable flotation separation zone. The agitation intensity in the agitation zone is 6 times as high as that in the separation zone (See Fig. 4); the turbulence intensity is in direct proportion to the impeller's peripheral speed (See Fig. 5); and by taking a constant peripheral speed of the impeller as the XJM-S cell similarity scale-up criterion, a similar turbulence intensity can be guaranteed. Through numerical simulation of the cell's flow field, more lights are shed on the flow field characteristics of the XJM-s cell. This provides a theoretical basis for the study and optimum design of the same types of cells.

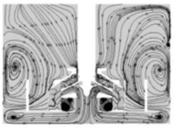

Fig. 1 Cell's Longitudinal Profile Flow Pattern

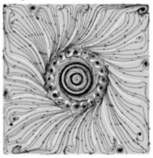

Fig. 2 The Cell's False Bottom Horizontal Profile Flow Pattern

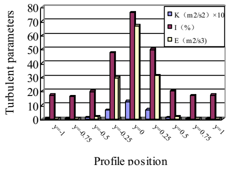


Fig. 3 Distribution of Lateral Turbulent Flow Characteristic Parameters inside Cell

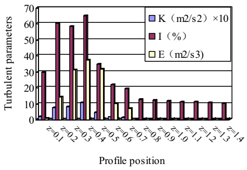


Fig.4 Distribution of Longitudinal Turbulent Flow Characteristic Parameters inside Cell

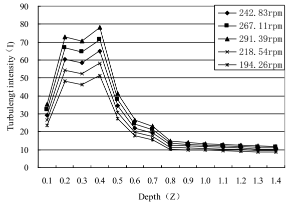


Fig. 5 Variation of Turbulent Flow Characteristic Parameters with the Change of Impeller Speed

**2. Design of Large-sized Flotation Cell**

Owing to the fact that the flow behavior of pulp in cell is quite complicated, the design and scale-up of flotation machine is still made using empirical formula as the basis both at home and abroad, and a generalized design method is still unavailable.

A.J.Jonnetis, Finland, described the design features of the 100~300-m$^3$ Tank Cell developed by Outokumpu Co. in a published literature [6]. Through improvement of the structural design of impeller and stator, 2 kinds of flow mechanisms, i.e. free flow mechanism and multiple mixing mechanism can be produced in the cell for treating respectively coarse ore and fine ore.

A. Vibert [7] of the US proposed a set of hydrodynamic parameters for the scale-up of the Wemco flotation machine and for giving guidance to the R&D of the Wemco 1+1 cell and Wemco Smart Cell ™. The hydrodynamic parameters proposed include: airflow velocity per unit froth surface area, residence time of air and pulp in disperser region, disperser's power intensity, cell's circulating intensity, velocity of pulp in standpipe, and air flowrate. The volume of the Wemco Smart Cell has now reached a maximum of 250-m$^3$.

The large-sized sub-aeration mechanically-agitated flotation cells with a volume of 130~200-m$^3$ have been developed by Shen Zhengchang and co-workers [8] using the empirical formula based on such scale-up factors as impeller diameter and linear speed, and agitation Reynolds number.

The XJM-S series self-suction mechanical agitation machines specifically designed for fine coal flotation have been developed by Cheng Hongzhi [9] and co-workers based on the following similarity criteria: under the prerequisite of maintaining a impeller-stator geometrical similarity, keep a constant impeller peripheral speed as the kinematic similarity criteria so as to maintain a similar state of motion; take the constant of air flowrate index and stirring power index as the dynamic similarity criterion in order to maintain a similar aeration rate and agitation intensity.

**3. XJM-S Flotation Cell Design Conception**

The XJM-S cell is a kind of self-suction sub-aeration cell which is designed in line with China's specific conditions as the coals differ widely in floatability and diversified coal cleaning processes are applied in coal cleaning plants. The cell developed proves to be highly adaptable to variation of coal floatability, pulp concentration and size range.

**3.1 Design principle**

(1) Maintaining a higher turbulence intensity and a thoroughly mixed flow pattern in the agitation zone

The function of the agitation zone is to disperse the air into micro bubbles so as to keep the coal particles in a suspended state and provide the opportunity and kinetic energy for air bubbles to collide with coal particles. Therefore, a higher turbulence intensity is required under this condition. Maintaining a complete mixed flow pattern can lead to uniform distribution of coal particles and air bubbles in the agitation zone, and a higher utilization rate of cell volume.

(2) Under a similar energy consumption condition, the self-suction mode is preferable to external aeration mode

The use of the former mode may lead to simplification of flotation process, reduction of number of equipment and convenience in operation and management. Moreover, a better air dispersion result can be expected because the air is sucked into cell under the effect of air entrainment. When the latter mode is used, the use of a blower is necessitated. The total power of the 2 modes is nearly the same [10].

(3) A bank of cells has 3~4 troughs operating in series

As the XJM-S flotation machine works with a thoroughly mixed flow pattern in its agitation zone, it is appropriate for the machine to work with 3~4 troughs arranged in tandem instead of the use of a single one. As evidenced by analysis of pulp flow regime and practical experience, such an arrangement may facilitate the re-adjustment of production quota and obtain a higher treating capacity per unit volume.

(4) Shallow trough

The use of shallow troughs offers a host of advantages including higher yield of flotation concentrate, higher adaptability to coarser feed size and higher air consumption.

**3.2 Structural features**

The structural design of the XJM-S flotation cell is schematically illustrated in Fig. 6. The special features of the cell are as follows: 1) The false bottom sucks downward the pulp flowing around. In this way, the machine is made highly adaptable to variation of feed coal property because a higher capacity can be expected when treating an easy-to-float coal and the pulp can undergo repeated circulation in cell in case of treating difficult-to-float coal; 2) The combined use of double-layer umbrella-shaped impeller and stator together with the false bottom-stabilizing plate combination, makes it possible for pulp to assume a W-formed flow pattern. This accords with the requirement for flotation process. In addition, this may lead to reduced power consumption and increased aeration capacity; 3) The cell has a rectangular cross section, with a large-volume agitation zone and capture zone. Other attractions include high capacity and less occupation of floor space; 4) The flotation system is greatly simplified due to use of self-aeration mode; 5) Each trough is provided with a reagent dosing point for flexible re-adjustment of reagent regime.

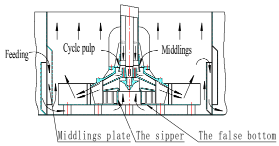


Fig. 6 Schematically Diagram Showing the Structural Design of XJM-S Flotation Cell and Pulp Flow Pattern

**3.3 Design of key parameters of XJM-S28 cell**

The 28m$^3$ cell's key parameters are designed using similarity scale-up method according to the established similarity criteria.

**1) Structural parameters**

Geometrical similarity criteria

$$\begin{cases} D/De = 0.237 \\ H = 0.6 + 1.2D \\ V = H \cdot \pi / 4 \cdot D_e^2 \end{cases} \tag{1}$$

Where

D – Impeller diameter, m

De – Diameter of equivalent circle area of cell's horizontal cross section, m

H – Cell depth, m

V – Cell volume, m$^3$

The empirical coefficient of the XJM-S version is taken as the constant in the formula.

At a cell volume of V=28-m$^3$, the following parameters can be derived using Equation 1:

D=1.050(m)，De=4.430(m)，H=1.860(m)

The cell's horizontal cross section is square-shaped with a computational cell side length of L=3.926-m.

**2) Dynamic parameters**

Kinetic and dynamic similarity criteria:

$$\begin{cases} U_2 = 8.90 \\ N_{Q_a} = Q_a / (ND^3) = 0.075 \\ N_P = P/(\rho N^3 D^5) = 1.60 \end{cases} \tag{2}$$

Where:
$U_2$ – Impeller's peripheral linear speed, m/s
N – Impeller speed, r/s
P – Agitation power, kW
ρ – Pulp density, $10^3 \cdot kg/m^3$
$N_{Q_a}$ – Number of air streams, dimensionless
$N_P$ – Number of power, dimensionless
The empirical coefficient of the XJM-S version is taken as the constant in the equation.

By substituting the structural parameters as derived above into expression (2), the following values are obtained: $N=U_2/(\pi D)$=2.698r/s, $Qa$=0.075×$ND^3$=0.234$m^3$/s, $P$=1.60×$(\rho N^3D^5)$=40.1kW.

The calculated aeration rate per unit area is $qa$=4$Qa/(\pi D_e^2)$×60=0.91$m^3/m^2$.min.

According to the result of test made on the performance of the XJM-S28 flotation machine with freshwater, the design error is less than 2.5% as evidenced by a comparison of the actually measured key data with the design values. For the result of the comparison, refer to Table 1 [11].

Table 1 Comparison of Actually Measured and Designed Values of Key Parameters of XJM-S28 Flotation Cell

| Key Parameter | Data | | Relative Error, % |
|---|---|---|---|
| | Designed | Actually Measured | |
| Impeller peripheral speed, m/s | 8.90 | 9.07 | 1.91 |
| Agitating power, kW | 40.1 | 40.9 | 1.96 |
| Aeration rate, $m^3/(m^2 \cdot min)$ | 0.91 | 0.89 | 2.25 |

The XJM-S28 flotation cell was first introduced by Luliangshan Coal & Electricity Co., Huozhou City, Shanxi Province, for use in its coal cleaning plant. The cell works with a pulp throughput of 900$m^3$/h, a pulp concentration of 70~90g/l. The flotation result is listed in Table 2. When used for treating either easy-to-float or difficult-to float coal, the cell can operate with equally remarkable performance.

Table 2 Performance of the XJM-S28 Flotation Cell for Treating Easy-to-float and Difficult-to-float Coal

| Item | Feed Ash, Ad% | | Concentrate Ash, Ad% | | Tailings Ash, Ad% | | Concentrate Yield, % | | Perfection Index, % | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Range | Avg. | Range | Avg. | Range | Avg. | Range | Avg. | Range | Avg. |
| Easy | 16.15-19.16 | 17.30 | 8.15-10.37 | 9.21 | 50.60-70.04 | 58.32 | 78.49-86.74 | 83.33 | 43.25-56.57 | 46.98 |
| Difficult | 30.14-32.89 | 31.53 | 9.40-12.56 | 11.02 | 40.66-57.87 | 47.97 | 25.46-58.32 | 43.18 | 26.96-53.37 | 40.66 |

## 4. Perspective of Research Work

**1) Further enlargement of cell size**

The XJM-S flotation machine developed has currently a cell volume of 60-90-$m^3$. A bank of 3~4 cells can attain a pulp throughput up to 1500~2400$m^3$/h. The designed agitation power per cell is 69-110 kW for catering to the needs of the development of large, energy-saving and high-efficiency coal cleaning plant.

**2) Optimization of key cell parameters through flow field numerical simulation**

Further work will be made to bring to light the laws governing the motion of pulp in cell by using computation fluid mechanics method and particle image velocity measuring technique (PIV) so as to improve cell structure and flotation performance.

**3) Further enhancement of flotation process automation level**

Work will be directed toward R&D and integrated use of sensitive testing and measuring elements and devices with an aim to realizing on-line liquid level and concentrate ash measurement to effect automatic control of flotation cell and process.

**5. Concluding Remarks**

The XJM-S flotation machine has become serialized in design, and different sizes of such cells have found widespread application in coal cleaning plants, bringing forth remarkable economic and social results. Through further R&D work, the use of such cells can gradually cater to the needs for the development of China's coal cleaning sector.

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# The Magnification Principles of the FJCA Series Coal Jet Flotation Cells

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**ABSTRACT**

The FJCA series jet flotation cells, which have the independent intellectual property rights of China, are different from the conventional mechanical-agitation flotation cells. Therefore, the magnification practice of the FJCA series jet flotation cells shall not simply copy from the conventional approach of the mechanical-agitation flotation cells. Instead, new magnification principles for the jet flotation cells need to be developed.

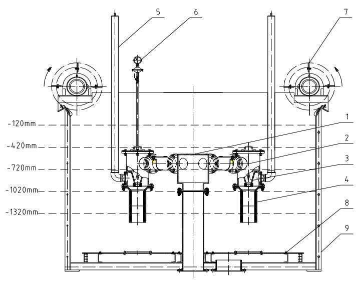
Three principles of the magnification of the FJCA series jet flotation cells are introduced in this article: 1. the similarity of the fluid dynamics and kinetics of the coal slurry; 2. the similarity of the geometries of the aeration agitator; 3. the similarity of the boundary parameters of the flowing velocity of the slurry. The first principle, which employs the circulation flowrate per unit volume as the base parameter of the similarity of fluid dynamics and kinetics, is considered as the most critical one for the magnification of the FJCA series jet flotation cells.

**Keywords**

jet flotation cells, magnification principle, circulation flowrate per unit volume, nozzle outlet diameter, similarity of the kinetics, similarity of the geometries, similarity of the boundary parameters

## Introduction

The FJCA jet flotation cells, with China's independent intellectual property rights (Patent No.: ZL200810054981.2), was developed in 2006 based on the design of the previous jet flotation cells.




1—distribution chamber;
2—jet cell;
3—nozzle;
4—nozzle throat tube;
5—air inlet tube;
6—pressure gage;
7—froth scraper;
8—false bottom;
9—chamber body

Figure 1 Schematic diagram of the FJCA series jet flotation cell

The FJCA jet flotation cell, which typically consists of four flotation chambers (Fig.1), utilizes direct fluid flow. At the bottom of each floatation chamber, four aerating agitators are installed symmetrically along the diagonal lines. Each aerating agitator is comprised of jet cell, nozzle, nozzle throat tube, and air inlet tube, etc. The pressurized slurry is injected into the distribution chamber through the nozzle with vanes at high speed. Due to the accelerated speed, a negative pressure area is created around the nozzle outlet, which forced the air to enter the air inlet tube and to be blended in the slurry jet. The spinning, aerated slurry is then ejected through the throat tube tangentially to the false bottom. As such, the aerating agitation is achieved.

Known benefits of the FJCA jet flotation cell include simplified structure, long service life, accurate selectivity, and less consumption of chemical agents and power, etc. However, during the technical appraisal meeting in 2010, it was observed that the maximum volume of a single cell was only 20m$^3$,



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longer serve the rapid development of coal industry. Therefore, the magnification of jet flotation cells becomes an urgent task to accomplish.

In spite of the great achievements in the research of flotation technologies, the design and magnification of the flotation cells still heavily rely on empirical formulas. There has not been a detailed, universal, and accurate method developed.

Due to its popularity, great efforts have been made on the magnification of the mechanical agitation flotation cell over the past half century. A number of similar guidelines were developed based on the fluid condition in the flotation devices. These guidelines usually consider the diameter and the rotating speed of the impeller as basic parameters. The FJCA jet flotation cells are categorized as non-mechanical agitation type, since there are no impellers installed. Thusly, the magnification of the FJCA jet flotation cells shall follow new principles. Three magnification principles of the FJCA series coal jet flotation cells have been introduced in this paper.

## 1. Similarity of Fluid Dynamics and Kinetics

### 1.1 The determination of unit volume circulation capacity $Q_0$

The hydraulic agitation process in the jet flotation cells is achieved by the circulating slurry, of which the capacity and intensity (pressure) define the fluid flow condition. The fluid flow condition is always complex due to the three-phase interactions between gas, liquid and solid, which directly impacts the uniformity of the aerating agitation, while the agitation uniformity is an important performance indicator of the flotation process.

The static aerating agitator is the core component of a jet flotation cell. The nozzle diameter and operating pressure of the aerating agitator have been optimized by industrial paired comparison experiments.

The paired comparison experiment is an experimental methodology using two identical FJCA20-4 jet flotation cells operated by one operator under different nozzle pressure, while employing the same coal slurry and flotation agent dosage. The flotation perfection index obtained from 5~6 paired tests was evaluated by t-test statistical method, indicating the optimum working pressure was 0.15MPa, compared with 0.12MPa and 0.18MPa.

On the basis of the previous finding, another set of paired comparison tests were set to determine the optimum nozzle diameter while the operating pressure was fixed at 0.15MPa.

According to conical nozzle flow equation in fluid dynamics:

$$Q_1 = 3600\mu \frac{\pi d^2}{4} \sqrt{200gH} \tag{1}$$

$Q_1$ — volumetric flowrate passing through a single nozzle, $m^3/h$;

$\mu$ — flow coefficient, $\mu=0.94$;

d — diameter of the nozzle outlet, m;

g — gravitational acceleration, $m/s^2$;

H — nozzle operating pressure, MPa.

$Q_1$ is the volumetric flowrate passing through each individual nozzle. Since each aerating agitator is equipped with one nozzle, the total capacity of the whole flotation chamber should be $Q=4Q_1$.

The circulation flowrate per unit volume $Q_0$ ($m^3/(m^3\cdot h)$) is defined as the ratio of the optimized circulation flowrate of each chamber, Q and the chamber volume, V,

$$Q_0 = \frac{Q}{V} = \frac{4Q_1}{V} \tag{2}$$

Since the circulation flowrate under the proper operating pressure has been optimized, enhanced operating performance can be achieved. The performance of the aerating agitator determines the flow condition of the gas-liquid-solid three-phase coal slurry. Thus, the circulation flowrate per unit volume $Q_0$ is employed as the base parameter, or the most fundamental indicator in the magnification of FJCA coal jet flotation cell.

**1.2 The determination of the cross-section shape of the flotation chamber**

The separation performance is largely affected by the cross-section shape of the flotation chamber. Thusly, the magnification of the flotation cell is actually accomplished upon the optimization of the cross-section shape of the flotation chamber.

In the first phase of the reconstruction project in Panyi coal preparation plant in China, the FJCA20 jet flotation cells with inverted trapezoidal cross-section were selected. During the second phase of the project, rectangular cross-section was installed (Fig.2). The depth of the cell of the newly installed flotation cell increased by 16.7%, while the volume and the bottom area remained unchanged. As a result, the separation selectivity was significantly improved. The width of the chamber also decreased, which reduced the horizontal travel distance of the froth layer by 25%, which facilitated the collection of the concentrates.

Figure 2 Cross-sectional shape of the flotation chamber.

Fourteen sets of paired experiments were conducted on the flotation machine No. 4108 with rectangular cross-section and flotation machine No. 4011 with inverted trapezoidal cross-section in Panyi coal preparation plant. The difference between the two flotation cells was assessed using statistical method (Table 1).

Table 1. Paired comparison test result of two cross-sectional shapes

| Index | No. 4108 (rectangular) | No. 4011 (invertedtrapezoidal) | Differential (No. 4108-No. 4011) | Prominent difference | T test, % |
|---|---|---|---|---|---|
| Rougher clean coal ash, % | 14.34 | 13.48 | 0.86 | Yes | 99.0 |
| Tailings ash, % | 73.41 | 63.28 | 10.13 | Yes | 99.5 |
| Clean coal yield, % | 65.36 | 57.33 | 8.03 | Yes | 99.5 |
| Combustible recovery, % | 85.62 | 75.46 | 10.16 | Yes | 99.5 |
| Clean coal ash recovery, % | 27.16 | 22.75 | 4.41 | Yes | 99.0 |
| Flotation perfect index, % | 58.46 | 52.71 | 5.75 | Yes | 99.0 |

The average ash content of the flotation feed in Panyi coal preparation plant is 34.44%. The fine particles (<0.03mm) account for as high as 53.73% of the feed, with an ash content of 54.28%. The flotation feed is uncommon in China. As a result, the rougher-cleaner process was utilized in the flotation circuit. The ash content of the clean coal in rougher circuit was required to be below 15%. As per China coal industry standard MT180: *Method for the evaluation of flotation process in Coal Preparation Plant*, when the fines with similar properties were tested using different equipment or circuits, the flotation perfect index should be considered as the indicator of the circuit performance.

Flotation perfect index:

$$\eta_{wf}=E_j-E_w \qquad (3)$$

combustible recovery of the flotation clean coal:

$$E_j = \frac{\gamma_c(100 - A_c)}{100(100 - A_f)} \times 100\% \qquad (4)$$

ash recovery of the flotation clean coal:

$$E_W = \frac{\gamma_C A_C}{100 A_f} \times 100\% \qquad (5)$$

$A_c$ —flotation clean coal ash conten, %;
$A_f$ — flotation feed ash content, %;
$\gamma_c$ — flotation clean coal yield, %, $\gamma_c = (A_t - A_f)/(A_t - A_c) \times 100\%$;
$A_t$ — flotation tailings ash content, %.

As shown in Table 1, the optimized rectangular cross-section shape has a significant beneficial effect on the separation performance. As a result, the rectangular cross-section shape is applied to all the FJCA series jet flotation cells currently in use.

The separation efficiency of the flotation machines depends on the performance of aerating agitator. The FJCA36-4 jet flotation machines with the cell depth of 2.3m was manufactured based on the magnification principle of jet flotation cells. The FJCA36-4 were evaluated and tested in 2014 according to China coal industry standard MT/T652 *Test method and decision rule for coal flotation cells with water-only*. The test results are shown in Table 2.

(1) Maximum aeration quantity per unit area, $\bar{Q}$>1.0m$^3$/(m$^2$·min) , which indicates the "potential capacity" of the unit.

(2) The aeration uniformity coefficient is as high as 91.74% at the testing plane, 120mm below the liquid surface.

As the testing depth increases, the aeration uniformity coefficient decreases due to the agitation turbulence. However, the value is still as high as 83.26% at the depth of 1320mm, which indicates an ideal aeration uniformity.

Table 2. The aeration index of the FJCA36 jet flotation cells

| Aeration Index | Depth under static liquid surface, mm | | | | |
|---|---|---|---|---|---|
| | 120 | 420 | 720 | 1020 | 1320 |
| Aeration uniformity factor K, % | 91.74 | 82.49 | 81.32 | 80.37 | 83.26 |
| Average aeration quantity per unit area $\bar{Q}$, m$^3$/(m$^2$•min) | 1.05 | | | | |
| Aeration Volume Utilization Factor F, % | 100.00 | | | | |

(3) The aeration volume utilization coefficient was measured to be 100%, which indicates the existence of air bubbles in the entire measured space in the flotation cell.

In addition, a regression relationship with tight correlation between the unit area aeration quantity and the depth of the aerating agitator was established by performance tests with clear water only.

$$\bar{Q} = a - 0.21H \qquad (6)$$

$\bar{Q}$ — aeration quantity per unit area, m$^3$/(m$^2$·min);
H — immersion depth (distance from nozzle throat outlet to liquid surface), m.

Each 0.1m increment in the immersion depth leads to a 0.021 m$^3$/(m$^2$·min) decrease in the unit area aeration quantity. The flotation chamber gets deeper as the flotation cell get magnified. The equation has provided reliable prediction of the unit area aeration quantity.

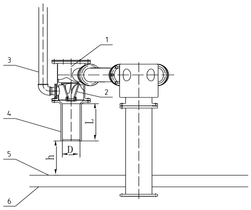
Lastly, the three-product heavy medium cyclones have been widely used as the major separation equipment in coking coal preparation plants in China. The bottom size for effective separation has long been controlled under 0.25mm. Therefore, the top size of the feed to the FJCA jet flotation cells is set to 0.25mm. Due to the finer feed size, coarse particle settlement on the bottom of the rectangular cross-section chamber has not yet been observed.

## 2 Similarity of Geometries

The similarity of geometries means the major components of the flotation cells are similar in both shape and dimension. As for the FJCA jet flotation separator, the aerating agitator is considered as a core component. The research on the agitator was conducted in a 6m$^3$ flotation cell based on the research against the XPM-8 jet flotation machine technically appraised in 1980s. The FJCA series has inherited the structural parameters of the nozzles with vanes from the XPM-8. As the flotation machines magnifies, the circulation quantity increases proportionally as the unit volume circulation flowrate remains unchanged. According to Equation 1, the nozzle diameter can be calculated using the equation below:

$$d = \sqrt{\frac{Q_1}{900\pi\mu\sqrt{200gH}}} \qquad (7)$$

When nozzle diameter is determined, the structural parameters of the jet chamber can also be finalized. Three parameters were optimized through tests (Fig.3).



1-jet cell;
2-nozzle;
3-air inlet tube;
4-nozzle throat;
5-false bottom;
6-bottom of experiment chamber

Figure 3. Structural parameters of the aeration agitator optimization

① D Nozzle throat inner diameter

The nozzle throat ensures the sealing of the spinning slurry ejected from the nozzle, which generates sufficient pressure drop and thus proper aeration quantity. Excessively small D will increase the frictional resistance of the slurry jet, while excessively large D will deteriorate the sealing.

② L Throat length

Excessively small length deteriorates the sealing, while excessively large L elevates the frictional resistance.

③ H Distance from throat pipe outlet to false bottom

When H gets too small, meaning the aeration agitators immersed too deep, which decreases the unit area aeration quantity. On the contrary, H too large, means the mixing force on false bottom will be weakened and result in the settling of coarse particles.

The coefficient of the geometric similarity of the aeration agitating devices is shown in Table 3.

Table 3. Coefficients of geometric similarity of aeration agitating device

| | |
|---|---|
| Ratio of throat inner diameter D and nozzle outlet diameter d | m=D/d |
| Ratio of throat length L and throat outlet diameter d | n=L/d |
| Ratio of the height between throat outlet and false bottom H and nozzle outlet diameter d | p=H/d |

**3 Similarity of Boundary Parameters**

The similarity of the boundary parameters means the flow status of the flotation feed is identical to the discharge. That is, to be specific, the coal slurry movement along the width of the flotation chamber at a low and constant velocity.

Feed box is installed at the head of the first chamber of each flotation machine, a baffle plate is mounted in the feed box to prevent the slurry from directly discharging into the flotation chamber. Since the FJCA jet flotation cell employs direct-flow pattern, wide circulation holes are opened at the chamber bottom for coal slurry flowing through the adjacent chambers, including the last chamber and the tailing collecting box. The opening size of the circulation hole is determined based on the assumption that the horizontal velocity of the slurry flow is less than 0.15m/s.

Lastly, in FJCA series jet flotation cells, the coal slurry flows directly from the feed box located at the first cell to the tailing collecting box at the last cell. It is the adjustable overflow weir in the tailing box controls the pulp level of the entire flotation machine. Thusly, the width of the overflow weir needs to be sufficient. The following rectangular thin wall weir equation was used to analyze the weir width.

$$Q = mB\sqrt{2g} \cdot h^{3/2} \quad (8)$$

Q — tailing flow rate, $m^3/h$;
m — coefficient of flowrate;
B — weir width, m;
g — gravitational acceleration, $m/s^2$;
h — pressure head above the weir, m.

The pressure head on the weir determines the pulp level in the flotation cell. When the flow rate of the overflow Q remains constant, the wider the weir, the narrower the varying range of the pulp level, which allows the flotation machine to be more adaptable to the waving feed rate.

**4 Conclusion**

Through long-term industrial study and systematic research, a series of scientific magnifying principles have been developed and applied. Nevertheless, the principles need to be further improved and optimized along with the times.

In China, about 300 sets of FJCA series jet flotation cells have been manufactured since 2008, in which 69 sets which were designed based on magnification principles, including FJCA28, FJCA32 and FJCA36. FJCA46 has already been selected in the engineering designs and can be expected to be manufactured this year. Technical design for FJCA52 and FJCA60 has also been completed.

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# Flotation of a Waterberg Coal using Release Analysis

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**ABSTRACT**

This paper evaluates the flotation response of a Waterberg coal using the release analysis technique. A bulk sample was taken from the mining area and prepared using the drop shatter and wet tumble technique. Samples were then screened into appropriate size fractions for feed to the flotation cell. A total of six different size fractions were tested in order to determine the optimal size fraction for flotation in the coal handling and process plant (CHPP).

Results indicated that operating on the -0.25+0.045 mm size fraction produced the highest yield over the flotation bank, however processing of the -1.0+0 mm material produced the highest yield overall. The -0.25+0.045 mm size fraction was selected as the optimal size fraction since the -1.0+0.25 mm material could be more economically processed in a conventional spirals and teetered bed separator circuit.

The capital and operating costs of the flotation circuit were determined for processing the -0.25+0.045 mm size fraction. These costs were weighed up against the revenue generated by producing a semi-soft coking coal, a Richards Bay Coal Terminal (RBCT) export coal, or an Eskom grade steam coal. Interestingly, production of an Eskom grade steam coal was found to maximise the project business case in the current economic environment.

**KEY WORDS**

Coal Flotation, Waterberg, Release Analysis, Coal Fines

## INTRODUCTION

With the ever greater depletion of coal reserves in the Witbank basin, South African coal mining companies are increasingly turning to new deposits in the Waterberg area to fill the declining production. Infrastructure in the Waterberg region is however limited and a significant capital outlay is required in order build a new mine, together with the associated infrastructure, such as roads and rail.

Furthermore, water supply is highly constrained in the area, and projects are under increasing pressure to minimise water losses. Costly technologies such as slimes filtration are a common addition to flow sheets in the Waterberg area; however these dewatering technologies contribute very little to the project economics. This is particularly true in cases where the slimes fraction is discarded and no revenue is derived from it.

Added to this, is fact that coal slimes were recently re-classified under Government Notice 635, National Environmental Management: Waste Act 59 of 2008. Under the new classification, coal slimes must be stored in a lined tailings pond if disposed of wet. The cost of lining a tailings pond is often prohibitive and several operations are therefore looking to install tailings filtration equipment to allow for disposal.

For this reason, mining companies are increasingly looking for new methods to derive value from the slimes fraction in order to cover the costs of expensive dewatering equipment. Flotation is one such technology which has been widely applied in operations in Australia and North America in order to generate a high quality product from the slimes fraction. The technology was initially installed at Grootegeluk in the Waterberg area in the early 1990s; however the process was later decommissioned due

 
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to high costs and difficulties in operability. Since then, the process has enjoyed a resurgence in application due to advances in both cell technology and reagents.
This paper aims to demonstrate that by following a rigorous testwork program early on in the project lifecycle, the efficacy of the flotation process on a given ore body, can be properly assessed and a proper economic evaluation can be performed. Testwork results can also be used to assess the sensitivity of the process to changes in product grade. This type of analysis aims to assist decision makers in making informed decisions, based on facts rather than on historical industry sentiment.

The paper focuses on a case study which Hatch Goba performed for a confidential client, evaluating the potential of the flotation process for improving the NPV for a project in the Waterberg area.

**TEST WORK METHODOLGY**

A large box cut was taken from the mining area in order to obtain sufficient sample for test work. The sample was divided into five different seams and each seam was subjected to a drop shatter and wet tumble test to provide a particle size distribution which was indicative of the feed to the future Coal Handling and Process Plant (CHPP).

The feed samples for flotation were prepared by screening at top sizes of 0.25, 0.5 and 1.0 mm. For each top size, an as-is and deslimed feed was prepared to yield a total of six feed size distributions.

A standard three litre Leeds flotation cell was used to conduct the batch flotation tests, using an initial solids content of 7% by mass. Flotation tests were carried out as per ASTM, D5114 – 90. The flotation release analysis was conducted according to the method of Forrest et. al. (1993), including a rougher and three stages of cleaning. A collector/frother combination from Betachem was used as the reagent for the testwork. Previous work by Forrest et. al. (1993) showed that the release analysis should be largely independent of reagent type and dosage, however since the reagent dosage had a significant impact on the total mass pull, it was decided to test the samples at both a high and low reagent dosage as follows:

**Table 1: Reagent Dosages during Release Analysis**

| | Rougher | 1st Cleaner | 2nd Cleaner | 3rd Cleaner |
|---|---|---|---|---|
| Low reagent dosage | 500 ml/t (initial)<br>+ 1000 ml/t (after initial float complete) | 500 ml/t | None | None |
| High reagent dosage | 2500 ml/t (initial)<br>+ 1000 ml/t (after initial float complete) | 2500 ml/t | None | None |

Two duplicates were performed at the higher reagent dosage setting. The flotation products (four tails samples and one concentrate sample), were submitted for proximate analyses, calorific value and sulphur assays.

**DATA ANALYSIS**

Initial data from the tests were mass balanced to obtain the ash vs. mass yield results for each seam and each size fraction. The method of Forrest et. al. (1993) suggests comparing the fixed carbon recovery with the percent ash rejection. However the fixed carbon recovery is only of limited interest in practical applications, as one cannot relate the fixed carbon recovery to the total mass yield. For this reason the analysis was performed on standard mass yield vs. ash curves. The figure below shows an example of the ash yield curves obtained.

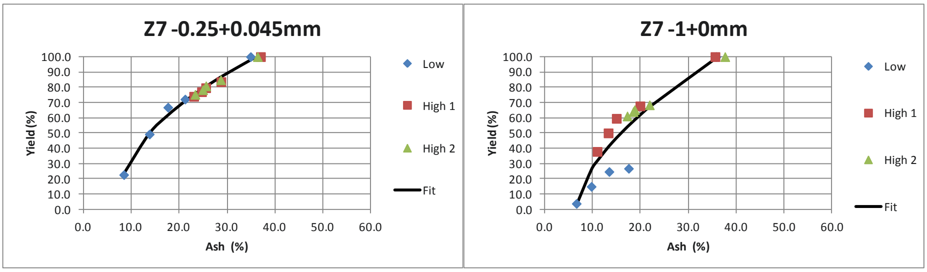


**Figure 1: Ash-Yield Curves Obtained for the -0.25+0.045 mm and -1.0mm Size Ranges**

Figure 1 illustrates the shape of the ash-yield curves obtained from the release analysis. In theory, curves which exhibit a sharp angle with points closer to the top left hand side of the graph indicate coal samples which were more liberated and therefore produced a higher yield at lower ash. Of particular importance for the analysis was the lower reagent dosage test, since this test typically resulted in lower ash products. As predicted by Forrest et. al. (1993), both the high and the low reagent conditions appear to fall on the same curve for the -0.25+0.045 mm fraction. This was found to be the case for all size fractions with top size of 0.25 mm. However, as the top size increased, the low reagent setting was often found to be insufficient to obtain the yields achieved in the high reagent tests. This is clearly shown in the results of the -1+0 mm size fraction in Figure 1.

In these cases, it is believed that the float was largely starved of reagent for the low reagent dosage test and only part of the recoverable material was floated in the rougher flotation stage. The observation that the release analysis curve is independent of reagent dosage therefore appears to only be valid for cases where the reagent dosage is sufficient to float the coarse particles in the feed.

**FLOTATION CIRCUIT MODELLING**

Flotation circuit modelling was performed using Limn: The Flowsheet Processor™. The model considered a standard coal handling process plant flowsheet processing -50mm material. A feed preparation screen diverted the -50+10mm material to a coarse dense medium separation (DMS) circuit and a deslime screen diverted the -10+1mm material to a small DMS circuit. Screening efficiencies of 90% were assumed. All classification units were modelled using a Whiten Efficiency Curve, to account for misplaced material. The yield in the flotation bank was taken from the release analysis curve for the given zone and size fraction.

The simulation also utilised a feed particle size distribution envelope to determine the variability of material reporting to the flotation. The flowsheet was tested all possible combinations of the three feed size distributions, five zones and six size fractions. This yielded a total of 90 flowsheet scenarios for testing.

**RESULTS AND DISCUSSION**

The results of the simulations are summarised in Figure 2 below.

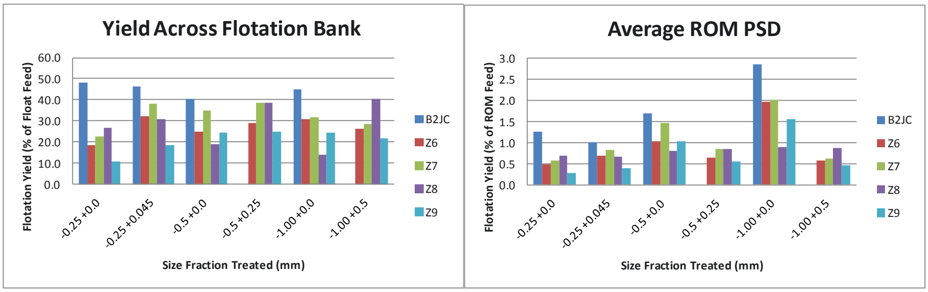

**Figure 2: Flotation Yield as a Function of Size, Zone and Plant Feed PSD.**

The yield across the flotation bank itself was found to vary widely, with yields between 17% and 54%. In most cases the B2JC zone was found to be the most amenable to flotation, with yields between 48% and 54%; however this fraction was not tested for the -0.5+0.25mm and -1.0+0.5mm size fraction. The zone 9 material was the least amenable to flotation with yields ranging from 17% to 35%.

In terms of yield across the flotation bank, the -0.25+0.045mm performed the best, with an average yield of 40%, followed by the -0.5+0mm and -1.0+0mm fraction with an average yields of 37% respectively. In general, desliming of the material was found to increase the flotation yield. This is to be expected as the majority of the ash is found in the finer size fractions.

When assessing the contribution of the flotation bank to the overall yield, the feed size distribution to the wash plant had a significant impact on the results. The -1.0+0mm fraction yielded the highest feed to the flotation bank and therefore the highest overall yield, followed by the -0.5+0mm fraction. Desliming of these two fractions was found to significantly decrease the feed to the flotation bank, and therefore the yield to final product was negatively affected.

Interestingly, desliming of the -0.25mm fraction at 0.045mm did not significantly decrease the mass reporting to the flotation bank. This is because a significant portion of the -0.045mm material is already removed by the screening and cycloning inefficiencies upstream of flotation, regardless of desliming. From the results in Figure 2 however, it is clear that these -0.045mm particles significantly decrease the yield achievable at the target ash of 13%.

When evaluating the results in Figure 2, it is clear that flotation of the -1.0+0mm fraction produces the highest contribution to the overall plant yield, followed by the -0.5+0mm fraction. However, if one considers that these size fractions were only able to achieve a yield of 37% across the float bank, whereas a more common technology such as a spirals and teetered bed separation (TBS) would expect to achieve a yield in the region of 60%, it is clear that flotation is not the most economical choice for these size fractions.

The -0.25+0.045mm fraction was able to yield on average, an additional 1.8% yield of 13% ash product. Since this material is normally considered too fine for a spiral-TBS circuit, this stream was considered to have the highest potential for adding to the project business case.

Processing of the -0.25+0.045mm fraction does however introduce a number of practical difficulties since it requires the use of small cyclones to achieve the desired desliming cut size. For example, desliming of 100 tph of -0.25mm material would require approximately 30 cyclones of 165mm diameter. Such a set up would add to the operational complexity of the flotation module, however this complexity is justified by the additional 7.7% yield across the flotation bank in this particular case.

**ECOMONIC ASSESSMENT**

The economics of including a flotation circuit in the final design were assessed relative to a base case scenario where the -0.25mm material was filtered and discarded. This is a common practice for most coal processing plants in the Waterberg area, since this material is often thought to be too difficult to process, due to the high ash values.

Costs for a flotation bank, including a rougher and scavenger were obtained from a vendor specialising in turnkey flotation plants using the Dual cell technology. The design included a concentrate thickener, as well as series of high pressure filter presses, to achieve final product moisture of 18-21% moisture. The flotation tailings is combined with the various cyclone overflow streams around the plant and be thickened and filtered with the existing tailings equipment.

In order to assess the full potential of the flotation module, the model was run for a variety of possible product grades, including a semi-soft coking coal (10.5% ash), a Richards Bay Coal Terminal (RBCT) export coal (13% ash) and a typical Eskom grade steam coal (25% ash). The table below summarises the outcomes of the assessment.

**Table 2: Results of Economic Analysis of Flotation Bank**

| | | | |
|---|---|---|---|
| Plant Throughput | 10 | Mtpa | |
| Operating Hours | 6 700 | hrs p.a. | |
| Size Fraction | -0.25+0.045mm | | |
| Nominal Throughput (tph) | 67 | | |
| Design Throughput (tph) | 86 | | |
| Product | Semi-Soft Coking | RBCT Export | Eskom |
| Product Ash (%) | 10.5 | 13 | 25 |
| Sales Price (R/t) | 850 | 650 | 300 |
| Logistics (R/t) | 295 | 295 | 20 |
| Ave Product Yield (%) | 31 | 40 | 68 |
| CAPEX (R 'mil) | 108 | 108 | 108 |
| OPEX @ R100/ton | 45 | 45 | 45 |
| Logistics (R 'mil) | 42 | 52 | 6 |
| Revenue (R 'mil) | 120 | 115 | 91 |
| NPV @ 10% (R 'mil) | 160 | 42 | 215 |
| IRR | 31% | 16% | 37% |

Table 2 illustrates the versatility of the flotation process for this particular ore type. The ash-yield curves derived from the testwork indicate that the process is able to produce a wide range of products to suit the prevailing market conditions.

In the case of the Waterberg, the high cost of logistics for transportation of export coal to RBCT significantly reduces the operating profit. This is particularly true in the current economic environment where the export price of thermal coal is constrained by an oversupply in the market. This situation is somewhat improved by targeting a higher value semi-soft coking coal for export, however the high cost of logistics still constrains the project business case.

Interestingly, the business case for targeting an Eskom product produces the best business case for the current price structure. This is because of the significantly reduced logistics costs for local sales in the Waterberg area, as well as the depressed export prices for RBCT coal. Targeting of a high ash product also allows the flotation plant to achieve a yield of 68%. This is highly significant when considering the environmental aspects of the project, as only 32% of the fine coal fraction will require disposal on site rather than the 69% required when targeting a semi-soft coking coal.

## CONCLUSION

A methodology for the evaluation of coal flotation results has been demonstrated. The method is based on the release analysis technique of Forrest et. al. (1993). During the analysis it was found that the usage of a low reagent condition was important in determining the minimum ash product that can be derived from a given coal seam. However, the reagent dosage needs to be sufficient to float the coarser particles in the feed. This could be determined by scouting floats prior to undertaking the detailed testwork program. Alternatively, timed flotation samples should be taken on the final cleaner float, as per the original method of Forrest et. al. (1993). The first timed sample is of the greatest importance, since this corresponds to the lowest ash product.

Results for this particular coal deposit indicate that flotation was unable to outperform conventional spiral and TBS performance for size fractions with material of greater than 0.25mm. The -0.25mm+0.045mm fraction was found to achieve an average 40% yield of 13% ash product.

In the current economic environment, the sales prices of export coal are relatively low compared to the high cost of logistics to transport coal from the Waterberg to Richards Bay Coal Terminal. For this reason, the use of flotation to produce an Eskom grade steam coal for local sales produced the best business case in this particular case. This has the added benefit of greatly reducing the volume of fine coal that must be disposed of on site and will therefore aid in the compliance with environmental regulations.
Given the combination of economic and environmental benefits for flotation of coal fines, it is believed that this technology will see a significant resurgence in the coming years. The combination of improved cell technology, reagents and process control have significantly improved the reliability of coal flotation.

## ACKNOWLEDGEMENTS

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# High end pneumatic flotation

Different features have to be considered by choosing the right flotation technology for coal. Material feed characteristics have to be considered. The flotation reagents have the most influence of a proper flotation process. Reagents, air intake volume and air bubble size have to be set up optimal to the feed material. Turbulent systems have lower efficiency for the very fine particles. Further the recycling load improves the concentrate in terms of yield. The correct design of the froth discharge system has a big influence of the concentrate yield, too.

Laboratory, semi industrial and industrial results are presented to underline the differences on a technical basis, when using pneumatic flotation. A new designed froth discharge system will be introduced which improves froth yield.

Beside the technical aspects to take the most efficient flotation system for the process, investment and operational cost have to be considered. The power consumption of different flotation technologies may differ enormous. In addition such characteristics like footprint, size and weight of steel supporting structure as well as wear part demand are important to compare prior selection of the flotation technology.

A simple guidance of different flotation technologies and its characteristics will be presented. Each flotation system has specific advantages and disadvantages to be considered.

**Key words:**

Ruzbass territory, coal flotation, pneumatic flotation, froth discharge system.

**Introduction**

In the Kuzbass territory in Russia mainly conventional agitator flotation machines are installed. Recently, column flotation and pneumatic flotation systems were introduced to the market.

Prior the installation of the first pneumatic flotation cell with vertical flow through a self-aspirated aerator, semi industrial test work was carried out. Target of the test work was to find out performance and efficiency of the flotation process. This test work was performed in a bypass to the existing mechanical agitator flotation. Different reagents and dosing rates were applied to find out optimal flotation parameters. The reagents have big influence on the results in terms of grade and yield.

In general, two stages of coal flotation are required to achieve acceptable yield, independent, which flotation technology was chosen. The cells/ banks have to be installed as rougher and scavenger flotation in series.

A combination of two different technologies causes most difficulties to be considered and solved. In the following, the pneumatic flotation called PNEUFLOT was combined with the mechanical agitator flotation as second stage.

 
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Optimal yield only can be reached, if the flotation technology is able to discharge coarse and very fine particles together with the froth. For agitator flotation it is in general not problematic to float coarse particles, column flotation are lower efficient for particles bigger than 0.3 mm. Pneumatic flotation is able to float minor particles up to 1 mm.

The coking coal KO (weakly caking bituminous type) which was in addition extremely fine (50% less than 40 micron) only gets good separation with non-turbulent flotation systems.

**Test work**

World standard and the fastest way of preliminary test results are in general generated in a small agitator Denver cell. The size should be two or three liter scale. Smaller cells are not suitable. This test work is used to determine the optimized reagent regime in general.

After performing it, columns and other pneumatic flotation technologies use own designed semi industrial machines to get proper test results. Agitator and column flotation needs upscale factors to industrial size. The tests carried out in semi industrial PNEUFLOT machine represent industrial results without needing a scale up factor.

The testing machine consists of the flotation cell and complete equipment like feed and tailings centrifugal pump as well as feed agitator tank with 1 m³ volume. The installation requires half a day. It is delivered in a 20” container. Experienced engineers are doing the tests. Usually three to four working days are sufficient to get the full overview of reachable targets.

Figure 1: Semi industrial testing machine PNEUFLOT

| Particle size [micron] | Yield [%] | Ash [%] |
|---|---|---|
| > 500 | 3.61 | 5.2 |
| 250 – 500 | 6.95 | 5.6 |
| 125 – 250 | 15.74 | 7.4 |
| 63 – 125 | 7.35 | 8.7 |
| 40 – 63 | 15.63 | 9.9 |
| < 40 | 50.73 | 22.4 |
| Total | 100.00 | 15.3 |

Figure 2: Feed particle size ash related analysis for coal type KO

The use of reagent Progress at a dosing rate of 1.000 – 1.100 gram per ton shows the following results for a one stage operation: ash grade of froth in the range of 9 – 10% at a yield of 60%. Together with the existing mechanical agitator flotation as second stage, the total yield was 87.5%.

Based on the test results, the corresponding flow sheet was developed. It was further found out, that the froth concentration of pneumatic flotation was with 210 gram per liter higher compared to the mechanical agitator flotation (140 gram per liter). Higher froth concentration results in higher discharge capacity of the filter and improved dewatering characteristics (22% surface moisture).

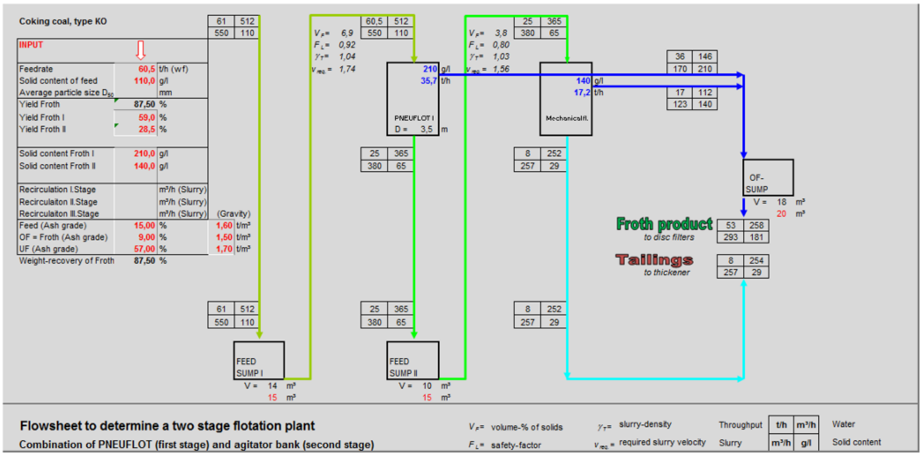


Figure 3: Industrial flow sheet, designed from test work

**Commissioning phase and improvement of PNEUFLOT machine and process**

Figure 4: Industrial flotation machine after improvement with a froth channel discharge device

In case two different technologies have to work together, some very important processing features must be considered.

1. The machine has to be set up for maximum efficiency.
2. The process around has to be adapted to continuous operation.

To set up the machine for maximum efficiency, a new designed discharge channel system was installed. By doing this, the yield could be improved from 45% to 59% in one stage. Prior designed cells with inner cone had partly losses in the coarse particles bigger than 0.5 mm. With the froth discharge channel system, up to 1 mm particles can be floated. Also the upstream-velocity out of the slurry distributor nozzles has a big influence on the flotation yield. If the upstream-speed is higher than 5 meter per second, big turbulence destroys a silent froth building zone.

