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**OPERATION TECHNIQUE TO MINE  
THIN SEAMS WITH WORKED-OUT AREA STOWING**

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**Operation** technique to mine thin seams with worked-out area stowing:  
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The monograph concerns substantiation of basic parameters and application area of thin coal seam mining with the worked-out area stowing by means of undercut rocks. As a result of the studies, regularities of changes in stress-strain state of rock mass depending upon operational parameters of mining procedures while the worked-out area stowing. The opportunity to use rocks of Western Donbas as stowing material has been substantiated. Basic design features of winning-stowing systems have been determined; operation schedules to mine thin seams with stowing for mines in Western Donbas have been developed.

The monograph is meant for students, engineers and technicians, academics, scientific-research institutes, and project organizations of coal industry.

44 figures, 111 reference sources

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## INTRODUCTION

Recently environmental problem has become crucial. During decades negative ecological developments accumulated in Ukraine; they are very serious today.

In Ukraine, mining enterprises are powerful sources of environmental contamination delivering substantial environmental, social, and economic damage. Land, water and air suffer. Large volumes of waste rock as well as undercut of many land areas and surface structures as a result of underground coal mining are the key factors aggravating ecological situation. As a number of thin seams and very thin ones are mined with enclosing roof and floor undercut, certain share of waste rock is hoisted from mines resulting in coal grade degradation, and increase in expenses connected with rock mass preparation and conveying.

Selective seam mining with stowing is well-known but underresearched technique able to simplify the negative consequences. Moreover, if individually problems of selective mining and stowing are somewhat understood, the problem of a stope undercut rock stowing stays to be unsolved. That especially concerns mines in Western Donbass. Lack of adequate scientific justification for technological parameters and frameworks prevents from such environmental technology large-scale implementation for thin seams. Thus, the problems solution is very important for coal industry of Ukraine.

*Staff of the Department of Underground Mining of the State HEI "NMU" and employees of Western Donbass mines took part in field studies concerning underground mine pressure manifestations. We highly appreciate their contribution to the research. We express individual thanks for D.S. Malashkevich and V.V. Kovbasa, students of State HEI "NMU" for their help rendered in the monograph design.*

## **CHAPTER 1. STATE-OF-THE-ART. OBJECT OF THE RESEARCH.**

### **1.1. Progress of the research and its importance**

It is impossible to put right extremely serious ecological setting in coal-mining regions without solving the two basic problems: decrease in volumes of hoisted waste rock and decrease in land surface deformation as a result of mining.

Volume of waste rock mined increases annually exceeding 140 mln tons per annum [60, 99]. 1500 Donbass waste dumps occupying almost 30,000 hectares of land contain more than 2.6 bln tons of hoisted waste rock [60]. For 300 fire waste dumps in Donbass emission into the atmosphere of gaseous air pollutant is up to 500,000 tons [90].

Almost 36% of ground transport and underground one and more than 30% of transport operators are involved in waste rock hoisting [60].

In Ukrainian mine volumes of waste rock in dumps attain their maximum, and it is prohibited to allot new land areas for waste rock. In fifty mines underground transport and hoisting are bottlenecks due to considerable volumes of balance weight conveying; that prevents from mine capacity increasing [99]. Operating experience of Donbass deposits shows that adequate performance of mines is possible if only volumes of hoisting waste rock is not more than 40% of mined coal; however, today the ratio is 75% [63, 99]. In certain mines it far exceeds the value. For instance, in E.T.Abakumov mine it is 132%, in “Mushketovskaia” mine it is 143%, and in “Kirovskaja” mine it is 128% [19].

According to prognosis of experts, if cardinal steps are not taken then volumes of waste rock hoisted by Donbass mines will exceed volumes of coal mined [99]; hence, the problem of waste rock becomes a factor which will have determining influence on both mining and adequate performance of mines [19, 86].

Considerable part of waste rock is hoisted using coal conveying lines causing sizeable coal dilution, and, consequently, its sharp degradation in grade.

According to information by Donenergo, for adequate coal combustion Ukrainian power plants have to consume almost 0.5 tons of oil fuel per ton of high-ash antracite. 1% increase in coal ash-content results in 80 kkal decrease of coal energy value, and 107 kkal decrease in antracite energy value [30, 54]. In terms of 1% increase of antracite ash-content, efficiency of boilers at power plants experiences 0.2% decrease. 1% increase in coke ash-content results in its 2-3% consumption increase. Pig iron output experiences similar decrease. In the context of 1% increase in mined rock ash-content, expenses connected with its hoisting and territorial distribution surface experience RUB 9.5 mln increase (hereinafter prices of 1990 are indicated). In addition 220-230 mln kWh of electric power are consumed. Expenses connected with mined rock conveying are RUB 2.3 mln. Mined rock preparation at

coal-preparation plant costs RUB 7 mln including RUB 1 mln spent for fuel and electric energy [54].

Mining is basic reason for dilution. Average dilution in stopes is about 2%, and rock yield is 77.5% of mined rock total volume hoisted using coal lines.

Availability of false roofs and unstable roofs is the reason of such coal dilution; as a result, average increase is 6%, and enclosing rocks undercut is 3%. That is why rock yield-total rock hoisting ratio is 42.3% and 21.1% correspondingly [6]. Coal dilution in the process of in-seam workings construction and maintenance is 3.2%, and rock yield from development headings is 22.5%.

Maximum dilution fall on Western Donbass mines (30.3%), “Selidovugol” (22.7%), “Donetskugol” (21.2%), “Sverdlovantratsit” (18.9%), “Dobropolieugol” (18.0%) “Krasnoarmeiskugol” (15.3%), and “Ukrzapadugol” (15.3%). Maximum dilution by enclosing roof and floor also takes place in Western Donbass mines (16.9%) [30].

The fact that basically mines in Western Donbass exploit thin seams and very thin seams with unstable enclosing roof and floor is the reason for such considerable coal dilution. In the form of histogram Fig. 1.1 illustrates allocation of balance reserves in Western Donbass in terms of seam thickness. Analysis of Fig. 1.1 shows that almost 50 % of the reserves are concentrated in seams which thickness is less than 0.8 m. It is impossible to mine them without enclosing roof and floor undercut in the context of available equipment.

Fig. 1.2 demonstrates changes in average dynamic values of geologic thickness and mining height of seams in Western Donbass from 1976 to 1991. For the most part, coal seams are of simple structure; that is why difference between extracting seam thickness and useful thickness is waste rock undercut thickness in stopes. If in 1976 average rock undercut value in longwalls of Western Donbass was 6 cm (that is 6% of mined seam thickness) then in 1991 it increased up to 25 cm to be 22% of mined thickness. In certain longwalls of the Association undercut values reached 40-50 cm. The fact is the key reason of loss in cost-performance ratio of operation of mines. Volumes of mined rock extracted from longwalls with undercut as well as total quantity of such longwalls increase year on year (Fig. 1.3). More than 70% of total output fall into longwalls operating with enclosing roof and floor undercut.

Mined rock ash-content in terms of the Association was 42-46%; as for “Zapadno-Donbasskaia” mine, it was more than 50%. There operated stopes in which mined rock ash-content reached 60% (when sheet one was 8-15%).

Ash-content increase has an adverse effect on key figures of mine performance; first of all it concerns profitability. Reduction from mined rock is in direct ratio to ash-content increase. In practice, customers excluded production of one or two longwalls from every mine; as for the Association, output of one or two mines is excluded.

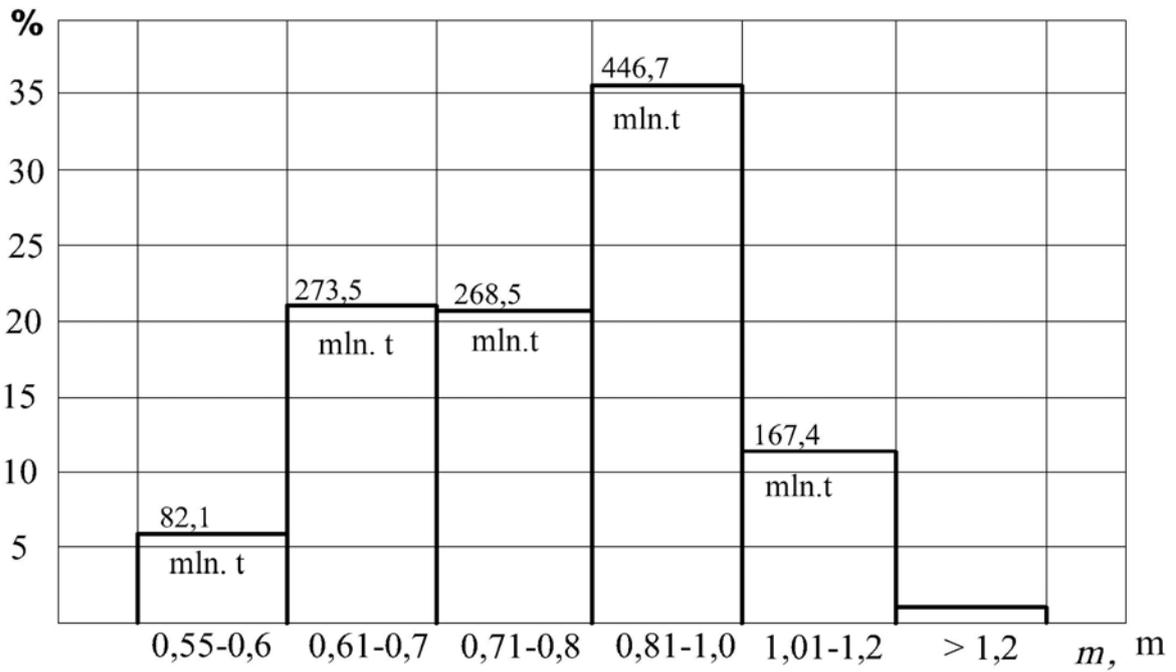


Fig 1.1 Allocation of reserves in Western Donbass mines in the context of seam thickness

