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Рецензенти:

М.С. Четверик – доктор технічних наук, професор, завідувач відділу геомеханічних основ технологій розробки родовищ Інституту геотехнічної механіки ім. М.С. Полякова НАН України (м. Дніпропетровськ);

В.І. Голінько – доктор технічних наук, професор, завідувач кафедри аерології та охорони праці ДВНЗ «Національний гірничий університет» (м. Дніпропетровськ);

Л.М. Ширін – доктор технічних наук, професор, завідувач кафедри транспортних систем і технологій праці ДВНЗ «Національний гірничий університет» (м. Дніпропетровськ).

Т38 **Selective** mining technique for thin coal seams :monograph / V.I. Buzilo, A.G. Koshka, A.V. Yavorsky, E.A. Yavorskaya, L.A. Tokar, V.I. Sulaev, V.P. Serdyuk; Ministry of Education and Science of Ukraine; National Mining University. – D.: NMU, 2015. – 131 p.

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Study devoted to the issues of basic parameters and application technology of selective mining technology of thin and very thin coal seams. The main parameters of selective technology of thin coal seams are determined in a result of carried out investigations. The principal design schemes of selective technology with undercut wall rocks based on the using of existing cutter technology is developed.

The monograph is for engineers, employees of higher educational institutions, research institutes, and engineering companies of coal industry.

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INTRODUCTION

Coal industry of Ukraine faces two major problems. One of them is comprehensive development of overall mechanization providing reduction of production cost and labour intensity of coal mining; another problem is improvement in mined coal quality. The problems, especially those, concerning thin seams, contradict. The matter is that normally their mining mechanization is followed by forced enclosing roof and floor undercut and, respectively, increases in ash-content to be one of the most important characteristics.

A post-event analysis shows that lately ash-content of coal mined, loaded, and used has experienced constant increase. And one of key reasons of such a situation is wide use of complete mining which results in mined coal pollution by diluting waste rocks.

Decrease in ash-content of coal while mining is well-known but insufficiently studied technique of coal quality improvement. It should be done by means of transition from complete mining to selective one, involving separate mining as well as transporting mineral and rocks. Moreover, the technique has been developed for medium thickness seams, and its basic parameters are sufficiently researched and scientifically grounded. As for the thin seams mined with enclosing roof and floor undercut, the problem stays to be open. Lack of adequate scientific grounding of operational parameters and structural schemes prevents developing more sophisticated technologies and machine systems to provide high-performance mining thin and very thin seams. Hence, solving the problems is actual task being of great importance for coal industry of Ukraine.

Staff of Underground Mining Department of the State Higher Educational Institution "National Mining University" and mines of Western-Donbass and Lvov-Volyn coal field took part in field studies of mine pressure. Authors highly appreciate them. The authors also extend appreciation to D.S. Malashkevich and E.V. Aksionov, students of State Higher Educational Institution "National Mining University", for their assistance in designing the monograph.

CHAPTER 1. STATE- OF-ART AND THE RESEARCH TASKS

1.1. State-of-art

Over 80% of commercial reserves are concentrated in Ukrainian mines which thickness is less than 1.2 m. The reserves are largely represented by high-rank coking coals (30%) and thermal coals [12].

To meet demands for coal, thin seams and very thin seams are involved in mining [43, 60, 66, and 84].

Narrow-web shearers applied with powered support (less frequently with single prop supports), plow equipment and scraper one are the mechanical means to mine the reserves. Within the seams which thickness is less than 0.7 m, mining takes place with enclosing roof and floor undercut. In some mines, where such seams are mined without undercut, wide-cut loaders, developed more than 50 years ago, are used [12,73]. In this context, daily face output is 2.5 less to compare with complex powered face. Moreover, labour intensity of such longwalls is extremely high.

Search results of new, including unconventional techniques of flat coal seams which thickness is less than 0.7-0.8 m prevent from ideas of their soon implementation [51,68,77]. Meanwhile, overall and powered narrow-web mining becomes more and more popular for thin seams. According to expert opinion [6, 61, 77], in the foreseeable future powered equipment to mine specified seams will be without a rival. However, powered support has bounded application area. Thus, papers [19,48,70,93] assess its lower application boundary within 0.75…0.8 m. Analysis of face faces activity in Ukrainian mines shows, that actual data of lower boundary is some higher being 0.95-1.0 m for complex KD80 support, 0.90-1.0 m for complex KMK97 support, 0.90-0.97 m for complex KD90 support, and 0.90-0.95 m for complex KM103 support. That is putting into operation complexes of new engineering level did not make it possible to extend substantially a sphere of overall and powered mining [102].

According to the data by Donetsk Coal Institute (DonCI) [62], plowing and scraper mining which provide activities with 0.7 m seams without enclosing roof and floor undercut has restricted field of application: a little more than 21% of faces of Ukrainian mines. Thereby, according to [62], due to various reasons nearly all coal seams in mines of Donbass and Lvov-Volyn coal field can not be mined by ploughs and scrapers.

Due to difficulties of overall mechanization of mining within thin flat seams, abroad they repaired to significant reduction or even abandoned to mine such seams. Thus, Germany despite having considerable coal reserves in thin seams (about 50%) [11, 49, 95], does not mine seams which thickness is less than 0.7 m; as for the seams with 0.7-1.0 m thickness, they have restricted number of faces [9, 29, 47, 66]. All the faces were equipped with plough facilities and powered supports of shield type. They are widely used for thin seams in Germany as free well-ploughed coal is available in the mined longwalls as well as lack of narrow-web cutter-loaders for seams which thickness is less than 0.8 m [11, 25]. However, significant reserves are in seams which mining is only possible by cutter-loaders. Accordingly, interest is taken in thin seam shearer mining [11] including those with enclosing roof and floor undercut [9, 10].

In Great Britain, seams having up to 0.91 m thickness are considered as thin ones. About 8% of total quantity of longwalls work for the seams [4, 47], and coal mining is 5-6% of total production despite reserves of thin seams in certain coal regions of the country are more than 50% [12] of total reserves. Thin seams are mainly mined by cutter-loaders with enclosing roof and floor undercut. Average undercut value is 15-20 m, and often mining thickness is more than 1 m (1.03-1.07 m) [4, 47]. Meanwhile, cost reduction of 1 ton of coal is 20% and more owing to its dilution.

In Czech Republic thin seams with 0.8-1.2 m thickness contain almost 42% of high-grade coking coal; however, winning is not more than 18% [47, 103]. The use of cutter-loaders able to break any coal is limited by minimum thickness (0.75-0.8 m). Hence, the problem of longwall mining of less thickness seams stays to be unsolved.

As the world practice of thin coal seam mining shows, modern means of overall mechanization give ability to mine seams of less than 0.8 m thickness under in favourable conditions (in cutter-loader longwall faces). Moreover, the factor experiences 0.9-1.0 m increase in terms of mining and geological environment deterioration. Thus, paper [16] notes that favourable environment of deposits in Great Britain makes it possible to use EDW170LN cutter-loader for thin seams starting from 0.85 m. That very time under the conditions of Ruhr coal field (Germany) the cutter-loader application for seams with less than 1 m thickness is impossible without enclosing rocks “seizing”.

Taking into consideration above-mentioned, one may conclude that after long-term efforts to lower the bottom of powered support use, in the world practice it is 0.7 m for plough longwalls [11, 47] and 0.76-0.8 m for cutter-loader ones[17, 30]. Thinner seams are not practically mined.

Some experts believe that in our country the highest rates can be obtained if flat seams with less than 0.8 m thickness will not be mined. However, as the analysis by DonCI [68, 70] shows, the idea would be resulted in almost 38% decrease of balance reserves of productive mines. When it happens, annual output of deficient coal grades would experience 41,000,000 tons reduce. To compensate the output owing to new mines construction, it would involve considerable investment and a period of 10-15 years. Hence, to provide high efficiency of mining coal seams which thickness is than 0.8 m, it is recommended [22, 65, 70, 78, and 93] to apply a technique of complete mining with enclosing roof and floor undercut being widely used lately for thin and very thin seams.

Up to 300 longwalls a year worked with undercut in Ukrainian mines [20], and annual increase in ash-content due to undercut was more than 0.4%. Ash-content of mined rock mass was constantly growing; as a result, its volume increase was more than 10,000, 00 tons [66]. Coal dilution in Western Donbass and Lvov-Volyn coal field resulting from undercut was 19.4 and 5.7% accordingly [61].

Increase in coal dilution is one of the basic reasons of decline in production facilities utilization and cost-performance ratios of Ukrainian mines performance. In addition, constant increase in ash-content of coal received by concentration mills, affects dressing process. As a result, output of commercial coal reduced by 10.4%; that very time their ash-content experienced 12.9 to 16.4% increase; concentration ratio reduced by 8.1%, and its ash-content increased from 9.3 to 16.3% [38, 41]. In turn, grade degradation of products from mines and concentration mills results in performance degradation of consumers. Therefore, a problem of a contraction of the solid volume mined with coal is severe. Urgent necessity to solve it is specified by technical-and-economic requirements as well as by requirements to cut overhead costs while coal mining, transporting, and dressing.

Every year funding for environmental protection increases as the number of waste piles is ever-expanding, and it makes a complicated problem to maintain them and liquidate. They are more than 1300 within Donbass, and many of them burn polluting the air [8, 27]. The solid from mines occupy huge territories suited for agriculture. In Donbass, total area occupied by waste piles is more than 3000 hectares [5]. A lot of human and material resources and engaged in the solid complex transport services (40% of electric locomotives and mine cars; 35% of underground transport workers) [87]. It takes much financial facilities, material, and manpower to bring to the surface 1 t of the solid [90]. In addition, up to 0.3 t of the solid is brought to grass a ton of coal [97].

Colossal material resources and labour ones are applied to mine such quantities of mineral mass together with fuel. Even now mines have reached their frontier in bringing the solid to the surface [87]. If adequate steps are not taken to discontinue increase in waste rock mined volumes, and then to scale it down, the problem will become the factor which will assert determining influence not only on mining industry progress but also on successful performance of mines.

Wide use of complete mining thin seams and very thin ones with enclosing roof and floor undercut is one of the key reasons for increasing total volume of waste rock mined.

Coal industry of our country as well as coal industries of a number of foreign countries takes measures to reduce waste rock mined, or, if it is technically possible, to implement operation schedules which prevent bringing waste rock to the surface [24, 25, 32, 57, 58, 82, 92]. Operation schedule of selective mining thin and very thin coal seams [35, 42, 86], proposed by us, is one such schedules. It provides separate mining and transporting coal and waste rock undercut. Besides, it provides using waste rock undercut for longwall stowage, improving mined coal grade, and increasing cost-performance ratios of coal enterprise performance.

1.2. Studying papers on mining thin seams with enclosing roof and floor undercut.

One of the key problems of thin and very thin seams mining is quality control of mined coal for which its ash-content is basic index. Its recent constant increase can be mostly explained by prevalence of complete seam mining with roof and floor undercut [46, 53, 61]. The key advantages of such mining are: relative simplicity of the technique and organization of work, ability to develop and implement simplified and high-duty powered systems, improved spatial labour conditions etc. [69, 96]. However, it also has a number of disadvantages. Among them are: decrease in feed rate and in cutter-loader capacity, and increase in tool consumption. But the most important thing is great dilution of coal mined which results in its grade deterioration, increase in traffic flow and, accordingly, cost escalation both in mine and at the surface, extra difficulties connected with waste rock dressing and yarding etc. [68, 69, 96].

Studies of papers on seeking rational techniques of powered mining thin and very thin seams make it possible to divide their authors into the three main groups. Representatives of group one [14, 70, 93] consider complete mining with enclosing roof and floor undercut as a nostrum. In their opinion, it is economically feasible to mine thin and very thin seams as they contain lion's share of reserves. Besides, they believe that complete mining has technological and technical advantages. To avoid the main disadvantage of the technique, i.e. significant increase in ash-content of the coal mined, they propose to extend available concentration mills, or build new ones. In this context, undercut degrees are not taken into consideration. However, as it follows from papers [20, 41, 85], dressing possibilities are limited. Uncontrolled and ill-founded increase in undercut and, accordingly, ash-content of the coal mined, result in deteriorating performance of concentration mills and other consumers of coal industry products [20, 31, 38, 63]. For this reason, solving the problem of seam mining technique choice, one should take into account enclosing roof and floor undercut degrees as the parameter is of primary importance for identifying rational parameters of areas for complete and selective mining.

Lately expenditures connected with waste rock transporting and dressing [31] have experience great increase, deteriorating mined coal and concentrate grade have taken place [20, 85] etc. Today it is possible to say that implementing overall mechanization for thin and very thin coal seams did not satisfy expectations. Recently loads on overall powered longwalls have dropped, and in some cases they are no more than loads on longwalls with single prop support; what is more, grade of rock mass mined in them is several times less. But the rock mass should also be brought to the surface, transported to concentration mill, and dress. Besides, waste rock should be removed and stockpiled at the surface. That is before miners ship their goods to consumers, they have to make extra colossal power, human, and material efforts. To avoid that, State Higher Educational Institution "National Mining University", DonCI, "Dongiprouglemash", and "Dneprogiproshakht" underway their efforts to develop efficient technique of thin and very thin seams selective mining to separate coal from undercut waste rocks at the stage of mining.

Falling back on above-listed groups of authors, mention that representatives of group two [7, 71, 80, 82, 91] speak for the necessity to design winning equipment fitting in mined seam thickness as well as for liquidating undercut as the key factor of the coal mined dilution. There is no denying that the latter comment is reasonable.

However, it should be noted that current level of mining machines can not give us such a chance. It will take many years to design new equipment fitting in thin seams. Bu the problem is that thin and very seams should be mined today. It especially concerns Donbass. From the viewpoint, technique of selective mining is ideal from the viewpoint of economic feasibility as it is based on the use of available systems of mining machines making it possible to avoid if not undercut but coal dilution.

Group three of authors [31, 33, 96] stands for so called golden mean of undercut intensity providing least mining cost, and the grade meeting the requirements of consumers. Their reason is that not always complete liquidating undercut is the best alternative.

We believe that such arguments are sound. However, under the conditions of thin and very thin coal seams it is not always real thing to limit undercut with the help of so called golden medium. It can be explained by the fact that minimum undercut is mainly limited by minimum mined thickness of the seam. As a result, a value of "required" undercut often goes beyond economically feasible golden medium.

We think that to provide required grade of coal it is more expedient at some stage to determine rational proportion of complete mining and separate mining for specific mine. That is for specified thickness seams it is more expedient determining ideal proportion for complete and separate mining in longwalls with undercut rather than searching for hardly determinable and sometimes practically impossible golden mean of undercut for the longwalls.

The idea of separate mining mineral and waste rock is not new at all. It was mostly implemented in the process of mining medium thickness seams with waste rock interlayers. For example, some mines of production association "Karagandaugol" [34] as well as mines of far-abroad countries applied downward technique of coal bench and waste rock mining (as they rest) by series-produced cutter-loaders. Shale deposits [72] applied a mining technique with initial shale mining and following intermediate rock mining on return travel of cutter-loader. In the process substantial improvement of the mineral grade as interlayer rocks could not dilute it.

Specific systems and cutter-loaders for selective composite seam mining were designed in the late 1960-s – early 1970-s [39]; they made it possible to perform simultaneous mining of all benches. However, field tests of the equipment were not successful.

There are several incomplete versions of selective mining various procedures for thin coal seams mined with enclosing roof and floor undercut. Thus, A.A. Skochinski Institute of Mining Art [99] proposes a technique of selective mining by cutter-loader having spaced end organs (Fig. 1.1).

In the process coal and rock mining along the length of a face takes place with their simultaneous parallel transportation with following coal delivery into conveyer face; rock is delivered into worked-out area.

Our objective is to improve the grade of mined coal. However, to implement the idea, it is required to have a conveyer with horizontally spaced branches. At the same time it is practically impossible to compose it with powered support. Furthermore, extra difficulties start up in the process of cutter-loader control through a conveyer line etc.

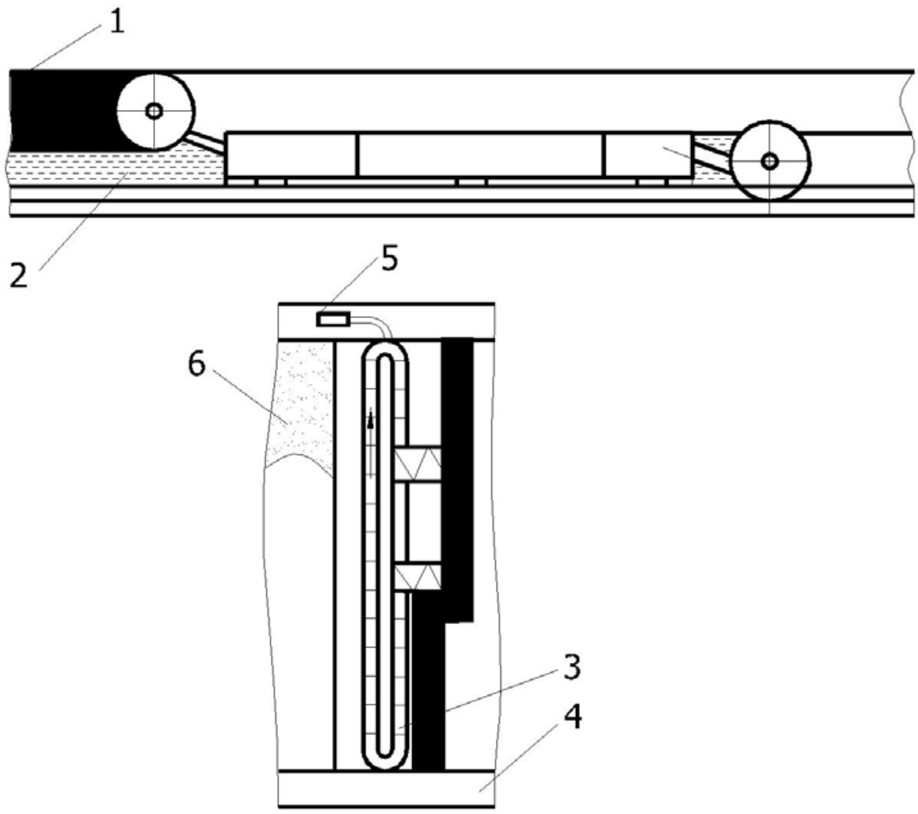


Fig. 1.1 – A technique of thin coal seam selective mining (A.A. Skochinski Institute of Mining Art): 1 – coal seam; 2 – undercut waste rock; 3 - conveyer; 4 – conveyer face; 5 - stower; 6 – stowed pack

Karaganda Research Coal Institute (KRIC) [34] developed techniques of selective mining thin seams with false roofs (Fig. 1.2, a) and soft floors (Fig.1.2, b) by such powered systems as KMK-97 and 1MKM with 1K101 cutter-loader.

The work technique provides selective coal and undercut enclosing roof and floor mining in two passes of cutter-loader. If rocks of false roof are undercut, then their preliminary mining (Fig. 1.2, a) is geared to the capacity of series-produced end organs of cutter-loader or specifically undersized ones. Floor undercut is performed after coal seam has been mined in the process of cutter-loader return pass (Fig. 1.2, b).

Dongiprouglemash [37] proposed a technique of thin seam selective mining when coal breaking, undercut, and rock transfer to stower in a face takes two passes of cutter-loader – during forward travel and reverse one (Fig. 1.3).

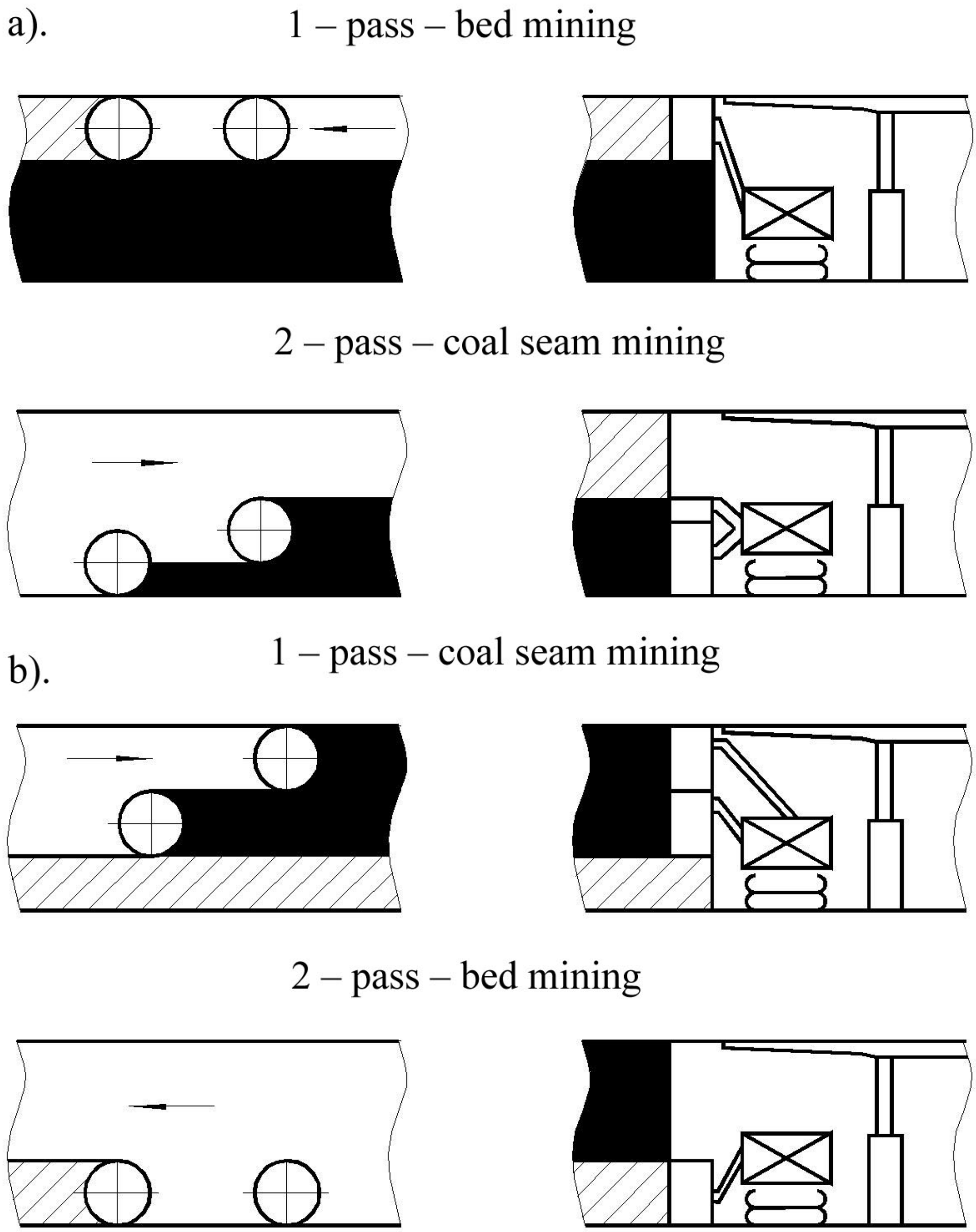


Fig. 1.2 A technique of coal seam selective mining Karaganda Research Coal Institute (KRIC)*: a) with false roofs; b) with soft floors*

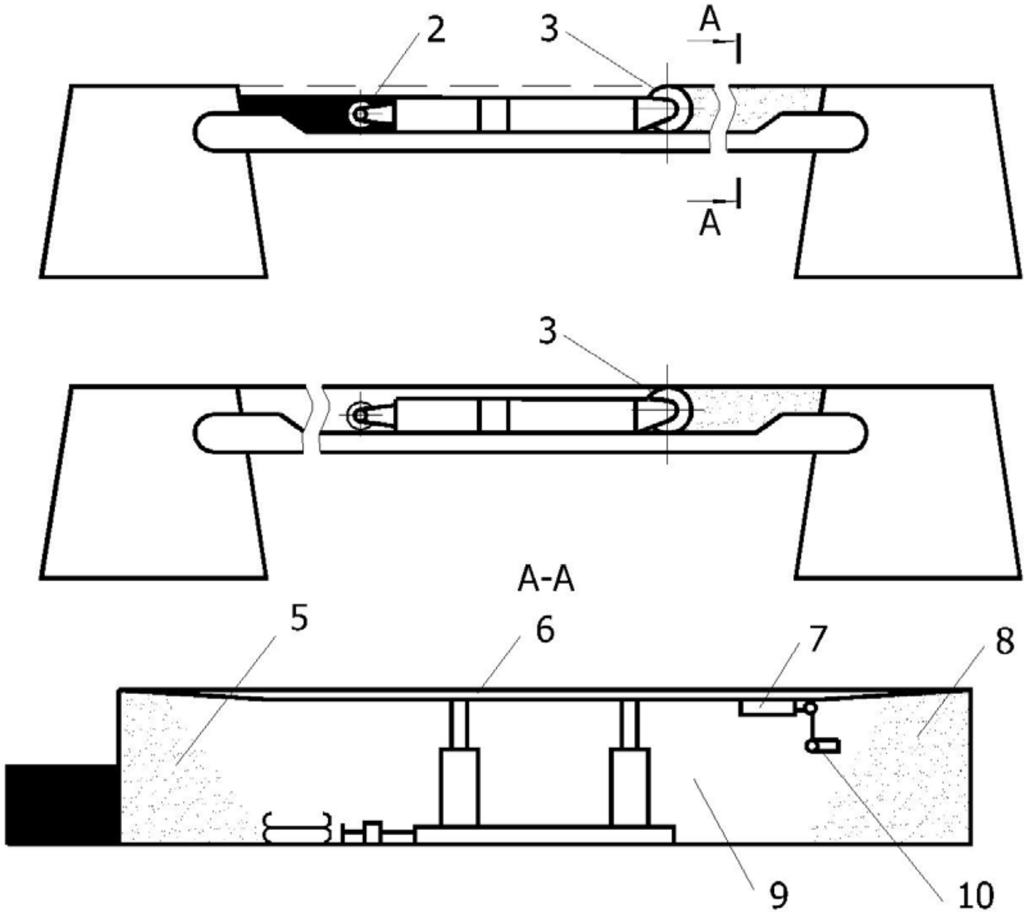


Fig. 1.3. A technique of thin seam selective mining (Dongiprouglemash):1 – cutter-loader; 2 – leading (coal) auger; 3 – backward (rock) auger; 4 - conveyer; 5 – face space; 6 – powered support; 7 - hydraulic advancing cylinder; 8 - filling mass; 9 – rubble-neighbouring area; 10 - stowing pipe

In the process of forward travel, coal breaking as well as its loading on a conveyer is performed by a leading (coal) auger controlled in steps depending upon the seam thickness. Simultaneously backward (rock) auger undercut either roof or ground; then, loosened rock is dumped in a face space. When the cutter-loader is run to start position, both augers perform dumped rock loading on face conveyor. The rock is delivered to a stower; then, it is stowed into mined-out space of double longwall.

From the mid of the 1970-s, DonCI made efforts to motivate rational spheres of complete and selective mining for thin and very thin seams with enclosing roof and floor undercut to compare with available wide-web winning. The new technique, basing upon application of such powered systems as KMK97 and "Donbass", had merits and demerits. Selective mining provided operations of cutter-loader as for coal and then rock continuous mining [54]. In this context it was proposed to leave rock in mined-out space using stowing system mounted in a haulage gate. Later, in the 1980 s, the technique (Fig. 1.4) was implemented on "Sukhodolskaia" mine, Liutikov colliery group (“Krasnodon” association), and mine No. 21 (“Sovetskugol” association).

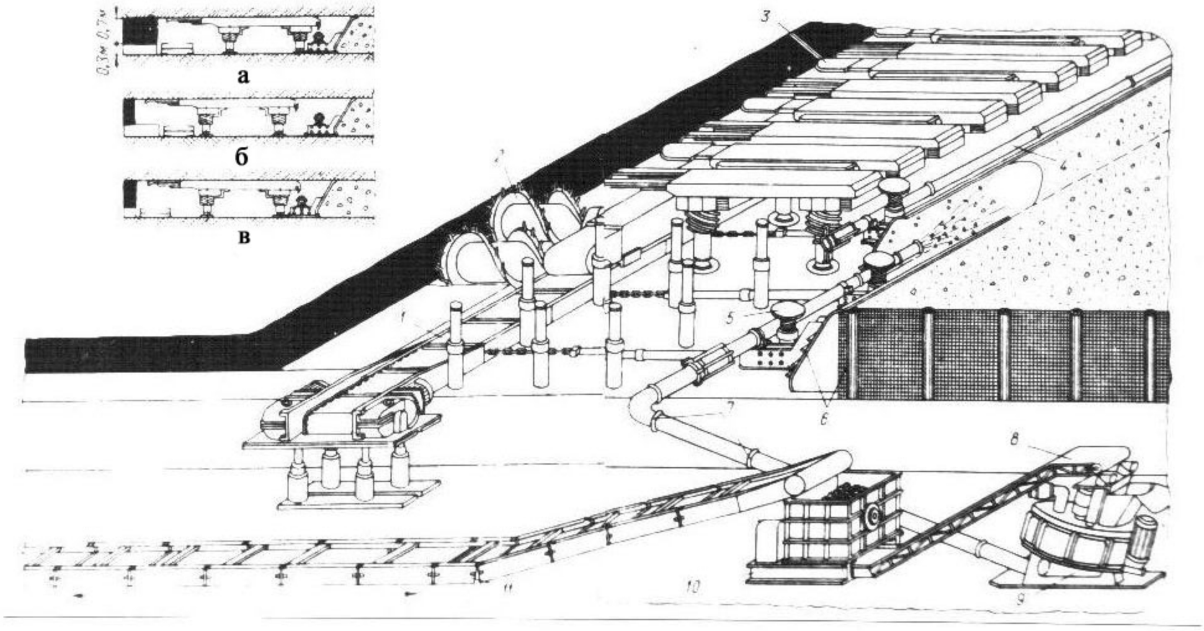


Fig. 1.4 A technique of seam selective mining (DonCI)

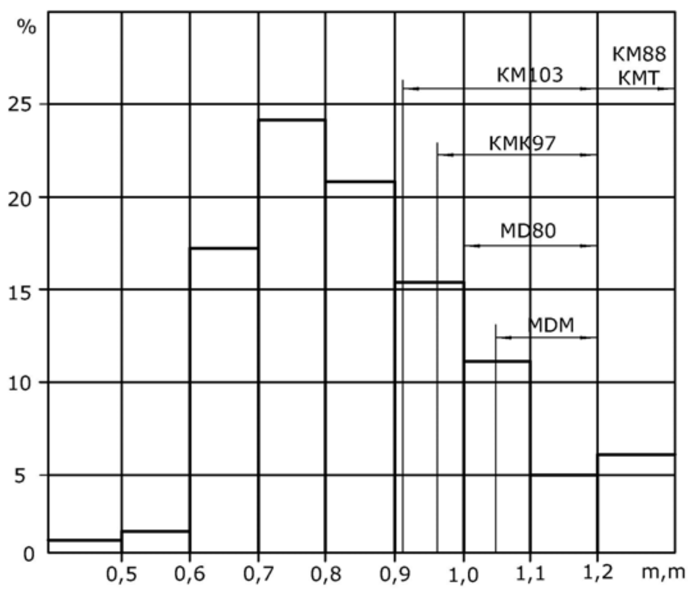
The research results confirm possibility in principle of coal undercut rock selective mining with the rock stowing into mined-out space by means of pneumatic technique. However, the conclusion is made that the process of a seam mining by available cutter-loaders with auger-type end organs is ineffective, and the technique has very limited field of use. As for rock stowing into mined-out space, movable pipeline and stowing operations mine testing show that largely the process solutions meet specified requirements. Besides, they may become a source for designing special-purpose stowing equipment [54].

Accordingly, research by DonCI shows that the problem of developing new efficient selective mining technique for thin and very thin seams stays to be open. That can be first explained by the fact that the operations did not have adequate rationale. Study of the works as well as literary review explains that while developing and implementing the technique of selective mining thin and very thin seams, only papers by DonCI make comprehensive analysis of the new technique certain parameters concerning undercut of floor rock [34, 37, 99] and false roof [34, 76]. Principally, they are poorly known. Even today there is no shared vision concerning minimum of mined seam thickness for specific powered support. However, the parameter is one of the most important while mining thin and very thin seams as it identifies undercut value in a longwall, and, consequently, works upon choice of a technique. Up to now principles of technique for thin and very thin seam selective mining have not been developed. The problem of determining rational fields and application extend for complete and selective mining thin coal seams with enclosing roof and floor undercut has not been solved yet. Available techniques [59, 67, 78, 79, 83] can not consider features of selective mining and its critical parameters. And that prevents from making evaluation of economically feasible areas and application extend for various techniques to be applied in certain mine environment, choosing the most efficient seam mining plans as well as waste recovery methods for undercut enclosing roof and floor, determining rational ways for mined product loading taking into account its grade. The most complete consideration of the problem and other ones is available for mining medium thickness seams with dirt beds [1, 2, 3, 40, 88]. Nevertheless, available projects can not be used in pure form in the process of a technique improving and grounding basic parameters of thin seams with enclosing roof and floor undercut as both technique and its application environment differ.

1.3. The problem actuality

Three fourth of Western Donbass and Lvov-Volyn coal field reserves are concentrated in seams which thickness is less than 1.0 m. Fig. 1.5 demonstrates diagrams of the reserves distribution according to their seam thickness; it also shows areas of available mining equipment application.

0.5 to 0.65 m thickness range was mined by longwalls equipped with off-market wide-web cutter-loaders CCTM).



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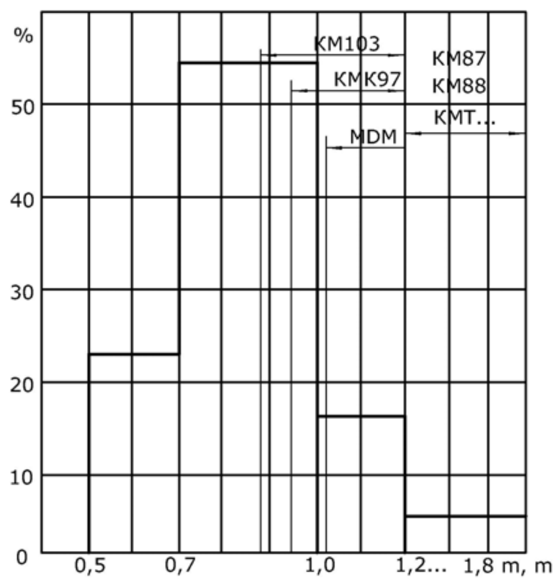


b

Fig.1.5 Diagrams of reserves distribution on their seams thickness: a. In mines of Western Donbass; b. In mines of Lvov-Volyn coal field

Such a technique is characterized by relatively low efficiency and high labour intensity. KD80, KD90, KM103, and KMK97 powered systems (with KA80, KA90, or 1K101UD cutter-loaders) were widely used for seams with 0.65…1.0 m thickness. For operational stability and maintenance of the systems under the conditions of mines within specified coal-mining regions, mined thickness of a seam should not be less than 0.90…1.05 m.

Both available and implemented systems of new engineering level, including foreign-made (DBT, Ostroy etc.), can mine seams which minimum thickness is 0.9…1.1 m. Forced undercut of enclosing roof and floor is applied while mining thinner coal seams. In certain cases, undercut may reach 40…50 cm being the main reason of mined coal dilution. While overall mechanization implementing, amounts of rock mass mined in longwalls with undercut as well as the total number of such longwalls was increasing year on year. Fig. 1.6 demonstrates their growth dynamics in terms of Western Donbass mines. In 1978, forty-eight longwalls worked there with undercut; their output was about 50% of total production. They were 116 in 1986, and their output was almost 80% of "Pavlogradugol" production. Accordingly, ash-content of mined rock mass was increasing (33.5% in 1978; 43.9% in 1986); it was 60 and more per cent in certain longwalls; in terms of sheet one it was 8 to 12%. Total coal dilution was 32%, including about 25% due to the undercut of roof rocks or a seam floor.

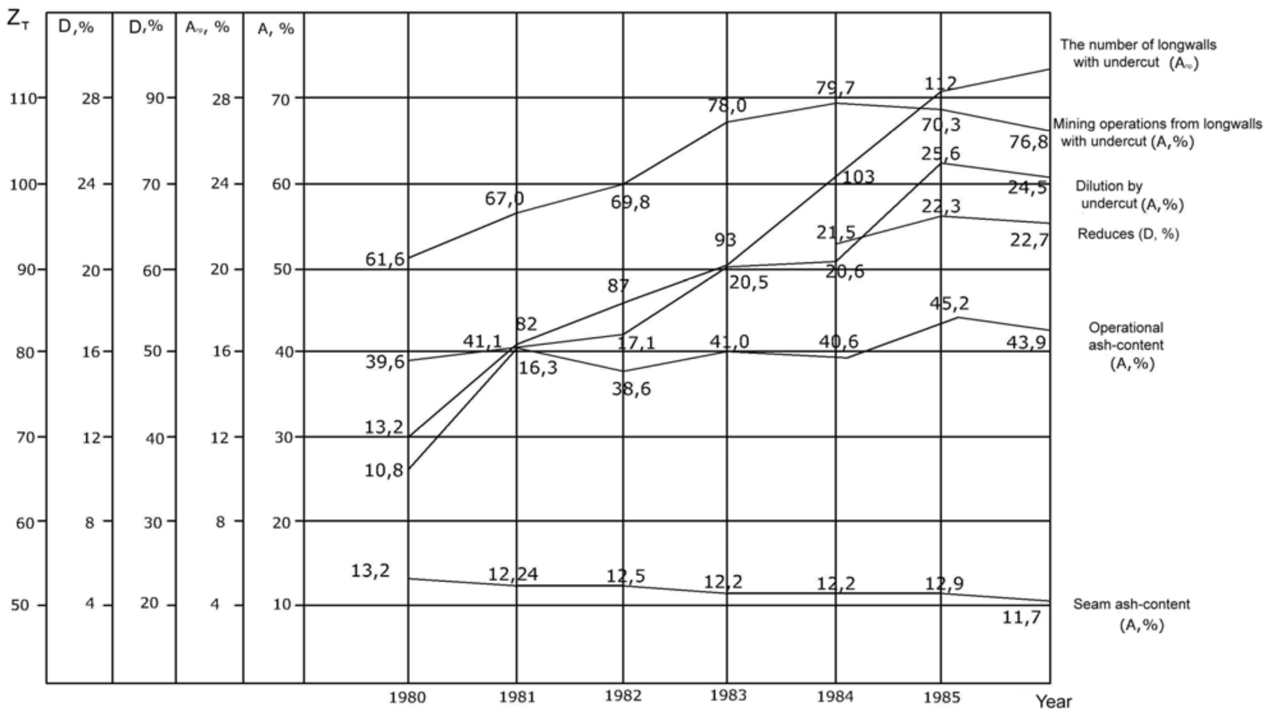


Fig. 1.6 Operational dynamics of longwalls with enclosing roof and floor undercut in Western Donbass mines in the period of overall mechanization implementing

Comparable situation is in mines of Lvov-Volyn coal field (Fig. 1.7).

Increase in ash-content adversely affects on the main aspects of mines performance; first of all, it concerns profit and economic viability. Synchronously, quality loss of washings takes place; in addition, their output cuts, and expenditures connected with dressing and transporting extra volumes of rock increase. In such a way, from 1980 to 1986 ash-content of rock mass delivered to concentration mills "Pavlogradskaia" and "Chervonogradskaia" experienced 9.8% and 6.4% increase accordingly and processing cost of 1t of concentrate increased from 20.47 RUB/t to 30.14 RUB/t , and from 12.31 RUB/t to 19.25 RUB/t.

Due to increase in ash-content, volume of rock mass decreases as each percent of planned ash-content excess results in 2.5% of output cut. Actually, every enterprise mining seams which thickness is less than 1.0 m, gives its consumers production of one or two longwalls; in terms of Association, output of one or two mines is "eaten away". For example, in 1986 cuts in "Pavlogradugol" PA were 2.5 million tons or 22.7% of total production. In "Ukrzapadugol" PA they were 2.2 million tons and 17.3% accordingly.