The feed material

  

Figure 5: Modification of froth discharge system

| 1st variant: original inner cone | 2nd variant: shorter inner cone | 3rd variant: froth channel |
|---|---|---|
| Tank volume: 22.1 m³ | 23.7 m³ (+ 7%) | 27.2 m³ (+23.1%) |
| Overflow lip length: 11 m | 11 m | 33.8 m |
| Retention time: 2.41 minutes | 2.59 minutes | 2.97 minutes |
| Yield: 45% | 51% | 59% |
| Mainly fine particles | Fine and coarse particles | Fine and coarse particles |
| Partly turbulent process | Minor turbulent process | Silent laminar process |

The gas hold up in the pneumatic flotation is with 20% approx. 5% higher, than in conventional mechanical agitator cells which improves yield in smaller tank volume. The feed pressure influences air bubble size and can be applied in the range of 2.5 – 3.0 bar with the use of a venture based self-aspirated aerator.

If the yield of the pneumatic flotation not exceeded 50%, the tails sump underneath which feeds the mechanical agitator flotation bank as second stage, was overloaded. Further to the tails sump underneath, big froth volume in addition led to overflow. After implementing the new froth channel discharge system, overload of tails sump and mechanical flotation was stopped.

Further, the froth has to be generally destroyed before feeding the vacuum disc filter(s). Special vertical or horizontal froth pumps with the capability to destroy the froth have to be included in the flow sheet. As other variant, a special designed slow running agitator mechanism can be used when feeding the disc filter by gravity.

As next problem, the intensive froth building at the mechanical agitator bank took place. Very low feed concentration of about 55 gram per liter for the second stage with approx. 75% less than 40 micron had to be processed in the existing machinery. Therefore, a reagent with low froth building characteristics must be applied. Further, the air intake plates of the agitator mechanism have to be closed more to minimize froth volume.

The following data of both flotation stages may give an overview about the process:

Figure 6: first and second stage of flotation

| | |
|---|---|
| 1st stage PNEUFLOT K-FV 35 NS | 2nd stage mechanical flotation |
| Feed rate: 550 m³/h | 380 m³/h |
| Retention time: 2.97 minutes | 5 minutes |
| Motor power: 110 kW | 330 kW (11 x 30 kW) |
| Specific power: 0.2 kW per 1 m³/h feed rate | 0.87 kW per 1 m³/h feed rate |
| Yield: 59% | 28.5% |

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# Flotability Sludge From Coal Waste Technogenic Deposits of Ukraine

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**Abstract**

Ukrniiugleobogashchenie (coal cleaning) institute experts investigated out-of-balance coal sludge from sludge pits in 16 coal cleaning plants in Ukraine. High content of fine coal particles sized less than 45 μ (microns) and increased oxidation level of their surface were identified that are the main reasons for the hard flotability tailings of technogenic deposits.

We have established dependence of the lifetime of the flotation complex from surface oxidation of the coal particles. The scheme of coal flotation with the withdrawal of finely divided sludge after preliminary enrichment in the first flotator chambers is developed. Showing rational modes flotation, which provide supply portion the most active foaming agent using efficient emulsification collector. A range of new generation radial flotation plants with one radial-axial aerator, and KPSh-100 processing complex for processing of sludge from sludge pits developed.

**Key words:**

Coal sludge, technogenic fields, technology, flotation, oxidation, aerator, emulsification, processing complex.

Tight fuel balance of Ukraine maintains a claim to develop efficient techniques for utilization not only natural but also technogenic coal fields in the fuel industry of the country.

There is about 85 million tons of fine-grained coal sludge with the ash content of 41 – 75% [1] contained in the sludge pits in Ukraine. Some sludge pits are actually technogenic fields where production of power-generating fuel is feasible.

"Ukrniiugleobogashchenie" institute experts investigated out-of-balance coal sludge from sludge pits in 16 coal cleaning plants in Ukraine. That sludge is quite representative in terms of its technical specifications and composition. Summarized granulometric and fraction composition data of that sludge is shown in Figures 1 and 2. Based on the study results the average ash content of the sludge contained in sludge pits is about 60 %, granular segment yield is around 29 %, ash content is up to 44 % (Figure 1).

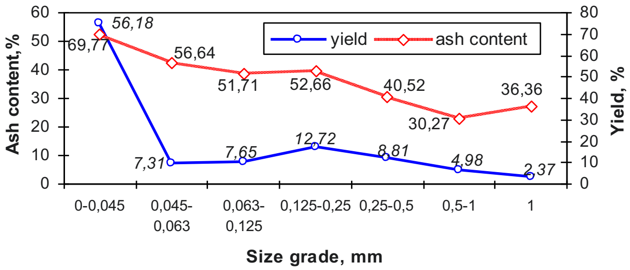


**Fig. 1** The average data for the granulometric composition of sludge

Average fine particle (sized less than 45 μ) content is 56.2 %, ash content is 69.8 %. Based on the fraction analysis results about 20% of the sludge has ash content up to 20% (Figure 2).

 
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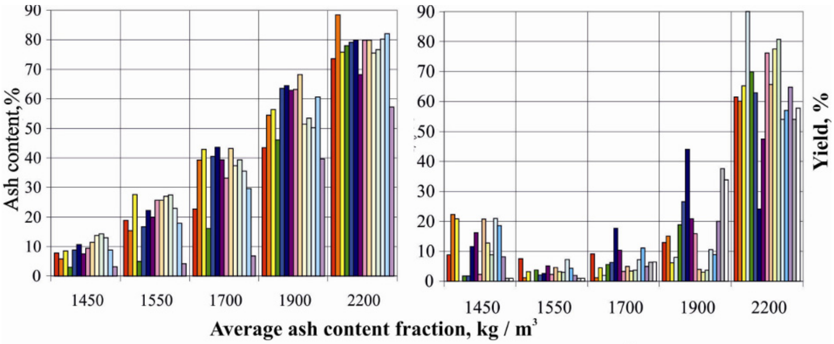


**Fig. 2** Fractional composition slurry from the sludge pit: ash content, yield fractions

Based on the analysis of the contents of sludge pits and the current status of their processing, it was found that the most effective process for the enrichment of sludge products are wet spiral separation and flotation. In this regard, the institute "Ukrniiugleobogaschenie" were investigated flotation properties of man-made sludge of this pits.

High content of fine coal particles sized less than 45 μ (microns) and increased oxidation level of their surface were identified that are the main reasons for the hard flotability of technogenic deposits. Well-known that fine particles have a considerable effect on the flotation rate, foam stabilization, chemicals consumption and other processes [2, 3]. Fine sludge adhesion to the surface of coal particle coal particles hinders the adhesion of the latter to air bubbles. As it is determined in the studies [4, 5] technogenic field coal particle surface oxidation have a negative effect on the floatation agent adsorption due to the lower coal natural hydrophobic properties.

“Ukrniiugleobogashchenie” institute experts made studies of the oxidation of fresh-mined coal particle surfaces and surfaces of sludge from sludge pits (Table 1). As the table reads during sludge transportation and storage their surface gets considerably oxidized.

In view of the above for Г, Ж, ОС and А grades of transported fresh-mined sludge the oxidation is 13-15 %, and for grades Г and Ж of sludge stored in sludge pits we have especially high oxidation values: 32-36 %. For grades ОС and А of sludge from sludge pits the oxidation values are a bit lower: 20-26 %.

Numerical simulation modeling with discrete units is applied for review of phase surface properties effect on flotation complex. The most detailed description of modeling technique and its fundamentals is given in studies [4, 5]. Based on the study results a floatation complex lifetime significantly depends on the coal particle surface oxidation in compliance with following exponential function:

$$\tau = 42{,}04\ OX_c^{\ -2{,}93}\ , \qquad (1)$$

where $\tau$- floatation complex lifetime, sec;

$OX_c$- coal particle surface oxidation, %.

Summary results of the floatation benefication of sludge of different grades and different surface oxidation levels when using the most efficient agent regimes are specified in Table 1.

Study of flotation of coal sludge with different surface oxidation levels was made using the following flotation agents currently used at Donbas coal-cleaning plants: collector agents (diesel fuel DF, TS-1 fuel, catalytic gas oil - RSL); blowing agents (T-80, KETGOL, KOBS). Besides, composite collecting agent widely used in Russia as the unified agent at Kuzbass coal-cleaning plants was also used [6].

Review of the results obtained enables us to make a conclusion that for the same floatation regimes processing parameters for high oxidation level coals are much worse than those for coals of the same grade with a low oxidation level. For the same regimes we failed to exceed 60% waste ash level for the oxidized sludge that serves as evidence of oxidized low-ash coal particles loss with the waste.

**Table 1** Flotation parameters for coals with different surface oxidation levels

| Cal grade | Sludge sampling location | Ash content, % | Agent regime | Oxidation, $OX_c$, % | Selectivity | Fuel mass extraction, % | Efficiency |
|---|---|---|---|---|---|---|---|
| Г | Novogrodovskaya mine raw coal | 48,4 | RSL 480, T- 80 | 13 | 27,4 | 86,0 | 70,2 |
| | | | KR +KOBS | | 30,97 | 87,9 | 71,8 |
| | PP Pioneer sludge pit | 61,4 | KR 600 | 36 | 11,89 | 54,5 | 62,0 |
| | Silidovskaya sludge pit CPP | 52,1 | KR 600 | 32 | 9,92 | 72,6 | 57,3 |
| Ж | Molodogvardeyskaya mine raw coal | 26,2 | TS 1100, KOBS 100 | 14 | 36,3 | 95,7 | 56,4 |
| | | | KR 850 | | 29,5 | 94,0 | 61,3 |
| | Combined sludge pit | 55,4 | KR+KOBS 150, DF 750 | 32 | 7,6 | 42,0 | 38,3 |
| OC | Kalinin mine raw coal | 28,6 | TS 1500, KETGOL 100 | 15 | 24,1 | 93,4 | 58,7 |
| | Kalinin mine outdoor storage area | 26,7 | TS 1500, KETGOL 100 | 20 | 20,1 | 92,4 | 57,7 |
| A | Dolzhanskaya-Kapitalnaya mine raw coal | 29,5 | KR 470 | 15 | 19,0 | 91,4 | 58,9 |
| | PP Mayak sump | 42,7 | KR 700 | 26 | 5,45 | 80,6 | 45,8 |
| | | | diesel fuel 1900, KOBS 100 | | 5,85 | 57,9 | 46,9 |

As Table 1 reads should surface oxidation level exceeds 15% all parameters shall go down, it is especially related to the separation selectivity deterioration.

In view of the above floatation beneficiation stale oxidized coal-bearing sludge technogenic deposits requires development of a technique applying the most efficient flotation agents, agent feeding regimes and methods, floatation procedures, schemes and use of upgraded floatation equipment.

Sludge low floatation rate is due to their production induced surface since an increased amount of small air bubbles is required. For that purpose a radial-axial impeller aerator was designed in the institute. Combination of two face-to-face axial and radial wheels in one impeller enables maximum air inflow with its subsequent dispergation and distribution in the liquid phase.

In order to create a compact flotation machine of the new generation to reenrichment the contents of the sludge pit with lower power consumption and metal content, the institute "Ukrniiugleobogaschenie" conducted research and developed a series of small radial parametric reenrichment mechanical flotation machines with one aerating device, optimal fluidic and technological characteristics of 2 $m^3$, 16 $m^3$, 25 $m^3$, 36 $m^3$ [7]. Radial floatation machine with one radial-axial aerator specific parameters and technical capabilities (performance of solids, degree of aeration, stable and sufficient air supply to support the processing) exceed floatation machines with six aerator.

The technogenic deposits flotation technique with collecting agent emulsification is developed in the institute to reduce agent consumption [8].

The technique enabling flexible control of agent action, feeding methods, metering and distribution throughout the floatation area is developed in the institute is critical for fine-grained coal cleaning intensification applying the floatation technique [9]. Application of the developed rational floatation regimes enables to reduce coal loss with floatation tailings and also reduce floatation concentrate ash content while decreasing agent mix consumption for up to 30-40%.

Besides, the scheme of coal flotation with removal of fine sludge from the flotation after its pre-cleaning in the first flotation machines chambers was elaborated in the institute to provide technogenic sludge cleaning (Figure 3) [10].

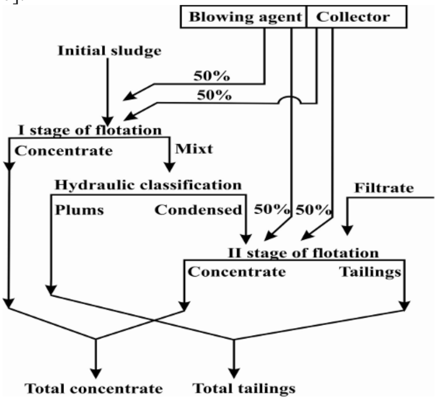


**Fig. 3** Technogenic coal-bearing sludge floatation schematic

As per the developed technique at the first flotation phase (process start-up) conditions for mainly fine sludge shall be created by using a small amount of agents, and then after slurry removal with small hydrocyclone draining more efficient coarse sludge flotation shall be performed at the second phase. Meanwhile low-ash floatation concentrate is generated in the last floatation chambers, and consequently making the total concentrate ash content gets reduced by 2-5 % (Table 2).

**Table 2** Sludge floatation processing parameters for the two-phase schematic

| Products of floatation | Single-phase floatation (base case) | | Two-stage floatation for 0.045 mm hydroclassifier grain boundary | |
|---|---|---|---|---|
| | yield, % | ash content,% | yield, % | ash content,% |
| CPP Oktyabrskaya | | | | |
| Initial sludge | 100,0 | 53,8 | 100,0 | 53,2 |
| Concentrate phase I | | | 17,9 | 21,1 |
| Concentrate phase II | | | 13,9 | 9,2 |
| Total concentrate | 32,4 | 22,4 | 31,8 | 15,9 |
| Tailings phase II | | | 7,4 | 73,7 |
| Tailings phase I (hydroclassifier draining) | | | 60,8 | 70,2 |
| Total tailings | 67,6 | 68,8 | 68,2 | 70,6 |
| CPP Selidovskaya | | | | |
| Initial sludge | 100,0 | 46,7 | 100,0 | 47,0 |
| Concentrate phase I | | | 32,5 | 19,8 |
| Concentrate phase II | | | 21,8 | 15,2 |
| Total concentrate | 48,6 | 20,4 | 54,3 | 18,0 |
| Tailings phase II | | | 7,6 | 76,6 |
| Tailings phase I (hydroclassifier draining) | | | 38,1 | 78,4 |
| Total tailings | 51,4 | 71,6 | 45,7 | 78,1 |

Data on technogenic sludge flotation performed applying the developed technique providing batch feeding of collector and blowing agent mix, and using efficient collector emulsification for the two-phase flotation schematic is specified in Table 3.

**Table 3** Sludge flotability test results (for different sludge pits)

| Test | Floatation agent description and max ratio | Flotation agent consumption, g/t | Ash content, % | | | Flotation concentrate yield, % | Combustible mass extraction in the concentrate, % |
|---|---|---|---|---|---|---|---|
| | | | initial sludge | flotation concentrate | tailings | | |
| Sludge pit CPP "Komendantskaya" (grade A, oxidation level $OX_c$ = 24%) | | | | | | | |
| 1 | Diesel fuel:KETGOL=8:1 | 1150 | 44,2 | 23,3 | 83,3 | 65,2 | 89,6 |
| Sludge pit GPP Luganskaya (grade Г, oxidation level $OX_c$ =38%) | | | | | | | |
| 2 | TC-1: Oksal =35:1 | 3600 | 53,9 | 19,6 | 77,5 | 40,8 | 71,2 |
| Sludge pit CPP Sverdlovskaya (grade A, oxidation level $OX_c$ =20%) | | | | | | | |
| 3 | KOBS | 250 | 41,3 | 19,9 | 87,6 | 68,4 | 93,3 |
| Sludge pit CPP Selidovskaya (grade Г, oxidation level $OX_c$ =32%) | | | | | | | |
| 4 | KR | 600 | 51,8 | 23,9 | 75,7 | 46,0 | 72,8 |
| Sludge pit CPP Dzerzhinskaya (grade Ж, K, oxidation level $OX_c$ =29%) | | | | | | | |
| 5 | KR | 600 | 52,9 | 25,7 | 79,3 | 48,7 | 77,5 |
| Sludge pit PP Pioneer (grade Г, oxidation level $OX_c$ =36%) | | | | | | | |
| 6 | KR | 600 | 61,4 | 23,6 | 78,6 | 31,3 | 62,0 |

Analysis results testify that saleable concentrate (ash content ranges from 19.6% to 25.7%, floatation tailing ash content ranges from 75.7% to 87.6%) can be produced from sludge sized less than 0.5 mm taken from all 6 sludge pits under test, floatation concentrate yield ranges from 31,3 % to 68,4.%.

Based on the study results and sludge cleaning operation synthesis Ukrniiugleobogashchenie experts developed KPSh-100 processing complex for technogenic sludge pit processing [11].

Complex design enables its transformation required for a specific sludge pit.

**Conclusions.** In view of the above we can make a conclusion that floatation of sludge taken from coal cleaning plant sludge pits is prospective and reasonable. Provided that optimal agent regimes are selected and developed floatation flow and radial-axial aerators are used additional marketable end product can be generated from Ukrainian technogenic coal fields for the power industry.

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# Flotation of Polish Hard Coal in Saline Water

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**Abstract**

With the use of process water (saline), the ash content in the concentrate was 6% and in the tailings up to 70%. Of course the upgrading plants which apply flotation for coal slurries still use mine water in technological processes. The water quality, particularly the content and type of salts in water, fundamentally influences the surface properties of phases which remain in contact during the process, that is the solid, gaseous and liquid phases. The properties of these phases and interactions between them determine the probability of occurrence of elementary flotation actions (Brożek and Młynarczykowska, 2005, Brożek and Młynarczykowska, 2008): collision, adhesion and separation, and ultimately the upgrading process effectiveness expressed by technological parameters. The paper presents the results of flotation effectiveness analysis for coal as a model raw material. The object of evaluation was qualitative parameters of the upgrading process products on the basis of flotation tests conducted for variable pulp salinity. The principal goal was to determine the impact of process water quality on the course and effectiveness of flotation.

**Keywords:**
(coal flotation, froth flotation, saline water, polish hard coal, flotation kinetics, upgrading curves, flotation parameters )

## Introduction

Hard coal is the main source of energy in Poland and its properties result from the condition of formation. The mineral matter of raw coal consists of hydrophilic minerals, mainly clays, quartz, carbonate minerals such as calcite and dolomite, gypsum and pyrite. Flotation is not the basic coal upgrading technology, but it is useful in pyrite removal and in case of culm as a method of upgrading which reduces the ash content in the concentrate. The use of surfactants in the coal flotation is justified from the physical and chemical point of view. One must also remember that in the industrial conditions the flotation is performed with the mine water which contains various well soluble inorganic salts which types and amounts depend on the mining conditions. Presence of salt in the flotation environment affects the action of flotation reagents, and consequently the process results and the quality of concentrate. The use of process water in coal upgrading in processing plants began as early as in the 1980's. However, the main reason to study it was treatment of small coal particles and regulation of water and slurry management in the plants in terms of solids content in concentrators and the quality of concentrate. Since the observation that presence of salts, mostly inorganic, improves the flotation of naturally hydrophobic particles, many theories have been formulated to explain this phenomenon. (Yoon, 1982; Yoon and Sabey, 1989; Laskowski, 2001. The research conducted by Paulsen and Pugh (1996) showed that the flotation recovery reaches its maximum at the minimum value of zeta potential. However, Harvey et al. (2002) proved at the same time that the improvement of coal flotability in the solutions of electrolytes cannot be explained exclusively by reduction of the zeta potential. Ozdemir et al. (2009) revealed the coal flotation results in brine (1 M NaCl) cannot be explained fully by physical and chemical surface properties, particularly because the wetting angle for coal in the saline water is the same as in the

 
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demineralized water. Hence, the explicit determination of mechanisms which improve the coal flotability under conditions of significant salinity is still an open question.

**Material**

The feed used in the flotation processes was grade 33 hard coal from one of Polish mines. The material preparation included comminution to obtain the grain size below 0.5 mm. The wet sieve analysis was performed and the following grain size classes were obtained: (0.5-0.4), (0.4-0.315), (0.315-0.2), (0.2-0.1), (0.1-0.063) which were tested for ash content. The material to be tested was kept in vacuum to prevent coal surface oxidation. All grain size classes were used in the flotation tests, analysing the impact of grain size and grain density on the upgrading results in the salt-free flotation environment. On the basis of previous results, two grain size classes (0.315-0.2) mm and (0.1-0.063) mm were selected for flotation tests in conditions of various pulp salinity. The flotation tests were also conducted for the grain size below 0.5 mm.

**Flotation Experiment**

The flotation tests were performed in the Denver pneumatic-mechanical laboratory machine. The work space volume was 1dm$^3$, rotor speed was constant 1850 rpm, and the pulp density was 70g/l. Using low flotation pulp densities allows to achieve better flotation upgrading results, and the constant rotor speed guarantees the unchanging amount and size of air bubbles (Młynarczykowska et al.,2013).

Individual flotation tests for selected narrow grain size classes were performed in distilled water, saline mine water and in brines made by adding NaCl to the mine water. The content of chlorine ions ($Cl^-$) in saline waters was 255.6 mg/dm$^3$, 23.4 g/dm$^3$ and 52.6 g/dm$^3$, and the waters were respectively referred to as "mine water I, II, III". The grains below 0.5 mm were floated in saline water which content 117 mgNaCl/dm$^3$ The products of flotation (concentrates and tailings) were analysed for ash content in accordance with PN-ISO 1171: 2002.

**Flotation Process Kinetics**

All flotation experiments were conducted in line with the procedure described below. Each time, specified coal samples were weighed, then process water with specific salt content was added, and the coal was soaked for 10 minutes. Then, a flotation reagent was added (hexanol). Hexanol is an aliphatic alcohol and in the flotation process it performs the function of a collector and a frothing agent which is related to its adsorption on both the solid-liquid and liquid-gas phase boundaries, thus increasing the coal hydrophobicity and reducing the speed at which air bubbles flow to the surface. In addition, it allows to form a durable flotation froth, ensuring stabilization of air bubbles and preventing their coalescence .

At specific pulp density, the dose of flotation reagent was specified as 0.08 g of hexanol per 1 dm$^3$ of solution, which corresponds to consumption of 1142 g per 1 Mg of dry feed. To ensure the reagent adsorption, the flotation machine was started and the solution was mixed for 3 minutes without access to air (Młynarczykowska et al. 2013). The air was delivered to the flotation tank after the set time and the fraction flotation was performed. The concentrates were collected at the following time intervals: 15, 15, 30, 30, 30, every 60 [s]. The last froth product was collected in the 6$^{th}$ minute of flotation. The concentrates and tailings were dried at about 60°C. The obtained results allowed calculating the basic indices to estimate the flotation process effectiveness, namely concentrate yields ($\gamma_k$) and tailings yields ($\gamma_o$), ash content in individual upgrading products ($\upsilon$–concentrate, $\beta$– tailings, $\alpha$–feed), and content of combustibles and volatiles in the concentrate ($\varepsilon_k$) as well as the ash content in the tailings ($\varepsilon_o$)

**Presentation of Results**

Tables 1 and 2 include the calculation results limited to yield, recovery of combustibles and volatiles and ash content in concentrates. The flotation kinetics curves were plotted in order to evaluate the upgrading progress in time. Figures 1 and 2 present the empirical recovery vs time relationships, respectively for grain size classes (0.315-0.2) and (0.1-0.063). The upgrading curves for individual grain size classes were plotted to illustrate the flotation effectiveness - Figures 3 and 4.

Table1. Selected coal flotation parameters: grain size (0.315-0.2) mm

| **Particie size (0,315-0,2) mm** | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **time** | **distilled water** | | | **Mine water I** | | | **Mine water II** | | | **Mine water III** | | |
| **t [s]** | **Σγ ↓ [%]** | **ϑ [%]** | **ε [%]** | **Σγ ↓[%]** | **ϑ [%]** | **ε [%]** | **Σγ ↓ [%]** | **ϑ [%]** | **ε [%]** | **Σγ ↓ [%]** | **ϑ [%]** | **ε [%]** |
| **0,** | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 |
| **15** | 4,90 | 5,13 | 4,80 | 9,54 | 18,67 | 9,73 | 7,48 | 25,37 | 7,57 | 9,77 | 33,36 | 8,34 |
| **30** | 8,07 | 5,07 | 7,92 | 11,48 | 17,99 | 11,80 | 10,72 | 26,86 | 10,63 | 14,54 | 29,12 | 13,19 |
| **60** | 12,39 | 4,86 | 12,18 | 15,63 | 14,49 | 16,75 | 14,71 | 24,91 | 14,98 | 18,80 | 26,25 | 17,74 |
| **90** | 17,00 | 4,60 | 16,76 | 18,12 | 14,36 | 19,44 | 17,96 | 25,13 | 18,23 | 22,56 | 23,74 | 22,01 |
| **120** | 19,88 | 4,67 | 19,59 | 20,89 | 13,94 | 22,52 | 21,20 | 25,61 | 21,38 | 25,31 | 22,50 | 25,10 |
| **180** | 26,51 | 4,27 | 26,23 | 26,14 | 13,52 | 28,33 | 26,68 | 25,82 | 26,84 | 31,08 | 20,85 | 31,47 |
| **240** | 32,28 | 4,19 | 31,96 | 30,29 | 13,15 | 32,97 | 31,42 | 24,44 | 32,19 | 35,34 | 21,08 | 35,69 |
| **300** | 38,62 | 3,96 | 38,33 | 34,72 | 12,70 | 37,98 | 35,16 | 23,44 | 36,51 | 39,10 | 19,91 | 40,07 |
| **360** | 44,09 | 4,02 | 43,74 | 40,25 | 12,10 | 44,33 | 38,65 | 22,44 | 40,65 | 43,61 | 18,82 | 45,30 |
| **tailing** | 100,00 | 3,24 | 100,00 | 100,00 | 20,20 | 100,00 | 100,00 | 26,26 | 100,00 | 100,00 | 21,85 | 100,00 |

Table2. Selected coal flotation parameters: grain size 0.1-0.063 mm

| **Particie size ( 0,1 -0,063) mm** | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **time** | **distilled water** | | | **Mine water I** | | | **Mine water II** | | | **Mine water III** | | |
| **t [s]** | **Σγ ↓ [%]** | **ϑ [%]** | **ε [%]** | **Σγ ↓[%]** | **ϑ [%]** | **ε [%]** | **Σγ ↓ [%]** | **ϑ [%]** | **ε [%]** | **Σγ ↓ [%]** | **ϑ [%]** | **ε [%]** |
| **0** | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 | 0,00 |
| **15** | 42,47 | 15,57 | 49,77 | 39,19 | 17,18 | 46,97 | 35,42 | 16,20 | 45,38 | 34,13 | 15,17 | 44,42 |
| **30** | 57,05 | 15,19 | 67,16 | 50,90 | 17,39 | 60,84 | 48,19 | 17,15 | 61,04 | 45,44 | 15,41 | 58,97 |
| **60** | 67,19 | 15,66 | 78,66 | 59,98 | 18,06 | 71,12 | 57,83 | 18,05 | 72,45 | 55,95 | 18,43 | 70,02 |
| **90** | 71,32 | 16,13 | 83,02 | 63,32 | 18,45 | 74,73 | 61,93 | 18,70 | 76,97 | 63,89 | 21,10 | 77,34 |
| **120** | 75,75 | 16,82 | 87,46 | 66,19 | 19,33 | 77,27 | 65,30 | 19,52 | 80,34 | 66,47 | 22,09 | 79,45 |
| **180** | 78,61 | 17,29 | 90,24 | 68,94 | 19,96 | 79,85 | 71,08 | 21,19 | 85,64 | 74,80 | 26,11 | 84,80 |
| **240** | 79,87 | 17,53 | 91,43 | 70,85 | 20,59 | 81,41 | 74,94 | 22,34 | 88,97 | 77,58 | 26,91 | 87,00 |
| **300** | 80,03 | 17,56 | 91,58 | 72,16 | 21,18 | 82,31 | 76,87 | 22,71 | 90,83 | 79,76 | 27,52 | 88,70 |
| **360** | 81,30 | 17,93 | 92,62 | 73,00 | 21,42 | 83,00 | 78,80 | 23,19 | 92,53 | 81,15 | 27,96 | 89,70 |
| **tailing** | 100,00 | 27,95 | 100,00 | 100,00 | 30,90 | 100,00 | 100,00 | 34,59 | 100,00 | 100,00 | 34,82 | 100,00 |

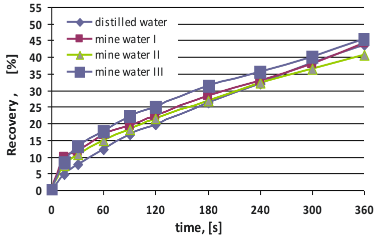


Figure 1. Flotation kinetics curves in waters with different salinity for grain size (0.315-0.2) mm.

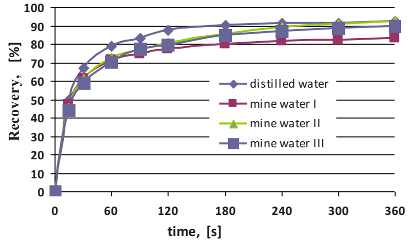


Figure 2. Flotation kinetics curves in waters with different salinity for grain size (0.1-0.063) mm

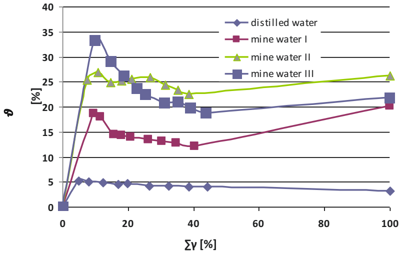


Fig.3 Changes of ash content in concentrate from yield, particle size (0.315-0.2)mm

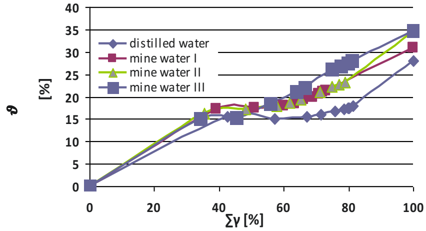


Fig. 4 Changes of ash content in concentrate from yield, particle size (0.1-0.063)mm

The graphical interpretation of flotation results for the below 0.5 mm grain size is performed by plotting the ~~flotation~~ kinetics curves (Fig.5) and the curves indicating the changes of ash content in concentrates (Fig.6) and in tailings (Fig.7).

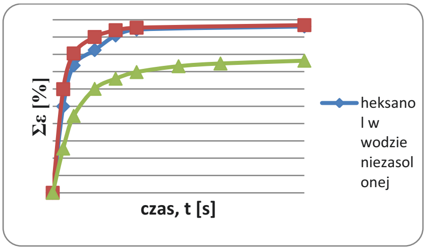


Fig.5.. Flotation kinetics curves

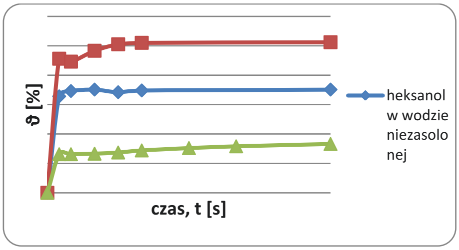


Fig.6. Changes of ash content in concentrate from time

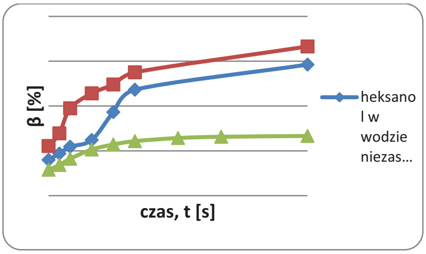


Fig.7. Changes of ash content in tailing from time

*Analysis of Results*

The analysis of results included in Table 1 and shape of curves in Fig.1 indicates a constant rising trend for recovery values in time. The shape of kinetics curves suggests that the upgrading process could continue because the balance recovery value (a plateau on the plot) was not reached. The indicated shape of kinetics curves should be expected in case of highly effective flotation (Brożek and Młynarczykowska, 2009). The course of flotation process is similar for each type of water used as shown by the shape of curves, and the highest intensity is observed in the first 3 minutes of flotation. The review of conditions improving the upgrading effectiveness indicates that in the most saline water (mine water III) the floatation reaches the best values of yield, recovery and concentrate parameters. This proves that the presence of NaCl assists in the flotation process.

It is also evident that the concentrate yield in the first 15 seconds of the process is lowest in case of distilled water (4.90%), whereas for mine water I it reaches 9.54 %, mine water II – 7.48%, and mine water III – 9.77%. The yield increase and also an analogous recovery increase by 1/3 or even 1/2 in comparison with flotation in distilled water clearly indicates a positive impact of salt on the properties of grains subjected to flotation and also an improvement of gaseous phase properties under such conditions. (Rodrigo et al., 2005, Brożek and Młynarczykowska, 2012).

Because in industrial conditions narrow grain size classes are not upgraded, the successive analyses include the results of flotation tests conducted for grains below 0.5 mm. The analysis of results showed in Fig.6 indicates that the greatest process speed for two flotations with hexanol was obtained in the first 2 minutes of upgrading, and the process without hexanol stabilized in the 3$^{rd}$ minute when 69.73% concentrate yield was achieved in flotation without reagent and almost identical result (difference of about 1%) for flotation in saline and non-saline water with addition of hexanol. It should be noted that both flotations with hexanol were similar, and the kinetics curves are practically identical "which would suggest similar properties of phases remaining in contact during the flotation" (Młynarczykowska et.all, 2013). Only during the first 15 to 105 seconds of the process, the obtained recovery results differed by about 10% in each successive concentrate collection. An initial recovery increase was observed for floatation in saline water, but it was later reduced which is shown in the plot as another convergence of curves in the same point (about 95% concentrate recovery). The final result was 96% recovery for flotation with hexanol in both the NaCl saline water and non-saline water. The recovery in flotation without reagent was however significantly lower and reached 76.29%. It is evident that hexanol increases the coal flotation effectiveness and at the same time it can be stated that water salinity facilitates the uplift of coal grains. The ash content in the concentrate in the non-saline water without hexanol varies in the 1.31-1.66% range, in the 3.28-3.51% range in non-saline water with hexanol, and in the 4.56-5.12% range in saline water with hexanol.

In order to evaluate the flotability of the tested material, the experiments results were presented as Henry curves. "The ideal upgrading on the Henry curve is when the relationship between the yield and the content of useful ingredient is a horizontal line" (Drzymała, 2001). If the relationship is like this, then we obtain only concentrate containing pure useful ingredient. When the upgrading line is vertical and the content of useful ingredient remains constant and equal to α, then there is no upgrading. The analysis of curves in Figures 8 and 9 indicates that the flotation reagent (hexanol in this case) gives high concentrate yields in both saline and non-saline water. The concentrate yield reached the 95% mark during the flotation, and 90% during the process in the saline water. Such yield values substantiate the statement that the coal floated with addition of hexanol becomes easily upgradeable.

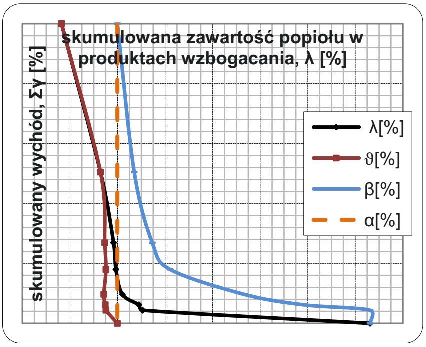

Fig.8.Washability curves in non saline water

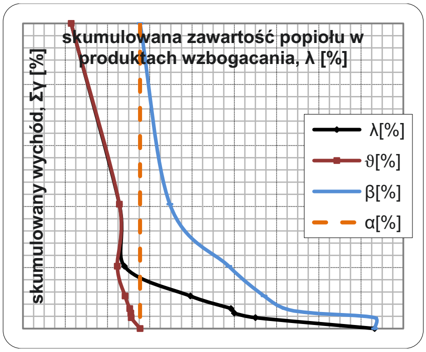

Fig.9.Washability curves in saline water

**Summary**

In the light of literature on the flotation upgrading in the presence of inorganic substances, mainly salt, it seems appropriate to distinguish the two terms: salt flotation conducted in conditions of increased surface tension in relation to water, and flotation conducted in the presence of salt in the pulp where the surface tension is reduced. The process conducted in the presence of salt is accompanied by increased froth stability, which rises as the concertation of frothing agent increases. The froth stability is also very closely related to the bubble size, namely: the smaller the bubbles, the more stable the froth. This conclusion should be verified for each frothing agent used (Wang and Qu, 2012). The froth stability in coal flotation and the increased content of NaCl in process water improves the flotation effectiveness, with maximum recovery at the 90% level. In addition, it should be noted that high flotation effectiveness depends on the feed grain size. Although narrow grain size classes are not upgraded in industrial conditions, a verification of process effectiveness for small grain size ranges allows to forecast their behaviour in real conditions. Hence, on the basis of the presented results one should expect the process effectiveness of 40-90% recovery when upgrading the (0.315-.0063)mm grain size class in saline water. Thus, coal flotation in saline waters brings benefits: conscious control of upgrading to increase its effectiveness, and reduced consumption of reagents which means lower costs. And most importantly, it allows the mines to recirculate the process water. The results of laboratory experiments prove that using an aliphatic alcohol, in both non-saline and saline waters, gives the concentrate yield in excess of 90%. This type of reagent improves the effectiveness of coal slurry flotation, and water salinity facilitates lifting the coal grains to the froth product. In addition, hexanol adsorbing on the surface of bubbles creates durable flotation aggregates, thus increasing the probability of floating the coal grains. Application of this flotation reagent gives high ash content in tailings, in excess of 16% in saline water, with the ash content in concentrates not exceeding 6%. The very purpose of coal upgrading is to maximize the content of combustibles and volatiles and to minimize the ash content in concentrates. Such results are guaranteed by the use of hexanol which has an apolar carbon chain as proven by the laboratory tests. In addition, it should be noted that:

- the process effectiveness is much better in case of upgrading the materials with a wider grain size range than for narrow ranges floated separately, giving much higher process effectiveness and good quality upgrading products. Wider grain size class and lower content of solids in the flotation pulp allows to reduce the reagent consumption, giving at the same time the better upgrading results;

- excessive salt content in process water causes a transfer of a significant amount of salt to the products of upgrading, thus increasing its yields and simultaneously reducing the technological parameters;
- use of saline mine waters reduces the costs of technological processes by limiting the demand for fresh water and improves the environmental protection by reducing the amount of mine wastewater discharged to rivers.

ACKNOWLEDGEMENTS

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# VOSTOCHNAYA COAL WASHING PLANT IMPROVEMENTS

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***Steve Frankland, Director Dargo Associates Ltd***

**Abstract**

As part of a continuing programme of modernisation, Arcelor Mittal has installed two Jameson flotation cells and a horizontal belt vacuum filter. The paper discusses some of the challenges faced and overcome when retro-fitting the equipment into the old plant.

The Vostochnaya plant is one of Arcelor Mittal's two coal preparation plant's in Kazakhstan, serving the steel plant in Temirtau as well as exporting coking coal. The plant treat a combined 10 million tonnes of coal per year providing high quality coking coal and middlings for power generation.

The results obtained from the new equipment will be presented as well as lessons learned to be applied to future upgrade contracts.

**Keywords**

**Flotation, Jameson, Filtration, Upgrade, Karaganda, Flowsheet, Coking Coal, Concentrate, Washability, Modernisation.**

## 1 BACKGROUND

Arcelor Mittal acquired the Temirtau Steelworks and associated iron ore and coal mines in Karaganda in 1994. The mines are located the Karaganda Region within 40 km of the town of Karaganda (see Figure 1).


**Figure 1 Map of Kazakhstan**

Figure 2 shows the location of the mines and coal cleaning plants around the town of Karaganda.

The complex includes 8 coal mines and two centralised coal cleaning plants. In 2008 a programme of improving the coal cleaning plants was commenced to bring the technology used in line with modern practice and to improve plant efficiency.

In 2010 fine coal treatment was improved at both plants by installing hydrosizers (Teeter Bed Separators or TBS). An improvement of 4% increase in plant yield was expected but the results was 5% increase[i].



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The Vostochnaya plant is the larger of the two plants and located closer to the mines. The plant contained some 88 mechanical froth flotation cells in 11 banks of 8. Each cell had a 50kW motor. The total amount of coal treated in the flotation section is 100 tph and it was felt that this could be replaced be more space efficient pneumatic flotation units. If space could be created then dense medium cyclones could be installed in the existing building.

This paper describes the implementation of pneumatic flotation and the results obtained with some speculation on future improvements.

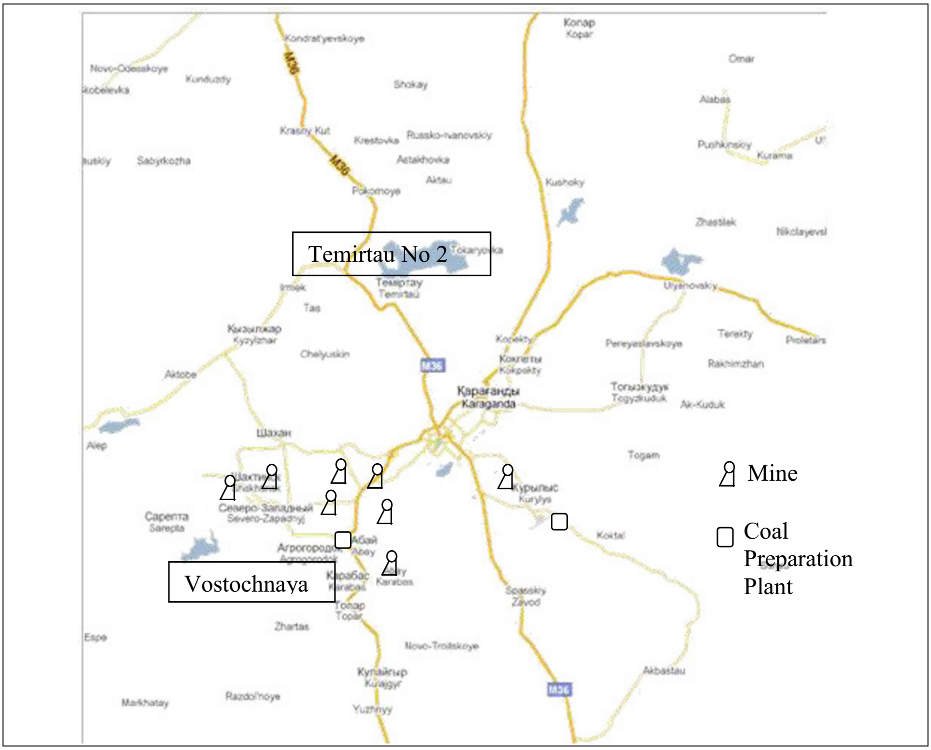


**Figure 2 Location of Mines around Karaganda**

## 2 PROCESS SELECTION

Processes considered were types of column flotation and Jameson cells. Considerable due diligence was done on the alternatives including a technical visit to 5 mines in Australia[ii].

After discussions with operators and consideration of the layout requirements for Vostochnaya, Jameson cells were chosen.

As it would be necessary to remove some of the existing disc filters, it was also decided to install a horizontal belt filter to replace all of the existing vacuum disc filters.

The existing water circuit at Vostochnaya is very complex with considerable recirculation between the different processes. This will be simplified in the future but for this project the water circuit must be generally maintained.

It was decided that two Jameson cells would be required to treat 100 tph of flotation feed with a nominal top size of 0.2 mm. After consultation with Australian operators and with Xstrata, a serial configuration was chosen in preference to operating the two machines in parallel.

After a tendering process, MEP was chosen as the main contractor to provide all engineering, equipment, construction oversight and commissioning.

## 3 CONSTRUCTION

Two half banks of flotation cells were removed from service and demolished together with three disc filters, in order to make room for the new Jameson cells and the Delkor Filter.

MEP's flow diagram is shown in Diagram 1.

Construction took 18 months from the award of contract to the start of commissioning.

The installation is compact fits well into the existing plant as shown in Figure 3, Figure 4 and **Error! Reference source not found.**.

**Figure 3 Jameson Cell and Delkor Filter Installation**

**Figure 4 Jameson Cell Installation**

All equipment has easy access for maintenance with the overhead crane.

The Jameson cells comprise two 7m diameter units with 24 downcomers per unit. Jameson cells work by pumping feed slurry through an orifice and entraining air pulled in by venturi action as shown in Figure 5.

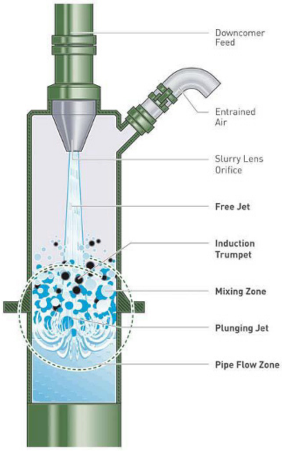


**Figure 5 Jameson Cell Downcomer[iii]**

Pressurized slurry enters downcomer through a nozzle at high velocity (typically 15-17 m/s) and a typical feed pressure of 130-170 kPa or 19-25 psi. The jet plunges into the slurry surface causing the entrained air to shear into fine bubbles. Residence time in each downcomer is only several seconds. Slurry and the collected particles exit downcomer into the tank where particle laden bubbles are separated from the pulp.

The Delkor Filter is 4m wide and 20m long. Vacuum is provided by a single liquid ring vacuum pump model 2BE4 from Nash.

## 4 RESULTS

Initial results with the Jameson cells were somewhat disappointing. Despite assistance from Xstrata, Nalco and other, final tailings ash content remained stubbornly below 60%. Froth ash was however, very good normally at about 6% to 8%.

The froth also contained a high percentage of coarse material (plus 0.5mm) demonstrating that the Jameson Cell is capable of floating coarser coal. However the presence of plus 0.5mm material indicated that the plant hydrocyclones were cutting at too coarse a size and new vortex finders were installed.

Following the change tailings quality improved significantly and now 70% ash tailings are commonly reported. The results for September 2015 are summarised in Table 1

| Stream | Average Ash % | Range Ash % |
|---|---|---|
| Feed | 23.7% | 13.3% - 34.5% |
| Primary Concentrate | 6.7% | 4.8% - 10.7% |
| Primary Tailings | 43.9% | 32.4% – 59.1% |
| Secondary Concentrate | 12.3% | 9.9% - 18.1% |
| Secondary Tailings | 61.5% | 44.5% - 72.5% |

**Table 1 Jameson Cell Results for September 2015[iv]**

Results continue to improve and as operator experience increases it is expected that results will also improve. There is no doubt that the variability of the plant feed has an adverse effect on the results. However, the mix of coal being fed to the plant varies from hour to hour as source coal availability changes. Homogenisation of the plant feed would improve consistency, but this will have to wait for a later project.

The Delkor filter has operated well from the start. Cake is consistently thick (see Figure 6) and with a cake moisture of 23% which is 3% lower than the vacuum disc filters previously in use.

Presently the Delkor Filter is not able to process all the flotation concentrate and the vacuum disc filters are still being used. A project is currently underway to utilise one the four tailings thickeners to thicken the feed to the filter. This will greatly increase the throughput of the filter and further reduce the moisture content of the cake.

**Figure 6 Delkor Filter Cake**

## 5 COMMENTS

The installation of the Jameson cells has been a great success for the Vostochnaya Washery. Despite some initial problems with lower than expected yield, changes to the plant circuit have resulted in much better results.

The solids content of the froth from the Jameson cell is less than expected. The froth does not overflow from the cell unless it is maintained with a high degree of mobility. This essentially means that the solids content must be kept at a low level. Attempts to increase froth depth have resulted in a lack of mobility and lower overflow rates.

The result of the lower solids content is that the Delkor filter cannot deal with the full output of concentrate from flotation. Fortunately Vostochnaya plant has four 30m diameter thickeners, which is more than it needs for thickening of flotation tailings. One of these will be converted to a froth thickener to reduce the water that needs to be removed by the Delkor.

## 6 FUTURE PROJECTS

The Temirtau No 2 Plant feeds directly into coke ovens and high moisture is causing long term damage to the ovens. To reduce the moisture, plate and frame filter presses have been installed to replace the existing vacuum disc filters. These should reduce the moisture content of the flotation concentrate from 25% to less than 20%, thereby reducing the total concentrate from 11.5% to 10.5% moisture.

At the time of writing this paper the filters have just been installed and no results have been confirmed. More details of the results will be presented at the conference.

It is still envisaged to install dense medium cyclones into Vostochnaya and a design study contract is about to be let. The cyclone plant will comprise one or two modules of two stage cleaning with a capacity of 600 tph per module.

Each module will be complete with desliming screens and new centrifuges and will fit in the space created by the replacement of the mechanical cells as previously described.

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# THREE-PARAMETR PHENOMENOLOGICAL COAL FLOTATION MODEL

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**Abstract**

Coal flotation process is a very complex one, the overall result of which depends on the partial results of many micro processes and parameters. The particle interaction which very often takes place in the flotation systems makes the phenomenon even more complex.
The two-parameter flotation kinetic model with particle interaction was already introduced by author in 1994 and further developed in 2000. In this paper based on the experimental results obtained with Hungarian Liassic bituminous coal the new, three-parameter phenomenological model is introduced. This general model based on the universal Tóth-isotherm, describes the flotation kinetics more precisely; all the parameters can be interpreted. One of the parameters is the interaction affinity, which can be experimentally determined much easily than by two-parameter flotation kinetic model with particle interaction
Keywords: flotation kinetics, two- and three parameter model, particle interaction and particle size.