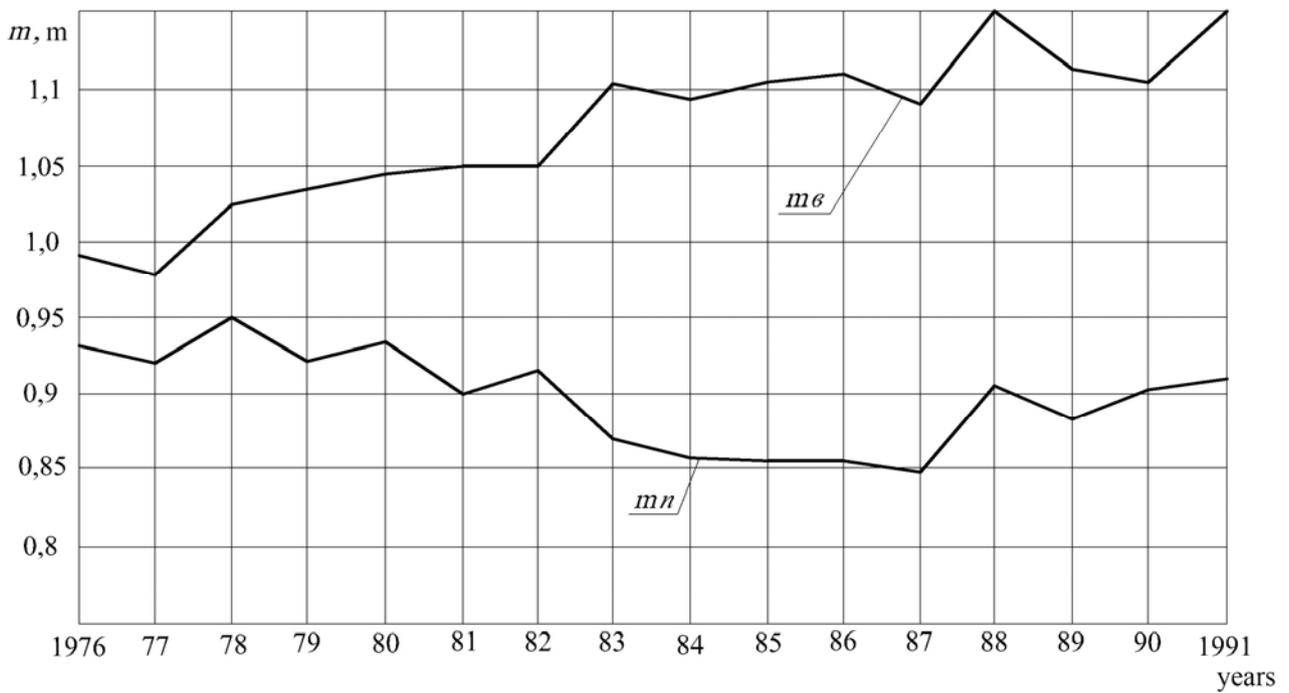


Fig. 1.2 Changes in average values of mined thickness ( $m_0$ ) and useful one ( $m_n$ ) from year to year

Increase in mined rock ash-content degrades mineral-dressing products and reduces their yield. Annually more than 5 mln tons of mined rock with average ash-content of 42-46% are delivered to “Pavlogradskaia” preparator. About 2.3 mln tons of concentrate with 11-11.5% ash-content is the outcome. Mined rock preparation is rather cost process being 60-65% of expenditures connected with its mining in stopes. Moreover, conveying of “refinement tailings” is quite cost plus procedure.

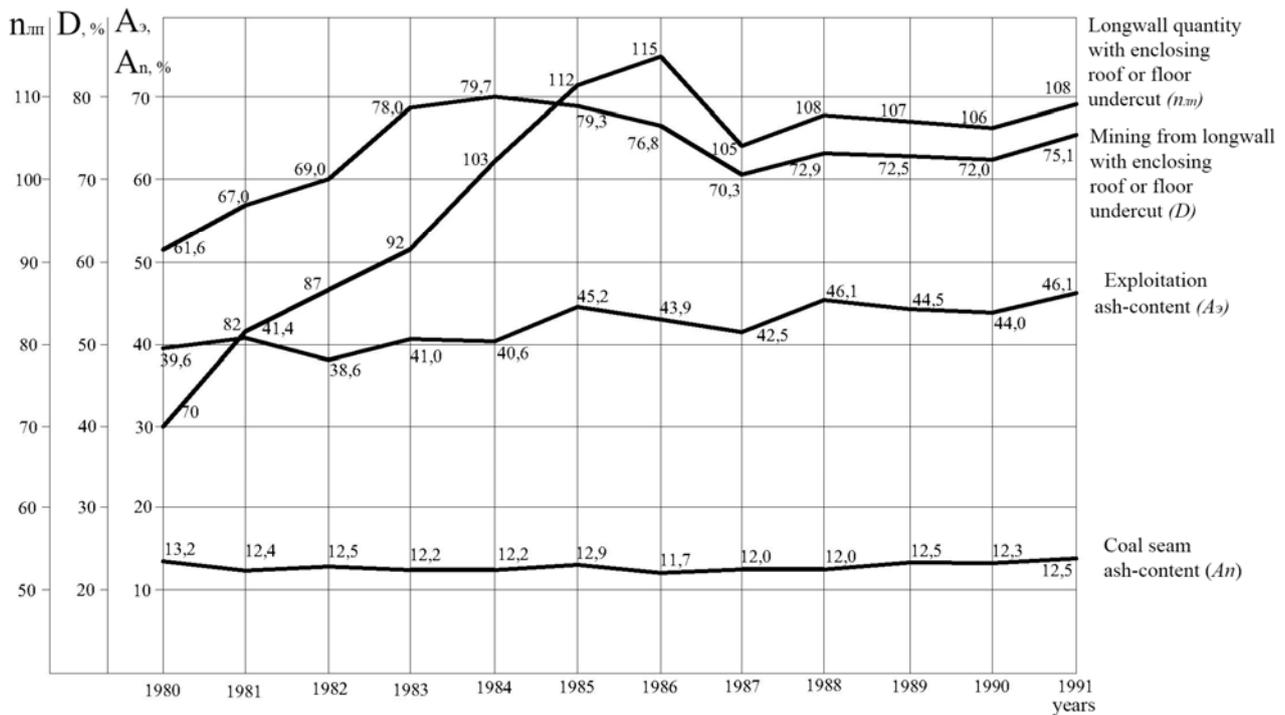
Not only constant decrease in geological thickness of working seams is the reason of increase in enclosing roof and floor undercut in stopes, and increase in mined coal operational ash-content. That also depends on the lack of efficient measures stimulating improvement in the quality of production. The latter factors into wide use of complete mining to be a technique providing simultaneous coal and rock extraction. It is even advantage for manufacturers to increase value of enclosing roof and floor undercut. The matter is that it can help obtaining increase in rock mass output with simultaneous creation of more favourable conditions for those who work in stopes.

As noted above, protection of natural objects, arable areas, and buildings and structures against undermining is integral part of measures on nature conservation. Donbass is located within a territory of Donetsk, Lugansk, and Dnipropetrovsk regions being densely populated and industrially developed regions. Hence less and less territories stay available today for mining without regard to surface deformation. 2.4 bln tons (27% of balance reserves) are located in pillars and areas of protected surface facilities [99]. Annually Donbass mines extract from under 40-50 mln tons of coal, and tens of thousands different-purpose objects experience undermining [82].

For example, mines in Donetsk operate under compact-planning townsite. Only during one year 5830 residential buildings, 295 industrial facilities, 30 km of railways, 85 km of major pipelines, and 18 water bodies were undermined [82].

A problem of land undermining is also severe for Western Donbass as the major part of the region land in plain floods of the Samara and Volchia rivers. The situation is dramatized by the fact that percarbonic strata are displayed as sandy-argillaceous differences being remarkable for its poor strength and low resistivity; hence, land settlings are 90-95 % of extracting seam thickness [57, 64].