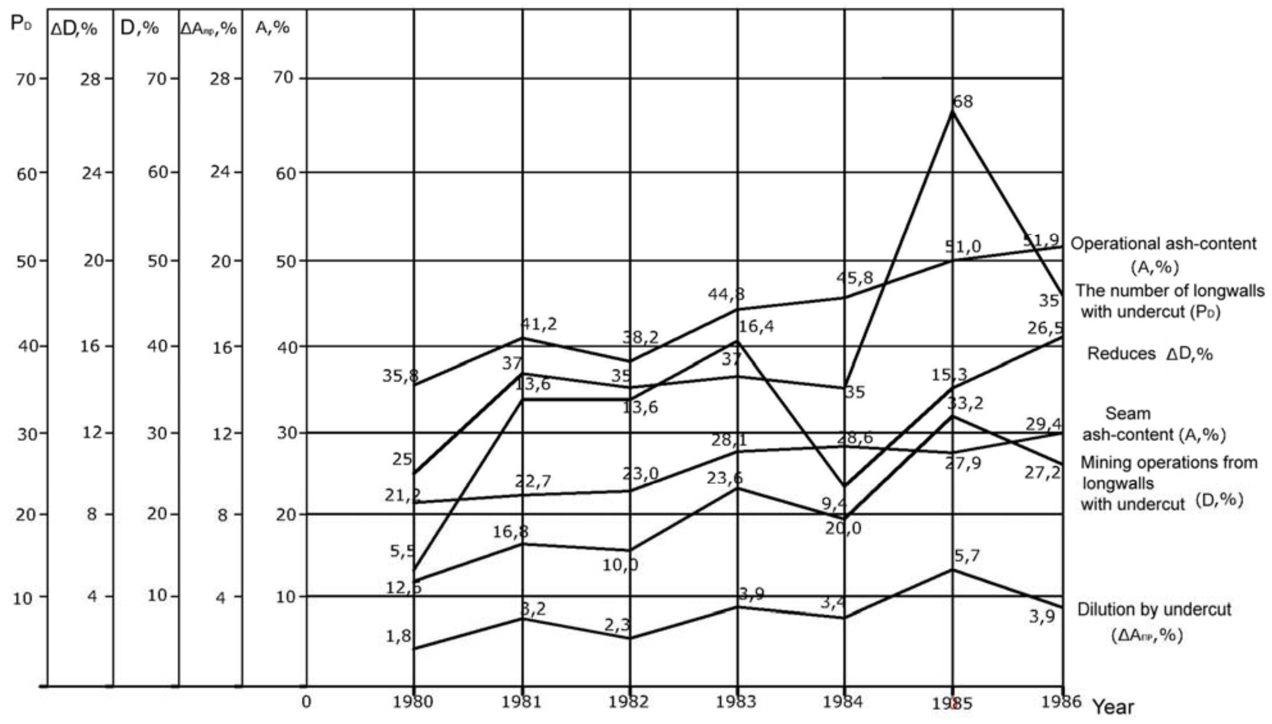


Fig. 1.7 Dynamics of operation of longwalls with enclosing roof and floor undercut in mines of "Ukrzapadugol" Production Association

This points to the fact that it is required to solve a problem of developing new techniques which provide such coal seams mining ($m \leq 1.0$ m) without dilution. The In collaboration with "Giprouglemash", "Dongiprouglemash", and "Dneprogiproshakht", the NMU developed such a technique allowable to the conditions under consideration.

The technique makes it possible to avoid three negative tendencies in mining industry helping to improve the grade of mined products, ergonomics, and environmental friendliness of mining practice on the whole. In addition, it will make it possible to mine very thin seams regarded as non-commercial reserves, and that will extend service life of a number of mines. And the fact is very important force for both considered coal-mining regions, and other ones.

Conclusions

Studies of technical and patent literature demonstrate that the problem of selective mining thin coal seams stays to be poorly explored to compare with medium thickness seam. Few papers concerned the problems are mainly engaged in developing certain procedures of selective mining without their parameters justification in their application area.

Basing upon studies of sources and real production situation, the key activities are determined. The question is scientific and technical reasoning the main parameters of thin coal seam selective mining, having used the information to develop principles of new technique and to determine rational areas and volumes of various mining procedures as applied to specific mine.

The work objective is to determine new rules and dependences required for justifying new parameters and application area for thin coal seam selective mining.

To succeed, following problems were set and solved:

- Preparing scientific and technical reasoning for the main parameters of mining in longwalls with enclosing roof and floor undercut;
- Developing principles of thin and very thin seam selective mining;
- Determining actual parameters of complete and selective mining seams with enclosing roof and floor undercut;
- Developing calculating method for rational areas and volumes of various mining procedures as applied to specific mine.

To solve the problems, complex research method involving analytical and field research as well as economic and mathematical modeling with the use of computer technology was applied.

CHAPTER 2. REASONING PARAMETERS OF SEAM SELECTIVE MINING TECHNIQUE

2.1. Common data

In Ukrainian mines balance reserves involve coal seams which thickness is 0.5 m and up. In productive mines, extracting seams which slope angle is up to 35°, balance reserves of A+B+C_1 grade are almost 9.2 billion tons. 17% of them are concentrated in seams which thickness is more than 1.2 m, 46% - in seams which thickness is 0.8 to 1.2 m, and 37% - in seams which thickness is less than 0.8 m. in addition, almost 2.6 billion tons of coal reserves are considered as non-commercial ones reserves according to their thickness.

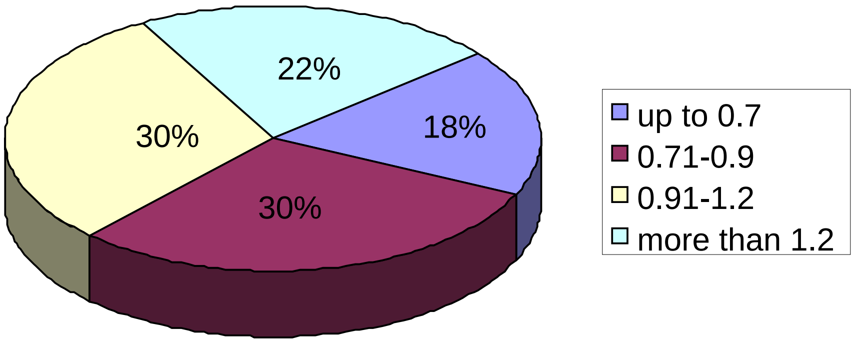
Thus, the majority of balance reserves of coal (about 7.6 billion tons or 83%) is concentrated in seams which thickness is up to 1.2 m, including 3.2 billion tons (42%) in seams which thickness is less than 0.8 m.

About 90% of all balance reserves of coal is deposited in seams with up to 18°slope angles. Table 1.1 and Fig. 1 demonstrate distribution of gently dipping (up to 35°) seams according to their thickness and slope angles. Table 2.1 and Fig.2.1 data show that of total quantity of seams – 566 or 77.8% which thickness is up to 1.2, including 129 or 17.7% – up to 0.7 m, slope angles of 71.2% of seams are up to15°, and 52.7% of them – up to 10°.

Table 2.1 Distribution of gently dipping (up to 35^0) seams in Ukraine according to their slope angles and thickness

Seam thickness, m															
Up to 0.7				0.71-0.9				0.91-1.2				Over 1.2			
Slope angle of a seam, degrees															
Up to 10	10-15	15-25	25-35	до 10	10-15	15-25	25-35	Up to 10	10-15	15-25	25-35	Up to 10	10-15	15-25	25-35
Ukrainian Donbass															
69	20	29	10	100	40	49	9	93	45	55	11	58	40	26	19
Lvov-Volyn coal field															
1	-	-	-	18	-	-	-	17	-	-	-	19	-	-	-
Total on mines in Ukraine															
70	20	29	10	118	40	49	9	110	45	55	11	77	40	26	19

a)



b)

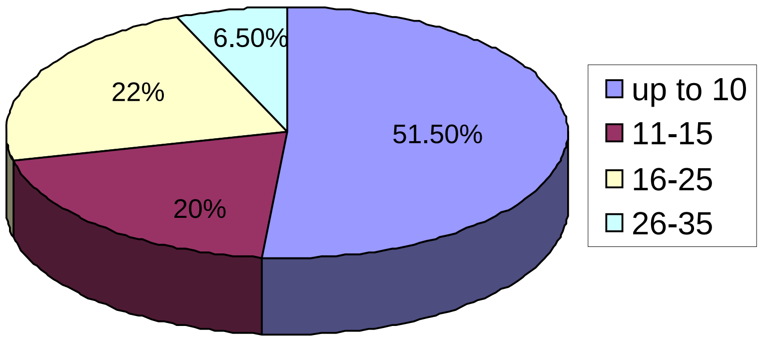


Fig. 2.1 Distribution of seams in Ukraine according to their thickness (a) and slope angles (b)

Having analyzed qualitative and quantitative composition of coal reserves in Ukrainian mines, one may say that the problem of thin coal seams effective mining stays to be urgent. The problem is the most severe in Western Donbass where the balance coal reserves are mainly concentrated in seams which thickness is less than 1 m.

The chapter makes justification of the key parameters for gently dipping thin seams with enclosing roof and floor undercut mining technique; besides, principles of selective mining thin and very thin seams are developed. Problem one is solved analytically with the use of known and new methodical aspects for every point in question. Following parameters will be considered: minimum and maximum undercut thickness, feed of cutter-loader, coefficient of cutting time, efficiency of cutter-loader, specific energy consumption, and ash-content of mined coal. As a result, the main parameters of thin coal seam selective mining are obtained. Their dependence on undercut rocks and applied technique is analyzed. Table 2.2 demonstrates initial data, typical for mines in Western Donbass and Lvov-Volyn coal field.

Table 2.2 The main initial data for analyzing parameters of seam selective mining technique

Factors	Units	Region	
		Western Donbass	Lvov-Volyn coal field
Coal seam thickness	m	0.6 – 0.8	0.7 – 0.8
Wall length	m	200 - 350	
Web width	m	0.7 – 0.8	
Coal resistance to cutting	kN/m	300	200
Resistance of rock to cutting	kN/m	200	300
Seam ash-content	%	15	
Undercut rock ash-content	%	90	
Coal density	t/m^3	1.4	
Rock density	t/m^3	2.5	
Type of mechanization: support		KD80, KD90, M103, DM, KDD и др.	
Cutter-loader		1K101UD, K103,UCD200-250 и др.	
Conveyer		CP63, CP250, CPC162	

To make analysis of calculations and comparative evaluation of result more convenient, initial data (coal seam thickness, wall length etc.) are specified as identical for both associations. The only difference is in coal and undercut rock cuttability values which can be explained by mining and geological peculiarities of operation objects.

Problem two is solved analytically with the use of obtained basic parameters of the new technique. As a result, 12 principal procedures of mining thin and very thin seams with enclosing roof or floor undercut are developed.

Admitted for future consideration powered system 1KM103 provides complete mining as well as three alternatives of selective mining: with floor rock undercut – in one and two cutter-loader passes, and in one pass of cutter-loader with roof rock undercut. Similar possibilities are also available for other powered systems equipped with powered supports working in accordance with a procedure, and cutter-loaders with spaced auger-type end organs.

2.2. Application and main features of the technique

The technique to mine thin and very thin flat and sloping seams is applied for improving a grade of coal produced in seams with enclosing roof and floor undercut, and for expanding the field for available systems of cutter-loaders for their efficiency in 0.4…0.9 m seams.

Selective mining is separate in time or in space coal seam and undercut rock mining with independent transportation of mineral and rocks. The technique is based upon appatch of coaling practically each available powered system; besides, specific system “Zapadny Donbass” (MKZD), designed in cooperation of employees of the Department of Underground Mining of the NMU with DonCI and “Dongiprouglemash” can also be applied. If it is economically feasible, narrow-web cutter-loaders, being a part of the systems, can mine coal seams of any thickness.

The use of available mining equipment makes the technique rather flexible: it provides transfer from selective mining to complete one and vice versa without any extra expenditure in a longwall. However, the technique has disadvantages. As a result, time consumption per one cycle in a longwall increases, and if rocks are left in worked-out space, labour intensity also grows.

Such cutter-loaders with spaced auger-type end organs as 1K103, 1K101UD, and UCD 200-250 are the most modified for selective mining coal seam and enclosing roof and floor undercut. Design of the cutter-loaders makes it possible to mine seams in two passes and in one as well.

2.3 Justifying basic parameters of technique for mining thin and very thin seams with enclosing roof and floor undercut

2.3.1 Minimums and maximums of enclosing roof and floor undercut values

Undercut minimum $m_{пр.min}$ is subject to lower bound of the support design $m_{min,}$ and coal seam thickness:

$$m_{np.min} = m_{min} - m_y \qquad (2.1)$$

Lower bound of the support feasibility according to the seam thickness can be determined by the expression:

$$m_{min} = h_{min}^{кр} + h_p + \Delta h \qquad (2.2)$$

where $h_{min}^{кр}$ is minimum structural depth of powered support (folded down), m; h_p is a reserve of hydraulic expansion to give relief to supports, m (if a seam thickness is less than 1.0 m, then it is taken to be 0.030 m; if thickness is more than 1.0, then it is 0.05 m); Δh is a value of rock fault at the level end support unit, m.

While solving the problem, a number of authors specify $h_{min}^{кр}$ and h_p as fixed values. However, they can not agree about determining Δh. Several of them [15] propose to determine mean values of rock fault within a face space, obtained by means of actual measurements in a line of end support unit when either this longwall’ advance or neighbouring one is 10…20 m; others [23, 26] propose determining Δh with the help of following formula:

$$\Delta h = \alpha \cdot m \cdot R, \qquad (2.3)$$

obtained in terms of statistical analysis of a number of observations; besides, it helps to determine a mean value of rock fault including the seam thickness, rock grade, and the face-area width. Using different approaches to solve the problem of evaluating m_{min}, [18,28,56,64,81] authors agree that to specify m_{min} it is required to take up maximum values of rock fault rather than mean ones.

Research by the NMU, concerning rock-pressure manifestation in longwalls of Western Donbass Lvov-Volyn mines, as well as analysis of results obtained demonstrate that in the majority of cases, the use of average values of rock fault is not permitted. Specifying $m_{min,}$ it is required to take up maximum values of Δh.

Research by All-Union Surveying Institute [64] demonstrates that rock fault in a face space of longwalls equipped with powered chock supports for flat coal seams is perfectly described by the dependence:

$$\Delta h = 0{,}01 \cdot (\alpha - \beta P + ce^{-np}) \cdot Rme^{-\frac{k}{t}} \tag{2.4}$$

where α, β, c, n, k are fixed coefficients determined empirically [64]; P is a support resistance, kN/m^2; R is a width distance of face space from a face itself to end row of legs, m; m is the seam thickness, m; t is a period of roof location within the longwall face space, hours.

The expression compares favourably with (2.3) expression as it takes into consideration the effect on Δh value, support resistance, and a period of roof location within the longwall face space. If we transform formula (2.4), symbolizing the value $(\alpha - \beta P + ce^{-np})$ through $k_{кр}$ (a coefficient, taking into consideration support resistance in terms of specified mining and geological conditions), we obtain:

$$\Delta h = 0.01 \cdot k_{кр} \cdot Rme^{-\frac{k}{t}}. \tag{2.5}$$

If we calculate $k_{кр}$ value for specified assumptions and powered supports, we obtain:

– $k_{кр} = 6.298$ for MK97 and MK97M;

– $k_{кр} = 5.705$ for KD80 and KD90; and

– $k_{кр} = 5.381$ for KM103.

Table 2.2 demonstrates Δh values obtained with the help of calculations from (2.5) expression for specific mining and geological conditions. The same table shows processed measurement results of factual rock fault; the measurements are taken longwalls of thin coal seams equipped with powered mining systems. As Table 2.2 demonstrates, in the majority of cases, maximums of factual rock fault, taking into account rock cushion, are almost 10% higher than their estimated values. Therefore, following expression may be applied for Δh specifying:

$$\Delta h = 0{,}1m + 0{,}01k^{-\frac{k}{t}}. \tag{2.6}$$

Inserting the value into (2.2) formula, and performing transformations, we obtain:

$$m_{min} = \frac{h_{min}^{кр} + h_p}{0{,}9 - 0{,}01k_{кр} \cdot Re^{-\frac{k}{t}}}. \tag{2.7}$$

In [15,23,26,28] calculations, a value of (2.7) expression numerator is considered as equal to minimum structural depth of the support (assembled) taking into account a degree of hydraulic expansion to lighten props. However, concerning powered supports meant for thin flat seam mining, such an approach is not completely reasonable. After all, in this case a pass for staff and cutter-loader are basic and required conditions.

Paper by DonCI [70] shows that on the basis of movement condition, a pass minimum height within powered support units for flat seams should not be less than 500 m. On the data of physiological research by DonCI, a process of human movement under such a height is considered as physically demanding job.

To make it possible not only moving within the system support but also its maintaining, pass height in units of powered support for flat seams should not be less than 550-600 mm [70].

Providing elementary conditions for miners' work rather than creating comfortable conditions in a longwall is meant (although, one should work for that). It is understood, that table values of minimum height of assembled support, assumed for the calculations, not always meet the requirements of thin flat seam mining; more specifically, they can not provide required a pass for staff and winning machines.

We believe that H_{min} value should be put in expression (2.7) numerator. The matter is, that together with design values of the support it would take into consideration a height required for winning machine pass as well as physiological parameters, that is creeping height in the support unit. Maximums of m_{min}, calculated inclusive of above parameters, are also involved:

$$m_{min} = max\left\{\frac{\left(h^{к} + h_{з}\right)k_i; \left(h_{min}^{пр} + h_{пер}\right)k_г; \left(h_{min}^{кр} + h_p\right)k_з}{0{,}9 - 0{,}01 k_{кр} R_i e^{-\frac{k}{t}}}\right\}, \tag{2.8}$$

where $h^{к}$ is height of cutter-loader frame, m; $h_з$ is required clearance between the cutter-loader frame and the support ceiling, m; $h^{пр}_{min}$ is minimum pass height in the support units, m; $h_{пер}$ is cumulative thickness of the ceiling and the support substructure, m; and k_i, $k_г$, $k_з$ are coefficients taking into account parameters applied.

Expressing value t (period of roof location within a face space) in terms of process parameters (the cutter-loader feed, machine time coefficient, the longwall length etc.) we obtain:

$$m_{min_i} = max\left\{\frac{\left(h^{k} + h_{з}\right)\cdot k_i; \left(h_{min}^{пр} + h_{пер}\right)k_г; \left(h_{min}^{кр} + h_p\right)k_з}{0{,}9 - 0{,}01 \cdot k_{кр} \cdot R_i\left(1 - \frac{T_{см} \cdot r \cdot V \cdot k_м}{2R_i \cdot l}\right)}\right\}, \tag{2.9}$$

Table 2.3 Theoretical and actual values of roof fault in terms of longwalls under study

Association, mine	Longwall	Seam	A system type	Average thickness, m		Roof fault, mm	Theoretical values of roof fault maximums (according to AUSI), mm	Rock cushion value, mm	Actual roof fault inclusive of rock cushion, mm	Maximum vibrations of roof fault, percent of mined thickness
				Coal seam thickness	Mined thickness					
Western Donbass										
"Ternovskaia" mine	665	C_6	KMK97	0.81	0.95	112-138	147	50-105	162-238	9.6
"Ternovskaia" mine	505	C_5	KD80	0.80	1.1	148-188	174	50-105	198-293	10.8
"Ternovskaia" mine	617	C_6	KD80	0.82	1.1	151-182	162	50-100	201-282	11.0
"Stepnaia' mine	623	C_6	KMK97	0.98	0.98	119-137	152	50-105	169-242	9.1
"Ternovskaia" mine	632		KD80	0.80	1.0	130-164	186	50-120	180-284	9.8
Lvov-Volyn coal field										
Mine no.5 "Velikomostovskaia"	21	n_7^8	KMK97	0.67	0.97	87-176	147	30-70	117-240	10.2
Mine no.5 "Velikomostovskaia"	24	n_7^8	KMK97	0.95	0.95	75-168	157	40-80	115-248	9.6
Mine no.5 "Novovolynskaia"	17	n_7	1KM103	0.70	0.95-1.0	9-189	179	50-100	146-289	11.0

where r is web width of cutter-loader end organ, m; V is the cutter-loader feeding velocity, m/min; $T_{см}$ is changeover time, min; $k_м$ is machine time coefficient; l is a longwall length, m.

The expression, contrary to available ones, takes into account maximum convergence of enclosing roof and floor, effect of the support force parameters and basic parameters of the technique applied involving time factor. That helps to improve calculation accuracy for mining seams in Western Donbass and Lvov-Volyn coal field.

Target value of minimum undercut $m_{пр\ min}$ is:

$$m_{пр\ min} = m_{min\ i} - m_y. \tag{2.10}$$

Maximum of undercut $m_{пр\ max}$ is subject to maximum adaptability of the support m_{max} design and coal seam thickness

$$m_{пр\ max} = m_{max} - m_y. \tag{2.11}$$

Maximum adaptability of the support on the seam thickness is determined by:

$$m_{max} = h_{max}^{кр} + \Delta h_n, \tag{2.12}$$

where $h_{max}^{кр}$ is maximum design height of powered support, m; and Δh_n is a roof fault at the level of the support front leg, m.

Making transformations similar to previous ones (while identifying m_{min}), we obtain:

$$m_{max} = \frac{h_{max}^{кр}}{1 - 0{,}01 h_{кр} \cdot R_n \left(1 - \dfrac{T_{см} \cdot r \cdot V \cdot k_м}{2R_n \cdot l} \right)}, \tag{2.13}$$

where R_n is distance on the face space width from a stope to a front line of props, m.

Then, maximum undercut $m_{пр\ max}$ is:

$$m_{пр.max} = \frac{h_{max}^{kp}}{1 - 0{,}01 \cdot k_{kp} \cdot R_n \left(1 - \dfrac{T_{см} \cdot r \cdot V \cdot k_м}{2R_n \cdot l} \right)} - m_y. \tag{2.14}$$

With stowing undercut rocks in worked-out space of a longwall, undercut maximums are limited by possible stowing volume, being determined as:

$$m_{пр.max} = \frac{(m_y + \Delta h_з) \cdot l_з \cdot k_{пз}}{l - l_з \cdot k_{пз}}, \tag{2.15}$$

where $\Delta h_з$ is roof fault at the level of the support back leg in the process of worked-out space stowing, m; $l_з$ is the longwall part stowed by undercut rock, m; and $k_{пз}$ is a coefficient which involves the stowing compactness.

Table 2.4 demonstrates minimum and maximum enclosing roof and floor undercut within longwalls of mines in Western Donbass and Lvov-Volyn coal field equipped with powered systems. The undercut are calculated according to expressions (2.10, 2.14).

Table 2.4. Minimum and maximum enclosing roof and floor undercut while mining seams which thickness is 0.6, 0.7, and 0.8 m

Type of system	Lvov-Volyn coal field						Western Donbass					
	0.6		0.7		0.8		0.6		0.7		0.8	
	$m_{np.\,min}$	$m_{np.\,max}$	$m_{np.\,min}$	$m_{np.\,max}$	$m_{np.\,min}$	$m_{np.\,max}$	$m_{np.\,min}$	$m_{np.\,max}$	$m_{np.\,min}$	$m_{np.\,max}$	$m_{np.\,min}$	$m_{np.\,max}$
Complete mining												
1KM103	0.27	0.56	0.17	0.46	0.07	0.36	0.30	0.60	0.20	0.50	.010	0.40
KD-80	0.37	0.67	0.27	0.57	0.17	0.47	0.38	0.72	0.28	0.62	0.18	0.52
KMK-97	0.31	0.68	0.21	0.58	0.11	0.48	0.35	0.71	0.25	0.61	0.15	0.51
KMK-98	0.30	0.66	0.20	0.56	0.10	0.46	0.33	0.79	0.23	0.59	0.13	0.49
KD80	0.42	0.69	0.32	0.59	0.22	0.49	0.45	0.72	0.35	0.62	0.25	0.52
Separate mining (one pass)												
1KM103	0.27	0.56	0.17	0.46	0.07	0.36	0.30	0.60	0.20	0.50	0.10	0.40
KD-80	0.37	0.67	0.27	0.57	0.17	0.47	0.38	0.72	0.28	0.63	0.18	0.52
KMK-97	0.31	0.68	0.21	0.58	0.11	0.48	0.35	0.71	0.25	0.61	0.15	0.51
KMK-98	0.30	0.66	0.20	0.56	0.10	0.46	0.33	0.69	0.23	0.59	0.13	0.49
KD80	0.42	0.69	0.32	0.59	0.22	0.49	0.45	0.72	0.35	0.62	0.25	0.52
Separate mining (two passs)												
1KM103	0.32	0.61	0.22	0.51	0.12	0.41	0.36	0.65	0.26	0.55	0.16	0.45
KD-80	0.43	0.74	0.33	0.64	0.23	0.54	0.46	0.77	0.36	0.67	0.26	0.57
KMK-97	0.37	0.72	0.27	0.62	0.17	0.52	0.42	0.74	0.32	0.64	0.22	0.54
KMK-98	0.35	0.72	0.25	0.62	0.15	0.52	0.40	0.75	0.30	0.65	0.20	0.55
KD80	0.49	0.75	0.39	0.65	0.29	0.55	0.53	0.79	0.43	0.69	0.33	0.59

Table 2.3 data demonstrate that undercut values are identical for one-pass complete and separate mining; however, if two-pass mining is applied, then undercut thickness should experience 0.05-0.08 m increase. That depends on extra increase in enclosing roof and floor convergence due to increase in face space and time of supported roof being in it. Smaller undercut values in longwalls of Lvov-Volyn coal field mines can be explained by increasing relative feeding speed of cutter-loaders because of lesser cuttability of primarily mined patch of coal and further harder rock mining weakened by two outcropping planes. Table 2.3 data prove that to provide 0.6 m pass height within a longwall, mined thickness of a seam should not be less than

0.90…0.95 m for КМК-97 and КМК-98; 1.02…1.05 m for KD80, and 0.87…0.9 m for KM103.

Minimum undercut thickness for mining seams with given 0.6 m, 0.7 m, and 0.8 m thickness using mechanized system KM103 is 0.30 m, 0.20 m, and 0.10 m; maximum thickness is 0.7 m, 0.6 m, and 0.5 m.

It should be noted that design features of cutter-loader 1K103 (located in the neighbourhood of a stope) can not provide floor undercut which height is more than 0.35 m. That is why, analyzing dependence of the technique basic parameters on enclosing roof and floor undercut thickness, we will vary it as Table 2.5 demonstrates.

Table 2.5 Limits for undercut thickness varying in a longwall equipped with 1KM103 system

Undercut of	Coal seam thickness		
	0.6	0.7	0.8
floor	0.3-0.35	0.2-0.35	0.1-0.35
roof	0.3-0.7	0.2-0.6	0.1-0.5

2.3.2 Feeding velocity of a cutter-loader

Feeding velocity is one of basic process parameters. According to available calculation methods, it is expressed in complicated mathematical equations. Paper [13] proposes dependence of feeding velocity on engine power of a cutter-loader, seam thickness and its specific cuttability derived on the basis of dimension theory and a theory of equation similarity:

$$V = \frac{P \cdot t_{cut}}{m \cdot r \cdot A} - 0.2V_{cut} \tag{2.16}$$

where P is total power consumed by a cuter-loader, kW; t_{cut} is a pitch between cutting lines, cm; m is mined seam thickness, m; r is web width of a cutter-loader, m; A is a seam resistibility to cutting, kN/m; $V_{рез}$ is cutting velocity, m/c.

In the process of complete mining, seam resistibility to cutting is determined involving undercut rock resistibility to cutting:

$$\overline{A} = \frac{\overline{A}_y m_y + \overline{A}_n m_{np}}{m_y + m_{np}}, \tag{2.17}$$

where $\overline{A}_y$ is coal seam cutting resistibility, kN/m; $\overline{A}_n$ is undercut rock cutting resistibility, kN/m; m_{np} is undercut rock thickness, m.

In the process of selective mining, determination of feeding velocity should take into account thickness of patches of coal and solid units as well as their cuttability. In this context it is required to take into consideration the rock mass weakening due to advance coal or rock mining; k_{app} coefficient is applied (Table 2.6).

Table 2.6 Values the rock mass weakening coefficients

End organ operative conditions	Cutting direction	
	Towards outcropping	Towards a pillar
The rock mass is weakened due to advance cut	0.6-0.7	0.6-0.7
The rock mass is weakened by heading lower end organ	0.64-0.68	0.75-0.8
The rock mass is weakened by heading upper end organ	0.72-0.77	0.85-0.3
The rock mass is not weakened	1.0	1.0

Involving k_{app}, expression (2.16) is:

$$V = \frac{P \cdot t_{cut}}{r\left(m_y \overline{A}_y k_{app}^y + m_{np} \overline{A}_n \cdot k_{app}^n\right)} - 0{,}2V_{cut}, \tag{2.18}$$

where k_{app}^y, k_{app}^n are coal and rock weakening coefficients, accordingly; $m_{пр}$, m_y are thickness of undercut rock or coal seam, m; $\overline{A}_y$, $\overline{A}_n$, are coal or rock cuttability, accordingly, kN/m.

Figures 2.2 and 2.3 demonstrate acceleration profile of 1K103 cutter-loader depending upon a technique applied and enclosing roof and floor undercut.

Cutter-loader feed velocity stays invariable if one pass independent from mining type; as the figures demonstrate, increase in undercut thickness results in it decreases in both cases. Under the conditions of Lvov-Volyn coal field, decrease in feed velocity becomes more intensive; that depends on greater cuttability of enclosing roof and floor undercut.

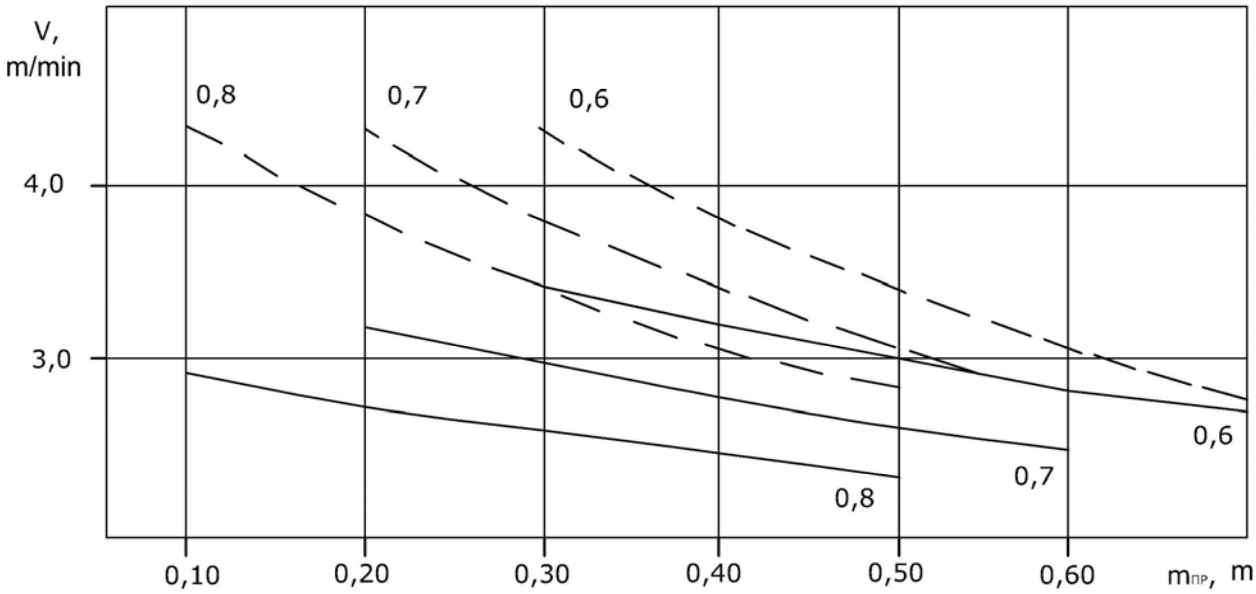


Fig. 2.2. Dependence of 1K103 cutter-loader feed velocity on the thickness of a seam and roof rock undercut if one-pass complete and selective mining:

________ -Western Donbass; __ __ __ - Lvov-Volyn coal field.

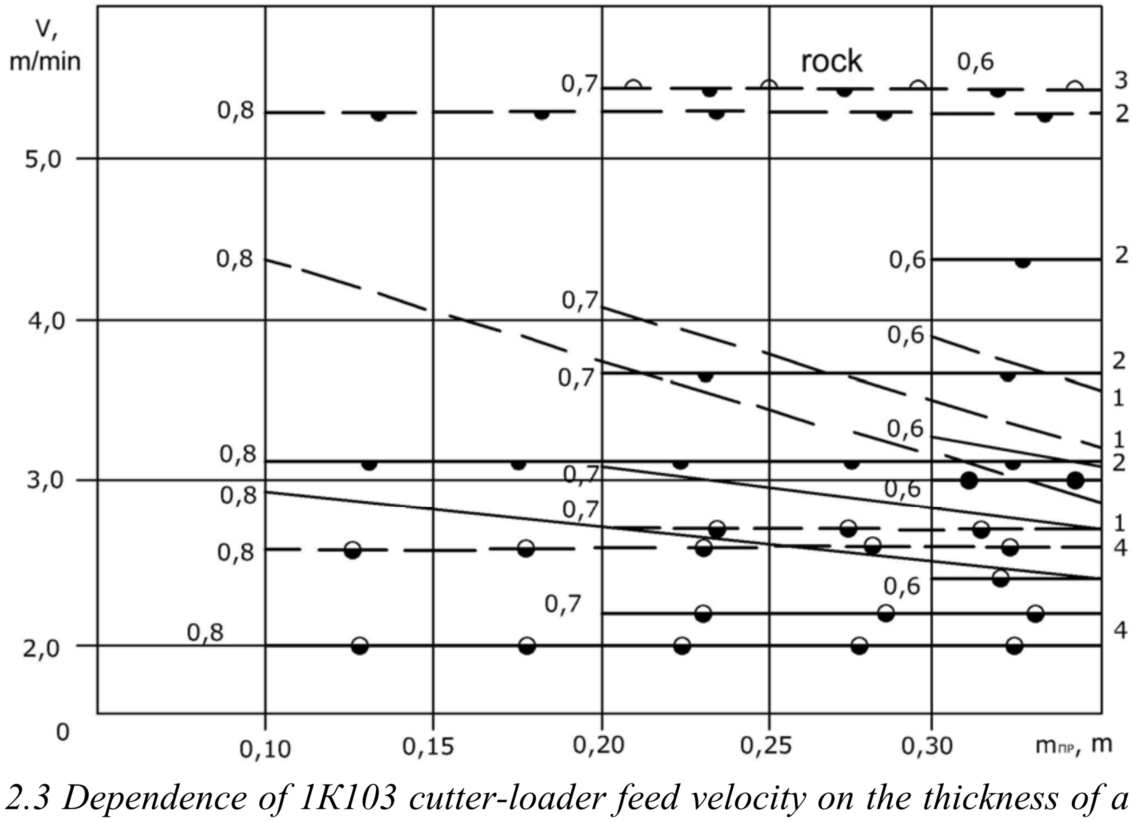


Fig. 2.3 Dependence of 1K103 cutter-loader feed velocity on the thickness of a seam and undercut floor:

1. *If one-pass complete and separate mining.*
2. *If split mining.*
3. *If undercut rock mining.*
4. *If common in the context of two-pass separate mining.*

________ Western Donbass; __ __ __ Lvov-Volyn coal field.

It should be noted that when seams having different thickness are mined (and mined thickness is uniform), then in the context of mines in Western Donbass, a cutter-loader feed velocity increases depending upon a seam thickness decrease and undercut increase; in the context of Lvov-Volyn coal field the situation is opposite – it either decreases (Fig. 2.2 and 2.3), or stays invariable. It can be explained by the fact that Western Donbass coal is harder than undercut rock, and vice versa in Lvov-Volyn coal field. Fig. 2.3 demonstrates that in terms of two-pass coal and floor rock mining, concerned undercut thickness has negligible impact on a cutter-loader feed velocity which can achieve maximum values (in calculations, V_n is 0.8 $V_{доп}$). While mining pure coal batch in terms of Lvov-Volyn coal field, a cutter-loader feed velocity is 20…40% higher to compare with Western Donbass; it also depends on higher cuttability of coal in mines of the region. Taking into account a cutter-loader feed velocity the excess is somewhat neutralized being 10…25%.

2.3.3 Machine Time Coefficient

Machine time coefficient $k_м$ is essential parameter providing estimation of approved operation schedule and equipment. It considers timing for auxiliary operations and for eliminating operational and technological delays independent of a cutter-loader operation (car exchange within a loading point, lack of empty cars, delays due to support lagging etc.). Following expression is used to determine its numeric values [98]:

$$k_м = \frac{1}{\frac{1}{k_г} + \frac{T_{мо} + T_{ко} + T_{зр} + T_{эо}}{l} \cdot V} \quad , \tag{2.19}$$

where $T_{мо}$ is cyclic timing for out-of-register switching, min; $T_{ко}$ is timing for end operations, min; $T_{зр}$ is timing for cutter replacement, min; $T_{эо}$ is time to eliminate operational problems (idle time) which are not directly connected with cutter-loader operation, min; l is a longwall length, m; $k_г$ is a cutter-loader availability factor:

$$k_г = \frac{T}{T - T_{ун}} \tag{2.20}$$

where T is operation activity of a cutter-loader:

$$T = l/V \tag{2.21}$$

$T_{ун}$, is time to fix operational problems of a cutter-loader, min.

In expression (2.19), value $k_г$ takes into account only a cutter-loader availability factor, when "Advanced programs of coal seam mining" recommend considering availability of a support and conveyor belt line as well. Consequent of expression (2.19) includes value $T_{зр}$ being practically an integral part of value $T_{ко}$. Together with the points, transformation of expression (2.19) is:

$$k_{м1} = \frac{1}{\dfrac{1}{k_{г.об}} + \dfrac{T_{мо} + T_{ко} + T_{эо}}{l} \cdot V}, \tag{2.22}$$

where $k_{г.об}$ is equipment availability:

$$k_{г.об} = k_{г.к} \cdot k_{г.кр} \cdot k_{гкл}, \tag{2.23}$$

$k_{г.к}$ is a cutter-loader availability; $k_{г.кр}$ is a support availability; $k_{гкл}$ is a conveyer belt line availability.

Availability factors $k_{г.к}$, $k_{г.кр}$, and $k_{гкл}$ are identified according to "Advanced programs of coal seam mining" or by other means.

Expression (2.22) can be used to determine $k_м$ value for one-pass complete and selective seam mining. For two-pass coal and undercut rock selective mining, (2.22) expression should be transformed with the use of time for rock mining. Finally, the value is expressed as following:

$$k_{мII} = \frac{l}{\dfrac{l}{k_{г.об}} + \left(T_{к.о} + T_{эо}\right) \cdot V + \dfrac{l \cdot V}{V_n}}, \tag{2.24}$$

where V_n is a cutter-loader feed velocity while rock mining, m/min.

According to (2.22, 2.24) expressions, value $k_м$ depends on a longwall length l, a cutter-loader feed velocity V, and established procedure of a seam mining. In our situation, longwall length is a constant, and feed velocity, among other things, depends on enclosing roof and floor undercut. It follows that after all machine time coefficient depends on enclosing roof and floor undercut. Fig. 2.4 and 2.5 demonstrate calculated dependences of machine time coefficient $k_м$ on undercut thickness $m_{пр,}$ and established procedure of mining seams having various thickness.