## Interactions between solid particles

The most practical flotation systems are *real* flotation systems in which the interaction of particles – mostly solid-solid ones – has to be taken into account (Bokányi, 1996), while in the *ideal* systems the particle interaction can be neglected.

Let us sum up the most important particle-particle interactions. The aggregation (adhesion) interaction of solid particles can be expressed as the sum of their particular interactions, according to the following formula (Parsonage, 1992):

$$V_{total} = V_A + V_R + V_B + V_{ass} + V_{hph} + V_{hydr} + V_{cap} + V_{floc}, \qquad (1)$$

$V_A$ - van der Waals interaction;

$V_R$ - Coulomb electrostatic interaction;

$V_B$ - Born electrostatic repulsion;

$V_{ass}$ - adhesion caused by the association of hydrocarbon chains

$V_{hph}$ - hydrophobic interaction;

$V_{hydr.}$ - interaction caused by hydration;

$V_{cap}$ - capillary interaction through the oil film;



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$V_{floc.}$ - flocculation caused by the attachment of polymer chains (bridge mechanism);

$V_{total}$ - aggregation (adhesion) interaction.

In the coal flotation systems, $V_{floc}$ is often negligible.
According to the DLVO theory (Verwey and Overbeek, 1948), the van der Waals and the electrostatic interaction depends on the distance between the particles, the Hamaker-London constant, as well as the *size of the particles*.

The Born repulsion, which is caused by the overlap between electron clouds when the particles get very close to each other, is, according to Feke and his fellow researchers (Feke, 1984), proportional to the Hamaker constant and depends on the distance between the particles and their *size*.

$V_{ass}$, the associative interaction of hydrocarbon chains, arises when the adsorbed hydrocarbon layers overlap each other. This interaction depends on the decrease of free energy when the hydrocarbon chain leaves the aqueous environment and enters the hydrocarbon one, the surface concentration of the -$CH_2$ - group, the volume of the overlap and the *size of the particle.* This associative interaction is a sort of hydrophobic interaction (Parsonage, 1987).

The long-range attractive hydrophobic interaction (Škvarla, 1993) is caused by the association of two adsorbed hydrophobic layers, the driving force of which results in the displacement of water between the particles and in an attractive force between the hydrophobic layers. This interaction primarily depends on the thickness of the adsorbed layers, the distance between the solid particles and the *size of the particles*.

The solid-solid hydration interaction is a short-range repulsive interaction between the hydrated surfaces which also depends on the hydration factor (Derjaguin and Churaev, 1974). The hydration interaction makes the adhesion of ultra-fine particles to hydrophilic surfaces difficult, which in turn takes place easily on hydrophobic surfaces. This leads to the phenomenon of „entrainment" when the ultra-fine hydrophilic particles adhered to the larger hydrophobic particles float, worsening the quality of the froth product. Derjaguin called the adhesion of ultra-fine particles to the surfaces of larger particles adagulation (Derjaguin et al., 1987).

The capillary interaction results in the agglomeration of solid particles through the oil film adsorbed on solid surfaces. It is a short- and long-range interaction. The magnitude of the attractive force depends on the *size of the particle* and the surface tension (Schubert, 1984).

### Competitive interactions of particles

Thus, the particle size is one of the most important factorof the individual solid particle-particle interaction. Dynamic flotation systems, however, represent a bulk of heterogeneous particles with different sizes being in turbulent flow. The individual particle-particle interaction only does not explain the experimental results for real flotation systems. So we can conclude that a bulk of particles of real flotation systems is dominated, beyond the individual particle-particle interactions, by the *competitive* interaction. The *competitive* interaction between heterogeneous particles with different sizes is further complicated by the presence of the poly-disperse gas and collector emulsion phases, as well as by the interactions involving them. As opposed to the interaction of individual particles, the literature of collective interactions in a bulk of particles is still insufficient.

The micro-process, determining the efficiency of flotation, is the collision of a solid particle with a hydrophobing particle (molecule, ion or drop) and the adsorption of the latter ensuring the hydrophobisation of the solid surface. The probability of collisions is primarily determined by the number of particles. In the unit volume of mono-disperse systems with the same concentration of solid material, the number of grains increases drastically with the decrease of the grain size and, for the same number of hydrophobing particles, the probability of collisions between solid particles and hydrophobing particles decreases continually. The probability of adsorption and hydrophobisation would favour the coarsest solid particles but, as they are mostly inhomogeneous (intergrown) particles, the medium-sized solid particles are preferred to the other ones, as far as this micro-process is concerned.

In the poly-disperse system, the probability of collision between solid and hydrophobing particles depends, beyond the particle number, on the relative velocity of solid particles. Thus the micro-processes of collision and hydrophobisation are *competitive*, not only because of the character of the particles (sort, intergrowth) but also because of the *grain size,* as well as the relative concentration of grains in the individual grain size ranges.

The collision of a solid particle with a gas bubble and its adhesion to it is the second fundamental micro-process determining the result of flotation. The probability of such collisions and adhesions largely depends on the number of gas bubbles, as well as the proportion of solid particles and gas bubbles. In the unit volume of mono-disperse systems with the same concentration of solid material, this micro-process has higher probability in the systems of coarser particles – at constant gas bubble number – with regard to particle-bubble collision and adhesion.

In the poly-disperse system, the fine and ultra-fine grains compete with the coarse and medium-sized ones because of their greater mobility and higher aggregative particle-particle affinity. Gas bubbles may also participate in the aggregation.

In the poly-disperse system, the micro-process of collision and adhesion between solid particles and gas bubbles is also the result of *competitive* interaction and depends on the size distribution of particles and the relative concentration of grains of a given size.

The third determinant micro-process of flotation is the vertical transport of particles adhered to gas bubbles and their remaining in the froth layer for a sufficiently long time. In the unit volume of mono-disperse systems with the same concentration of solid materials, the probability of this vertical transport increases with the decrease of grain size (Rubinstein and Philippov, 1980).

In the poly-disperse system, the vertical particle transport has *competitive* character, too. The large grains can only be transported by sufficiently large bubbles or several small bubbles, one by one. The smaller grains, particularly the fine and ultra-fine ones, can be floated by means of bubbles of any size available. So, the competition of solid particles for the bubbles of an appropriate size – in case of appropriate relative concentration of fine and ultra-fine particles – favours them, especially in self-aerated cells with the Gaussian air bubble size distribution.

**Experiments and flotation kinetics results**

The flotation-kinetic curves per grain fraction for coal slurry from the Hungarian Mecsek Mountains and its combustible and mineral matter components were determined in course of two-step flotation experiments in a 1-litre laboratory self-aerated flotation cell of type VRF 2 ($n = 2054\ min^{-1}$, $\dot{Q}_{air} = 100\ dm^3/h$), using 1200 g/t of BKI reagent (70 % gas oil, 20 % kerosene and 10 % amyl alcohol). We fed 40 percent of the fine emulsion of the reagent (emulsification for 20 minutes at 10.000 $min^{-1}$) into the 1-minute conditioning and the rest (30-30 percent) at the 3rd/6th minute of the 10-minute rougher flotation. During the 8-minute cleaning flotation, five froth products were obtained. The solid material concentration of rougher flotation was 125 g/L. The experimental circumstances were identical in both mono-disperse (narrow size fractions and the poly-disperse (≤0,63 mm) systems. The elementary fractions were defined according to the ash content for the mono-disperse systems and according to the grain composition and the ash content of the grain fractions for poly-disperse systems.

The values of elementary floatability and equilibrium yield according to the Huber-Panu equation

$$m(t) = A\,[1 - \exp(-K\,t)] \qquad (2)$$

(m(t)-froth product vs. time, A-equilibrium yield, K-flotation rate constant in $min^{-1}$ )fitted to these experimental flotation-kinetic curves by non-linear regression were determined.

It was established from the change of floatability and equilibrium yield per grain fraction that the floatability of particles in the same grain range is *different* in mono-disperse and poly-disperse systems, for both combustible and ash-forming components (Bokányi, 1996).

According to the further examination of floatability, it can be stated that both coal components in the ultra-fine <10 μm range of the poly-disperse system have a flotation rate constant 1.33…2.76 times higher than that of the mono-disperse system. For coarser particles, just the opposite is true, except for the ash-forming 20…32 μm particle fraction.

The elementary floatability of relatively coarse, 0.315…0.63 mm particles is about 3.2…4.2 times, that of the medium-sized ones about 4.7…7.5 times and that of the 0.01…0.1 mm ones 2.5…3.6 times higher in the mono-disperse system than in the poly-disperse one.

The combustible component of the 20…32 μm grain fraction, showing a behaviour, which is different from that of the above ones, has nearly the same floatability in both systems while its mineral matter has a floatability 1.4 times higher in the poly-disperse system than in the mono-disperse one.

Of the grains in the same fraction, usually more are floated in the mono-disperse system than in the poly-disperse one, as can be seen from the change of equilibrium yields. At the same time, from the combustible and ash-forming components in the ultra-fine <5 μm range, as well as the combustible component in the 20…32 μm range, the total mass of grains getting into the froth product is 1.25…1.77 higher in the poly-disperse system than in the mono-disperse one.

The equilibrium yield of the coarser combustible particles is about 3 times and that of the ash-forming ones is 5.4…5.8 times higher in the mono-disperse system than in the poly-disperse one. This ratio is 1.4…1.9 for medium-sized combustible particles, 3.7…5.4 for ash-forming particles and 1.08…1.23 and 1.95…2.62 for fine particles. With regard to the fact that the density of ash-forming particles is twice of the density of combustible ones, the changes for combustible and ash-forming particles can be considered as adequate.

Thus, the interaction between particles is considerable and not negligible. The examined flotation systems with considerable interaction between solid particles can be defined as *real* flotation systems.
Thus, the interactions between particles and particle collectives result in different floatability orders which can be determined experimentally. According to this, the increasing floatability order of mono-disperse systems, for both combustible and ash-forming components, is as follows:

ultra-fine < fine < coarse < medium-sized,

while for the poly-disperse system, this order is as below:

medium-sized< coarse < fine < ultra-fine.

The difference of flotation rate constants of elementary fractions determined experimentally in mono-disperse and poly-disperse systems means that the flotation rate constant comprises the result of the overall interaction between particles, as well as the resultant of the particular micro-processes of flotation. The interaction of particles with different compositions and sizes during flotation depends on the interaction affinity of individual particles and particle bulks (collectives).

**Improved kinetic model with particle interaction**

The flotation rate constant for real systems is the flotation rate constant for ideal systems corrected with the interaction affinity (Bokányi, 2012):

$$K_{real.} = k_B \cdot K_{id}, \qquad (3)$$

where $K_{real}$ is flotation rate constant in poly-disperse, real system; $K_{id}$ is flotation rate constant in mono-disperse, ideal system and $k_B$ is the ineraction affinity. The values of the latter are shown in Fig.1.

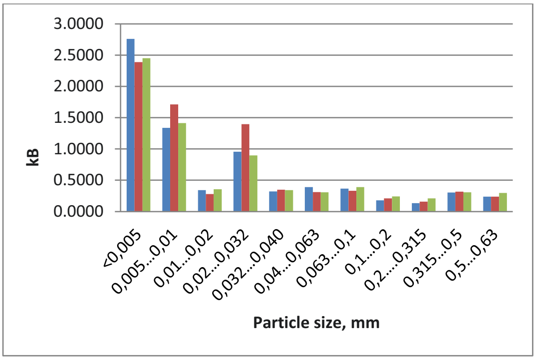


Figure 1 The $k_B$ interaction affinity for combustibles (blue), mineral matter (red) and coal (green)

It can be seen that generally the finest particles have higher interaction affinity. Exception is the fraction between 10 and 20 micro meters. The introduction of the interaction affinity is an improvement of the kinetic model published in 1996.

**Three parameter phenomenological coal flotation model**

It was József Tóth who advised the author to test his universal isotherm equation (Tóth, 1971) as a base for a phenomenological flotation model. The aim of the creation of a newer model was to better, more precisely describe the flotation kinetics and to simplify the work- and time-consuming experimental determination of the interaction affinity.

The symbols of the parameters were changed to adjust them to the flotation process in the form below:

$$m(t) = (B^{1/k}t)/(1/D+t^k)^{1/k}, \quad (4)$$

where B, D and k –parameters, m(t)-froth product vs. time.

The equation is better fitting the kinetic profiles as compared with Huber-Panu eq. as it was proved ($R^2$ =0.999). To interpret the flotation meaning of the B parameter of the Tóth-flotation model to be applied for flotation, the numerical calculation was carried out at the constant D and k values (Bokányi, 2012). Only the equilibrium yield has been varying in this case, the slope of the curves has not. It was established therefore, that the B parameter of the Tóth-flotation model is the 1/k-th root of the equilibrium yield:

$$B^{1/k} \equiv A. \quad (5)$$

To interpret the flotation meaning of the D parameter the numerical calculation, as well as the dimensional analysis was carried out as follows:

$$m \equiv (1*min)/[(1/1/min^k)+min^k]^{1/k};\ m \equiv (min)/[min^k+min^k]^{1/k}\ ;\ m \equiv 1$$

The dimension of the D parameter of the Tóth-flotation model is $1/min^k$, what means that this parameter is adequate to the flotation rate constant K raised to the 1/k-th power:

$$D^{1/k} \equiv K. \quad (6)$$

It was proved experimentally by flotation with different solid concentration, that the flotation 1/k parameter is the particle interaction affinity parameter. This parameter is the adequate to the Tóth adsorption interaction affinity parameter and the relationship between it and the $k_B$from two-parameter kinetic model is illustrated by Fig. 2.

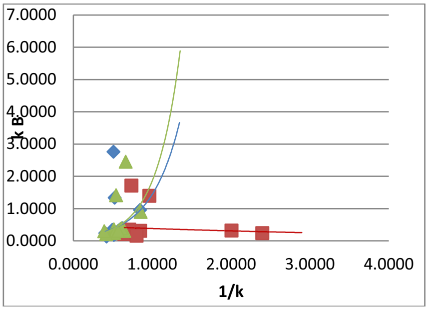


Figure 2 The $k_B$ interaction affinity for combustibles (blue), mineral matter (red) and coal (green) as a function of 1/k interaction affinity (Tóth)

## Conclusions

Based on the experimental results obtained with Hungarian Liassic bituminous coal the new, three-parameter phenomenological model is introduced. This general model based on the universal Tóth-isotherm, describes the flotation kinetics more precisely than Huber-Panu equation. All the three parameters can be interpreted. The k parameter is the interaction affinity, which can be experimentally determined much more easily than the $k_B$ interaction affinity by two-parameter flotation kinetic model with particle interaction. This parameter is an adequate one to the Tóth adsorption interaction affinity parameter. The k and $k_B$ are related to each other.

## Acknowledgement

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# "Forensic engineering science" to aid process control of flotation circuits when processing difficult-to-float coal blends.

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**Abstract**

*Keywords: flotation, coal characterization, ash minerals characterization, slurry rheology, petrographic analysis.*

This paper presents an evaluation of flotation and downstream circuits from a coal processing plant producing metallurgical coal in Western Canada. The objective of the test work was to understand the causes for the occasional poor flotation performance when certain coal blends were processed at this plant.

The specific objectives were to determine what differences existed in the composition of coal between the poorly- and normally-processing "standard" coal blends that can explain the observed poor flotation performance. For this study, the samples were taken from around the flotation circuit and from the thickener when "standard" blends were processed at the plant and subsequently, after the problematic blend entered the plant. Also, immediately after trouble shooting responses were triggered at the plant to control the situation.

In order to accomplish this, a complete mineralogical, petrographic and surface properties' assessment of coal samples was performed and this was complemented by the mineralogical analyses of ash forming minerals, slurry rheology on the samples from around the flotation circuit and the thickener at the time when the typical standard blend (non-problematic) and the troublesome blends were processed at the plant.

## Introduction

Coal production in Western Canada comes mainly from higher rank coal deposits that are of metallurgical quality and these coals are highly sought after and are used in blends to produce high quality coke to produce steel elsewhere (British Columbia Coal Industry Overview, 2014). These coals are classified as medium volatile bituminous and are highly floatable, thus flotation is suitable to process fine fractions of these coals. However, periodically, mining operations encounter parts of the coal strata that have a high content of clay minerals which are either being placed as partings between seams or exist within the seam as sheared coal strata.

As described elsewhere (Bustin, 1982; Stach et al., 1982), sheared coal particles have in their composition a finely-disseminated mineral matter; minerals are embedded into the organic phase of the coal and this usually leads to difficulty in cleaning these coals. The sheared coal particles also have a tendency to breakdown into very fine size fractions and they are difficult to handle during processing (Holuszko et al., 2004).

The presence of clays in the coal as well as badly sheared particles in the feed to the flotation circuit was suspected to be the cause for the poor flotation and the poor thickener performance at this particular plant. Apart from physical attributes and wettability characteristics of coal (Laskowski, 2001), there is always a possibility that flotation circuits and up and downstream



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process conditions could have been responsible for the poor flotation and this has also become an important element of the investigation.

To fully understand how all these different aspects affected processing, the characterization was done on four different samples; two samples representing regular “standard” coal blends (blend 1 and blend 2) and two others representing “poorly performing” coal blends (blend 3 and blend 4) which affected flotation and thickener operation at the time when they were treated at the plant.

Initially, blend 3 was recognized as “troublesome” and plant engineers have taken several steps to alleviate the effects of poor flotation of this blend and periodic overloading of the thickener. While one of the steps was to increase flocculants addition to the thickener, the other step was to request a change of the coal blend entering the processing plant from the mining operations.

Since at this mine, coal seams are blended according to the Volatile Matter (V.M.), the blend that was chosen by mining operations represented coal with V.M. content close to the previously mined blends (sample 1 and 2).  At the same time, the coal that was chosen was a blend with lower ash content, but also of different quality in terms of its composition. This blend, which will be referred to as blend 4, was expected to be easier to process. The samples were taken from around the flotation circuits and from the thickener immediately after the trouble shooting responses were triggered at the plant. Characterization was divided into several parts: coal characterization; mineral matter (ash forming minerals, clay identification) characterization; and rheological characterization of the feed to the flotation.

**Objectives**

The objective of this project was to determine the reason for the poor flotation performance at this plant when certain coal blends were treated.

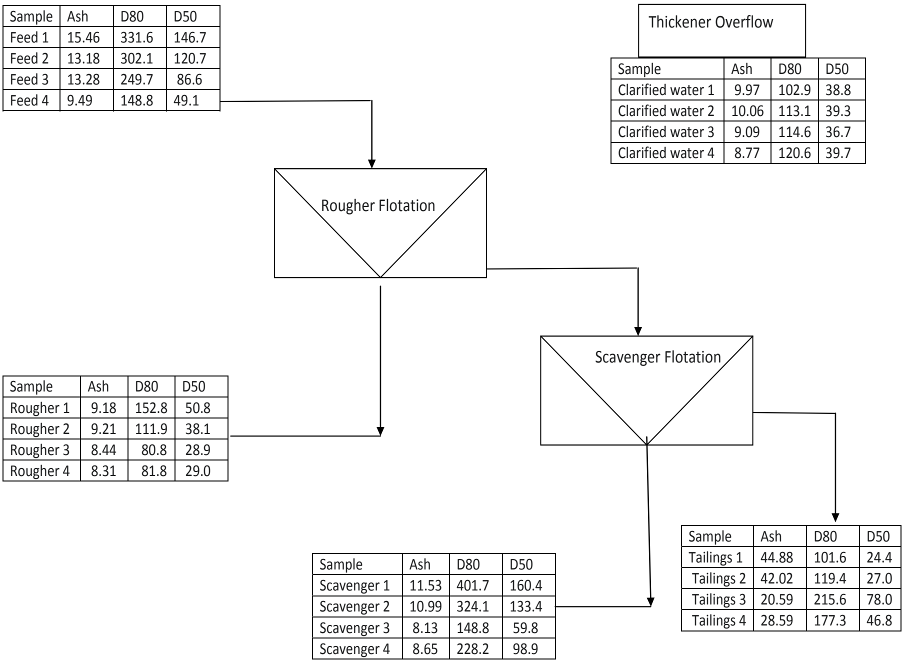
**Methodology**

Two sets of samples were collected during the period when troublesome blends were processed at the plant. These samples were obtained from the flotation cells (rougher and scavenger) and from the thickener overflow (referred to as clarified water samples). Similar sets of samples were collected on a day (morning and afternoon) in which typical flotation performance was observed for comparison. Samples representing the standard blend were identified as sample 1 and 2, while coal samples representing troublesome blends were identified as samples 3 and 4.

**Results**

*Coal characterization*

Figure 1 presents data for samples representing flotation and adjacent circuits. This data represents products from both rougher and scavenger flotation as well as for clarified water samples from the thickener. In terms of ash content blends 1,2 and 3 were consistent with the standard feed to flotation at this plant. Ash content in tailings in both blend 3 and 4 (20.59% and 28.59%) indicated that not all clean coal particles were recovered, as compared to the standard blend 1 and 2 samples (44.88% and 42.02%).




| Sample | Ash | D80 | D50 |
|---|---|---|---|
| Feed 1 | 15.46 | 331.6 | 146.7 |
| Feed 2 | 13.18 | 302.1 | 120.7 |
| Feed 3 | 13.28 | 249.7 | 86.6 |
| Feed 4 | 9.49 | 148.8 | 49.1 |

| Sample | Ash | D80 | D50 |
|---|---|---|---|
| Clarified water 1 | 9.97 | 102.9 | 38.8 |
| Clarified water 2 | 10.06 | 113.1 | 39.3 |
| Clarified water 3 | 9.09 | 114.6 | 36.7 |
| Clarified water 4 | 8.77 | 120.6 | 39.7 |

| Sample | Ash | D80 | D50 |
|---|---|---|---|
| Rougher 1 | 9.18 | 152.8 | 50.8 |
| Rougher 2 | 9.21 | 111.9 | 38.1 |
| Rougher 3 | 8.44 | 80.8 | 28.9 |
| Rougher 4 | 8.31 | 81.8 | 29.0 |

| Sample | Ash | D80 | D50 |
|---|---|---|---|
| Tailings 1 | 44.88 | 101.6 | 24.4 |
| Tailings 2 | 42.02 | 119.4 | 27.0 |
| Tailings 3 | 20.59 | 215.6 | 78.0 |
| Tailings 4 | 28.59 | 177.3 | 46.8 |

| Sample | Ash | D80 | D50 |
|---|---|---|---|
| Scavenger 1 | 11.53 | 401.7 | 160.4 |
| Scavenger 2 | 10.99 | 324.1 | 133.4 |
| Scavenger 3 | 8.13 | 148.8 | 59.8 |
| Scavenger 4 | 8.65 | 228.2 | 98.9 |

**Figure 1** Schematic flow sheet for flotation circuits at the processing plant providing information on the product samples from blends 1,2,3 and 4.

*Mineral Matter characterization*

The XRD analysis indicated that kaolinite is the main ash forming mineral in all samples (Table 1). The highest content of kaolinite is in blend 4, while the ash of this sample is the lowest at 9.45%.

F*TIR absorption spectra of coal*

The FTIR absorption spectra were collected for all feed to flotation samples and are presented in Figure 2. FTIR spectroscopy has been proven to be a useful technique in identifying flocculants such as polyacrylamides used in the processing of coal (Rogers and Poling, 1978). From the FTIR spectra of all the samples, it was evident that sample 4 had visible peaks at frequencies corresponding to the polyacrylamide presence, a flocculant that was used at this plant, hence indicating that the polymer was present at the surface of coal particles in sample 4 while entering flotation circuit.

*Petrographic Analyses of Coal*

Distributions of random reflectance in feed to flotation samples confirmed that blend 4 was not consistent with the rank of blends 1 and 2 or blend 3 (Figure 3). Maceral composition of all samples revealed a much higher content of semifusinite in blends 3 and 4, while sample 4 had the highest overall content of inertinite macerals as presented in Table 2. Increasingly more shattered, fractured and porous particles were found in the poor performing blends 3 than in the "standard" blends 1 and 2. Figure 4 shows the example of sheared particles.

Table 1: Results of semi-quantitative phase analysis (wt.%)

| | Sample1 | Sample 2 | Sample 3 | Sample 4 |
|---|---|---|---|---|
| Macerals | Feed | Feed | Feed | Feed |
| Vitrinite | 68.20 | 66.20 | 63.80 | 60.60 |
| Liptinite | 0.80 | 0.60 | 1.20 | 0.80 |
| Semifusinite | 22.40 | 25.60 | 28.80 | 33.20 |
| Fusinite (1) | 4.00 | 5.00 | 3.20 | 1.80 |
| Inertodetrinite (2) | 4.60 | 2.60 | 2.60 | 3.60 |
| Total (1+2) | **8.60** | **7.60** | **5.80** | **5.40** |

Table 2: Maceral composition data for feed samples

| | Sample1 | Sample 2 | Sample 3 | Sample 4 |
|---|---|---|---|---|
| Macerals | Feed | Feed | Feed | Feed |
| Vitrinite | 68.20 | 66.20 | 63.80 | 60.60 |
| Liptinite | 0.80 | 0.60 | 1.20 | 0.80 |
| Semifusinite | 22.40 | 25.60 | 28.80 | 33.20 |
| Fusinite (1) | 4.00 | 5.00 | 3.20 | 1.80 |
| Inertodetrinite (2) | 4.60 | 2.60 | 2.60 | 3.60 |
| Total (1+2) | **8.60** | **7.60** | **5.80** | **5.40** |

*Slurry rheology*

Rheology of the slurry was carried out on the four samples representing feed to flotation. Rheological measurements were conducted using a Haake Rotovisco VT550 rotational viscometer., Flow curves were obtained from shear stress versus shear rate tests according to Haake, (1989) and Klein et al, (1995).

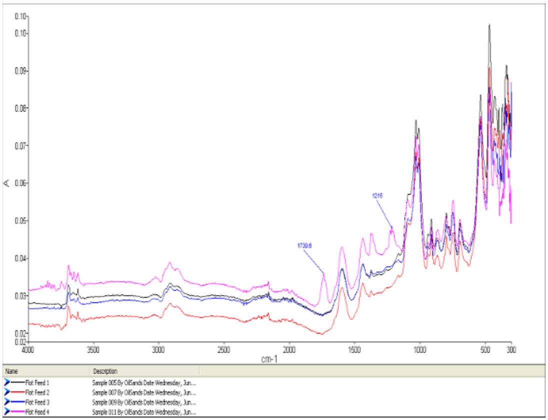



**Figure 2** Spectroscopic spectra obtained from FTIR for feed to flotation samples (blend 1,2,3 and 4).

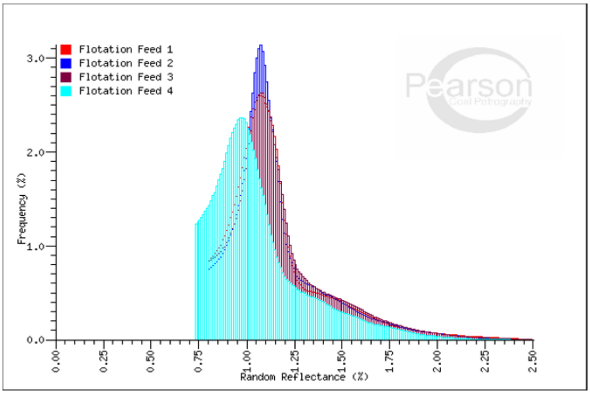



**Figure 3.** Random reflectance distributions for feed to flotation samples (blends 1,2,3 and 4).

Table 2. Maceral composition data for feed to flotation samples.

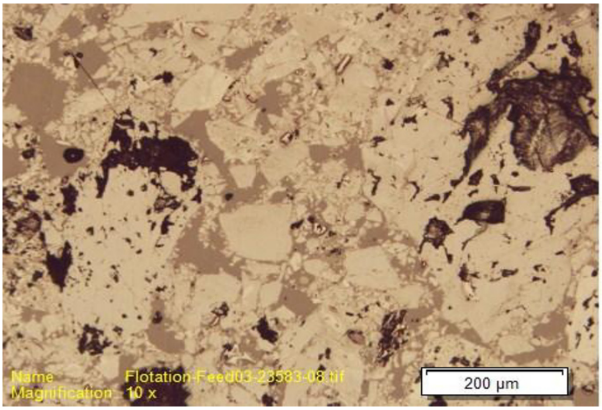



**Figure 4.** Photomicrographs of sheared particles found in blend 3

The apparent viscosity obtained from rheological measurements for both feed to flotation samples of “poor performing” blends was much higher than for “standard blends”, indicating different behaviour in the flow under mixing conditions for the poor performing blends. While samples were tested (at “as received” slurry content) it was not clear whether the slurry solids content represented the actual solids content in the flotation cell or if the received solids contents were a result of the sample handling conditions (evaporation, spilling, etc). Figure 5 presents the correlation of the apparent viscosity versus the shear rate for the tested coal samples at the slurry level at “as received” basis and in addition, for sample 4 at a slurry density comparable to the 3 other samples. Shear stress-shear rate curves were obtained using the Casson model and were used to derive apparent viscosity versus shear rate curves as presented in Figure 5.(Casson, 1959).

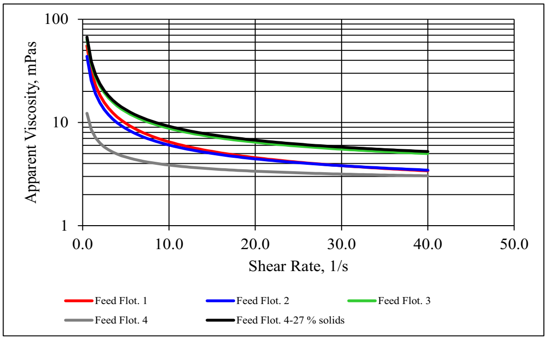



**Figure 5**. Apparent viscosity vs. shear rate curves for tested coal samples as obtained from the Casson model. Slurry solids content was as follows: feed 1=27.8%; feed 2=26.4%; feed 3=27.5%; feed 4=20.4%; feed 4=27% adjusted

**Conclusions**

The following conclusions regarding the characterization of coal samples from flotation circuits treating difficult-to-float coal blends were drawn:

- Sheared coal particles in the blend 3 along with the fine size consist of this coal and coupled with a high clay minerals content lead to the poor flotation response at this plant. An increased content of semifusinite possibly also contributed to the inferior flotation response, as semifusinite is a slower floating maceral (Holuszko and Mastalerz, 2015).
- Coal blend 4 was intentionally sent to alleviate problems that arose during the processing of the troublesome blend 3. Upon investigation, the distribution of (Ro), random reflectance revealed that blend 4 was of a lower rank, but due to its high content of inertinite macerals still conformed to the required V.M content and was used as the feed to the plant.
- Although coal blend 4 had low ash, its flotation performance was also very poor. It was concluded that the presence of flocculant found on the surface of coal particles from sample 4 indicated that flocculant was present during flotation, hence adversely affected flotation performance.
- Rheology of the coal slurries indicated that poorly performing blends (3 and 4) had higher apparent viscosity, which could be the reason for experiencing high torque condition in the thickener when these blends were processed at the wash plant.
- This study showed how the systematic investigation of the coal samples around the processing circuits revealed the causes of poor flotation of studied coal blends.

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# Part XI
# Dry Coal Separation

# Perspectives of Reduced Water Consumption in Coal Cleaning

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## ABSTRACT

During cleaning of high-ash coal mainly "wet" processes are used which require 5 to 10 tons water consumption per 1 ton of coal. Arrangement of recycling water supply reduces demand in "fresh" water, but transportation of huge volumes of water slurry requires high energy consumption. Dry cleaning of low rank coal which has not been exposed to preliminary preparation is inefficient. It was suggested that to provide dry cleaning of high-ash coal it would be reasonable to expose it to chemical heat treatment first, and then to direct the treated coal mass for physical and mechanical cleaning to get low-ash high-caloric product.

It has been determined that in black coal exposed to medium temperature pyrolysis, as well as in brown coal, improvement of incombustible mineral fraction liberation is observed that facilitates further beneficiation with the use of combination of high intensity magnetic separation and triboelectrostatic separation.

It has been determined that cleaned semicoke substantially exceeds both initial and cleaned coal by its qualities as a solid fuel, and tailings of semicoke dry cleaning can be utilised.

Key words: Low Rank Coal, Dry Cleaning, Dry Processing, Semicoke, Magnetic Separation, Triboelectrostatic Separation.

## INTRODUCTION

Mined coal is exposed to long processing flowsheet ending with its use as a fuel in energy industry, metallurgy or chemical industry. There are several stages in this processing flowsheet at which the largest economic waste occurs and environment is subjected to damage. For example, during coal cleaning mainly "wet" processes are used which require 5 to 10 tons water consumption per 1 ton of coal. Arrangement of recycling water supply reduces demand in "fresh" water, but transportation of huge volumes of water slurry requires high energy consumption.

Long-distance transportation of commercial coal is associated with expenses occurring due to movement of relatively low-caloric product containing in addition from 15 to 25% of ballasting ash fraction.

It is a real disaster for the companies of energy industry to store large volumes of coal combustion waste including ash and slag.

Considering the above processing flowsheet of coal usage after mining, one may pinpoint the following technical issues, solution of which would significantly increase its efficiency:

- transition from wet to dry coal cleaning at the places of coal mining;
- producting high caloric low-ash fuel during cleaning, which is suitable for usage both in energy industry and metallurgy;
- separation of ash fraction of coal during its deep cleaning in the form which allows to use it as raw material for commercial product manufacturing.

The accumulated experience shows that dry cleaning of high-ash low rank coal which has not been exposed to preliminary preparation is inefficient [1,2].

At the same time, there is a current steady trend in world practice to enhance coal beneficiation improving its qualities and coal product range expansion [3–5].

 
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Analysis of previously performed studies shows [6–8] that on the basis of the task set, i.e. processing of coal without water use, the following processes are of the greatest interest:

- the Green Fields Coal Co. (USA) process, including deep drying and fine grinding of coal followed by its gravity separation in air cyclones collectors;
- the Convert Coal, Inc. (USA) process, including coal pyrolisis and separation of pyrrhotite by means of magnetic separation;
- the SynCoal (USA) process of the Rosehud SynCoal Partnership (USA) company, including coal pyrolisis and separation of ash by means of pneumatic separation;
- the Thermocoke process of the Sibtermo (RF) company including partial gasification of brown coal with the subsequent separation of ash by pneumatic separation.

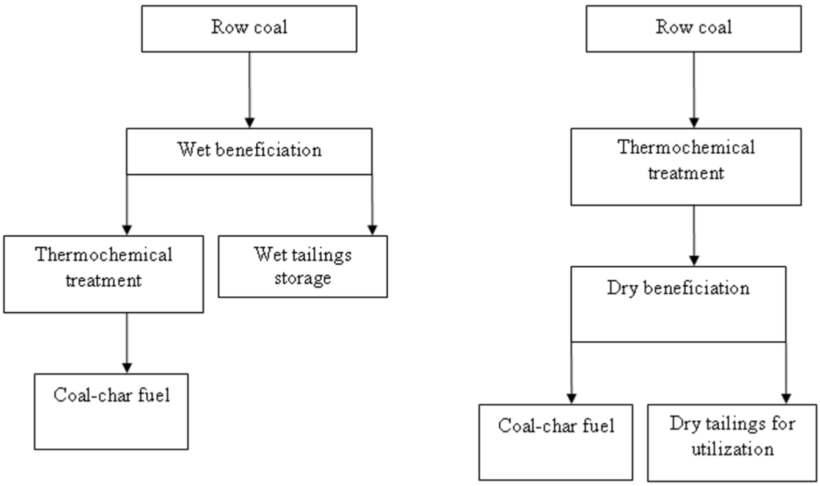
It is worth mentioning that the main purpose of the above technological approaches was production of water-free high caloric fuel on the basis of low-ash coal, mainly brown.

At the same time, testing of all the above technologies proved that during heat-treatment of coal which corresponds to the mode of coal pyrolisis, physical and chemical transformations of coal mass take place which substantially influence its further processing, i.e.:

- decrease of mechanical strength of coal due to moisture and volatile matters removal;
- exposure of particles of non-combustible mineral fraction along the boundaries of contact with the carbon part due to differences in physical and chemical properties resulting from heat impact;
- increase of calorific value of residual coal due to moisture and volatile matters removal;
- change of physical and chemical properties of ash forming minerals due to heat impact.

On the basis of the above, one may suggest that to provide for dry cleaning of high-ash coal, it is reasonable to expose it first to heat treatment, and then direct the treated coal mass for physical and mechanical cleaning to get low-ash high-caloric product (coal char fuel).

This approach is shown in Fig. 1.




a. b.

Figure 1 -High ash coal processing flowsheets: a. Wet beneficiation; b. Dry beneficiation

Implementation of the above chemical transformations provides new process potential in further processing of heat-treated coal, i.e.:

− decrease of power consumption for crushing and grinding due to decrease of mechanical strength;
− possibility of deep removal of ash fraction from coal with the use of only dry cleaning processes;
− possibility of agglomerated semicoke production;
− possibility to use burnt ash forming minerals fraction for production of special binding agents, additives to concrete and construction materials.

## EXPERIMENTAL

Analysis of the previous studies [6–8] shows that medium temperature pyrolysis at a temperature of 450–600°C is the most reasonable method of heat modification of coal.

Within this temperature range, the following process problems are solved:

− coal mass porosity increases and mechanical strength decreases, which provide good exposure of ash fraction at subsequent grinding;
− sulphide minerals being part of non-combustible fraction acquire higher magnetic susceptibility;
− clay minerals being part of non-combustible fraction lose crystal moisture, enlarge and lose capacity to water regain;
− the generated carbon material – semicoke – has high calorific value;
− the volume of generated volatile fractions is sufficient for provision of autothermal flow of medium temperature pyrolysis process.

For liberation of ash fractions in high-ash coal, it is usually required to grind material up to particle size less than 1 mm. To separate ash fractions with such particle size, it is possible to use pneumatic separation, magnetic separation and triboelectrostatic separation. The use of pneumatic separation for extraction of ash fraction out of coal has been studied rather well [9, 10] and has not been considered in this study. Studies on the use of magnetic and triboelectrostatic separation for extraction of ash fraction from coal proved their prospectivity [10–14]. The main difficulty for the processes of separation of mineral powders with the particle size less than 1 mm by their magnetic and electrical properties is provided by availability of internal friction forces in powders which hinder effective separation of mineral particles. Studies proved that, to overcome the internal friction forces in mineral powders, one may use the effect of vibrofuidization occurring during overlapping of certain vibrations [15]. The use of this effect allowed creation of effective separators for separation of fine mineral powders by magnetic and electrical properties described in [16] and used in this study.

The studies have been conducted on samples of hard coal having humidity of 1.8%, ash content of 14.8%, devolatalisation of 34.7%.

Experiments on determination of optimal temperature of coal pyrolisis have been performed in standard vessel as per Fisher. On the basis of data given in Fig. 2, t° = 550°C has been accepted as the optimal temperature of coal pyrolisis.

Semicoke sample for conducting process studies has been made at a laboratory pyrolysis unit having a 4 litres chamber, externally heated.

Pyrolisis of the coal sample under test at a laboratory unit showed that at t° = 550°C, semicoke yield makes 63.8%, yield of pyrolysis oil is 11.4%, pyrolysis gas – 20%, pyrogenetic water – 4.7%, that is in good correspondence with the results received by Fisher's method.

Coal with particle size 10–20 mm has been treated. Samples of raw coal and semicoke were grinded in laboratory hammer grinder up to 0–2 mm particle size. The results of classification of grinded products by particle size and distribution of ash fraction by particle sizes are given in Fig. 3. The data in Fig. 3 shows that thermo chemical treatment of coal contributed to substantial improvement of ash fractions liberation – recovery of ash fractions with grinding of semicoke to particle size 0–0.5 mm makes more than 90% if compared to 70% for raw coal.

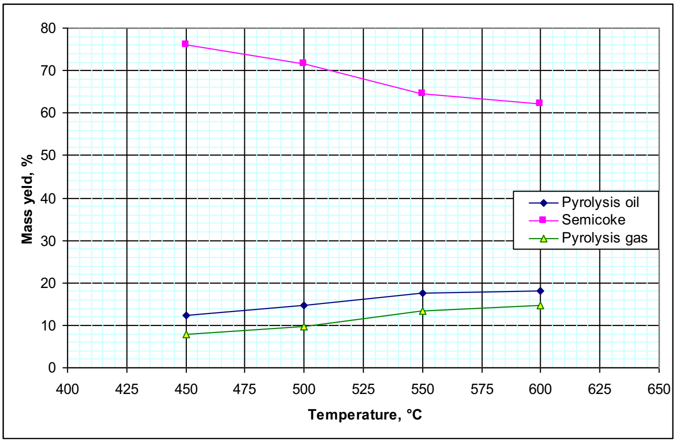



Figure 2 - The coal pyrolysis by Fisher method results.

Magnetic separation of raw coal and semicoke grinded up to particle size 0–0.5 mm was conducted in a laboratory magnetic separator in two steps – the first was at magnetic field induction of 0.35 T, the second – at 1.7 T.

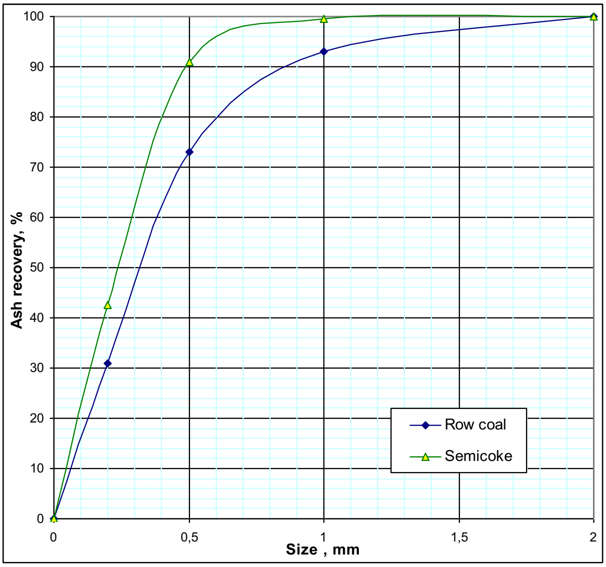



Figure 3- Size and ash analysis of row coal and semicoke grinded to 0-2 mm size

Results of magnetic separation are given in Table 1 and show that during separation of coal exposed to heat-treatment, recovery of ash fraction into low intensity magnetic product increases almost 3 fold and recovery into high intensity magnetic product increases 2 fold.

Table 1 - Results of raw coal and semicoke cleaning

| Products | Raw coal | | | Semicoke | | |
|---|---|---|---|---|---|---|
| | Mass Yield, % | Ash Content, % | Ash Recovery, % | Mass Yield, % | Ash Content, % | Ash Recovery, % |
| Low intensity | 0,6 | 55,0 | 2,1 | 2,1 | 60,2 | 6,0 |
| High intensity | 3,1 | 67,2 | 13,3 | 9,2 | 63,4 | 27,8 |
| Conductor | 9,7 | 52,2 | 32,3 | 10,1 | 58,7 | 28,3 |
| Non-Conductor (concentrate) | 86,6 | 9,4 | 52,1 | 78,6 | 10,1 | 37,9 |
| Feed | 100,0 | 15,6 | 100,0 | 150,0 | 21,0 | 100,0 |

Triboelectrostatic separation of non-magnetic product allows removal of about 30% ash fraction, that allows to get semicoke of the satisfactory quality (Table 1 and 2).

Table 2 - Comparative characteristic of studied products by ash specific yield

| Parameters | Raw coal | | Semicoke | |
|---|---|---|---|---|
| | Without cleaning | With cleaning | Without cleaning | With cleaning |
| Calorific capacity, MJ/kg | 12,55 | 15,06 | 23,72 | 25,94 |
| Ash Content, kg/kg | 0,158 | 0,100 | 0,195 | 0,100 |
| Ash Specific Yield, kg/MJ$\cdot 10^3$ | 12,6 | 6,6 | 8,2 | 3,9 |

## RESULTS DISCUSSION

The results received prove the practicability of using thermo chemical modification of high-ash black coal to increase efficiency of its dry cleaning with the use of physical and mechanical processes.

It has been determined that in black coal exposed to medium temperature pyrolysis, as well as in brown coal [8], improvement of incombustible mineral fraction liberation is observed that facilitates further beneficiation.

It has been proved that high-intensity magnetic separation and triboelectrostatic separation are effective methods of cleaning fine-grained high-ash coal. It has been determined that the efficiency of using these methods for non-combustible mineral fractionation increases after thermo chemical heat modification of high-ash hard coal. Combining high-intensity magnetic separation with triboelectrostatic separation increases efficiency of high-ash coal and semicoke cleaning. To evaluate efficiency of thermo chemical modification of coal and of cleaning both coal and products of its modification, it is reasonable to introduce such a parameter as ash specific yield per calorific capacity unit:

$$A_w = \frac{A_c}{W} \cdot 10^3, \quad (1)$$

where: $A_w$ – ash specific yield per calorific capacity unit, kg/MJ;
$A_c$ – ash content in product, kg/kg;
W – product calorific capacity, MJ/kg.

Use of this parameter (Table 2) shows:

- combined dry cleaning of coal allows to decrease ash specific yield 1.5–1.7 fold;
- semicoke from high-ash coal without cleaning insignificantly exceeds initial coal by ash specific yield, i.e. 3.44 against 4.39;
- cleaned semicoke 2.5 times exceeds initial coal by ash specific yield.

Final tailing of semicoke dry cleaning containing more than 50% of combustible mineral fractions are raw material for production of binding agent for construction, i.e. cement analogue, which can be derived by means of their combustion without additional fuel and used for back filling of coal mines.

## CONCLUSIONS

Cleaning high-ash coal without using water requires new approach to organisation of its conversion. To create conditions providing possibility of dry cleaning of high-ash coal requiring its fine grinding for exposure non-combustible mineral fractions, it is reasonable to perform thermo chemical modification of coal before cleaning. Medium temperature pyrolysis at t° = 450 – 600°C is an effective method of modification of high-ash coal providing possibility of dry cleaning derived semicoke with the use of high-intensity magnetic separation and triboelectrostatic separation.

## ACKNOWLEDGEMENTS

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# Dry destoning of coal based on XRT-separation method

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**Resume:**

The article studies the technique of dry coal preparation implemented in the sorting equipment of TOMRA radiometric separation method. The principle of work of sorting equipment is shown. The comparison with conventional technologies of coal beneficiation is provided. The possibility of various designs of the separation complex is considered. Depending on the requirements of a subsoil user a separator can be completed in a modular complex based on sea container.

The text provides brief technical results of a test work on coal deposits in Russia and the CIS. Positive aspects of the use of sorting equipment at the facilities of the coal industry are formulated.

**Key words:**
Coal, XRT-separation, coal dry processing, ash content, concentrate, rock.

Despite intensive search and active development of alternative energy sources, that has been continuing the last decade, coal stays one of the main source of heat and power in the world. Besides coal is the process feedstock for energy, chemical and other major industries.

In the context of continuous improvement of the technical level of the coal industry, the mechanization of the main and auxiliary processes in the coal production and deterioration in the quality of extracted rock mass beneficiation becomes an important step in the production of fuel that meets the requirements in terms of quality(1).

The existing worldwide practice of preparation of coal of various grade composition involves the use of gravity (wet) benefication processes, carried out in a gravitational or centrifugal media. The most widely used ones are:

- Dense media separation (DMS);
- Jigging (jig).

The main advantage of these technologies is high efficiency of processes and an ability to process the coal in a wide range of sizes. However, the use of these methods involves significant capital costs for the construction of processing plant and the establishment of the necessary infrastructure, which, in conjunction with high operating costs, significantly increases the cost of the products obtained, and in



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case of finding the field in difficult geological conditions, casts doubt on the profitability development of such deposits using conventional technology.

Dry coal beneficiation based on the X-ray absorption method (XRT) of separation is devoid of these shortcomings and is an alternative to traditional methods of wet benefication of coal, that deserves special attention (2). This method of separation is based on the difference in attenuation of the intensity of X-ray radiation flux by pieces of rock and coal (3).

Today, a European company TOMRA Sorting GmbH is the world's leading manufacturer of sorting equipment for the mining industry on the basis of radiometric benefication.

TOMRA equipment allows refining in the size range of 8-100 mm with the use of the X-ray absorption method (XRT) of separation. It is recommended to process the coal classified by class sizes with the classification module no higher than 3, for example -100 + 50mm, -50 +25 (20) -25 mm, and (20) +8 mm.

Raw material by the vibratory feeder is being discharged to the transport node of the separator (conveyer belt or sloping shoot), and then the material on the conveyor is being delivered to the radiation and registration point (Figure 1). Radiographs are processed by a special algorithm. The data transferred to the graphical view of the computer and analyzed separator. In the next step the computer makes a decision on the allocation of each individual piece of the total air flow via nozzles.

X-ray diagrams of the rocks are being processed according to a special algorithm. The data is being transferred to the graphical view and being analyzed by the computer of the separator. Next the computer makes a decision on the ejection of each individual rock of the total flow by means of jet-nozzles.