Of total quantity of “Pavlogradugol” Association balance reserves 1.44 bln tons or 29% are located under plain flood; as for such mines as “Blagodatnaia”, “Pavlogradskaia”, “Ternovskaia”, and “Samarskaia” it is up to 80%. Seam mining under the Samara plain flood without specific measures will result in downstream effect for almost 11.8 thousand hectares of land of which 1.6 thousand hectares are woods, 2.3 thousand hectares are acres, and 1.4 thousand hectares are human settlements [82].



*Fig. 1.3 Dynamics of operation of longwalls with enclosing roof and floor undercut in Western Donbass mines*

Town of Ternovka and nine villages which population is over 15 thousand people are situated within zones of downstream effect and undermining of productive mines and constructed ones.

Over 5.3 thousand hectares of farmland have been disturbed and flooded; 677 buildings (647 of them are residence buildings) have been destroyed and cannot be corrected [82]; agriculture has experienced serious damage.

Board of Disturbed Soils Recultivation was established in 1974 in Pavlograd. RUB 32.6 mln were spent to recultivate 2923 hectares of land; 198 apartments were given [82]. Hoisted waste rock and tails (almost 21 mln cubic meters) were used to bank up, fill settling areas, back fill gullies, and perform other reclamation procedures.

Reclaimed territories are used in national economy. For example, 151 hectares render suitable for agriculture, 82 hectares render suitable for woods, and 682 hectares are used for commercial fish farming and collection of sewage. About 1.3 thousand hectares have been dried up and got back to agriculture.

In sum, RUB 45.6 mln were spent in Donbass from 1976 to 1992 for provisional measures aimed at protection of buildings and constructions, post-setting repair and rehabilitation operations, and payment of indemnities [82].

As it follows from above data, much money is spent for environmental protection measures to mitigate consequences of coal mines operation. However, no money can eliminate all negative effects as their majority is of irreversible character.

Wide use of mining technique providing either complete stowing or partial one (it should be combined with coal and rock selective mining in terms of thin seam extraction) is one of most practicable procedures to mitigate abovementioned

negative effects of mining enterprise operation, and cut expenditures connected with undermined land and structures recultivation.

Certain background for wide use of such technique has already been developed. 700 thousand tons of coal had been mined under central part of Donetsk using complete stowing technique; 4600 thousand tons had been mined under metallurgical plant, coking plant, and cement plant in Enakievo [99].

More than 400 scraper stowers of ZK-02 and ZK-03 type are widely used for flat seams [13, 24, 38]; they are applied for stowing to drive development workings.

“Titan-1” pneumostowing system is extensively used to stow and leave rock after development workings were driven. About fifty units operated [13, 99]; moreover, twenty-three of them worked in six mines of “Donetskugol” Association. Operation of “Titan-1” system helped leave almost 300,000 tons of rock [63]. As experts believe [4], scraper stowers and pneumostowing systems “Titan-1” will be in great request.

Small amounts of rock left after permanent mining opening construction was used for goaf pack building [13]. To this effect, A.G. Stakhanov mine (“Krasnoarmeiskugol” Association) and “Komsomolets Donbassa” mine (“Oktiabrugol” PJSC) applied interlinking PSS crushing and stowing system with DZM-2 pneumostowing machinery [13, 99].

M.Gorki mine (“Donetskugol” PJSC) applied a technique of complete stowing. The matter is that starting with 1970 in-site fixed crushing and stowing system (CSS) [13, 63]. For the system, stowing material was prepared according to one-stage schedule with the use of two ДО1 run in parallel crushers. Prepared stowing material was conveyed to stowing machine ZS -240 with following stowing [47]. Such stowing technique was combined with complete stowing of short longwall faces using “Titan-I” systems [13, 47].

According to data by [63], eight longwalls were mined, 932,000 tons of coal was extracted, and 682,000 tons of waste rock was left in mined-out space in 1988.

Hydraulic stowing system HSS was exploited in “Krasnyi Oktiabr” mine; the system performed stowing with the help of waste rock prepared for that at the surface [13].

However, despite certain practice stowing is rarely used in Ukrainian mines. Annually only 8,000,000 to 10,000,000 tons of waste rock was left in mines to be extremely inadequate for successful performance of mine enterprises [13].

Following things were the key reasons why the mining technique with stowing or waste rock leaving in worked-out area applied rarely:

- Lack of series-produced powered longwall sets of equipment to work with stowing [13, 43, 63, 104]; it notably concerned thin seam mining;
- Problems with waste rock transportation within mine workings due to their poor conditions [13];
- Either absence of reduced output of compression facilities in mines [13];
- Insufficient amount of research and elaboration of a design concept concerning technologies and equipment for stowing etc. [13, 63].

Selective extraction of coal and undercut rocks is applied by certain mines for thin seams. For example, “Dobropolskaia” mine (“Dobropolieugol” Association) applied such a technique to mine  $m_5$  seam, and in “Belitskaia” mine where  $l_8$  seam with 0.63 m thickness was mined using selective technique [30]. A number of other Ukrainian mines individually applied selective technique. Lack of transportation facilities as well as striving of manufacturers to execute a plan at any price blocks large-scale implementation of such a technique. In recent years more and more attention is paid to the improvement of cost-performance ratios rather than to gaining in yield; thus, interest in selective mining technique will grow.

Solving the problem of undercut rock leaving in working-out areas may considerably contribute to it; however, only research work and experiments are available up to date [26, 32, 44].

A number of mines in Western Donbass (for instance, “Zapadno-Donbasskaia”, “Dneprovskaia”, “Blagodatnaia”) used selective coal mining technique with undercut hoisting for thin seams and very thin ones [37, 108]; nevertheless, activities connected with stowing or rock leaving in working-out areas were not performed regardless of their acute need.

Hence, development of low-waste and environmentally-friendly technology providing high technical-and-economic factors and quality ones is of current interest for the majority of mines in Western Donbass; it particularly concerns mines in west region of Donbass. The development of such a technology should follow the substantiation of its parameters as well its efficient application in the context of Western Donbass mines.

## **1.2. Analysis of papers concerning thin seams mining with worked-out area stowing**

Many papers concern problems of thin and very thin seams mining [32, 37, 62, 64, 87]. Several of them propose traditional techniques for such seams mining with the use of powered systems and machinery where shearers [32, 37, 44], plows [50], and scrapers boxes and conveyor plows [76, 77, 87] are applied as mining assemblies. Other papers propose mining of thin seams (and especially very thin ones) with the use of nontraditional recovery methods [77] including those ones without continuous presence of personnel in stopes [87, 64].

However, results of application of nontraditional recovery methods cannot hold out hope for their manufacturing application in the near future [61, 71]; besides, many scientists and researches believe that in the foreseeable future powered mining will be applied for the majority of thin seams and very thin ones [4, 71].

In spite of the fact that plowing can be effectively used for thin seams and very thin ones, it has restricted application area [50] (about 21% of stopes in Ukrainian mines); meanwhile, the technique is unsuitable for the majority of seams in Western Donbass as they are of high strength.

Thus, taking cognizance of the above, it is required to be geared to application of powered mining while selecting technology for extraction of seams in mines of Western Donbass. However, as it was noted above, in the majority of cases, thin

seam and very thin seam mining is followed by forced undercut of enclosing roof and floor.

To solve the problems and therefore to reduce coal dilution many researches speak for necessity to design powered systems fitting in the thickness of seam under mining [78, 96]. Such viewpoint is correct but current level of development of mining facilities, automatic equipment and proficiency of attending personnel one can hardly rely on early progress.

Taking this into account, the majority of scientists and experts consider necessary further progress of selective seam mining technique [13, 27, 30, 32, 37, 44].

Idea of selective seam mining is the most popular for complex-structure seams bearing dirt beds. A number of mines from “Karagandaugol” Association [26] applied adown successive mining of coal patches and waste rock with the help of standard coal shearers. Shaly deposit development involved one-pass shaly mining and interstratified rock mining in the course of coal shearer return [68].

In early 1970s there were attempts to design specific equipment for complex-structure seams selective mining; the machines had to perform simultaneous mining of coal patches and waste rock [33]. However, production tests were not successful; as a result, they are not currently applied.

There are also several alternatives of selective mining technique for thin coal seams being mined with enclosing roof and floor undercut.

For example, staff members of KRCI [26] have developed techniques to mine thin seams with false roof or soft floor with the help of powered systems KMK-97Д and 1MKM and coal shearer 1K101. The procedure involved two-pass mining. Pass one consists of roof rock breaking; pass two – coal seam breaking. Roof rock mining depends on capacity of either standard screws of coal shearers or undersized ones. In the case of floor undercut, coal seam is mined first.

In the majority of cases, selective mining technique for thin seams and very thin ones provides floor undercut [27, 30, 37, 41, 44, 45]; roof undercut is uncommon [32, 37]. As a rule, seam mining involves two passes of coal shearer.