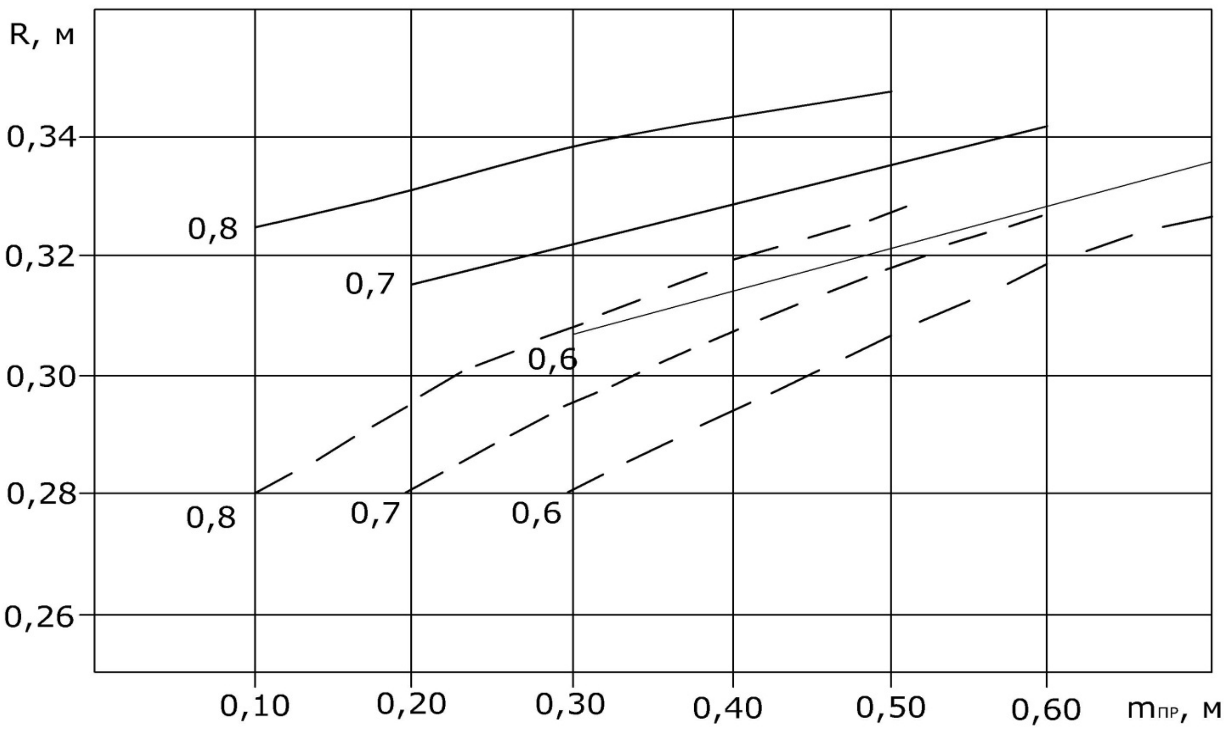


Fig. 2.4 Dependence of machine time coefficient on the thickness of a seam and undercut roof rocks while one-pass complete and separate mining:
________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

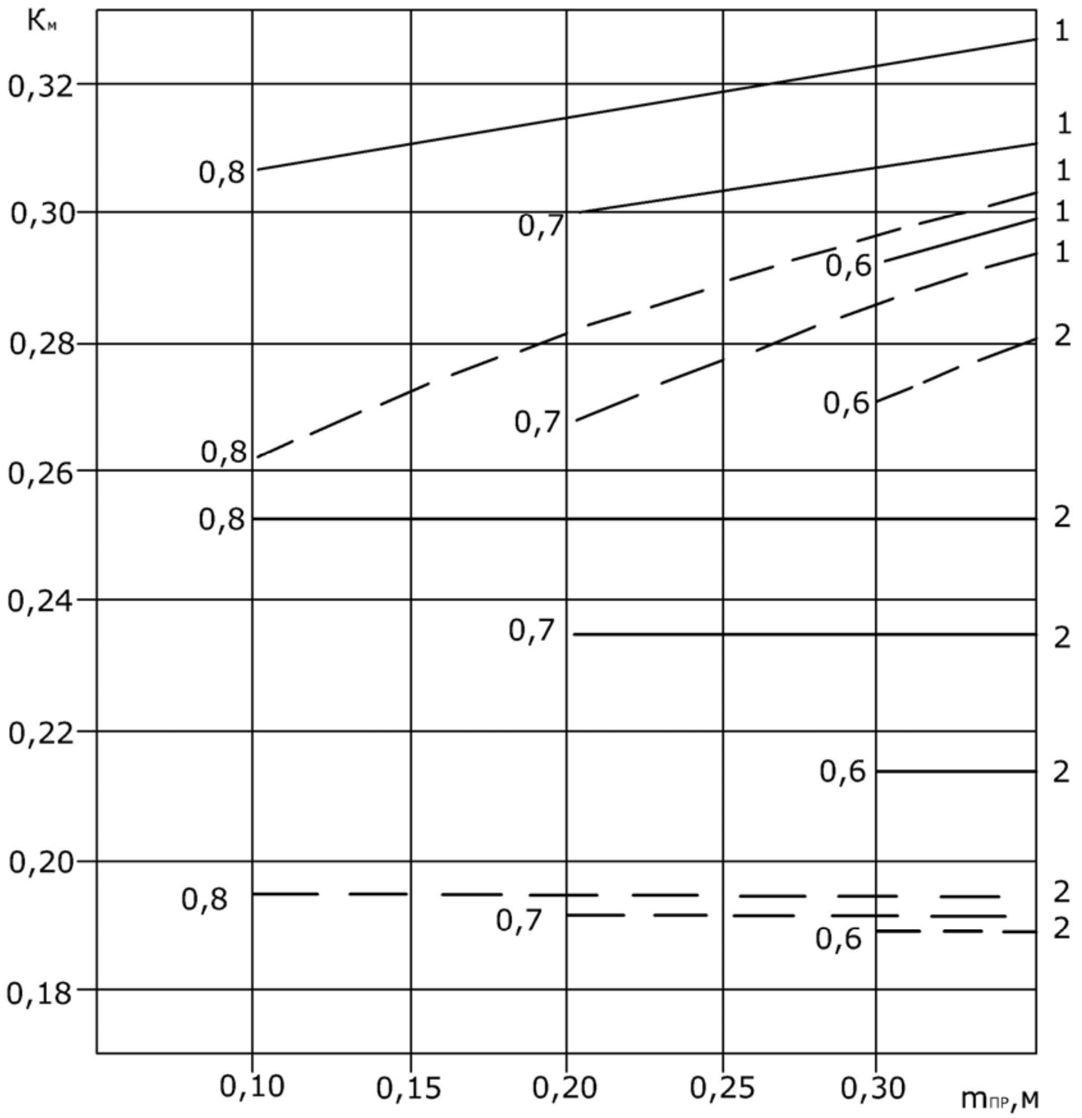


Fig. 2.5 Dependence of machine time coefficient on the thickness of a seam and floor rock undercut:

1. If separate one-pass mining;

2 If separate two-pass mining.

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

Increase in $k_м$ values takes place with increase in mined thickness in the process of one-pass complete and selective mining. It depends neither on increase in coal seam thickness nor on that of undercut rocks. In terms of Western Donbass, when mined thickness of seams is similar, then machine time coefficient reduces if coal seam thickness decrease, and undercut value increases (Fig. 2.4, 2.5). On the contrary, under similar conditions of Lvov-Volyn coal field mines, machine time coefficient either experiences some increase (Fig. 2.5), or stays constant (Fig. 2.4) depending upon a plan of seam mining (undercut of floor or roof).

$k_м$ values stay constant if separate two-passage mining is applied. In this case they in ratio depend only on coal batch thickness; by no means they depend on undercut thickness (Fig. 2.5).

2.3.4 Mined Coal Grade

Ash-content of coal is one basic grade factors which can be controlled while mining. In this context scheduled ash-content of rock mass is determined with the help of expression (2.25) showing net-weighed ash-content of mined coal seam and undercut rock:

$$A_{г.м} = \frac{A_у \cdot m_у \cdot \gamma_у + A_п \cdot m_{пр} \cdot \gamma_п}{m_у \cdot \gamma_у + m_{пр} \cdot \gamma_п}, \qquad (2.25)$$

where $A_у$ and $A_п$ are source ash-content of coal seam and undercut rock correspondingly, %.

However, as research by the NMU in mines of Western Donbass and Lvov-Volyn coal field shows, roof falls in a longwall face zone exercise a significant influence on a value of mined coal ash-content. Due to it, coal dilution is 5 to 10%. That is why, (2.25) expression should be completed with value $A_з$ involving increase in coal ash-content in a longwall caused by falls. Then (2.25) expression is:

$$A_{г.м} = \frac{A_у \cdot m_у \cdot \gamma_у + A_п \cdot m_{пр} \cdot \gamma_п}{m_у \cdot \gamma_у + m_{пр} \cdot \gamma_п} + A_з, \qquad (2.26)$$

where $A_з$ is a value of dilution caused by roof falls in a longwall face zone, %.

According to approach by such institutes as “UkrNIIugleobogashchenie”, DonCI, and A.A. Skochinski Institute of Mining, $A_з$ value can be determined by:

$$A_з = \frac{A_n - A_y}{1 + \dfrac{m_y \cdot \gamma_y}{m_{\sigma n} \cdot \gamma_n}}, \tag{2.27}$$

where $m_{\sigma n}$ is a value of enclosing roof and floor dilution, m;

$$m_{бп} = \frac{l_{лк}}{l} \cdot m_{лк} \cdot k_{лк} + \frac{l - l_{лк}}{l} \cdot m_{зуд} \cdot k_{Voз} k_{мбп} \tag{2.28}$$

where $l_{лк}$ is length of a longwall with a false roof, m; $m_{лк}$ is the false roof thickness, m; $k_{лк}$ is a coefficient involving dilution by enclosing roof and floor; $m_{зуд}$ is tolerated dilution of coal by enclosing roof and floor in a longwall, m; $k_{Voз}$ is a coefficient involving advance rate;

$$k_{Voз} = 1 + \left(40 - V_{oз}\right) \cdot \frac{\Delta k_V}{10}, \tag{2.29}$$

where $V_{oз}$ is advance rate, m per month; Δk_V is adjusted coefficient; and $k_{мбп}$ is a ratio of ash-content increase depending upon enclosing rock type.

Inserting expression (2.27) into formula (2.26), and performing required transformations, we obtain:

$$A_{гм} = \frac{A_y m_y \gamma_y + A_n \gamma_n \left(m_{np} + m_{бп}\right)}{m_y \gamma_y + \gamma_n \left[m_{np}\left(1 - k_n\right) + m_{бп}\right]}, \tag{2.30}$$

In the process of separate coal seam and undercut rocks mining, seeing that no dilution is owing to undercut, one can determine coal ash-content by:

$$A_{ду} = \frac{A_y m_y \gamma_y + A_n \gamma_n \left[m_{np}\left(1 - k_n\right) + m_{бп}\right]}{m_y \gamma_y + \gamma_n \left[m_{np}\left(1 - k_n\right) + m_{бп}\right]}, \tag{2.31}$$

where k_n a coefficient involving conveyer belt complete charging with undercut rock.

As it follows from expressions (2.30 and 2.31) and dependences resulting from the expressions (Fig. 2.6 and 2.7), thickness of rock undercut exercises a significant influence on mined coal ash-content.

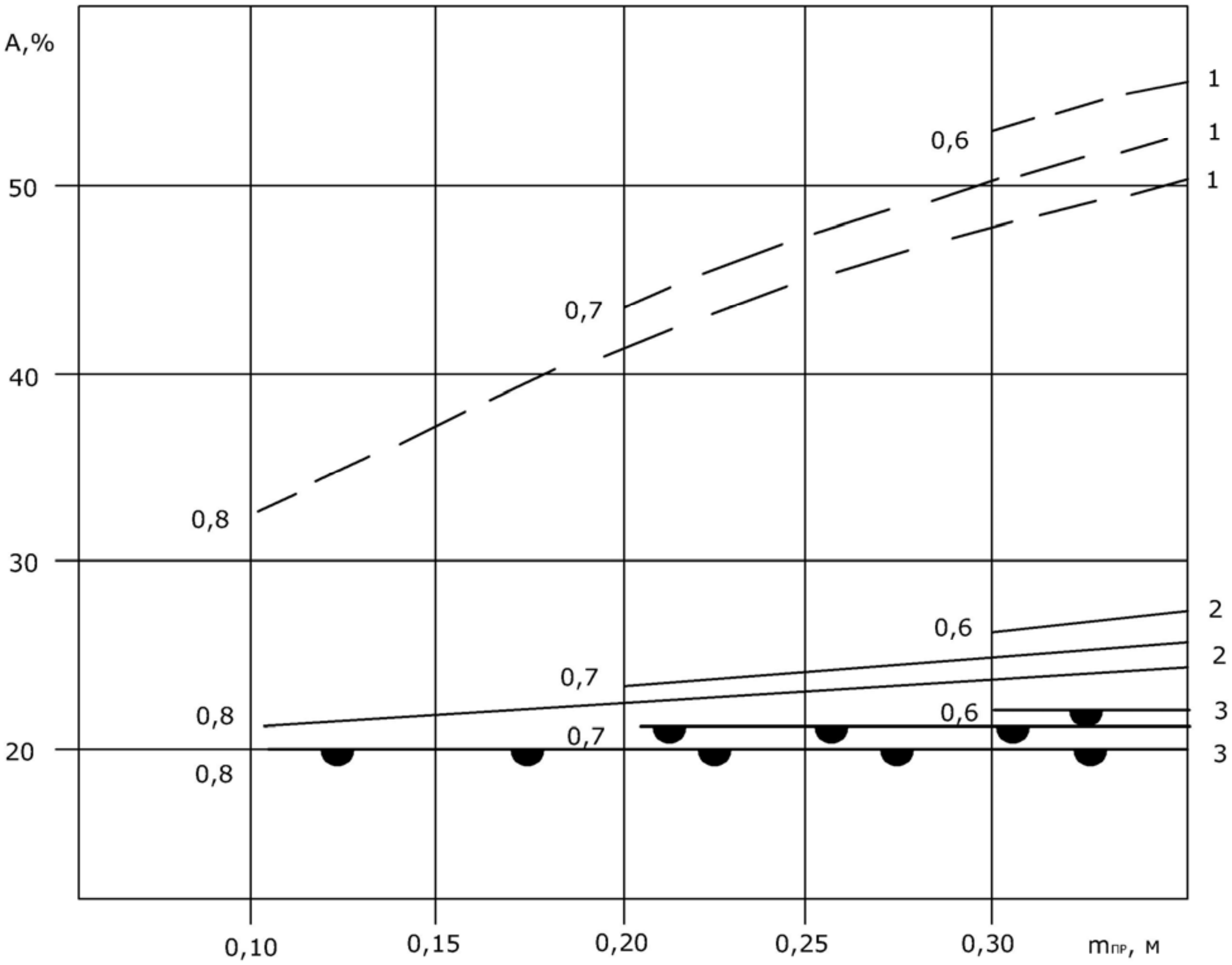


Fig. 2.6. Dependence of output ash-content on seam content and undercut rocks:

1. *Rock mass in the process of complete mining.*
2. *Coal, mined in the process of one-pass separate mining.*
3. *Coal, mined in the process of two-pass separate mining.*

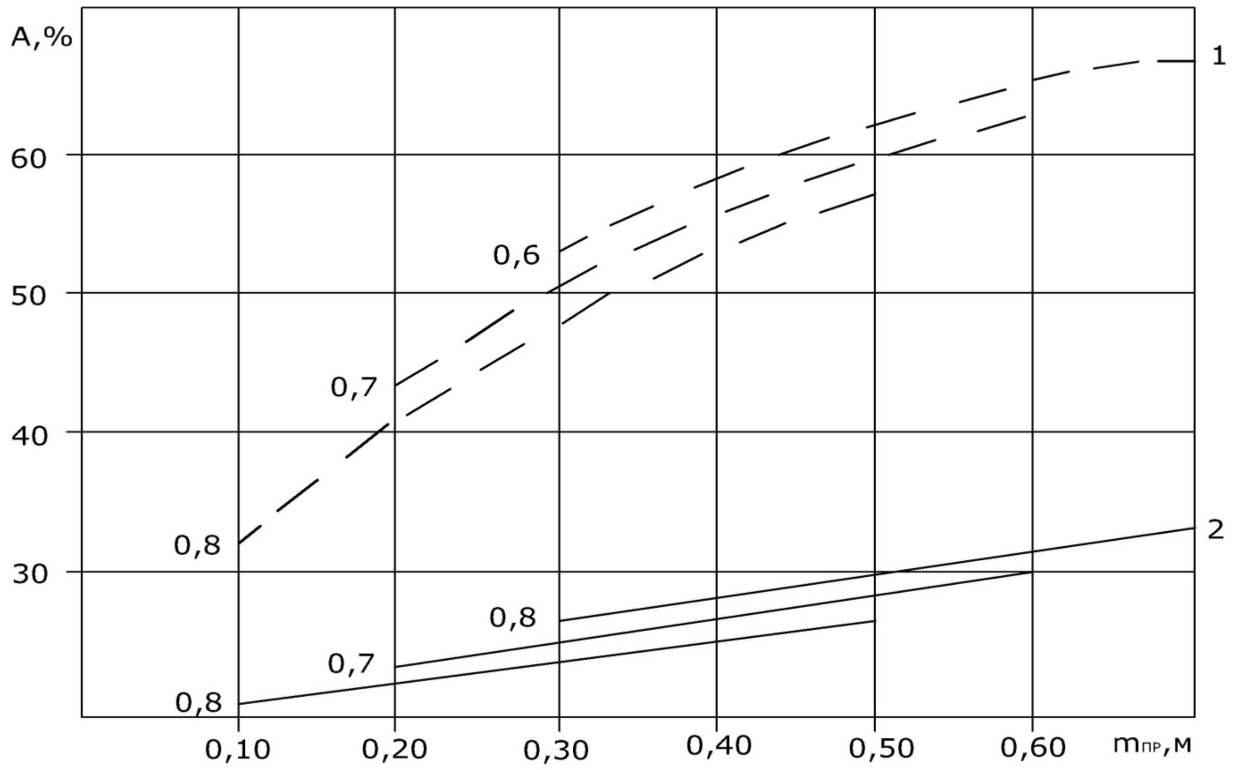


Fig 2.7. Dependence of output ash-content on seam thickness and roof rock undercut: 1. If complete mining; 2. If separate mining.

However, it should be noted that established procedure of a seam mining rather than undercut thickness is a determinant for mining quality formation. Thus, 1 cm increase in undercut taking place in the process of complete mining results in extra 0.4-1.2% mined coal dilution; separate one-pass mining gives only 0.1-0.2% increase; and separate two-pass mining makes it possible to avoid practically dilution resulting from undercut. As Figures 2.6 and 2.7 demonstrate, in addition to abovementioned factors, ash-content of mined coal is also affected by a coal seam thickness; the less it is, the higher is a level of coal dilution. In this context, intensity of the process depends on established mining procedure. Thus, it is two times higher in the process of complete mining to compare with one-pass separate mining; also, it if four times higher to compare with two-pass separate mining [101].

Transition from complete mining with enclosing roof and floor undercut to their selective mining will give twofold mined coal grade improving (Fig. 2.6 and 2.7).

2.3.5. Cutter-Loader Capacity

In general, cutter-loader capacity depends on the quantity of mineral mined in time unit. It is common practice to distinguish theoretical capacity, technical capacity, and working one. Department of Mining Machines and Systems of Moscow Mining Institute proposed the idea and developed standard practice to calculate the capacity of mining machines, systems, and assemblies [74, 89].

Mining and geological, operational, and design parameters are involved in the process of capacity calculation; however, quality factors are not taken into account. But it is known that each per cent of planned ash-content excess mined in a longwall with rock mass undercut cuts 2% of total mining. That is, in a finite case, ash-content of rock mass mined exercises significant effect on a cutter-loader capacity. Hence, it is proposed to introduce the idea of reduced capacity for longwalls with enclosing roof and floor undercut.

Reduced capacity of cutter-loader $Q_{пр}$ (t per shift) is determined taking into account rock mass cuts for planned ash-content excess differing from working capacity by $k_{ск}$ value being a cut coefficient:

$$Q_{пр} = k_{ск} \cdot Q_{э} = k_{ск} \cdot k_{м} \cdot Q_{theor} \cdot T_{см}, \tag{2.32}$$

where $T_{см}$ is a shift duration, min; and Q_{theor} is theoretical capacity of a cutter-loader, t/min:

$$Q_{theor} = V \cdot r\left(m_y \gamma_y + m_{пр} \gamma_п\right); \tag{2.33}$$

$k_{ск}$ is a coefficient of cuts by rock mass for planned norm of ash-content excess

$$k_{ск} = 1 - 0.02\left(A_{гм} - A_{пл}\right); \tag{2.34}$$

where 0.02 is a cut coefficient for 1% ash-content excess; $A_{гм}$ is actual ash-content of mined rock mass, %; $A_{пл}$ is a planned norm of ash-content in the longwall, %.

Daily average load on a stope $Q_н$, working with complete coal and rock mining is taken to be equal to daily reduced capacity of a cutter-loader:

$$Q_н = k_{ск} \cdot k_{м} \cdot Q_{theor} \cdot T_{см} \cdot n_{см} \tag{2.35}$$

where $n_{см}$ is the number of shift per day.

Capacity calculation for selective mining needs some correctives involving specifity of the technique. Thus, taking into account the fact that theoretical capacity of a cutter-loader is used to select the equipment for the whole manufacturing chain, it is required to calculate a cutter-loader capacity on coal as well as on undercut rock.

Theoretical capacity is determined on the quantity of coal or rock mined by a cutter-loader per time unit in terms of its continuous productive work:

$$Q_{theor} = m_y \cdot r \cdot V_y \cdot \gamma_y, \qquad (2.36)$$

$$Q_{theor} = m_{np} \cdot r \cdot V_n \cdot \gamma_n, \qquad (2.37)$$

where $V_{y,n}$ is feed velocity of a cutter-loader while mining coal and rock, accordingly, m per minute.

Feed velocity of a cutter-loader is:

in terms of one-pass coal mining:

$$V_{yI} = \frac{P \cdot t_{cut}}{r\left(m_y A_y k_{app}^y + m_{np} A_n \cdot k_{app}^n\right)} - 0{,}2V_{cut}, \qquad (2.38)$$

in terms of two-pass mining:

$$V_{yII} = \frac{P \cdot t_{cut}}{m_y \cdot r \cdot A_y \cdot k_{app}^y} - 0{,}2V_{cut}; \qquad (2.39)$$

in terms of rock mining:

$$V_n = \frac{P \cdot t_{cut}}{m_{np} \cdot r \cdot A_n \cdot k_{app}^n} - 0{,}2V_{cut}. \qquad (2.40)$$

Feed velocity, calculated on pointed expressions, can not be more than technically possible velocity of specific cutter-loader corresponding to a support velocity (depending upon accepted operation schedule $V_{кр} \geq V$).

Working (shift) capacity of cutter-loader (tons per shift) is determined involving each timing for auxiliary operations and for organizational and technical problem solving under certain conditions of a stope which are not directly connected with a cutter-loader operations (car exchange within loading point, empty stock wait, deenergizing, support delay, avoidance of rock fall etc.).

The timing is taken into account by a cutter-loader continuity operation coefficient; if the cutter-loader operates – by a machine time coefficient $k_м$ (2.22, 2.24). Then:

$$Q_э = k_м \cdot Q_{theor} \cdot T_{см} = k_м \cdot m_у \cdot r \cdot V_у \cdot \gamma_у \cdot T_{см}, \tag{2.41}$$

Volume of mined in the process rock may be determined by:

$$Q_п = Q_э \cdot \frac{m_{пр} \cdot \gamma_п}{m_у \cdot \gamma_у}, \tag{2.42}$$

where Q_n is mined rock volume, t.

Fig. 2.8-2.11 demonstrate behaviour of a cutter-loader capacity depending upon a seam thickness, undercut thickness, and a technology applied.

Fig. 2.8 demonstrates dependences of capacity in terms of complete and separate mining seams with enclosing roof and floor undercut. As you can see in Fig. 2.8, if complete mining, on rock mass, a cutter-loader capacity experiences sharp increase depending upon undercut increase. In terms of similar mined seam thickness and various thicknesses of coal batch and under rocks, a cutter-loader capacity depends proportionally on the undercut thickness being inversely proportional to coal thickness. It is connected with the fact that undercut rock density is almost two times higher than coal density; that is volume of rock mined by a cutter-loader is almost two times heavier than comparable volume of pure coal.

In the context of Lvov-Volyn coal field, increase in mined thickness owing to undercut increase results in some drop of relative intensity of a cutter-loader capacity. Thus, if a seam thickness is 0.9 m for coal seams which thickness is 0.8 m, 0.7 m and 0.6 m, and which undercut is 0.1 m, 0.2 m and 0.3 m accordingly, difference in a cutter-loader capacity was about 32 t, then if mined thickness of a seam is 1.3 m and undercut is 0.5 m, 0.6 m and 0.7 m accordingly, that very difference is about 22 t for seams which thickness differs. Alternatively, similar conditions of Western Donbass make some increase in a cutter-loader capacity to be explained by different level of cuttability of undercut rocks within the environment involved. In this context, mines of Lvov-Volyn coal field demonstrate significant increase in a seam cuttability resulting from undercut increase to compare with mines of Western Donbass, and, as a consequence, sharp decrease in a cutter-loader capacity.

A cutter-loader capacity, reduced to ash-content norm in the process of complete mining for each undercut (0.10 to 0.7 m) is much lower to compare with separate seam mining. Depending upon increase in undercut thickness, capacity of a cutter-loader on coal as well as that reduced to ash-content norm reduces. Moreover, while separate mining, a cutter-loader capacity changes to constant depending upon undercut increase.

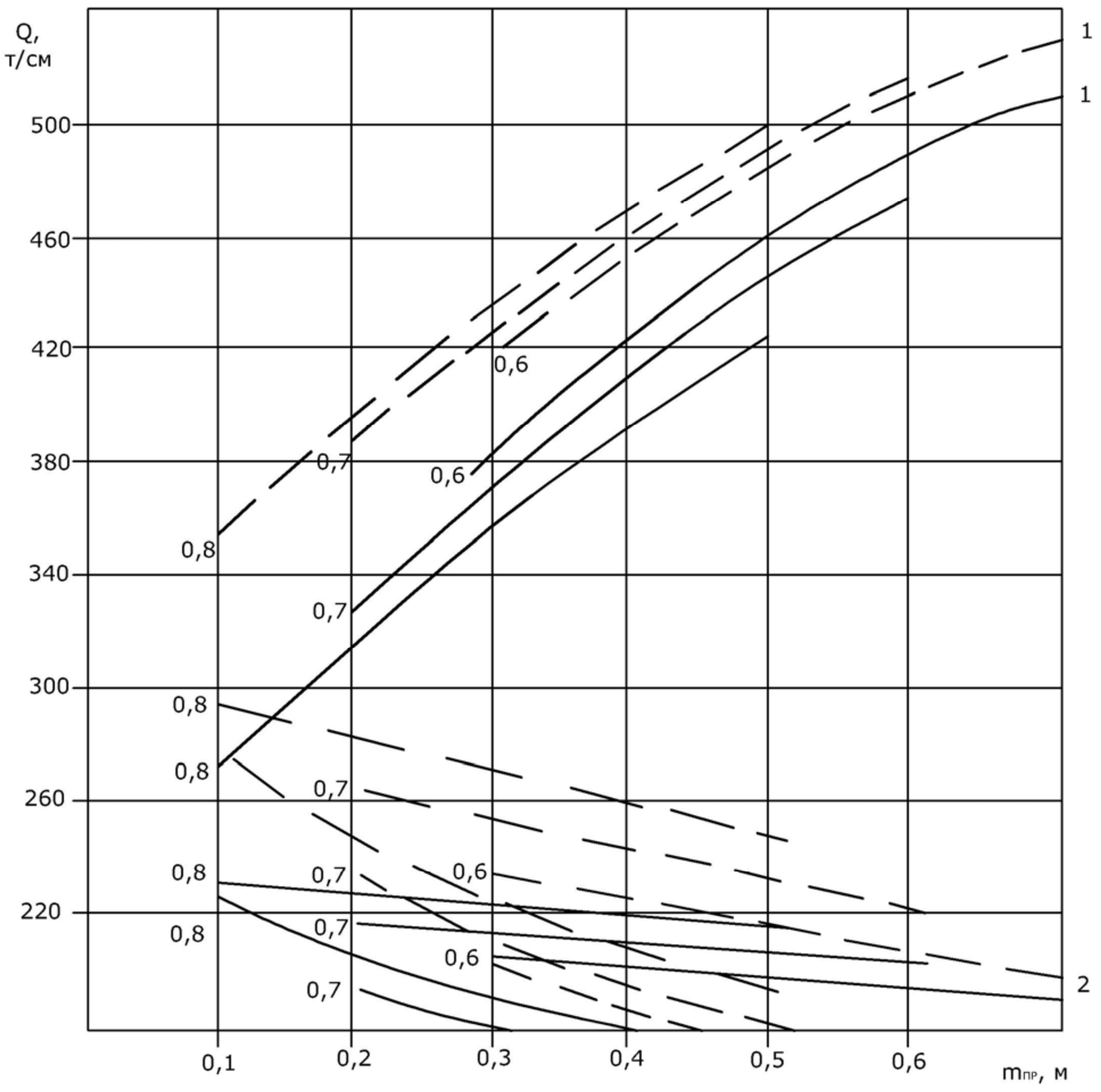


Fig. 2.8. Dependence of 1K103 cutter-loader capacity on thickness of a seam and undercut roof rocks:

1. *On rock mass in the process of complete mining.*
2. *On coal in the process separate mining.*
3. *Reduced on ash-content in the process of complete mining.*

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

The value is larger in the context of Lvov-Volyn coal field, and smaller in the context of Western Donbass being 10 t and 3 t for each 10 cm of undercut, accordingly. While complete mining, capacity of a cutter-loader on coal, reduced to unified ash-content, experiences non-unique change depending upon undercut increase. Owing to undercut value increase, capacity curve flattens out (Fig. 2.8).

Fig. 2.9 and 2.10 demonstrate dependences of a cutter-loader capacity on undercut thickness in the process of complete mining and two techniques of separate mining with floor rock undercut for one pass of a cutter-loader (Fig. 2.9) and two its passes (Fig. 2.10). In the both cases a cutter-loader capacity on rock mass is much higher to compare with that on coal; in the process of separate mining, a cutter-loader capacity is generally higher than the capacity in the process of complete mining. While mining a seam with 0.8 m thickness and less than 0.1 m undercut thickness in the context of Lvov-Volyn coal field, and less than 0.12 m in the context of Western Donbass, a cutter-loader capacity is higher in the process of complete mining to compare with two-pass separate mining.

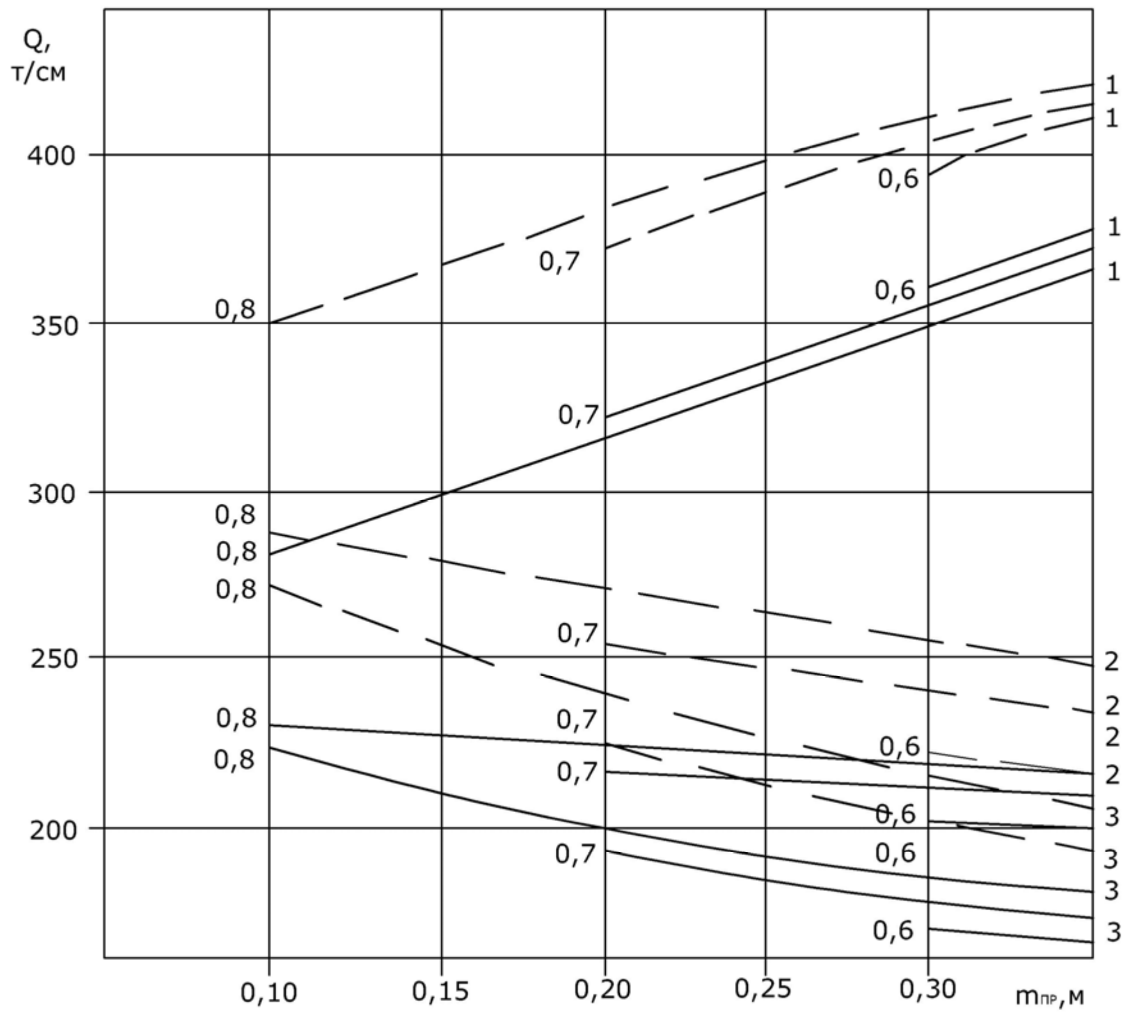


Fig. 2.9. Dependence of 1K103 cutter-loader capacity on thickness of a seam and undercut floor rocks:

1. *On rock mass in the process of complete mining.*
2. *On coal in the process on one-pass separate mining.*
3. *On ash-content reduced in the process of complete mining.*

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

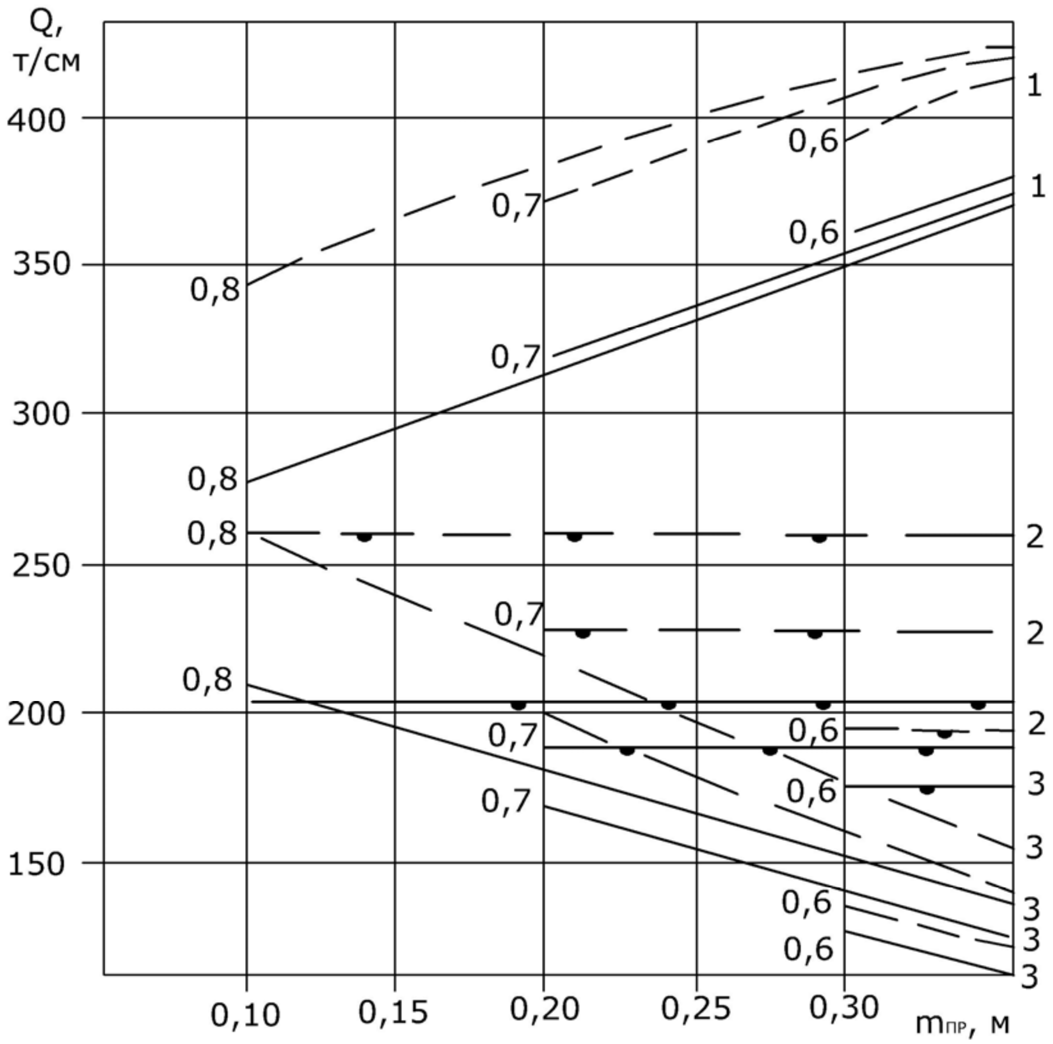


Fig. 2.10. Dependence of 1K103 cutter-loader capacity on the thickness of a seam and undercut floor rocks:

1. *Rock mass while complete mining.*
2. *Coal while two-pass separate mining.*
3. *Reduced on ash-content while complete mining.*

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

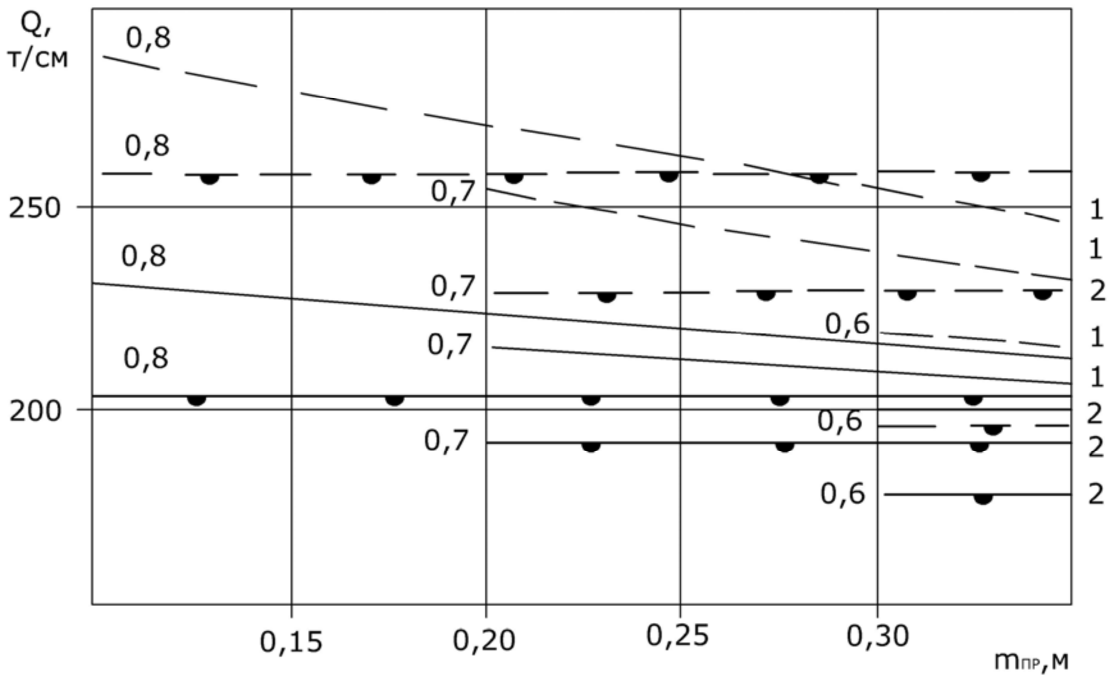


Fig. 2.11. Dependence of 1K103 cutter-loader capacity on the thickness of a seam and undercut floor enclosing roof and floor while separate mining:

1. One pass.

2 . Two passes.

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

Fig. 2.11 demonstrates dependences of a cutter-loader capacity on the thickness of undercut and coal seam using two techniques of separate mining. In terms of Western Donbass, a cutter-loader capacity in the process of one-pass mining is higher to compare with two-pass mining; in Lvov-Volyn coal field it is lower than in the process of two-pass mining if a seam thickness is less than 1.07 m. The fact can be explained by higher cuttability of undercut rocks.

It should be noted that if rock cuttability is higher than that in calculations (over 300 kN/m), a cutter-loader capacity in terms of two-pass separate mining will be more than one-pass mining capacity in the context of mined thickness, or, identically, in the context of less undercut thickness.