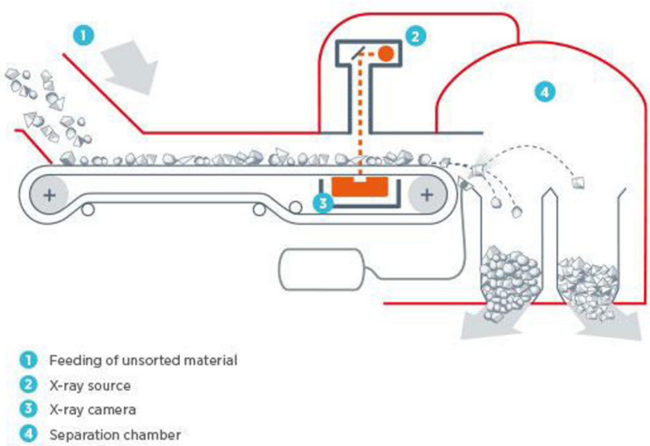


Figure 1 – Operation scheme of separator TOMRA Sorting GmbH

In general the data processing activities of XRT-separators for raw minerals are similar to scanning systems used by security services during the inspection of luggage at airports.

Volumetric analysis of the material by X-ray absorption method also influences the preparatory operations: the material fed does not require pre-washing and surface cleaning.

To optimize the conditions for measurement of rocks of different size and to reduce the influence of the material density on the signal level, TOMRA Sorting uses two different radiation detectors: a channel with low energy and a high energy channel. The computer of the separator combines the radiographs obtained through various channels, processes them, and defines materials with different atomic densities.

The XRT-separation makes it possible to extract from the stream both all rock material and coal with any amount of gangue impurities. During the separation the volume of the rock is being analyzed and the coal of the required quality is being extracted to the concentrate (according to the percentage of high-density inclusions to the total area of the piece). An example radiograph of coal samples obtained on a computer of the separator is shown in Figure 2, the red color corresponds to the atomic density of low-ash coal of the represented deposit - 1300-1400 kg/m3, and the blue color corresponds to the rock material of the atomic density more than 2000 kg/m3.

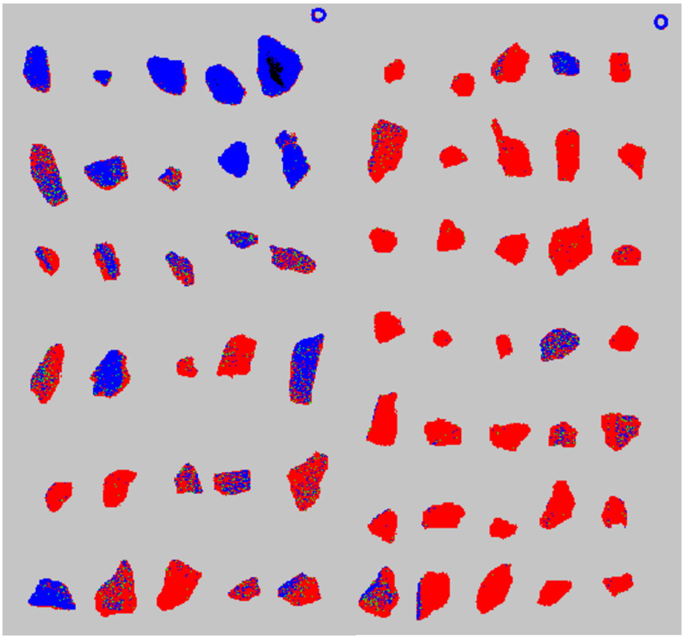

Figure 2 – An example radiograph of coal samples

The design of TOMRA XRT-separator is shown in Figure 1. Depending on the separation task and feed size the maximum feed performance of one separator with a belt working width of 2.4 m for coal is up to 300 t/h.


Figure 3 – The design of TOMRA XRT-separator

The design of TOMRA Sorting equipment and its technological features renders it possible to use separators in proximity to the mine, which allows to receive additional benefit through the reduction of transport costs. Application of the technology allows to increase productivity while maintaining the processing redistribution of production costs, or to obtain the equivalent in the form of reduced operating costs associated with the consumption of reagents, water, electricity, costs for the maintenance of the sludge storage. Separators can be mounted in a room with an ambient temperature of + 5°C or in a container, heated version (Figure 4).


Figure 4 – Containerized TOMRA separator

Option of modular unit on the basis of marine container is preferable for small deposits or those in hard geological conditions, when the construction of traditional factories is not economically feasible or technically not realizable. The use of sorting equipment is also justified in conditions of water scarcity. For example, there are two XRT-separators for sorting of thermal coal in the province of Mpumalanga, South Africa. The size classes are -120 + -50 + 50mm and 12mm, the feed performance is 150 and 80 t/h, respectively (4).

A representative office of TOMRA in Russia and CIS is based on the test center of “Trane Teknikk”.

Over the last two years Thrane Teknikk company have conducted a number of studies on the possibility of XRT-separation application to coal ranks K, T, B3 and A. In the course of each operation qualitatively-quantitative characteristics of destoning were defined. The results are presented below briefly:

– When separating the machine size fractions of -100+25mm of the K-rank coal of 17,5-16,5% initial ash content a concentrate of less than 10,7-11% was produced with the output of 65 and 70%, respectively.
– When separating machine size fractions of -100 + 10mm B3-rank coal with initial ash content 28,7-30,6% a concentrate of less than 14-19% ash was produced with the output of 60 and 70%, respectively.
– When separating the machine size fractions of -100 + 10mm of the T-rank coal of 39-42% initial ash content a concentrate of less than 26-32.5% ash was produced with the output of 60 and 70%, respectively.
– When separating machine size classes of -70 + 25mm of the T-rank coal of 15% initial ash content a concentrate of less than 8.2% at its output of 60% was produced.
– When separating machine size classes of -80 + 13mm of the A-rank coal of 11,1% initial ash content a concentrate of less than 4.3% at its output of 80% was produced.

Despite the relatively short history of its existence, the sorting equipment based on XRT-technology for the separation of coal, has attracted wide interest of subsoil users worldwide.

Rapidly developing methods of radiometric beneficiation makes it possible to achieve acceptable technological and economic results for different types of minerals.

The use of TOMRA XRT-separation renders it possible to increase an economic efficiency of coal products due to:

- low operation cost;
- high productivity;
- module and mobile features of the system allow installation in close vicinity to the mine;
- transportation cost reduce;
- elimination of water use from the process;
- reduce the negative environmental impact from wastes of the traditional "wet" coal benefication schemes.

Implementation of XRT-separation in the processing chain of the coal plants, taking into account the alleged benefits of the technology, will render it possible to essentially improve the efficiency of such plants in the part of subsoil use, benefication and provide long-term competitiveness.

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# «SEPAIR» pneumatic separation complex for dry coal beneficiation

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**Abstract**

From year to year the amount of coal which in its quality can fully satisfy customer and does not require processing is growing smaller. It is possible to provide high quality of saleable coal only by maximum beneficiation. In the context of inaccessibility of water resources and necessity to reduce treatment charge of ores and nonmetallic materials more attention is paid to the methods of beneficiation which do not imply the use of process water as operating or auxiliary environment. This problem is particularly relevant for enterprises of Siberia and high latitudes where technologies and devices resistant to significant subzero temperatures are high demanded.

Experts of «Gormashexport» company developed radically new technology of dry coal beneficiation - «SEPAIR» [1,2].

A distinctive feature of the technology and the device is brand new process aerodynamics. The construction of operating chamber of the unit combines traditional concepts of pneumatic separation and vortex technologies. This allows to separate components of initial material into products with density difference from 0,01 t/m$^3$ by means of dry coal cleaning. The efficiency is 95% and higher.

**Key words:** pneumatic separation, coal beneficiation, coal, vortex technologies, suspension bed, ash content, effectiveness of coal classification.

Back in the 20-s of the last century the attempts were made to separate coal and waste using the force of air flow but that method was accompanied by too much loss from the beginning [3,4]. In the 50-s at Kuzbass beneficiation plants 61.7% of all coals were processed by «dry» method, 8.4% were processed by combined method and 29.9% were processed by wet method [5]. Currently volumes of coal beneficiation by dry method are practically reduced to zero. This is due to the fact that the hitherto existing technologies of dry coal beneficiation allowed significant loss of salable coal together with cleaning rejects. According to the existing data content of coal in middling product and cleaning rejects ranged from 40% to 73%. For instance studies of middling product and tailings from «Ziminka 3-4» beneficiation plant produced with ПОМ-1 pneumatic separator in 1958 showed presence of coal Ж grade 41% with ash content 8.2%.

Previous machines for dry coal beneficiation like those produced nowadays in Ukraine and China simply copied the functional principle of jigging machines of hydraulic type ignoring significant change of viscosity and density of carrier medium. It resulted to low efficiency of the processes implemented in these machines. The basic differences between the various types of machines were mainly related to the unloading mechanism of cleaned products [6].

Experts of «Gormashexport» company developed radically new technology of dry coal beneficiation - «SEPAIR».

A distinctive feature of the technology and the device is brand new process aerodynamics. The construction of operating chamber of the unit combines traditional concepts of pneumatic separation and vortex technologies. This allows to separate components of initial material into products with density difference from 0,01 t/m$^3$ by means of dry coal beneficiation. The efficiency is 95% and higher. «SEPAIR» technology and pneumatic separation complex of the same name implements a completely new concept previously not used in machines for pneumatic separation of coal. Concept is in separation



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of products by density in upward air current in a vortex chamber with fluidized bed located above perforated screen [7,8]. High efficiency of the process is provided not by separation of the whole run-of-mine ore with one air flow but by double separation on the given density border for each grain in particular of the processed material in semifluidized bed.

The flow in the layer of dispersed solid phase can be described by Navier-Stokes equation with the appropriate boundary conditions. In the recent years direct numerical integration of equations of motion plays an increasingly important role for description of suspension bed. In simulation one has to use a number of hypotheses for the closure of equations of motion [9].

Despite a relatively large number of publications devoted to theoretical analysis of gas motion in suspension bed development of the theory is still far from being complete. For correct calculation of the process it is necessary to have reliable calculation methods at the design stage. Mechanical transfer of the available literature data to a particular technological problem or unfounded boundary expansion of application of specific models leads to serious discrepancies between a calculation and a fact.

Mathematical model should meet the requirements of reliability. On the other hand the model should be simple enough in mathematical respect so it could be solved with available methods and resources. When choosing a model one should rely on a reasonable compromise between the complexity of the model, the fullness of characteristics of an object received with the help of the model, and the accuracy of these characteristics. In the end the advantage of one or another model is determined by practice criterion taken in a broad sense. Complexity of mathematical modeling associated with uncertainty of some constants in equations of motion forced us to do research of aerodynamic stability conditions of suspension bed in the device, i.e. to directly conduct physical modeling of the process.

Repeatability of the process is confirmed by tens of thousands of experiments conducted in laboratory during creation of the technology as well as by tests of coals from tens of coal deposits conducted on «SEPAIR» pilot plant. Coal beneficiation is carried out on products that have passed preliminary classification, for example 1-4, 4-10, 10-25, 25-50 and 50-100 mm.

It is possible to produce any amount of products of various density on one plant. For instance if necessary it is possible to separate coal in 4-12 fractions of different density and consequently of different ash content. Currently the models of separator are developed that separate primary product in 2 and more products of different density, see table 1.

During beneficiation of easy thermal coals that contain no splices and middling product fraction a single-zone plant «SEPAIR» is used which allows to separate coal and enclosing rocks. See Figure 1.

During beneficiation of medium and difficult coals it is possible to produce low-ash coal, medium-ash coal, middling product and enclosing rocks (tailings) separately. For instance when beneficiation coking coals on three-zone plant it is possible to produce coal concentrate suitable for smelting purposes, concentrate for energy purposes, middling product and high-ash product.

Table 1 – Results of beneficiation of different coals on the plant

| Name of coal | Ash content of run-of-mine ore, A, % | Products of beneficiation |
|---|---|---|
| Bungursk coal mine. Grade T | 22.6 | Concentrate A=11.5%, second concentrate A=17.0%, middling product A=36.0%, tailings 82% |
| Chul’makansk coalpit. Grade Ж | 21.6 | Concentrate A=9.5 %, middling product A=35.0%, tailings 78.8% |
| Kusheyakovsk coalpit Grade Г | 16,4 | Concentrate A=8,2 %, middling product A=29.0%, tailings 77.6% |
| Gorlovsk deposit Grade A | 6.2 | Concentrate A=3.6%, middling product A=33.9%, |
| Kolmagorovsk coalpit Grade ДГ | 33.4 | Concentrate A=10.9%, tailings 77.8% |
| Sarykolsk deposit Grade 3Б | 23.5 | Concentrate A=13.2%, tailings 83.0 % |

Figure 1 – Coal and tailings of ДГ grade.

«SEPAIR» technology shows equally good results in beneficiation of coal both with low and high initial ash content. When beneficiation the coal obtained from «Chulmakanskaya» coalpit and the coal mine located in the same coalbed a stable quality of concentrate was obtained regardless of ash content of the coal used for beneficiation: coal selected manually from the coalbed and having a relatively low initial ash content, coal produced in mechanized mining and having initial ash content of 30-35% or coal, selected from the side of coal mine with initial ash content of about 50%. See Figure 2.

Processing of middling products and wastes from the existing beneficiation plants also confirmed efficiency of the technology because up to 20% of coal was successfully extracted.

The first commercial equipment commissioned in Kuzbass at the Bungursk coal mine, see Figure 3, was designed for beneficiation and processing of coal with capacity of 120t/h. Raw mine coal with initial ash content of 22–23%, class 14–25 and 25–40 mm is separated by «SEPAIR» equipment into low-ash coal (ash content is 8–12%), clean coal (16–18%), middling product (35%), host high-ash rock-tailings (75% and higher).

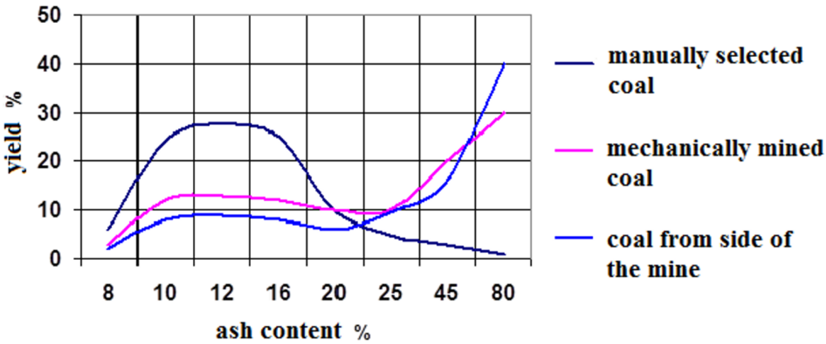


Figure 2 – Results of coal separation from Chulmakansk deposit with different initial ash content.

Figure 3 – «SEPAIR» complex. Bungursk coal mine. Kuzbass.

According to expert conclusion of the enterprise technical capabilities of «SEPAIR» allow to control the beneficiation process continuously yielding coal of necessary ash content without stopping the equipment. After setting the process does not change maintaining stability of all main indicators which is of great practical importance [10].

With all the variety of evaluation criteria of equipment performance the main criterion is always price and quality. Commercial operation of the complex confirmed the initial estimates. According to the analysis the expenses of one coal-preparation plant using dry cleaning are 3-4 times lower than the expenses of a traditional plant. At the same time dependence of production on foreign equipment manufacturers is sharply reduced. None of the technologies available today is able to show better results.

«SEPAIR» can be used in all climates. Installation of the complex requires minimum number of buildings and constructions. For operation at temperatures from 30°C below zero to 50 °C above zero it is necessary only to shelter the complex from precipitations and wind. Only the console should be in a heated room. Productivity of the proposed technology equipment varies from 10 to 500 t/h.

Not having analogues in the world the technological process and the device are protected by Russian and foreign patents. Due to the new technology incorporated in the design of the separator finishing beneficiation of raw materials allows to perform dry separation of the components efficiency of 95% at minimum cost.

The application of «SEPAIR» is not limited by the above examples. «SEPAIR» is not just a device, it is a unique technology that fits easily into almost any technological conditions. Any granular materials having different aerodynamic properties can be separated this way. It can be either ores or slags, or crushed recycled products, or granular alimentary products. This is one of rare cases in beneficiation technology when positive result is guaranteed regardless of the task and operating conditions.

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# Small Coal Dry Cleaning Jig

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**Abstract:** The small coal dry cleaning jig is a technologically innovative and fully-automatic equipment. The paper presents an introduction to the operating principle, mechanical structure, system make-up, technical features, separating performance, scope of applicability, product series and prospect of application of the small coal dry cleaning machine.The jig can be used in combination with conventional coarse coal dry cleaning system for larger scope of applicability and higher separating performance of pneumatic cleaning processes. Besides the application of the jig as an independent cleaning system, the jig can also operate in combination with a coarse coal wet cleaning system---a technical approach for drastic reduction of initial capital cost. Now a complete array of the small coal dry cleaning jig is available. The jig is a novel equipment specifically developed for treating small coal, and its use has successfully tackled the technical bottleneck previously encountered in dry cleaning of small coal. Field application indicates that the use of the jig not only enlarges the scope of applicability of dry cleaning technology, but also pushes such a cleaning technology to a higher level.

**Keywords:** Small coal dry cleaning jig Separating bed Vibration exciter Suspension mechanism Discharge mechanism Pulsating air supply system Automatic control system Open-circuit dust collection system

## Introduction

The small coal dry cleaning jig (See Diagrams 1, 4&5) works on a principle similar to that of a wet cleaning jigging machine. It is an independently developed new-generation small cleaning equipment. The successful application of the jig overcomes the technical bottleneck in the field of pneumatic cleaning of small coal. Result of widespread field application shows the jig offers a remarkable separating performance, and is low in cost. The jig is comprised of separating bed, air distribution system, discharge unit and control system. It operates with air supply and dust collection system. After entering the separating bed, the feed undergoes 3 separating processes under the combined effect of gravitational, vibrating and frictional forces, and upward air stream. Through each process a heavier

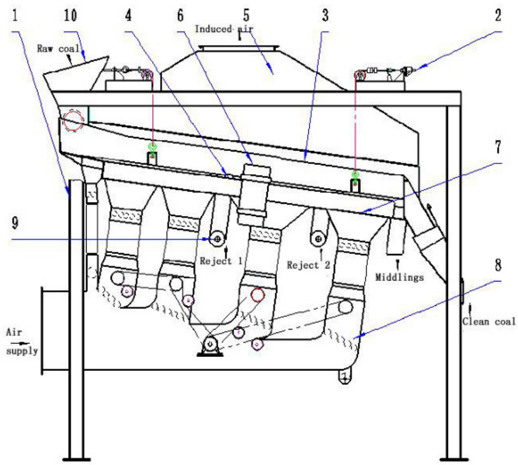


**Diagram 1 The Main Structure of the Small Coal Dry Cleaning Jig**

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product is produced and finally cleaned coal is produced. The jig is adaptable to a raw coal feed with a size of <13mm and a surface moisture of <8%. When used for treating a 13-0mm raw coal, the machine works effectively with a capacity up to 130t/h and a remarkable separating performance (approximately, Ep=0.203, organic efficiency η=89.53%). For its high capacity and performance, the application of the machine can well cater to the needs of large-, medium-and small-sized coal sectors.

## 1. Structural Design

As shown in Diagram 1, the jig is comprised of the framework 1, suspension unit 2, separating deck 3, leveling mechanism 4, dust collection hoods 5, exciter 6 (8-stage vibrating motor), adjustable air chamber 7, pulsating air supply mechanism 8, discharge unit 9 and feed unit 10. The jig body is made up by rectangular box, manually-operated uniform air distribution screen-plates, small separation boxes in between upper and lower screen-plates, intermediate small screen-plates carrying balls and the adjustable small air chambers to which the corresponding individual small boxes are connected. Connected with the jig body are vibration exciter 6, leveling mechanism 4, material distributor and discharge unit 9. The separating deck is suspended on framework 1 via flexible suspension unit 2. Fixed onto framework 1 are feed unit 10 and dust hood 5. The pulsating air supply mechanism 8 is flexibly coupled with the air chamber of separating deck. The material leveling mechanism 4 is fixed in separating deck body for ensuring that the bed material is evenly distributed across the deck. All the associated equipment are driven via frequency converters.

## 2. Working Principle

The dry cleaning jig operates on a principle which is basically similar to that of a wet cleaning jigging machine. The only difference between them is that the former uses air instead of water as separating medium. The fine size material on the deck tends to become gradually loosened and stratified under the combined effect of vibration force and the pulsation produced by the upward air-stream coming from deck bottom. The heavier material gradually sinks down to bed bottom while the lighter material gradually floats up to the top of the bed. During the proper jigging process, a stable material bed is finally formed. The heavier material at bed bottom is discharged through the discharge unit while the other materials move forward to go through further separating process under the effect of vibration force. After secondary bottom discharge process, the final products, i.e. reject 1, reject 2, middlings and clean coal are produced (See Diagram 1).

## 3. Jigging System

The jigging system is a complete cleaning circuit composed of the basic unit, and feeding, air supply, dust removal, dust collection and ash discharge and product discharge systems. For avoiding clogging of screen-plates by coal dust in air chamber beneath screen-plates, use is made of an open-circuit pulsating air supply system to ensure that fresh air is supplied and air passageway is unimpeded. The dust collection system works with cyclones and pulsating bag collectors and forms an open-circuit system with series draft fans. By using the dust collection system, the dust emission is effectively reduced to $17mg/m^3$. The jigging system operates under PLC control. The complete set of the TFX-8 dry cleaning jig is shown in Photos 4 and 5.

## 4. Technical Features

The series jigs in 4 different specifications are currently available. For their models and technical parameters, refer to Table 1.

**Table 1 Technical Parameters of TFX Series Small Coal Dry Cleaning Jigs**

| Model | TFX-1 | TFX-3 | TFX-5 | TFX-8 |
|---|---|---|---|---|
| Deck Area, $m^2$ | 1 | 3 | 5 | 8 |
| Capacity, t/h | <6 | <30 | <80 | <130 |
| Feed Size, mm | <13 | <13 | <13 | <13 |
| Effective Separation Size, mm | 1-13 | 1-13 | 1-13 | 1-13 |
| Surface Moisture Adaptable, % | <8 | <8 | <8 | <8 |
| Ecart Probable, Ep, Kg/L | 0.203 | 0.203 | 0.203 | 0.203 |
| Organic Efficiency η, % | <90 | <90 | <90 | <90 |
| Driving Power, kw | 20 | 40 | 52 | 91 |
| Overall Dimensions (LxWxH), m | 6x2x3 | 6x5x7 | 9x8x8 | 10x9x9 |

**5. Separating Performance**

In order to verify the jig's small coal cleaning performance, two kinds of sampling checks are made: one on samples taken from different kinds of coals; one on samples taken from the jig when a particular raw coal in treated. The former inspection result is listed in Table 2.

**Table 2 Result of Checks Made on Samples of Products Obtained in Treating Different Kinds of Coals**

| Coal | Size Fraction (mm) | Surface Moisture (%) | Raw Coal Ash (%) | Clean Coal | | Middlings | | Reject | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | Yield (%) | Ash (%) | Yield (%) | Ash (%) | Yield (%) | Ash (%) |
| Coking Coal | 13-0 | 7.29 | 19.75 | 77.49 | 9.10 | 7.50 | 28.43 | 15.01 | 70.41 |
| Power Coal | 13-0 | 7.81 | 42.53 | 58.73 | 27.26 | 21.93 | 54.88 | 19.39 | 73.47 |
| Power Coal | 13-0 | 7.17 | 35.85 | 60.97 | 20.71 | 20.90 | 46.27 | 18.13 | 74.73 |
| Average | | 7.42 | 32.71 | | 19.02 | | 43.19 | | 72.87 |

As can be seen from Table 2, the jig's performance obtained in treating either coking coal or power coal can well meet the requirements of users. Under normal working conditions, the average figures of moistures of feeds and ashes of the 3 sampled products are respectively as follows: surface moisture – 7.42%; ash of rejects – 72.87%; ash of raw coal – 32.71%; ash of clean coal – 19.02%; and ash reduction level – 13.69%. It can be seen that the jig is highly adaptable to variation of surface moisture of feed, high in ash reduction capacity, and a purer reject product can be expected.

The check in the latter case is made with a 130t/h capacity jig treating a raw coal feed with a moisture of 7.29%. The result of the size analysis of the sample is listed in Table 3. For the density consist of the 13-1mm size fraction, refer to Table 4. The washability curve of the 13-1mm size fraction is shown in Fig. 2. The density consists of the different size fractions are listed in Fig. 5, and the corresponding partition curve is shown in Fig. 3. The calculated values of Ecart Probable Ep and organic efficiency are tabulated in Table 6. It can be seen that the value of Ep=0.203 and the organic efficiency η is as high as 89.53%.

**Table 3 Size Composition of the Sample**

| Size Fraction, mm | Yield, % | Ash, % |
|---|---|---|
| 13-1 | 86.80 | 18.75 |
| <1 | 13.20 | 30.61 |
| Total | 100.00 | 20.32 |

**Table 4 Density Consist of the 13-1mm Coal Sample**

| Density Fraction kg/L | Density Composition | | Cumulative | |
|---|---|---|---|---|
| | Yield in that Fraction, % | Ash, % | Yield in that Fraction, % | Ash, % |
| <1.3 | 21.95 | 3.29 | 21.95 | 3.29 |
| 1.3-1.4 | 55.68 | 4.33 | 77.63 | 4.04 |
| 1.4～1.5 | 2.73 | 16.45 | 80.36 | 4.46 |
| 1.5～1.6 | 1.13 | 27.69 | 81.49 | 4.78 |
| 1.6～1.7 | 0.84 | 35.97 | 82.33 | 5.10 |
| 1.7～1.8 | 0.73 | 45.82 | 83.06 | 5.46 |
| <1.8 | 16.94 | 83.93 | 100.00 | 18.75 |
| Total | 100.00 | 18.75 | | |

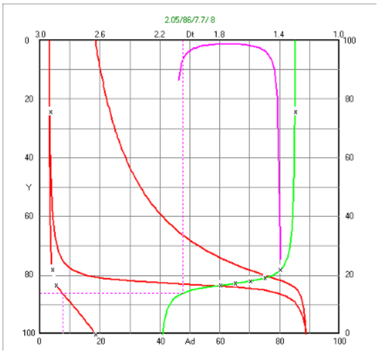

Fig.2 Washability Curve of the 13-1mm Size Fraction

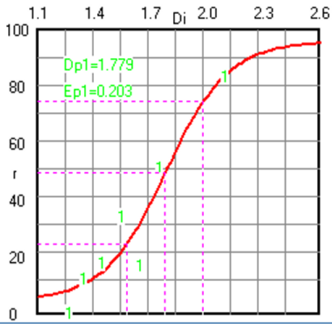


Fig. 3 Partition Curve of the 13-1mm Coal (Clean Coal + Middlings + Reject)

**Table 5 The Density Composition of Each Product Obtained by Treating 13-1mm Size Coal Using The Small Coal Dry Cleaning Jig**

| Density Fraction kg/L | Clean Coal | | | Middlings | | | Reject | | |
|---|---|---|---|---|---|---|---|---|---|
| | Yield in Total Fractions (%) | Yield in That Fraction (%) | Ash (%) | Yield in Total Fractions (%) | Yield in That Fraction (%) | Ash (%) | Yield in Total Fractions (%) | Yield in That Fraction (%) | Ash (%) |
| <1.3 | 18.89 | 28.30 | 3.28 | 0.08 | 1.21 | 4.54 | 0.08 | 0.59 | 4.47 |
| 1.3-1.4 | 42.01 | 62.95 | 4.28 | 4.27 | 59.64 | 3.98 | 2.05 | 15.91 | 6.13 |
| 1.4～1.5 | 1.92 | 2.88 | 16.36 | 0.27 | 3.74 | 14.68 | 0.17 | 1.29 | 20.52 |
| 1.5～1.6 | 0.65 | 0.97 | 26.98 | 0.18 | 2.52 | 24.13 | 0.16 | 1.24 | 34.52 |
| 1.6～1.7 | 0.60 | 0.90 | 35.85 | 0.12 | 1.72 | 36.33 | 0.01 | 0.10 | 37.75 |
| 1.7～1.8 | 0.31 | 0.46 | 45.58 | 0.18 | 2.52 | 45.54 | 0.14 | 1.09 | 46.70 |
| <1.8 | 2.36 | 3.54 | 78.95 | 2.05 | 28.64 | 83.17 | 10.30 | 79.78 | 85.22 |
| Total | 66.74 | 100.00 | 7.67 | 7.15 | 100.00 | 29.18 | 12.91 | 100.00 | 70.22 |

**Table 6 Calculated Values of Ecart Probable Ep and Organic Efficiency**

| Clean Coal Ash, % | Ep, kg/L | Actual Yield, % | Theoretical Yield, % | Organic Efficiency, % |
|---|---|---|---|---|
| 7.67 | 0.203 | 76.89 | 85.88 | 89.53 |

## 6. Application in Prospect

Apart from its application simply for ash reduction and coal upgrading for higher calorific value, the small coal dry cleaning jig can also find applications in the following aspects: 1) To form an integrated coarse coal and small coal combined dry cleaning system for treating 75-0mm size coal (Cleaning >13mm coarse coal with ordinary dry cleaning machine and ≤13mm small coal using the small coal dry cleaning jig); 2) To set up a wet and dry cleaning combined system for simplification of coal slurry treatment circuit (Wet cleaning of >13mm coarse coal and dry cleaning of ≤13mm small coal with the small coal dry cleaning jig); 3) To form up a difficult-to-clean coal dry cleaning system to achieve an improved performance in treating such kind of coal (Crush the coal down to a size of <13mm for better liberation of its inter-grown constituents and then have it cleaned with small coal dry cleaning jig).

Photo 4 The Complete Set of the 130t/h TFX-8 Small Coal Dry Cleaning Jig

Photo 5 The Main Structure of the TFX-8 Small Coal Dry Cleaning Jig

## 7. Conclusions

The jig is a novel equipment specifically developed for treating small coal. Its debut has successfully tacked the technical bottleneck previously encountered in dry cleaning of small coal. The use of the jig not only enlarges the scope of applicability of dry cleaning technology, but also pushes such a cleaning technology to a higher level.

The jig operates with all-round innovative air supply and distribution technology, fixing mode and dust collection system and in a fully automatic manner. The provides a technical guarantee for the efficient dry cleaning of small coal.

Field operation shows that the jig is a novel equipment capable of effectively cleaning small coal.

The jig is suitable for treating raw coal with a size of -13mm and a surface moisture of less than 8%. When used for treating 13-1mm coal, the jig can operate with an Ep around 0.203 and an organic efficiency η around 89.53%.

The estimated investment for a complete set of such a jigging system is around Rmb 1.75/t (raw coal). The power consumption and operating cost of the jig are approximately 0.715 Kwh/t (raw coal) and Rmb 1.92/t (raw coal) respectively.

The jig is provided with an open-circuit fresh air supply system working with cyclone and bag collectors and series-connected draft fans. With the use of the high-efficiency dust collection system, the concentration of gas emission is reduced to about 17mg/m$^3$.

Now, an array of such jigs are available, which fall into 4 specifications, with a maximum capacity of 130t/h. The use of the jig can well cater to the needs of large-, medium-and small-sized coal sectors.

Besides the application of the jig as a independent cleaning system, a coarse and small coal combined dry cleaning system can also be set up with the use of the jig, so as to meet the specific market requirements. The jig has a bright future of development.

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# The Progress of the Pulsing Airflow Fluidized Bed for Separating the Fine Coal

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**Abstract:** Because of the bubbles and intense backmixing of the medium in the conventional air dense medium fluidized bed (ADMFB), the alternative size range of the feeding is +6mm. The innovation of employing the pulsing airflow to the fluidized bed (PFB) can significantly strengthen the gas-solid contact and reduce the formation of the bubbles. To study the bubble behavior and energy transfer in the fluidized bed, the simulation of the Eulerian- Eulerian (EE) model was conducted on the gas-solid system. The results show that the lower bed height can decrease the energy collision and mergers. The comparison between ADMFB and PFB indicates that pulsing energy flow can effectively decrease the vortexes and make the energy flow transfer along with the vertical direction of the bed. The software of "Design Expert" was adopted to conduct an orthogonal test with the factors of gas velocity and pulsing frequency. As for the best separation results, the E value of 0.093g/cm$^3$ and the clean coal ash content of 33.69% were achieved with the yield 52.04% and maximum ash content drop 22.46%.

**Key words:** dry beneficiation; pulsing airflow; fine coal; simulation; bubbles; energy transfer; vortexes;

## 1. Introduction

Coal is one of the most important primary energy in China and the consumption accounted for 66.03% in the primary energy consumption in 2014[1]. As most coal resource distributes in the arid area worldwide, wet beneficiation is restricted for being short of water resource but dry coal preparation without using water owns a huge potential market and good market prospect. China University of Mining and Technology has begun developing the air dense medium fluidized bed (ADMFB) since 1980s. Based on 30 years of research on fundamental theory [2,3], laboratory test[4-6], pilot test and industrial test [7,8], the first industrial dry coal preparation system was established to separate the coal with size fraction of -50+6mm efficiently[9]. However, traditional ADMFB technology is difficult to separate the fine coal, especially for the size fraction of -6mm. Based on the ADMFB, the pulsing airflow is introduced to substitute the constant airflow, namely PBF which was a hotpot recently[10,11]. The pulsing airflow can significantly decrease the diameter of the bubbles and short circuit in the bed[12,13], making the bed fluidized more uniformly .

Computational fluid dynamics (CFD) software was adopted to analyze the bubble behavior and energy transfer within the bed. At present, the common simulation method used in the gas-solid system can be divided into two categories including Eulerian-Eulerian (EE) model and Eulerian-Lagrangian (EL) model[14]. The EE model also be called two fluid model (TFM) which considers both gas phase and solid phase as interpenetrating continuums using Navier-Stokes equations in each phase[15]. The EL model generally treats solid phase as discrete entities such as discrete element method (DPM) and discrete particle model (DPM) while the gas phase is still taken as a continuum employing CFD software[16,17]. PFB belongs to dense suspension fluidized bed and EL model not easily track so large amount of particles, therefore in this article the EE model was conducted to simulate the PFB because of its low computational costs as well as the reliability[18,19].

 
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**2. Experimental and material**

**2.1 Experimental system**

The experimental sketch is shown in Fig. 1. The fluidized bed consists of a gas distributor and a transparent Plexiglas container with a cross section of 300 mm×200 mm and height of 600 mm. A frequency converter was used to adjust the frequency of the pulsing airflow. The pulsing airflow was generated by the rotation of a motor-driven ball valve and the pulsing frequency was controlled by an inverter. The relationship between the gas velocity and time can be illustrated using Eq. (2-1):

$$v = v_0 + v_c\left[1 - \left|\cos(2\pi f t)\right|\right] \tag{2-1}$$

where $v$ is the instantaneous velocity of the pulsing airflow, $v_0$ is the continuous velocity of airflow, $f$ is the pulsing frequency and t is the time, so $v_{max} = v_0 + v_c$.

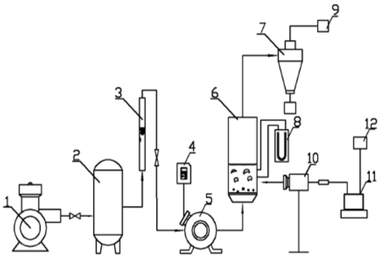


**Fig. 1.** Schematic of experimental system: 1-blower, 2-tank, 3-flow counter, 4-inverter, 5-ball valve, 6-fluidized bed, 7-cyclone dust collector, 8-U-tube manometer, 9-bag filter, 10-high-speed camera.

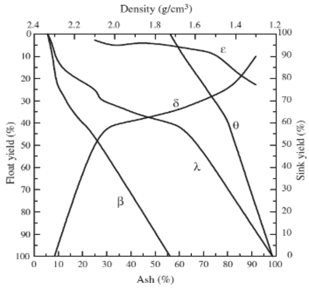


**Fig. 2.** Washability curves of raw coal.

**2.2. Material**

The average size of the magnetite powder used in the experiments was 232 $\mu m$ and the true density was 4600 kg/m$^3$. The heavy medium with the size fraction of -300 + 60$\mu m$ was confirmed to be the best for separation [20]. Fine coal with size fraction of -6 + 1 mm was used in the separation experiments. The washability curve of the raw was drawn as shown in Fig. 2

**3. Effect of the pulsing mechanism**

Because the weight of fine coal is very light, it is difficult to separate the fine coal in conventional ADMFB for various bubbles in the bed can easily interfere with the motion of the fine coal particle. The airflow imparted into the ADMFB can be divided into two parts: interaction of the gas and solid which promotes the fluidization and superfluous airflow which can aggregate together and form bubbles. Because the pressure of the bed decreases along the vertical direction, these bubbles may grow larger when they move upward shown in Fig. 3(a). By contrast, the energy introduced into the PFB is periodic and most airflow would act on the solid particle to fluidize the medium instead turn into the bubbles as shown in Fig. 3(b). Besides, the pulsing airflow make the density of the fluidized bed maintain in a range of high density and low density, which is beneficial to the settlement of particles with different densities.

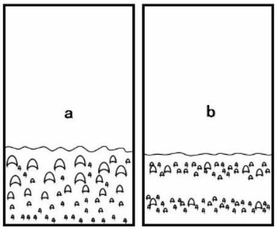


**Fig. 3.** The comparison of bubble behavior of ADMBD (a) and PFB (b).

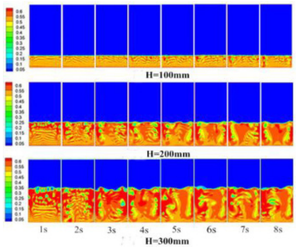


**Fig. 4.** Diagram of bubble behavior under different bed height.

**4. Simulations**

**4.1 Simulations under different bed height**

In order to investigate the best bed height to separate the coal particles, a series of experiments were simulated with different height (100mm, 200mm and 300mm) and the gas velocity plus frequency were set as $1.15U_{mf}$ ($U_{mf}$ is the minimum fluidization velocity) and 1.8Hz respectively. Fig. 4 shows that the number and diameter of the bubbles in the bed increase as the height rising. In ideal conditions where the energy flow (airflow) could transfer from the bottom of the bed to the surface along the vertical direction and ultimately those exceeded energy would disappear beyond the surface, but the energy flow easily alters the transfer direction in this process because the resistance of the medium and the intensive energy mergers would occur, which can produce bubbles and influence the stability of the bed. Before the energy mergers the optimal time $t_z$ is assumed, in other word, it is until the energy flow have already arrived at the surface of the bed that the energy transfer occurs. The equation for the $t_z$ can be established:

$$t_z = \frac{H_z}{v_o} = \varepsilon\beta \frac{d_j \delta_j}{\theta_k d_k \varphi_k v_0} \tag{4-1}$$

The height of the bed $H_z$ is related to medium density $\delta_j$, the equivalent diameter of medium particle $d_j$, the aperture ratio of the air-distributor $\theta_k$, the diameter of opening aperture $d_k$, well-distributed coefficient of the opening aperture $\varphi_k$, relevant coefficient of fluidized bed size $\varepsilon$ and drag coefficient of gas and solid $\beta$. If the bed is fluidized uniformly and there is less or no energy merger, then $t_0<t_z$ and some equations can be derived as follows:

$$\frac{1+\theta_c}{v_0} H \le \frac{H_z}{v_0} = \varepsilon\beta \frac{d_j \delta_j}{\theta_k d_k \varphi_k} \tag{4-2}$$

$$H = \frac{H_z}{1+\theta_c} = \varepsilon\beta \frac{d_j \delta_j}{(1+\theta_c)\theta_k d_k \varphi_k}$$

(4-3)

From the equations above, we can find that less higher bed contribute to decreasing the energy merger, but a necessary bed height for different coal particles should be settled. The bed height of 100mm seems to be better in the simulation but the best height is closely to the structure of the bed itself.

**4.2 The comparison of the ADMFB and PFB**

The simulations on the ADMFB and PFB were performed respectively with relevant parameters setting as follows: gas velocity $1.15U_{mf}$, frequency 1.80Hz and bed height 100mm. It is not easy to observe the internal conditions of the bed in actual experiments(including the motion of the medium and the airflow), but the simulation may present these cases. As referred above, the energy flow (airflow) may change direction and collision or merger, forming into vortex, which may worsen the separating conditions. This phenomenon can be found in the diagrams of flow streamline contour as shown in Fig. 5.

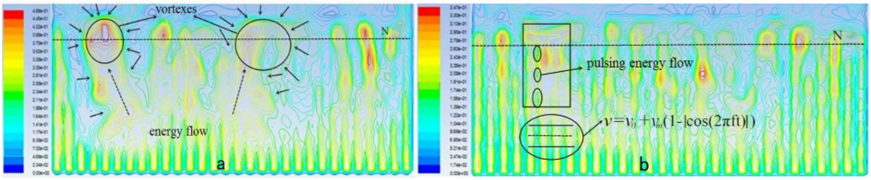


**Fig. 5.** Diagram of flow streamline contour of ADMFB (a) and PFB (b) at 5.0s.

The Fig. 5(a) shows that the energy flow only has the clear transfer direction in the bottom and margin of the ADMFB and there are many vortexes in the middle . Although some vortexes can be seen in PFB, the energy flow can generally transfer uniformly through the bed as shown in the Fig. 5(b). In the middle of two transverse lines, there will be a pulsation period when pulsing energy flow can hold back the formation of vortex. Actually, when the energy flow arrives at N layer as shown in the Fig. 5, the vortexes in ADMFB are much larger than PFB. This is because the former energy flow in ADMFB has already make the medium fluidized, but subsequent equal energy makes the incompact medium to be more active and produces vortexes. By contrast, the pulsing energy flow in PFB allows time to make medium return to inactive condition and avoid the vortexes.

## 5. Results and discussion

Based on the analysis of the simulations, an orthogonal test was conducted to separate the fine coal with the size -6+1mm. The bed height was set as 100mm while the pulsing frequency and gas velocity were independent variable with probable error E as evaluation. After the separation, the bed was divided into two layers: the upper layer of above 40mm is clean coal and the lower layer of 60mm is gangue. The basic parameters of the gas velocity and pulsing frequency and separation results are shown in Table. 1

**Table.1.** Basic parameters and the result of E value

| No. | A: gas velocity (cm/s) | B: pulsing frequency (Hz) | E ($g/cm^3$) |
|---|---|---|---|
| 1 | 6.5 | 1.36 | 0.178 |
| 2 | 6.5 | 1.82 | 0.125 |
| 3 | 6.5 | 2.27 | 0.275 |
| 4 | 6.9 | 1.36 | 0.150 |
| 5 | 6.9 | 1.82 | 0.093 |
| 6 | 6.9 | 2.27 | 0.120 |
| 7 | 7.9 | 1.36 | 0.305 |
| 8 | 7.9 | 1.82 | 0.335 |
| 9 | 7.9 | 2.27 | 0.365 |
| range/R | 0.642 | 0.207 | |

It can be clearly seen that when the gas velocity and pulsing frequency are 6.9cm/s and 1.82Hz respectively, the optional separation efficiency was obtained with the E value as 0.093g/cm$^3$ as shown in Fig. 6 (a) . Under this optional condition, the clean coal ash content is 33.69% with the yield 52.04% and maximum ash content drop 22.46%. The comparison experiment was also carried out in AMDFB and the best separation results were shown in Fig. 6 (b) with E value as 0.19g/cm$^3$. The range R of the each factor reflects the fluctuation of E value when the value of the factors change. So the gas velocity influences the separation results more significant than the pulsing frequency as 0.642 > 0.207.

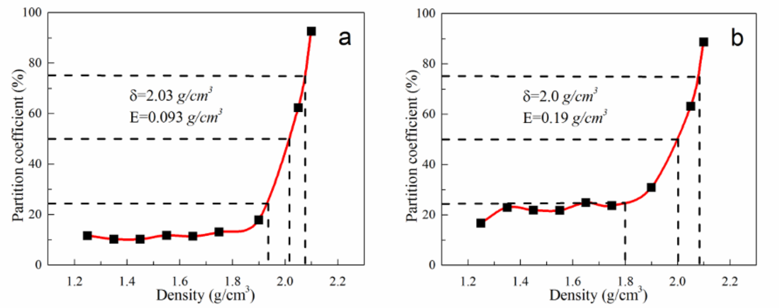


**Fig.6.** Partition coefficient of fine coal of -6 +1 mm.

**6. Conclusion**

1. APFB was proposed for fine coal beneficiation, which introduced an active pulsing airflow to the ADMFB. An equation about the height of energy transfer was established to illustrate that proper bed height can decrease the energy collision and mergers contributing to better beneficiation.

2. The results of the simulation indicate that the number and size of bubbles in PFB decreased significantly. Pulsing energy flow can control the formation of vortexes and create more steady conditions for separating fine coal.

3. The orthogonal test shows that the gas velocity influences the separation efficiency more significant than pulsing frequency. Under the optimized combined conditions, the E value of 0.093g/cm$^3$ and the clean coal ash content of 33.69% were achieved in PFB comparing the E value of 0.19g/cm$^3$ in ADMFB .

**Acknowledgments**

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# Fine Coal Beneficiation with the Application of a Pulsing Air Dense Medium Fluidized Bed

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**Abstract:** Coal is one of the most important energy in the world. However, without separation, the coal would cause serious environmental pollution while in utilization and combustion. Because of the increasing shortage of water resource, research on high effective dry coal beneficiation technology becomes increasingly important. In this study, pulsing air flow with periodical velocity is introduced to air dense medium fluidized bed (ADMFB) to separate fine coal and the separation effect of the velocity and pulsation frequency of air flow are discussed. The results show that within the appropriate fluidization number range[1.1,1.3] of PADMFB, with the increase of pulsation frequency, under the condition of different air flow velocity and feed particle size, the standard deviation of ash segregation first increases and then decreases. N = 1.2, f =4Hz is proved to be the optimal operating condition. -6+3mm fine coal is separated effectively in pulsing air dense medium fluidized bed (PADMFB). The clean coal ash content and yield are respectively 15.03% and 64.07%.

**Keywords:** pulsing air dense medium fluidized bed, fine coal, dry coal beneficiation, vibration energy, air velocity, pulsation frequency, beneficiation effect

Coal is one of the three key energy resources. In 2014, coal consumption accounted for 30.0% of the primary energy consumption in the world [1]. China is the largest producer and consumer of coal, accounting for more than a half of world coal consumption [2]. Hazardous materials, such as smoke and dust, NOX and SO2, are released in low grade coal combustion [3]. Sulfur is the primary harmful element and SO2 emanated from coal combustion is the main source of acid rain. Coal preparation is the most economic and effective coal utilization technology. Developing coal preparation technology is of great significance to coal mining, energy conservation and environment protection. Wet dressing plays a dominant role in coal preparation at present. However, it needs a large amount of water, 3-5 m3 water is required for the preparing a ton of coal, while in China, 2/3 coal storage is buried in western area where is short of water. Therefore, with increasing shortage of water resource, research on high effective dry coal beneficiation technology is urgent.

Fine coal and air are taken as medium in compound dry coal preparation technology [4] to separate 80-0 mm materials. China University of Mining and Technology [5-8] has developed a novel ADMFB in 1980s and established a modularized dry coal beneficiation system in 2007 [9]. The modularized system makes it possible to realize effective dry coal beneficiation for 50-6 mm coal with E value of 0.05 g/cm$^3$ and large separation density, meeting the demands of different coal property and production quality. With the widespread use of mechanized mining technology, -6 mm fine coal occupies a relatively high proportion in raw coal, which reaches to more than 60%. Pyritic sulfur is mainly distributed in fine coal while it is not easy to realize the separation of fine coal with traditional ADMFB. In this case, introducing external energy becomes an effective way to improve the separation effect of fine coal.

In the research, pulsing air flow with periodical velocity was introduced to ADMFB, which effectively weakens the channeling and short-circuit of the bed material. The mass and heat transfer efficiency is enhanced and the dense medium fluidizes well, as a result, an appropriate fluidized bed is formed to separate fine coal. The effect that the air flow velocity and frequency act on separation result is explored



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as well.

**1. PADMFB separation system**

PADMFB separation system [10] is shown in fig. 1. The testing system is composed of air-supply system, pulsating air flow device, flow control device, fluidized bed model, dedusting system et.al. Active pulsating air flow is generated from rolling valve, whose velocity fluctuates periodically over time. Vibration energy is introduced into ADMFB by vibrating air flow, keeping the dense medium uniform and stable. Coal particles are separated according to the density. In the experiment, dense medium is magnate powder with true density of 4600 kg/cm$^3$ and bulk density of 2300 kg/cm$^3$, whose dominant particle size is 100~500μm. The size distribution [10] is analyzed with laser particle size analyzer, as shown in Fig. 2. According to particle classification method proposed by Geldart, the magnate powders are Geldart group B particles.