Papers [37, 41] include a number of schemes for selective seam mining and substantiate thoroughly basic parameters of coal and rock selective extraction. The papers conclude that standard mining and hoisting mechanisms with upgraded separate units can be used for selective mining; undercut rock cuttability should not be more than 450 N/mm [41]. For extra hoisting it is recommended for coal mining direction to coincide with travel direction of haulage chain of a conveyor [41]; as for rock mining– in the course of executive device rotation towards stripped area.

Doctoral dissertation by A.G. Koshka in full deals with problems of selective seam mining in Western Donbass [37]. The publication determines dependences of mining machine feed velocity while rock extracting, coefficient of machine time of longwall, face productivity in terms of selective seam mining with undercut rock hoisting. It has been concluded that use of twin-screw shearers of 1K103 type may give the best results. Reasonable application area for selective seam mining in Western Donbass has been theoretically substantiated.

We believe that under the conditions of stowing or leaving undercut rocks in worked-out area, expressions proposed in [37] paper should be corrected. More specifically, it is required to take into consideration loading capability of mining machine, interaction between processes of mining and stowing as well as other features of the technique.

The majority of proposed alternatives of selective mining [30, 41, 45, 68] involves rock hoisting. Just a few abovementioned papers [26, 37] as well as [11, 32, 44] proposed to leave undercut rock in worked-out area.

IM named after A.A. Skochinski proposed thin seam mining technique with rock left in worked-out area [105]. If it is applied coal and rock breaking is performed with the help of spaced executive devices with simultaneous separate conveying along coal arm and rock one being spaced horizontally. Coal is conveyed to transport working and rock is conveyed to stowing machine; the next step is its stowing into worked-out area of a longwall. Advantage of the idea is possibility to perform one-pass selective mining of coal and rock that is output per stope stays unchangeable. However, implementation of the method involves a conveyor with horizontally spaced arms but it is practically impossible to put it together with powered support. Moreover, extra difficulties emerge connected with coal shearer control through conveyor flight [37].

Idea by “Dongiprouglesh” [32] is also based on selective but simultaneous mining of coal and rock. However, broken rock is loaded when coal shearer is back to start position having both its screws down. It is proposed to stow undercut rock into worked-out area of double longwall. Pneumatic stower, undismountable stowing pneumatic pipeline with remotely controlled facilities for side discharge of material, and special two-cantilever hydroficated support should be used for it.

DonCI achieves great success in terms of development of selective mining technique when rock is left in a mine [18]. The Institute has developed and tested operation schedule with the use of advanced KMK97Д system and standard stowing equipment. F.P. Liutikov mine (“Krasnodonugol” PJSC) and a mine No. 21 (“Sovetskugol” PJSC) tested one of the schedules [44]. Coal shearer 1K101, scraping longwall conveyor СИ202, powered support МК97, rock breaker ДО (to break and stow undercut rock), chain loader СИМ46, belt loader ПЛ4,5, and pneumatic drum-type machine ZP 200 were applied to mechanize mining operations. Pilot stowing pipeline with mobile powered wall was also used. The pipeline sections were remotely jointed with the help of hydraulic clamps.

DonCI experts [44] believe that tests of the technique were positive; however, poor loading capacity of a coal shearer and, consequently, considerable coal losses as well as much dust-rich air in a stope are among its disadvantages.

We suppose that despite positive results, several problems stayed underresearched. Mutual effect of mining and stowing; changes in power parameters of support when its section canopy area increase and when stowing mass is available; parameters of the latter when undercut rocks of the stope are used as basic material for stowing, and a number of other problems are among them.

Moreover, equipment used to implement the technique may be applied in those longwalls where strength of enclosing rocks is at least average one; hence, application of such technique in the context of unstable enclosing rocks of Western Donbass is rather problematic. Thus, effective stowing operations require designing of mining and stowing facilities on the basis of other systems capable of operating under the conditions of the coal-mining region.

Above technological concepts concerning rock left in worked-out area are based on rock air conveying to the place where stowing mass will be constructed. Besides, a number of solutions based upon application of mechanical facilities to leave rock are available. Thus, engineering solutions by Dnipropetrovsk Mining Institute (the National Mining University today) [53] were to apply special scraper conveyor with unloading apertures fixed within extended goaf overhang of powered support section. KRCI experts proposed to place rock left after coal selective mining in worked-out area using screw device located along a longwall from a goaf side of powered support [2].

Despite certain advantages (i.e. simplicity and low level of dusting) those solutions will hardly be commercialized in the years coming. In addition, they keep from the developing of tight stowing mass throughout the length of worked-out area; they only can be used to leave some few of rock in a mine. But we should find universal solutions to be used both for small amount of rock leaving in a worked-out area and for complete stowing.

Upwards of mid-1970s DonCI [44, 66], A.A. Skochinski IM [69, 92], KRCI [35, 65, 67], and Dongiprouglesh [43, 58] have been engaged in the designing of mining and stowing systems for seams having various thicknesses and dips.

DonCI developed operation schedules of pneumatic stowing [66] based on SS “Donbass” system application using features of standard “Donbass” complex; its application area has been substantiated. It is provided to perform stowing by means of integrated stowing pipeline when rock is discharged sideways; the pipeline is mounted to reverse overhangs of powered support. Besides, the activity involves large-scale underground investigations as for mine pressure manifestation in one of longwalls of Gorki mine (“Donetskugol” Association). Among other things, values and nature of enclosing roof and floor displacement within stowing mass, working place of longwall and immediate mine workings have been determined; earth’s surface deformation values have been defined. Such a comprehensive approach to studying of technique parameters can be used to analyze parameters of the proposed technique.

Unfortunately, it turned out to be impossible to test “Donbass” systems due to a number of design defects and organizational reasons; hence, further activities connected with the system development were interrupted.

Of all works by A.A. Skochinski IM and KRCI those related to the design of mining and stowing systems for flat medium-thickness seams with stowing on the basis of KM87 and KM88 systems warrant great attention [35, 65, 67, 69]. Available mining-and-stowing systems provide either hydraulic stowing [67, 69] or pneumatic

stowing [35, 65] of worked-out area. Technique using the systems was tested in Karaganda coal basin.

Subsequently, in 1984 there was developed a project of pneumatic stowing powered system PNSS [35] which included powered support M87YMF, coal shearer 1ГШ68, face conveyor ЦП87П-31, cable layer ЦТ4, electrical equipment, face-end supports KC1MY, and stowing devices.

Powered support is designed on the basis of standard M88YM support; contrastingly it was equipped with elastically mounted back overhang under which preventive panel and stowing pipeline with rock sublevel caving were located.

A design of mining and stowing system is well thought-out having a number of advantages; moreover, several its features we will use in our future work.

Activities by Dongiprouglesh concern the design of mining-and-stowing system for thin seams [43, 58]. It is proposed to develop the system basing upon КД-80 system, and subsequently – using МКД-90 system. Availability of back overhangs of powered support units, modified lemniscate mechanism, and stowing pipeline with filling material sublevel caving are distinctive features of the mining-and-stowing system. The system can mine seams which thickness is 0.95 to 1.4 m. Functional test of experimental units with pipeline performed in “Kommunarskaia” mine (“Shakhtiorskugol” Association) was positive.

КД-80 systems have been and stay in successful operation in mines of Western Donbass. Hence, it can be expected that mining-and-stowing system being designed using it may achieve high technical and economic parameters. However, variation one of the mining system involved application of KA-80 coal shearer having drum-type working members; that makes it impossible to implement selective mining for seams which thickness is less than 0.95-1 m without enclosing roof and floor undercut. Nothing but its operation as part of mining and stowing system of winning machine with screw effectors enabled to provide separation of rock and coal while mining which makes it possible to obtain rock required for stowing.

Monograph by Yu.A. Korovkin is devoted to the problems of substantiation of structural parameters of powered supports including stowing ones [36]. The author takes up the position that if longwalls require pneumostowing then powered support should perform extra support for roof above stowing line by two cutting depths of winning machine. Therefore, length of goaf overhangs should be 1.2 to 1.6 m; it should be 1.8 to 2.6 m if stowing pipeline is required. Yu.A. Korovkin believes that when possible second row of props in a support unit should be closer to stowing mass with possible shortening of canopy; it is expedient for goaf overhang to have extra support. It is reasonable to consider working resistance of second row of props as that being larger to compare with the first one; if strength is similar in terms of rows of props then power parameters of stowing powered supports should be excessive.

The monograph notes that if complete stowing of worked-out area takes place then in the context of controllability, roof grade experiences one order decrease; i.e. working strength of stowing powered support operating under the conditions of “tight” roofs should be considered at the level of semi-controlled grade while dealing with caving. If roof grade is estimated as semi-controlled, then basic power

parameters of stowing powered support should be specified as those being no lower than requirements for easy-controlled grade.