2.3.6. Specific Energy Consumption in the Process of Complete and Separate Mining

Coal enterprises are both suppliers of energy feedstock and large consumers of electric power [17]. Thus, mines of involved coal regions consume almost 1 kWh to mine a ton of coal. In this context, dead rock mining in longwalls with undercut takes 20 to 60% of energy consumption. Hence, annually mines of the associations only unproductively consume millions of kilowatt hours.

To select rational technique for thin flat seams mining, analyze effect of mined rock mass effect on specific energy consumption.

Specific mining energy consumption may be determined by:

$$H_w = \frac{P}{60 \cdot r \cdot m \cdot \gamma \cdot V}; \tag{2.43}$$

where r is web width of end organ, m; m is a seam thickness, m; γ is coal density, t/m^3; V is feed velocity of a cutter-loader, m per minute.

While coal and undercut rock mining, (2.43) formula is:

$$H_w = \frac{P}{60 \cdot r \cdot (m_y \gamma_y + m_{np} \gamma_n) \cdot V}, \tag{2.44}$$

where m_y and m_{np} are the thickness of a seam and undercut accordingly, m; γ_y and γ_n are the density of coal and undercut rock accordingly, t/m^3.

(2.44) formula explains energy consumption for a ton of rock mass with $A_{г.м}$ ash-content assumed according to actual data or determined by (2.30) formula.

To identify specific energy consumption for mining coal with target ash-content norm, that is coal involved in (2.44) formula, it is required to introduce cut coefficient $k_{ск}$ (2.34). Then, (2.44) formula is:

$$H_{wв} = \frac{P}{60 \cdot r \cdot (m_y \gamma_y + m_{np} \gamma_n) \cdot V \cdot k_{ск}}, \tag{2.45}$$

In the process of separate mining coal and undercut rock, (2.43) formula is applied to make distinct determination of energy consuming for coal and rock mining. Following formulae are used:

$$H_{wy} = \frac{P}{60 \cdot r \cdot m_y \cdot \gamma_y \cdot V_y}, \tag{2.46}$$

$$H_{wn} = \frac{P}{60 \cdot r \cdot m_{np} \cdot \gamma_n \cdot V_n}, \tag{2.47}$$

where V_y and V_n are a cutter-loader feed velocity in the process of coal and undercut rock, m per minute.

Total energy consumption per a ton of mined coal is:

$$H_{wc} = \frac{H_{wy} \cdot m_y \cdot \gamma_y + H_{wn} \cdot m_{np} \cdot \gamma_n}{m_y \cdot \gamma_y}. \tag{2.48}$$

Ash-content of mined coal is determined either on actual basis, or by (2.31) formula.

To compare values of specific energy consumption in the process of complete and separate mining, it is required to reduce ash-content of rock mass $A_{гм}$ not to target ash-content norm but to ash-content of coal mined separately ($A_{ду}$); then, cut coefficient $k_{ск}$ is:

$$k_{ск} = 1 - 0.02 \cdot (A_{гм} - A_{ду}). \tag{2.49}$$

In terms of known values of energy consumption, it is possible to determine such a difference ($\Delta A_{г.м}$-$A_{д.у}$), when energy consumption values are similar even if various mining techniques are applied:

$$\Delta A = \frac{1 - \dfrac{H_{wв}}{H_{wc}}}{0.02}. \tag{2.50}$$

$H_{wв} > H_{wc}$ if actual ΔA values exceed that one, calculated according to (2.50) formula; if ΔA is less than calculated value, $H_{wв} < H_{wc}$.

With the help of the expressions one can determine rational mining techniques for thin flat seams as well as minimum specific energy consumption.

Fig. 2.12 - 2.15 demonstrate dependences of specific energy consumption on the thickness of undercut and coal seam as well as mining technique applied. In each case, H_w for coal mining with the use of separate technique is lower to compare with that to mine coal reduced to ash-content norm in the process of complete mining; that very time it is higher to compare with specific energy consumption to mine rock mass.

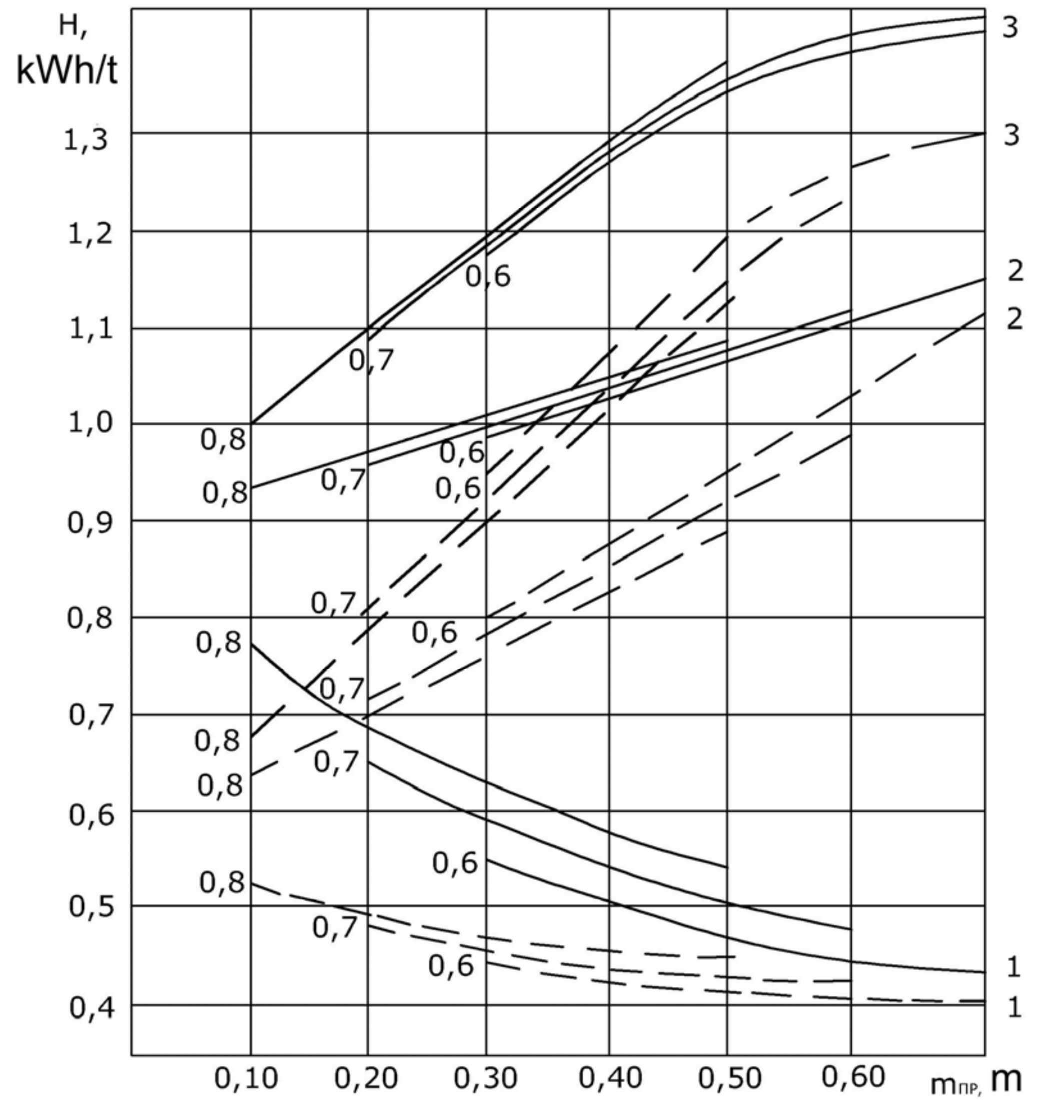


Fig. 2.12. Dependence of specific energy consumption on the thickness of a seam and undercut roof rocks:

1. *On rock mass if complete mining.*
2. *On coal if separate mining.*
3. *On ash-content reduced if complete mining.*

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

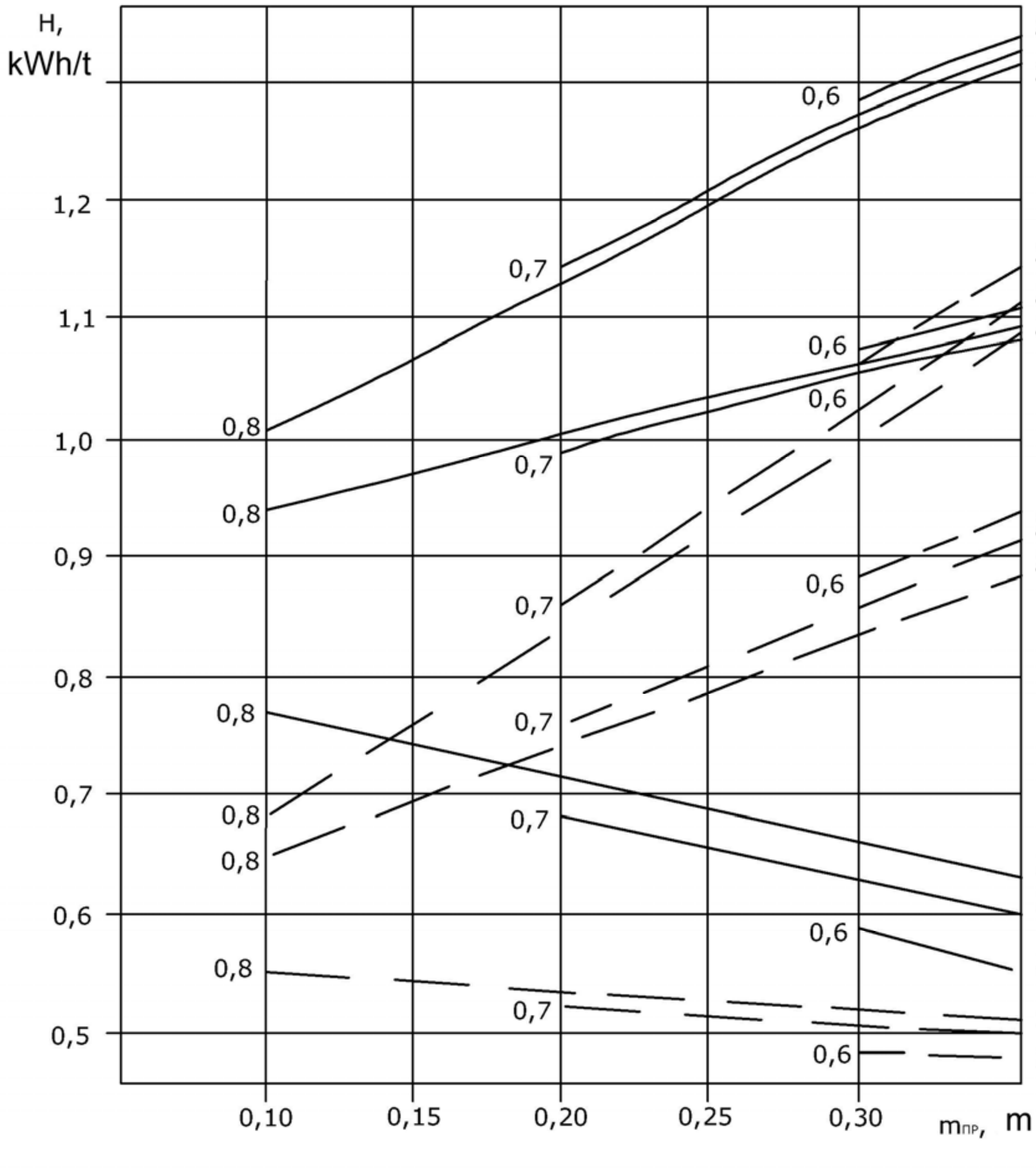


Fig. 2.13. Dependence of specific energy consumption on the thickness of a seam and undercut floor rocks:

1. *On rock mass if complete mining.*
2. *On coal if one-pass separate mining.*
3. *On ash-content reduced if complete mining.*

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

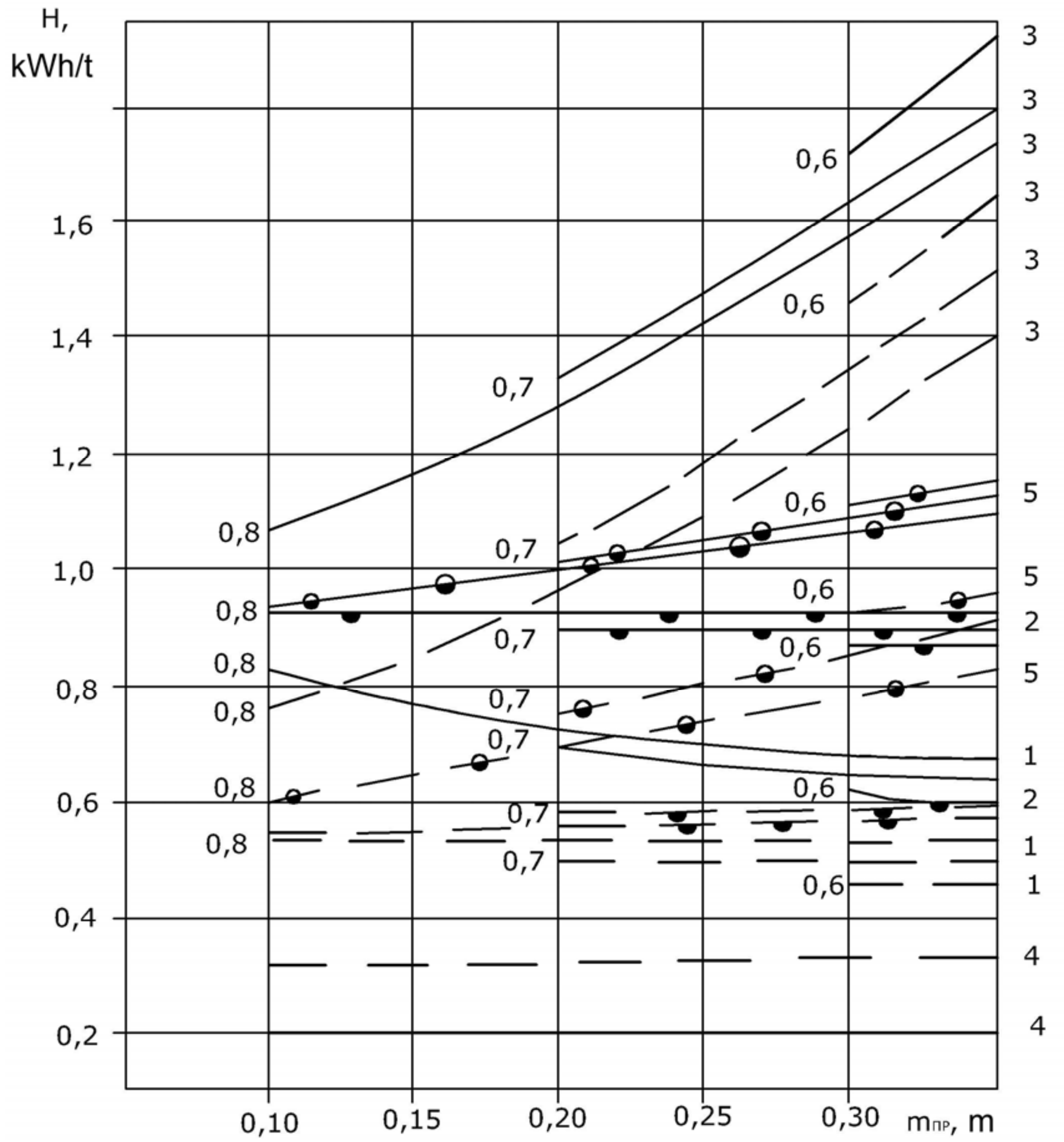


Fig. 2.14. Dependence of specific energy consumption on the thickness of a seam and undercut floor rocks:

1. *On rock mass if complete mining.*
2. *On coal if two-pass separate mining.*
3. *On ash-content reduced if complete mining.*
4. *On rock.*
5. *Total if separate mining.*

________ - Western Donbass; __ __ __ - Lvov-Volyn coal field.

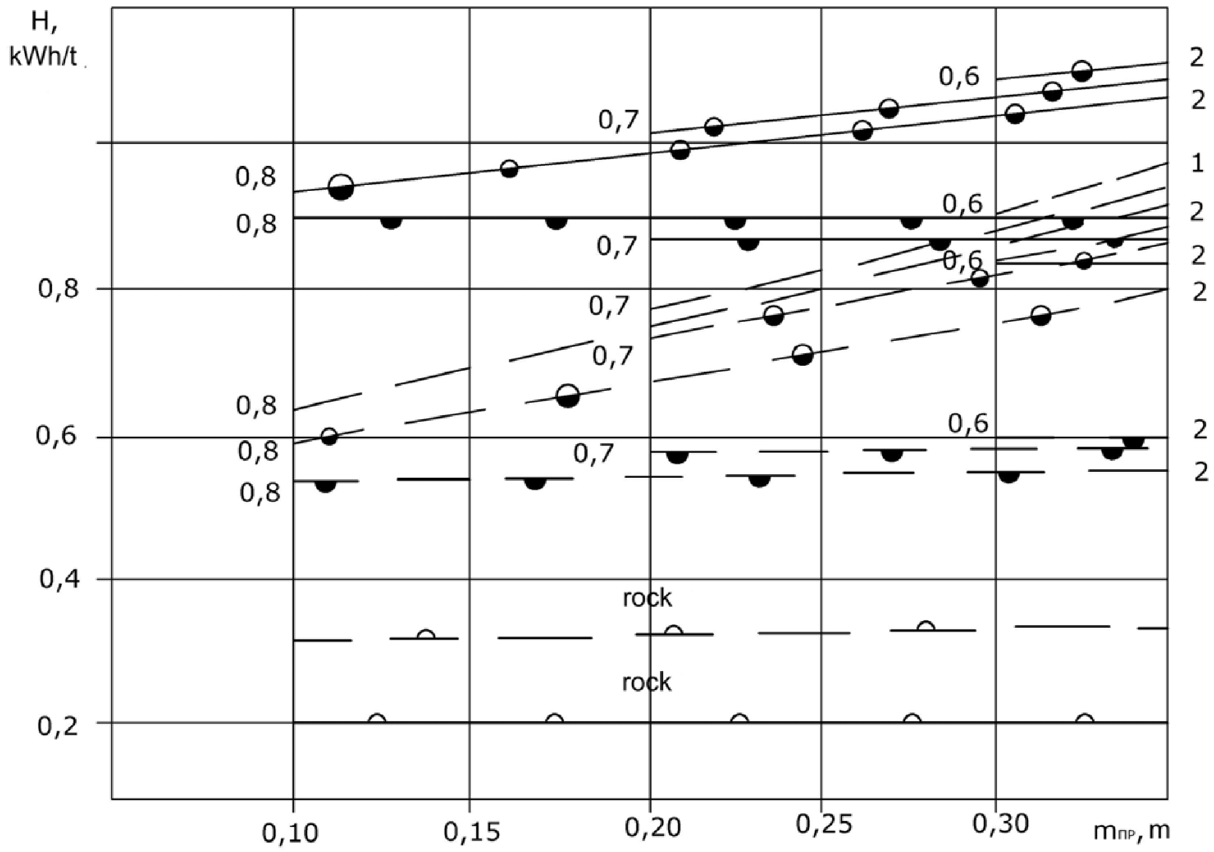


Fig. 2.15. Dependence of specific energy consumption on the thickness of a seam and undercut floor rocks if separate mining:

1. *One-pass technique.*
2. *Two-pass technique.*

________ *- Western Donbass;* __ __ __ *- Lvov-Volyn coal field.*

Fig. 2.15 demonstrates dependences of specific energy consumption while using two mining techniques (one-pass technique and two-pass technique). It is understood that in the context of Western Donbass specific energy consumption for two-pass mining is some higher to compare with one-pass technique. As for the Lvov-Volyn coal field, one-pass mining is more energy intensive to compare with two-pass technique. Moreover, the difference rises if the thickness of undercut increases. Hence, from the viewpoint of specific energy consumption decrease per a ton of mineral mining, the following are the most rational: two-pass separate seam mining for mines in Lvov-Volyn coal field, and one-pass technique for mines in Western Donbass.

2.4. Principal Techniques for Selective Thin and Very Thin Flat Coal Seams Mining

Twelve principal techniques for selective mining seams with enclosing roof and floor have been developed basing upon the research of basic parameters. Within the classification each technique is subdivided according to its nature and mining plan. The two mining types are considered: with undercut of floor rocks and roof rocks (with advancing coal or rock mining). Fig. 2.16 demonstrates principal techniques for seam selective mining.

Technique 1.1 provides advancing coal seam mining alongside a longwall with following break of undercut floor rocks. With this capability of roof support after mining is provided. Dimensions of cutter-loader end organs should match up the coal seam thickness.

Technique 1.2 provides advancing undercut rock cut along the whole longwall with following cut of coal seam. The technique is recommended to be applied in longwalls with hard coal and soft floor rocks. It provides not only decrease in mined coal ash-content but also increase in coarse grain yield, and energy consumption for coal cutting decreases.

Technique 1.3 (1.3.1 and 1.3.2) provides simultaneous coal and floor rocks with advancing coal seam cut. A cutter-loader with spaced end organs is applied for simultaneous cut. Size of advancing end organ should match the thickness of coal seam. After the cutter-loader pass, face space is supported. When specific conveyer, equipped with devices providing cut rock loading, is used, then the conveyer is advanced to a new road without the cutter-loader moving. That is 1.3.1 technique provides the capability of one-pass coal seam and undercut rock selective mining.

Technique 1.4 (1.4.1 and 1.4.2) provides simultaneous coal and floor rock mining with undercut rock advancing cut. As in previous case, cutter-loaders with spaced end organs are applied. Undercut value depends on used organ dimensions. It is recommended to be applied if hard coal and soft floor rocks.

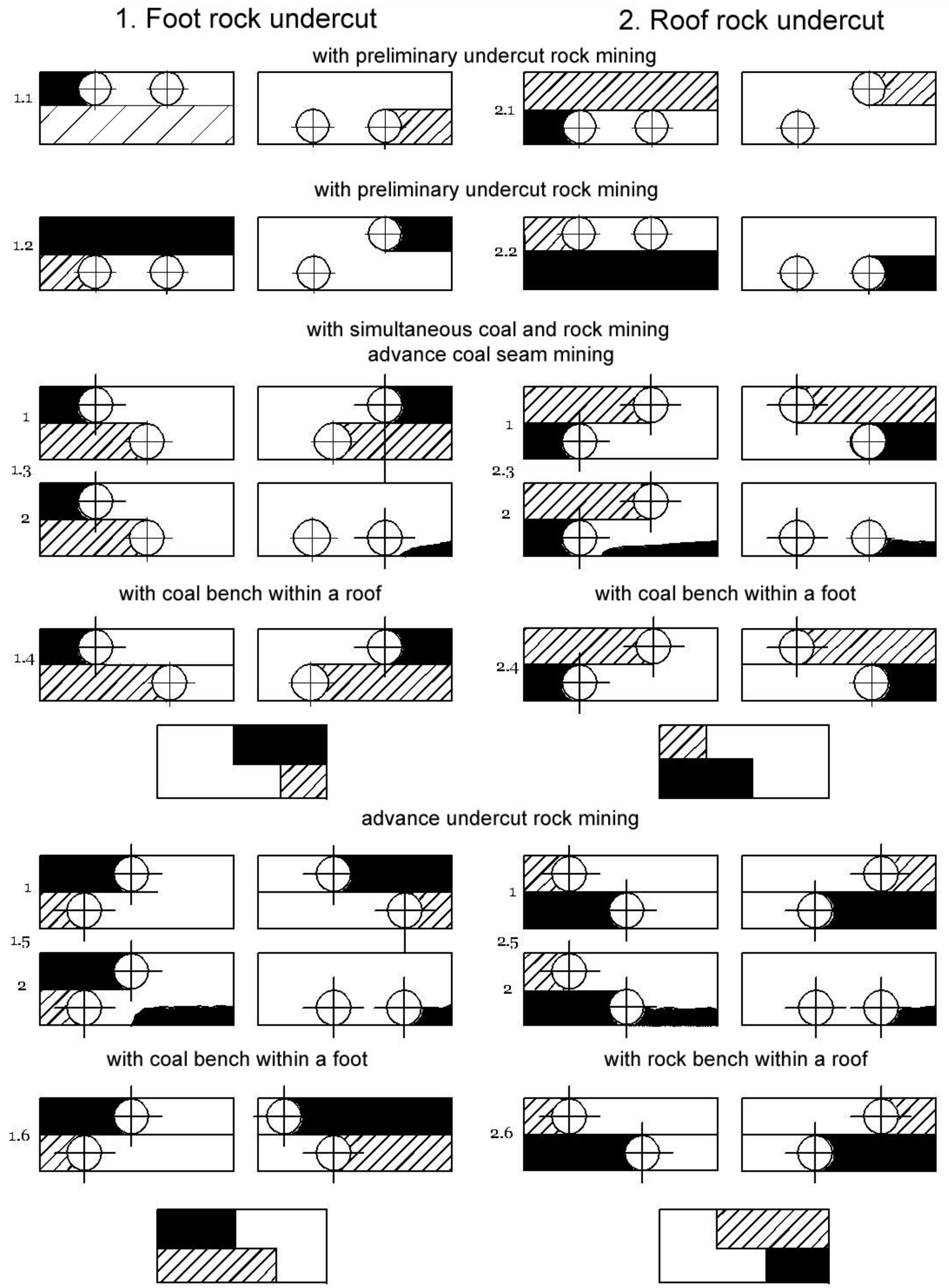


Fig. 2.16 Principal techniques for thin and very thin seam selective mining

It increases yield of coarse grain, and decreases material and energy consumption in the process of hard coal cut. As in the previous technique, unidirectional coal and rock mining (1.4.2) as well as one-pass shuttle one is possible (1.4.1).

Technique 1.5 provides simultaneous coal and floor rock mining with web width coal seam advancing. To the effect, cutter-loaders with horizontally spaced end organs are applied. The technique provides combined one-pass coal and rock mining. While using mechanized system of MKZD type, it is recommended to mine thin and very thin seams under complicated mining and geological conditions.

Technique 1.6 provides simultaneous coal and floor rock mining with web width undercut rock advancing. It is recommended for hard coal seams and soft floor rocks mining.

Technique 2.1 provides preliminary coal seam mining along the whole longwall with following cut of undercut roof rocks. It is recommended for seams with hard floor and roof rocks. The technique provides capability of hard roof rocks cut by means of production cutter-loaders with massive dust reduction.

Technique 2.2 provides preliminary mining undercut roof rocks with following coal seam mining. That provides exposing roof support, and increase in coarse grain yield. While hard coal seam mining, material and energy consumption reduce for coal cutting. It is recommended to be applied for seams with low-thickness false roof. It this context, a diameter of a cutter-loader end organ should match the false roof thickness.

Technique 2.3 (2.3.1 and 2.3.2) provides simultaneous coal and roof rock mining with advancing coal seam mining. The technique provides one-pass capability for coal and rock mining (2.3.1). It is recommended if soft coal and hard enclosing floor and roof rocks.

Technique 2.4 (2.4.1 and 2.4.2) provides simultaneous coal and roof rock mining with advancing undercut rock mining, and capability to apply shuttle technique (2.4.1). It is recommended if weak and false narrow roofs and hard coal. It provides decrease in mining energy consumption and increase in coarse coal yield.

Technique 2.5 provides simultaneous coal and rock mining with web width advancing undercut rock mining. A system of mining equipment of MKZD type can be applied for that as it provides one-pass combined coal and rock mining. The use of mining pressure to break coal bench makes it possible to decrease significantly material and energy consumption for mining, and continuous roof of face zone helps to liquidate roof inrush. The technique is recommended if mining thin and very thin coal seams takes place under complicated mining and geological conditions.

Technique 2.6 provides simultaneous coal and roof rocks mining with web width undercut rock mining. The technique is recommended for seams with hard enclosing roof and floor and stable roof [100].

Conclusions

1. A dependence of minimum mined seam thickness basing upon data by DonCI and AUIC as for analogous m_{min} dependences on design parameters of a stope equipment and mining and geological conditions has been identified. It differs in the fact that makes synchronous consideration of physiological parameters required to provide high labour efficiency, technological and time factors characterizing the technology features as well as maximums of longwall rocks convergence. That helps to apply it to determine minimum mined thickness and required enclosing roof and floor undercut in the process of both complete and selective seam mining.

2. A dependence of a cutter-loader feeding velocity on power, consumed by its engines, web width, and mining and geological conditions is determined. It differs in taking into consideration an order of coal and rock benches mining as well as their weakening because of advancing cut. As a result, an expression obtained makes it possible to determine a cutter-loader feeding velocity depending upon a type of separate seam mining.

3. A dependence of machine time of technological parameters of a stope is determined. It differs in taking into consideration the effect of coal seam and enclosing roof and floor undercut, mining technique, and time to mine coal as such and rock undercut which helps to use it for selective seam mining.

4. A dependence of specific energy consumption on energy input by a cutter-loader, web width, and mining and geological conditions is determined. It differs in taking into consideration the applied mining technique as well as a grade of coal mined. That helps to apply the technique for various seams mined with rock undercut, and to make comparative analysis of the techniques involving the product grade.

5. A dependence of mined coal ash-content basing upon adequate recommendations of known techniques. It differs in taking into account the completeness of undercut rock while separate mining. That helps to determine ash-content more accurately while applying various techniques to mine thin and very thin coal seams with enclosing roof and floor undercut.

6. It is established that while mining seams with rock undercut, capacity of cutter-loaders should involve mined coal grade; to do that, a coefficient, taking into account cuts

for increased ash-content, is introduced into a known formula of working capacity. That helps to evaluate performance of stopes and a mine on clean coal rather than on total output or rock mass.

7. Among other things, analysis of results calculated with the help of dependences obtained, shows that:

- Thickness of thin flat seams mined in Western Donbass and Lvov-Volyn coal field, should not be less than 0.90…0.95 m for KMK97 and KMK98 systems; 1.02…1.05 m for KD80 system, and 0.87…0.9 m for 1KM103 system. Thinner seams mining are possible if only enclosing roof and floor undercut is applied;

- Feeding velocity of a cutter-loader to mine a seam having two outcropping flats, slightly depends on hardness and thickness of coal and mined rock (if $m \leq 0.8$ m and $m_{np} \leq 0.35$ m) reaching its maximums ($V \leq V_{\partial on}$);

- Almost in each case specific energy consumption to mine clean coal using separate mining technique is lower to compare with complete mining the coal reduced to ash-content norm;

- 0.01m undercut increase in the process of various-thickness seams complete mining ($0.6\,\text{m} \leq m_y \leq 0.8\,\text{m}$) results in 0.4…1.2% extra dilution of mined coal; if one-pass separate mining, the figure is 0.1…0.2%; two-pass mining slightly effects the grade of mined coal;

- Almost within the whole range of 0.10 to 0.70 m undercut, a cutter-loader capacity, reduced to ash-content norm, is much lower than that in the process of separate mining.

8. Selective mining thin and very thin seams, using available mechanized systems, makes it possible to apply twelve principal techniques; six of them apply floor rock undercut, and other six – roof rock undercut.

CHAPTER 3. UNDERGROUND INVESTIGATIONS OF SEAM SELECTIVE MINING

3.1 General Conditions

A number of full-scale investigations were carried out in mines of Western Donbass and Lvov-Volyn coal field to review basic theoretical principles obtaining values of actual parameters for thin coal seam mining. In addition, the investigation main aim was to recognize technical and economic feasibility for selective seam mining, to obtain factual values of different techniques parameters justifying rational fields and volumes for their in situ application.

The investigations were governed by dedicated procedure involving basic rules and demands of known branch techniques.

3.2 Conditions and Site

The investigations were performed in longwalls, mining thin flat seams being the most typical for Western Donbass and Lvov-Volyn coal field. Fig. 3.1 demonstrates mining and geological characteristics of the longwall as well as stratigraphic columns of enclosing roof and floor seams. Table 3.1 shows that thickness of mined coal seams varies within 0.65 to 0.82 m; undercut thickness varies within 0.2 to 0.8 m, and hardness of undercut enclosing roof and floor varies within $f = 1.5$ to $f = 5$ according to classification by professor M.M. Protodiakonov. Hardness and cuttability of coal seams and enclosing roof and floor vary in longwalls of the Associations. Thus, in mines of Western Donbass coal cuttability is almost twice higher to compare with enclosing roof and floor cuttability. On the contrary, in mines of Lvov-Volyn coal field, rock cuttability is twice higher to compare with coal seam cuttability. Hence, it is permitted to compare some results of the investigations (e.g. feeding velocity of a cutter-loader, energy consumption, mining time etc.) obtained in one regions with other region conditions replacing in this context relative location of coal and undercut rock. In such a way, results obtained in the process of preliminary seam mining and floor rock undercut in terms of Western Donbass may be considered as those obtained in terms of Lvov-Volyn coal field with preliminary roof rock mining and subsequent seam mining, and vice versa. The idea may help broadening application field of the techniques.

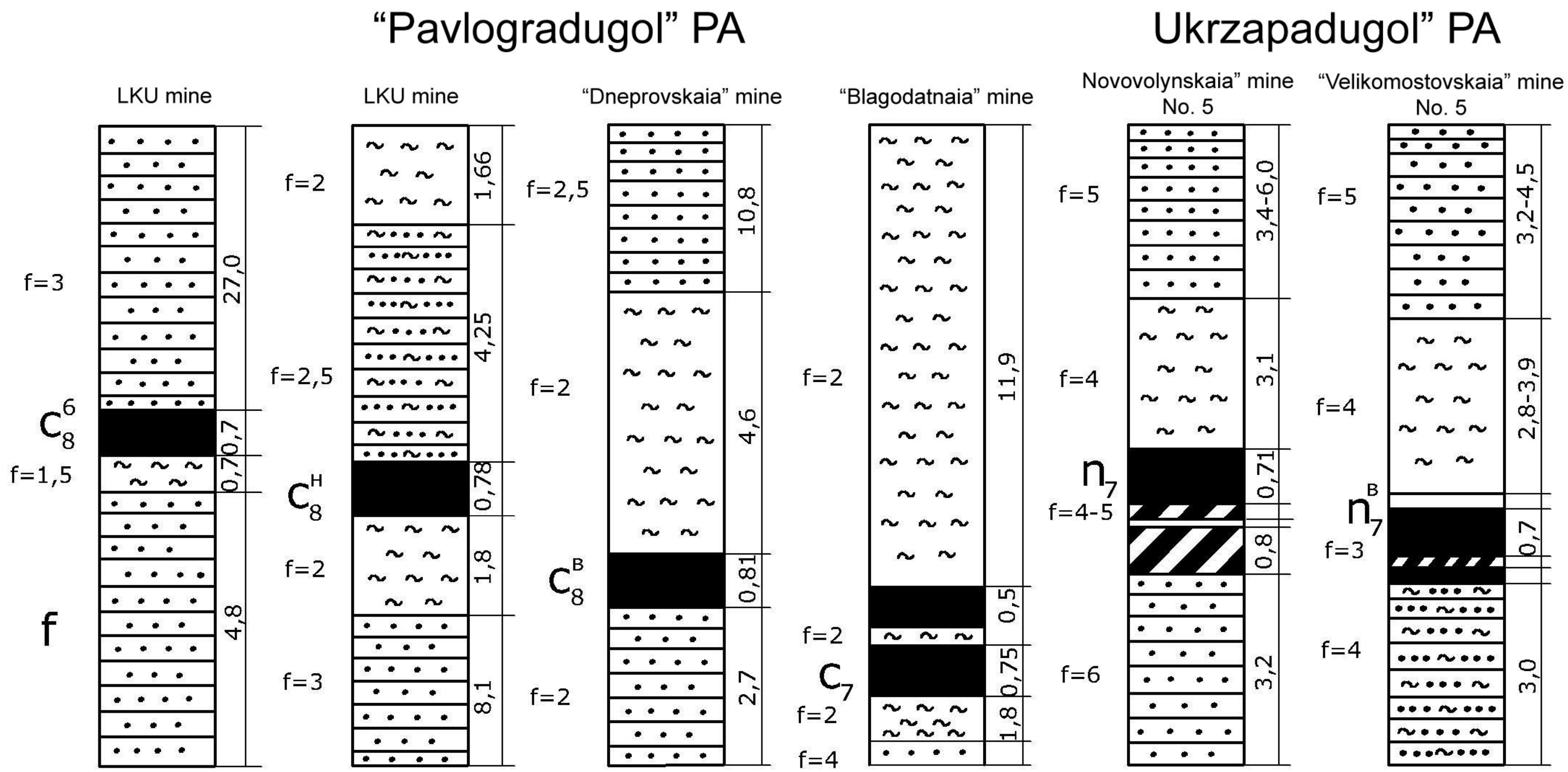


Fig. 3.1. Stratigraphic columns of seams and enclosing roof and floor

Table 3.1. In-situ characteristics of considered longwalls

Factors	"Pavlogradugol"				"Ukrzapadugol"			
	"Zapadno-Donbasskaia" mine		"Dneprovskaia" mine	"Blagodatnaia" mine	"Velikomostovskaia" mine No. 5		"Novovolynskaia" mine No. 5	
	seam $C_8^{В}$ longwall 905	seam $C_8^{Н}$ longwall 814	seam $C_8^{В}$ longwall 859	seam C_7 longwall 708	seam $n_7^{В}$ longwall 21	seam $n_7^{В}$ longwall 24	seam n_7 longwall 17	seam n_7 longwall 26
Coal seam thickness, m	0.7	0.7	0.82	0.7-0.8	0.68-0.75	0.65-0.72	0.68-0.8	0.7-0.77
Undercut rock thickness, m	0.4	0.4	0.28	0.7-0.8	0.3-0.35	0.3-0.35	0.3-0.22	0.32-0.25
Undercut location	floor	floor	floor	roof	floor	roof	floor	floor
Coal hardness coefficient	3	3	3	3	1.5-2	1.5-2	2	2
Rock hardness coefficient	1.5	2	2	2	3	3-3.5	4-5	4-5
Coal cuttability, kN/m	280	290	30	300	180	180	200	200
Rock cuttability, kN/m	160	170	180	170	320	340	450	450
Coal seat	aleurolite	aleurolite	aleurolite	aleurolite	sandy shale		coal shale	
Immediate roof	argillite	argillite	argillite	coal batch	aleurolite	aleurolite	aleurolite	aleurolite
Longwall length, m	170	180	180	175	170	175	150	150
Mining equipment	KD80 1K101	KD80 1K101	KD80 1K101	KM88 1K101	KMK97 1K101(2)	KMK97 1K101(2)	1KM103 1K103	1KM103 1K103
Web width, m	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8
Mined coal ash-content, %	56.0	–	–	47.9	–	61.0	43.9	–
Coal seam ash-content, %	12	–	–	19.5	–	36.2	17.8	–
Undercut rock ash-content, %	93.0	–	–	89.0	–	90.0	79.0	–

3.3. Selecting Operation Schedule

To select rational operation schedule, for each case the analysis of mining and geological, and economic parameters is carried out. The key criterion of this or that schedule is its technical and economic feasibility. Selected operation schedule should meet the highest economic performance both for the longwall and the whole mine. Mining and geological conditions as well as the longwall equipment potential are the constraint factors in the process of transition from complete mining to separate one. However, even the case implies more than one feasible operation alternative making it possible to select the most reasonable.

In terms of Western Donbass, schedules with floor rock undercut are rational ones from the viewpoint of mining and geological conditions. The matter is that in the case of weak roof, its discontinuity is possible. From the viewpoint of decreasing energy consumption per mining a ton of coal, operation schedules with soft rock preliminary or advanced mining are the most efficient; that helps to weaken hard and viscous coal seam. However, available mining equipment makes it possible to mine first floor rocks which minimum thickness is not less than 0.6 m. As a consequence, it is required to make technical and economic substantiation of the technique comparing it with simpler ones which provide preliminary coal seam mining. If latter, we lose on increase in specific energy consumption benefitting from decreasing volumes of mined rock as it becomes possible to control the undercut thickness.

In terms of Lvov-Volyn coal field the majority of roof rocks are of mean stability, and they have the opportunity to apply both techniques with floor undercut, and with roof undercut as well.

As experiments demonstrate, available mining equipment enables enclosing roof and floor undercut if their hardness is not more than f =3…4 by Professor M.M. Protodiakonov.

Set forth, critical assessment of all feasible mining techniques and analysis of their technical and economic efficiency make it possible to choose the most reasonable operational schedules of selective mining for mining and geological conditions under study (Table 3.2).