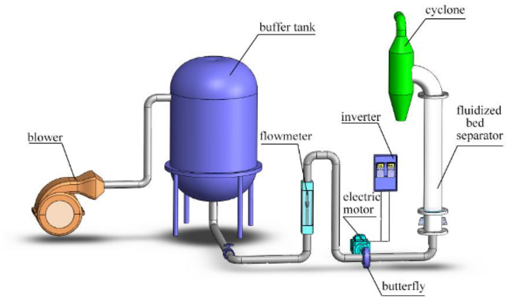


Fig. 1. Schematic of gas-vibro fluidized bed separation system

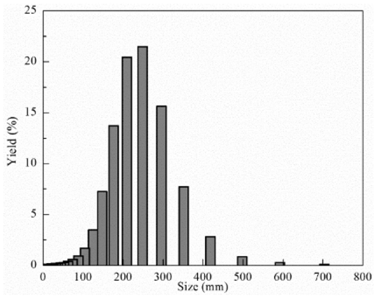


Fig. 2. Size distribution of magnetic powder.

**2. Separation mechanism**

In PADMFB, coal materials are separated on the basis of bed density. Ash content is the key index to reveal product quality. Clean coal with low density and ash content go up to the top while the gangue with high density and ash content sink to the bottom. The fine coals would fluidize for a moment, when the air flow stops suddenly, the fluidized bed becomes fixed bed. The bed was divided into *n* layers from top to bottom and particles in each layer were sieved and analyzed. The higher proportion the ash content occupies in every layer deviate from the raw coal ash, the better the separation effect could be. According to the Separation mechanism shown above, the standard deviation of ash segregation is proposed to evaluate the influence that the superficial gas velocity and pulsation frequency act on separation, as shown in equation (1):

$$\sigma_{ash} = \sqrt{\frac{1}{n-1}\sum_{i=1}^{n}\left[A(i)-\overline{A}\right]^2} \tag{1}$$

Where,

$$\overline{A} = \sum_{i=1}^{n} A(i)\gamma(i)$$

Where n is layer number, A( i ) is the ash content of the ith layer, γ( i ) is the yield of the ith layer, $\overline{A}$ is the weighted average of the ash content, which is theoretically equal to that of raw coal. The standard deviation of ash segregation represents the extent that the ash content deviate from the raw coal in the vertical direction. When the particles mix uniformly, ash content in different height won't deviate from the raw coal and $\sigma_{ash} = 0$ . A large $\sigma_{ash}$ value represents a high deviation extent and a great separation effect.

## 3. Results and analysis

Fine coal particles with -6+3 mm size fraction were used to separate in PADMFB. The properties of raw coal are shown in table 1. The ash content of raw coal is 33.69% and the total sulfur content is 3.69%, which can be concluded that the raw coal is high-ash and high-sulfur coal. The washability curves are obtained with float-and- sink test and chemical analysis, as shown in Fig. 3. δ curve indicates that the proportion of +1.8 g/cm$^3$ is more than 30% , that is, they are pure gangue particles. The intermediate density fraction (-1.8+1.5 g/cm$^3$) particles are quite few, proving that the gangue could be removed easily.

Table 1 Properties of raw coal

| **Size mm** | **$M_t$ %** | **$A_{ad}$ %** | **$V_{daf}$ %** | **$S_{t,d}$ %** | **$\rho_s$ g/cm$^3$** |
|---|---|---|---|---|---|
| **-6+3** | 0.82 | 33.69 | 15.46 | 3.69 | 1.70 |

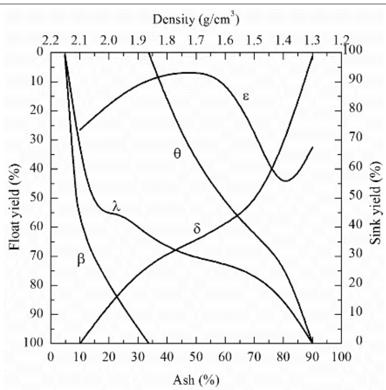


Fig. 3. Washability curves of raw coal.

### 3.1 Effect of air flow velocity on separation

Air flow velocity is one of the key factors that influence the separation performance of ADMFB. The air velocity exerts a direct influence on bed expansion ratio and viscosity. Fluidization number N is introduced to represent the influence of the air flow velocity on separation effect. Equation (2) is shown as below.

$$N = \frac{v_p}{v_{mf}} \qquad (2)$$

Where $v_p$ is air flow velocity in pulsing fluidized bed, $v_{mf}$ is the minimum fluidization velocity.

The influence of the air flow velocity on separation effect of −6+3 mm fine coal is studied. Coal particles with distinct fluidization number in each layer are analyzed. The standard deviation of ash segregation is calculated with equation (2) and the changes are shown in Fig. 5. As the velocity increases, the standard deviation of ash segregation first rises and then falls dramatically. When N = 1.2, standard deviation of ash segregation reaches its peak, after then it decreases. When N>1.2, separation effect deteriorates linearly. To obtain a better separation effect, the range of fluidization number should be controlled within [1.1, 1.3] in this study.

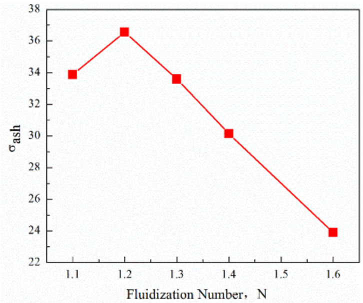


Fig. 5. Effect of gas velocity on σ ash

**3.2 Effect of pulsation frequency on separation**

The forced vibration energy of pulsing air flow has a direct relation with fluidization effect and the degree of vibration energy depends on pulsation frequency and fluidization velocity. Within the appropriate fluidization number range [1.1, 1.3], the effect of different pulsation frequencies on -6+3mm fine coal are shown in following figures.

As shown in Fig. 6, when the frequency is low, the increasing trend of ash production is in tune with that of the velocity. The higher the velocity is, the larger the change of ash production could be. When the frequency is no less than 3.49Hz, an increasing velocity would lead to back-mixing which might deteriorate the separation effect. To make a comprehensive estimation on the influence of air flow velocity and pulsation frequency on separation result, the standard deviation of ash segregation of -6+3mm fine coal is calculated according to the Fig. 6 and equation (2), as shown in Fig. 7. With the increase of pulsation frequency, under the condition of different air flow velocity and feed particle size, the standard deviation of ash segregation first increases and then decreases, reaching the peak when f = 4Hz in the mediate frequency area. Within the mediate frequency area, when N = 1.2, a better separation effect can be obtained compared to other conditions. Through the comparison, N = 1.2, f =4Hz is chosen as the optimal operating condition.

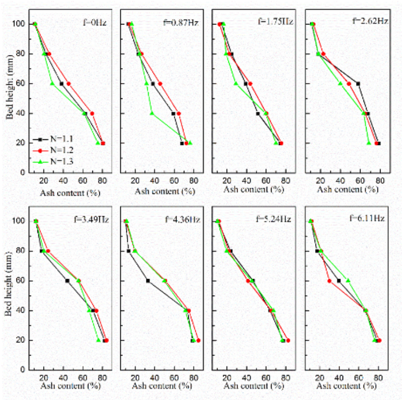

Fig. 6. Separation results at different gas velocities and pulsation frequencies

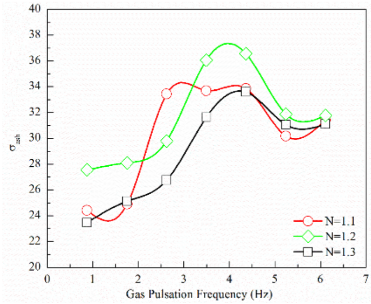

Fig. 7. Effect of gas pulsation frequency on σ ash (−6+3 mm)

### 3.3 Partition coefficient

Under the optimal operating condition (N = 1.2, f = 4Hz), fine coal particles with −6+3 mm size fraction are separated to produce clean coal, middling and gangue. The ash content and yield of products are shown in table 3. For −6+3 mm fine coal, the clean coal ash content and yield are respectively 15.03% and 64.07%. The density fractions of the products are calculated with the float-and-sink test. By calculating the partition coefficients of every density fraction, the partition curves of the two density fractions are drawn as shown in Fig. 8. For high density separation, the actual separating density is 1.89 g/cm$^3$ and the E value is 0.22 g/cm$^3$. For low density separation, the actual separating density is 1.68 g/cm$^3$ and the E value is 0.19 g/cm$^3$.

Table 3 Yield and ash content of product

| **Size** | **Clean coal** | | **Middlings** | | **Gangue** | |
|---|---|---|---|---|---|---|
| **/mm** | Yield % | Ash % | Yield % | Ash % | Yield % | Ash % |
| **-6+3** | 67.40 | 15.03 | 14.48 | 50.92 | 20.82 | 79.70 |

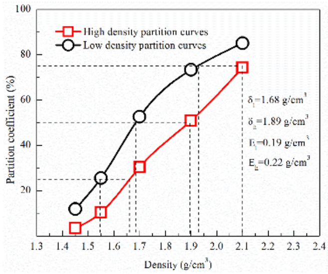


Fig. 8. Partition curves of separation for -6+3 mm coal

## 4. Conclusions

Vibration energy is introduced into ADMFB by way of air flow vibration. The influences of air velocity and pulsation frequency on -6+1mm fine coal separation in PADMFB are explored. The conclusions are as follows:

(1) As air flow velocity increased, the standard deviation of ash segregation first increases and then decreases. When N = 2, standard deviation of ash segregation reaches the peak.

(2) Within the appropriate fluidization number range[1.1,1.3] of PADMFB, With the increase of pulsation frequency, under the condition of different air flow velocity and feed particle size, the standard deviation of ash segregation first increases and then decreases. N = 1.2, f =4Hz is proved to be the optimal operating condition.

(3) Under the optimal operating condition, for −6+3 mm fine coal, the clean coal ash content and yield are respectively 15.03% and 64.07%.

**Acknowledgments**

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# The Application and Research on Before-Furnace Coal Desulfurization by ZM Dry Separator in Guizhzou Tongzi Power Plant

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**Abstract**

This paper introduces working mechanisms and system configuration of ZM coal dry separator. An industrial desulfurization test adopting dry separation process is conducted for the fuel coal in Guizhou Huadian TongZi power plant through conducting a coal quality and washability analysis according to its characteristics of high sulfur content in the fuel coal in the power plant and by incorporating mineral separation characteristics and the system separation process of the latest developed ZM high efficient mineral separator manufactured by Tangshan Shenzhou Manufacturing Co., Ltd. Then, the overall operating benefit of power plant is systematically analysed through this industrial test of desulfurization before combustion. It comes to a conclusion that the dry separation process is the best choice to achieve the optimum benefit for power plant coal desulfurization before combustion, it removes most of inorganic sulfur and cut the load of desulfurization system of the power plant. Moreover, it also provides certain technical support for the before-combustion desulphurization technology for China's power plant.

**Keywords:** power plant, coal fuel, coal dry separation, mineral separation, desulfurization, separation process, experimental study

**Introduction:**

Guizhou Huadian Tongzi Power Generating Co., Ltd. is located in TongZi county in the north of Guizhou province. The plant has 2 x 600 MW unit generators [1]. At present, the high sulfur and ash content in the fuel coal is an important factor restricting the operation of the power plant, because burning high sulfur content coal in the power plant has resulted in the emission of flue gas that contains a large amount of sulfide and dust, which has not only enhanced after-furnace desulfurization difficulty, at the same time, it has also caused serious jam in the air pre-heater due to the negative effect of coal dust on SCR catalysts. When proper coal preparation process is adopted, the materials such as inorganic sulfur and high ash coal in the raw coal can be removed in advance. As a result, it prevents most of the waste rock from entering into the coal pulverizer, which has greatly reduced the energy consumption. Moreover, it will also greatly reduce the sulfur and ash content in flue gas [2-4]. Therefore, the proper separation before combustion for power plant fuel coal plays a very important role in enhancing the utilization efficiency of coal before combustion, reducing pollution and comprehensive utilization of rejects and so on [5-6]. Although laboratory test of biological desulfurization method can remove some organic sulfur [7], it fails to get the actual industrial application. Dense medium, jig and flotation combined process are now most used process in desulfurization, the capital cost and operation cost are high [8-10]. This article mainly discusses and explores the industrial experiment of desulfurization that conducted by ZM high efficient mineral separator (developed by the joint effort of Shenzhou Manufacturing Co., Ltd and FGX SepTech.LLC) for the fuel coal in Guizhou Huadian Tongzi Power Generation Co., Ltd.

## 1. The Proposal of Desulfurization by ZM High Efficient Separator

In consideration of the characteristics of raw coal quality in Guizhou Tongzi Power Plant, system

 
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analysis of current coal desulfurization technology and the present development of new dry-process desulphurization equipment, it is proposed to adopt such a process route to utilize ZM high efficient mineral separator in dry desulphurization of fuel coal in this power plant. Moreover, an industrial test is also conducted for the coal in Sifeng mine.

**1.1. Separation Principle of ZM High Efficient Mineral Separator**

Developed by the joint effort of Shenzhou Manufacturing Co., Ltd and FGX SepTech.LLC, ZM High Efficient Mineral Separator is suitable for the dry separation of materials with high temperature and high density contents. This equipment adopts a multistep separation principle and the main separation bed is divided into three stepwise parts. Under the separation deck, it is fitted with several wind chambers, through which proper weak wind is blown upwards to the separation deck surface, achieving an ideal stratification condition; appropriate wind force can reduce the segregation phenomenon generated by vibration effect, which makes the coal bed loose and materials with fine size fraction and low density is blown to the upper layer, thus, it optimizes the stratification of mineral particles in the coal bed, consequently, the separation effect is enhanced. Therefore, by introducing a weaker updraft air flow, taking good advantage of autogenic medium formed by the fine particles of feed material, and finally utilizing the combined function of air blow function and autogenic medium function, the separation medium is fluidized. Thus, the mineral material in the fluidizing medium on each part of the separation deck is stratified according to densities. The structure of main separator is shown in Figure 2:

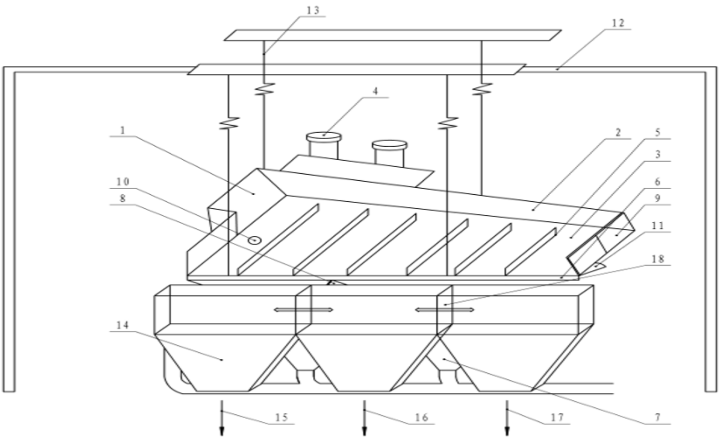



**Figure 2.** Main Separator Structure of ZM High Efficient Mineral Separator

As shown: 1-coal feeding port; 2-back plate; 3-separation deck; 4-vibration motor; 5-riffle; 6-discharge baffle; 7-air chamber; 8-thrust plate; 9-refuse door; 10-air holes; 11-transverse slope α; 12-separator frame; 13-hanging mechanism; 14-discharge chute; 15-clean coal discharge port; 16-middlings discharge port; 17-refuse discharging port; 18-adjustable damper

**2.2. Industrial Desulfurization Test conducted by ZM High Efficient Mineral Separator**

**(1) Conditions of Industrial Test**

The industrial desulfurization test was carried out in No.3 coal yard of Guizhou Huadian Tongzi Co., LTD. The coal sample for the test is collected from the power plant fuel coal supplied by Sifeng main mine. The coal quality data are shown in table 1 and the washability curve is shown in Figure 3:

**Table 1.** 80~6mm Size Fraction Raw Coal F&S Test Analysis

| Density Class | Yield | Ash content | Sulfur content | The Floats Yield | The Floats Ash content | The Sinks yield | The Sinks Ash content | Separation density±0.1 | |
|---|---|---|---|---|---|---|---|---|---|
| $g \cdot cm^{-3}$ | % | % | % | % | % | % | % | $g \cdot cm^{-3}$ | |
| <1.4 | 3.08 | 4.04 | 2.18 | 3.08 | 4.04 | 100.00 | 21.38 | 1.4 | 41.23 |
| 1.4～1.5 | 38.15 | 10.06 | 3.05 | 41.23 | 9.61 | 96.92 | 21.93 | 1.5 | 60.30 |
| 1.5～1.6 | 22.15 | 16.73 | 4.67 | 63.38 | 12.10 | 58.77 | 29.64 | 1.6 | 29.53 |
| 1.6～1.7 | 7.38 | 23.74 | 8.03 | 70.77 | 13.31 | 36.62 | 37.45 | 1.7 | 13.53 |
| 1.7～1.8 | 6.15 | 31.17 | 8.88 | 76.92 | 14.74 | 29.23 | 40.92 | 1.8 | 15.38 |
| 1.8～2.0 | 9.23 | 36.69 | 14.73 | 86.15 | 17.09 | 23.08 | 43.52 | 1.9 | 9.23 |
| >2.0 | 13.85 | 48.07 | 31.74 | 100.00 | 21.38 | 13.85 | 48.07 | | |
| Total | 100 | 21.38 | 9.16 | | | | | | |

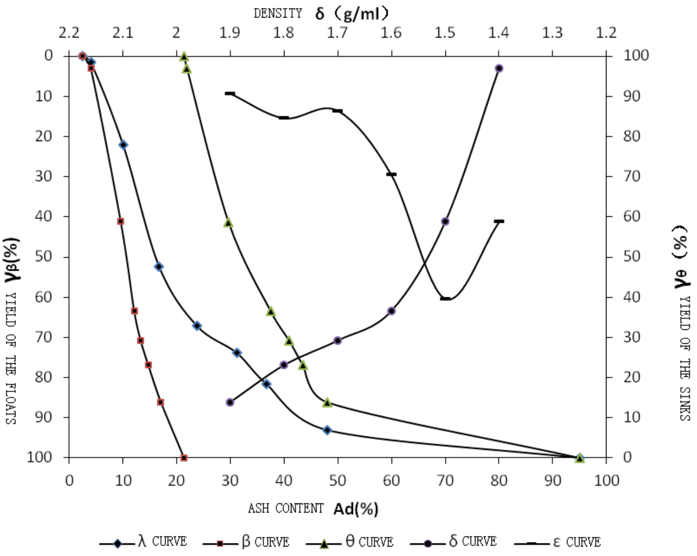
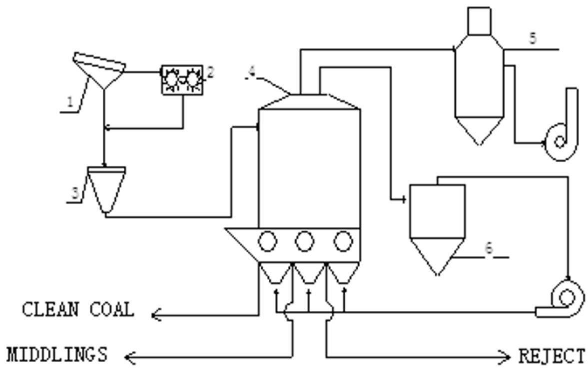



**Figure 3.** Sifeng Mine 80～6mm Size Fraction Raw Coal Washability Curve

It is shown in Table 1 that: for the mineral material of low density class: -1.4 g·cm$^{-3}$—yield：3.08%，ash content: 4.04%, sulfur content: 2.18%; -1.5 g·cm$^{-3}$ —yield: 38.15%, ash content: 10.06%, sulfur content: 3.05%; for the density class of -1.8 g·cm$^{-3}$—yield：6.15%， ash content: 31.17%, sulfur content: 8.88%; for the density class of -2.0 g·cm$^{-3}$—yield：9.23%， ash content: 36.69%, sulfur content: 14.73%. It can be concluded in the table that there is a high ash content in high density materials and its total sulfur content is obviously higher than low density materials, and the sulfur content concentrates mainly in high density materials, therefore, all of the above indicates that the sulfur mainly exists in high density materials in the form of inorganic sulfur. Considering the above mentioned coal quality data analysis, we can draw a conclusion that this type of coal is absolutely suitable for the dry desulfurization separation using ZM high efficient mineral separator.

**（2）Process Flowsheet**

The before-combustion industrial desulfurization test was conducted by ZM High Efficient Mineral Separator developed by Tangshan Shenzhou Manufacturing Co., Ltd. The system flowsheet is shown in Figure 3. Production process: the raw coal was classified by a 80 mm sizing screen, of which + 80 mm was fed into the crusher and then after crushing they are sent to the coal conveyor belt together with -80mm, finally all the raw coal under 80mm will be fed into the high efficient mineral separator through a vibrating feeder. The coal dust generated by process of separation is dedusted by a two-phase dusting system of both cyclone dust collector and bag dust collector, and the recycled coal dust can be reused. After separation, clean coal, middlings and reject are conveyed respectively to the stock yard by clean coal belt conveyor, middlings belt conveyor and reject belt conveyor.




Classifying screen 2-crusher 3-surge bin 4-mineral separator 5- bag filter 6-dust cyclone

**Figure 4.** Process Flowsheet of ZM Mineral Separator

## 3. Industrial Test Result Analysis

The industrial desulfurization test was carried out in No.3 coal yard of Tongzi power plant by ZM high efficient mineral separator. The test results are shown in Table 2. According to different parameters requirements of coal separation test, the settings of the equipment was properly adjusted, when it ran stable, sampling work was started and then quality of clean coal, middlings and rejects analyzed.

**Table 2.** Separation Result of ZM Separator

| Name | Yield/% | Ash/% | Sulfur/% |
|---|---|---|---|
| Clean Coal | 73.06 | 14.74 | 4.43 |
| Middlings | 15.30 | 33.46 | 13.27 |
| Reject | 11.64 | 47.21 | 33.46 |

According to the test result, we can get to know that for this kind of high-sulfur coal, after separation by ZM high efficient mineral separator, the clean coal yield is 73.06%, ash content is 14.74%, and sulfur content is 4.43%, which is reduced by 4.73% in comparison to the raw coal. Meanwhile, the ash and sulfur content in discharged reject are 47.21% and 33.46% respectively. Apparently, it indicates that a relative ideal separation effect has achieved.

## 4. Conclusions

1. It is completely feasible to use ZM high efficient mineral separator to conduct a before combustion separation process to remove sulfur and ash for the high sulfur fuel coal of Guizhou Huadian Tongzi power plant.
2. After separation, the sulfur content of clean coal is 4.43%, and it has reduced by 4.73% in comparison to the raw coal, which has fully meet the sulfur content requirements of power plant fuel coals.
3. ZM high efficient mineral separator is able to achieve an effective separation for high density materials and most of the waste rocks can be rejected.
4. The desulfurization cost of the power plant can be significantly reduced by means of before combustion dry desulfurization and deshaling process to remove sulfur and ash content from the power plant fuel coal.

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# Dry Cleaning, an Affordable Separation Process for Deshaling Indian High Ash Thermal Coal

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**Abstract:** Coal continues to play a major role in the economic development of India, especially in conventional power generation plants. The wet beneficiation process for coal cleaning is currently the predominant method of purification of coal in the world. However, dry beneficiation of coal has obvious advantages over wet processes. The potential of deshaling high ash Indian thermal coal with FGX separator has been evaluated in this paper. The problems caused by high ash content in Indian thermal coal are reviewed. The evaluation has been performed on raw coal from Adani by simulation study of different deshaling process options. Regardless of the types of mineral matter, a clean product having acceptable market quality was generated and pure rock removal into the reject stream was achieved with little coal loss by FGX separator. The simulation results show that FGX separator has the capability of maximizing the reject amount of high-density rock prior to transportation and processing. This paper also provides an overview of raw coal quality, FGX dry separator working mechanism, and process simulation of deshaling. Furthermore, it discusses the potential usage of dry separation in coal industry of India.

**Keywords**: dry coal cleaning, deshaling, computer simulation, flowsheet, coal preparation, thermal coal, high ash coal, India

## 1. Introduction

Even though India has a dominant share in word reserves, Indian coal deposits are generally of high ash content varying from 36 to 50 percentage and it features in poor washability, which poses significant challenges to end users [1]. For example, Kalyan Sen et al. assessed washability characteristics of representative Indian thermal coal seams of West Bokaro to predict separation results during coal beneficiation using gravity methods [2]. The index for washability (IW) of different coal seams varied between 8 and 41 with an average of <20. These features result in high ash and the high percentage of near gravity material (NGM) at 1.45-1.65 cut density range and thus the coals become difficult to wash. Furthermore, due to predominance of open pit mining, the 'out of seam' dilution is also high, which results in the supply of coal with high ash and inconsistent quality to most consumers. The typical distribution of such mineral matter demands judicious preparation and beneficiation of effective end use.

In coal preparation industry, it tends to use heavy medium separation to process those raw coal showing difficult washability. Large amount of reject rock entering into the dense medium system and no matter if it is a heavy medium vessel or a heavy medium cyclone, either will bring huge disadvantage and harm to the whole system. And it even causes system failure. The disadvantages of treating high ash raw coal include limiting system throughput and causing system jam, causing equipment wearing, degradation of rock, and increasing operation cost.



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Dry deshaling offers significant advantages over wet cleaning operations, which includes very economic capital cost and operation cost, reducing product surface moisture, enhancing heating value, eliminating of expensive coal slurry water processing system, and reducing transportation of large amounts of ash-forming minerals.

Several dry processing technologies such as air jig [3], air-dense medium fluidized bed separator[4-7] and FGX separator [8] have been successfully tested and some technologies commercialized in the past. The All-air Jig has been successfully applied in the U.S. for coal cleaning [9]. FGX coal dry separator is potentially one kind of very attractive method for upgrading Indian coal quality. The FGX Separator is a successful example of a Chinese dry, density-based separation technology that has several hundred commercial installations [10].

To evaluate this potential, a process calculation simulation of dry separation and a pilot-scale air table deshaling unit was tested for one Indian thermal coal in the 60x0mm size range.

**2. Structures of Compound Coal Dry Separator**

As shown in Fig. 1, the compound dry separator is composed of separation deck, vibrator, air chambers, frame and hanging mechanism, raw coal surge bin and other parts. Vibrator is fixed on the separation deck electric motor frame by two vibration electric motors and the separation deck with vibrator is suspended on the machine frame by four steel wire ropes with damping springs.

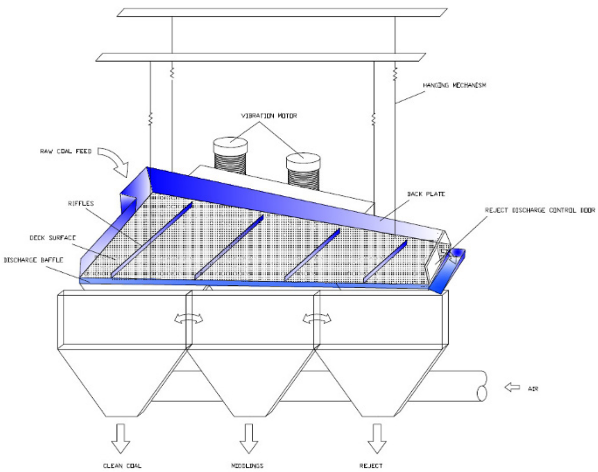


Fig.1 Compound dry separator structure drawing

**2. Raw Coal Quality**

For comparison of separation of different processes, a typical Indian thermal coal sample from Adani is used for this study. Its quality is summarized in Tables 1 to 2.

Table 1 Screen analysis of -50mm raw coal

| Size(mm) | Wt, % | Ash, % |
|---|---|---|
| 50-25 | 54.2 | 49.9 |
| 25-13 | 19.5 | 47.0 |
| 13-6 | 10.7 | 43.7 |
| 6-3 | 5.8 | 34.1 |
| 3-0.5 | 6.8 | 25.7 |
| -0.5 | 3.0 | 25.1 |
| Total | 100.0 | 45.4 |

Table 2 The Combined Washability data developed from F&S test results of 0.5-50mm

| Sp.Gr | | | Cumulative Float | | Cumulative Sink | |
|---|---|---|---|---|---|---|
| | Wt, % | Ash, % | Wt, % | Ash,% | Wt, % | Ash,% |
| <1.40 | 7.80 | 14.70 | 7.80 | 14.70 | 100.0 | 40.8 |
| 1.40-1.50 | 16.60 | 25.10 | 24.40 | 21.78 | 83.6 | 46.3 |
| 1.50-1.60 | 14.90 | 36.30 | 39.30 | 27.28 | 62.8 | 52.5 |
| 1.60-1.70 | 13.70 | 45.30 | 53.00 | 31.94 | 43.7 | 59.4 |
| 1.70-1.80 | 6.50 | 51.70 | 59.50 | 34.10 | 29.9 | 65.3 |
| 1.80-1.90 | 12.60 | 56.50 | 72.10 | 38.01 | 21.5 | 70.7 |
| >1.90 | 27.90 | 70.40 | 100.00 | 47.05 | 14.9 | 75.4 |
| Total | 100.00 | 47.05 | | | | |

The clean coal quality requirement is 32-35% ash, and according to the float & sink test result analysis in Table 2, the separation density must be higher than 1.80 g/cm$^3$ in order to get washed coal of accepted quality.

## 3. Simulation Results

### 3.1 Comparison of different deshaling methods

For comparison purpose, at same deshaling cutting density pf 1.75 Sp. Gr., the separation performance of same 50-13mm size range coal by dry separator, jig and heavy medium vessel are compared in case 1, case 2 and case 3. It is evident that heavy medium vessel has the best deshaling effect (highest clean coal yield at highest refuse ash content). However, the clean coal yield difference between dry separation and heavy medium separation is small (less than 1.5 percentage points) and the corresponding clean coal ash difference is less than 1.2 percentage points. The separation performance difference is attributed to high separation efficiency of heavy medium vessel.

Table 3 Comparison of different deshaling process options

| | Product | Yield, Wt % | Ash% |
|---|---|---|---|
| FGX | Clean coal | 65.34 | 36.56 |
| | refuse | 34.66 | 61.99 |
| Jig | Clean coal | 65.29 | 35.82 |
| | refuse | 34.71 | 63.36 |
| Heavy medium vessel | Clean coal | 66.77 | 35.48 |
| | refuse | 33.23 | 65.26 |

### 3.2 Deshaling density

Dry separator can efficiently process raw coal having a size range of 50-3mm. The dry separation is simulated at deshaling density of 1.8, 2.0 and 2.2 in case 4, 5 and 6 respectively. At cutting density of 1.8, dry separation can produce a clean coal product with ash content of 34.96% and clean coal yield is 61.74%. Increasing the cutting density from 1.8 to 2.2, the clean coal yield is increased to 82.68% while clean coal ash is increased to 40.94%. This hints that: 1) for producing qualified clean coal product (ash <35%), the deshaling density must be less than 1.8; 2) the applicable bottom size of dry separation bottom size can be 3mm; 3) the high ash reject 66.54% can be removed at cutting density of 2.2. Dry deshaling can remove high ash rocks in raw coal due to out-of seam dilution to provide a consistent quality of coal.

Table 4 Effects of deshaling density on dry separation

| Deshaling density | Product | Yield, Wt % | Ash% |
|---|---|---|---|
| 1.8 | Clean coal | 61.74 | 34.96 |
| | Refuse | 38.26 | 62.19 |
| 2.0 | Clean coal | 73.16 | 38.16 |
| | Refuse | 26.84 | 65.04 |
| 2.2 | Clean coal | 82.68 | 40.94 |
| | Refuse | 17.32 | 66.54 |

### 3.3 Effect of feed size range

At same clean coal ash=32%, the dry separation of 50-3mm raw coal can produce a slight higher clean coal yield than processing 50-6mm raw coal. Due to difficulty in classification of raw coal at 3mm, for this coal, the appropriate separation bottom size limit is 6mm.

Table 5 Effects of feed size range on dry separation

| Process | Product | Yield, Wt % | Ash% |
|---|---|---|---|
| 50x6mm was separated by dry separation, -6mm raw coal is blended with clean coal | Clean coal | 46.46 | 32.00 |
| | Middlings | 34.40 | 50.91 |
| | Refuse | 19.14 | 67.90 |
| 50x3mm was separated by dry separation, -3mm raw coal is blended with clean coal | Clean coal | 47.79 | 32.00 |
| | Middlings | 32.48 | 51.51 |
| | refuse | 19.72 | 67.69 |

## 4 Pilot Scale Testing

The deshaling test was completed on pilot scale testing unit. The model is FGX-1 which has a capacity of about 10 TPH. Raw coal is provided by Adani Company from India. Raw coal has a high content of 50.90%; the HHV and LHV of raw coal are 3692 and 3243 kcal/kg respectively. Ten tons of high ash raw coal was tested, and then four samples were collected along deck edge ranging from feed end to refuse discharge end of dry separator. They are clean coal 1, clean coal 2, middlings and refuse. The total clean coal yield is 60.43% at clean coal ash of 34.85%, which meets clean coal quality requirement (ash is less than 35%). As shown in Table 6 most of high ash refuse can be removed from raw coal and LHV of cumulative light product (clean coal +middlings) is 4063 kcal/kg. The rejected high ash rock consists of 33.58% of raw coal feed and reject ash is as high as 79.16%.

Table 6 Deshaling Testing Results of Adani High Ash coal

| PRODUCT | FRACTIONAL | | | | CUMULATIVE | | | |
|---|---|---|---|---|---|---|---|---|
| | Wt% | Ash % | HHV, kcal/kg | LHV, kcal/kg | Wt% | Ash% | HHV, kcal/kg | LHV, kcal/kg |
| Clean Coal 1 | 45.74 | 32.11 | 5072 | 4374 | 45.74 | 32.11 | 5072 | 4374 |
| Clean Coal 2 | 14.69 | 43.37 | 4169 | 3648 | 60.43 | 34.85 | 4852 | 4197 |
| Middlings | 5.99 | 54.44 | 3056 | 2708 | 66.42 | 36.61 | 4690 | 4063 |
| Reject | 33.58 | 79.16 | 904 | 778 | 100.00 | 50.90 | 3419 | 2960 |
| Total | 100.00 | 50.90 | 3419 | 2960 | | | | |

## 5 Conclusions

High ash Indian thermal coal is categorized as difficult to wash coal and there is a strong need for pre-deshaling of raw coal and providing deshalled coal of quality consistent to customers. Dry separation is a density-based separation that utilizes the combined separating principles of an autogenous fluidized bed and a pneumatic table concentrator. Dry separation shows many advantages over conventional deshaling methods in terms of separation performance, operation complexity, capital cost and operation cost. A process simulation of dry separation has been evaluated at several scenarios for the treatment of high ash run-of-mine coal from India. The simulation results show that FGX separator can produce a clean coal having qualities that meet contract specifications and has the capability of maximizing of the reject amount of high-density rock prior to transportation and processing. The FGX Separator provides a relatively efficient separation at high separation density values of around 1.8 RD to 2.2 RD.

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# Destoning of fine coal in a fluidized bed

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**Abstract**

Water scarcity is driving the development of dry coal beneficiation processes. A lot of research has gone into the development of dry dense medium fluidized bed technology (DMFB), especially in China. However, these processes focus mainly on +6mm particles while little work has gone into the development of dry processes for -2mm particles. This paper focusses on the possibility to remove mineral matter (high density particles) from valuable coal fines (-2+1mm and -1+0.5mm) by using a fluidized bed operated with and without added vibration, while adding dense media (magnetite, sand and fine coal discards). As a control the bed was also operated without any dense media.

A fluidized bed column was designed and constructed by clamping several individual rings on top of one another. This design helped with the sampling of the coal in the bed. After a test run, each individual ring could be removed and the coal inside analysed for ash percentage and calorific value. The results clearly indicated that this process is viable by removing high ash value material in the bottom layer of the bed, leaving the rest of the bed to be significantly lower in ash value and as a result higher in the calorific value of the coal. It confirmed previous work by the authors that showed a negative overall performance of the bed when media was added, due to difficulty in separating the media from the fine coal particles. By vibration the bed, the sharpness of separation did increase slightly.

**Key words**: dry coal beneficiation, fine coal, fluidized bed

## Introduction

Although water is one of the most abundant resources on the earth, there are several arid regions that sees very little to no rainfall per annum. This is particularly true for some countries with vast coal reserves such as India, China, Russia, Mongolia and South Africa, some of which have a shortage of clean process water near coal reserves. Transporting water to these areas with a shortage thereof would be very expensive and unpractical to maintain (Houwelingen & de Jong, 2004:335).

Throughout most of South Africa water is scarce during winters (Philander, 2010). A detailed study into South Africa's water resources revealed that only 1200m$^3$ of fresh water is available per capita per year. Moreover, South Africa has an average of 464mm of rainfall per year (Zhao et al., 2010a). To put this into perspective, a study indicates that for one ton of coal, 3 – 5 tons of process water is needed in a wet jigging process (Chen & Wei, 2003). In the Waterberg area, which has one of the largest coal deposits in South Africa, large scale plant development could be limited due to insufficient water supplies (Eberhard, 2011). The same applies to coal resources in Botswana.

 
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New discoveries of vast coal reserves were made in Mongolia recently. More than 200 coal deposits were found which consists of 152 billion tons of coal (Erdenetsogt et al., 2009). China, which is geographically situated next to Mongolia, is the largest steel producer in the world. According to Levin (2012) there is sufficient coal in Mongolia to fuel China's substantial demand for the next 50 years. However, Mongolia is a perfect example of a country with enormous coal reserves but not enough water to run economically viable wet beneficiation processes in some regions. The only viable alternative is to implement dry beneficiation.

Coal is primarily washed using dry or wet processes. Wet beneficiation methods require vast amounts of process water as emphasized by Chen & Wei (2003), but yield much sharper separation efficiencies than dry processes. Considering the major problem of a clean water shortage worldwide, the focus of research should be on effective dry beneficiation methods of coal (Yang et al., 2012a).

A dry beneficiation method which could be a possible solution to the problem is the dense medium fluidized bed (DMFB) technology. This technology is classified as a dense medium beneficiation, which uses gas and solid particle interactions as well as the law of gravity to stratify coal according to density (Luo et al., 2007).

Over the years several research articles have been written on every aspect of fluidization, and it was found that this technology holds many advantages especially in the coal washing industry (Mohanta et al., 2013). Chen & Yang (2003) describes these advantages to be:

**High precision**: Coal with a size range of 50 mm x 6 mm can effectively be separated with Ep values of 0.05-0.07. These values compare favourably with the existing heavy medium wet beneficiation.

**Low investment**: The same capacity dry beneficiation plant can be constructed for half the cost compared to a wet beneficiation plant. This is due to the fact that no complicated and costly slurry treatment is needed when handling dry coal.

**No environmental pollution**: This technology only requires low pressure compressed air. It also operates smoothly with very little noise pollution. The dust emitted by the equipment is within environmental laws.

**Wide ranges of beneficiating densities**: Beneficiating densities ranging from 1.3 to 2.2 g/cm3 can be created by adding magnetite powder to the bed. Thus this technology can either remove heavy gangues or lower density clean coal depending on the required product.

**No moisture penalties**: Because only air is used in this process, cleaning water is not needed and the product coal is not penalized due to exceeding moisture levels (Luo et al., 2008).

**Product thermal quality**: A dryer product with a higher calorific value per ton of coal is produced (Sahu et al., 2009).

**Transportation costs**: Due to low moisture content, transportation of the coal does not include the additional cost of transporting the weight of water (England et al., 2002).

However, research on dry coal beneficiation methods is mostly conducted on coarse coal (+6 mm) which yielded promising results but are not applicable to fine coal fractions (-2000 + 500µm). In recent years coal fines produced by the modern mechanized mining procedures has increased dramatically (Le Roux et al., 2005). Due to these modern methods, up to 15% of run-of-mine coal is in the -500 µm size fraction (England et al., 2002). Methods of dry fine coal beneficiation are therefore important to consider due to the vast amount of valuable fine coal that is discarded every year.

**Experimental**

A 150mm inner diameter fluidized bed column was designed and constructed from poly-carbonate material (Figure 1). It was assembled in a series of rings, clamped on top of one another to aid in the sampling after fluidization. After feeding the fluidized bed, it was operated for 15 minutes at a velocity just above the incipient fluidization point, where after the airflow was closed and the contents inside the chamber were allowed to settle. The material contained in different sections was then carefully removed and analyzed for density, calorific value and percentage ash (mineral content). As an added variable, vibration was added to the bed during some of the tests.

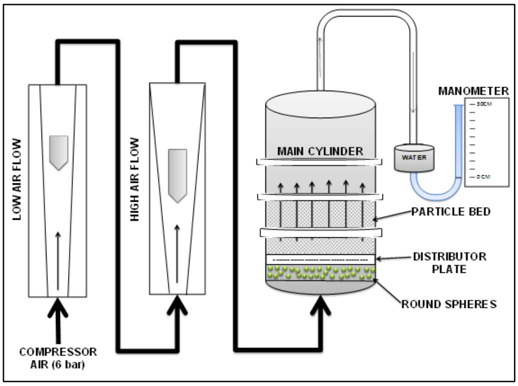


**Figure 1: Fluidized bed setup.**

Coal with two particle size distributions namely -2+1mm and -1+0.5mm was used together with three different media; fine sand (-212µm), magnitude (-75µm) and discard fine coal (-500µm with SG=1.7). The mixtures were done on a weight base ratio medium to coal of 70:30 and 50:50. As a control a series of tests was done without media. The coal used in this study a typical South African bituminous coal of which the ash yield and calorific values are given in Table 1.

**Table 1: Partial proximate and CV analysis**

| | Value | Standard |
|---|---|---|
| % Inherent moisture content (air-dried) | 2.4 | ISO 11722: 1999 |
| % Ash content (air-dried) | 21.6 | ISO 1171: 2010 |
| Calorific value (MJ/kg) (air-dried) | 24.5 | ISO 1928: 2009 |

**Results and discussion**

An extensive set of results was generated during this project, of which only selected highlights are presented here.

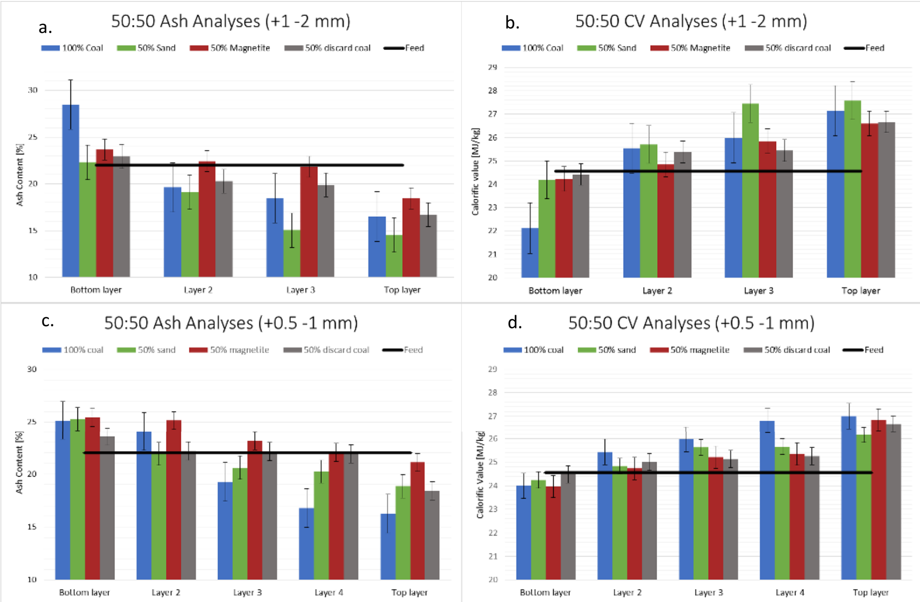


**Figure 2: Results obtained for different PSD and mediums fed to a fluidized bed in a 50:50 ratio.**

Figure 2 indicates the possibility of destoning fine coal using a fluidized bed. The graphs in Figures 2a. and 2c. show a definite increase in ash yield percentage in the bottom part of the bed while Figures 2b. and 2d. indicate the corresponding increase in calorific value of the coal at the top part of the bed. The larger particles clearly show an increase in separability in relation to the smaller particles. This is attributed to smaller particle-particle contact surfaces for the larger particles, leading to lower frictional forces between the particles; increasing the ease in separation.

An unexpected result that was obtained for both PSD ranges was the increase in the sharpness of separation for the coal only fluidized beds in comparison to any of the dense media beds. The reason for the apparently poor performance of the dense media beds are attributed to the difficulty in recovering the dense media from the coal after fluidization. This was quantified by observation of the various colours of the residual coal ash during analysis as well as XRD performed on the coal ash. The combination of fine coal particles and ultra-fine dense media, especially the magnetite, increase the cohesion forces between the coal-coal, coal-media and media-media particles. Some of the dense medium particles also get trapped into the macro pores of the coal particles and then remain lodged there. Similar results were found for a bed with a medium to coal ratio of 70:30 and are therefore not shown.

The addition of vibration to the fluidized bed increased the destoning efficiency of the bed when operated both with and without any media. It opens up additional pathways for denser particles to settle to the bottom of the bed, therefore eliminating the amount of high density particle carry-over. This is especially true for smaller size percentages in each of the specific PSD feeds as well as when magnetite was added as a dense medium (Figure 3). This said, the resulting error-bars for both data sets do overlap to such a degree that a clear conclusion is difficult to reach. However, previous work done by Le Roux et.al. (2015) showed this effect to be real.

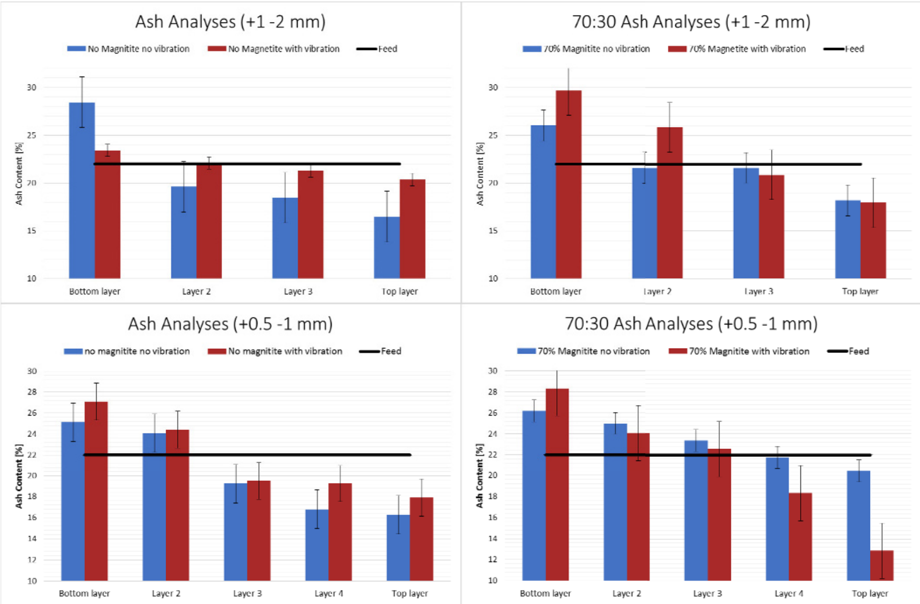


**Figure 3: Comparison between vibration added to the bed and no vibration.**

## Conclusions

This paper investigated the possibility of upgrading -2+1mm and -1+0.5mm coal in a dry fluidized bed with and without added dense media. It showed that this technology is viable to achieve destoning in the bottom 20 to 30% of the bed especially if no dense medium was added. The addition of vibration to the bed does seem to increase the sharpness of separation during operations. The difficulty in recovery of the ultra-fine dense medium from the coal particles due to particle-medium adhesion and entrapment of the medium in the coal macro pores gave a false indication of the performance of the bed, but should be kept in mind in terms of overall plant performance.

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# DRY ALLAIR® WASHING PILOT PLANT TEST RESULTS ON NARYN SUKHAIT'S MULTI LAYER COAL

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**ABSTRACT:** Coal is the main source of energy vitally important asset that drives the economic development of the country. Mongolia's geological probable reserves of coal are estimated at 173.3 billion tons which about 70% is lignite. There are over 300 deposits and occurrences in 15 different basins at which over 21.5 billion tons of coal reserves are estimated through detailed exploration work. Presently about 20 coal companies export unprocessed and processed coals.

Surface water expected in Mongolian coal and other minerals and less environmentally friendly, low-cost and dry washing issues are standing a problem. Regarding the Naryn Sukhait coal washing project multilayer coal of seam taken and send to Allmineral German company Allair® dry technology in order to possibility of washing pilot plant tests.

Due to lack of surface water in Mongolia and to be environmentally friendly and economically towards the others minerals, the dry processing technology is at of great interest.

In order to determine suitability of allair® dry technology from Allmineral Germany, these pilot plant testing is performed for the multi-layer, high ash contented coal within Naryn Sukhait Washing Plant Project.

***Key Words: Mongolian coking coal, dry process, allair®, sampling, pilot test, clean coal, ash, pilot plant.***

## I. ALLAIR DRY TECHNOLOGY

Idea of allair® technology arose originally in 2000 and has been commercialized since 2002 for the pilot plant testing purpose.

As of today, there are in total 75 allair® units are operating around the globe, such as in USA, Ukraine, India, Turkey, Spain, Colombia and Brazil.