The latter cannot involve seams where unstable rocks form immediate roof and main roof. Thus, substantiation of parameters of stowing powered supports for such environment requires a number of studies to determine regularities for rock mass stress-and-strain state changes depending upon operational parameters in the process of working-out area stowing in terms of Western Donbass mines.

In the eyes of Yu.A. Korovkin [36] use of stowing support of КДЗ-90 type (with 380-420 kN/m<sup>2</sup> strength) is expedient for thin seams which thickness is up to 1.5 m.

World practice of selective seam mining with working-out area stowing is of some interest. Such a technique was applied by “Park Mill” mine (Great Britain) while mining composite seam [18, 44, 108]. Selective mining was performed using two-pass operations. Pass one involved simultaneous mining of lower and upper coal benches; then during return pass involved mining of rock interlayer with 0.8 to 0.9 m thickness using forward screw. Upon that rear screw was scraping the seam floor. Coal shearer loaded rock of the interlayer on a longwall chain-and-flight conveyor; following operation was reloading to gate belt within haulage gate. Belt conveyor hauled the rock to a conveyor located in a cross-hole and then to a crusher for its reducing into less than 50 mm fragments. After crushing has been performed, rock was fed to a conveyor mounted on vent drift; then through a system of short conveyors it was fed into pneumostowing plant “Bayen” (Germany) with further stowing into worked-out area of longwall.

Application of such a technique results in high performance indicators of a stope. Efficiency of the longwall was 1100 to 1200 tons of coal per day and almost 85% of rock being a component of interlayer was remained in worked-out area.

Other countries of the world apply a technique with worked-out area stowing only when seams with more than 1.6m thickness are being mined [18, 25, 97, 102]; small number of seams with minor thickness are being mined using complete caving of roof rocks.

Certain research concerning the development of environmental protection techniques for worked-out area stowing in the context of Western Donbass conditions should be distinguished. Such Ukrainian scientists and researchers as V.V. Vystorop, V.E. Zhukov, A.V. Zaria, I.A. Kiiashko, O.V. Kolokolov, N.V. Kuznetsov, I.G. Lisitsa, Yu.S. Makarevich, N.V. Mishchenko, I.A. Sadovenko, K.F. Sapitski, V.I. Stytsin, Yu.M. Shenderovich and others contributed much to the problem solution. The majority of scientists and researchers believe that such techniques should be implemented [20, 27, 40, 44, 57, 64, 77].

As far back as 1970s Mine Surveying Department of DMI carried out a number of research to determine areas of potential flooding when coal seams are mined in the bottom of the Samara River with complete roof caving and worked-out area stowing. In this regard it was assumed that stowing took 50% of surface settling of mined seams thickness. The calculations helped to conclude that stowing of worked-out area in such mines as “Samarskaia”, “Pavlogradskaia”, “Blagodatnaia” and “Ternovskaia”

may reduce flooding area by 3174 hectares. That is almost 64% of land within the bottom of the Samara River will be preserved [57, 64].

Problem of stowing is the most acute for “Blagodatnaia” and “Samarskaia” mines where its effect is the greatest [20, 64].

In this connection different alternatives of environmental protection techniques aimed at surface preservation were proposed [20, 64, 77].

Experts of DPI proposed shortwall technique for coal seam mining [77]. Its authors believe that if width of interchamber pillars is equal to width of chambers, it is possible to avoid surface displacements. However, experts of DonCI [20, 44] and DMI [64] showed that finally rock displacements will result in the displacement of overlaying rock mass. In this context, displacement nature will be similar to that one as in terms of complete caving in longwalls. If interchamber pillars of specified geometry are left then rock displacement experiences 50% reduce, and a process of rock deformation elongates [20, 44]. The idea was proved in practice during test coal mining using chambers in 1974 under the conditions of “Blagodatnaia” mine [64].

Decrease in surface deformation while using shortwall technique is possible at the expense of worked-out area chamber stowing [20, 44]. However, high-performance mining and stowing equipment to apply the technique is not available today. Hence, it is unlikely for such a technique to become widely used in the near future.

Bearing this in mind, experts of DMI headed by Professor O.V. Kolokolov proposed to use augering combined either with pneumatic stowing or with drilled-out cavities stowing for seam extraction in mines of the Samara River basin [64]. The idea is also based on the author’s statement that construction of stowing mass with side caving of stowing material generally applied in combination with powered supports is unacceptable for Western Donbass. The matter is that weak roof rocks cave right after support unit has been shifted. Thus, they infill worked-out area closing out to feed stowing material. The fact also presents a problem for sublevel caving application which traditional use will require major change [64].

Such staff members of DonCI as V.E. Zhukov and V.V. Vystorop [20, 44] did not support last-mentioned opinion; on the contrary, they preferred a scheme involving powered support and pipeline for stowing material front caving. It is beyond doubt that application of augering with stowing may result in the least settlement of ground. Moreover, such a technique is simple and less labour intensive; it also has high cost-performance ratios of coal mining. However, augering application area is limited by rather weak seams with their occurrence quiet hypsometry. We suppose that under the conditions of Western Donbass the technique is only expedient to mine non-commercial reserves or drill out remained coal pillars.

All specialists engaged in problems of stowing in Western Donbass prefer pneumostowing [20, 40, 44, 64]; however, they suggest that in their pure state rocks which swell on wetting fast as well as those inclined to adhesion rocks of the region will form substandard stowing mass making difficulties for their pneumatic

conveying [64]. The same is also true for fine sand and close sand available in Western Donbass. The matter is that due to high content of clay particles in their pure state they are unusable as stowing material [20, 44].

In this connection, two-component burden consisting of 75-80% of crushed mine rock and 20-25% of sand are recommended for stowing in Western Donbass mines [20, 40, 44, 64]. In this case stowing mass compression will not be more than 30%.

Papers [44, 64] state that Western Donbass mines surface almost 40% of mined coal as a result of working construction and maintenance. In the context of “Samarskaia” and “Blagodatkinaia” mines figures are 44% and 3% respectively [64].

Practice of worked-out area stowing shows that each ton of mined coal requires a ton of stowing material. In this connection, paper [64] mentions that mine rock is not sufficient for complete stowing of worked-out area; other sources are needful to deliver rock. It also states that shafts in mines of the Association are unappropriated to move stowing material down having limited possibilities to equip them with extra facilities [64]. That prevents them from providing complete stowing within pre-bottomland. In this context the fact that rather substantial amount of rock as a result of enclosing roof and floor undercut is surfaced is not taken into account. However, its average quantity in the context of Western Donbass mines is almost 17% [30] of total output.

Once more the fact confirms necessity to apply selective seam mining technique within the region. The matter is that in such a case, mine rock volume will be quite sufficient for worked-out area stowing under protected objects.

We believe, that the necessity to apply two-component burden (mine rock and sand) for stowing is understandable. However, complexity of such stowing materials application is in limited capacities of pit shafts to run sand. Thus, only mine rock should be used at stowing stage one. To do that it is required to test its capabilities as well as parameters of stowing mass erected using mine rock.

Short analysis of works and literary sources may help conclude that the problem of environmentally friendly and low-waste technique development for Western Donbass is unsolved; however, it should be noted that certain activities were carried out. Despite selective coal and rock mining technique has been deeply analyzed and its efficiency for Western Donbass has been proved, it is not widely use because:

- Neither technique nor means to stow undercut rocks of a stope into worked-out area are available;
- Infrastructure of stowing activities is not available;
- Parameters of mining and stowing technique are of poor substantiation;
- Under-search into a matter of Western-Donbass mine rocks use as stowing material; this is especially true for the case of their producing in a stope.

Solution of the problems creates prerequisites for large-scale implementation of techniques which improve environmental friendliness of mining with similar coal grade improvement.

### **1.3. Objective of the research and its techniques**

Analysis of considered works help determine principal line of the research – scientific and technical basis for major parameters of technique to mine thin flat seams with worked-out area stowing as well as using them to develop structural schemes of the technique and requirements for mining and stowing equipment.

The research objective is to determine rules and dependences required to substantiate basic parameters for thin seams mining with worked-out area stowing.

To accomplish the objective following problems haven raised and solved:

- To determine rules of changes in rock mass stress and strain state depending upon technological parameters of mining in terms of worked-out area stowing;
- To form scientific and technical basis of principal parameters to mine seams with worked-out area stowing;
- To substantiate possibility of Western-Donbass mine rocks use as stowing material;
- To identify basic structural features of mining and stowing equipment and develop operation schedules to mine thin flat seams with stowing for mines in Western Donbass.

Analytical, full-scale, and laboratory research as well as computer-based mathematical simulation have been used to solve the problems.

## **CHAPTER 2. ANALYTICAL RESEARCH OF PARAMETERS OF TECHNIQUE TO MINE THIN FLAT SEAMS WITH WORKED-OUT AREA STOWING**

### **2.1. General remarks**

The chapter substantiates basic parameters of technique to mine seams in Western Donbass with worked-out area stowing. Covered parameters are nominally divided into three groups.