The Table data demonstrate that for Western Donbass where the majority of mines with undercut have mechanized support “Donbass” and 1K101UD cutter-loader, technique 1.1 is the simplest with advanced coal seam mining and subsequent soft floor rock breaking. Such a technique is officially accepted for five stopes. Fig. 3.2 shows a fragment of a longwall support pattern.

Table 3.2 Applied operation schedules

Region, mine, longwall	Seam No.	Mining equipment	Technique
Western Donbass			
“Zapadno-Donbasskaia” mine			
1. Longwall 905	C_8^B	"Donbass" 1K101	complete; selective (1.1)
2. Longwall 814	C_8^H	"Donbass" 1K101	complete; selective (1.1)
“Dneprovskaia” mine			
3. Longwall 1026	C_{10}^B	"Donbass" 1K101	complete; selective (1.1)
4. Longwall 859	C_8^B	"Donbass" 1K101	complete; selective (1.1)
“Blagodatnaia” mine			
5. Longwall 708	C_7	KM-88, 1K101	complete; selective (2.2)
Lvov-Volyn coal field			
Mine 5 “Velikomostovskaia”			
6. Longwall 21	n_7^B	KMK-97, 1K101	complete; selective (1.1)
7. Longwall 24	n_7^B	KMK-97, 1K101	complete; selective (2.1)
Mine 5 “Novovolynskaia”			
8. Longwall 17	n_7	1KM103, 1K101	complete; selective (1.1, 1.3.1)
9. Longwall 26	n_7	1KM103, 1K103	complete; selective (1.1)

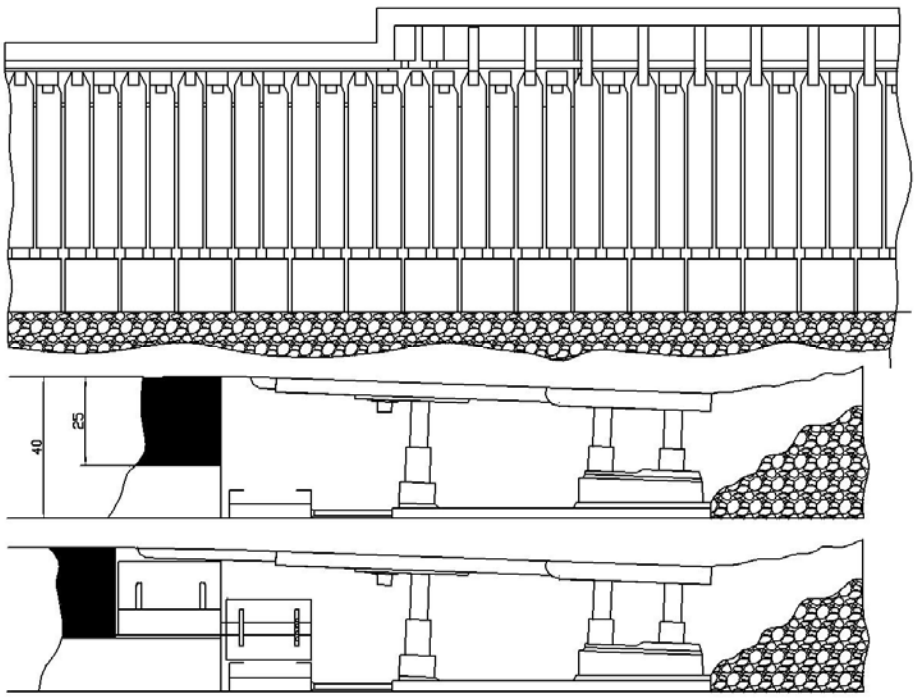


Fig. 3.2 A technique of selective seam mining with advanced coal winning and subsequent undercut of floor rocks in a longwall equipped with mechanized system “Donbass”

Initially sections of "Donbass" support are screwed to a conveyer, and the conveyer is close to a stope. A cutter-loader is in a rock shelter at the boundary entry; starting, it mines a coal seam which thickness is 0.7 and 0.8 m along the whole length of the longwall. Backing 1.0…1.5 m out of end organ of the cutter-loader, bars of sections of mechanized support "Donbass" protract. When ribbing has been mined and composed entry has been reached, the end organs get down. Moving in the opposite direction, the cutter-loader mines 0.3…0.4 m rock bench left in a floorwall. Subsequent to the cutter-loader move, the support sections and the cutter-loader flight are shifted to a new path. Then the cycle is restarted.

Specific mining and geological conditions of longwall 708 (c_7 seam in "Blagodatnaia" mine) make it possible to test technique 2.2 with advanced roof rock mining. Fig. 3.3 shows fragments of the longwall support pattern.

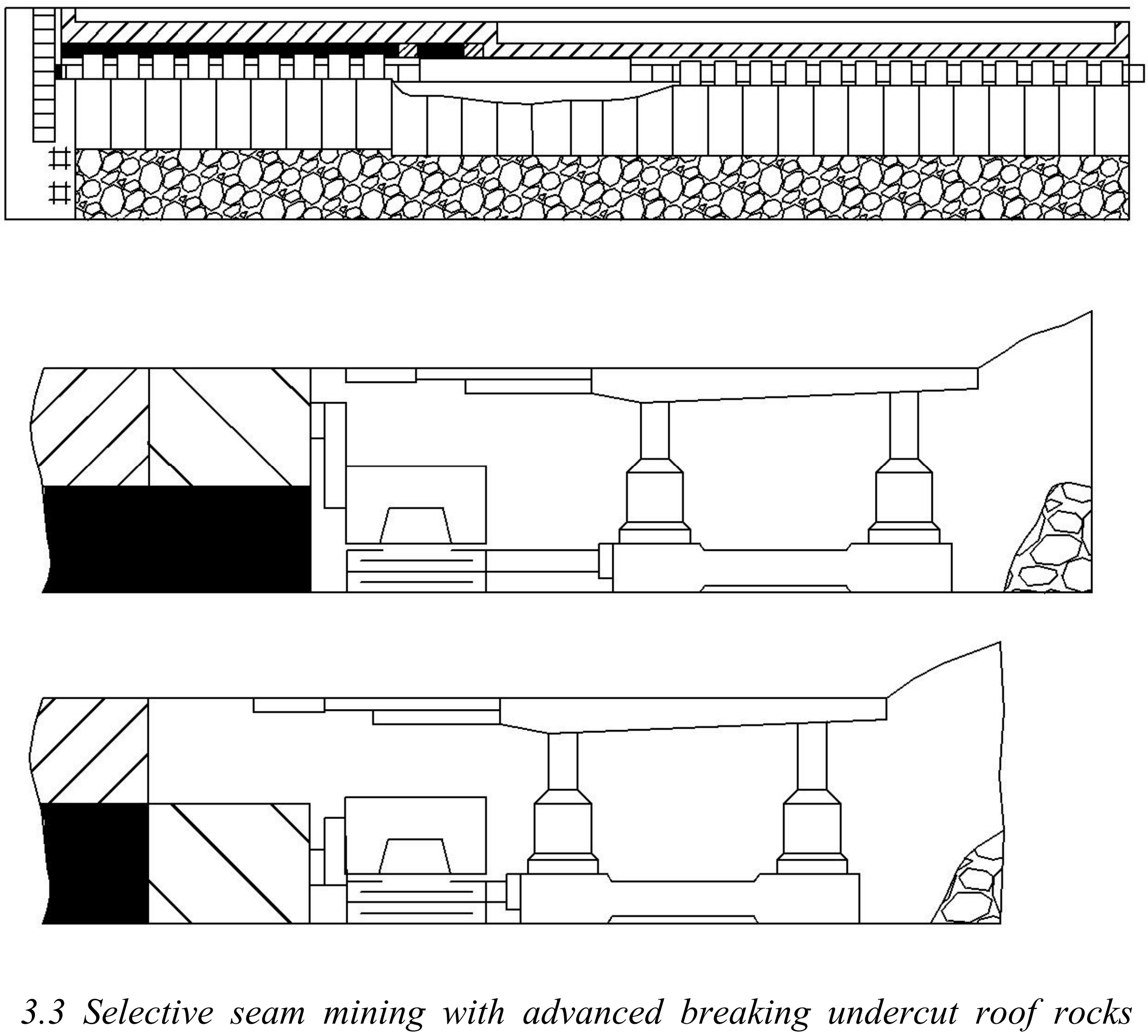


Fig. 3.3 Selective seam mining with advanced breaking undercut roof rocks and subsequent coal seam mining in a longwall equipped with mechanized system KM88

Initially sections of M88 support are spaced from a conveyer per advance increment (loaded technique). Moving from haulage roadway to boundary entry, 1K101U cutter-loader mines undercut rock as well as top coal; total thickness is 0.7…0.8 m. Support sections follow the cutter-loader. After the rock has been mined, coal seam is excavated in the opposite direction; then multi-point conveyor advancing to a new roadway takes place. Then the cycle is restarted.

Under the conditions of Lvov-Volyn coal field, 1.1 and 2.1 techniques were officially accepted in longwalls equipped with mechanized system KMK97, and 1.1 and 1.2 techniques were officially accepted in longwalls equipped with 1KM103 mechanized system. Figs. 3.4, 3.5, 3.6 and 3.7 demonstrate fragments of supports patterns of stopes equipped with KMK97 systems, and performance of two 1K101cutter-loaders in the process of mining a seam with floor and roof undercut.

Position I (Fig. 3.6) is considered as the start of winning cycle if floor rock undercut takes place. In this context basic cutter-loader is in undercut zone, 20 to 30 m spacing boundary entry and auxiliary cutter-loader is in a high side. The conveyer flight is flexed within a cutting area, retracing the working face form. Basic cutter-loader extracts coal seam from the cutting area to boundary entry. Sections of the support with mounting bases are resited behind the cutter-loader. Mining and cleaning up of coal seam within upper share of the longwall takes place after the cutter-loader made its move over sectionalizing arrangement (positions II and III). When coal seam has been mined, then basic cutter-loader is left in a lower part of the longwall; auxiliary cutter-loader extracts and cleans up rock bench formed in the seam floor from boundary entry up to a cutting area (positions IV and V). After that, basic cutter-loader performs beating and loading rock of residual bench from haulage roadway up to cutting area. The support sections with spring consoles and the conveyer flight are moved next. At the cutting area, the cutter-loader is stopped; retraling support sections and the conveyer flight are shifted to a new path. Then the cycle is restarted.

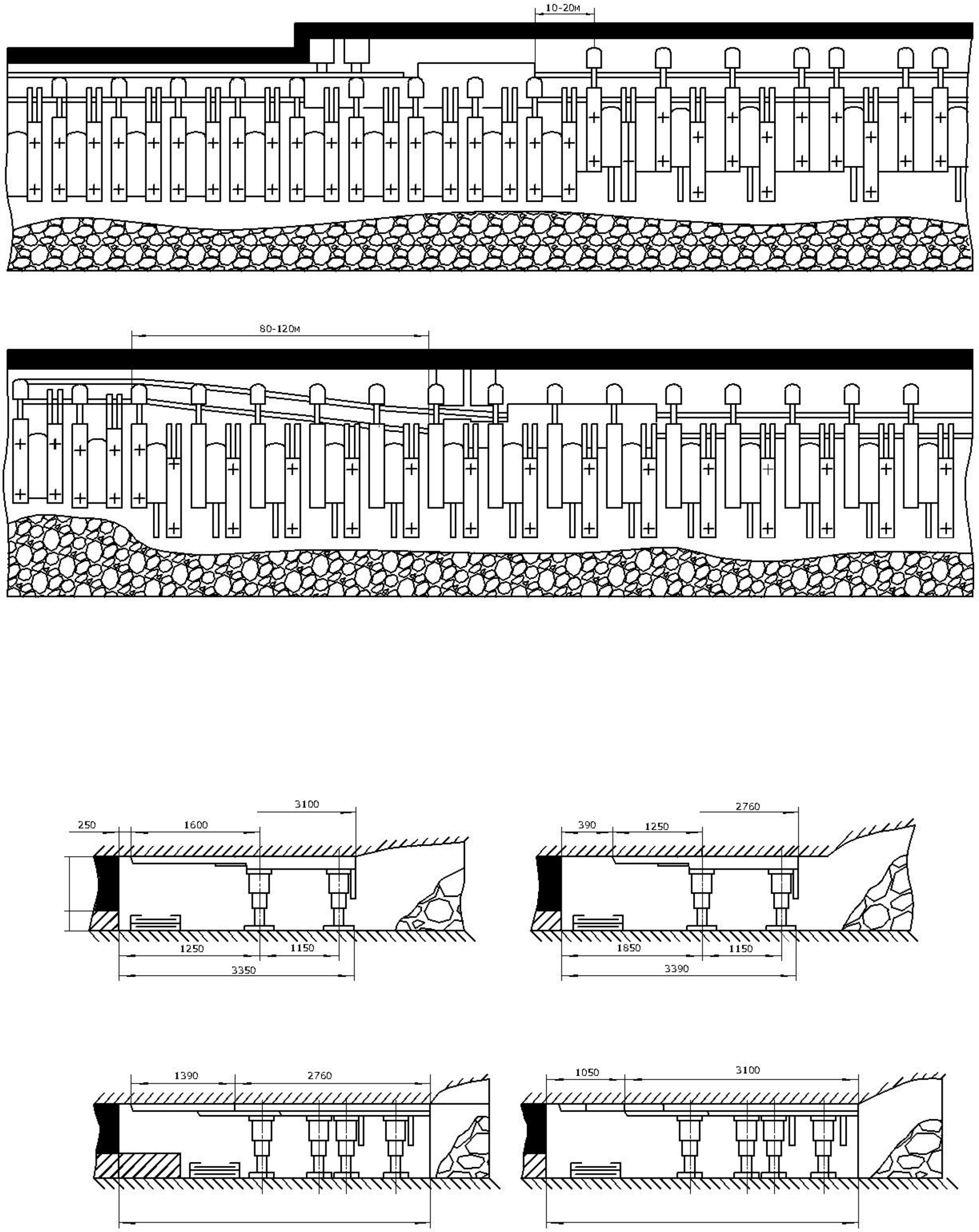


Fig. 3.4 A technique of two-pass selective mining with floor rock undercut

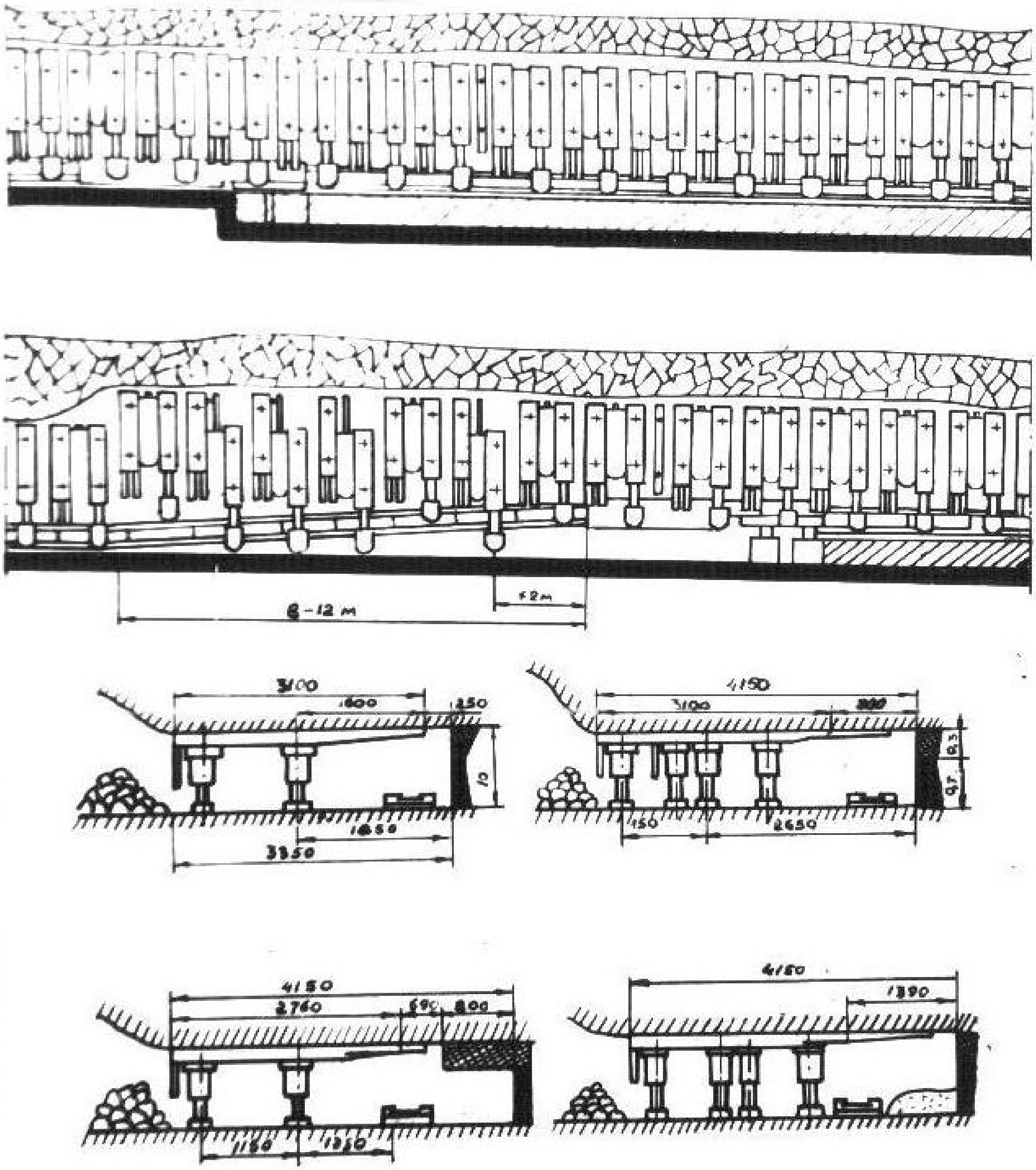


Fig. 3.5 A technique of two-pass selective mining with roof rock undercut

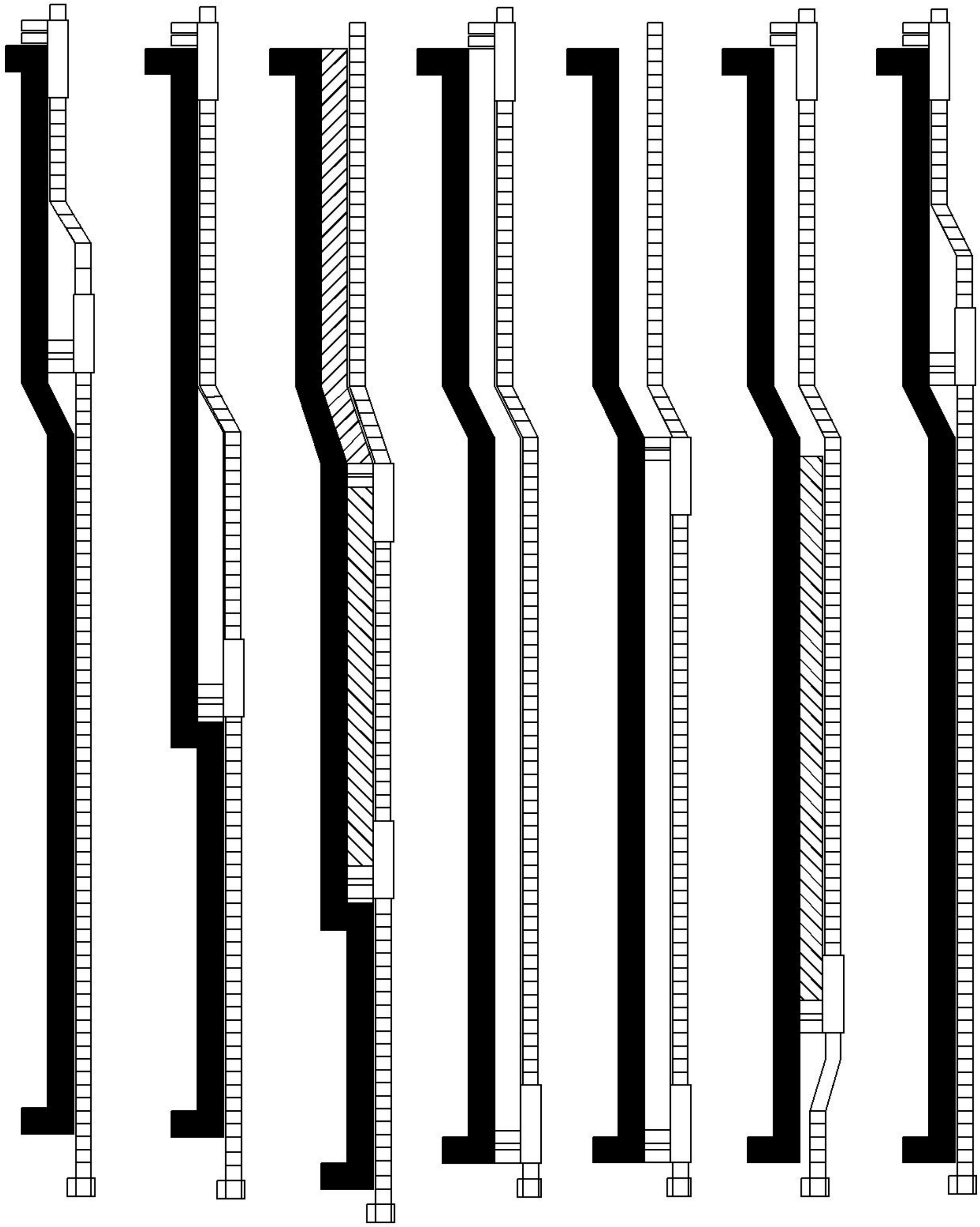


Fig. 3.6 Operation scheme of two 1K101 cutter-loaders in the process of selective seam mining with floor rock undercut

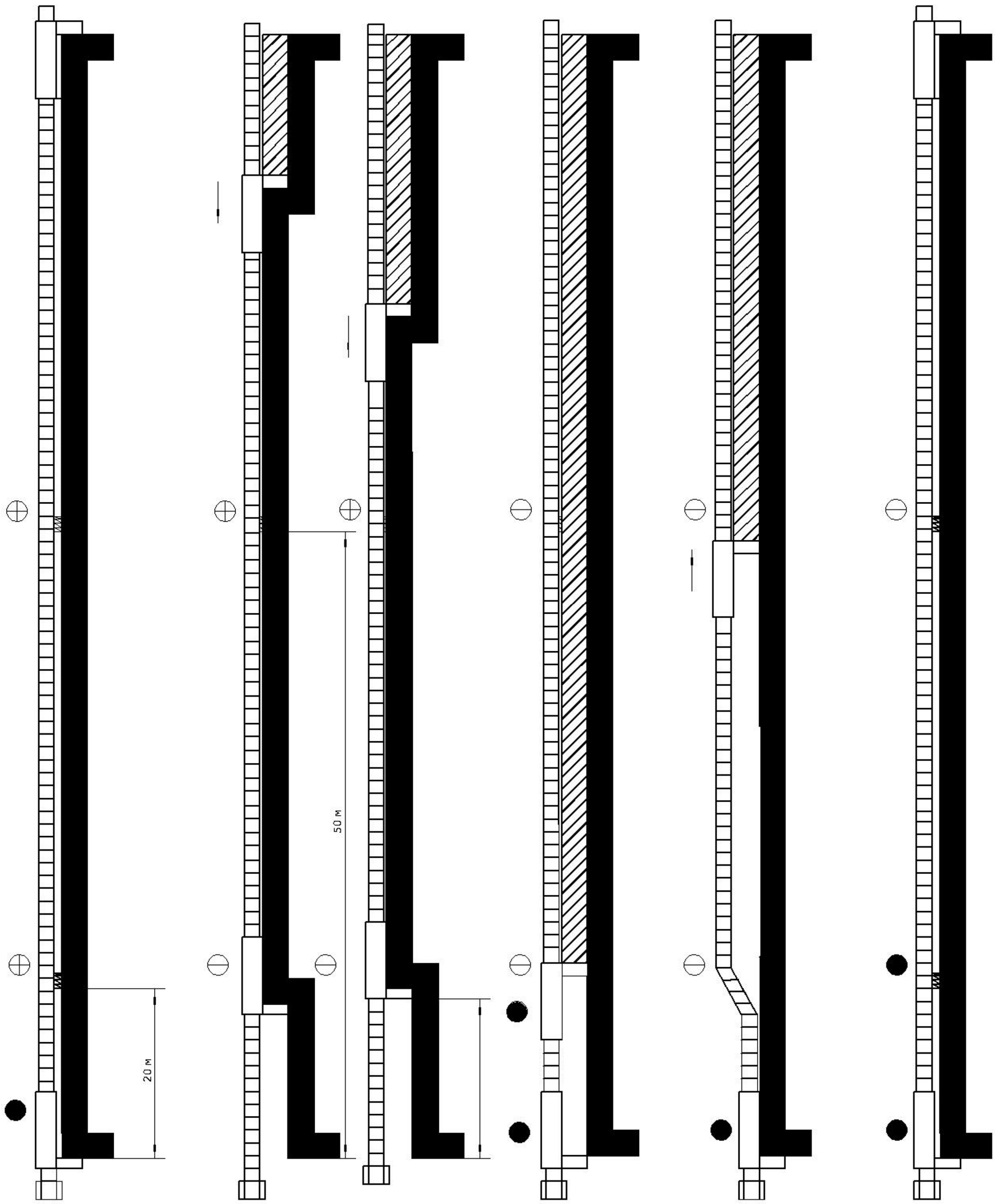


Fig 3.7 Operation scheme of two 1K101cutter-loaders in the process of selective seam mining with roof rock undercut

Position I (Fig. 3.7) is considered as a starting point of mining cycle when basic cutter-loader is in upper part of a longwall in a cutting state, and auxiliary cutter-loader is in a lower part. Heading, basic cutter-loader and auxiliary one perform coal mining. In this context, basic cutter-loader mines coal which thickness 0.7 to 0.8 m is leaving a roof rock bench with 0.25 to 0.30 m; auxiliary cutter-loader mines coal for the whole mining height (position II). When auxiliary cutter-loader prepares cutting area moving away from haulage roadway at 15 to 20 m, it backs up effecting the seam floor cleaning-up (position III). Sections of powered system follow the auxiliary cutter-loader. When basic cutter-loader finishes coal extraction moving to a cutting area, routing inspection of the cutter-loader, loading colter resetting and other required operations take place (position IV). Lifting advance end organ to a seam roof and moving from cutting area to a boundary entry, the cutter-loader breaks roof rock bench conveying rock (position V). Sections of powered system with controlled consoles follow the cutter-loader. With 10 to 12 m back, conveyer flight is moved to a working face; then sections with spring consoles are moved. When cutting by basic cutter-loader is over, set of actions aimed at preparing for the following cycle is performed (position VI). Then the cycle is restarted.

Figures 3.8 and 3.9 demonstrate fragments of charts of stopes equipped with powered system 1KM103 while one-pass and two-pass selective mining.

One-pass seam mining is almost similar to technique of complete coal mining and undercut rock mining. The difference is that extracting a seam, advance screw conveyors coal, and retarding screw cuts floor rocks (or roof rocks) without conveying.

Two-pass mining takes place in the following order: 1K103cutter-loader mines coal seam which thickness is either 0.6 or 0.8 m (depending upon screw diameter); then support sections are moved (scheduled technique of support movement). When coal has been mined, cutter-loader backs up cutting and conveying undercut floor rocks which thickness is up to 0.36 m; when a cutter-loader passes to a new path, then face conveyor is moved there as well. Then the cycle is restarted.

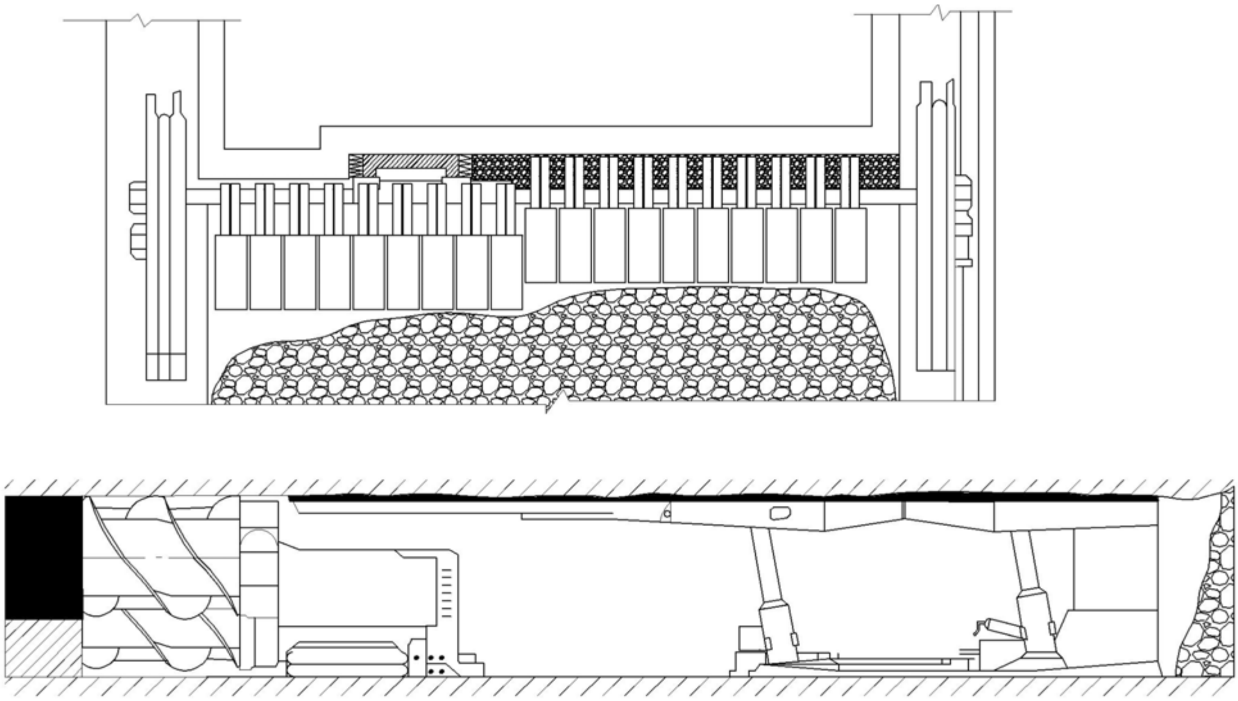


Fig 3.8 A technique of one-pass selective seam mining with floor rock undercut in a longwall equipped with powered system KM103

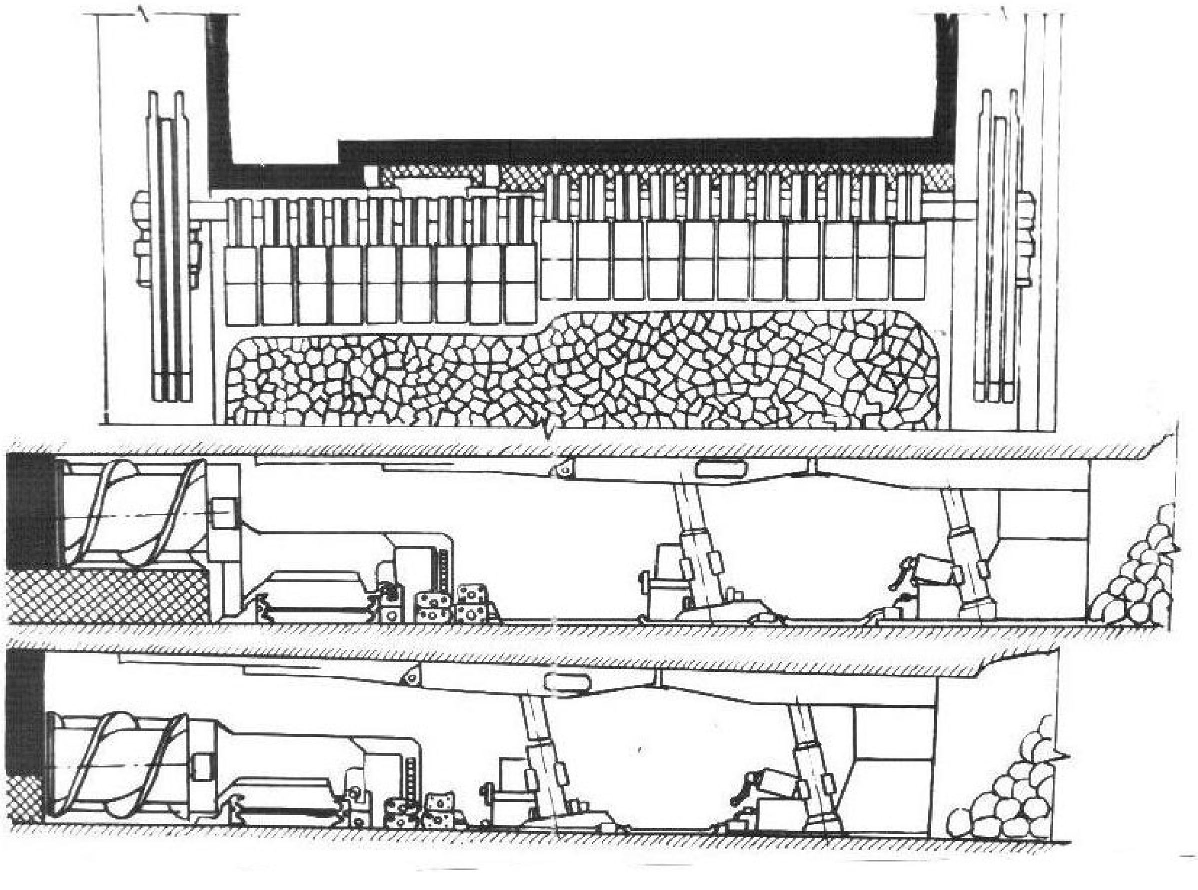


Fig. 3.9 A technique of two-pass selective seam breaking with floor rock undercut in a longwall equipped with powered system КМ103

3.4. Principles of the Research

Field observations were performed to determine practical parameters of complete and selective seam mining in stopes being representative for considered production associations. For example, in Western Donbass there were chosen longwalls mainly equipped with KD80K system widely used in mines of the association. In Lvov-Volyn coal field there were researched longwalls equipped with KMK97 and 1KM103 systems being typical for their mines.

The following was subject to intensive research:

1. Mining and geological conditions with the use of both mine documentation and own observations;

2) State, stability, nature of movement and fall of immediate roof, rock or coal bench;

3) Values and rates of enclosing roof and floor displacement;

4) Actual resistance and pliability of support legs.

The measurements were performed with the help of manometers, loggers, multi-purpose indicator piece legs, and other equipment. Three or four support sections located in the center of the longwall were equipped with measuring instruments.

In the process of coal and rock transporting, the conveyer efficiency was measured visually. In this context running speed of haulage chain and a conveyer no-contact with a stope as well as amount of coal and rock (which stays to be unloaded after the conveyer was moved to a new path) were measured.

Both duration and amount of the research depended on specific conditions, lasting from several shifts up to several months. Moreover, in some longwalls the research was not carried out in full as the technique stipulates, but selectively which depends on organizational problems. However, identity of mining and geological conditions made it possible to use them together with those carried out in full to prove actual parameters for complete and selective seam mining.

3.5. Underground Research Results

3.5.1 Values of Enclosing Roof and Floor Superimposition

Research concerning values of enclosing roof and floor superimposition and nature of powered support-adjacent strata interaction was carried out in three longwalls of Western Donbass mines, and two longwalls of Lvov-Volyn coal field. The research was implemented in terms of complete and selective seam mining.

Table 3.3 and Fig. 3.10 demonstrate averaged values of enclosing roof and floor superimposition in one complete mining cycle at various distances from a working face.

Total superimposition of enclosing roof and floor at the boundary of operational space and mined-out space in terms of various mining techniques and mechanical means were: 149…261 mm in the process of complete mining in longwalls equipped with a support of KD80 system; 77…164 mm if MK97 support; and 175…201 mm if M88 support.

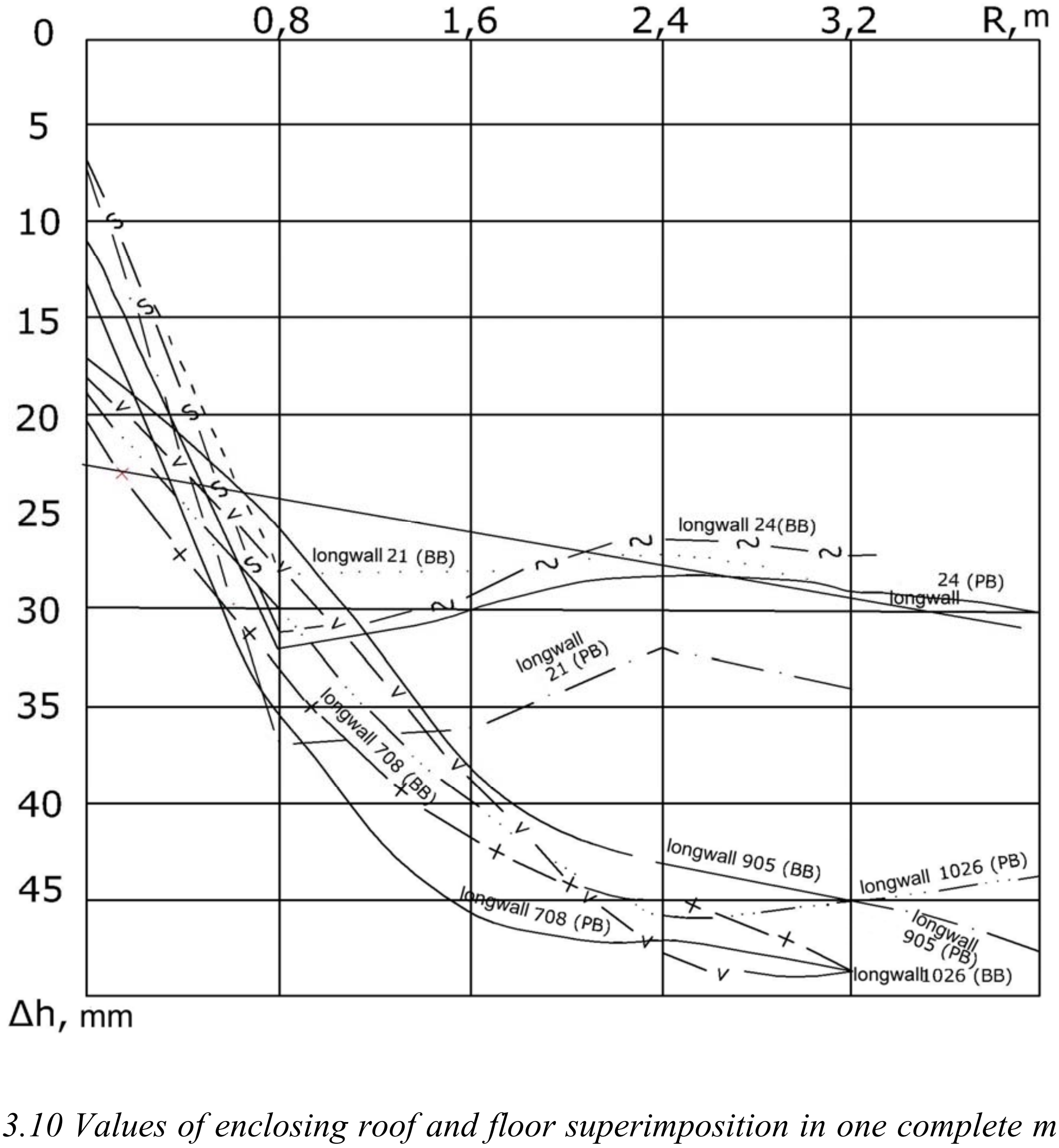


Fig. 3.10 Values of enclosing roof and floor superimposition in one complete mining cycle at various distances from a working face

If selective mining, then the values were accordingly: 195…261 mm, 103…211 mm, and 189…214 mm. The greatest superimposition values were observed in 905 and 1026 longwalls equipped with KD80 system; in the former case separate mining was applied, and in the latter case both separate and complete mining were applied. That can be explained by the fact that the technique can not allow a support to follow coal-extraction operations.

Both support and conveyor were moved after rock bench has been mined in the back engine. From coal mining to rock mining only slide roof support bars of KD80 system maintained the stope roof. In this case, face space of the longwall experience 0.8 m increase to be web width of end organ of a cutter-loader. Moreover, as a rule, period of one strip mining increased. Hence, increase in cumulative approximation values can be explained by prolongation of one and the same roof zone location the longwall face space.