Raw material requirement for the technology is to have coal moisture less than 7%, particle size within 0-50 mm and each unit has capacity of 5-100 tph.

FGX technologies mean differentiating feature is that the raw coal is analyzed instantly through nuclear sensor and air pulsed coming from the bottom is regulated automatically. General operating scheme of the unit is shown below on figure 1.

Today, there are smallest allair® unit (5tph) operating in Ukraine, while the largest allair® unit (600 tph) running in Spain and are shown below on figure 2 [2].



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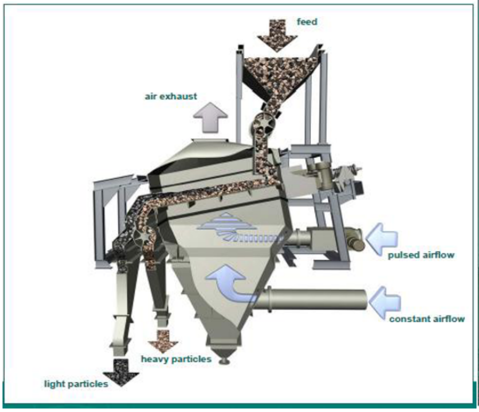


**Figure 1. allair® technology principle** **Figure 2. allair® plant, capacity 50 tph, Ukraine**

## II. SAMPLING

As requested by the Allmineral for the pilot plant testing, total of 300.5 kg representing bulk samples were sent and they were extracted from 3 points in mine-1 area.

The Samples were packaged in closed plastic package with tags and were sent to Germany on 6th April 2013.

**Sample area, methods:**

1. **Mine-1 mid area between (1-7) (1-8) at 18 C block:**
   Representing samples from bottom, middle, upper and the outer areas were taken from the dump formed by excavator
2. **Floor, Ceiling, dump polluted with explosives (1-10) (1-11) line:**
   Shovel were utilized to get samples from floor, ceiling areas and samples containing explosive stones.
3. **Mine-1's South west wall between (1-13)(1-14) line:**
   Up to 5 m thick coal from multi-layer area exposed to atmosphere from Mine-1, South west were taken with shovel (Figure 3)

In order to determine properties of the original samples, 2 samples from each points were collected for analyzing and their results are shown on table 1:

**Figure 3. Sampling**

**Table 1. Raw Coal quality**

| Location | Standard | | | | | | |
|---|---|---|---|---|---|---|---|
| | MNS 655:79 ISO, 589 | MNS 652:1979, ASTM D3174 | | MNS 654:1979, ISO 17246, ASTM D 3175 | | MNS ISO 501: 2003 ASTM D 720 | ISO 15585:2006 |
| | Moisture,% | Ash, % | | Volatile matter, % | | Free Swelling Index | Coking Index |
| | War, | Aad | Aar | Vad | Vdaf | FSI | G |
| 1-1 | 1.76 | 8.2 | 8.1 | 31.7 | 35.1 | 1 | 11 |
| 1-2 | 2.23 | 11.2 | 11.2 | 31.1 | 35.8 | 0 | 0 |
| 2-1 | 5.51 | 31.7 | 31.1 | 26.1 | 40.3 | 0 | 0 |
| 2-2 | 8.92 | 25.4 | 24.7 | 27.4 | 39.9 | 0 | 0 |
| 3-1 | 3.29 | 36.7 | 36.1 | 27.7 | 45 | 0 | 0 |
| 3-2 | 4.15 | 21.6 | 21 | 32.3 | 41.9 | 1 | 11 |

## III. PILOT PLANT TEST

The pilot plant testing were carried out by Allmineral's experts on pilot plant testing unit located in Duisselberg on 20-24th April, 2013.
For testing bulk sample were blended and mixed until particle size had become 0-40 mm.

After that, the samples were divided into 0-5mm, 1-5mm, 5-40mm particle sizes and the experiment were performed. The results are shown on table 2, 3, 4 [4].
Based on the test result, correlation between clean coal yield and ash is illustrated on figure 4 [3].

**Table 1. Test results (0-5mm)**

| Products | Yield, Y, % | Ash, $A^{dry}$, % | Float | | Sink | |
|---|---|---|---|---|---|---|
| | | | Yield, Y, % | Ash, $A^{dry}$, % | Yield, Y, % | Ash, $A^{dry}$, % |
| Product - 1 | 4.64 | 11.6 | 4.64 | 11.6 | 100.0 | 18.90 |
| Product - 2 | 13.68 | 12.22 | 18.32 | 12.06 | 95.36 | 19.26 |
| Product - 3 | 14.48 | 12.8 | 32.8 | 12.39 | 81.68 | 20.44 |
| Product - 4 | 14.8 | 14.5 | 47.6 | 13.04 | 67.2 | 22.08 |
| Product - 5 | 14.8 | 15.21 | 62.4 | 13.56 | 52.4 | 24.22 |
| Product - 6 | 15.41 | 15.68 | 77.81 | 13.98 | 37.6 | 27.77 |
| **CLEAN COAL** | **77.81** | **13.98** | | | | |
| Product - 7 | 16.08 | 36.13 | 93.89 | 17.77 | 22.19 | 36.17 |
| Middling | 5.84 | 34.81 | 99.73 | 18.77 | 6.11 | 36.26 |
| Sand | 0.27 | 67.71 | 100.0 | 18.90 | 0.27 | 67.71 |
| **REJECT** | **22.19** | **36.17** | | | | |
| **TOTAL** | **100.0** | **18.90** | | | | |

**Table 2. Test results (1-5mm)**

| Products | Yield, Y, % | Ash, $A^{dry}$, % | Float | | Sink | |
|---|---|---|---|---|---|---|
| | | | Yield, Y, % | Ash, $A^{dry}$, % | Yield, Y, % | Ash, $A^{dry}$, % |
| Product - 1 | 10.42 | 13.37 | 10.42 | 13.37 | 100.0 | 25.16 |
| Product - 2 | 13.98 | 16.39 | 24.4 | 15.10 | 89.58 | 26.53 |
| **CLEAN COAL** | **24.4** | **15.10** | | | | |
| Product - 3 | 14.02 | 17.06 | 38.42 | 15.82 | 75.6 | 28.41 |
| Product - 4 | 14.6 | 17.25 | 53.02 | 16.21 | 61.58 | 30.99 |
| Product - 5 | 14.81 | 17.41 | 67.83 | 16.47 | 46.98 | 35.26 |
| Product - 6 | 15.21 | 31.93 | 83.04 | 19.30 | 32.17 | 43.48 |
| Product - 7 | 16.96 | 53.84 | 100.0 | 25.16 | 16.96 | 53.84 |
| **REJECT** | **75.6** | **28.41** | | | | |
| **TOTAL** | **100.0** | **25.16** | | | | |

**Table 3. Test results (5-40mm)**

| Products | Yield, Y, % | Ash, $A^{dry}$, % | Float | | Sink | |
|---|---|---|---|---|---|---|
| | | | Yield, Y, % | Ash, $A^{dry}$, % | Yield, Y, % | Ash, $A^{dry}$, % |
| Product - 1 | 7.57 | 6.16 | 7.57 | 6.16 | 100 | 25.39 |
| Product - 2 | 13.46 | 6.56 | 21.03 | 6.42 | 92.43 | 26.97 |
| Product - 3 | 13.97 | 7.94 | 35 | 7.02 | 78.97 | 30.45 |
| Product - 4 | 14.67 | 8.41 | 49.67 | 7.43 | 65 | 35.29 |
| Product - 5 | 14.3 | 18.41 | 63.97 | 9.89 | 50.33 | 43.12 |
| **CLEAN COAL** | **63.97** | **9.89** | | | | |
| Product - 6 | 15.73 | 33.95 | 79.7 | 14.64 | 36.03 | 52.93 |
| Product - 7 | 20.3 | 67.63 | 100 | 25.39 | 20.3 | 67.63 |
| **REJECT** | **36.03** | **52.93** | | | | |
| **TOTAL** | **100** | **25.39** | | | | |

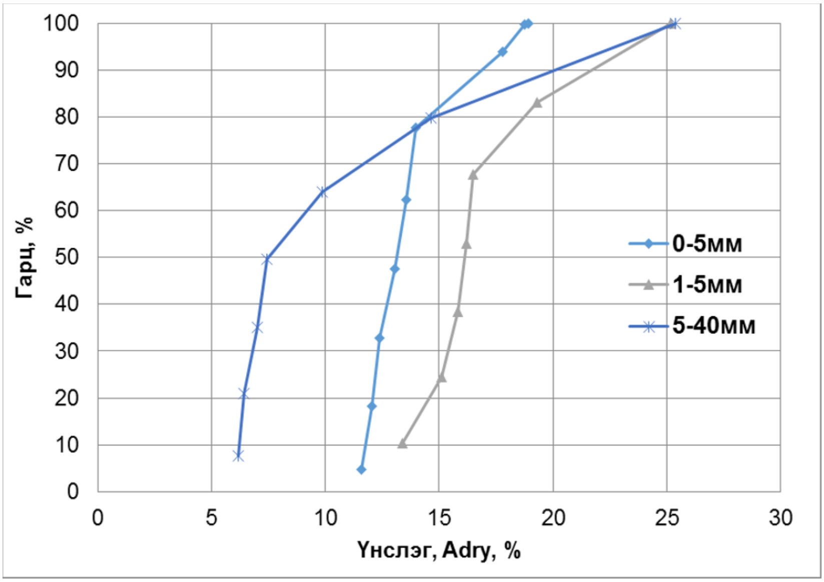


**Figure 4. Clean coal yield from ash (0-5mm, 1-5mm, 5-40mm)**

## SUMMARY

1. It is determined that clean coal containing 9.89% ash with 63.97% yield is possible to achieve by allair® dry technology from Naryn Sukhait multi-layer raw coal with 25.39% ash, 5-40mm particle size (60%).
2. Moreover, the same results can (clean coal ash 13.98%, yield 77.81%) be achieved from Naryn Sukhait multi-layer raw coal with 18.9%, particle size of 0-5mm.
3. Investment required for dry technology in comparison to wet technology is 3-4 times less, operation cost 5-7 times less.

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# APPLICATION OF VIBRATION PNEUMATIC SEPARATION

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## ANNOTATION

The represented material is devoted to disclosure of practical application of a method of vibration pneumatic preparation of coals, features of influence of parameters of regulation of production technological process. Actuality of application of this way of preparation consists in opportunities of application of the same installations with a low indicator of cost of processing of production in comparison with "wet" methods of preparation. It becomes attractive when processing the waste storages of coal rock that is at the same time and the solution of environmental problems. For improvement of an ecological situation in the world it is necessary to take measures for recycling of this wastes.

In this article expounded the developed theoretical regulations on management of process of vibration pneumatic separation on the basis of the analysis of mechanotronic system of the device. During factorial experiment is obtained the polynom equation testifying to significant influence of parameter of intensity of fluctuations of working body on indicators of efficiency of preparation of mineral taking into account a factor of looseness of the layer of separation material.

**Keywords: kinematic system, vibration pneumatic separators, working body, mineral processing, deck, process management, efficiency, selectivity**

## 1. PROBLEM AND ITS COMMUNICATION WITH SCIENTIFIC AND PRACTICAL TASKS

The expert assessment for the last decade testifies to the organization of the general storing of industrial wastes in Ukraine within 650 million tons year. Directly for the coal industry there is a task additional recovery of a power coal component from rock waste storages the mines and coal preparation factories, decrease in an ash-content of the wined original coals before sending to their consumers by cheap gravitational methods remains an actual task for many coal-mining company. The most acceptable method for the solution of the put task is vibro-pneumatic separation.

## 2. ANALYSIS OF RESEARCHES AND PUBLICATIONS

The analysis of constructions and the principle of operation of various devices of dry "waterless" - "dry" preparation of coal materials to contain in works [1, 2, 11, 12]. In work [3] are published results of comparative tests of separators of various producers and was shown that the separator of SVP-5,5x1

 
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provides high technological results. On the basis of a separator of SVP-5,5 x 1 is realized the project of the modular installation KPO-50 for "dry" preparation of original coals and off-balance carboniferous waste.

In this connection a certain interest represents research of influence of various parameters on indicators separation in separator SVP-5,5x1 in industrial conditions. Theoretical researches of interaction of particles in working space of a separator allowed to develop basic parameters of technological regulation of installation [4].

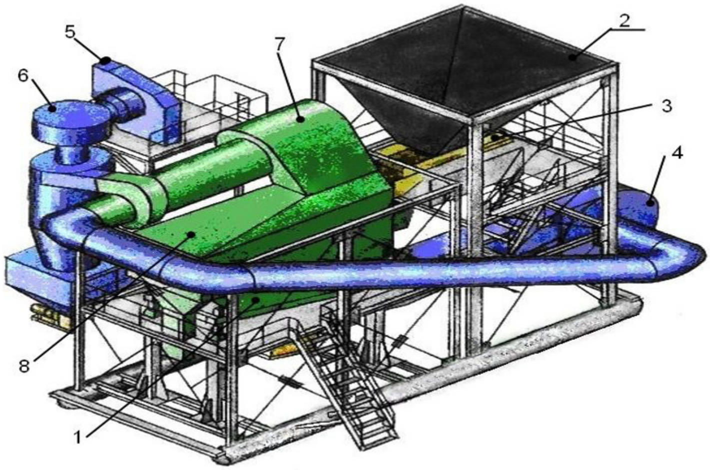


Fig. 1. Block scheme of a complex of vibro-pneumatic preparation of KPO-50
1 - SVP -5,5x1; 2- bunker; 3 – feeder; 4 – smoke sucker; 5 – aspirator; 6 – whirler; – spiral dust arrester; 8 – dust outtake cowl.

## 3. PROBLEM STATEMENT

The purpose of this work is implementation of the analysis of extent of influence of components of parameter intensity of swings of working body – a deck of a vibro-pneumatic separator on results of preparation. The condition of looseness of the layer of separation material of the processed material depends on dynamic parameters of excitement – amplitude of fluctuations and frequency of swings of a deck, pulsation mode of an air stream and its expense for the purpose of giving of a bed "pseudo the boiling layer". The fluidization condition of loose layer on a working surface allows to improve considerably to mend mutual penetration of layers from "easy" and "heavy" particles on a vertical component. Also essentially effects movement of vibration character of heavy fractions under action "tribo" effect in unloading area deck. These problems are solved by development of nonlinear mathematical model of research of a condition of mechatronny system and industrial tests. Industrial tests allow to study the principles and the nature of separation of loose coal material on a deck of a vibration pneumatic separator, and also to define the importance of the operating parameters.

## 4. STATEMENT OF MATERIAL AND RESULTS

On the “Mineral processing department” of “Donetsk National Technical University” is done work on improvement of a method of vibro-pneumatic separation, and also its introduction at utilization of rock dumps [5, 6, 11, 12].

In figure 2 is represented the three-dimensional model of improvement vibro-pneumatic separator of SVP-5,5x1.

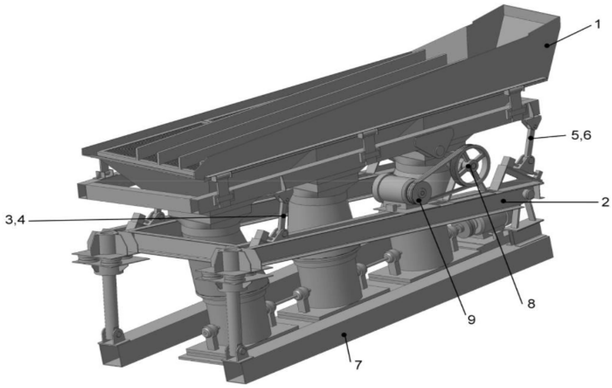


Fig. 2. Construction of vibro-pneumatic separators SVP -5,5x1.
1 – deck with the perforated surface; 2 – frame turning; 3, 4 – front hinges;
5, 6 – back hinges; 7 – supporting frame; 8 – vibroexciter; 9 – electric motor.

Application of a method of vibropneumatic separation allows to create small-sized, compact and mobile concentrating installations with a year-round cycle of work [7, 8].

The separator of the type SVP -5,5x1 works with the pulsing supply of technological air under the punched deck.

When swings the direction of the movement of a deck periodically changes thanks to what the bed by inertia is thrown up in the direction, perpendicular to the plane of support. As a result of tossings and simultaneous influence of a stream of air material of a bed is loosened and gets "fluidity" [9, 10, 11].

Thanks inclination to a deck in the cross direction and progressive motion of a bed the layer of easy particles which is settling down above directing, gradually "slips" down, at an angle to an axis of a separator and unloads along a board of separator in forward part of a deck. The lower layers of a bed which are between slideways, advance along them.

## 5. DEVELOPMENT OF NONLINEAR MATHEMATICAL MODEL OF THE MOVEMENT OF WORKING BODY OF MECHATRONICS SYSTEM

The working body of system on which moves the divided material, makes the plane-parallel movement. In assumptions of consideration of the kinematic scheme is entered consideration of flat section S into plane $O_1x_1y_1$ For determination of speed of any point of a deck we will consider the point M (*x,y*) deck, radius vector which F, also we will calculate its speed (fig. 3).

According to the theorem of speeds of points of the body under flat movement, speed of M is equal

$$\overline{V} = \overline{V}_A + \overline{\omega} F, \tag{1}$$

where $\overline{V}_A$ - strip speed $\overline{V_A} = V_{Ax_1}\bar{i}_1 + V_{Ay_1}\bar{j}_1$; $\overline{\omega}$ - vector of deck angular speed: $\overline{\omega} = \omega \overline{k}_1$.

Projections of $\overline{V_A}$ on axes $x_1$, $y_1$:

$$\overline{V_A} = V_{Ax_1}\bar{i}_1 + V_{Ay_1}\bar{j}_1 + \begin{vmatrix} \bar{i}_1 & \bar{j}_1 & \bar{k}_1 \\ 0 & 0 & \omega \\ x_1 & y_1 & 0 \end{vmatrix}, \tag{2}$$

The module or size of speed of any point of deck can be calculated using formula $V = (Vx_1^2 + Vy_1^2)^{1/2}$. Deck angular acceleration $\varepsilon = \dot{\omega}$.

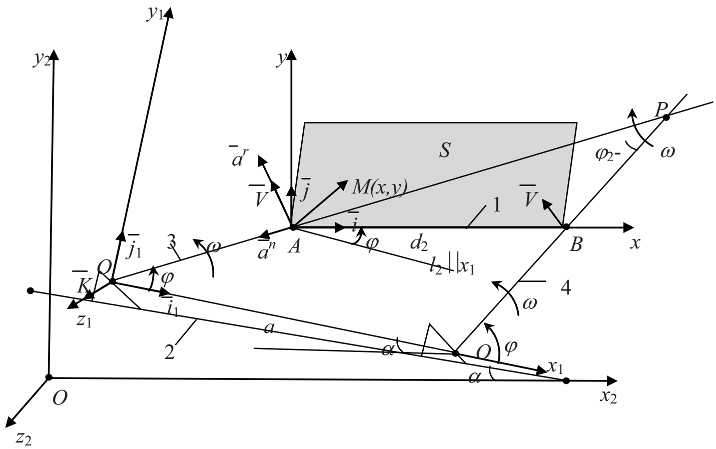


Fig. 3. Design model of determination of speed of a pole of a deck and its angular speed

Let's write the vector products in projection on the axis $x_1$, $y_1$ :

$$\overline{\omega} \times (\overline{\omega} \times \bar{r}) = \begin{vmatrix} \bar{i}_1 & \bar{j}_1 & \bar{k}_1 \\ 0 & 0 & \omega \\ -\omega y_1 & \omega x_1 & 0 \end{vmatrix} = -\omega^2 y_1 \bar{i}_1 - \omega^2 x_1 \bar{j}_1 . \tag{3}$$

Module of acceleration of any point on the deck:

$$\dot{a} = \sqrt{a_{x_1}^2 + a_{y_1}^2} . \tag{4}$$

Without restrictions on the movement is received the nonlinear equation of the movement of the mechanism of a vibro-pneumatic separator which has an appearance:

$$\ddot{x} + a_1(x)\dot{x}^2 + b_1(x)\dot{x} + c_1(x) = 0 \tag{5}$$

Having designated $\dot{x} = y$ and from differentiation on time to pass to differentiation on x, this differential equation of the second order will be transformed to the differential equation of the first order

$$\frac{dy}{dx} + a(x)y^2 + b(x)y + \frac{c(x)}{y} = 0 \tag{6}$$

Coefficients $a(x)$, $b(x)$, $c(x)$ with the equations depend from $x$ they include general quantities of the mechanism of a vibro-pneumatic separator of $\varphi_{10}$, $\varphi_{20}$, $\varphi_{30}$, $\varphi_{40}$, and $\gamma$, defining its configuration and on which depends separation process.

## 6. CONCLUSIONS AND DIRECTIONS OF FURTHER RESEARCHES

The analysis of data of work of installation testifies that the separator of SVP-5,5x1 provides high technological results when processing various on structure carboniferous raw materials. Efficiency and selectivity of separation in this case maximum in comparison with other series. At decrease in an ash-content of supply to 39% receiving a power concentrate in number of 66% is also possible.

It should be noted that the pneumatic installations on preparation created on the basis of a vibration separator of fan type are compact, don't demand considerable floor spaces and communications, are rather mobile, can be operated for the different purposes. Possibility of change-over of parameters of work of a separator in quite wide limits allows to operate quickly process of separation depending on properties of the arriving raw materials.

Solving the equation [6] it is possible to define a field of speeds and a field of accelerations of points of a deck by means of which it is possible to calculate forces of inertia of particles of the material which is on it in process of separation. This mathematical apparatus will allow to modeling process at preparation of loose raw material will allow to develop further theoretical bases of design and creation of vibration concentrating apparatus with mobile flat working body.

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# A STUDY OF THE DESHALING OF POLISH HARD COAL USING AN FGX UNIT TYPE OF AIR CONCENTRATING TABLE

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**ABSTRACT**

In 2012 the Institute purchased via Warkop Sp. z o.o. (limited liability), a distributor of this type of equipment in Eastern and Central Europe, an FGX-1 dry concentrating table - the smallest model with a throughput of 10 Mg/h. This unit was the basis for devising a semi-industrial test stand equipped with a feed module (hopper, inverter-driven belt feeder), a classification module (double deck vibration sieve), a crushing module (jaw crusher), a beneficiation module (dry concentration table) and a control module. This is the first installation of this type in Poland as well as within the EU (Baic i in., 2014).

The paper will present a set-up of the test stand where tests have been carried out since 2013. The study is focused at the removal of gangue from various types of coal using a dry method and the research methodology developed by scientists from the Institute.

The paper will present the results of studies on: obtaining clean coal products with high calorific value, removing sulphur from raw coal, deshaling of raw coking coal and obtaining a low coal content refuse.

**KEY WORDS**: coal deshaling, dry coal separation, concentrating tables, removal of pyrite sulphur, rock grains, steam coal, coking coal, middlings.

## 1. INTRODUCTION

For several years now technological solutions have been sought in Poland in order to reduce the production costs of various commercial size grades of hard coal. One of the methods to do so can be supplementing wet beneficiation processes with a preliminary dry deshaling process using dry concentrating tables. Air beneficiation has experienced a revival in recent years. It is used in many countries for preliminary removal of some part of both rock grains and part of fine coal grains sized below 0.5 mm from raw coal. The removal of waste material grains and grains of fine coal below 0.5mm using dry beneficiation leads, in many cases, to obtaining commercial products with quality satisfactory to end users (power stations, combined heat and power stations, etc.).In cases where high quality of commercial products (concentrates) is needed, dry separation can be used for preliminary removal of mineral impurities and fine coal smaller than 0.5 mm before the raw coal is directed to wet beneficiation processes. This is because the removal of most of the rock grains and part of fine coal grains simplifies the process set-up: the amount of feed directed to wet processes decreases, which consequently reduces the number of necessary units and equipment, also reducing the use of energy and water and the load on the water-slurry system. All this contributes to the reduction in the overall beneficiation costs. It should also be pointed out that waste materials obtained by dry beneficiation, due to having no contact with



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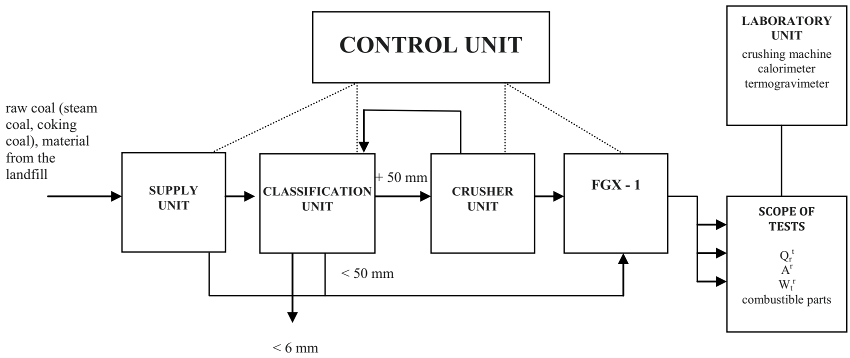
water, can be successfully used as a substitute to natural aggregate for engineering work or as a material for filling goafs. In papers presented at the International Coal Preparation Congresses (Johannesburg (Schen, 2002), Beijing (Li Gongamin, 2006), Lexington (Caner 2010), Istanbul (Ghost, 2013)) issues of dry coal separation were discussed. These papers have inspired scientists from the Katowice Branch of the Institute of Mechanised Construction to take steps to use FGX type of concentrating table for beneficiation/deshaling of Polish hard coals (Baic 2013,2014,2015).

## 2. RESEARCH INSTALATION FOR DESHALING OF COAL

The Tangsan Shenzou Machinery Co. Ltd. produces series of ten models of the air concentrating tables of the FGX type. Out of these models the IMBiGS chose the FGX-1 model with the lowest capacity (up to 10 Mg/h). The choice of this model was determined by the assumption that it will be a unit designed for the research purposes. The capacity of the FGX-1 allows to use it to the quarter- and semi-industrial tests. The air concentrating table of the FGX-1 type was complemented with the Polish auxiliary units. The power supply module, scales, systems for raw coal transport and for discharging the obtained separation products were constructed. A screen extracting the grains exceeding, determined at a given stage of tests, size (+50, +25, +6) was installed. A crushing machine grinding the feed of raw coal to the required grain class was also built-up. The block diagram of the research installation are shown in the picture 1.




**Picture 1 The block diagram of the research installation for preliminary dry coal deshaling**

## 3. THE WAYS OF CONDUCTING TESTS

As known form the Chinese experience, in case of the air concentrating table of the FGX type, the most effective is separation of the grains of the limit size 80-6 mm. The grain class 80-0 mm may also be a subject to enrichment process under the assumption that the share of the grain class 6-0 mm in total feed directed to the process does not exceed 15 %. On the test unit, equipped with the air concentrating table of the FGX-1 type, that belongs to the IMBiGS the operating parameters are set separately for each type of tested coal. For this purpose, from the point of view of a given research objective, the optimal values of the following parameters are determined experimentally:

- angles of inclination of the separation deck,
- amount of air supplied to the separation zones,
- frequency of vibration of the separation deck,
- height of the baffle plate in the discharge zone of the rock, middling's and coal.

The researches on coal deshaling that lasted almost a year enabled to gain a rich experience in conducting separation processes and to develop a research methodology for obtaining optimal separation for the particular fractions of raw coal from the Polish coal mines. So far, the studies were conducted on samples from the several coal mines. Some of the results are presented below.

## 4. STUDIES AIMED AT OBTAINING POSSIBLY CLEAN COAL PRODUCTS

Experience gained by IMBiGS shows that it is possible to produce clean coal concentrates that can be considered ecological friendly fuels. The results of such studies are shown in Table 1. The table does not take into account the amount of separated dust product, whose yield in some studies was as high as 3% to 8%. As a result of air separation, very clean concentrates were obtained with a low ash content of about 20%, as well as middlings with a high content of coal particles. Depending on the ash content, these middlings can be saleable products for utility power stations or can be further beneficiated using air concentrating tables operating in parallel. Very clean concentrates, known as ecological friendly fuels, are in high demand for heating houses equipped with appropriate furnaces.

**Table 1 Test results for the raw steam coal of granulation of 20-0 mm**

| 20 – 0 mm raw steam coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 31.7 | **21.2** | 29.5 | 80.5 |
| 2. | Total moisture [%] | 9.6 | **9.0** | 8.4 | 5.3 |
| 3. | Sulphur content [%] | 0.56 | **0.62** | 0.68 | 0.44 |
| 4. | Net calorific value [kJ/kg] | 15,412 | **21,558** | 18,885 | 3,498 |
| 5. | Yield [%] | 100 | **77** | 3 | 18 |

## 5. STUDIES AIMED AT OBTAINING PRODUCTS WITH HIGH CALORIFIC VALUE

Some raw coal produced in Poland within particle size of 20-0 mm have a relatively low net calorific value, which ranges from 16 to 19 MJ/kg. Currently, the energy industry is interested in coals with calorific values in excess of 22 MJ/kg. The studies performed by IMBiGS using the FGX -1 installation showed that the removal of even small quantities of stone raises the calorific value to levels interesting for Polish power stations and combined heat and power stations. Table 2 shows selected results of the studies aimed to increase the calorific value of raw coal within the particle size 20-0 mm. The table does not include the dust product separated in the FGX-1 installation.

**Table 2 Test results for the raw steam coal of granulation of 20-0 mm**

| 20 – 0 mm raw steam coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 30.7 | 24.8 | 38.6 | 85.7 |
| 2. | Total moisture [%] | 10.3 | 5.0 | 4.4 | 2.2 |
| 3. | Sulphur content [%] | 0.65 | 0.62 | 0.61 | 1.28 |
| 4. | Net calorific value [kJ/kg] | **17,596** | **22,383** | 17,459 | 1,307 |
| 5. | Yield [%] | 100 | 76 | 15 | 7 |

## 6. STUDYING THE POSSIBILITY OF REMOVING SULPHUR FROM RAW COAL

Polish hard coal deposits in eastern part of the Upper Silesia Basin are characterised by high sulphur content. In Poland, sulphur levels above 1.2% are considered high, but there are also coals where the sulphur content exceeds 4%. There even exists a seam where coal contains more than 12% of sulphur. Such coals are obviously not mined. In such coals sulphur occurs mainly in the form of pyrite. As its

density is greater than that of coal, pyrite can be removed from coal using gravity-based methods. IMBiGS studied the possibility of removing pyrite with the FGX-1 air concentrating table. For this purpose the parameters of the installation were adjusted so as to separate pyrite most efficiently.
Table 3 shows the results of the study aimed to reduce sulphur content of raw coal from one of the Polish mines with sulphur content of more than 1.8% (without taking into account the dust product).

**Table 3 Test results for reduce sulphur in the raw steam coal of granulation of 25-6 mm**

| 25 – 6 mm raw steam coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 10.5 | 5.5 | 10.5 | 23.5 |
| 2. | Total moisture [%] | 17.9 | 18.2 | 17.8 | 14.2 |
| 3. | Sulphur content [%] | **1.90** | **0.86** | 1.70 | 5.88 |
| 4. | Net calorific value [kJ/kg] | 21,488 | 23,074 | 22,424 | 18,038 |
| 5. | Yield [%] | 100 | 20.6 | 67.7 | 9.7 |

## 7. STUDIES AIMED AT OBTAINING LOW COAL CONTENT REFUSE

According to the information received from the producer the FGX air concentrating tables make it possible to separate from raw coal refuse products with low coal content, which can be used in place of natural aggregate in engineering and construction work. The possibility of obtaining such that products was studied by IMBiGS. Table 4 presents selected results of this study (without taking into account the dust product). The process was considered effective if ash levels in the refuse products exceeded 80%.

**Table 4 Test results for obtain low coal content refuse from the raw steam coal of granulation of 25-8 mm**

| 25 – 8 mm raw steam coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 35.9 | 28.0 | 65.8 | **86.1** |
| 2. | Total moisture [%] | 8.7 | 6.9 | 3.8 | **2.3** |
| 3. | Sulphur content [%] | 0.55 | 0.6 | 0.33 | **0.39** |
| 4. | Net calorific value [kJ/kg] | 16,291 | 20,914 | 5,494 | **948** |
| 5. | Yield [%] | 100 | 81.4 | 1.6 | **14.0** |

## 8. STUDIES ON DESHALING OF COKING COAL

Coking coal users demand commercial products with ash content not greater than 7–7.5%. Such a product is needed to obtain high quality coke. Note that Poland is currently one of the largest coke exporters and this coke must comply with the most stringent quality standards. Obviously, with the use of dry separation methods it is not possible to produce concentrates with ash content below 7.5% without considerable loss of coal in the middlings. Consequently, this method cannot replace wet processes of coking coal beneficiation in dense media, jigs or flotation machines. These processes are very expensive. To reduce the overall cost of coking coal production, IMBiGS undertook studies to examine the possibility of removing some part of refuse material particles from coking coal. The removal of part of the refuse material will reduce the load on the jigs and flotation machines, thus increasing their efficiency.

Tables 5 to 7 show the results of coking coal deshaling. The results do not take into account the dust product in the balance of materials. The results demonstrated that it is possible to use the dry coal separation in parallel with a wet beneficiation line.

**Table 5 Test results of raw coking coal of granulation of 50-25 mm**

| 50 – 25 mm raw coking coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 38.4 | 19.9 | 33.3 | 65.0 |
| 2. | Total moisture [%] | 4.0 | 1.7 | 2.6 | 2.0 |
| 3. | Sulphur content [%] | 0.53 | 0.62 | 0.62 | 0.38 |
| 4. | Net calorific value [kJ/kg] | **19,260** | **26,076** | 21,336 | 9,251 |
| 5. | Yield [%] | 100 | 49.2 | 13.2 | 37.1 |

**Table 6 Test results for the raw coking coal of granulation of 25-6 mm**

| 25 – 6 mm raw coking coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 28.15 | 19.8 | 56.6 | 82.2 |
| 2. | Total moisture [%] | 5.8 | 4.8 | 2.2 | 1.6 |
| 3. | Sulphur content [%] | 0.64 | 0.63 | 0.50 | 0.87 |
| 4. | Net calorific value [kJ/kg] | **22,761** | **25,847** | 12,052 | 3,004 |
| 5. | Yield [%] | 100 | 83.0 | 8.4 | 8.4 |

**Table 7 Test results for the raw coking coal of granulation of 25-0 mm**

| 25 – 0 mm raw coking coal | | | | | |
|---|---|---|---|---|---|
| **No.** | **Property** | **Feed** | **Coal** | **Middlings** | **Refuse** |
| 1. | Ash content [%] | 18.4 | 12.2 | 27.0 | 75.6 |
| 2. | Total moisture [%] | 7.8 | 7.0 | 3.3 | 2.2 |
| 3. | Sulphur content [%] | 0.65 | 0.66 | 0.63 | 0.57 |
| 4. | Net calorific value [kJ/kg] | **25,538** | **27,991** | 23,668 | 4,921 |
| 5. | Yield [%] | 100 | 77.0 | 16.8 | 5.8 |

## 9. CONCLUSION

The study, which has been in progress for more than a year and a half, on using the air concentrating table in Poland revealed the possibility of using this technology and its advantages. IMBiGS researchers have published the results of this study in Polish scientific journals and presented them at conferences in Slovakia and the Czech Republic (Baic el al. 2014, 2015 [4-13].

The process of dry coal separation with the use of air concentrating tables has generated great interest of the coal industry. As a result of studies on coal from the mines interested in this process, the purchase of industrial-scale FGX table is currently being considered. The interest in Chinese technology resulted in a research trip to China, which was organised in April 2014. Representatives of eight Polish mining companies participated in the trip to the factory of Tangsan Shenzou Machinery Co. Ltd and to several Chinese mines where this particular machine is used. The trip provided Polish engineers with more knowledge of dry coal separation of raw coal. The goal of the studies currently conducted by IMBiGS is to justify the need to buy the FGX air concentrating tables.

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# NOVEL DRY SORTER FOR COAL PROCESSING AND COAL RECOVERY FROM MINE ORIGINATING WASTES

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Abstract:

Coal excavated from a mine contains an average of 30% of polluting materials. To become a commercial product with the required quality parameters, this coal must be subjected to cleaning processes in a coal mechanical processing facility. In Poland hard coal processing and its recovery from mining waste is conducted on industrial scale, applying chiefly the gravity separation methods in a dense liquid or water medium. The coal processing in water medium raises many problems i.e. high energy consumption, necessity to apply dewatering processes, the maintenance of the water-sludge cycle, high processing costs. Thus, in the recent years, there is a growing demand for "dry" cleaning processes in air medium. Currently in the world, new constructions of this type of devices such as the FGX air concentrating table, sorter TOMRA, OSX can be found. Another type of machine used in air coal separation is a pneumatic sorter, constructed in the frames of the KIC project: "*Novel dry sorter for coal processing and coal recovery from mine originating wastes"* - AMSEP. A coal separation installation for coal processing as well as for recovery of coal from coal waste and the production of qualified coal fuels and mineral aggregates will be applied.

Key words:
coal, coal processing, "dry" cleaning processes, coal separation, sorter, pneumatic sorting, coal waste

## 1. Introduction

Coal excavated from a mine contains an average of 30 % of polluting materials in a form of gangue. To become a commercial product with the required quality parameters, this coal must be subjected to cleaning processes in a coal mechanical processing facility. Cleaning processes in industrial conditions usually employ the method of gravity separation in heavy liquid as well as in a water and air medium (Blaschke 1976, 2009.; Mishra S. K., Klimpel R., 1987; Baic, Blaschke 2013). Gravity separation methods are also used for the recovery of coal from mining waste deposited in the natural environment. In Poland coal with a particle size distribution of 200-20mm is most often cleaned in a dense liquid washers, while more fine-grained coal with a particle size distribution of 20-0.5 mm (culm) is most often cleaned in fines jiggers, in water medium. In the case of some of the mines, the finest-grained coals, mostly coke coal with a particle size distribution of 0.5-0 mm are cleaned through flotation (Baic, Góralczyk 2009). The coal processing in water medium raises many problems i.e. high energy consumption, necessity to apply dewatering processes, the maintenance of the water-sludge cycle, high processing costs. The primary downside of water-based cleaning processes is the necessity to maintain the water-slurry balance and to mechanically or thermally drain the separated products. The draining processes for large quantities of products are difficult to conduct and very expensive. For that reason effective "dry" methods of cleaning in air

 
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medium have been investigated. These methods have been known for more than 100 years. However, the machines used in the interwar period were not very precise, which caused a high pollution of the produced coal concentrates and big losses of coal in the form of waste. Currently new designs of this type of machinery are appearing all over the world, such as FGX concentration tables (Baic, Blaschke 2013, Baic i in. 2014a,b, Baic i in. 2015), optical sorters, e.g. TOMRA, OSX sorting system (http://www.comex-group.com/Comex/system_sortujacy_serii_osx.html), or Bradford crushers, used for initial stone washing. In some countries (China, the USA, the Republic of South Africa) these machines are used to initially prepare the feed for the cleaning processes (Zhenfu, Qingru 2001; Haibin at all 2011; Yang at all, 2013).

Another type of machine used in aerial coal separation is a pneumatic sorter, built as part of the KIC InnoEnergy project entitled "*Novel dry sorter for coal processing and coal recovery from mine originating wastes"* (acronym AMSEP), which allows for the separation of material mixtures differing in density by 0.1 g/cm$^3$ (patent RU 22822503C1). The plant consists of a separating machine, serving as a separator and auxiliary equipment, such as: a crusher, classifying sifters, belt conveyors transporting the material into the plant and belt conveyors transporting the processed materials to the storage yard. The principle of the plant operation is based on a multi-step use of aerial separation of grains placed on a moving sieving belt. The principle has been shown in Figure 1.

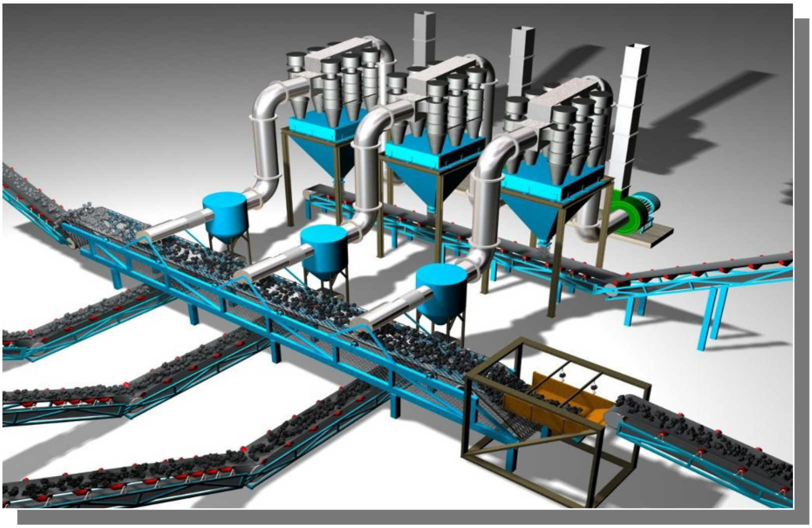

Fig. 1. The design of an AMSEP pneumatic sorter
(developed by the project partner – Company ASM)

The pneumatic plant is designed to clean the run-of-mine coal excavated from the underground mine in a mechanical processing facility as well as for recovering the coal from coal waste and the production of qualified coal fuels and mineral aggregates.

## 2. Preliminary laboratory tests of pneumatic sorting

In order to establish the concept of pneumatic coal cleaning, laboratory tests have been conducted to check the possibility of using this process in coal grain separation. The scope of the tests conducted using a laboratory sorter based on vacuum grain sorting by means of nozzles included:

- establishing the methodology for the pneumatic sorting process testing,

- designing and building a laboratory workstation for testing the pneumatic sorting process,
- preparing the (artificial) reference mixes containing coal grains of specific grain classes,
- preparing the natural material (run-of-mine coal) of specific grain classes,
- conducting sorting tests for reference and natural materials,
- separation process efficiency assessment,
- process optimisation – determining the sorting efficiency and product purity for the optimised material (material with the most favourable parameters).

The research itself was preceded by the design and construction of the testing workstation. The essential elements of the laboratory set for pneumatic sorting were:

- the suction fan,
- the suction nozzle,
- the bag filter.

The setup consisting of those three parts allowed us to control the position of the suction nozzle both vertically and horizontally. The layout of the testing workstation has been shown in Figure 2.

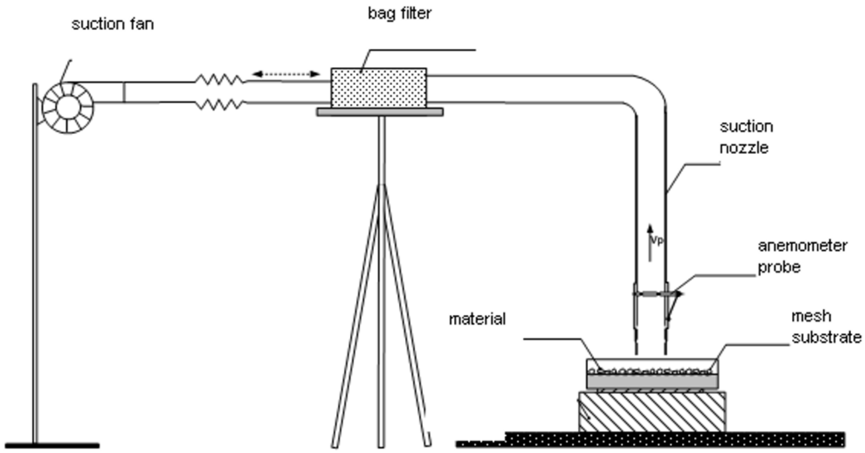


Fig. 2. Testing workstation layout

The testing workstation allowed for conducting the process for feed heap on mesh bases. The laboratory set allowed us to produce a suction force caused by the air flowing through with a speed of 19 m/s as measured in the tube, at the end of the nozzle.

**2.1. Preparing the feed for the sorting process**

The testing feed consisted of:

- a mix of grains of materials of significantly different density: coal concentrate from a Polish coal mine and quartz gravel, in a 1:1 ratio.
- a mix of grains of coal concentrate from a Polish coal mine and shale grains, in a 1:1 ratio.
- run-of-mine coal from a Polish coal mine.

Each reference material was prepared in a quantity of 140g. The research material was prepared in the following grain classes: 6,3-5 mm, 8-6,3 mm, 10-8 mm, by sieving it on a vibration sieve. Pneumatic sorting tests were conducted for each of the aforementioned grain classes separately. The tests were conducted for materials in air-dry state.

**2.2. Pneumatic sorting**

Pneumatic sorting consisted in sucking the concentrate from the feed heap and evaluating its efficiency and purity. It consisted in moving the nozzle above the feed layer according to the pre-set parameters of the sorter. Pneumatic sorting was conducted for all the materials in the same conditions of suction nozzle airflow speed *Vp*. The changing parameter was the height of the nozzle above the mesh base.

The reference materials were manually divided into separate components of each mixture (based on the difference in colour) and weighed. For the natural materials, the percentage of the coal fraction in the product was measured using the densimetric method.

## 3. Research results

Table 1. as well as Figures 3a,b. the results of tests conducted for reference mixtures. The "efficiency of the pneumatic sorting process" has been defined as a percentage ratio of the mass of coal concentrate sucked in during the test to the mass of coal concentrate left on the mesh base.

Table 1. Results of the pneumatic sorting test for the reference mixtures

| Material | Size grade [mm] | Yield of the sucked-in product [%] | Yield of product on the substrate mesh [%] | Efficiency of the pneumatic sorting process [%] | Concentrate purity [%] |
|---|---|---|---|---|---|
| 50% quartz (70g) 50% coal (70g) | 6,3-5 | 38,31 | 61,69 | 76,6 | 96.9 |
| 50% quartz (70g) 50% coal (70g) | 8-6,3 | 45,18 | 54,82 | 90,3 | 92,4 |
| 50% quartz (70g) 50% coal (70g) | 10-8 | 34,21 | 65,79 | 68,4 | 89,4 |
| 50% shale (70g) 50% coal (70g) | 6,3-5 | 57,60 | 42,40 | 82,3 | 83,1 |
| 50% shale (70g) 50% coal (70g) | 8-6,3 | 45,71 | 54,29 | 91,4 | 78,0 |
| 50% shale (70g) 50% coal (70g) | 10-8 | 37,42 | 62,58 | 74,8 | 83,9 |

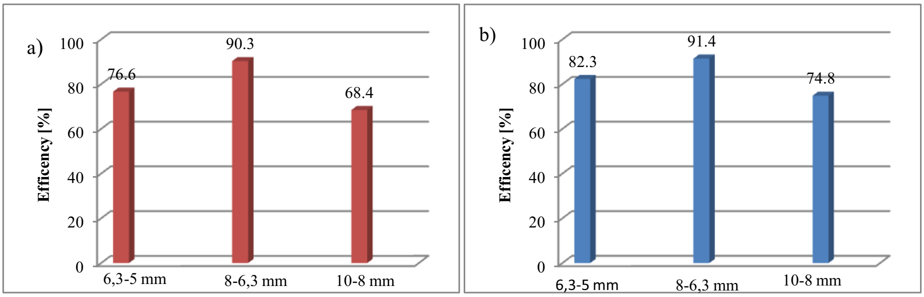


Fig 3. The pneumatic sorting efficiency profile for the reference material sample for the 5 - 6.3mm, 6.3 – 8mm and 8 – 10mm fractions a) coal+ quartz, b) coal+ shale

Table 2 contains the efficiency test results of the run-of-mine coal pneumatic separation conducted in air-dry conditions. The "efficiency of the pneumatic sorting process" has been defined as a percentage ratio of the mass of coal concentrate sucked in during the test to the mass of the coal concentrate left on the mesh base while “concentrate purity” has been defined as the percentage ratio of the mass of coal concentrate present in the sucked-in product to the total mass of the sucked-in product.

Table 2. The results of densimetric tests of run-of-mine coal pneumatic separation

| **Lp.** | **Sample weight [g]** | **Yield of the sucked-in product [%]** | **Yield of product on the substrate mesh [%]** | **Efficiency of the pneumatic sorting process [%]** | **Concentrate purity [%]** |
|---|---|---|---|---|---|
| 1 | 433.3 | 87,45 | 12,55 | 99.6 | 92.5 |
| 2 | 415.4 | 83,42 | 16.58 | 98.3 | 96.0 |
| 3 | 428.1 | 84,71 | 15,29 | 98.2 | 89.5 |

## 4. Summary and conclusions

The preliminary laboratory investigations into the possibility of implementing the pneumatic sorting process in coal cleaning allow us to formulate the following conclusions:

- The pneumatic sorting process can be used to efficiently separate the coal substance from the stone substance. Tests conducted on reference materials, i.e. coal-quartz and coal-shale have indicated that as a result of optimisation of the process conducted in static conditions, it is possible to achieve a high efficiency of over 90% as well as a high purity of the product, reaching more than 90%. Similar values have been obtained for the product purity parameter.
- High sorting efficiency and purity coefficients have been achieved for all three separate grain fractions from the 5 – 10mm range.
- The pneumatic sorting process in laboratory conditions conducted for run-of-mine coal was also characterised by high efficiency and purity of the final product. As a result of optimisation, a peak efficiency of 99% and peak product purity of 96% were achieved. Based on the conducted tests, we ought to conclude that in the optimisation of the process in laboratory conditions, the following factors are of particular importance:
  - conducting the process in a monolayer of the feed on a mesh base,
  - determining, based on empirical data, the right geometry of the suck-in setup, in particular, the height at which the nozzle is to be located above the mesh base for a given suction force.
  - proper preparation of the feed for the sorting process (isolating narrow grain classes and the sorted material dampness). Any prospective large-scale laboratory research should include investigations into the efficiency of sorting, depending on the method of feed preparation (the way it is divided into fractions) as well as its degree of dryness.
- The conducted laboratory tests provide a basis for further research aimed at developing a “dry” coal cleaning technology using a pneumatic sorter, which shall be continued as part of the KIC InnoEnergy project entitled *“Novel dry sorter for coal processing and coal recovery from mine originating wastes”* – AMSEP.