Group one covers parameters of geomechanical processes within coal-overlying mass. They include: values of standard loads on coal-overlying mass seams; geometry of loading diagrams in the process of worked-out area stowing; convergence and deformations of rocks both in a longwall and stowing mass; load taking up by stowing mass; and lowering movement of surface located over it. The parameters, determined in the process of stowing were compared with analogous one when roof is controlled by means of complete caving; that helped determine differences and efficiency of proposed technique in comparison with technique being traditional for Western Donbass.

Group two involves geometry of the technique. It includes: length of a stope, width of a pack in terms of partial stowing of worked-out area with rock remained in it as a result of separate mining or as a result of construction of workings adjoining to a stope.

Operating parameters of the technique united in group three involve: feed velocity of stoping machine combined with rock stowing and machine time coefficient of a longwall.

The parameters have been analyzed analytically. Their substantiation should be validated in the process of the technique underground analysis.

### **2.2. Analytical studies of the technique parameters according to rock pressure factor**

#### **2.2.1. Subject for analytical studies**

Various reasons may stipulate the necessity to apply mining technique with worked-out area stowing in mines of Western Donbass. The necessity to remain rocks in a mine; mining of seams with tight roofs ( $C_5$  seam in “Blagodatnaia” mine) or mining of superimposed seams ( $C_8^B$  and  $C_8^H$  seams in “Zapadno-Donbasskaia”, Stashkov and other mines) and, what is more important, mining of seams under protected objects are among them.

Currently such mines as “Samarskaia”, “Blagodatnaia”, “Ternovskaia”, “Stepnaia” and “Zapadno-Donbasskaia” are engaged in mining of seams located under the Samara River bottom. Problems of undercut are the most severe for such mines as “Samarskaia”, “Blagodatnaia” and “Zapadno-Donbasskaia” where the major part of reserves is either under the Samara River bottom or under built-up territories. Moreover, as it was mentioned, technique with worked-out area makes it possible to

conserve the major share (up to 70-90%) of underworked land; hence, the result of stowing use will be the greatest in the mines.

Mining and geological conditions of “Blagodatnaia” and “Samarskaia” mines are generally identical; that’s why analysis of recommended technique parameters in terms of rock pressure factor carried out for one of the mines can be used for another one. “Zapadno-Donbasskaia” mine has great mining depth and somewhat different mining and geological conditions. Hence, it is reasonable to determine stress and strain state of percarbonic mass rocks in terms of worked-out area for such two mines as “Blagodatnaia” and “Zapadno-Donbasskaia”.

Studies carried out for the two mines will be useful for the whole Western Donbass as they involve the major part of conditions of seams being under mining in the region.

Seams for which stowing is the most promising idea and which major share under protected objects has not been mined yet were selected for the studies. The seams are:  $C_7^H$  in “Blagodatnaia” mine and  $C_8^H$  in “Zapadno-Donbasskaia” mine.

### **2.2.2. Technique selection to calculate stress and strain state for rock mass**

A number of prognosis methods as for rock mass state in terms of mining are available. Possibility to solve each involved problem with maximum precision is that fundamental requirement taken into consideration in the process of a technique selection.

A good deal of operations performed in the area may be nominally divided into three basic groups. Group one represents rock mass as continuum where deformations take place without any discontinuity. Group two considers rock mass as layered medium represented with the help of overhanging beams or platforms. Works by group three are connected with research of rules for stress and deformation distribution within index zone; they are based on experimental data. The latter approaches can be either used only under certain conditions or involve labourious experiments for each case. Consequently, they are not popular and the study will not consider them.

Group one considers rock mass as a unit from a roof of a seam under mining up to surface. Rock mass is represented as continuum offering plastic [1, 5], elastic [48], or elastic-plastic [14, 15] properties.

Certain works based upon continuum mechanics methods [15, 42, 49] give analytical explanation for stress and strain state of rock mass in terms of worked out area stowing. Work [72] considers interaction between enclosing rocks, stowing mass and coal seam taking into account its out-of-limit deformation. Mathematical tools of theory of elasticity and plasticity is rather complicated and labourious; however, its results are only approximate as sedimentary rocks are of heterogeneous properties [9, 80]. Moreover, L.G. Fisenko [95] believes that representation of rock mass in the form of continuum makes it impossible to involve such fundamental mining and geological factors as availability of rock bridges, their thickness, height location etc.

In recent years, “Finite element method” [2, 34, 55] applied to calculate stress and strain state of rocks has become popular. It is based upon continuum approximation by means of discrete elements being of arbitrary shape which interact through groups. The method has a number of advantages. It provides possibility to define nonuniformity of mass and regulate calculation accuracy by means of element network refinement; it provides freedom in defining of area configuration as well as boundary conditions etc.

It is known that rock deformation within bending area is followed by disintegration between rock seams and formation of cavities [22, 72] which parameters depend on a number of geological and mining conditions. The disintegration takes place both over worked-out area and rock mass. That results in random load distribution within percarbonic mass. Thus accurate calculation with the help of finite element method should involve dimensions of disintegration cavities as well as loading function across the width of mine working.

Group two includes research of scientists who consider percarbonic mass as discrete layered medium. Works by V.D. Slesarev [80] are worth noting as he proposed to use approximate approach based upon platform theory; works by G.N. Kuznetsov [39] whose calculation methods rely on hypothesis of “articulating blocks”; works by A.A. Borisov [7] who determined a number of regulations for nature of main roof rock deformation basing on his own 3D modeling. Using experimental data and analysis of thin platforms theory, A.A. Borisov proved allowability of thin platforms theory to solve geomechanical problems and developed calculation approach being a model of overhanging beam resting on elastic foundation. The approach means parting of undermined rock mass into formation members basing on rock bridges and having clamping load within bindings.

Works by AURSI experts [7,9] are also worth mentioning. They consider rock mass as layered medium with deformation continuity and creeping on stratification contacts. Process of percarbonic mass strata movement takes place in the form of systematic platform bending; if a stope is sizeable, then the process takes place in the form of beams. It is assumed in this context that vertical subsidence of each seam is equal.

Works by F.F. Shalamberidze [100] concern the development of calculation technique for rock pressure and load on powered support in terms of worked-out area stowing. Using components of articulating blocks hypothesis, the author states that support in a stope should props up only strata immediate roof divided into individual blocks between which horizontal thrust is available and main roof rests on virgin coal. Basing upon the assumption, the author has developed expressions to determine enclosing roof and floor convergency and support load. However, the author confines application area for deep mining of formation subject to numerous underworking or overworking.

Thus, representatives of the group consider percarbonic mass as layered medium which deformation takes place with discontinuity on stratification contact. Paper [95] notes that such model of medium helps involve following key mining and geological factors: structural features of rock occurrence, their mechanical properties, effect of thick rock bridges on a process of percarbonic rock mass.

However, calculation approached described by the majority of the papers help solve only individual problems; for example, it concerns main roof fault pitch,

enclosing roof and floor convergency, and powered support load. They prevent from solving all geomechanical problems and ignore certain mining and technological factors.

Each geomechanical parameter which should be substantiated by this work can be determined using a technique developed by professor A.V. Savostianov from the National Mining University. The technique takes into account effect of geological, mining, and production factors on rock mass state while mining which facilitates solving a number of technological problems even those when roof control methods vary.

A number of papers [73, 74, 75] give details of the calculation technique. We only mention that in the context of the technique percarbonic mass is represented as a system of thin platforms mildly pinched on a stope boundary under redistributed load resulting from underworked rock mass.

To analyze rock state in the process of mining, percarbonic mass is divided into seams according to rock types. First, geometry of basic load distribution involving depth of a seam, its thickness, rock type, face advance, wall length and mining completion in the context of section under consideration, extracting seam thickness, and roof control technique are determined. Next, in terms of specified wall length a rule of basic loads distribution as well as their values within characteristic points at the level of any specified bed is determined.

Changes in a stope advance are taken into account using dependences of changes in module of rock deformation over worked-out area  $E_{\pi}$  and the system stiffness coefficient  $\beta$  on time obtained on the basis of theoretical research and experiments.

In terms of known parameters of load diagram and specified coefficients characterizing a rule of the load diagrams distribution, subsidence and deformations of bed are calculated. Calculations to determine rules of basic loads distribution and calculation of parameters characterizing rock state are performed using PC according to operation programs developed by the NMU.

### **2.2.3. Results of analytical research**

To make analytical research of rock mass stress and strain state, conditions of 736<sup>th</sup>-bis longwall of  $C_7^H$  seam (“Blagodatnaia” mine) and 837<sup>th</sup> longwall of  $C_8^H$  seam (“Zapadno-Donbasskaia” mine) were accepted. Stratigraphic sections of percarbonic mass under specified conditions were initial calculation data (Figures 2.1, 2.2).

According to [8, 73] definitions, mass of free caving rocks being overhang is immediate seam roof. From the overhang load is taken by mass and support, and immediate roof takes load of tight rocks having force action on mass and caved rocks or stowing.