It should be noted that in 708 longwall values of enclosed roof and floor approximation decreased in the neighbourhood of the stope. It can be explained by the fact that 0.3…0.5 m top coal was left in the process of separate mining. As a result, no “cut” of soft roof rocks along a seam edge, being typical for Western Donbass, took place. As in the first example, 24 longwall experienced increase in approximation values only owing to increase in face space. Rock bench, left in a seam roof, blocks on-time relocation of support section with hold-down consoles. Table demonstrates two approximation values at 0.8 m from a stope. The matter is that in this context both values of rock bench settling (numerator) and roof itself (denominator) were measured. Values of rock bench approximation are some higher than respective values for immediate roof of a seam to be explained by formation of horizontal joints within the bench. However, they could not affect its stability seriously. In the central part of the longwall, rock bench was preserved during 2 to 6 hours. Its breakage on end organs of a cutter-loader was observed only in areas of mining and geological displacements; mainly, they took place within end sections of longwalls when feeding velocity of the cutter-loader was 1.2…2.0 m per minute. When velocity increased up to 2.8…3.5 m per minute, the bench failed at the distance of 1…3 m from end organ of the cutter-loader in 0.05…0.10 m layers or completely.

Table 3.3 Results of enclosing roof and floor convergency measuring

Mine, seam, longwall, powering type	Technique	Value	Values of enclosing roof and floor approximation. Distance from sighting point to a stope, m						Total approximation, mm
			0	0.8	1.6	2.4	3.2	4.0	
1	2	3	4	5	6	7	8	9	10
“Zapadno-Donbasskaia” mine; No.905 longwall; KD80, 1K101; floor rock undercut	complete mining	minimum	14	25	32	41	37	–	149
		average	17	26	39	43	45	–	170
		maximum	20	28	44	49	50	–	191
	selective mining	minimum	16	28	33	44	42	46	209
		average	19	30	40	46	45	48	228
		макс.	24	32	48	52	51	54	261
“Dneprovskaia” mine; $C_{10}^{в}$ seam; No. 1024 longwall; KD80, 1K101; floor rock undercut	complete mining with cleaning	minimum	14	25	32	46	42	44	203
		average	18	28	39	48	49	48	230
		maximum	22	30	46	50	52	51	251
	selective mining	minimum	15	26	32	44	41	37	195
		average	19	30	40	46	45	44	224
		maximum	22	32	48	49	50	48	249
“Blagodatnaia” mine; C_7 seam; No. 708 longwall; KM88, 1K101; roof rock undercut	complete mining	minimum	19	31	40	41	44	–	175
		average	20	33	42	45	49	–	189
		maximum	21	36	44	48	52	–	201
	selective mining	minimum	11	33	44	45	46	–	179
		average	13	35	46	47	49	–	190
		maximum	14	39	49	50	52	–	204

1	2	3	4	5	6	7	8	9	10
Lvov-Volyn coal field; No. 5 “Velikomostovskaia”mine; n_7^B seam; No. 21 longwall; KMK97, 1K101; floor rock undercut	complete mining	minimum	4	17	20	16	20	–	77
		average	7	28	28	27	29	–	119
		maximum	9	43	39	36	77	–	164
	selective mining	minimum	5	24	26	23	25	–	103
		average	7	37	36	32	34	–	146
		maximum	10	51	47	47	43	–	192
No.5 “Novovolynskaia”mine; n_7 seam; No. 24 longwall; KMK97, 1K101; roof rock undercut	complete mining	minimum	4	18	19	16	21	–	78
		average	6	31	29	26	27	–	119
		maximum	8	40	38	35	36	–	157
	selective mining	minimum	9	27/24	25	21	25	27	199
		average	11	38/22	30	28	29	30	160
		maximum	13	52/44	43	38	40	39	211

While transferring from complete mining to selective one, increase in a longwall face space often took place; that depended on unfixed area left in the neighbourhood of the stope. However, as it is known [21], maximum increase in enclosing roof and floor approximation value is in this very area. As research, carried out in longwalls of mines in Western Donbass shows, the use of extendable consoles can not take effect. As a result, increase in roof rock outburst is possible [21]. Moreover, the most inrush-dangerous are those which can not involve the roof pickup immediately after a cutter-loader pass; also, techniques using a bench with unstable rocks leaving in the roof are among them. To decrease possibility of roof rock outburst in the process of selective mining, it is required to apply operation schedules providing relocation of a support section immediately after a cutter-loader pass.

Interaction between a support and enclosing roof and floor was evaluated visually; besides, their contacting types were sketched and rock cushion measuring took place. Rock cushion on upper roof support canopy with 40 to 80 mm thickness was available practically in each case; 30 to 80 mm height dust coal is available under lower roof support canopy of KD80 and KM88 systems. Effect of mining technique of the thickness of rock cushion and dust coal can not be identified.

As the data demonstrate, transfer from complete mining to separate one needs 5 to 8% increase thickness to be mined to provide required height of operating space depending upon applied technique of separate mining. That can be explained by certain increase in available approximation values while transferring to separate mining.

Following conclusions can be drawn while comparing results of interaction between enclosing roof and floor, and powered supports in the process of complete and selective seam mining.

Available powered supports are sufficient for complete and selective mining. In the process of selective seam mining which involves two passes of a cutter-loader, mined seam thickness should be increased by 5 to 8% to compare with complete mining. That depends on providing required height of face space.

In the context of soft floor rock preliminary mining in terms of Western Donbass, certain decrease in enclosing roof and floor approximation in the neighbourhood of a stope is possible owing to the roof cut liquidation. It is required to provide roof rock supporting after a cutter-loader pass in the process of transition from complete mining to selective one. Application of supports, working as scheduled, is expedient.

3.5.2 Performance Factors of Complete and Selective Seam Mining

Eight stopes with different techniques of coal and rock mining were involved in the research. Feeding velocity of cutter-loaders; time for one strip mining; power consumed by cutter-loaders; specific energy consumption; stability of a cutter-loader and its loading capability; ash-content of rock mass and coal; cutter-loader efficiency; and air dustiness in a longwall were determined in the research.

Feeding Velocity of a Cutter-Loader

Feeding velocity of cutter-loaders was measured in all eight longwalls under study both in terms of complete mining and selective one. Moreover, in a longwall No. 17 of a mine No. 5 "Novovolynskaia" the research was carried out in the process of complete mining and two techniques of separate mining: one-pass and two-pass. Table 3.4 shows the results. The data demonstrate that transition from complete mining to separate one (excluding one-pass technique) results in relative increase of feeding velocity of a cutter-loader (15…85% for coal, and 30…100% for rock); so, in a number of cases, increase in feeding velocity is notable. Longwalls in mines of Lvov-Volyn coal field demonstrated maximum absolute values of feeding velocity. There coal seam with 180…200 kN/m cuttability was mined first; then hard rock having two cropping out flat surfaces was mined. That also concerns a longwall No. 708 in "Blagodatnaia" mine where soft roof rocks were mined first followed by hard and tough coal. However, almost in all cases despite significant (sometimes up to 100%) increase in absolute values of feeding velocity – both in process of two-pass separate coal and rock mining – their values allocated to one running meter of a longwall, are practically always less by 50% than while one-pass complete or separate mining. It can be explained by the fact that one cycle needs two passages of a cutter-loader in a longwall; one pass to mine coal, and another one is mine rock.

The information can help to conclude that from the viewpoint of a longwall operation improving it is expedient to apply techniques of selective mining which provide one-pass coal and undercut rock mining. If two-pass selective seam mining, then technique for foremost and advance mining of softer undercut rocks is the most efficient for mines in Western Donbass, and for soft rocks in mines of Lvov-Volyn coal field.

Time for One Strip Mining

Time for one strip mining or time of technological cycle in stopes under study varied greatly depended not only on the technique applied (complete mining or selective one) but also on organizational (work discipline, shift change time, leisure time, time for car feeding/changeover etc.), mining and geological (enclosing roof and floor inrush, band pyrites etc.), and process-oriented (extraction and hauling equipment stoppage and breaking etc.) factors. However, it is possible to generalize specific features of separate mining. Thus, cycle period maximizes owing to two-pass of a cutter-loader if correspondent separate technique is applied. For this reason, extra time losses depend of equipment and technique applied; their value is inversely proportional to a cutter-loader feeding velocity (Table 3.5).

Table 3.4 Results of Cutter-Loader Feeding Velocity Measurements in Terms of Complete and Separate Seam Mining

Region, mine, longwall, seam	Cutter-loader feeding velocity, meters per minute											
	Compete mining			Separate mining								
				coal			rock			reduced		
	minimum	maximum	average	minimum	maximum	average	minimum	maximum	average	minimum	maximum	average
Western Donbass												
“Zapadno-Donbasskaia” mine												
longwall No. 905, $C_8^в$ seam	1.1	2.5	1.7	1.4	2.7	2.0	1.8	4.2	3.8	0.8	1.6	1.3
longwall No. 814, $C_8^н$ seam	1.0	2.4	1.6	1.5	2.6	2.1	2.0	4.4	3.7	0.9	1.6	1.3
“Blagodatnaia” mine												
longwall No. 708, пласт C_7 seam	1.0	2.3	1.6	1.6	3.4	2.9	1.9	3.6	3.0	0.9	1.8	1.5
“Dneprovskaia mine												
longwall No. 859, $C_8^в$ seam	1.1	2.2	1.5	1.4	2.6	2.1	2.1	4.4	3.6	0.9	1.6	1.3
Lvov-Volyn coal field												
“Velikomostovskaia”mine No. 5												
longwall No. 21, $n_7^в$ seam	1.6	2.9	2.3	2.4	4.1	3.4	2.8	4.1	3.1	1.3	2.1	1.6
longwall No. 24, $n_7^в$ seam	1.6	2.7	2.2	2.2	3.9	3.2	3.0	4.2	3.3	1.3	2.0	1.6
“Novovolynskaia” mine No. 5												
longwall No. 26, n_7 seam	0.6	3.6	2.0	1.9	4.5	3.7	1.4	6.0	4.0	0.8	2.6	1.9
longwall No. 17, n_7 seam	1.4	2.7	1.9	1.7	3.6	2.5	2.1	6.0	3.8	09 / 1.4	2.3 / 2.7	1.5 / 1.9

Table 3.5 Net Time for a Seam Mining

Region, mine, longwall, seam	Complete mining	Selective technique			Увеличение времени выемки, %
		coal	rock	total	
Western Donbass					
"Zapadno-Donbasskaia" mine					
longwall No. 905	100	85	45	130	30.0
longwall No. 814	113	86	49	135	19.5
"Blagodatnaia" mine					
longwall No. 708	109	60	58	118	8.3
"Dneprovskaia" mine					
longwall No. 859	120	86	50	136	13.3
Lvov-Volyn coal field					
"Velikomostovskaia" mine No. 5					
longwall No. 21	65	44	48	92	41.5
longwall No. 24	68	47	45	92	35.3
"Novovolynskaia"					
longwall No. 26	75	41	37	78	4.0
longwall No. 17: - two passes - one pass	 79 79	 60 -	 40 -	 100 79	 26.6 0

In absolute values (if random values of time losses are ignored) period to perform one cycle experienced 1.1 to 1.5 times increase when transition from two-pass complete mining to selective one took place; if one-pass separate mining took place, then one strip extraction period was practically identical with compete mining.

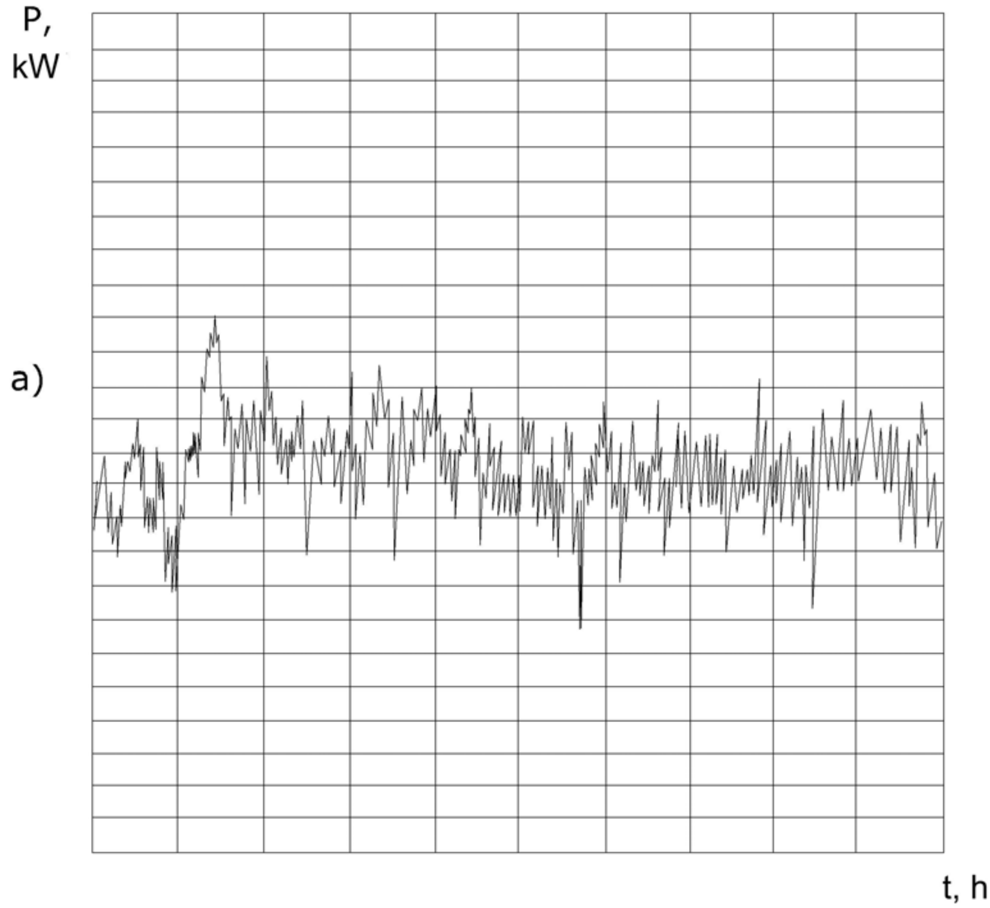
Consumed power

In longwalls Nos. 17 and 26 of "Novovolynskaia" mine No. 5 actual power consumed by 1K103cutter-loader engine and EFS in terms of one-pass and two-pass complete mining and two techniques of separate mining (longwall No. 17). Table 3.6 demonstrates the results; Fig. 3.11 shows model entries.

Table 3.6 Values of Power Consumed, kWh

Indices	Technique		
	Complete	Selective	
		Two-pass	One-pass
Longwall No.26			–
a) cutter-loader engine			–
- minimum	60.0	90.0…54.0 / 144 total	–
- maximum	187.0	142.0…158 / 300 total	–
- average	120.6	115.0…95.6 / 210.6 total	–
b) Extended Feeding System			
- minimum	–		–
- maximum	–		–
- average	–		–
c) total	120.6	210.0	–
Longwall No.17			–
a) cutter-loader engine			
- minimum	39.0	30.6…15.3 / 105.9 total	39.8
- maximum	207.3	140.1…24.7 / 164.8 total	207.3
- average	97.3	119.5…19.9 / 139.4 total	97.3
b) Extended Feeding System			
- minimum	9.0	2.0…1.0 / 3.0 total	9.0
- maximum	15.0	9.0…2.0 / 4.0 total	15.0
- average	12.0	6.2…1.5 / 7.7	12.0
total	109.3	146.7	109.3

Numerator shows factors on coal and rock, respectively; denominator shows total ones.



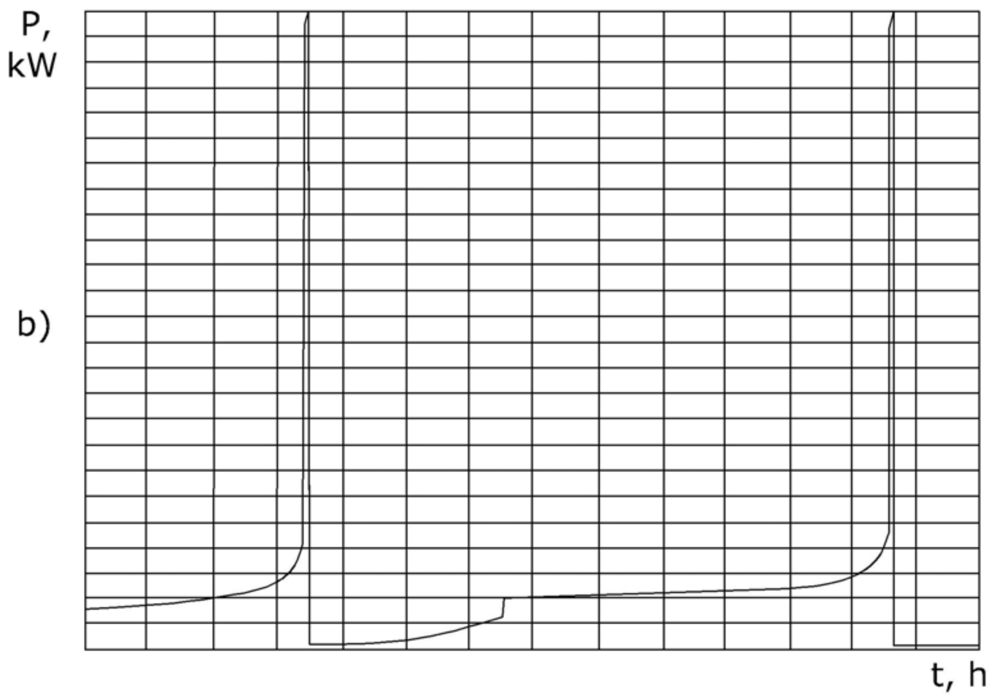


Fig. 3.11 model entries for power consumed:
a) by a cutter-loader engines; b) by EFS

One-pass technique in terms of complete and separate mining demonstrates identical values of consumed power as mining technique varies slightly. 25% increase in power consumption in the context of separate mining can be explained by 0.6 to 0.8 m increase in web width as well as by the necessity to distance cutter-loader twice within a longwall to perform one cycle.

Data from Table 3.6 show that utilization of cutter-loader engines while mining is 80 to 110%; the figure is 20 to 40% while mining undercut rock which has two flat outcropping surfaces. That is, 1K103 cutter-loader can perform powerful undercut of hard enclosing roof and floor (in this context $f = 4…5$) with high feeding velocities.

Fig. 3.12 demonstrates distribution of actual consumed power along the length of a longwall depending upon feeding velocity, thickness of coal seam and undercut, and cuttability of coal and rock in terms of one-pass selective seam mining.

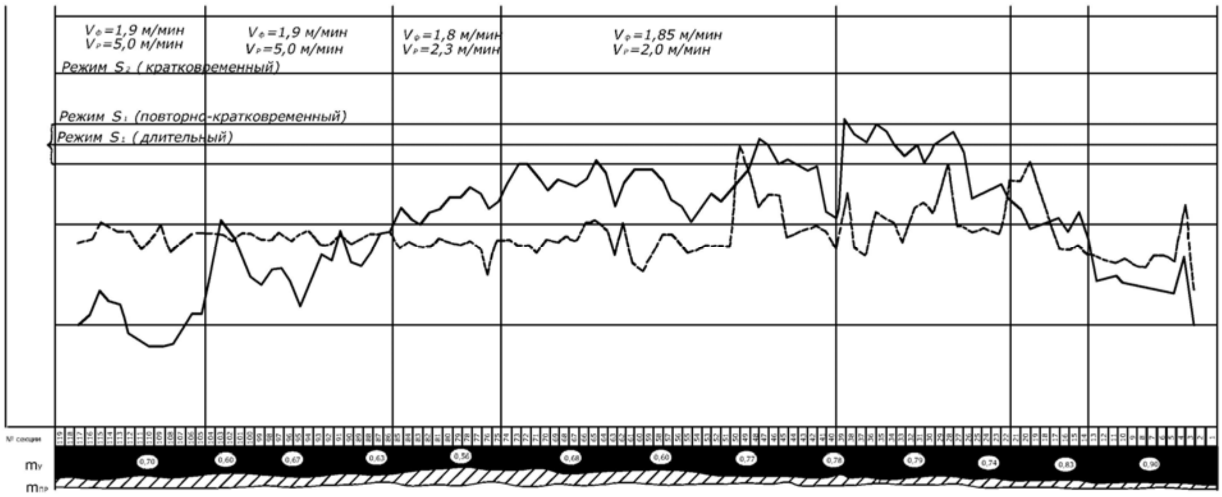


Fig. 3.12 Distribution of actual consumed power along the length of a longwall No. 17 depending upon feeding velocity, and coal seam and undercut floor rock thickness:——— consumed power; — — feeding velocity of a cutter-loader

Cutter-Loader Stability and Haulage Capacity. Stability of winning machine and haulage capacity of end organs of a cutter-loader and loading colters of conveyor were estimated visually. In longwalls of mines in Western Donbass, 1K101cutter-loaders demonstrated stable operation both in terms of complete and separate mining irrespective of undercut rock location and thickness. Low hardness of undercut rocks can explain the fact, and that makes it possible to beat in any direction. Another situation was observed in mines of Lvov-Volyn coal field. Mining a seam with floor rock undercut in terms of a longwall No. 20 in “VM” mine No. 5 demonstrated cases of 1K101cutter-loader crawl on undercut rock without complete its brake. The phenomenon took place when a cutter-loader moved towards screw rotation, or when feeding velocity was increased up to 4.0-4.2 meters per minute. When mining direction varied, or when feeding velocity decreased down to 2-2.5 meters per minute, 1K101 cutter-loader demonstrated operational stability.

In the context of roof rock undercut (longwall No. 24), hanging rock bench caved in (completely or partially) under gravity in certain parts of the longwall. Moreover, bedding joints were formed, and the cutter-loader operated steady within the whole velocity range in practice performing only rock cleaning and conveying.

One-pass complete and separate operation of 1K103 cutter-loader according to technique for n_7 seam mining in the context of “NV” mine No.5 (longwalls Nos. 17 and 26), unsteady operation of the cutter-loader took place if feeding velocity was 2.8 to 3.0 meters per minute and higher. Due to design features of the cutter-loader (i.e. rotational direction of end organs), tail screw crawled on a bench left in the seam floor without complete its break. The tail screw and rear body were jumped to the seam roof, and combs of unbroken rock stayed in the soil. When feeding velocity dropped to 2.5 meters per minute, the cutter-loader becomes stable; however, 0.15 to 0.20 m height and 0.10 to 0.15 m width rock edge continued in the soil near the stope (Fig. 3.13). While two-pass coal and rock mining, a cutter-loader demonstrated steady operation within the whole range of velocities, and rock edge in the neighbourhood was not observed. It can be explained by the fact that end organ performed compete break of rock bench when screw rotated bottom-up – from rock mass to outcropping plane.

It should be noted that each case when transition from complete mining to separate one (excluding one-pass mining technique) took part, increase in actual web width was registered. That means end organs hoisting capacity upgrading, and as a result – loading colters installed on a conveyor. Fig. 3.14 shows the most typical locations equipped with 1K101 cutter-loader in terms of various mining techniques application. As the research demonstrates when 1K101 cutter-loader operates without loading colter, it dumps 50 to 80% of coal (or rock) from a bench left in floor. If a cutter-loader moves towards end organs, their hoisting capacity is 1.5 to 2 times higher than when it moves invertedly. Only coal (or rock) cushion which thickness is 0.05 to 0.10 m (Fig. 3.1 b) remains on a bench; in the process of opposite direction move broken-down coal (or rock) are placed as it is shown in Fig. 3.14 a.

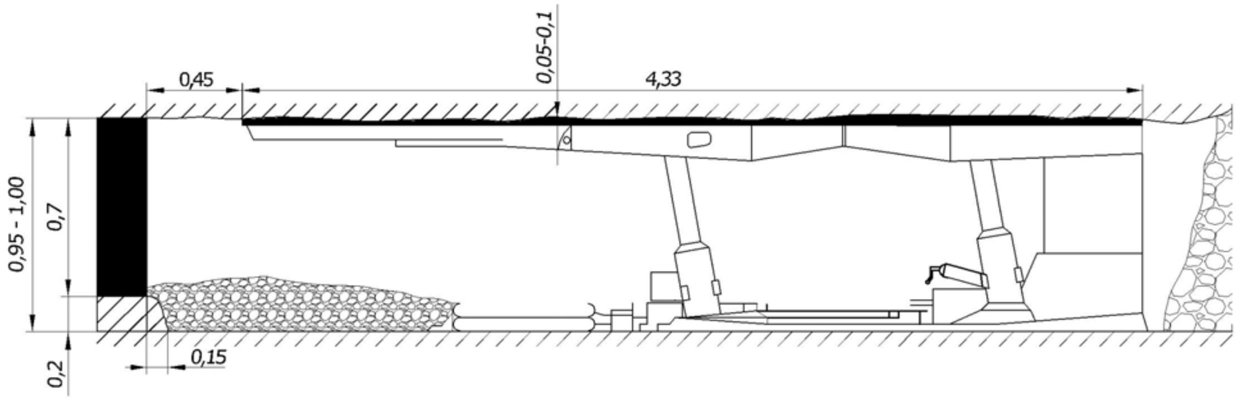


Fig. 3.13 A view of a longwall equipped with 1KM103 powered system after coal seam has been extracted, and undercut floor rocks have been cut

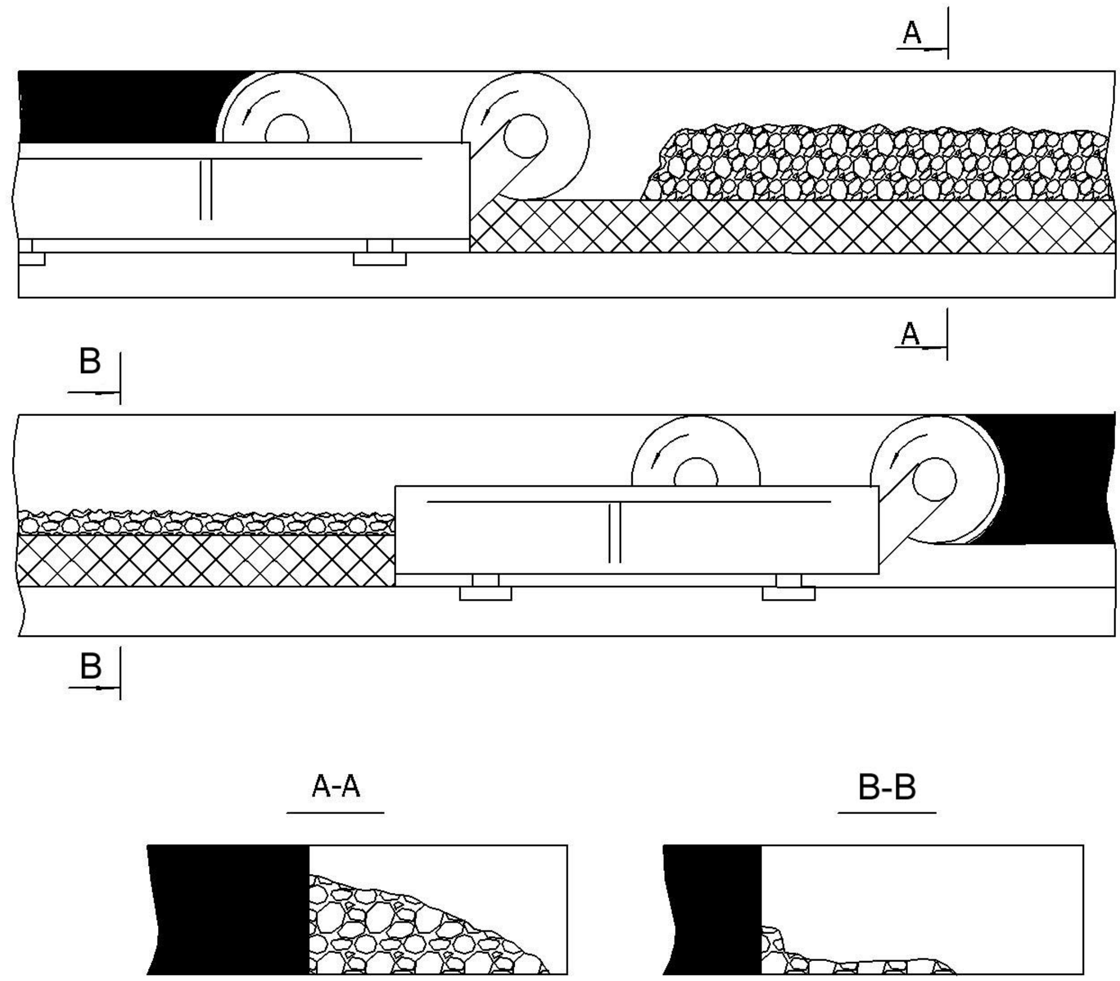


Fig. 3.14 Hoisting capabilities of 1K101 cutter-loader end organs while coal seam extracting:

a) Towards terminal switch;

b) Towards end organs

While coal mining with following roof rock breaking (longwall No. 124), 1K101 cutter-loader was equipped with loading colter which helped to provide complete coal and rock conveying.

In terms of 1K103cutter-loader one-pass operation, advance screw conveyed nearly the whole broken coal, and back one just cut rock without conveying it (Fig. 3.15). Broken rock left on a seam floor between a conveyor and a stope forming 0.35 to 0.45 m layer. When a conveyor resited, 50 to 70% of rock was conveyered with loading colters. A bed which average width was 0.52 m left unconveyered between a conveyor and a stope. Conveyor no-contact was 0.2 m, that is productive web in terms of one-pass complete and separate mining was 0.6 m instead 0.8 m. While transiting to two0pass separate mining, coal and rock breaking as well as their conveying were mainly performed by end organs of cutter-loader. Advance increment and productive web width of cutter-loader along with other separate mining techniques increased up to 0.8 m.

Ash-Content of Rock Mass and Coal. Determination of actual ash-content of mined rock mass in terms of complete mining as well as ash-content of coal in terms of separate mining was performed in four longwalls. Table 3.7 demonstrates results of the determinations.

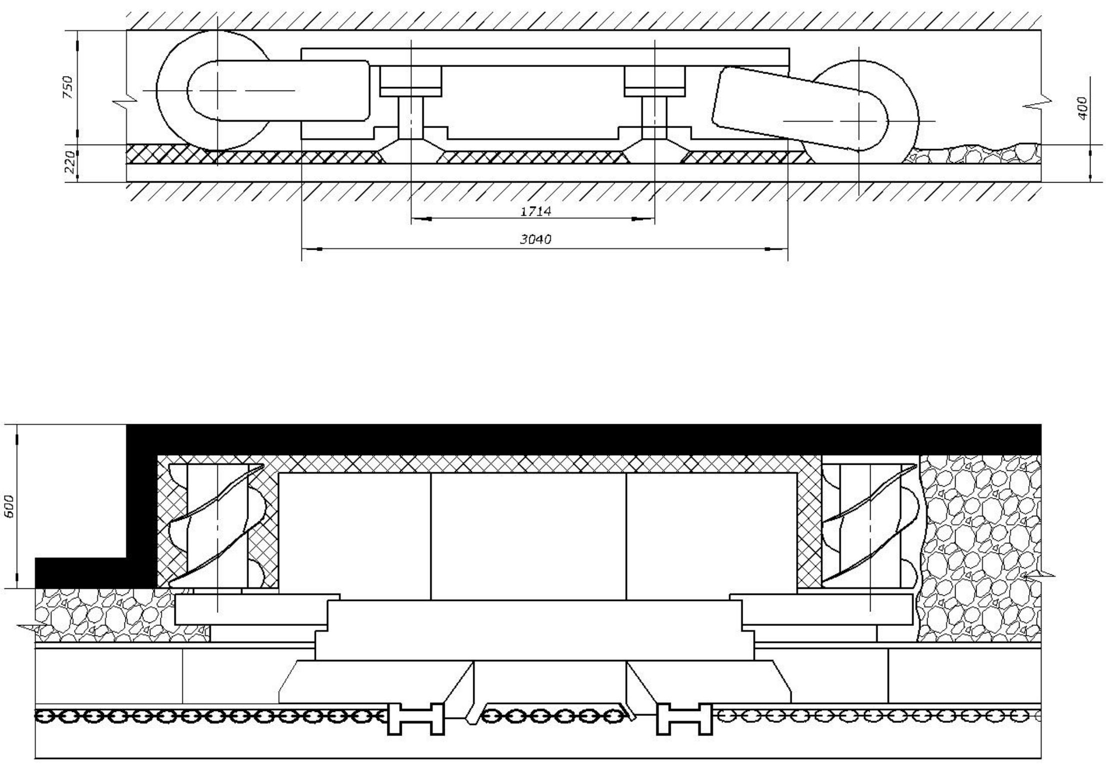


Fig. 3.15 Separate Seam and Undercut Floor Rocks by Means of 1K103 Cutter-Loader

Table 3.7 data explain that maximum dilution (28 to 44%) took place in longwalls of Western Donbass mines. For the most part it results from higher thickness of undercut rock geological ash-content. In those mines ash-content of coal is small (12 to 15%) in contrast to mines in Lvov-Volyn coal field (18 to 37%). To large extend, the factor defines new technique efficiency. While transiting for complete seam mining to selective one, operative ash-content of mined coal decreases by 12 to 38%; that is dilution drops form 28-44 to 6-16%.

Table 3.7. Operative Ash-Content Measurement Results

Mine, longwall, technique	Thickness, m			of rock mass	Ash-content, %			
	of a mined seam	of coal seam	of undercut rock		of coal		of rock	
					geological	operating	geological	operating
"Zapadno-Donbasskaia" mine								
longwall 905								
complete	1.17	0.78	0.39	56	12	–	93	–
separate	1.17	0.78	0.39	–	12	18	93	–
"Blagodatnaia" mine								
longwall 708								
complete	–	–	–	47.9	19.5	–	89	–
separate	–	–	–	–	19.5	35.6	89	–
"VM" mine No.5								
longwall 24								
complete	1.05	0.7	0.35	61	36.2	–	90	–
separate	1.05	0.7	0.35	–	36.2	41	90	–
"NV"mine No.5								
longwall 17								
complete	0.95	0.73	0.22	43.9	17.8	–	–	–
separate:								
- one-move	0.95	0.73	0.22	–	17.8	28.2	–	68.4
- two-move	0.96	0.73	0.23	–	17.8	23.6	–	73.3

Estimated values of mined coal and rock mass ash-content correspond to actual values with the specified degree of accuracy. Error is less than 8 to 10%.

Air Dustiness

Air dustiness was measured together with MRT staff in accordance with current GOST in two Western Donbass longwalls (Nos. 814 and 159), and one longwall in Lvov-Volyn coal field (No. 17). Table 3.8 demonstrates results.

Table 3.8 Air Dustiness Measurement Results

Mine, longwall	Sampling site	Airflow rate	Operations performed	Cutter-loader type	Dust control	The number of people in measurement location	Dust content, mg/m^3
“Dneprovskaia” mine, longwall 859	behind a cutter-loader	1.94	coal mining	1K101	sprinkling	5	221.0
	behind a cutter-loader	1.94	coal mining	1K101	sprinkling	2	176.0
	behind a cutter-loader	1.94	complete mining	1K101	sprinkling	5	247.0
“Zapadno-Donbasskaia” mine, longwall 814	behind a cutter-loader	2.3	coal mining	1K101	sprinkling	6	256.0
	behind a cutter-loader	2.3	coal mining	1K101	sprinkling	2	192.0
	behind a cutter-loader	2.3	complete mining	1K101	sprinkling	6	283.0
“NV”mine No.5, longwall 17	behind a cutter-loader	1.8	coal mining	1K101	sprinkling on advance screw	1	134.0
	behind a cutter-loader	1.8	coal mining	1K101		1	29.0
	behind a cutter-loader	1.8	complete mining	1K101		1	227.0

Table 3.8 explains that while transiting from complete mining to separate one (two-pass operation), air dustiness in longwall drops by 10 to 40%; when undercut rocks are mined, the figure is 30-80%. It should be noted that air dustiness values in longwall No. 17 involve 79 mg/m^3 background; values on longwalls Nos. 859 and 814 do not take into account.

Machine Time Coefficient.

Machine time coefficient values were determined for each longwall by means of H-358 device and according to the data from registry of cutter-loader operation shift time. Table 3.9 shows actual values in stopes under study.

Table 3.9 explains that in the process of two-pass selective seam mining, mean values of machine time coefficient are by 28.5% less than under the conditions of complete mining. It can be accounted for the fact that undercut rock extraction time is considered as routine break time. That is, in the context of separate mining only coal extraction time comes into account.

In the context of one-pass selective mining, machine time coefficient is more or less identical with complete mining.

Cuter-Loader Efficiency.

The research also involved moment and mean-shift cutter-loader efficiency under the conditions of various mining techniques. Table 3.10 shows Q_{max} and $Q_{см}$ values for stopes under study.

Table 3.10 explains that while transiting from complete mining to separate one, moment cutter-loader efficiency drops by 8-24% (16% in the mean). However, it should be noted that increase in efficiency in the context of complete mining results from undercut of denser rocks. The same situation takes place when mean-shift efficiency is compared. It would seem that substantial (40% in the mean) decrease of the parameter is a result of dilution avoiding. Table 3.11 demonstrates that illustrating mining volumes normalized to uniform ash-content in terms of complete and separate mining.

As it follows from Table 3.11, only a longwall No. 708 continues certain excess in mean-shift cutter-loader efficiency (15% in the context of complete mining); in three other longwalls it is lower to compare with separate mining (5 to 57%).

Table 3.9 Minimum – Maximum Value of Machine Time Coefficient
Average

Technique	Stope							
	No. 905	No.814	No. 708	No. 859	No.21	No. 24	No. 26	No. 17
Complete	0.26-0.36 0.33	0.28-0.37 0.34	0.24-0.34 0.31	0.28-0.38 0.36	0.24-0.32 0.28	0.26-0.33 0.30	0.27-0.34 0.31	0.28-0.33 0.30
Two-pass selective	0.17-0.26 0.24	0.19-0.27 0.24	0.18-0.30 0.25	0.20-0.26 0.23	0.18-0.24 0.20	0.17-0.29 0.21	0.19-0.25 0.22	0.19-0.24 0.22
One-pass selective	-	-	-	-	-	-	-	0,28-0,33 0.30

Table 3.10 Value of Moment and Mean-Shift Cutter-Loader Efficiency

Технология	Stope															
	No. 905		No. 814		No. 708		No. 859		No. 21		No. 24		No. 26		No. 17	
	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т	Q_{min}, т	$Q_{см}$, т
Complete	2.5	297	2.3	281	2.9	323	2.0	252	2.8	262	2.6	281	1.8	195	1.9	205
Two-pass selective	1.9	164	2.0	173	2.3	207	1.8	149	2.5	171	2.3	174	1.4	110	1.75	140
One-pass selective	-	-	-	-	-	-	-	-	-	-	-	-	1.4	150	1.5	160

Table 3.11 Values of Uniform Mean-Shift Cutter-Loader Efficiency

Technique	Stope			
	No. 905	No. 708	No. 24	No. 17
Complete	71	244	168	122/141
Two-pass selective	164	207	174	140
One-pass selective	–	–	–	160

Specific Energy Consumption. Specific energy consumption experienced changes in two longwalls equipped with 1KM103powered systems. Table 3.12 shows measurement results which exclude and include ash-content of coal and rock mass mined. However, in terms of complete mining rock mass ash-content is normalized to ash-content of coal extracted with the help of separate mining technique.

Table 3.12 Specific Energy Consumption per a Ton of Output

Factors	Complete technique		Selective technique	
	Ash-content excluded	Ash-content included	Two-pass	One-pass
Longwall No. 26				
minimum	0.49	–	0.62	–
maximum	2.34	–	2.12	–
average	1.12	–	0.92	–
Longwall No. 17				
minimum	0.44	0.74	0.76	0.59
maximum	2.67	4.48	1.91	2.98
average	1.02	1.71/1.48	1.01	1.2

Table 3.12 explains that in the process of selective mining in a longwall No. 26 specific energy consumption per a ton of output is lower by 16% to compare with complete technique. Significant (25%) increase in mined coal in terms of separate technique depends on increase in actual web width (from 0.6m to 0.8 m). For a longwall No. 17 one strip extraction increase turned out to be sufficient to cover increase in energy consumption. In the context of two-pass separate mining specific energy consumption was almost equal to that in the context of complete mining. However, it is drastically lower if ash-content is taken into consideration (40%). One-pass selective mining provided 20% drop in specific energy consumption. That is from the viewpoint of energy consumption reduction a two-pass technique of selective mining is the most expedient.