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# *INVESTIGATION OF DRY COAL BENEFICIATION WITH OPTICAL SORTER*

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**Abstract:**

There are significant environmental and operational advantages when coal is subjected to the separation process right after it is extracted from earth. Hence, run-of-mine coal is usually put to the HMS (heavy medium separation) after size classification if necessary. Use of HMS in coal beneficiation has some known operating difficulties resulting from water and magnetite requirement, waste formation, dewatering, and climate challenges. Additionally, HMS plants cannot be constructed in underground. Automated optical sorting of coal is a competing technology with HMS when the colour difference between coal and its impurities is visible. Therefore, optical sorting depending on colour differences between minerals can be introduced to coal beneficiation systems.

In this study, lignite samples obtained from different regions of Turkey was sorted with color sensor mounted optical sorter and results were evaluated. Laboratory studies included selection of the material, sample preparation, heavy liquid tests, optical sorting, optical sorting plant simulations, and image acquisition of products. Results showed that there was a remarkable difference between sorting products under distinct conditions. The experimental studies under specific conditions also suggested that lignite sorting with visible light sensor could be evaluated as an alternative to the dry coal cleaning applications.

**Keywords:** Optical Sorting, Sensor Based Sorting, Visible Light Sensor, Coal, Lignite, ROC, Sorting Performance

## Introduction

Conventional wet coal cleaning is predominantly performed with jigging and HMS (heavy medium separation) all over the world (Wills, 2006). Inefficiencies due to insufficient density difference between contents, chemical breakdown of contaminants, environmental concerns, and difficulties in operation may lead to the use of a dry method for coal washing (Houwelingen, 2004). Hand sorting, dry jigging, FGX (air tables) (Young, 2012), berrisford separators, XRT (X-Ray Transmission) sorters (Kleine, 2010), and dry fluidized bed separators are some of the dry separating methods used in dry coal cleaning. Automated optical sorting of coal is a competing technology with HMS when the color difference between coal and its impurities is visible. Preferring ore sorting has significant benefits over conventional concentration methods (Salter and Wyatt, 1991). An optical sorting pre-concentration plant reduces the head-feed, accordingly, size of plant is reduced and certain process streams for more efficient valuable mineral recovery are increased. The purpose of the optical sorting operation is usually to reject barren waste or to separate the ore into high-grade and low-grade components in terms of a more general expression. Hence, overall efficiency of mining, milling and processing operations can be improved considerably by using of automated optical sorting (Lessard J., 2014).

Color sensors detect the visible light spectrum (400-750 nm wavelength). Thereby, visible difference between subjected particles which can be detected with naked eye is a strong indication for successfully implementing the color sorter. Previous studies performed with various lignites showed that optical sorting can efficiently achieve a significant separation with lignite depending on the color differences between high ash bearing particles and low ash bearing coal grains. Exemplarily, lignite from Ilgın-Turkey region having a low degree of alteration totally tended to float in heavy medium at low densities. Therefore cleaning of this lignite with conventional methods was not possible (Gülcan, 2014). But



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depending on the color differences, ash in feed could be removed by obtaining a relatively clean concentrate with color sensor mounted optical sorter. Beside the suitability of Ilgın lignite to optical sorting, most of the coal samples are not suitable for color sorting because of the very low visual differences among the coal particles, which are mostly dark colored.

Given a sorting operation, there are four possible outcomes. These are target particles reporting in concentrate, target particles reporting in tail, non-target particles reporting in concentrate and non-target particles reporting in tail. A ROC graph visualizes tradeoffs between true positives, false positives, true negatives and false negatives (Fawcett, 2005). Generally, this approach is used in binary classifiers, when particle-based sorting is concerned. Grains are classified as "good (positive grains)" or "bad (negative grains)". For example, an apple is rotten and should be removed from the system, or is fresh. On the other hand, grade is important in terms of mineral processing. Target mineral grain is exceptionally good or bad according to its valuable mineral content. Therefore, the optical separation results should be evaluated on the basis of grade and recovery in addition to ROC curves.

Considering positive fraction as “accept” and negative fraction as “reject”, a threshold (t) of sorting products is defined in Equation 1 and 2 (Ooms, 2010 and Fawcett, 2005)

y ≥ t : Dx  Accept
(1)

y < t : Dx  Reject
(2)

Where $D_x$ is each object data and $y$ indicates the probability of object belonging the target class.

Confusion matrix of results are given in Table 1. These parameters are also used to define the ROC parameters given in Equation 3, 4, and 5.

**Table 1.** Confusion matrix to define ROC parameters

| | Class | |
|---|---|---|
| Accept | True Positives (TP) | False Positives (FP) |
| Reject | False Negatives (FN) | True Negatives (TN) |
| Total | Positives (P) | Negatives (N) |

$$Accuracy = \frac{TP+TN}{P+N} \quad (3)$$

$$Specificity = \frac{TN}{FP+TN} = \frac{TN}{N} = 1 - \frac{FP}{N} = 1 - FPr \quad (4)$$

$$Sensitivity = \frac{TP}{FN+TP} = \frac{TP}{P} = TPr \quad (5)$$

Where *FPr* is false positive rate and *TPr* is true positive rate. The ROC graph is expressed by TPr over FPr in a two dimensional plot. The performance of the discrete classifier is discussed on the ROC area. One certain approach is that it is not possible to maximize both sensitivity and specificity at the same time.

## Optical Sorting Study

### Sorting Equipment

In the sorting equipment, the images of the freely falling particles are taken by a line-scan sensor/camera to be digitalized and analyzed. After a sample preparation by size classification, classified material is fed into the sorter via a vibrating feeder. The images of the particles in the stream are identified by installed image analyzing software separately and decision is made according to predefined parameters whether to blow the particles out by the air valves in the suitable position (rejected) or leave them to follow their own natural orbital (accepted). Rejecting the light colored rocks from feed stream is the most preferential operation type (Mular, 2002).

The schematic view of the gravity type single line scan COLOR sensor mounted optical sorter used in this study is seen in Figure 1.

The main separation parameter, which is the reflected amount of the radiation, is detected by the sensor/camera. Within the application range of the sensor and light source, each mineral’s own spectrum is detected and analyzed. Hence the separation is provided according to the sensed differences among the images. When using a COLOR camera, images are the digitalized versions of the visible light spectrums.

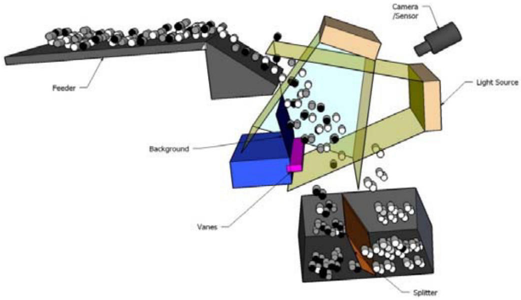



**Figure 1.** Schematic view of the optical sorter

## Seyitömer coal optical sorting tests

Seyitömer coal provided by Turkish Coal Enterprises was chosen due to color differences between the individual particles of the slightly dry run of mine coal. There are two hard clay layers which has light color between the coal seams in this region. But sometimes the color of these layers is intertwining with the coal and the recognition of the differences between coal and clay particles is getting very difficult. A representative sample having approximately 50-55 % ash and 2500-3000 Kcal/kg heating value was screened to narrow size fractions. Characteristics of the prepared lignite sample is as seen in Table 2.

Test results obtained for each size fraction and images of sorted product and discarded waste after optical sorting were summarized in Figure 2. When Seyitömer optical separation test products were examined, it was seen quite clearly that visible light sensor detected even delicate color variations that allow separation. Combined accepted and rejected products of sorted fractions had ash contents of 38.53 % and 74.96 %, respectively. Also 54.50 % weight of total sorted sample was collected as accept.

**Table 2.** Sample properties

| Particle Size, mm | -150+100 | -100+75 | 75+50 | -50+25 | -25 | Feed |
|---|---|---|---|---|---|---|
| Weight, % | 19.76 | 21.08 | 18.4 | 20.14 | 20.63 | 100 |
| Dry Ash, % | 60.61 | 62.65 | 50.55 | 44.59 | 44.33 | 52.6 |
| LHV*, Kkal/kg | 1920 | 1931 | 2679 | 3135 | 3072 | 2544 |

## Ilgın coal optical sorting tests

Prepared sample pile of Ilgın coal having approximately 45-46 % ash and 2700-3000 Kcal/kg heating value was screened to narrow size fractions. After a sample preparation by size classification, the classified feed materials except -9.5 mm which was too fine for an efficient separation were weighed and analyzed in order to feed to the optical sorter. Particle size fractions, weights, ash and calorific values of Ilgın coal is as seen in Table 3. Each size fraction of Ilgın coal with known weights was fed to the optical sorter at an optimum feed rate which was chosen with respect to particle size. Test results obtained for each size fraction and images of sorted product and discarded waste after optical sorting were summarized in Figure 3.

**Table 3.** Properties of Optical Sorter Feed

| Particle size (mm) | -150+100 | -100+75 | 75+50 | -50+38 | -38+19 | -19+9.5 | -9.5 | Feed |
|---|---|---|---|---|---|---|---|---|
| Weight (%) | 16.84 | 16.66 | 21.52 | 10.17 | 20.4 | 6.88 | 7.53 | 100 |
| Dry Ash (%) | 47.24 | 47.53 | 46.32 | 44.64 | 43.9 | 49.73 | 55.17 | 46.91 |
| LHV (Kcal/kg) | 2720 | 2697 | 2791 | 2923 | 2980 | 2525 | 2102 | 2745 |

As seen in Figure 3, images of the concentrates and tails were differing in shades of grey. While colour difference was strongly noticed at coarse particle sizes, it became hardly noticeable when particle size was -19.5+9.5 mm. But the differences between concentrate ash contents indicate the separation performance for these fractions.

## Results and Discussions

ROC graphs for each sorting operation is plotted as tp rate versus fp rate are given in Figure 4. Any sorter on or close to inefficient classifier traverse performs random guessing. In other words, at (0.5;0.5) point sorter sorts half of the positives and half of the negatives correctly. It acts like a splitter. If separation hints at inverted classifier zone, true positives (TP) and false positives (FP) become false negatives and true negatives, respectively. Differing coal samples are showing specific characteristics in ROC space and discussed together. Data obtained from tests with these samples is quite far away from inefficient separation boundary.

| Feed | | Products | | | |
|---|---|---|---|---|---|
| Particle size (mm) | Dry Ash (%) | Operation | Weight (%) | Dry Ash (%) | LHV* (Kcal/kg) |
| -150+100 | 60.61 | Accepted | 41.45 | 40.86 | 3377 |
| | | Rejected | 58.55 | 74.59 | 888 |
| -100+75 | 62.65 | Accepted | 52.25 | 44.28 | 3240 |
| | | Rejected | 47.75 | 82.76 | 499 |
| -75+50 | 50.55 | Accepted | 63.26 | 35.51 | 3772 |
| | | Rejected | 36.74 | 76.44 | 799 |
| -50+25 | 44.59 | Accepted | 61.64 | 34.71 | 3791 |
| | | Rejected | 38.36 | 60.45 | 2080 |

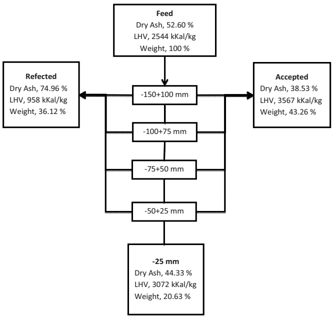
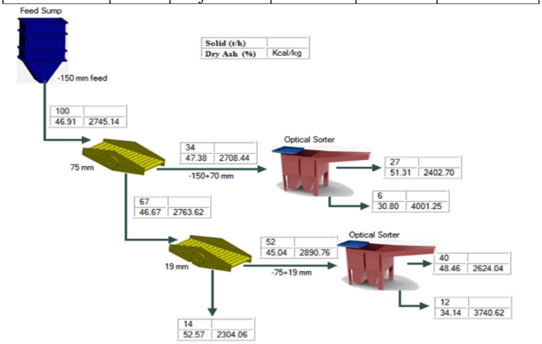



**Figure 2.** Seyitömer coal optical sorting test results (*Left:* Color sorting test results on fractional and combined basis. *Right:* Images of the obtained accept and reject products after sorting)

Classifications appearing on the left-hand side of an ROC graph make positive classifications only with strong evidence so they make few false positive errors, but they often have low true positive rates as well. Classifiers on the upper right-hand side of an ROC graph may be thought of as ‘‘liberal’’: they make positive classifications with weak evidence so they classify nearly all positives correctly, but they often have high false positive rates. Starting from that point, sorting -150+100 mm of Seyitömer coal is more conservative than -100+75 mm. Similarly, -100+75 mm shows more positive classifications with few false positive errors than -75+50 mm Seyitömer coal. This is a result of the increasing liberation in finer lumps. Informally, -50+25 mm sorting results has the lowest tp rate value. For understanding this behavior, which is valid for most operations, geometric and operational characteristics of the sorter should be considered. In descrete sorters, liberation is a key factor for an efficient separation. Recovery is expected to increase with increasing liberation/decreasing particle size. However, upper and lower particle size limits of the sorter due to mechanical limitations and spectral sensitivity range of the sensor also decrease the recovery. Also, the increasing distance between air valves reduces the possibility of accurate air blow to fine particles. The distance between air valves of available optical sorter is designed as 10 mm, which is critical particle size. Although Ilgın coal show similar sorting characteristics with Seyitömer coal in terms of particle size differentiation, overall trend shifts to the inefficient classification zone due to the coal characteristics.

To conclude, Seyitömer coal exceptionally is more suitable for sorting in terms of true and false alarm rates of the sorting equipment. Resulting with the sharp color differences between ash forming and coal particles in Seyitömer coal, less mistakes were made during selective sorting. Ilgın coal has incomplete coal characteristics and it is at early stages of carbonization. Thus, color of some particles are not directly related to ash content, resulting with false alarms. On the other hand, when the ash removal efficiencies of both products are taken into account (Figure 5), sorting of Ilgın coal is clearly removed more ash from the feed due to the mass pull during sorting opertions.

| Feed | | Products | | | |
|---|---|---|---|---|---|
| Particle size (mm) | Dry Ash (%) | Operation | Weight (%) | Dry Ash (%) | LHV (Kcal/kg) |
| -150+100 | 47.24 | Accepted | 36.88 | 27.89 | 4228 |
| | | Rejected | 63.12 | 58.54 | 1839 |
| -100+75 | 47.53 | Accepted | 39.64 | 33.53 | 3788 |
| | | Rejected | 60.36 | 56.73 | 1980 |
| -75+50 | 46.32 | Accepted | 39.04 | 32.03 | 3905 |
| | | Rejected | 60.96 | 55.47 | 2078 |
| -50+38 | 44.64 | Accepted | 48.03 | 33.15 | 3818 |
| | | Rejected | 51.97 | 55.25 | 2095 |
| -38+19 | | Accepted | 56.86 | 36.08 | 3589 |
| | 43.90 | Rejected | 43.14 | 54.20 | 2177 |
| -19+9.5 | | Accepted | 43.68 | 43.41 | 3018 |
| | 49.73 | Rejected | 56.32 | 54.64 | 2143 |




*1: -150+100 mm concentrate, 2: -150+100 mm tail, 3: -100+75 mm concentrate, 4: -100+75 mm tail, 5: -75+50 mm concentrate, 6: -75+50 mm tail, 7: -50+38 mm concentrate, 8: -50+38 mm tail, 9: -38+19.5 mm concentrate, 10: -38+19.5 mm tail, 11: -19.5+9.5 mm concentrate, 12: -19.5+9.5 mm tail*

**Figure 3.** Ilgın Coal optical sorting test results (*Left:* Color sorting test results on fractional and combined basis. *Right:* Images of the obtained accept and reject products after sorting)

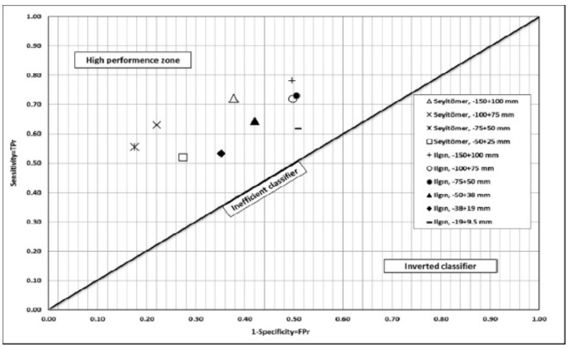



**Figure 4.** ROC graph of magnesite and quartz samples

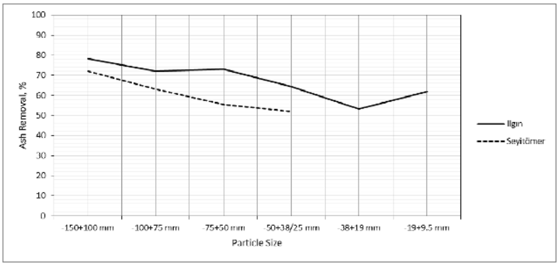



**Figure 5.** Ash removal performances of sorting operations

Ash removal indicates the percent ash which is removed from the feed to obtain a low ash bearing product. For instance, when -150+100 mm size fraction of Ilgın coal was fed to the optical sorter, %78.23 of the ash in feed could be removed by obtaining a concentrate having 27.89 % dry ash. More ash is removed from the feed when particle size is coarser. In other words, coal separation efficiency of the optical sorter increased when coarse particle size fractions were fed.

## Conclusion

Test results showed that a successful sorting operation can be conducted with Ilgın region's low rank coal (which does not respond to heavy medium separation because of the insufficient density difference between contaminants) and Seyitömer coal (which is problematic when cleaned with HMS). On the other hand, optical sorting can efficiently achieve a significant separation with Ilgın and Seyitömer coal depending on the color differences between ash bearing content and coal. Results underline that since there is a color difference between coal and its impurities and common wet or dry cleaning methods are not applicable or feasible, optical sorting is a good option for coal cleaning. Particularly, it is liable to perform an optical sorting efficiently in a plant operation when certain conditions are met.

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# ArdeeSort – Next Generation Coal Dry Beneficiation Technology

**By**
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*ABSTRACT:*

***Coal is the cheapest energy source. However, it has an image issue of being a dirty fuel, which needs to be addressed. While coal washing to bring down ash levels is being taken up in most parts of the world, washing per se, pitches the industry against rest of society in water-challenged communities, especially against the farming sector. Further, coal is seen as a conservative industry which is not sensitive to societal perceptions of its impact on society or environment and keeps doing business-as-usual. This perception needs to be reversed or there is likelihood of the industry facing punitive measures which will make survival difficult. Out-of-the-box solutions are required to improve coal quality in an enivornment-friendly manner.***

***Dry beneficiation of coal, is one such small step which is immediately achievable. Within the family of dry beneficiation methods, recent technologies like ArdeeSort have emerged as game changers which apart from improving the techno-economic efficiency of coal beneficiation without use of water, have made clean coal process very affordable.***

***Keywords: Dry Beneficiation; Coal Sorting; ArdeeSort; X-Ray Sorting; Radiometric Sorting; DE-XRT Sorting, Hybrid Washing.***

**Introduction:**

Global warming has caused governments, organizations and people across the globe to get their clean up act together rather than just talk. It is important for the coal industry to fall in line with this movement and contribute to it since all members present here have stakes in the preservation of environment. Coal is seen as a dirty fuel and a prime contributor to emissions. Western countries are shifting from coal to other energy forms such as shale gas, oil, solar and wind power. However, developing countries like China and India, despite pressures to the contrary, will continue to largely depend on coal which is the cheapest source of energy. Since economic development is still a primary objective, use of coal cannot be prohibited or discouraged. At the same time, there cannot be a blatant neglect of the environment. The message is clear – protection of environment and the economy have to go hand in hand – it is not possible to divorce one from the other.

One way of reducing emissions from coal is to upgrade its quality and ensure 100% beneficiation of coal prior to use. Different coals have varied washability characteristics and based on the target ash% in clean coals, the yield and process adopted may vary. However, once better coal is used in power generation, it leads to an absolute reduction in quantum of coal required per unit of power generated. Second, it reduces carbon dioxide and flyash emissions.

Beneficiation of coal through conventional wet technologies requires addressing the issue of water requirement. About 160 litres of water per tonne of coal input is required even with zero effluent discharge systems. Water is a scarce resource and most washery projects find themselves in direct competition and conflict with local communities, especially farmers. This is why coal dry beneficiation has emerged as a key focus area in coal preparation.

**Dry Beneficiation of Coal – Outline of Different Technologies:**

Dry beneficiation encompasses a number of methods such as Rotary Breaker, Accelerator, Air Jig, Air Dense Media, Colour Sorting, Radiometric (Gamma ray) and Dual Energy X-Ray Transmission (DE-XRT) Sorting. While the first four are bulk processing techniques, the latter two are selective methods

 
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based on individual particle analysis and ejection. The choice of techniques largely depends on the type of coal being handled.

In bulk coal processing, a mass stream containing a mix of coal and contaminants is subjected to a mechanical process with the objective to force a separation into two different and distinct streams – cleans and rejects. This technique is largely valid if there are sufficient density or friability differentials between usable coal and contaminants. However, when dealing with lower quality coal having a preponderance of near gravity material, there is likelihood of increased cross-migration or misplacements of coal into rejects and vice versa which puts these systems into the technically sub-optimal zone. Certain techniques like Air Jig or Air Dense Media are economical to invest and operate. However, their efficiencies are not high and are not suitable for coals where total moisture exceeds 12%. This is where sorting has emerged as a superior option in dry beneficiation range of techniques.

Colour sorting or optical sorting can be done using cameras, photodiodes and near infra-red sensors. It is successful with grains and certain ores where there is a clear cut demarcation between what is acceptable and what is not. For example, the most prevalent form of colour sorting is grain sorting where black spots, chalk white, brown or red colour in normal white rice is considered a contaminant and has to be ejected. In Near Infra Red (NIR) sorting techniques, the reflection of the NIR beam after striking the surface of the object bears the signature of the material composition of the object. It may be used to determine the type of plastic, say PVC or PET, or the percentage of magnesium oxide or silicon dioxide contamination in limestone. The system is programmed to trigger a signal for acceptance or rejection of the particle if the contamination level crosses a threshold value.

The same principle cannot be directly applied to most coals. First, surface characteristics do not determine the quality of coal. Second, colour differential between coal and its contaminants can be low. While shale is grey in colour and stone can be anything from white to red to black, the colour of coal can also vary depending on the degree of contamination. During rainy seasons, all the particles look alike. In some mines, there are clear observable differences between coal, shale and stones and some employ manual labour to pick shale or stone from a stockpile or a slow moving conveyor belt. The effectiveness or utility of manual picking is questionable, but in several cases it is being done to merely demonstrate the intention to improve quality.

Radiometric sorting is the last to arrive in this family of dry beneficiation technologies since the last two decades. It started with gamma ray attenuation techniques combined with ultrasonic height sensor for calculating the apparent density of the particle. Here, mass per unit area of the particle was derived from attenuation of gamma rays which when compensated by average height of particle resulted in its apparent density. Problems of cross-talk of ultrasonic sensors due to beam scatter put the radiometric technique into a realm of relative inefficiency. Ardee replaced ultrasonic with specialized point laser sensors, which unlike normal optics, is capable of measuring heights of black surfaces. This improvement helped substantially enhance accuracy. The main plus point is that it is a low cost technique. Two plants based on this technology were commissioned in 2008 in India – one for lignite and the other for coal – and both plants are working successfully till date.

The basic handicap of the gamma ray-based sorting system was minimum size limitation of + 50 mm while top size could go upto 400 mm. At around this time, several Indian mines decided to restrict the top size of coal dispatch to – 50 mm to improve handling capabilities and reduce en route pilferage. Effectively, this change reduced the potential for Gamma Ray beased radiometric techniques in dry beneficiation. Further, the issue of radiation safety came to the fore in India with a series of unfortunate incidents where radioisotopes were improperly handled by universities and research institutes, causing accidents and panic among general public. This has caused users to be wary of anything involving high levels of radiation.

The Dual Energy X-Ray Transmission technology (DE-XRT) sorting is the latest and most promising dry beneficiation technique for coarse coals (+ 6 mm) where efficiencies can be as high as the best conventional wet practices at a fraction of the cost. This is more in synchrony with market requirement since it works well on coals having < 100 mm size, there is no stray radiation continuously emanating and the energy levels at which these operate are far less than gamma ray systems.

One reason why DE-XRT technology could not be successfully introduced into coal sorting till recently is that these require a huge amount of data processing before accept or reject decisions can be taken. The software and hardware complexity is much more than simple colour sorting or non-coal mineral sorting. Till Ardee took up development of DE-XRT coal sorting technology, ore sorting with DE-XRT was successful only in some non-coal minerals – manganese, iron ore and diamonds. This is because gangue and ores are discrete. Hence, analysis of any part of the particle is likely to give a fairly accurate estimate of its composition. In short, composite particle-wise averaging of quality is not required to distinguish between ore and gangue. Coal is a more complex material to sort and more so Indian coals which have more interspersed bands of shale both in seams and even within the particles. This causes quality of coal to vary from particle to particle in a mass stream and even within a particle there can be substantial variation. Therefore, it is imperative to capture the composite quality of each particle before determining whether it can be accepted or rejected. Most DE-XRT techniques are unable to do this on account of the complexity of the calculations, the limited time frame in which to do it and the appropriate ejection technology which enables a highly accurate, fast and pinpointed ejection to eliminate the contaminant without touching clean coal.

Ardee is the first company to come out with such a solution that encompasses identification of composite particle size, shape and quality measurement, adoption of high speed algorithms to calculate the average organic ratio of the particle and a solenoid-based pneumatic nozzle array technology with a narrow pitch that enables accurate ejection of particles without causing undue misplacement of particles. The initial research and development works were conducted with generous financial and technical assistance from the Department of Scientific & Industrial Research, Ministry of Science & Technology, Government of India. Once the technique was fine-tuned, Ardee started commercialization of the technology. It has already initiated two commercial scale plants in India for lignite and coal which are being commissioned and is exploring options for some more plants.

**How ArdeeSort Dry Beneficiation Helps?**

The use of ArdeeSort dry beneficiation technology – as a standalone or a primary coal preparation system – involves by-passing substantial part of Run-of-Mine (ROM) coal which need not be washed. By introducing dry beneficiation in the process flow, washery size, water and land requirement, capex and opex all can be substantially reduced without compromising on quality or yield. We give examples from the case study of two installations – lignite and coking coal respectively – that illustrates the feasibility of using this technology in an intelligent fashion to help save capital, land space, water, power, operating costs and reduce the adverse impact on the environment. Further, since no water is used in the process, there is no requirement for dewatering and drying.

**Case Study 1 – GMDC's Lignite Mine at Surka North, Gujarat, India**

Gujarat Mineral Development Corporation operates the Surka North lignite mine in the western state of Gujarat, India. The deposits are fairly huge but there are random occurrences of pyrites which has caused reduced marketability of the lignite produced in this mine. Ardee set up a 2 million tonne per annum capacity plant at this mine to remove pyrites and restore marketability of the lignite. In this case, the dry beneficiation system is a standalone plant whereby the output is directly dispatched to customers with no further intermediate processing. The same can be planned for some coal mines where the removal of contaminants in a single stage operation can help make the clean coal output conform to the quality stipulations of customers and give the mine operator an edge. Since lignite cannot be washed, dry beneficiation is perhaps the only affordable solution. Also, lignite contains 30 – 50% moisture and hence, dry beneficiation technologies like Air Jig and Air Dense Media are not applicable.

**Case Study 2 – BCCL's Madhuban Washery, Jharkhand, India**

Coal India's subsidiary, Bharat Coking Coal Ltd. operates a 1.5 million tonne per annum coking coal washery at Madhuban. This washery was originally designed for treating raw coking coal with an ash range of 28 – 30% to be brought down after washing to 18%. Over time, mined coal quality deteriorated and today substantial part of raw coal is having more than 40% ash. The result is that the washery is severely challenged in maintaining ash% and yield in clean coal. A deshaling circuit has been added to remove extraneous contaminants in the 50 – 13 mm size fraction (after crushing) so that feed to washery gets restored to original design levels. In this instance, the ArdeeSort dry beneficiation system is an intermediate process to prepare the raw coking coal, make it more amenable to washing and in the process help improve washery performance and techno-economics. Here it plays a complementary role to conventional washing.

**Hybrid Technologies – The Future ?**

Coal India Ltd., the world's largest coal miner, has decided to dispatch only washed coal to customers in future. This has to be achieved at least cost with maximum efficiency. The progress on bids floated for nearly 100 million tonnes of washing capacity is not very encouraging. Many of them are not close to the finishing line even after 5 years since the original proposal was mooted. This places at risk the Indian government's target of enhancing clean coal supply to protect the environment.

One way out of this quagmire over extended deadlines is introducing dry beneficiation plants at a number of locations in the first phase of an accelerated programme that needs to be put into place. What this effectively does is to press into service Dry Beneficiation technology on an express mode which can start generating output within 6 – 9 months after zero date. These plants will yield a certain quantum of clean coal and byproducts consist of middlings and rejects. The latter can again be re-processed to separate middlings and rejects. While rejects can be dumped in voids in mined areas, middlings can be sold separately through auction method based on declared quality. Cleans can be dispatched to distant consumers, as per the objectives of the original clean coal programme, to minimize transport costs and prevent emissions at distant locations. This ensures compliance with mandates stipulated by the Govenent of India.

In the second phase, the middlings can again be re-processed after suitable re-sizing to achieve maximum liberation, and generate two products – clean coal and rejects. This effectively extracts maximum carbon from the coal, minimizes water requirement at reduced capital and operating costs. And as outlined earlier, land space requirements can also be reduced substantially. The hybridization of the washed coal programme enables quicker delivery of some quantity of clean coal in Phase I at the least cost and complete washed coal in Phase II when middlings are re-processed. In the bargain, Coal India gets a bonus in terms of a huge drop in capex, slashing of water and magnetite requirement for the washing of coal and reduced opex.

Why discuss an issue concerning Coal India Ltd. at an international forum? Simple reason is that this methodology holds out a lot of promise for other countries also. While some scholars, in India and world over, have advocated dry beneficiation of coal in water-challenged areas, this is a narrow way of looking at the issue, ignoring the larger picture that can help the coal industry. It can be demonstrated that dry beneficiation be taken up as the primary coal preparation technology to reduce quantity of coal that requires to be treated further. As raw coal is first re-sized using primary and secondary crushers, similarly futuristic way of looking at DE-XRT sorting is that raw coal after re-sizing will be first sorted whereby usable clean coal will be selected and bypassed from washery circuit. This helps reduce washery load and facilitates preserving the environment. The upshot of this approach is that it also ensures that high costs of coal washing do not discourage miners from taking initiative to improve quality of dispatched coal.

The following table contains the data relating to thermal coal quality from some mines in different sizes and the generation of two products – rejects and cleans. However, by introducing a secondary stage of sorting, it is possible to generate three products – cleans, middlings and rejects. While cleans are rejects are not treated further, only middlings are to further extract clean coal from this fraction.

**Table 1: Typical Testing Results for Different Indian Coals**

| Coal From | Size (mm) | Feed Ash% | Cleans Ash% | Rejects Ash% | Yield% |
|---|---|---|---|---|---|
| Chandrapur | 13 – 25 | 27.01 | 22.07 | 77.56 | 87.58 |
| | 25 – 50 | 22.11 | 18.11 | 72.05 | 92.39 |
| Ramagundam | 13 – 25 | 51.38 | 29.29 | 78.51 | 55.11 |
| | 25 – 50 | 57.97 | 30.10 | 80.77 | 45.00 |
| North Karanpura | 13 – 25 | 47.01 | 39.79 | 78.17 | 81.18 |
| | 25 – 50 | 40.50 | 34.04 | 60.98 | 76.02 |
| Talcher | 13 – 25 | 53.31 | 39.60 | 78.62 | 66.70 |
| | 25 – 50 | 42.16 | 33.21 | 61.74 | 68.60 |
| Madhuban (Coking Coal) | 13 – 25 | 43.62 | 30.20 | 80.29 | 73.20 |
| | 25 – 50 | 46.31 | 29.58 | 73.94 | 62.30 |

Source: Testing Results from Ardee's R&D Prototype Testing Centre, Visakhapatnam

Above table illustrates the possibilities of using ArdeeSort to improve techno-economics of washing. In India, the target ash for clean coal in power plants located at 500 kilometres away from mine, is 34%. By appropriately changing the settings, it is possible to generate 34% ash coal from most of the above mines. The yield may vary but ultimately there is a possibility of a certain amount of coal being segregated as clean coal having < 34% ash. The remaining coal can be sold in the nearby areas either as a separate arrangement or through e-auction which is the current practice for non-Fuel Supply Agreement linkage coal.

A study by R.K. Sachdev, President of Coal Preparation Society of India (to be published in a forthcoming paper), has advocated a more flexible approach towards coal preparation and beneficiation. Sachdev has demonstrated that a Hybrid Washery can cut down capital costs by 31% and operating costs by 28% per tonne while at the same time enhancing the capacity of the washery 2.5 times with a marginally enhanced footprint and no compromise on yield or quality. In fact, quality of cleans is better due to no added moisture in sorted coals.

ArdeeSort technology is presently working at two locations and it is hoped that in the coming period, it transforms from being a niche technology to its projected primary role in coal preparation and beneficiation. This will help preserve role of coal in energy sector especially in developing countries.

The global coal community need contribute their bit towards mitigating the environment by use of appropriate technology rather than walk along the beaten path. As the ferocity of green warriors increase, coal industry needs to demonstrate that they too are concerned and doing their bit towards making the environment for the global citizen better.

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# Coal Preparation in India : New Business Opportunities & Need for Dry Separation Technology

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## Abstract

Coal is the prime fuel for generation of commercial energy in India. Coal India Limited (CIL) and Singareni Collieries Company Limited (SCCL) are the two government controlled public sector companies, accounting for over 85 percent of the total coal production in the country. The coal resource available within the geographical boundary is characterized by high percentage of inert mineral contents and difficult washing amenability, mainly because of its drift origin, wherein the extraneous impurities were intimately mixed in the coal matrix during the formation stage itself, causing high ash content in the run-of-mine coal.

The country has undertaken a massive programme of setting up new power plants. Nearly 90% of the additional power generation capacity is planned through coal. As a result, the requirement of coal is expected to increase manifold in the years to come. Non-availability of low ash coal is a deterrent for efficient technological application at the end of power producers, whereas incremental coal production will be from the sources mostly producing high ash coal in general. To address concerns on environmental issues for which stringent regulations is already in place, use of high ash coal has been permitted only to the fewer power plants such as those located near pit-head. This has resulted into spurt in demand for washed coal.

Coal India Limited is setting up as many as 15 coal washeries with 112 Million tonne annual throughput capacity. Such a large venture has faced challenges of managing the associated resource requirements, especially water. Dry Beneficiation technology may be a viable alternative. Some of the dry beneficiation technologies have been considered for development in India with lab-scale design, modelling and testing, along with simultaneous demonstration of available technology at plant level. Coal India Limited is further looking for an established and efficient dry beneficiation technology provided there is significant potential in Indian scenario.

**Key Words: Build-Own-Operate, Washery, New Business Opportunity, Dry Beneficiation, Winnowing, Air-jig, X-Ray**

## 1. INTRODUCTION

The energy sector in India is coal intensive. More than 50% of energy demand is met by this solid fossil fuel. The coal sector in India is passing through sea change with opening to non-governmental operators too. Presently Coal India Limited (CIL) and Singareni Collieries Company Limited (SCCL) are the two government controlled public sector companies, accounting for over 85 percent of the total coal production in the country. Most of the resource available is not very friendly for direct application because of inferior grades. The most common undesirable matter in Indian coal is the noncombustible inorganic mineral matters, that is, the 'ash', which got associated with coal during its origin in geological past. The ash content is usually high; 20 to 40 % in coking coal and 30 to 50 % in non-coking coal. Sulphur is, by and large, not an issue.

 
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## 2. COAL WASHING – APPROACH & BUSINESS MODEL

The challenge is to use such coals in efficient and clean manner with due regards to the environmental issues. Attention has been focused on processes that add value to coal by reducing the percentage of mineral matters or impurities ingrained in coal. For the purpose, Coal India Limited is setting up as many as 15 coal washeries having aggregate capacity of 112 Million ton raw coal throughput, as an initiative in first phase.

With the priority on technology as well as operational efficiency, the business model adopted for setting up washeries is "Build-Operate-Maintain' (BOM), in which bidder has been given freedom for selection of efficient state-of-the-art technology and is responsible to operate & maintain the washery at the guaranteed operational parameters including efficiency. CIL has extended full financial supports in setting up of washeries.

Such an ambitious venture has, however, encountered several challenges in the way of managing the associated resource requirements, especially water. The potential coal bearing areas in India are mostly water scarce. For dealing with setting up of washeries in subsequent phases, water availability is likely to aggravate further. Dry beneficiation technology may be a sensible alternative. It has an edge over conventional wet technologies of coal beneficiation due to inadvertent addition of moisture in later, which offsets the benefit of ash-reduction to some extent as ash-forming mineral matters and moistures both have no fuel value and cause wastage of heat when coal is incinerated in boilers. That is, overall GCV of the solid fuel is affected adversely due to moisture addition. Dry Beneficiation promises to address these issues in an environment-friendly manner and is likely to circumvent these problems.

## 3. DRY BENEFICIATION

Different technological equipment have been developed for dry beneficiation, based on the particles physical properties such as density, size, shape, magnetic susceptibility, electrical conductivity etc. They are broadly classified as air jigs, sorters, magnetic separators, air-dense medium fluidized bed separators, electrostatic separators etc.

Coal India Limited and Ministry of Coal have given due consideration to adopt dry coal beneficiation and a number of projects have been approved for research supports. The approach is oriented towards development of new concept as well as embracing the existing technology for pit-head application in Indian coal mines, which is widely spread in small mining capacity. Dry beneficiation washery usually has simpler circuit and plant set up time is also less comparative to wet washing plants.

### 3.1. NEW DEVELOPMENT

Some of the dry beneficiation technologies have been considered for development in India with lab-scale design, modelling and testing, along with simultaneous demonstration of available technology at plant level.

#### a) Coal Winnowing

The winnowing technology has been in use in agriculture sector for sorting of the chaff or husk and rice from rice stalk after harvest and threshing since time immemorial. Separation of particles occurs in horizontal air stream due to their differential specific gravity.

Efforts is being made to sort coal and ash particles by application of winnowing technique in association with Central Institute for Mining & Fuel Research, Nagpur. The basics have been established with the outcome that beneficiation of coal is possible with the application of winnowing technology after due understanding of governing parameters such as the dynamics of air flow around coal particles.

In the winnowing type air stream separator, air is blown horizontally or at an inclined angle to the horizontal against mixed feed (coal) injected along the vertical plane. The aerodynamic drag force $\boldsymbol{F_d}$ exerted upon a particle by a stream of air is a function of the frontal area $\boldsymbol{A}$ of the particle, the particle density

$\boldsymbol{\rho_a}$ and the relative velocity between the air and the particle $\boldsymbol{V_r}$ and is governed by the formula:

Fig-1: Coal Winnowing Laboratory set-up

$$\boldsymbol{F_d} = \frac{1}{2} C_d \boldsymbol{\rho_a} A V_r^2,$$

where ($\boldsymbol{C_d}$) is drag coefficient.

The drag force ( $\boldsymbol{F_d}$ ) accelerates the particle until it acquires the velocity of air stream. The heterogeneous materials are displaced along the horizontal plane at various distances based on their aerodynamic properties. This has advantage of producing two or more products based on gravity within a short time. There were seven boxes placed one after another in a row to collect particles sorted based on density. Experiments in coal winnowing machine were carried out at different coal size fractions such as 50-25 mm & 25-13 mm on different coal samples. One of the preliminary test data is presented in Table -1 below.

| | Box | | | | | | | Total |
|---|---|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | |
| Wt of coal, gm. | 1151 | 3160 | 2407 | 1689 | 789 | 446 | 386 | 10000 |
| M % | 6.4 | 6.2 | 7.0 | 7.4 | 7.8 | 8.4 | 8.4 | |
| A % | 38.7 | 37.1 | 28.3 | 25.7 | 21.9 | 16.6 | 18.8 | 30.5 |

Tab -1: Test Data with Laboratory set-up, CIMFR-CSIR

Further work is in progress for obtaining consistent performance.

**b) X-Ray based Radiometric Technology**

Detection of coal and mineral laden ash with the application of dual energy x-ray technique is another area where CIL has recently extended its support for indigenous development in association with M/s Ardee Hi Tech (P) Ltd. The sorter has been given a name of 'ArdeeSort'. Multi-energy radiometry has been established as an ideal system for rapid quality determination of smaller coal particles that has been used for subsequent acceptance or ejection based on pre-set parameters. The know-how encompasses – coal chemistry, radiation physics, sensor technology, electronic hardware and software, mechanical handling systems, pneumatic sorting technology, interface systems and final integration of all these know-how into one composite equipment.

Attenuation of x-rays by particles depends primarily on their atomic numbers. The primary combustibles in coal, viz., carbon, oxygen and hydrogen and derived elements, have relatively low atomic numbers. The incombustibles like silica, alumina, iron, calcium etc. have higher atomic numbers. Therefore, the attenuation of a particle will depend primarily on the constituents - higher the ash content, higher the attenuation. Dual energy sensor technology is utilized to neutralize the effects of size of particle on attenuation. The mathematical algorithm that analyses the dual energy signal automatically computes the composite ratio of organics to inorganic in the particle. The operator has the freedom to set the limits of inorganic (ash) percentage in particles that can be accepted, which is akin to setting the specific gravity in a dense media bath. In fact, the distinct advantage of such a radiometric technologies is that the target for clean coal or the threshold value for rejection can be planned and set as per need. After consistent research and series of tests followed by modifications, the technology available now is efficient, dust-free and energy friendly.

Having achieved lab-scale viability of identification and removal of heavier particles using focused pneumatic jet nozzles, the ArdeeSort modules are being commissioned at existing Madhuband washery for demonstration at full scale industrial application.

## 3.2. DEMONSTRATION OF EXISTING TECHONOLOGY

Development of a technology from fundamental and subsequent validation for confidant industrial application is resource intensive; especially significant time and substantial money is required. As such, Coal India Limited as a commercial organization is simultaneously striving for introduction of established dry coal beneficiation technology. A project for demonstration of air-jig has been taken up. Other efficient technologies can also be considered for adoption after pilot scale demonstration of their potential for Indian non-coking coal for power sector.

## 4. CONCLUSION:

The coal washery business in India is passing through sea change in almost all aspects, viz. business growth, open market scenario, technology freedom, financial support etc. Dry beneficiation technology is the need of posterity for more effective resource management. Involvement of all stakeholders will boost the efficient utilization of large resource of low grade coal in India. There is a vast opportunity for the globally established players in the coal washing sector.

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# Dry Coal Preparation of Fine Particles by KAT Process

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## Abstract

In this study, effectiveness of KAT Process, dry coal preparation process developed by KIGAM (Korea Institute of Geoscience and Mineral Resources), was assessed for bituminous coal from MAK-Naryn Sukhait (NS) coal mine, Mongolia. 1-25 mm raw coal delivered from Mongolia was screened and classified with 1-5 mm, 5-10 mm, 10-20 mm, and over 20 mm particle size range. Separation tests were conducted for 1-5 mm and 5-10 mm size fractions using two types of pilot scale KAT Process units, suitable for each size range. Prepared test samples were concluded as well liberated samples having less than 3 % of weight material between specific gravity range of 1.6 to 2.0 g/cm$^3$. Test results conducted under optimum conditions observed at previous investigations. Overall, the average ash rejection and combustible recovery for 1-5 mm fraction was 81% and 77% respectively and the feed ash content was decreased from 35% to 12% in product and the average product yield was 56.3%. The 5-10 mm size fraction average ash content was 48.4%. After the dry separation, about 41% of the feed was recovered as product with average 9.4% ash content. Ash rejection and combustible recovery was 91.7% and 76.8% respectively. Conducted tests showed that KAT process can effectively remove large amount of dense impurities from low grade coal thus producing an acceptable ash content clean coal product.

***Keywords : Low grade coal, Dry coal beneficiation, Air table, Autogenous medium***

## 1. Introduction

Coal is the one of the main fossil fuel for energy and most of worldwide energy consumptions still depend on coal. Coal needs to be cleaned from its gangue minerals in order to increase its quality and to meet the industrial standards through many kinds of coal beneficiation techniques. Today the dominant coal beneficiation technologies mostly rely on wet methods which requires huge amount of process water and inescapable high operation cost during processing procedures. Waterless coal beneficiation technologies have some notable advantages compared to wet technologies. The dry beneficiation methods are based on the differences in physical properties between coal and mineral matters such as density, size, shape, lustrousness, magnetic susceptibilities, electrical conductivity, frictional coefficients. [4]. The dry coal-cleaning separators can be broadly classified into three groups: air tables, air jigs, and dry dense medium separators [5]. Optical sorting, magnetic and electrostatic separators also considered as part of the dry processes. Dry preparation of coal can be economical by reducing capital and operational costs. It will not utilize water and no dewatering of the product will be required. Although having above economic and environmental advantages, dry separation processes still quite inefficient in separation accuracy for

 
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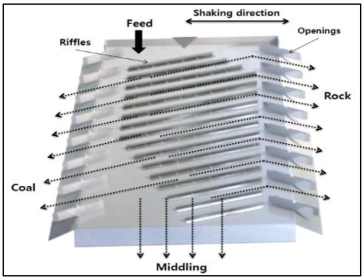
finer size material comparing to modern wet processing technologies with more sharper separation performances. This study investigates the air table separation performance of fine particle (1-10mm), low grade coal with large amount of dense impurity. KAT (Korean Advanced Technology) table is a type of pneumatic table separator. It separates the components in the ROM coal according their specific density differences. The denser material settle down to form a lower layer and moves to the oscillating direction of the table deck by the action of eccentric motion. The upper layer of light particles roll down by the means of airflow and table shaking to the lower part of the deck and accumulates along the blocking wall, and when the level of accumulated light particles exceed the height of blocking wall it will start overflow. This specific design of KAT table generates an autogenetic medium of light particles along the lower end of the deck thus increases a product quality. KAT table is separating ROM coal into 3 different fractions. The light fraction-coal, the heavy fraction-reject and the middling.




a) b)

**Fig.1** a) Illustration of dry separator unit. b) Material separation trajectory on the table deck.

## 2. Material preparation

A low grade 1-25 mm ROM coal sample was collected from Naryn Sukhait coal mine of Mongolyn Alt Corporation, Mongolia. Delivered sample was screened to obtain +20mm, 10-20 mm, 5-10 mm and 1-5 mm size fractions. 5-10 mm and 1-5 mm size fractions were used for dry beneficiation tests. Float and Sink tests were carried out on every size fractions using prepared dense solutions of Tetrabromoethane and Tetrachlorethylene with density ranges between 1.3g/cm$^3$ o 2.0g/cm$^3$ at 0.1g/cm$^3$ intervals. Each relative density float and sink materials were measured for proximate analysis and calorific values using Leco TGA701 Thermogravimetric analyzer and Leco AC600 Calorimeter.

**Table.1.** Analysis of the 1-5 mm size fraction test samples.

| 1-5 mm size fraction | | | |
|---|---|---|---|
| Test-1 | Weight (%) of 2.0 g/cm³ dense material | Cumulative Calorific Value (Kcal/kg) | Cumulative Ash content (%) |
| 1 | 35.4 | 4746 | 37.5 |
| 2 | 32.8 | 4996 | 34.7 |
| 3 | 33.3 | 4989 | 34.7 |

**Table.2.** Analysis of the 5-10 mm size fraction test samples.

| 5-10 mm size fraction | | | |
|---|---|---|---|
| Test | Weight (%) of 2.0 $g/cm^3$ dense material. | Cumulative Calorific Value (Kcal/kg) | Cumulative Ash content (%) |
| 1 | 51.8 | 3485 | 51.1 |
| 2 | 50.3 | 3705 | 49.3 |
| 3 | 48.5 | 3792 | 48.2 |
| 4 | 47.4 | 3961 | 46.4 |
| 5 | 47.5 | 3928 | 46.9 |

The KAT air table experiments were aiming to remove over 2.0 $g/cm^3$ specific density fractions from feed coal thus decreasing ash content of product. From Table.1 and 2, it can be seen that both size fraction feed has large amount of dense noncombustible particles, over 33% in 1-5 mm and 47-51% in 5-10 mm size fractions.