Following this definition and taking into consideration the fact that in the context of Western Donbass, rock mass of 3-4 m thickness caves after powered support haulage, layer of 3m thickness argillite is considered as a part of immediate roof controlled by complete caving (for “Blagodatnaia” mine) (layer 1 in Fig.2.1). As for “Zapadno-Donbasskaia” mine, it is layer of 4 m thickness argillite occurring between  $C_8^H$  and  $C_8^B$  seams (layer 1 in Fig.2.2).

Upper layer is considered as a part of main roof.

Generally, when worked-out area is under stowing, roof rocks fall on stowing mass without caving. Hence, first rock layer is integrated into main roof. Besides, it is common practice that in terms of worked-out area stowing immediate roof is not available.

Calculations have been performed for various techniques of roof control.

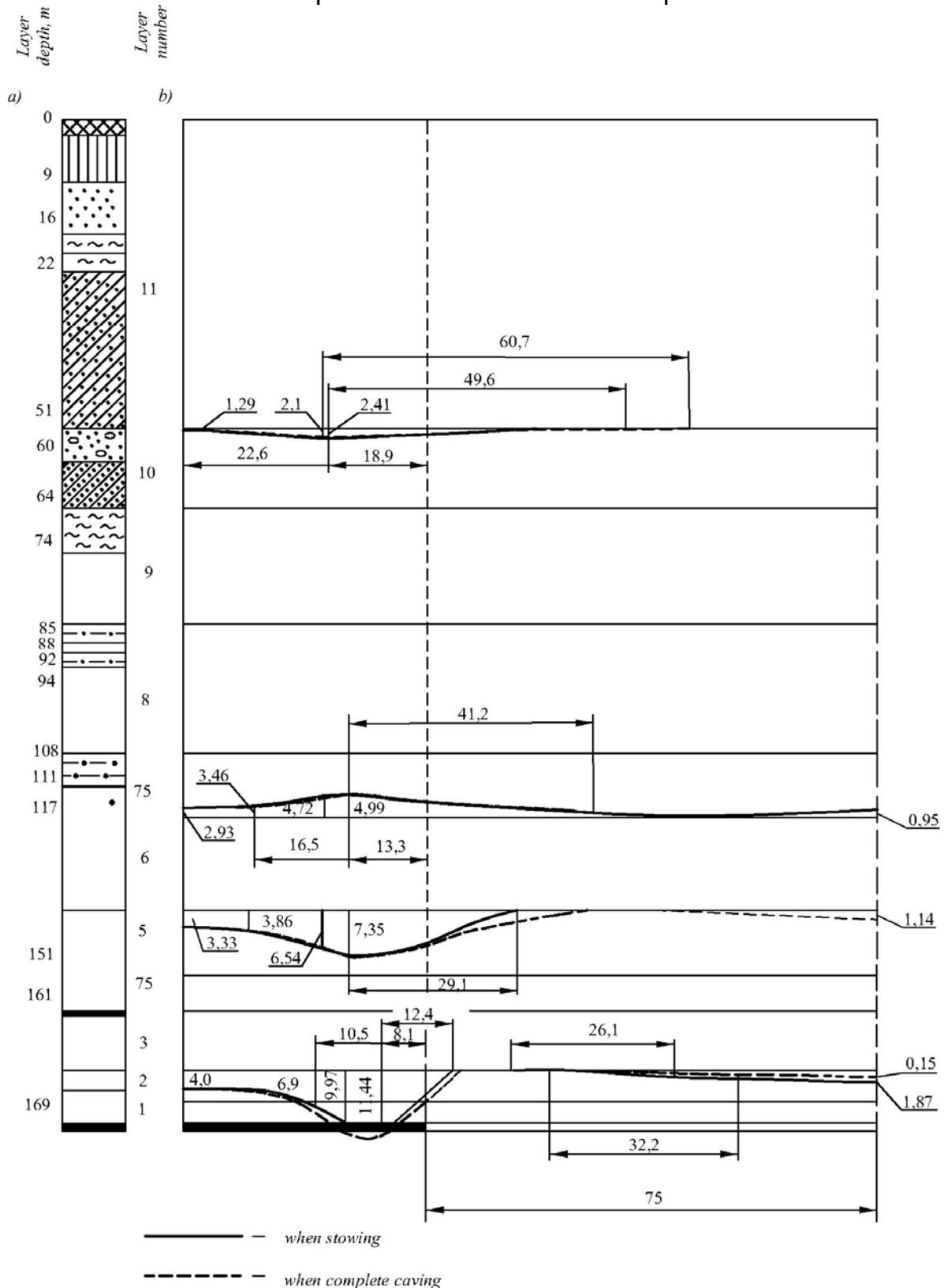


Fig.2.1 Stratigraphic section (a) and distribution of basic loads within rocks of percarbonic mass (b) of C<sub>7</sub><sup>H</sup> seam (“Blagodatnaia” mine)

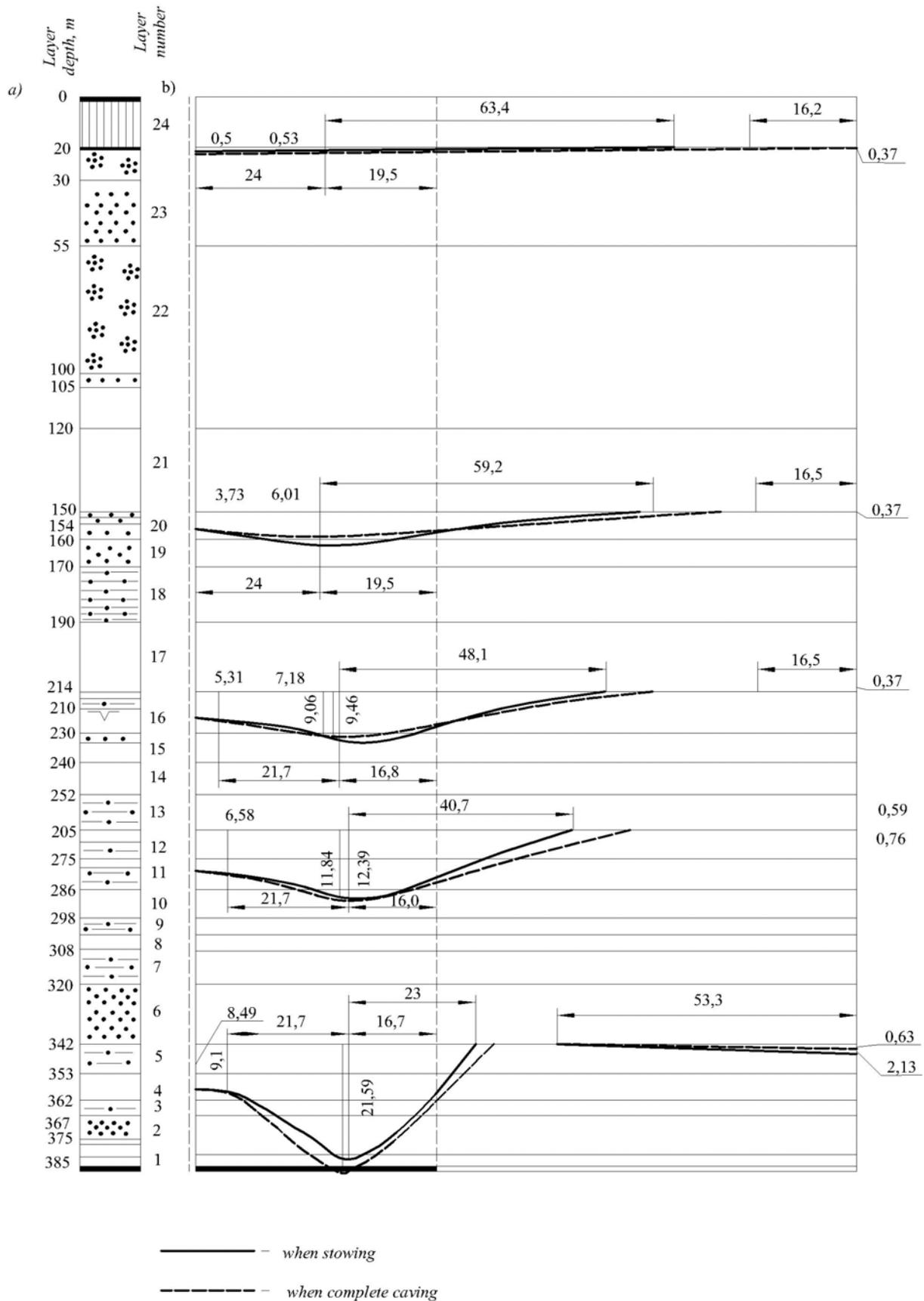


Fig.2.2 Stratigraphic section (a) and distribution of basic loads within rocks of percarbonic mass (b) of  $C_8^H$  seam ("Zapadno-Donbasskaia" mine)

In this context, stope advance was varying from 1 to 3m/d; wall length was varying from 100 to 200 m.