3.6. Analysis and Estimation of the Research

Actual parameters of complete and selective mining obtained as a result of mine research are analyzed in comparison with theoretical parameters being estimated adequately.

As theoretical analysis shows (Chapter 2), feeding velocity of cutter-loader depend on a coal seam and undercut rock thickness and cuttability, cutter-loader type, and mining technique. Actual values of a cutter-loader feeding velocity being a result of mine research confirm the theoretical background. Table 3.13 demonstrates calculated values and actual ones of 1K103 cutter-loader feeding velocity V for n_7 seam complete and selective mining in terms of "Novovolynskaia" mine No. 5

Table 3.13 Feeding Velocity Actual and Calculation Values

Longwall; technique	Feeding velocity		Deviation value	
	actual	calculated	absolute	%
Longwall No.26; n_7 seam				
Complete mining	$\frac{0.6\text{-}3.6}{2.0}$	$\frac{0.64\text{-}3.91}{2.12}$	0.12	6.0
Separate mining	$\frac{0.8\text{-}2.6}{1.9}$	$\frac{0.77\text{-}2.68}{1.94}$	0.04	2.1
Including that for coal	$\frac{1.9\text{-}4.5}{3.7}$	$\frac{1.82\text{-}4.60}{3.63}$	0.07	1.9
Including that for rock	$\frac{1.4\text{-}6.0}{4.0}$	$\frac{1.46\text{-}6.0}{4.16}$	0.16	4.0
Longwall No. 17; n_7 seam				
Complete mining	$\frac{1.4\text{-}2.7}{1.9}$	$\frac{1.36\text{-}2.64}{1.94}$	0.04	2.1
One-pass separate mining	$\frac{1.4\text{-}2.7}{1.9}$	$\frac{1.36\text{-}2.64}{1.94}$	0.04	2.1
Two-pass separate mining	$\frac{0.9\text{-}2.3}{1.5}$	$\frac{0.97\text{-}2.42}{1.62}$	0.12	8.0
Including that for coal	$\frac{1.7\text{-}2.6}{2.5}$	$\frac{1.67\text{-}2.72}{2.55}$	0.05	2.0
Including that for rock	$\frac{2.1\text{-}6.0}{2.8}$	$\frac{2.0\text{-}6.0}{3.69}$	0.11	2.9

Table 3.13 demonstrates that deviation of feeding velocity calculated value from its actual values is not less than 8.0% varying from 1.9 to 8.0%. 98.1 to 92% reproducibility favours legitimacy of expression (2.18) use while calculating cutter-loader feeding velocity. Table 3.14 shows both actual and calculated values for machine time coefficient.

Table 3.14 Actual and Calculation Values

Longwall; technique	Machine time coefficient		Deviation value	
	actual	calculated	absolute	%
Longwall No. 905; C_8^B seam				
Complete mining	$\frac{0.26-0.36}{0.33}$	$\frac{0.248-0.370}{0.345}$	0.015	4.5
Two-pass separate mining	$\frac{0.17-0.26}{0.24}$	$\frac{0.170-0.262}{0.236}$	0.004	1.7
Longwall No.814; C_8^H seam				
complete mining	$\frac{0.28-0.37}{0.34}$	$\frac{0.276-0.384}{0.352}$	0.012	3.5
Two-pass separate mining	$\frac{0.19-0.27}{0.24}$	$\frac{0.180-0.263}{0.226}$	0.014	5.8
Longwall No.708; C_7 seam				
Complete mining	$\frac{0.25-0.34}{0.31}$	$\frac{0.261-0.362}{0.336}$	0.025	8.1
Two-pass separate mining	$\frac{0.18-0.29}{0.25}$	$\frac{0.174-0.276}{0.231}$	0.019	7.6
Longwall No.859; C_8^B seam				
Complete mining	$\frac{0.28-0.38}{0.35}$	$\frac{0.276-0.372}{0.354}$	0.004	2.1
Two-pass separate mining	$\frac{0.30-0.26}{0.23}$	$\frac{0.196-0.252}{0.229}$	0.006	2.6
Longwall No.21; n_7^B seam				
Complete mining	$\frac{0.24-0.32}{0.28}$	$\frac{0.256-0.342}{0.304}$	0.024	8.6
Two-pass separate mining	$\frac{0.18-0.24}{0.20}$	$\frac{0.174-0.258}{0.212}$	0.012	6.0
Longwall No.24; n_7^B seam				
Complete mining	$\frac{0.26-0.33}{0.30}$	$\frac{0.246-0.340}{0.312}$	0.012	4.0
Two-pass separate mining	$\frac{0.17-0.23}{0.21}$	$\frac{0.177-0.241}{0.219}$	0.009	4.3
Longwall No.26; n_7^B seam				
complete	$\frac{0.27-0.34}{0.31}$	$\frac{0.280-0.322}{0.291}$	0.019	6.1
Two-pass separate mining	$\frac{0.19-0.25}{0.22}$	$\frac{0.176-0.251}{0.212}$	0.008	3.6
Longwall No.17; $n_7$seam				
compete	$\frac{0.28-0.33}{0.3}$	$\frac{0.292-0.338}{0.306}$	0.006	2.0
Two-pass separate mining	$\frac{0.19-0.24}{0.22}$	$\frac{0.183-0.266}{0.242}$	0.022	10.0
One-pass separate mining	$\frac{0.28-0.33}{0.3}$	$\frac{0.292-0.338}{0.306}$	0.006	2.0

It follows from Table 3.14 that deviation values of calculated machine time coefficient from its actual values vary within 1.1-10.0%. Such convergence makes it possible to apply (2.22, 2.24) expressions to calculate $\kappa_м$ in terms of compete and separate seam mining.

Table 3.15 demonstrates rock mass (mined coal) ash-content values as well as actual ones calculated according to (2.30, 2.31) expressions.

Table 3.15 explains that calculated values of rock mass and mined coal ash-content correspond to actual ones with the specified degree of accuracy. Deviation maximums are no more than 10.0%.

Table 3.16 shows cutter-loader efficiency values.

Table 3.15 Actual Calculation Values of Ash-content

Longwall No; technique	Ash-content, %		Deviation value	
	actual	calculated	absolute	%
Longwall No. 905; $C_8^в$ seam				
Complete mining	56.0	56.94	0.94	1.1
Two-pass separate mining	24.2	26.62	2.42	10.0
Longwall No.708; $C_8^н$ seam				
Complete mining	47.9	48.48	0.54	1.1
Two-pass separate mining	35.6	34.27	1.47	4.1
Longwall No.24; $n_7^в$ seam				
Complete mining	61.0	59.61	1.41	2.3
Two-pass separate mining	41.0	42.88	1.28	3.1
Longwall No.17; $n_7$seam				
Complete mining	43.9	46.33	2.43	5.5
Two-pass separate mining	23.6	25.26	1.66	7.0
One-pass separate mining	28.2	30.46	2.26	8.0

Table 3.16 Efficiency Actual and Calculation Values

Longwall No; technique	Cutter-loader efficiency		Deviation value	
	actual	calculated	absolute	%
Longwall No.26; n_7 seam				
Complete mining	195	199.49	4.49	2.3
Two-pass separate mining	110	116.84	6.84	6.2
Longwall No. 17; n_7 seam				
Complete mining	$\frac{206^*}{1222\text{-}241}$	$\frac{194.56}{112.57\text{-}136.7}$	$\frac{10.44}{9.42\text{-}2.3}$	$\frac{7.4}{7.7\text{-}3.1}$
Two-pass separate mining	140	127.03	12.97	9.3
One-pass separate mining	160	159.98	0.02	0.01

* Cutter-loader efficiency for rock mass is in numerator; cuter-loader efficiency in ash-content for two-pass and one-pass separate mining is in denominator.

Table 3.16 explains that convergence of calculated and actual values of cutter-loader efficiency is quite satisfactory. Hence, (2.26) expression as well as its values adequately indicate actual value of efficiency.

Thus, estimating the results of mine research concerning actual parameters in terms of seam complete and separate mining, one may conclude that their basic theory is true, and procedural weight is proved. Reproducibility is within ±10% to be reasonable to calculate mining technique parameters.

Conclusions

1. Mine research as well as its results confirms functionality of thin and very thin seams selective mining with the help of available winning equipment. Such a technique allows widening area of powered systems application without making any changes in their design.

2. From the viewpoint of mining and geological conditions, techniques with floor rock undercut are the most expedient for mines in Western Donbass; the matter is that when weak roofs are undercut, then their continuity can be troubled.

3. To reduce a risk of roof rock inrush in terms of separate seam mining, it is required to apply techniques providing support units resiting right after cutter-loader pass.

4. While advance soft floor rock mining in terms of Western Donbass, certain decrease (30-40%) in enclosing roof and floor convergence in the neighbourhood of a stope is possible owing to roof cut liquidation.

5. While two-pass seam selective mining, thickness of extracted seam should experience 5-8% increase to compare with complete mining to provide available height of face space. Techniques providing one-pass extraction should be applied to liquidate the phenomenon.

6. From the viewpoint of making operations more intensive, it is the most reasonable idea to apply one-pass coal mining and enclosing roof and floor undercut techniques.

7. It is the most efficient to apply prior and advance mining of weaker undercut rocks in the process of one-pass and two-pass separate techniques in terms of Western Donbass and relatively soft coal mining in terms of Lvov-Volyn coal field.

8. In the process of separate two-pass mining, 1K103 cutter-loader can undercut hard thick (up to 0.35 m) enclosing roof and floor (f =4-5) with maximum feeding velocities.

9. When one-pass complete and separate mining techniques are applied for a seam with hard floor rock undercut, operation of 1K103 cutter-loader is unsteady (if V= 2.8 – 3.0 meters per minute).

10. Selective mining technique makes it possible to decrease drastically (down to 40%) ash-content of coal mined in longwalls with enclosing roof and floor undercut.

11. With the specified degree of accuracy analytically obtained dependences describe actual values of complete and separate mining parameters for coal with enclosing roof and floor undercut. Reproducibility is within ±10% to be rationally for performing calculations.

CHAPTER 4. SUBSTANTIATION OF RATIONAL AREA AND RANGE OF SELECTIVE SEAM MINING TECHNIQUE

4.1 New Technique Application Restrictions

Application of selective mining technique for thin coal seams by available powered systems is limited by:

- Thickness of mined coal seam and enclosing roof and floor undercut.
- Stability of enclosing roof and floor, and hardness of undercut.
- Economic expediency.

Rock bottom of mined coal seam thickness, or in the case of enclosing roof and floor advance undercut, the undercut thickness is limited by minimum dimensions (diameter) of a cutter-loader end organs; for the above 1K103 cutter-loader it is m_{min}=560 mm. Maximum depth of enclosing roof and floor undercut, or in the case of following coal breakage, maximum mined thickness of coal seam is limited by a cutter-loader end organs extension being for screws with 560; 630; 710 and 800 mm diameters respectively; if advance mining coal or rock top patch takes place, then diameters are 555, 510, 470 and 425 mm (Fig. 4.1).

Application range of KM103 system is limited by thin seams where roof rocks are either stable or of mean stable. The new technique allows extracting seams with false roof; in particular, when its thickness is less than undercut maximum (Fig. 4.1).

As results of underground investigations and experiments show, 1K103 cutter-loader can mine seams with enclosing roof and floor undercut which hardness is f=4-5. In this context, one-pass complete and separate mining demonstrated following negative things: partial breakage of undercut floor rocks by back end organ of a cutter-loader; conveyor no-contact and, as a result, almost 25% decrease in productive web width; unsteady operation of a cutter-loader at increased feeding velocities due to back screw scrawl on a floor rock bench; and increase in pick and electric power consumption.

Selective two-pass seam mining with hard floor rocks undercut (f=4-5) when one pass is for coal and another one is for rock, makes it possible to close the gaps. In the context of such a technique, floor rock breakage is performed when screw performs bottom-up rotations towards outcropping (Fig. 4.2) rather than adown toward rock mass (Fig. 4.2) as it is done in the process of one-pass complete and separate mining. If hard roof rocks are undercut or hard and tough coal is mined (Western Donbass mines), then breakage of coal or rock batch left in a roof by means of back end organ of a cutter-loader is performed adown towards outcropping (Fig. 4.2); that makes it possible to perform more efficiently one-pass separate mining.

a) With floor rock undercut

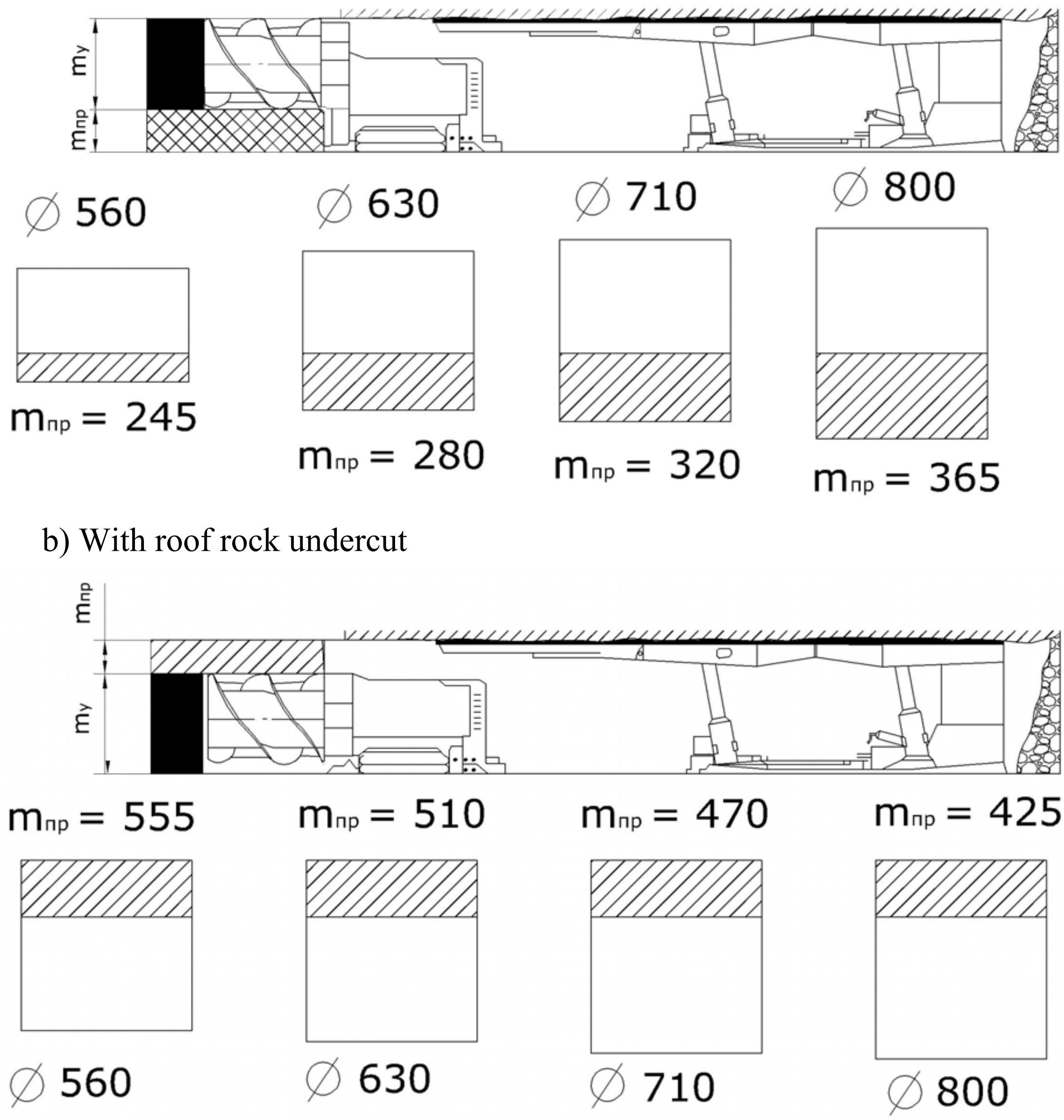
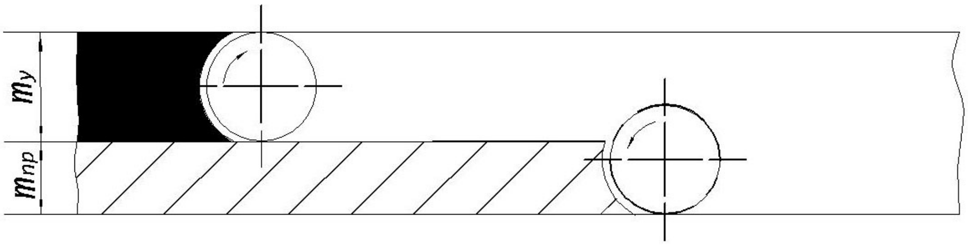


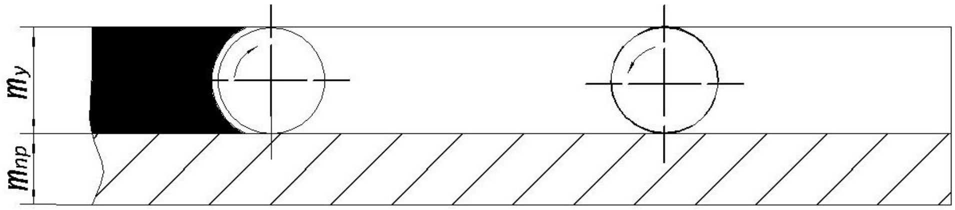
Fig. 4.1 Limits of a seam thickness mined by1K103 cutter-loader:

Economic expediency of compete or selective mining techniques as well as their areas and volumes for specific underground environment should be each time substantiated and determined basing upon economic and mathematic simulation. First, price depends on production quality directly influencing on a coal producer profit.

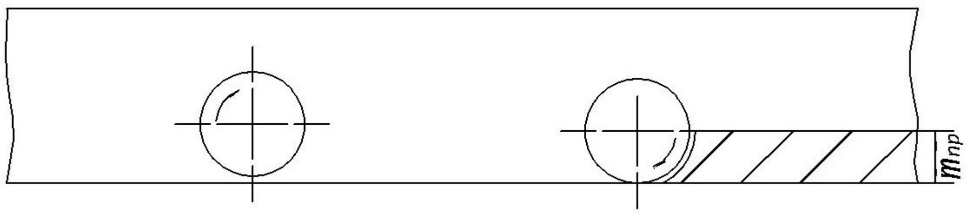
a) one-pass compete and selective mining a seam and undercut floor rocks



b) two-pass separate mining a seam and undercut floor rocks:
- for coal



- for rock



c) one-pass complete and separate mining a seam and undercut roof rocks

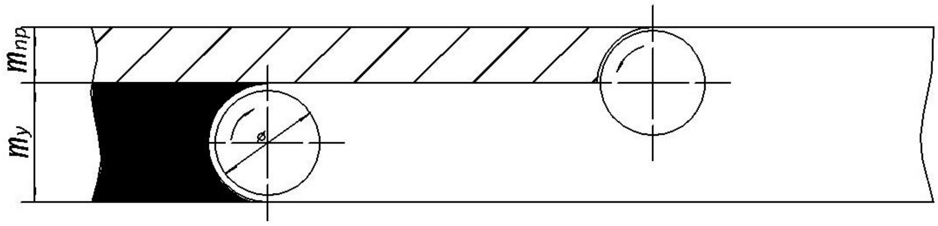


Fig. 4.2 Breakage techniques for coal and enclosing roof and floor by means of 1K103cutter-loader end organs

4.2. Economic and Mathematic Simulation for Rational Areas and Volumes to Apply Complete and Selective Seam Mining

4.2.1 The Problem Formulation and the Simulation Procedure

The work tries a shot to determine by theory economically expedient area as well as selective technique application volumes under specific mine condition using economic and mathematic simulation. Generally, the problem is engineering and economical comparison of calculated alternatives basing upon accepted optimization criterion.

Economic and mathematic simulation model consists of the two modules: determination of economic area to apply selective seam mining technology and estimation of rational utilization capacity to apply the new technology within a mine. Together with the basic task performance, module one involves selection of longwalls which parameters meet the requirements of transition to selective mining technique separating longwalls dissatisfying the demands. Both selection and simulation of small amount of alternatives to determine economic expediency of new technique application allows simplifying significantly the model; helps to make it more controllable; reduces time consumption to calculate and analyze calculation data. Moreover, the selection method excludes the risk for optimum alternative not to be involved in the calculations. The model is developed to mine thin and very thin coal seams in terms of mines in Western Donbass and Lvov-Volyn coal field.

Annual profit is selected as optimization criterion taking into full consideration changes in output quantity and quality to estimate final results of the enterprise activity. The problem is solved not only for a mine but also for such systems as "mine-washhouse", "mine-consumer", and "washhouse-consumer" using a criterion of maximized total profit.

Thickness of undercut enclosing roof and floor is basic varied parameter; it varies from minimum to maximum in terms of specific powered system operations under specific mining and geological conditions. As the paper should obtain comparative economic characteristics to estimate rated alternatives rather than to determine specific profit margin for them, only those cost items varying in terms of mining technique (either complete or selective); as for the price – it is wholesale nominal price. To reduce to ash-content norm (scheduled ash-content for mine), 2% cut for each per cent of scheduled ash-content norm excess is used.

Economic and mathematical model makes it possible to identify boundaries of economically attractive area of selective mining technique application as well as its rational utilization capacity for specific mine.

4.2.2. Economic and Mathematical Model

Developed expression of economic and mathematical model goal function is:

$$I = \left(\sum_{i=n}^{0} L_{B_i} n_i + \sum_{j=0}^{n} L_{cj} n_j \right)(P_w - C) + \sum_{j=0}^{n} Q_{nj} n_j C_{ymj} \rightarrow max, \tag{4.1}$$

where i is a symbol of longwalls applying complete seam mining; j is a symbol of longwalls applying separate seam mining; L_{B_i} is load on i^{th} longwall, tons per day; L_{cj} is load on j^{th} longwall, tons per day; n_i and n_j are the number of working days per year for i^{th} and j^{th} longwall respectively; P_w is wholesale price for a ton of end product taking into account its grade, UAH per ton; C is expenditures connected with extraction, transportation, and preparation of a ton of coal (rock mass), UAH per ton; C_{ym} is expenditures connected with recycling of a ton of coal, UAH per ton; Q_{nj} is rock volume extracted from j^{th} longwall, tons per day.

1. Borders of undercut thickness variation. Undercut thickness is varied with specified step: from minimum ($m_{np_{\min}}$) to maximum ($m_{np_{\max}}$) depending upon assumed type of powered support. Value $m_{np_{\min}}$ is determined as follows:

$$m_{np_{min}} = max\left\{\frac{(h^k + h_3)k_1;(h^{np}_{min} + h_{nep})k_2;(h^{\kappa p}_{min} + h_p)k_3}{0{,}9 - 0{,}01k_{\kappa p}R(1 - \frac{T_{cм}rVk_м}{2Rl}}\right\} - m_y \ , \qquad (4.2)$$

where h^k is cutter-loader height, m; h_3 is distance between cutter-loader frame and support ceiling, m; h^{np}_{min} is minimum channel height in the support units, m; h_{nep} is cumulative thickness of ceiling and support basis, m; $h^{\kappa p}_{min}$ is minimum constructive depth of the support (assembled), m; h_p is reserve of hydraulic extension to take weight of support walls, m; $k_{\kappa p}$ is a coefficient taking into account support resistance; R is distance across the width of face space from a face to location of measurement, m; $T_{cм}$ is shift period, min; r is web width of a cutter-loader end organ, m ; V is a cutter-loader feeding velocity, meters per minute; $k_м$ is machine time coefficient; l is length of a longwall, m; k_1, k_2, k_3 are coefficients taking into account parameters involved; and m_y is coal seam thickness, m.

Value $m_{np_{max}}$ is determined by:

$$m_{np_{max}} = \frac{h^{\kappa p}_{max}}{1 - 0.01k_{\kappa p}R_n(1 - \frac{T_{cм}rVk_м}{2R_nl})} - m_y \ , \qquad (4.3)$$

where $h^{\kappa p}_{max}$ is maximum structural depth of a support, m; and R_n is distance across the width of face space from a face to front columns, m.

2. Output per longwall. Computation of daily average output per longwall (D) is required if the value is not determined by initial data

$$D = min(D_{п.к.}; D_{г.ф.}) \qquad (4.4)$$

where $D_{п.к.}$ is output per longwall depending upon a cutter-loader feeding velocity, t/day;

in terms of complete mining:

$$D_{п.к.} = (m_y \gamma_y + m_{пр} \gamma_n) rVk_м T_{см} n_{см} \, , \tag{4.5}$$

in terms of separate mining:

$$D_{п.к.} = m_y \gamma_y rVk_м T_{см} n_{см} \, , \tag{4.6}$$

where $m_{пр}$ is undercut rock thickness, m; γ_y, γ_n are density of coal and undercut rocks, respectively, t/m^3; $D_{г.ф.}$ is output per longwall depending on gas factor, t/day

$$D_{г.ф.} = \frac{3456 S_{O4} l}{0{,}75 g_{пл} (l + 43{,}2 rVk_м)}, \tag{4.7}$$

where S_{O4} is crosscut nominal area of a stope face space, m^2; $g_{пл}$ relative methane emission, m^3/t.

<u>3. A cutter-loader feeding velocity.</u> A cutter-loader feeding velocity is calculated depending upon accepted extraction technique:

in terms of one-pass complete and separate mining

$$V = \frac{Pt_{cut}}{r(m_1 \bar{A}_1 + k_{app} m_2 \bar{A}_2)} - 0{,}2V_{cut} \, , \tag{4.8}$$

where P is total power consumed by a cutter-loader engines, kW; t_{cut} is distance between operating cutting lines, cm; , m_1, m_2 is thickness of a seam extracted with the help if forward and back end organ respectively, m; $\bar{A}_1, \bar{A}_2$ is cuttability of a seam extracted with the help of forward and back end organ respectively, kN/m; k_{app} is a coefficient of rock mass wreaking; and V_{cut} is cutting velocity, m/c;

in terms of two-pass separate mining (forward and reverse)

$$V = \frac{V_y + V_n}{V_y V_n}, \tag{4.9}$$

where V_y is a cutter-loader feeding velocity in the process of coal seam mining, m/min

$$V = \frac{Pt_{cut}}{rm_y \bar{A}_y k_{app}} - 0.2V_{cut} \, , \tag{4.10}$$

where $\bar{A}_y$ is a coal seam cuttability, kN/m; and V_n is a cutter-loader feeding velocity in the process of undercut rock mining, m/min

$$V_n = \frac{Pt_{cut}}{rm_{np}\,\bar{A}_n\,k_{app}} - 0{,}2V_{cut}, \qquad (4.11)$$

where $\bar{A}_n$ is undercut rock cuttability, kN/m.

If V_n, calculated on above expression, is more than $0.8V_{gon}$ value (being maximum feeding velocity), then it is be equated with $0.8V_{gon}$.

4. Ash-content. Ash-content is calculated for each stope and the whole mine:

ash-content of rock mass in longwalls with complete seam mining ($A_{гм}$)

$$A_{гм} = \frac{A_y m_y \gamma_y + A_n \gamma_n (m_{np} + m_{бn})}{m_y \gamma_y + \gamma_n (m_{np} + m_{бn})}, \qquad (4.12)$$

where A_y, A_n is source ash-content of coal seam and undercut rocks respectively, %; and $m_{бn}$ is thickness of enclosing roof and floor dilution, m;

ash-content of coal mined in longwalls where separate seam mining is applied ($A_{д.у.}$):

$$A_{д.у.} = \frac{A_y m_y \gamma_y + A_n \gamma_n \left[m_{np}(1-k_n) + m_{бn} \right]}{m_y \gamma_y + \gamma_n \left[m_{np}(1-k_n) + m_{бn} \right]}, \qquad (4.13)$$

where k_n is a coefficient involving mined coal dilution due to incomplete undercut rock loading;

ash-content of coal laded in a mine ($A_{Ш}$):

$$A_{Ш} = \frac{\sum_{i=0}^{0} D_{вi} A_{г.м.i} + \sum_{j=0}^{n} D_{cj} A_{д.у.j}}{\sum_{i=n}^{0} D_{вi} + \sum_{j=0}^{h} D_{cj}}, \qquad (4.14)$$

where $A_{г.м.i}$ is ash-content of rock mass extracted in i^{th} longwall, %; $A_{д.у.j}$ is ash-content of coal mined in i^{th} longwall, %.

5. Trade price for a ton of end product. Trade price depends on a goal set in economic and mathematical model. A mine can lade coal either for concentration plant or for a consumer. Optional version is when pure coal is laden from concentration plant to a consumer as well as related supplies. Common expression to identify trade price for end product (P_w) laded for a consumer is:

$$P_w = k_{on} P_{tp} + k_{оф} \cdot k_{ск_ш} (k_{гм} P_{tpp} + k_к k_{ВК} P_c), \qquad (4.15)$$

where k_{on} is a factor involving quantity laden for a consumer:

$$k_{on} = 1 - k_{оф}, \qquad (4.16)$$

where $k_{оф}$ is a factor involving quantity laden for concentration plant;

P_{tp} is trade price for a ton of coal laden by a mine for a consumer, UAH/t:

$$P_{tp} = P_{tlp}\left[1 + 0.025(A_{пру} - A_{Ш})\right], \tag{4.17}$$

where P_{tlp} is trade listed price for a ton of coal, UAH/t; 0.025 is reduction coefficient for listed ash-content excess; $A_{пру}$ is listed ash-content of coal, %; and $k_{ск_ш}$ is a mine reduction coefficient for scheduled ash-content excess:

$$k_{ск_Ш} = 1 - 0.02(A_ш - A_{пл.ш}), \tag{4.18}$$

where 0.02 is reduction coefficient for scheduled ash-content excess; $A_{пл.ш}$ is scheduled ash-content in mine %; and $k_{к_2}, k_{гм}$, are coefficients involving end product type;

- while calculating in accordance with rock mass transported to preparation plant:

$k_{гм} = 1; k_к = 0$;

- while calculating in accordance with concentrate transported from preparation plant to a consumer:

$k_{гм} = 0; k_к = 1$.

P_{tpp} is trade price for a ton of rock mass transported to a preparation plant, UAH/t:

$$P_{tpp} = P_{tlp}\left[1 + 0.025(A_{пр_у} - A_{пл.ш})\right]; \tag{4.19}$$

$k_{ВК}$ is coefficient involving concentrate yield from rock mass being prepared

$$k_{ВК} = 1 - (A_{пл.ш} - A_к)k_{об} - k_{пот}, \tag{4.20}$$

where $A_к$ is ash-content of concentrate, %; $k_{об}$ is coefficient involving mine refuse yield; and $k_{пот}$ is coefficient involving preparation loss.

P_c is trade price for a ton of concentrate transported to a consumer, UAH/t:

$$P_c = P_{ltpc}\left[1 + 0.025(A_{пр\cdot к})\right], \tag{4.21}$$

where P_{ltpc} listed trade price for a ton of concentrate, UAH/t; and $A_{пр\cdot к}$ is listed ash-content of concentrate, %.

6. <u>Expenditures connected with a ton of coal (rock mass) extraction, transportation, and preparation.</u> In common with trade price, expenditures connected with a ton of coal (rock mass) extraction, transportation, and preparation depend on product transportation variant rated by:

$$C = k_{оп}C_{гм} + k_{оф}(k_{гм}C_{гм} + k_к C_к), \tag{4.22}$$

where $C_{гм}, C_{к}$ are own costs of a ton of rock mass and concentrate being transported, respectively, UAH/t.

$$C_{гм} = \frac{\sum_{i=n}^{0} C_i D_{вi} + \sum_{j=0}^{n} C_j D_{cj}}{\sum_{i=n}^{0} D_{вi} + \sum_{j=0}^{n} D_{cj}} + C_{тр}, \qquad (4.23)$$

where C_i, C_j are own costs of coal (rock mass) extraction in i^{th} and j^{th} longwalls, respectively, UAH/t; $C_{тр}$ are expenditures connected with a ton of coal (rock mass) transportation, UAH/t.

$$C_{к} = C_{гм} + C_{тр.оф} + C_{об} + (A_{пл.ш} - A_{к})k_{об} \cdot C_{тр.хв}, \qquad (4.24)$$

where $C_{тр.оф}$ are expenditures connected with a ton of rock mass transportation from a mine to a preparation plant, UAH/t; $C_{об}$ is own cost of preparation, UAH/t; $C_{тр.хв}$ expenditures connected with a ton of waste refuse transportation and impoundment, UAH/t.

7. Expenditures connected with rock utilization. Total cost value for utilization of rock extracted from longwalls applying separate seam mining ($C_{ут.общ}$) is determined as follows:

$$C_{ут.общ} = \sum_{j=0}^{n} Q_{П} n_j C_{ут_j} = \sum_{j=0}^{n} Q_{П_j} n_j \left[R_{П_j} - \left(C_{П_j} + E_{н} \frac{K_j}{Q_{П_j} n_j} \right) \right] \qquad (4.25)$$

where $Q_{П_j}$ is volume of rock extracted in j^{th} longwall, tons per day;

$$Q_{n\,j} = D_{cj} \frac{m_{npj} \cdot \gamma_{nj}}{m_{yj} \cdot \gamma_{yj}},$$

$R_{П_j}$ is utilization effect of a ton of rock extracted in j^{th} longwall, UAH/t;

$$R_{П_j} = \left(\frac{C_{штрj}}{\gamma_{nj} l_j m_{npj} r_j} + C_{тр_{nj}} \right) k_{3j} + C_{троф} + C_{об} + (A_{пл.ш} - A_{к}) k_{об} C_{тр_{хв}}, \qquad (4.26)$$

where $C_{штрj}$ are expenditures connected with a running meter haulage roadway drivage in j^{th} longwall, UAH; l_j is j^{th} longwall length, m; r_j is web width in j^{th} longwall, m; $C_{тр_{nj}}$ are expenditures connected with a ton of rock transportation from j^{th} longwall, UAH/t; k_{3j} is coefficient involving a technique of rock utilization extracted in j^{th} longwall: k_{3j}=1 when rock stowing into worked-out space of a longwall; k_{3j}=0 otherwise. C_{nj} are expenditures connected with utilization of a ton of rock extracted in j^{th} longwall, UAH/t;

$$C_{nj} = C_{3nj} k_{3j} + C_{mp_{nj}} k_{mp_j}, \quad (4.27)$$

where C_{3nj} are expenditures connected with a ton of rock stowing into worked-out space of j^{th} longwall, UAH/t; k_{mp_j} is coefficient involving a technique to utilize rock extracted in j^{th} longwall: k_{mp_j}=0 when rock stowing into worked-out space of a longwall; k_{mp_j}=1 otherwise.

E_H is capital investment standard efficiency coefficient; K_j is investment connected with j^{th} longwall transfer to separate seam mining, mille UAH.

4.2.3. Algorithm of Economic and Mathematical Model Analysis Using PC

Algorithm of the model analysis using PC to determine effective area and extent of selective seam mining technique application consists of successive blocks; each of them performs its own functions. In addition to sequencing of computations in accordance with mentioned formulas (subsection 2), algorithmization provides identification of required auxiliary values and parameters. Below you can find explanation how blocks of the algorithm operate and interrelate.

Block of Initial Data Preparation and Input. The block covers initial data characterizing mining and geological, mine technical, qualitative, and cost parameters of association, mine, stopes, and preparation plant. Some data are input in the form of constants, and others – in the form of variables varied depending upon specific situation and techniques of seam mining.

Block Determining Economically Feasible Application Area For Selective Seam Mining. The block is the first active block of economic and mathematical model. In addition to basic problem solution based on results obtained, longwalls are selected to transfer to separate mining. As it has been already mentioned, thickness of undercut enclosing roof and floor is running parameter for economic and mathematical model and for the block. The variation is not performed for longwalls with less than 5 cm undercut.

Maximum profit is the optimization criterion used to estimate economically feasible application area for selective seam mining technique in terms of certain mine. Computations are gradually performed for each longwall where undercut thickness is more than 5 cm. It is done as follows.

1. Thickness of enclosing roof and floor in terms of complete seam mining determines both boundaries and variation step.

2. Involving undercut thickness variation rated values for a cutter-loader feeding velocity in terms of complete seam mining are evaluated.

3. Design output of a stope is determined involving undercut and feeding velocity.

4. Ash-content of rock mass extracted in a longwall is defined.

5. A mine output is determined involving changes in i^{th} longwall output and ash-content of rock mass extracted in it.

6. Ash-content for a mine is calculated involving changes in i^{th} longwall ash-content.

7. Trade price for a ton of end product is determined involving computed transportation option.

8. Value of cost specific charges for a ton of coal (rock mass) extraction, transportation, and preparation is defined.

9. Then profit margin from end product sales is determined for each running value of undercut enclosing roof and floor thickness in i^{th} longwall.

When variant with complete seam mining in this longwall was calculated, profit calculation in terms of selective seam and rock mining is performed. Moreover, two mining variants are calculated: one-pass and two-pass. In the context of one-pass mining profit is calculated similarly to abovementioned (i.e. for complete seam mining). The only difference is that after item 8 expenditures connected with rock utilization are estimated. In the context of two-pass mining profit is calculated as follows:

10. First cutter-loader feeding velocity is calculated in terms of coal seam and undercut enclosing roof and floor mining.

11. Then value of enclosing roof and floor is determined in terms of selective seam mining.

12. Capacity of j^{th} longwall is calculated.

13. Ash-content of coal extracted from j^{th} longwall is evaluated.

14. The mine capacity is identified.

15. Ash-content of the mine is evaluated.

16. Trade price of a ton of end product is determined.

17. Specific expenditures connected with a ton of coal extraction, transportation and preparation are evaluated.

18. Expenditures connected with rock utilization are determined.

19. The mine profit margin is evaluated.

20. In terms of profit margin obtained determination of efficient areas for complete and selective mining techniques takes place; it also involves thickness of undercut enclosing roof and floor in j^{th} longwall.

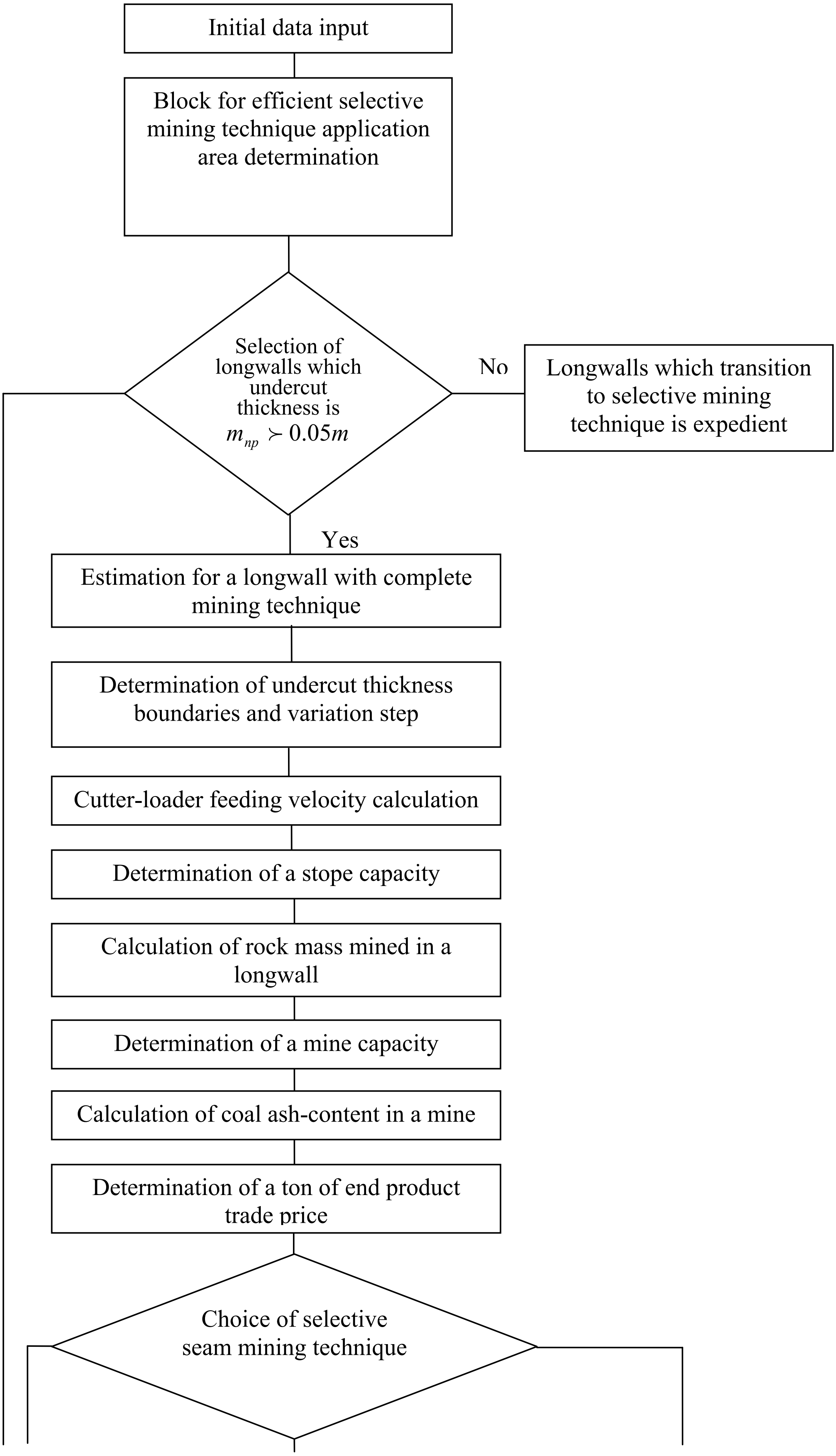
21. The cycle is performed for each longwall where undercut is more than 5 cm.