## 3. Experimental

All tests were conducted in a dry coal preparation pilot plant installed at Hanbit KSE, Ulsan, Korea. Tests were purposed to investigate KAT table's effective feeding rate with optimum ash rejection value. For each test about 1 ton of prepared and weighted raw coal fed onto the table deck through belt conveyor from feed hopper. After feed coal poured on the deck, coal dust and ultra fine particles were blown upward by airflow and enters to the dust hood placed above the deck. These dust and fine material further delivered to cyclone separator. KAT-1 table operating parameters, other than feeding rate, were fixed based on previous test observations. Feeding rate was increased from 3.7 t/h to 5t/h for 1-5 mm and from 5t/h to 31 t/h for 5-10 mm.

**Table.3.** KAT-1 and KAT-2 table operating parameters

KAT-1, Naryn Sukhait, 1-5 mm

| Test | Test-1 | Test-2 | Test-3 | Unit |
|---|---|---|---|---|
| Feeding rate | 3.7 | 4 | 5 | Ton/hour |
| Airflow frequency | 60 | 60 | 60 | Hz (0-60) |
| Table frequency | 40 | 40 | 40 | Hz (0-60) |
| Cross table slop | 5.5 | 5.5 | 5.5 | Degree |
| Longitudinal slope | 5 | 5 | 5 | Degree |

KAT-2, Naryn Sukhait 5-10 mm

| Test | Test-1 | Test-2 | Test-3 | Test-4 | Test-5 | Unit |
|---|---|---|---|---|---|---|
| Feeding rate | 5 | 6.5 | 10 | 20 | 31 | Ton/hour |
| Air frequency | Full | Full | Full | Full | Full | Full open |
| Table frequency | 53 | 53 | 53 | 53 | 53 | Hz (0-60) |
| Cross angle | 7 | 7 | 7 | 7 | 7 | Degree |
| Longitudinal angle | 5.5 | 5.5 | 5.5 | 5.5 | 5.5 | Degree |

Product, middling and reject were collected in a ton bag and sampled for analysis. Reference samples were analyzed for density distributions and ash contents. The actual yield and ash rejections were calculated by using an ash balance equations 1 and 2. After every test the feed ash content was recalculated using weight and ash (%) of product, middling and reject material.

$$Yield\% = \frac{Reject\ Ash - Feed\ Ash}{Reject\ Ash - Product\ Ash} \times 100 \quad [1]$$

$$Ash\ Rejection\% = \frac{Yield \times Product\ ash}{Feed\ ash} \quad [2]$$

## 4. Results

Dry coal cleaning tests with Naryn Sukhait 1-5 mm size fractions showed that KAT air table is effectively removing high density reject materials from feed coal. Test results for 1-5 mm size coal test shown in Table.4 and 5.The run No.2 gives most favorable results with product ash content of 9.4% and 85.5% ash rejection rate. The heat content of feed coal was increased from 4996 Cal/g to 7214 Cal/g with 74.1% combustible recovery.

**Table.4.** KAT-1 table products

| Naryn Sukhait,1-5 mm | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Test | Product | | | | Middling | | | | Reject | | | |
| | 2.0 g/cm³ Sink fraction | Calorific value (Kcal/kg) | Ash (%) | Yield (%) | 2.0 g/cm³ Sink fraction | Calorific value (Kcal/kg) | Ash (%) | Yield (%) | 2.0 g/cm³ Sink fraction | Calorific value (Kcal/kg) | Ash (%) | Yield (%) |
| 1 | 10.3 | 6627 | 16.0 | 57.2 | 41.8 | 4170 | 44.5 | 20.4 | 93.7 | 475 | 86.1 | 22.4 |
| 2 | 4.3 | 7214 | 9.4 | 53.4 | 32.7 | 4748 | 37.9 | 22.9 | 97.2 | 231 | 88.7 | 23.7 |
| 3 | 5.6 | 7107 | 10.7 | 58.3 | 33.8 | 4751 | 37.3 | 14.0 | 91.5 | 647 | 84 | 27.6 |

**Table.5.** KAT-1 test results, Naryn Sukhait,1-5mm

| Test | 1 | 2 | 3 | AVG |
|---|---|---|---|---|
| Feed Ash (%) | 37.48 | 34.73 | 34.72 | 35.6 |
| Product Ash (%) | 15.96 | 9.42 | 10.7 | 12.0 |
| Reject Ash (%) | 66.25 | 63.74 | 68.3 | 66.1 |
| Feed Cal/g | 4746 | 4996 | 4990 | 4911 |
| Product Cal/g | 6627 | 7214 | 7107 | 6983 |
| Actual Yield (%) | 57.21 | 53.41 | 58.3 | 56.3 |
| Recovery of ash in Product (%) | 24.36 | 14.49 | 17.97 | 18.9 |
| Ash Rejection (%) | 75.64 | 85.51 | 82.03 | 81.1 |
| Ash Reduction (%) | 57.42 | 72.88 | 69.18 | 66.5 |
| Combustible Recovery (%) | 76.90 | 74.12 | 79.75 | 76.9 |

2.0 g/cm³ sink material fraction decreased to 4.3% in product and increased to 97.2% in reject.

Table.6 and 7 shows test results for Naryn Sulhait 5-10 mm size fractions. From conducted tests run No.4 gives most favorable results. Feeding rate was 20 ton hour and feed ash 46.4% decreased to 9.4% in product and ash rejection rate was 92%. Run No.5, test with feeding rate of 31 ton hour, also gives good results. About 47.8% of feed was recovered to product with 10.6 % ash and ash rejection was 89.2%.

**Table.6.** KAT-2 table products

| Naryn Sukhait,5-10 mm | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Test | Product | | | | Middling | | | | Reject | | | |
| | 2.0 g/cm³ Sink fraction | Calorific value (Kcal/kg) | Ash (%) | Yield (%) | 2.0 g/cm³ Sink fraction | Calorific value (Kcal/kg) | Ash (%) | Yield (%) | 2.0 g/cm³ Sink fraction | Calorific value (Kcal/kg) | Ash (%) | Yield (%) |
| 1 | 6.3 | 6721 | 13.3 | 41.1 | 52.8 | 3385 | 52.9 | 18.2 | 97.3 | 258 | 88.6 | 40.7 |
| 2 | 1.3 | 7432 | 6.9 | 34.5 | 52.4 | 3357 | 53.4 | 32.8 | 99.6 | 130 | 89.8 | 32.7 |
| 3 | 1 | 7336 | 7.7 | 36.7 | 42.9 | 4001 | 46.1 | 26.1 | 99.1 | 151 | 89.6 | 37.2 |
| 4 | 3.3 | 7290 | 8.4 | 44.4 | 57.1 | 3069 | 56.8 | 18.9 | 95.9 | 350 | 87 | 36.6 |
| 5 | 5.1 | 7116 | 10.6 | 47.8 | 67.8 | 2195 | 66.9 | 13.5 | 92.8 | 585 | 84.7 | 38.6 |

**Table.7.** KAT-2 test results, Naryn Sukhait,5-10 mm

| Test | 1 | 2 | 3 | 4 | 5 | AVG |
|---|---|---|---|---|---|---|
| Feed Ash (%) | 51.1 | 49.3 | 48.1 | 46.4 | 46.9 | 48.4 |
| Product Ash (%) | 13.3 | 6.9 | 7.7 | 8.4 | 10.6 | 9.4 |
| Middling mixed to Reject Ash (%) | 77.7 | 71.6 | 71.6 | 76.7 | 80.1 | 75.5 |
| Feed Cal/g | 3485 | 3705 | 3794 | 3961 | 3928 | 3775 |
| Product Cal/g | 6974 | 7432 | 7338 | 7290 | 7116 | 7230 |
| Actual Yield (%) | 41.1 | 34.5 | 36.8 | 44.4 | 47.8 | 40.9 |
| Recovery of ash in Product (%) | 11.7 | 4.8 | 5.9 | 8.0 | 10.8 | 8.3 |
| Ash Rejection (%) | 88.3 | 95.2 | 94.1 | 92.0 | 89.2 | 91.7 |
| Ash Reduction (%) | 71.5 | 86.0 | 84.0 | 81.8 | 77.4 | 80.1 |
| Combustible Recovery (%) | 72.7 | 89.8 | 65.4 | 75.8 | 80.4 | 76.8 |


**Fig.1.** KAT process during operation

## 6. Conclusions

The low grade high ash coal cleaning and de-ashing study was conducted using a dry air gravity table. It is found that KAT table is able to reduce significant amount of high ash dense particles from 1-5 mm and 5-10 mm size fractions of Naryn Sukhait mine multi seam and high impurity coal.
Overall, the average ash rejection and combustible recovery for 1-5 mm fraction was 81% and 77% respectively and the feed ash content was decreased from 35% to 12% in product and the average product yield was 56.3%. The 5-10 mm size fraction average ash content was 48.4%. After the dry separation, about 41% of the feed was recovered as product with average 9.4% ash content. Ash rejection and combustible recovery was 91.7% and 76.8% respectively. Closely sized feed material gives better separation results. It is found that KAT table effective feeding rate is 4.5-5 ton/h for 1-5 mm size fraction and 20-30 ton/h for 5-10 mm coal. Conducted tests showed that KAT process can effectively remove large amount of dense impurities from low grade bituminous coal thus producing an acceptable ash content clean coal product.

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# Part XII
# Preparation and Processing of Carbonaceous Ores

# Key issues for improving of carbonaceous ore beneficiation processes for the extraction of valuable components

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**Abstract**
The results of mineralogic-technological and geochemical research of black shale ores are presented. It has been proved that black shale is a new promising unconventional source of noble and rare metals. Black shale ores of the subjects under study are classified as highly refractory ores and therefore require a special technological approach. To substantiate effective recovery methods, we have studied flotation and extraction options to recover valuable trace elements. Carbonaceous shale of the Kimkan occurrence of the noble Au – Pt mineralization (the Far East) and dictyonema shale of the Leningrad region constituted the main subject of the research. According to the ore composition and structural features of black-shale blocks, the studied subjects are associated with big large-volume gold deposits belonging to black-shale formation: 1) dispersed, finely disseminated condition of noble metal mineralization; 2) presence of carbonaceous matter in ore-bearing rocks; 3) presence of native metals in ore (molybdenum, lead, zink, bizmuth, as well as gold and platinoids).

The obtained results predetermine the general guidelines for creating new methods and techniques of processing carbonaceous material for integral development of solid minerals in the mining and oilfield areas of Russia.

**Key words:** black shale, carbonaceous matter, noble metals, rare elements, flotation, nano-fraction

## Introduction

The problem of processing refractory (rebellious) ores is important for all the countries involved in recovering precious metals from ores. The prospects of enhancing the recovery of noble and rare metals depend on the existing raw material, its quality and the possibility to enlarge it. Complex noble metal deposits in carbonaceous ores, which have been discovered recently, are regarded as a new promising source of gold, platinum, and other useful elements [1]. In order to develop the deposits of carbonaceous ores in the Far East [2, 3] a series of studies is required to develop processing techniques for carbonaceous ores which will ensure maximum recovery of useful components, primarily graphite, gold, and platinum-group metals.

Black shales are presently regarded as a new, promising and unconventional source of noble and rare metals. However, as some researches justly point out, they have been but fragmentarily studied [4-6]. It can be assumed that the considerable amount of platinum group minerals and rhenium is a typomorphic feature of these deposits which not only confirms the rightness of their selection as an independent «black-shale noble- and rare-metal formation", but significantly increases their practical value and makes their development cost-effective in the short term.

The conditions in which black-shale ores were formed, and especially the genesis of their mineralization, have been but little studied and therefore present most controversial issues, which hamper effective search and assessment. There is no consensus about the forms in which noble and rare metals occur, nor about the effective techniques to determine their real concentration in ferrum oxides, clay minerals, biotides, chlorites, alunite, quartz-calcedony, jarosite, some other metal-bearing minerals, and carbonaceous-bituminous inclusions in shales.

 
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Black-shale ores of the subjects under study are classified as refractory ores and therefore require a special technological approach. To substantiate effective recovery methods, we have studied flotation and extraction options for the recovery of valuable trace elements and assessment of typomorphic associations of trace elements The term “typomorphic association of trace elements (TE)” refers to a set of elements whose concentration in useful minerals, their mined or processed products (e.g. gravity, flotation, etc.) is either higher than background values, i.e. average content in minerals, or equals or higher than the content which could be of interest for the production of their product compounds from raw material [4,5]. There are natural ($Q_n$) and technological ($Q_t$) associations of elements, i.e. in the mined ores and their processed products. Ya.E. Yudovich [7] suggested calling the elements “typomorphic” if their content in the ash of carbonaceous ores is higher than their clarke in sedimentary rocks. The term is used in this paper to classify TE in shales. The main strategic metals are shown in Table 1.

TABLE 1 – Rare and strategic metals found in shales

| | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Metals considered strategic in Russia | Be Li | Mo | - | Zr | - | Sc | TR (Y) | Ge | Sb | - | Ni Co Mn Cr | Nb | Re | U |
| Rare metals considered strategic in the US | - | - | Sr | - | Cd | - | - | Ge | - | Bi | V | Nb | - | - |
| Metals found in shales* | Be | Mo | Sr | - | Cd | Sc | Y, Eu, Yb, La | Ge | Sb | - | V Ni Co Mn | Nb | Re | U |

** according to the data published [8]*

Vanadium (V) and nickel (Ni) were among the first metals found in bituminous shales and heavy oil, probably because their concentration was higher than that of other metals.

**Subjects and methods of research:** carbonaceous shales of the Kimkan occurrence of noble Au-Pt mineralization (Far East), dictyonema shale of the Leningrad region.

The test sample of carbonaceous shale of the Kimkan occurrence mostly contains phyllitic muscovite-graphite-quartz shale with variable content of muscovite, graphite and quartz, while chlorite-sericite-quartz phyllits (the general name is black shales) are less common. Shale content: muscovite 5 – 20%, quartz 25 – 70%, graphite 5 – 40 %, biotite 5 – 10 %, sericite 0 – 35 %. Ore minerals: magnetite and metapyrite in the form of impregnations: 0 – 5%. Accessory minerals: zircon, leucoxene, sphen, xenotime, monazite, rutile, apatite, orthite. Graphite is found in the form of thin flakes of 0,01– 0,02 mm and their crystallites, more often in earthy aggregate of 0,01 – 0,05 mm, and is often distributed in all rock-forming minerals, especially in mica. Black shales are often altered by leaf injection, less often by a crosscutting quartz- and muscovite-quartz injection which includes ore mineralization. Metasomatites form clustered, vein-clustered, and vein-lenticular assemblages 0,2 – 25 mm thick. The grain size of the granoblastic quartz II which forms them is 0,1 – 2 mm, the size of a muscovite flake is 0,1 – 1 mm. The structure of hydrothermal and metasomatic assemblages and their interaction with shales were observed in transparent sections. Metasomatites are formed either by transparent and transparent-gray granoblastic quartz, or by muscovite-quartz, which characterizes them as high-temperature assemblages of greisen type [8].

Dictyonema shales of the Leningrad region present an unconventional type of mineral resources, which can be classified as refractory lean uranium ores. They have been regarded as non-profitable for working out, especially considering the modern environmental requirements. The rock contains clay minerals (kaolinite, hydromica, montmorillonite, chlorite), silt to sand-sized detritus (quartz, feldspar, apatite) and authigenic minerals (anthraconite, gypsum, anhydrite, pyrite, siliceous and phosphate concretions). Besides, sporadic grains of gold, platinum, and rare-earth minerals were diagnosed. The organic component percent varies from 6 to 15 %. The shales of the region are characterized by abnormal and increased content of U, V, Mo, Ni, Zn, Pb and noble metals, as well as some other precious metals and associated elements, in particular rare-earth elements (up to 200 g/t), which had not presented commercial interest earlier due to their problematic recovery. However, in the recent years the deficit of the above-mentioned metals has made the issue of integral processing of such metal-bearing formations quite urgent, and it could be solved by treating such formations as polymetallic reserves [9].

The studies of carbonaceous ore beneficiation have been focused on:

1. Carbon flotation using non-ionized and cationic collectors (Fig. 1, Table 1).
2. Stage sulfide-graphite flotation of gravity concentration tailings
3. Extraction of nano-fraction from the raw material.

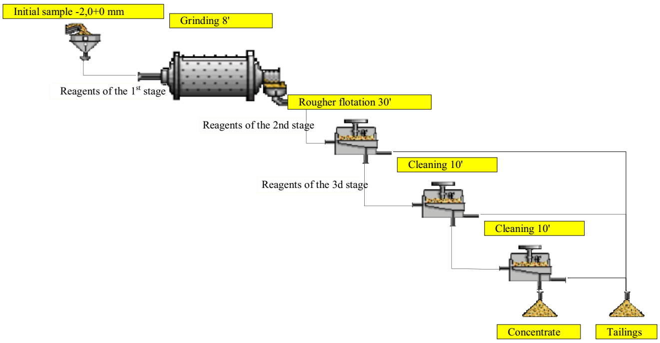


Fig. 1 Carbon Flotation Scheme

The preparation of process samples included grinding, splitting and selecting subsamples for technological and chemical tests. Gravity concentration experiments were conducted using standard laboratory equipment (concentrating table SKL – 2; Knelson concentrator); flotation concentration was conducted using Laarmann Flotation Bench Test Machine. Flotation extraction of graphite products was performed using long-chain amines, kerosene, sodium fluorosilicate and pine oil in various pH-range formed by lime and sulfuric acid. The concentrates were analyzed using atomic absorption and X-ray fluorescence. Optical and electron-microscopic methods with electron microprobe analysis were used in mineralogical-petrographic examination.

In nano-fraction analysis the ore was crushed and grinded to the particle size <75μm. One part of the sample was tested using nanotechnology methods, the other one – by the standard scheme of “complete” decomposition using concentrated nitric, hydrofluoric and perchloric acids. The solutions were tested by inductively coupled plasma mass-spectrometry method.

**Results and Discussion**

Thus, stage sulfide-carbon flotation of gravity tailings is an acceptable method of re-extraction of noble metals. The analysis of data of sulfide flotation of the initial sample reveals low silver recovery, which suggests its presence in gravity-extracted forms. Platinoids and rhenium are partially extracted into graphite concentrate, gold is extracted by gravity methods with re-extraction of finely dispersed gold and gold attached to sulfides by flotation.

The form in which trace elements occur in minerals and ores is an important issue both for geochemists and analytical chemists. A considerable part of chemical elements occur in minerals as isomorphic impurities that replace macro components in the crystal structure.

The results of flotation concentration of carbonaceous shales are shown in Table 2 and Fig. 2.

TABLE.2 Flotation concentration results according to the scheme (Fig. 1)

| Item number | Product | Output | | **Ag** | **Cd** | **Sn** | | **Sb** | **Pt** | **Pb** | Mineralogical analysis of flotation products |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | g | % | | Average element content, g/t | | | | | | |
| 1 | Concentrate | 473,5 | 15, 9 | 2 | 4,3 | 8,3 | | 43, 7 | 0,01 | 40, 3 | Graphite~ 55% Muscovite ~40% Quartz ~5% Pyrite, limonite, metapyrite |
| 2 | Tail | 2510 | 84, 1 | – | – | – | | 44,3 | – | 17, 3 | Graphite-quartz shale~20% Muscovite-graphite-quartz shale~30% Graphite ~10% |
| | Total | **2983,5** | **100** | | | | | | | | |

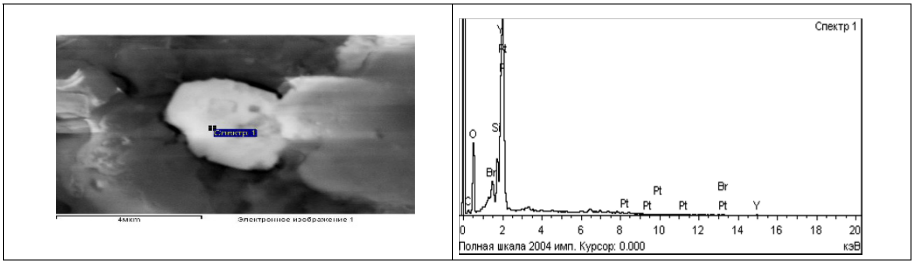


Fig. 2 Microphotographs of noble metal inclusions in flotation concentrates

Some of them are accumulated in gas-liquid inclusions, and others are presented in colloid-dispersed form in the pore space of rock. Besides, for a number of chemical elements the dispersed form is the prevailing one. Moreover, the less the average content of an element in the earth crust is, the higher its degree in the dispersed form [10]. It should be noted that the elements occurring in ultra-dispersed form have never been regarded as independent elements of the system of prospecting of rare and trace elements. Neither have they been extracted in ore processing. Meanwhile, these elements could be the most flexible and the most bioactive ones in different ecosystems [11,12].

**Concluzion**

Therefore, the use of flotation methods under reasonable reactant treatment allows for effective concentration of microinclusions of noble and rare metals.

In general, geochemical and technological studies of laboratory samples of carbonaceous shale revealed some common microelement associations, producing the commercial compounds of which could have commercial importance:

- associations of chalcophylic elements caused by their accumulation on recovery or hydrosulfuric barriers including U – Se – Mo – Pb – Zn –Re – Ag, where Mo, U, Se have the highest content (as compared to background values);

- technological associations of noble metals Au-Ag, platinoids and platinoid metals (Pt – Pd – Ir – Os);

- some associations, for example, Ni – C r– Co, which were formed, apparently, as a result of considerable basic and ultrabasic rock mass in alimentation area and are found both in gravity and flotation concentrates. In associations with high content of Mo – Re – Ag – Hg – Pb – Zn – Sn in shale mass, the highest concentration rate (as compared to their background concentration) was found for Mo.

According to the ore composition and structural features of black-shale blocks, the studied subjects are associated with big large-volume gold deposits belonging to black-shale formation: 1) dispersed, finely disseminated condition of noble metal mineralization; 2) presence of carbonaceous matter in ore-bearing rocks; 3) presence of native metals in ore (molybdenum, lead, zink, bizmuth, as well as gold and platinoids).

The obtained results predetermine the general guidelines for creating new methods and techniques of processing carbonaceous material for integral development of solid minerals in the mining and oilfield areas of Russia.

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# Title: A CYCLONE FOR A REASON – Dense Medium Cyclone efficiency.

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**Abstract for Coal Prep. Congress,**

Cyclones have been in mineral processing since the first patent in 1912 and amongst the many uses of the configuration of the cyclones is the DENSE MEDIUM CYCLONE, used for separation of particles of different densities; separation enhanced by the use of either magnetite or ferrosilicon as an essential element of the process to create the desired effect.

The MULTOTEC dense medium cyclone has been successfully applied in, primarily, coal washing but also in hard rock beneficiation, specifically diamonds, iron ore and platinum, to enhance and improve the overall process resulting in improved revenues for the user-plant.

The initial ratios between the various components making up the dense medium cyclone (DMC) were developed by the Dutch State Mines Research Institute and still apply today. Thus DMC may be said to be "old technology" but improvements in materials of construction (linings, patterns), methodology of manufacture and vital ancillary components (pumps, vibrating screens and magnetic separators) have enabled the technology to be applied in different configurations worldwide at higher capacities with improved life cycles and efficiencies.

This paper will briefly describe some of MULTOTEC's experiences and applications of DMC in coal, specifically touching on "high density cut points" in coal and detail some current successful experiences in coal plants.

The paper will include technical data comparing proposed performance parameters with actual results from operating plant and conclusions to be drawn from the data.

**Key words**: dense medium, cyclones, process, coal washing, high density, yield, clean coal, ash.

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**Introduction**

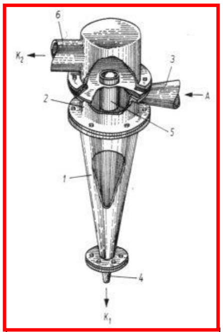



Fig 1
CYCLONE PATENT

The patent for a hydrocyclone registered in 1912 and the cyclone has been evolving ever since. Initially the unit was very much simpler design but it certainly provided a quantum leap in process technology and enabled improvements in following years.

In the mid-1940's the Dutch State Mines (DSM hereafter) identified that cyclones had potential use for the separation of coal/ash and subsequent research established the various ratios between

- inlet opening : inlet head diameter : vortex finder diameter : spigot diameter that gave the optimal efficiency of separation.

Research also showed that, for coal processing, magnetite is the favoured dense medium, which, added to the water, creates slurry of suitable capability to separate sinks (ash) and floats (coal).

In South African research in 1950 by the then Fuel Research Institute of South Africa, tests concluded that led to the application of dense medium cyclones in coal processing. For South Africa this meant the first thermal coal washing using dense medium cyclones began in the Witbank Coalfields.

Power station feed coal has increasingly come under the microscope as environmentalists query the emissions. Not only that, but increasing attention is paid to the quality of the coal burned specifically ash content because, in both South Africa and India, coal is transported over long distances which causes road deterioration. Further, costs of electric energy, driven by increasing demand, dictates that power stations be more diligent about the quality of coal received from suppliers.

There are many benefits to using washed coal in power plants including:

- Consistent power plant performance and improved thermal efficiency
- Reduced ash handling
- Reduced emission issues
- Lower maintenance resulting from coal's lower wear characteristics than "stone"

For the country, washed coal results in:

- Less road damage from transportation of "stone"
- Cleaner air
- Better power generation costs
- Better coal reserve utilization through more efficient lower grade reserve exploitation.

As with all good things, there is a downside and that is that the coal washeries have a more complicated process with potentially higher costs unless due diligence is applied to process and resultant equipment selection.

Thus more efficient process systems were sought to produce clean coal.

**Early observations**

Once the efficacy of Dense Medium Cyclones (DM) was established and cyclone washing of coal was put into practice certain limitations raised their ugly heads.

The established ratios of inlet throat : inlet head diameter : volumetric capacity of DM cyclone : vortex finder : spigot and the considered best ratio between medium and ore of 3.5:1.0 meant that DM cyclones had to be manufactured in specific proportions.

This size limitation also dictated the largest size of coal that could be fed because there is a direct relationship between inlet throat size and coal lump size beyond which the cyclone will choke.

Similarly, the inlet diameter, which is also proportional to the inlet throat, determines the volumetric capacity of the cyclone, and therefore effective throughput of solids because, following the rules developed for best separation, the ratios of medium to ore should be a minimum of 3.5:1.0 for proper washing and separation of the coal.

**Cyclone configurations.**

In the early phases, the DM cyclones were made of cast iron with high chrome and nickel content. Castings were favoured technology because manufacturers had been able and competent at using this methodology for some years.

The high chrome and nickel were added as a wear resisting element ensuring that the cyclones lasted some time rather than requiring continuous maintenance and spares replacement and inventory.

The downside of this manufacturing method is that the maximum size of cyclone was limited by the mould required to make the unit and the handling capability of the foundry.

Thus, initially, DM cyclones were at a maximum size of 710 mm inlet head and capacity of the order of 100 tph of raw coal feed, maximum lump size approx 50 - 60 mm.

Consequentially, coal washeries therefore needed a combination of technologies to wash the coal from 150 or 80 mm down. Flowsheets were seen to include drum washers like the Teska, Wemco, Norwald and so on for the plus 10 to 20 mm coal, and the less-than-that size in DM cyclones. Fines below 2.0 mm presented further issues like simple, cost-effective separating equipment, dewatering operating cost constraints vs. clean coal returns.

Focusing on cyclones, we turn to the Grootegeluk Colliery in South Africa. This mine has the largest coal output in South Africa and probably the southern hemisphere. Accordingly, any efficiency implemented on the plant has high value returns.

**Operational considerations**

Initially, large size coal (150 x 28 mm) was washed in drums mainly because there is no limitation on the feed inlet. Cyclones of the day had a small feed inlet (20% of inlet head diameter by DSM rule) which resulted in choking if the 1:3 ratio of throat area : top size was not followed.

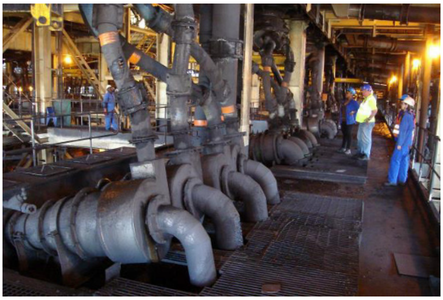


Fig 2
4 x 510mm DM cyclones
Grootegeluk [6]

This meant that the secondary beneficiation at Grootegeluk was carried out in multiple modules each equipped with 4 x 510 mm cyclones because of the high throughputs required by high plant output to power station.

Distribution was identified as another crucial element of good performance, especially as the cyclones were gravity fed (for wear considerations).

With due consideration being given to the conditions, a decision was taken to replace the 4 cyclones with one larger 810 mm unit; pump-fed because of higher pressure needed but essentially replacing each module of 4 with 1 unit. Tests would also be conducted on efficiencies with and without the barrel. [4]

The results speak for themselves and the supplier now had to take the step towards manufacture of larger diameter cyclones.

| Parameter | Reference module | Test 1 | Test 2 |
|---|---|---|---|
| NAME | 4 X 510mm | 1 X 800mm | 1 X 800mm + Barrel |
| FEED RATE | 125t/h | 135t/h | 135t/h |
| M:C | 4 | 4.2 | 4.2 |
| EPM | 0.020 | 0.015 | 0.021 |
| SG50 | 1.373 | 1.382 | 1.366 |
| EPM/SG50 | 0.0146 | 0.0109 | 0.0154 |

Fig 3
Cyclones results comparison, Grootegeluk [4]

The barrel did not, in these tests, substantially improve performance.

**Improved configurations**

Simultaneously, attention was being given by cyclone manufacturers to the cyclone's configuration, methods of manufacture and materials of construction.

Early cyclones followed the initial inlet head configuration and were tangential to the main separating zone in the inlet head. This created turbulence, which resulted in high wearing areas but more critically interfered with the efficient separation of coal product and discards.

Larger DM cyclones were being considered but the limitation of cost caused by large castings prescribed against these larger cyclones – an alternative needed to be found.

Developmental work followed and the subsequent cyclones progressed through involute entry on 90$^{o}$ to evolute entry on 180$^{o}$ (Fig 4) to the current scrolled evolute entry (Fig 5) which has an arguably 35% improvement on the initial tangential capacity and a higher efficiency.

The scrolled entry has an efficiency effect that begins the downward reporting of high density material immediately on entry and therefore improves separation efficiency in the DM cyclone.

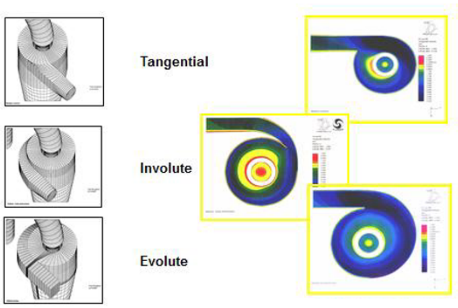



Fig 4
INLET HEAD CONFIGURATIONS

**Materials of construction**

The general use of improved wear resistant linings was also on the upswing at this time and DM cyclones were no exception.

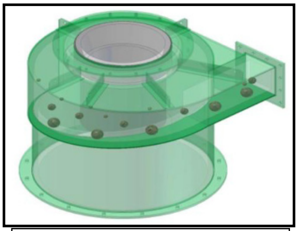


Fig 5
SCROLLED EVOLUTE INLET

Ceramic-filled epoxies had made their entry, as had dense alumina ($Al_2O_3$) ceramics and with the advent of tiles as a lining material, the manufacture of DM cyclones changed.

Now the units were made from fabricated mild steel shells with ceramic linings, installed in increasingly improved patterns e.g. staggered horizontal joints to prevent fine slurry attacking the epoxy adhesive; tapered and angled cone tiles for smoother surfaces, improved wear life and less turbulence.

The ceramic tiles now allowed manufacture of larger diameter DM cyclones, especially as larger pump sizes and greater screen widths were now becoming available from leading equipment suppliers.

The advent of the large diameter, high capacity, very efficient dense medium cyclone was at hand.

**Essential ancilliaries.**

Together with dense medium separation comes a host of support equipment without which the DM cyclone or even the drum, cannot function. These relate to primary classification and desliming, washing and recovery of medium, pumping and naturally drying.

In the early stages of dense medium coal washing, pumps had not yet been developed to match high volumes of slurry, screens for washing and recovery of medium (drain & rinse screens) were limited in width and therefore capacity and, crucially, distributors for feeding multiple smaller cyclones were inefficient.

Increasing the sizes of all this equipment now enabled a look at larger diameter DM cyclones and the trend became a rush.

The DSM rules still applied however, so the larger cyclones still required the same minimum medium to ore ratios, still required all the ratios between components but now, with the larger throat the cyclone could digest a larger top size of coal (the Multotec 1450 mm cyclone is suitable up to 95 or maximum 100 mm coal).

**Large diameter DM cyclones**

The sizes of the cyclones now proceeded to grow through 1000 to 1150, 1300 and even the monster, 1450 mm. These sizes were to have their own challenges however, as South African studies showed a new phenomenon, the so-called "breakaway size". This hypothesizes that the large diameter cyclones are less efficient at washing coal from 10 mm down and that the plant should be split into two fractions:

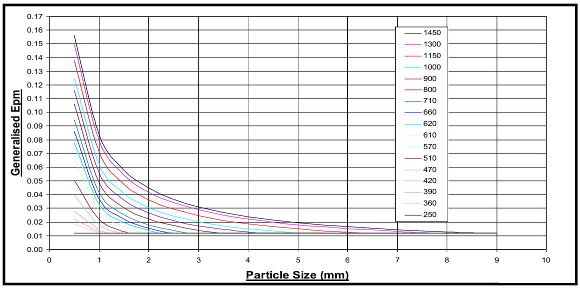



Fig 6
Cyclone efficiency curves

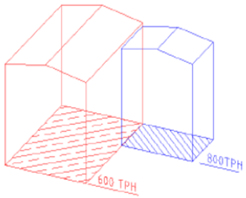
- large coal (plus 10 – 15 mm but less than 95 – 100 mm) and

- fine coal below 10 or 15 mm but above 1.5 or 2.0 mm (the threshold of efficient medium removal/recovery, of dewatering ability and of clean coal to cost efficient return)

Despite the Australians showing that the breakaway size had limited affect in their washeries (Fig 7) [5], the South African investigations showed a definitive improvement in a split process and further that the use of spirals on the -1.5 or 2.0 to plus 100 µm fractions, flotation on the extremely fine coal, produced high yielding plants that are more cost effective (Fig 8). [6 & 7]

| | *Goonyella* | | *Riverside* | |
|---|---|---|---|---|
| | Twin | Single | Twin | Single |
| | 710mm | 1m | 710mm | 1m |
| Pressure(cyclone diameters) | 10 | 11 | 10 | 10 |
| Size Fraction | Probable Error | | | |
| +16 | 0.028 | 0.025 | 0.020 | 0.014 |
| -16+ 4 | 0.030 | 0.026 | 0.025 | 0.020 |
| -4 + 2 | 0.038 | 0.028 | 0.045 | 0.026 |
| -2 + 0.5 (ww) | 0.055 | 0.045 | 0.070 | 0.049 |

Fig 7
Ep comparison
710 vs. 1000 mm DM cyclones




**Structural Volume as a Function of the Feed Capacity**

Fig 8
Capacity

The benefits according to the Australians regarding plant footprint, individual module size vs. throughput, plant operator numbers, maintenance and a host of other things, far outweighed the potential inefficiency losses when Australian coal is considered (Fig 7) [5].

However, South Africa begs to differ and thus still looks to a split flowsheet for best practice.

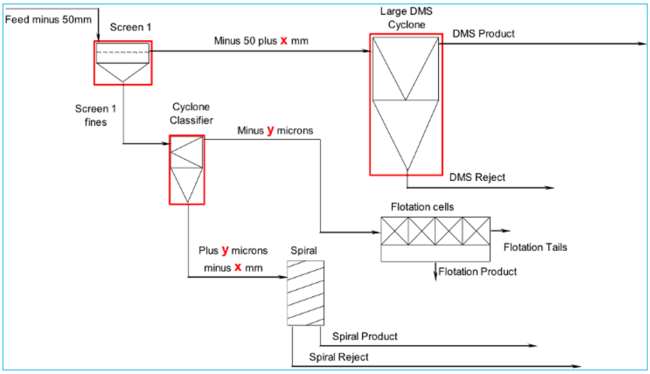



Fig 9
Split process
flowsheet

In the background, the question is still being investigated of high density cuts giving best returns for power station feed-coal.

Grootegeluk had investigated the cut density as early as November 2009 and in a paper presented to the South African Coal Processing Society technical meeting [2], the vexaceous issue of washing at densities of 2.0 and above was examined.

The reason for this is that ESKOM (South Africa's power utility) required a 35% ash product for its Matimba Power Station and this could be washed from all benches of the Grootegeluk quarry at a density of > 2.0 with an acceptable yield (for power station coal) of 47%. Thus the investigation detailed how this difficult to achieve but possible to achieve target was reached.

The conclusion delivered in the paper [2] is high density cuts are possible with existing DM cyclone configurations (both large and small diameter) if:

- densifiers are the key for producing correct medium at 2.3
- the dilute circuit must have sufficient capacity to control magnetite consumption
- ceramic cyclones have the best life/cost performance

The benefits are:

- magnetite consumption does increase at high densities but are insignificant when compared to improved plant yield
- densities of 2.05 are achievable with standard grade magnetite (85% < 45 µm)

| Cyclone diameter (mm) | Plant | Feed size | Average particle size of feed | d50* | EPM* | Imperfection* |
|---|---|---|---|---|---|---|
| 1450 | Newlands[6] | 50 x 8 | 29.0 | 1.644 | 0.0170 | 0.0103 |
| | | 8 x 4 | 6.0 | 1.647 | 0.0262 | 0.0159 |
| | | 4 x 2 | 3.0 | 1.676 | 0.0353 | 0.0211 |
| 1500 | Benga[7] | 50x4 | 27.0 | 1.628 | 0.0114 | 0.0070 |
| | | 4x1.2 | 2.6 | 1.704 | 0.0528 | 0.0310 |
| 1300 | Benga[7] | 50x4 | 27.0 | 1.390 | 0.0121 | 0.0087 |
| | | 4x1.2 | 2.6 | 1.417 | 0.0280 | 0.0198 |
| 1500 | Drayton[8] | 50x31.5 | 40.8 | 1.575 | 0.0160 | 0.0102 |
| | Test 1: Aug 2005 | 31.5x8.0 | 19.8 | 1.581 | 0.0210 | 0.0133 |
| | | 8 x 1.0 | 4.5 | 1.580 | 0.0300 | 0.0190 |
| 1500 | Drayton[8] | 50x31.5 | 40.8 | 1.615 | 0.0200 | 0.0124 |
| | Test 2: Aug 2005 | 31.5x8.0 | 19.8 | 1.600 | 0.0140 | 0.0088 |
| | | 8 x 1.0 | 4.5 | 1.600 | 0.0350 | 0.0219 |
| 1500 | Drayton[8] | 50 x 16 | 33.0 | 1.690 | 0.0130 | 0.0077 |
| | Test 3: Aug 2006 | 16 x 8 | 12.0 | 1.725 | 0.0200 | 0.0116 |
| | | 8 x 4 | 6.0 | 1.690 | 0.0280 | 0.0166 |
| | | 4 x 1.4 | 2.7 | 1.700 | 0.0300 | 0.0176 |

Fig 8
DM cyclone results [7]

**Conclusion**

The design, configuration and materials of construction used in the modern day MAX dense medium cyclones is true to the original DSM concepts but with necessary upgrades and modernization.

The use of large diameter cyclones with the concomitant benefits is here to stay but should be applied with a diligent eye on the type of coal.

Finally, high density cuts are possible, have their place and definitely improve overall plant yield.

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## Production of Shungite Concentrates – Multifunctional Fillers for Elastomers

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**Abstract**
The chemical composition of shungite from "Bolshevik" deposit was studied by energy-element analysis. It was established by X-ray diffraction method that in addition to the carbon component of shungite has a mineral part: hydromica type of illite, gidromuskovit, quartz, dolomite, chlorite, pyrite and siderite. Enrichment by flotation of shungite was carried out to concentrate of carbon and stabilize composition. The possibility of partial or complete replacement of technical carbon by new fillers based on shungite in rubber mixes was studied. The influence of shungite filler on the strength properties of rubber was studied. Strength properties of rubbers filled with shungite are above, compared to the properties of rubber filled with carbon black. Reinforcing effect of elastomers filled shungite is above compared reinforcing effect of the of elastomers filled standard technical carbon black.
**Key words**: Shungite, Elastomer, Enrichment, Carbon, Concentrate, Filler.

Despite the fact that carbon material which is similar to the Karelian shungite was found on the territory of Kazakhstan, it continues to appear in mass media that the world's only field of shungite is in Russia, Karelia. We have been working with it for several years. In a comparative study of local carbon material ("Bolshevik" mine, East-Kazakhstan region) with Karelian shungite and several other well-known carbon materials it was confirmed that it belongs to a class of shungite. Karelian shungite themselves are very different both in structural state and on the supramolecular organization.

Shungite occur in nature as thick shungite rocks, which are densified mechanical mixtures of carbon and mineral agglomerates of various chemical composition [1,2]. Natural shungite is a multi-phase system by results of X-ray analysis (Figure 1) [3].

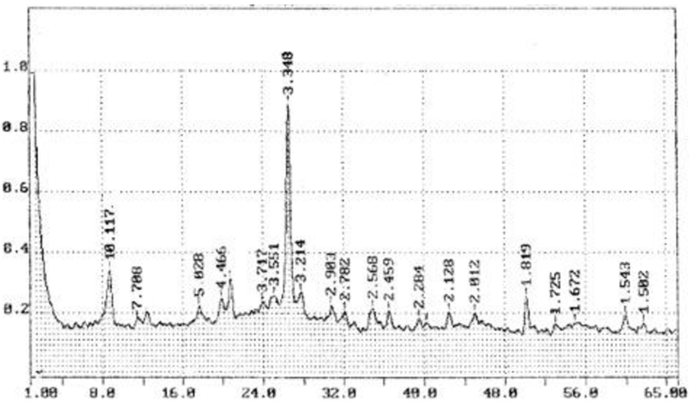


Figure 1 - Radiograph of shungite rock

 
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A number of mineral components were determined by characteristic diffraction peak at 10.1 A (hydromica type of illite , gidromuskovit), besides of carbonaceous material wich diagnosed by reflex with an interlayer distance d ~ 3,5 A. Diffraction reflexes of 4.26; 3.34; 1.37 A testify the presence of quartz in the rock. The diffractograms also demonstrate the presence of dolomite (3.70; 2.9; 2.012 A). Chlorite (3.53 A), pyrite (2.71; 2,41A) and siderite are present in minor amounts [4].

The given rocks are identical in qualitative composition however, are not uniform in a quantitative sense. Carbon content changes widely from a few percent to 44% by weight (Table 1). The heterogeneous composition of the mineral substrate and a wide range of changes in the amount of carbonaceous substances explain a wide variety of shungite differing appearance, physical, mechanical and chemical properties.

Table 1 - Chemical composition of shungite

| No | Component content, % wt. | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Sample | C | $SiO_2$ | $TiO_2$ | $Al_2O_3$ | FeO | $Fe_2O_3$ | MgO | CaO | $Na_2O$ | $K_2O$ |
| 1 | 44.2 | 34.7 | 0.4 | 9.8 | 1.2 | 0.3 | 0.6 | 1.1 | 0.2 | 1.8 |
| 2 | 27.6 | 46.2 | 0.5 | 10.6 | 1.4 | 2.4 | 0.9 | 1.6 | 0.4 | 1.9 |
| 3 | 12.3 | 50.4 | 0.8 | 14.0 | 1.7 | 5.5 | 2.0 | 2.6 | 0.4 | 2.5 |
| 4 | 6.3 | 59.6 | 0.7 | 12.0 | 1.5 | 5.6 | 3.6 | 3.3 | 0.7 | 2.6 |

A necessary requirement for raw materials used in production is constancy of chemical and grain-size composition. Therefore, we carried enrichment by flotation of shungite for concentrating carbon content and stabilize composition of shungite [5, 6]. Chemical compositions of products after enrichment by flotation are presented in Table 2.

Table 2 - Average composition of products after enrichment by flotation

| Products of enrichment by flotation | Component content, % wt. | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | C | $SiO_2$ | $TiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | FeO | CaO | MgO | $Na_2O$ | $K_2O$ | $M_{Al}$ |
| Tailing | 19.1 | 50.7 | 0.6 | 12.3 | 4.6 | 2.0 | 3.3 | 2.8 | 1.0 | 2.2 | 0.28 |
| Concentrate | 43.8 | 32.1 | 0.5 | 8.6 | 2.6 | 2.2 | 2.8 | 0.9 | 1.5 | 1.8 | 0.19 |

Then we obtained experimental batch of the shungite concentrate wich was tested for rubber mix preparation. Pilot tests of carbon concentrate were carried out on the Issyk plant of the rubber products in the workshop of rubber compounds. The formulations of standard rubber compounds used in the processes of the enterprise were taken as a basis in the work.

Batch of the rubber mixes, curing and subsequent study of the physical and mechanical properties of the finished experimental rubbers were performed in accordance with current technological reglaments and methodics of enterprises.

Thus, 120 kg of rubber mix based on Kazakhstan shungite was obtained. Rubber irrigation pipes and insulating mats were vulcanized as the products.

The elastic-strength characteristics of experimental products and products based on standard mix are given in Table 3 [7]. According to these data, physical and mechanical properties of the experimental rubbers substantially above those for the products prepared by the standard formulation (i.e. only filled with carbon black mark P 324), and meet the requirements of GOST 126.

Table 3 - Physical and mechanical properties of rubber dielectric rugs

| Characteristics | Rubber rugs | | | |
|---|---|---|---|---|
| | experimental | | standart | |
| | upper mix | plantaris mix | upper mix | upper mix |
| Rupture strength, MPa | 7.84 | 9.33 | 7.84 | 7.35 |
| Relative elongation at rupture, % | 520 | 320 | 300 | 300 |
| Relative residual elongation, % | 19 | 21 | 25 | 40 |

CPCMRA al-Farabi KazNU has implemented the project of pilot plant construction for the production of carbon materials based on shungite rocks up to 4 million tons per year [8] in the framework of commercialization of research projects supported by the Ministry of Education and Science of Kazakhstan, the World Bank and the Scientific and Technological Center "Parasat". Finely Dispersed Shungite Concentrates (FDSP) with 40 ± 2% of carbon were obtained. Purpose of finely dispersed powders is to be used as a filler of elastomers in rubber and polymers industry [9].

Work on the shungite powders implementation [10] on rubber industries has been going on for a number of years. Powders obtained by different technologies of grinding do not differ in constancy of composition and dispersion. Table 2 presents the main properties of a commercially available FDSP produced by CPCMRA KazNU.

Table 4 - Properties of FDSP

| Indicator name | Norm |
|---|---|
| 1. Appearance | Finely dispersed black powder without impurities. |
| 2. The mass fraction of carbon (%), not less than | 35.0± 2.0 |
| 3. Mass fraction of losses at 105°C (%,) no more than | 1.5 |
| 4. Cinder content (%), no more than | 60.0 |
| 5. Packed density, g/dm$^3$ | 350-450 |
| 6.pH of aqueous extract | 6.5-7.0 |
| 7. Mass fraction of residue,%, after sieving through a sieve with a mesh<br>0.14<br>0.045<br>no more than | none<br>0.05 |
| 8. Dibutyl phthalate absorption, cm$^3$ / 100g | 32 |
| 9. Specific adsorption surface | 23 |

In spite of the fact that developed standard of the Customs Union (Active carbonaceous filler of elastomers - ST TOO 111040004929-04-2015) for manufactured products contains a limited list of standardized indicators, it remains a good basis for the widespread commercial introduction of FDSP in the rubber industry as a filler of polymers.

In the manufacture of rubber compounds filled with technical carbon in combination with active types of FDSP concentrates show properties of technologically active additives. Thus, they accelerate the implementation, distribution and dispersion of fillers, reduce "dusting", increase the ductility and fluidity of elastomeric composites, increase adhesion to metals, reduce the phenomenon of dangling mixture over the roll gap of rubber compositions during their processing on a roller equipment and generally improve performance of millability, the surface quality of the extruded and calendered workpieces. Therefore, the introduction of significant FDSP dosage when used as filler either improves the "processability" parameters of rubber mixes or remains at a level of ethalon.

The obtained data of the effect of high carbon shungite powders on the structure and properties of the elastomeric composites show that FDSP is a perspective component of the rubber mixes with multifunctional actions. It combines properties of the new "diphilic" filler, plasticizer and structural technologically active additives. Certain shungite stocks with more than 30 million tons in each mine are leading to perspective industrial use along with a different stable grades of carbon blacks.

**Conclusion**

1 Researches shown that shungite can be used as reinforcing filler of the elastoplastic composite materials.

2. Strength properties of rubbers filled with shungite are above, compared to the properties of rubber filled with carbon black.

3. Reinforcing effect of elastomers filled shungite is above compared reinforcing effect of the of elastomers filled standard technical carbon black.

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