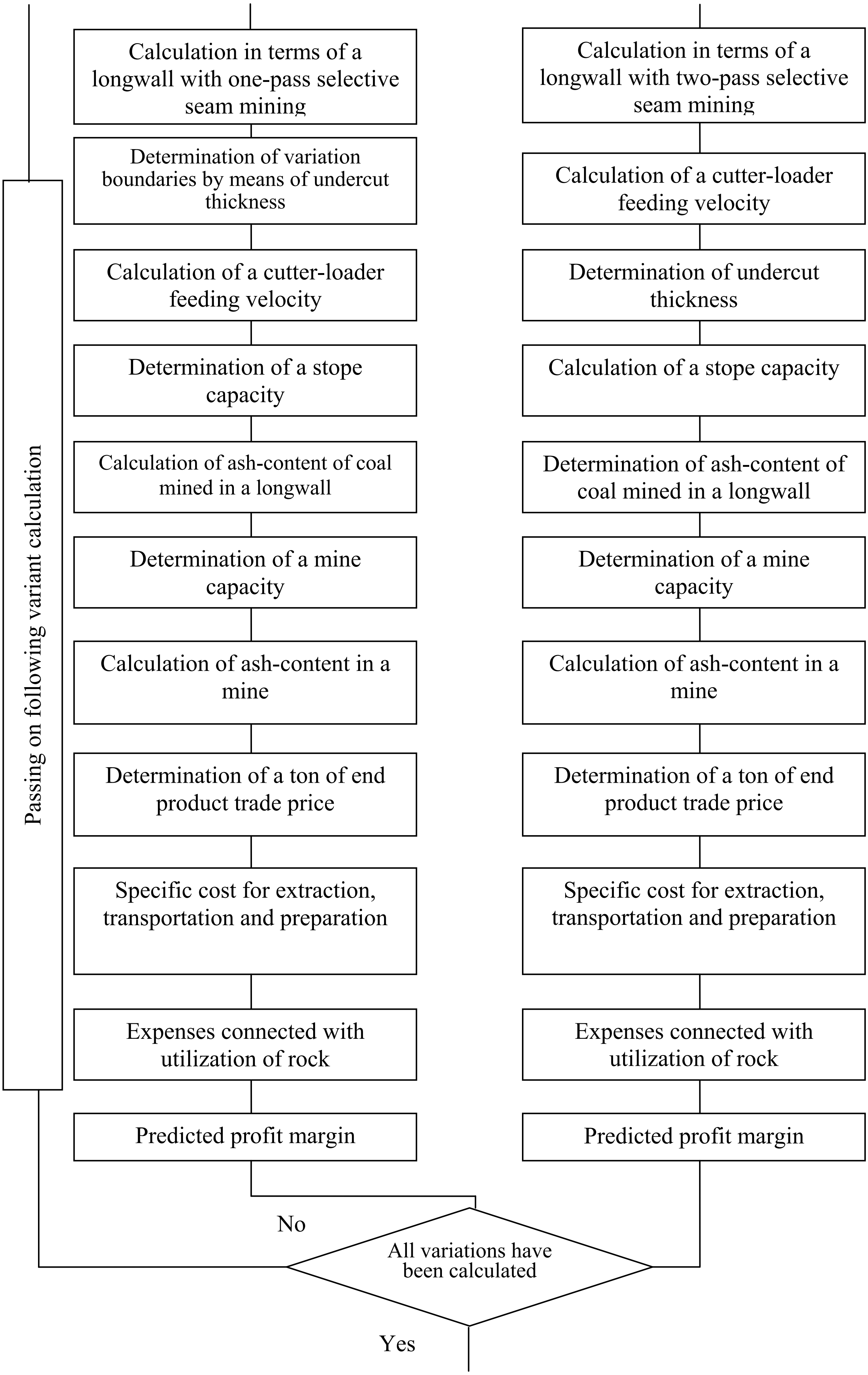
22. Longwalls are selected in which transition to selective technique is efficient.

Estimation Block for Selective Technique Efficient Application. Estimation is performed using a technique of search. The number of variations to be considered depends on the number of selected longwalls. Comparison technique helps to determine such a ratio of longwalls where complete mining and selective mining are applied to ensure maximum profit. Calculations are performed as follows:

1. Extent of output in a mine is determined.
2. Ash-content of coal transported by a mine is calculated.
3. Trade price of a ton of end product (e.g. coal or concentrate) is determined.
4. Cost for extraction, transportation, and preparation is calculated.
5. Expenses connected with rock utilization are estimated.
6. In terms of applicable selective technique profit is fixed.
7. Optimum technique is determined.
8. Output data in terms of applicable techniques are printed out.

Fig. 4.3 demonstrates structure flow chart algorithm to analyze economic and mathematical model for determining efficient application area of selective seam mining technique and its rational utilization capacity.


Initial data input
Block for efficient selective mining technique application area determination
Selection of longwalls which undercut thickness is $m_{np} \succ 0.05m$
No
Longwalls which transition to selective mining technique is expedient
Yes
Estimation for a longwall with complete mining technique
Determination of undercut thickness boundaries and variation step
Cutter-loader feeding velocity calculation
Determination of a stope capacity
Calculation of rock mass mined in a longwall
Determination of a mine capacity
Calculation of coal ash-content in a mine
Determination of a ton of end product trade price
Choice of selective seam mining technique


Calculation in terms of a longwall with one-pass selective seam mining
Determination of variation boundaries by means of undercut thickness
Calculation of a cutter-loader feeding velocity
Determination of a stope capacity
Calculation of ash-content of coal mined in a longwall
Determination of a mine capacity
Calculation of ash-content in a mine
Determination of a ton of end product trade price
Specific cost for extraction, transportation and preparation
Expenses connected with utilization of rock
Predicted profit margin
Calculation in terms of a longwall with two-pass selective seam mining
Calculation of a cutter-loader feeding velocity
Determination of undercut thickness
Calculation of a stope capacity
Determination of ash-content of coal mined in a longwall
Determination of a mine capacity
Calculation of ash-content in a mine
Determination of a ton of end product trade price
Specific cost for extraction, transportation and preparation
Expenses connected with utilization of rock
Predicted profit margin
Passing on following variant calculation
All variations have been calculated
No
Yes

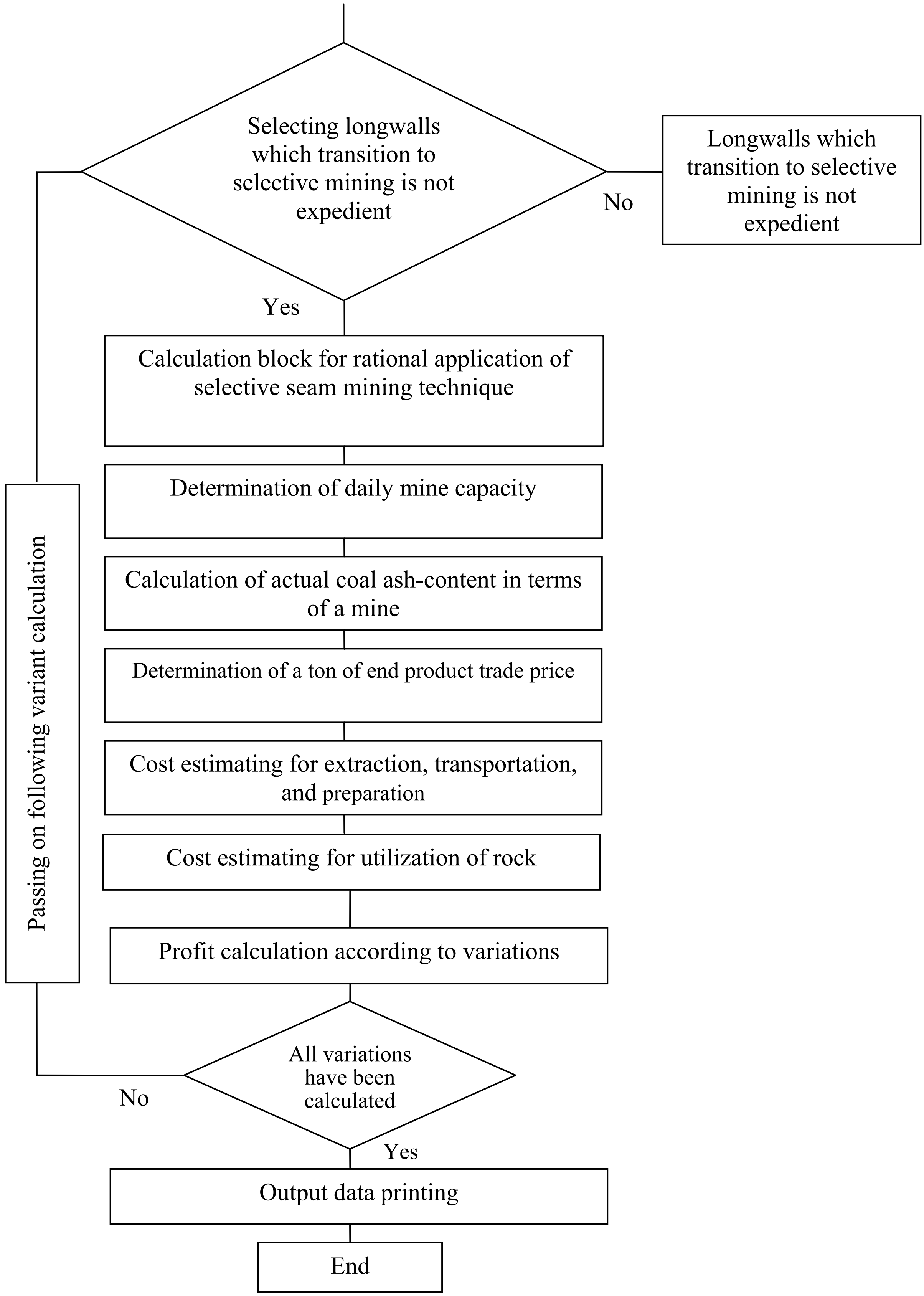


Fig. 4.3 Total block representation of economic and mathematical model algorithm to reason rational area and volumes for selective seam mining technique.

4.2.4. Modeling Results Analysis

Tables 4.1 and 4.2 demonstrate initial data received in the process of analyzing PC-based economic and mathematical model to reason rational area and volumes of selective seam mining technique application for mines in Western Donbass and Lvov-Volyn coal field.

Table 4.1. Initial Data for Mines in Western Donbass

Factors	Units of measurements	Longwall			
		1	2	3	4
1.Coal seam thickness	m	0.6	0.7	0.8	0.9
2. Undercut thickness plan/actual	m/m	$\frac{0.10}{0.30}$	$\frac{0}{0.20}$	$\frac{0}{0.10}$	$\frac{0}{0}$
3. Longwall capacity:					
Plan	tons per day	810	660	700	1000
Actual (in terms of rock mass)	tons per day	1074	954	840	1000
Reduced	tons per day	720	509	623	1000
4. Ash-content of filling up coal:					
Plan	%	37.1	20.7	20.0	21.5
Actual (in terms of rock mass)	%	53.6	44.0	32.9	21.5
5. Average mine capacity:					
Plan	tons per day		3170		
Actual (in terms of rock mass)	tons per day		3968		
Reduced	tons per day		2852		
6. Ash-content in a mine:					
Plan	%		25		
Actual	%		38.4		

Table 4.2. Initial Data for Mines in Lvov-Volyn Coal Field

Factors	Units of measurements	Longwall			
		1	2	3	4
1. Coal seam thickness	m	0.6	0.7	0.8	0.9
2. Undercut thickness plan/actual	m/m	$\frac{0.10}{0.30}$	$\frac{0}{0.20}$	$\frac{0}{0.10}$	$\frac{0}{0}$
3. Longwall:					

Plan	tons per day	720	900	940	1000
Actual (in terms of rock mass)*	tons per day	$\frac{887}{1182}$	$\frac{842}{1122}$	$\frac{785}{1047}$	1000
Reduced*	tons per day	$\frac{549}{792}$	$\frac{450}{599}$	$\frac{562}{777}$	1000
4. Ash-content of filling up coal:					
Plan	%	37.1	20.7	20.0	21.9
Actual (in terms of rock mass)	%	53.6	44.0	32.9	21.9
5. Average capacity of a mine:					
Plan	tons per day		3480		
Actual (in terms of rock mass)	tons per day		$\frac{3514}{4351}$		
Reduced	tons per day		$\frac{2626}{3141}$		
6. Ash-content of :					
Plan	%		25.0		
Actual	%		38.9		

* Numerator is 0.6 m web width; denominator is 0.8 web width.

Tables 4.1 and 4.2 describe only basic data taken up in the process of modeling; other data required to perform PC-based calculations as for mining, geological, and economic factors are specified as those typical for mines in Western Donbass and Lvov-Volyn coal field. Some of them are fore-quoted in Table 2.5 (Chapter 2).

In both cases, calculations are made for coal seams with 0.6 m, 0.7 m, 0.8 m, and 0.9 m thickness. Undercut height calculated with the help of expression (2.10) is 0.3 m, 0.2 m, 0.1 m, and 0 m respectively if seams of specified thickness are mined with the help of 1KM103 powered system. That is in the context, minimum mined seam thickness is 0.9 m when m_{min} is 0.7 m is recommended by specifications of 1KM103 system.

Planned and actual ash-content of mined coal were determined on 2.56 and 2.57 expressions involving undercut thickness according to the system specifications and calculations by the NMU. Capacity of stopes was calculated according to 2.27and 2.33 expressions taking into account mined coal ash-content and ignoring it. Moreover, planned capacity was calculated for thickness of seams with minimum provided for the system specifications (i.e. 0.7 m); in terms of rock mass actual capacity was calculated for extracted seam thickness depending upon calculations (i.e. 0.9 m). Reduced capacity was determined by means of rock mass ash-content normalizing as well as planned ash-content of coal mined with the help of 1KM103 system when 0.7 m is minimum extracted thickness. It should be noted that calculations for actual and reduced capacity for coal mining in Lvov-Volyn coal field involve negative phenomena as a result of underground investigations. The phenomena depend on hard enclosing roof and floor undercut, specifically deceleration of a cutter-loader feeding velocity and decrease in web width of end organ (0.8 m to 0.6 m). As Table 4.2 demonstrates sizable effect on a value of actual and reduced capacity is available.

Area of Economical Application for Selective Seam Mining Technique

Table 4.3 demonstrates results of implementing block one of economic and mathematical model to determine economic area for selective seam mining technique depending upon its thickness and enclosing roof and floor value involving various techniques and ways of coal haulage taking into account mined coal preparation and without it. Figures 4.4 and 4.5 describe certain averaged dependences typical for coal-mining regions under study. It follows from Fig. 4.4 that one-pass selective seam mining technique with floor rock undercut is economic for mines in Western Donbass. Such a technique ensures top profits practically within the whole range of undercut thickness. It should be noted that in the process of undercut waste stowing profit experiences minor changes as when the rocks are transported to a surface and undercut value increases, profit drops sharply. It depends on the fact that one-pass selective seam mining minor dilution takes place which reduces profit preventing from covering expenses connected with stowing and waste rock transportation. As a rule, two-pass selective seam mining provides higher profits adequate for the expenses covering.

Table 4.3 Values Undercut Thickness Economic for Transition from Complete Mining to Separate One

Haulage types	Selective Extraction and a Seam Thickness																	
	Seam Extraction with Floor Rock Undercut												Seam Extraction with Roof Rock Undercut					
	One-pass technique with stowing			One-pass technique with haulage to the surface			Two-pass technique with stowing			Two-pass technique with haulage to the surface			One-pass technique with stowing			One-pass technique with haulage to the surface		
	0.6	0.7	0.8	0.6	0.7	0.8	0.6	0.7	0.8	0.6	0.7	0.8	0.6	0.7	0.8	0.6	0.7	0.8
Mine-consumer	0.07	0.10	0.06	0.08	0.11	0.09	0.10 / 0.15	0,06 / –	0,03 / –	0,11 / 0,17	0,07 / –	0,04 / –	0.15	0.13	0.11	0.18	0.15	0.12
Mine-preparation plant	0.1	0.11	0.11	0.13	0.13	0.16	0,12 / 0,15	0,12 / 0,14	0,12 / 0,12	0,15 / 0,17	0,15 / 0,16	0,15 / 0,14	0.18	0.16	0.12	0.23	0.20	0.14
Mine-preparation plant - consumer	0.1	0.13	0.11	0.13	0.18	0.16	0,12 / 0,15	0,08 / –	0,05 / –	0,14 / 0,16	0,09 / –	0,06 / –	0.18	0.16	0.13	0.23	0.30	0.14

*Value of undercut thickness in terms of which transition from complete mining to two-pass separate one is reasonable

** Value of undercut thickness in terms of which transition from one-pass separate mining to two-pass separate one is reasonable.

In the context of such a technique, profits experiences sharp increase; notably it concerns the situation when stowing takes place. However, under considered circumstances the technique prevents from reaching such profits as it is possible for one-pass selective mining. If only undercut values are close to maximally accepted (that is m_{np}= 0.35) then values of achievable profits experience certain equalisation. That makes it possible to say that if undercut thickness is 0.35 m and more in some cases selective two-pass mining is more reasonable for Western Donbass than one-pass technique. According to Fig. 4.4, lower boundary of such technique reasonability is within 0.15 m in terms of stowing, and 0.2 m if undercut rocks are conveyered to the surface.

Roof undercut is the most reasonable in the context of Lvov-Volyn coal field. In such a case rather high profits are made in terms of complete mining and separate one. Fig. 4 demonstrates dependences of mine profits on mining technique and undercut location confirming the fact. Reasonable boundary of transition from complete mining to separate one is within 0.19 to 0.25 m; it depends on a technique of rock utilization (Fig. 4.5 b). So, it can be said with confidence that either one-pass complete mining technique (if $m_{np} \leq$ 0.19-0.25 m) either selective one (if $m_{np} \geq 0.19 - 0.25$ m) with roof rock undercut is the most reasonable for Lvov-Volyn coal field. The techniques help to avoid disadvantages typical for mining with hard floor rocks reaching higher cost-performance ratios. However, their application area is restricted as roof undercut may disturb its continuity having negative consequences.

Techniques with floor rock undercut are common practice. In this situation the three mining techniques are applicable: complete mining, one-pass separate mining, and two-pass separate mining. Fig. 4.5 a helps evaluate constructive efficiency of this or that technique application, or boundaries of reasonability to transform from one technique to another. It is understood that complete mining if efficient if undercut thickness is 0.09 to 0.12 m.

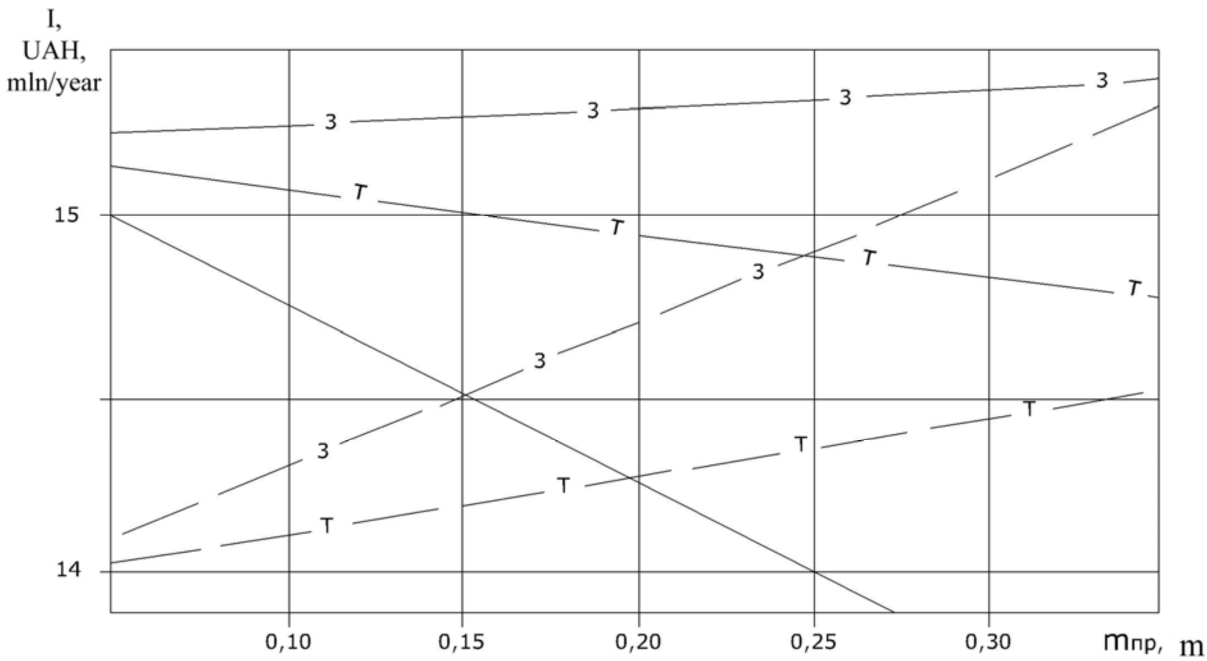


Fig. 4.4 Reasonable application areas for complete and selective mining in Western Donbass with floor rock undercut:_____ -complete seam mining; – 3 – 3 –separate seam mining (one-pass and two-pass respectively) with stowing; – T – T – with rock haulage to the surface

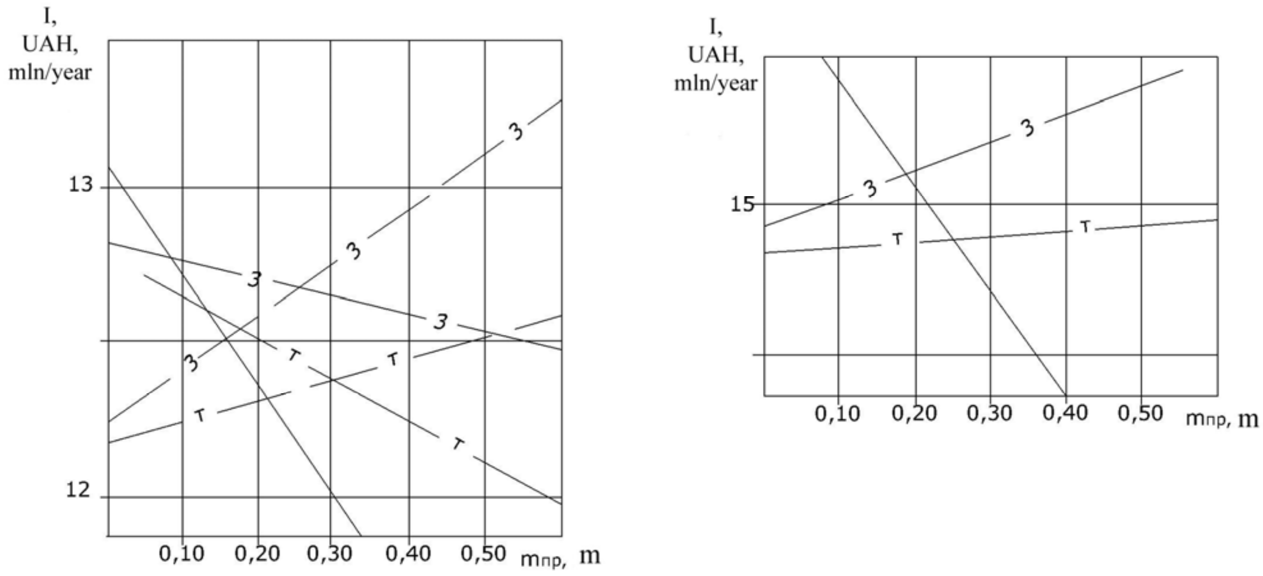


Fig. 4.5 Reasonable application areas for complete and selective seam mining in Lvov-Volyn coal field with floor rock undercut (a) and roof floor undercut (b): — *- complete seam mining; – 3 – 3 – separate seam mining (one-pass and two-pass respectively) with stowing; – T – T – with rock haulage to the surface*

Transition to selective mining is more reasonable for seams with thicker undercut; moreover, if undercut is 0.09-0.12 m to 0.17-0.20 m, then one-pass mining is more efficient; if $m_{np} \geq 0.17 - 0.20m$, then two-pass mining is more functional.

In each considered case smaller boundary undercut thickness value corresponds to alternatives with undercut waste stowing, and larger corresponds to their haulage to the surface.

Operations with stowing are the most efficient for the Associations.

It should be noted that each case requires reasoning of this or that technique technological expediency depending upon a mine specifity.

Table 4.4 and Fig. 4.6 demonstrate results of economic and mathematical model (block two) implementation to identify reasonability of complete and selective seam mining in terms of conventional mine.

Table 4.4 Results of economic and mathematical model (block two) implementation as for reasonability of complete and selective seam mining techniques application

Factors	Units of measurements	Estimated alternative							
		1 complete.	1 complete.	1 complete.	1 complete.	1 complete.	1 complete.	1 complete.	1 complete.
		2 complete.	2 complete.	2 complete.	2 complete.	2 complete.	2 complete.	2 complete.	2 complete.
		3 complete.	3 complete.	3 complete.	3 complete.	3 complete.	3 complete.	3 complete.	3 complete.
1. A mine capacity									
- Actual in terms of rock mass	tons per day	3868	3394	3563	3722	3089	3417	3248	2943
- Reduced	tons per day	2831	2946	2856	2866	3015	2932	2975	2943
2. Ash-content in terms of a mine									
- Actual	%	38.4	31.6	34.3	36.5	26.2	32.1	29.2	23.3
- Plan	%	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0
3. Trade price of a ton of coal	RUB/t	14.27	16.89	16.45	15.28	20.77	17.63	19.17	22.31
4. Trade price of a ton of coal									
- trade price of rock mass	RUB/t	5.93	6.75	6.51	6.31	7.81	7.00	7.40	9.03
- trade price of coal with reduced ash-content	RUB/t	8.11	7.75	7.95	8.13	8.00	8.16	8.08	9.03
- trade price of concentrate	RUB/t	13.46	13.10	13.30	13.48	13.35	13.51	13.43	14.38
5. Profit									
- if it is transported to a customer	RUB, mln	11.59	$\frac{14.00}{14.54}$	$\frac{12.99}{13.31}$	$\frac{12.11}{12.24}$	$\frac{15.04}{16.16}$	$\frac{13.84}{14.84}$	$\frac{14.52}{15.52}$	$\frac{14.83}{15.82}$
- if it is transported to preparation plant	RUB, mln	13.45	$\frac{14.84}{15.38}$	$\frac{14.23}{14.56}$	$\frac{13.82}{13.85}$	$\frac{15.31}{16.30}$	$\frac{14.74}{15.74}$	$\frac{15.03}{16.03}$	$\frac{14.83}{15.82}$
- involving preparation	RUB, mln	12.85	$\frac{14.06}{14.60}$	$\frac{13.52}{13.85}$	$\frac{13.07}{13.20}$	$\frac{14.64}{15.64}$	$\frac{14.14}{15.13}$	$\frac{14.40}{15.39}$	$\frac{14.20}{15.19}$

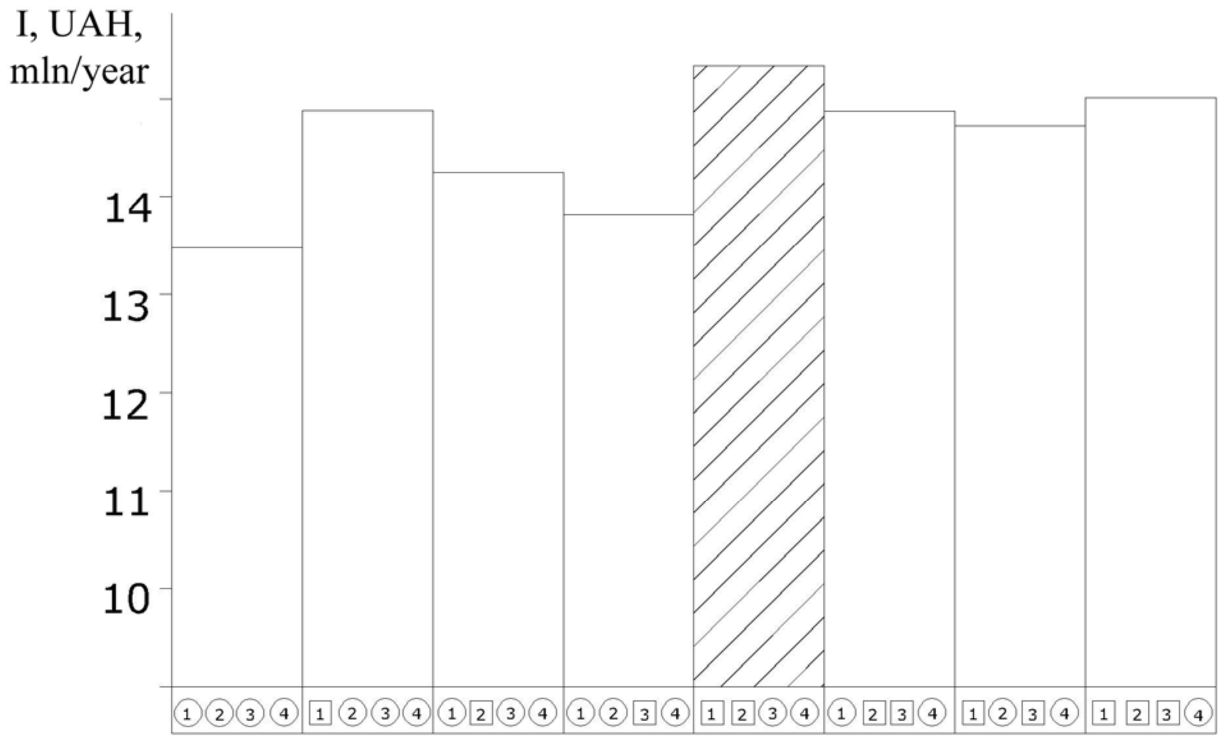


Fig. 4.6 Reasonable capacity to apply complete and selective seam mining techniques in terms of conventional mine where seam thickness is 0.6 m-1, 0.7 m-2, 0.8 m-3, and 0.9 m-4:○ – complete mining; □ – separate coal and undercut rock mining.

Represented data confirm that the following is the most reasonable: two longwalls perform selective mining (seam thickness is 0.6 and 0.7 m), and other two longwalls respectively perform complete mining (seam thickness is 0.8 m and thickness in terms of coal is 0.9 m). If so, annual profit of the mine is almost UAH 3.5 billion.

Thus, it is possible to determine a set of techniques in terms of each specific mine to provide maximum possible profits.

4.3. The Potentials to Apply Selective Seam Mining Technique

In Ukrainian Donetsk coal field, reserves of seams with less than 0.6 m thickness are 51.4% of all balance flat and steep seams [94]. According to siddles, distribution of reserves is as follows: 68.6% up to 18°, and 31.4% within 19-35°.

Coking coal (56.1%) followed by thermal coal (28.8%) and anthracite (15.1%) prevail in reserves of flat and steep seams which thickness is up to 0.8 m. Table 4.5 demonstrates distribution of reserves within up to 0.8 m thickness seams according to their grades.

Balance reserves with enclosing roof and floor of mean stability are 72.4% (in terms of m=0.51-0.6 m seams they are 75.9%; 0.61-0.70 m and 0.71-0.80 m they are 71.4% and 71.7% respectively). More stable rocks are largely typical for seams with coking coal; 75.2% of their reserves are in seams with mean stability rocks. As for seams with thermal coal and anthracite they are 74.5% and 58.3% respectively.

Table 4.5 Distribution of coal reserves depending upon a seam thickness, coal grades, and enclosing roof and floor stability, %

Coal grades	Enclosing roof and floor stability	Seam thickness, m			Total
		0.51-0.60	0.61-0.7	0.71-0.80	
Coking	Stable and mean-stable	9.8	15.8	16.6	42.2
	Hazardous	4.2	4.3	5.4	13.9
	Stable and mean-stable	3.9	7.0	10.5	21.4
Thermal	Hazardous	0.2	3.4	3.8	7.4
	Stable and mean-stable	0.2	2.7	5.8	8.7
Anthracite	Hazardous	-	2.5	3.8	6.3
	Stable and mean-stable	13.9	25.5	33.0	72.4
	Hazardous	4.4	10.2	13.0	27.6

Analogous situation is with Lvov-Volyn coal field where commercial coal reserves in seams with less than 1 m thickness were 77.5 %; among them 22.8% were in 0.51-0.70 m. Total percentage of coking coal in commercial reserves was 43.7%; of them, 44.7% were deposited in less than 1 m thickness, and 58.4% in seams with 0.51 to 0.70 m thickness.

All the reserves are concentrated in flat seams with 0 − 8° dips. Roof stability of thin seams is basically mean and higher; it favours application of powered systems. It specifically concerns 1KM103 system with selective seam mining technique. However, in Ukrainian mines (particularly, it is related to Western Donbass and Lvov-Volyn coal field) complete mining with enclosing roof and floor is widely used (almost 150 longwalls annually).

At the same time, as above results explain, selective seam mining technique is quite possible to be applied in each longwall working with enclosing roof and floor undercut.

Conclusions

1. Application area for separate seam mining with the help of 1KM103 powered system depends on:

- Mined thickness of coal seam and enclosing roof and floor undercut thickness;

- Enclosing roof and floor stability and undercut rock hardness.

2. Lower boundary of mined coal seam thickness or in the context of advance undercut of enclosing roof and floor, undercut thickness is limited by minimum dimensions (diameter) of powered system 1K103 end organs being m_{min}=560 mm.

3. Maximum of enclosing roof and floor undercut or in the context of subsequent break, maximum mined thickness of coal seam is limited by a maximum of a cutter-loader end organs extension; for screws with 560, 630, 710 and 800 mm diameters it is:

- 245; 280; 320 and 365 mm in the context of advance extraction of upper patch of coal or rock; and

- 555; 510; 470 and 425 mm in the context of advance extraction of lower patch of coal or rock.

4. Application area of 1KM103 system is limited by thin and very thin gently dipping seams where roof rocks are either stable or midstable. In some cases new technique makes it possible to mine seams with false roof; particularly it concerns those where its thickness is not more than maximum undercut values.

5. 1K103 cutter-loader is able to mine seams where hardness of enclosing roof and floor undercut is up f=4…5. Moreover, in the context of hard floor rocks undercut two-pass separate seam mining; in the context hard roof rocks one-pass technique is reasonable.

6. Powered system 1KM103 is able to mine very thin seams in Western Donbass classified as non-commercial reserves (which thickness is 0.1-0.55 m).

7. If undercut floor rocks are of considerable thickness, and undercut rocks are very hard then separate two-pass mining is more reasonable to compare with one-pass technique. Mostly it has relation to seams in Lvov-Volyn coal field.

8. Rational application area for complete and separate mining both depends on thickness of a seam and undercut of enclosing roof and floor, and on mined coal consumer. Thus, when coal is directly transported to a consumer, complete technique

(in terms of Western Donbass) is efficient if undercut value is 0.08 to 0.11 m; if transported to preparation plant, it is 0.13 to 0.19 m respectively.

9. Economically separate one-pass seam mining with more than 0.1-0.19 m undercut of enclosing roof and floor is more reasonable for mines in Western Donbass (depending upon specific conditions):

a) for seams where coal thickness is 0.6 to 0.8 m with advance coal seam extraction;

b) for seams where coal thickness is less than 0.6 m with advance extraction of soft undercut rocks.

10. Economically one-pass separate coal seam mining is the most reasonable for mines in Lvov-Volyn coal field if the thickness of the seams are 0.5 to 0.8 m, and undercut of hard roof rocks is more than. When roof undercut is either impossible or undesirable it is reasonable to apply two-pass separate mining for seams with specified thickness where floor rock undercut is more than 0.15-0.09 m.

11. There is no any necessity to apply separate mining for each longwall to realize maximum profits; one or two longwalls in a mine is quite enough.

12. For each particular case values of undercut thickness to be reasonable for transition from complete mining to separate one and vice versa have different values. That is why rational areas and capacities for different techniques should be calculated individually depending upon each specific longwall and a mine at large.

CONCLUSION

The monograph is completed research setting and solving topical scientific and practical problem of substantiating basic parameters and application area for selective mining thin and very thin coal seams.

Following tangible results have been obtained.

1. Basic parameters of selective mining technique for thin coal seams:

- Dependence of minimum thickness extracted with the help of powered system on design parameters of a stope equipment as well as mining and geological conditions of a seam mining. The difference is that it considers physiological parameters required for high labour efficiency, operational and time parameters characterizing features of the technique as well as maximum approximation of enclosing roof and floor in a longwall;

- Dependence of a cutter-loader feeding velocity $\bar{v}$, machine time coefficient $k_м$, specific power consumption $H_ω$, and ash-content of mined coal A on mining and geological and design parameters. They differ in the fact that they consider peculiarities of separate coal seam mining and undercut rock including seam weakening due to advance cut, mining plan as for benches, completeness of coal and rock haulage on conveyor etc.;

- Efficiency of a cutter-loader in the process of mining coal seams with enclosing roof and floor undercut should involve its dependence on mined coal grade.

Among other things, analysis of calculations using the dependences shows that:

- For the conditions of thin coal seam mining thickness should not be less than 0.90 – 0.95 m for KMK97 and KMK98 cutter-loaders; 1.02 – 1.05 m for KD80 powered system, and 0.87 – 0.90 m for KM103; mining seams having less thickness is impossible without worthless enclosing roof and floor;

- 0.01 m increase of undercut in terms of complete varying-thickness seam mining results in extra dilution of mined coal (0.04-1.2%); in terms of one-pass separate mining it is 0.1 – 0.2 %. Two-pass mining slightly effects on mined coal grade;

- Feeding velocity of a cutter-loader in the context of extraction having two outcropping flats slightly depends on its hardness and thickness (if $m_{пр} \leq 0.35$) reaching its maximums ($\bar{v} \leq \bar{v}_{доп}$).

Analytical dependences make rather accurate explanation of actual parameters of complete and separate seam mining with enclosing roof and floor undercut. Convergence of results is within ±10% of limiting accuracy to be quite acceptable for the calculations.

2. Methodic principles and appropriate solver using PC have been developed to identify rational application areas and capacities to use complete technique and separate technique in specific mine. They also can determine the most reasonable operation schedules for selective seam mining, utilization techniques for enclosing roof and floor to be undercut, and rational transportation procedures for mined coal depending on its grade. The model implementation helps determining the most expedient techniques to mine thin and very thin coal seams in Western Donbass and Lvov-Volyn coal field.

3. Principle schemes of selective seam mining with enclosing roof and floor undercut basing upon available winning technique have been developed. Several of them have been proven in mines of Western Donbass and Lvov-Volyn coal field. Among other things, 1КМ103 powered system was involved for selective mining seams with 0.6-0.8 m thickness and enclosing roof and floor undercut with up to f=4÷5 hardness according to M.M. Protodiakonov scale making it possible to decrease mined coal ash-content substantially. A technique of powered mining seams with 0.6 m thickness and soft enclosing roof and floor (f≤3) is innovative.

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Яворська Олена Олександрівна
Токар Лариса Олександрівна
Сулаєв Віктор Іванович
Сердюк Володимир Петрович

ТЕХНОЛОГІЯ СЕЛЕКТИВНОГО ВІДПРАЦЮВАННЯ ТОНКИХ ВУГІЛЬНИХ ПЛАСТІВ

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