The calculation helps determine nature of basic loads distribution within percarbonic mass rock in terms of worked-out area stowing and in terms of complete caving for comparison. Figures 2.1 and 2.2 illustrate distribution of the loads within layers when a stope advance is 2 m/d and 150 m wall length.

It has been determined that worked-out area stowing results in deloading within index zone and the zone shortening. For example, for “Blagodatnaia” mine basic loads over worked-out area experienced 0.15 MPa up to 1.87 MPa increase at the level of 3<sup>rd</sup> rigid layer (Fig.2.1); for “Zapadno-Donbasskaia” mine (Fig. 2.2) basic loads increase was 0.63 up to 2.13 MPa (6<sup>th</sup> rigid layer).

Similar situation is with next to coal seam first layer (Figures 2.3 and 2.4). In terms of worked-out area basic loads on coal seam roof within index zone decrease by 11-20% in the context of its narrowing; within worked-out area loads experience 0.27 MPa up to 2.6 MPa for “Blagodatnaia” mine and 2.03 MPa up to 3.35 MPa for “Zapadno-Donbasskaia” mine.

Such changes in load diagrams parameters can be explained by the fact that in the process of worked-out area stowing values of seam settling decrease which results in reduce of rocks hovered over the mass as well as size reduction of stratification cavities between rock layers.

Table 2.1 shows fixing parameters and values of standard loads within 1<sup>st</sup> rock layer in terms of various operating conditions. The Table data were applied to determine rules for changes in geometrical and physical parameters of load diagrams depending upon longwall advance and the longwall length (Fig. 2.5).

The data analysis demonstrates that under considered conditions geometrical parameters of load diagrams (that is distance from maximum load to a point where they correspond to gravity ( $a$ ) and distance from a stope to maximum load ( $d_o$ )) do not depend upon a longwall length depending significantly upon a stope advance and a method of roof control.

When increase in a longwall advance is 1 to 3 m/day, index zone ( $a + d_o$ ) experiences 12% narrowing in terms of “Blagodatnaia” mine, and 10% narrowing in terms of “Zapadno-Donbasskaia” mine. Worked-out area stowing results in drastic (down to 20-40%) narrowing of index zone to compare with complete caving (Table 2.1) and decrease in maximum loads upon coal seam ( $\square_2$  and  $\square_{12}$  parameters).

It should also be noted that in terms of worked-out area stowing, loads experiences less intensive changes when longwall extends to compare with roof control by means of complete caving (Fig. 2.5).

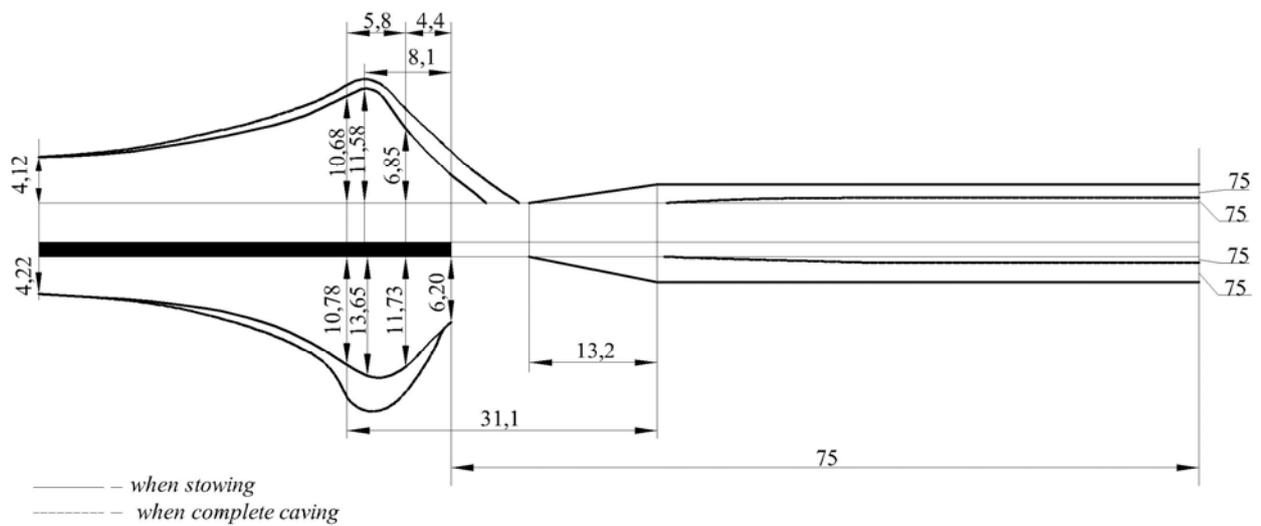


Fig. 2.3 Distribution of standard loads within main roof of  $C_7^H$  seam ("Blagodatnaia" mine)

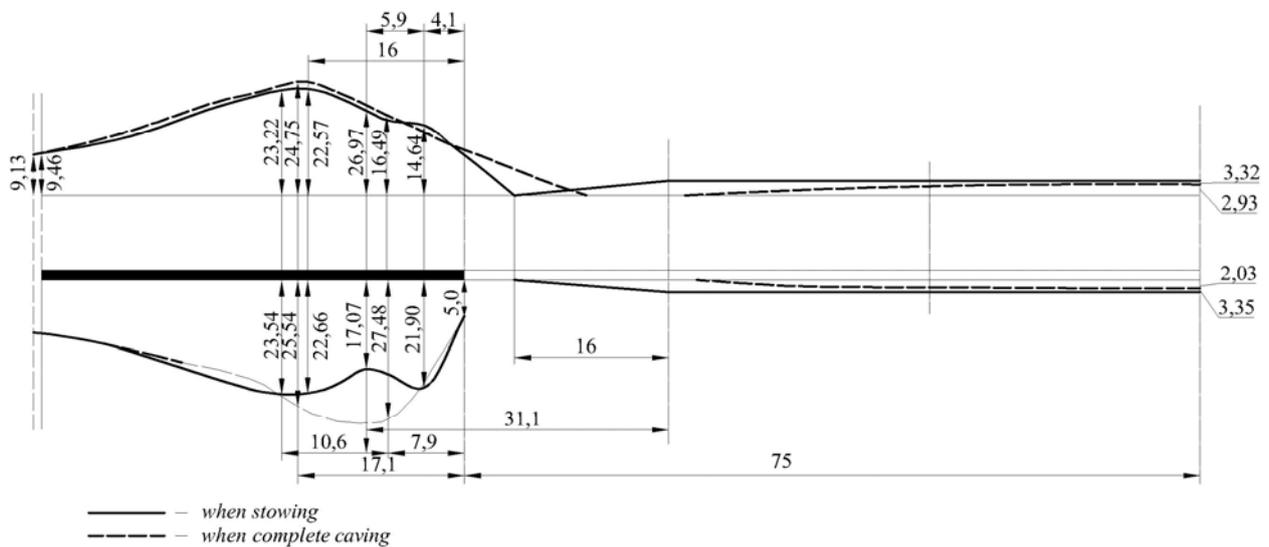


Fig. 2.4 Distribution of standard loads within main roof of  $C_8^H$  seam ("Zapadno-Donbasskaia" mine)

Table 2.1 Parameters of load diagrams within 1<sup>st</sup> rock layer in terms of various operating conditions

Mine	Longwall length, m	A stope advance, m/day															
		1					2					3					
		$a$ , m	$d_0$ , MPa	$\square_{2,}$ MPa	$\square_{12,}$ MPa	$a$ , m	$d_0$ , m	$\square_{2,}$ MPa	$\square_{12,}$ MPa	$a$ , m	$d_0$ , m	$\square_{2,}$ MPa	$\square_{12,}$ MPa	$a$ , m	$d_0$ , m	$\square_{2,}$ MPa	$\square_{12,}$ MPa
"Blagodarnaya"	100	<u>6.2*</u>	<u>4.7</u>	<u>8.93</u>	<u>11.51</u>	<u>5.8</u>	<u>4.5</u>	<u>10.37</u>	<u>11.98</u>	<u>5.3</u>	<u>4.3</u>	<u>13.07</u>	<u>12.40</u>	<u>5.3</u>	<u>4.3</u>	<u>13.07</u>	<u>12.40</u>
		7.3	5.8	10.95	12.86	6.8	5.5	12.28	13.44	6.3	5.2	13.02	13.63	6.3	5.2	13.02	13.63
	150	<u>6.2</u>	<u>4.6</u>	<u>8.76</u>	<u>11.33</u>	<u>5.8</u>	<u>4.5</u>	<u>11.73</u>	<u>13.65</u>	<u>5.3</u>	<u>4.2</u>	<u>14.51</u>	<u>14.22</u>	<u>5.3</u>	<u>4.2</u>	<u>14.51</u>	<u>14.22</u>
7.3		5.8	13.16	15.82	6.8	5.5	15.12	16.93	6.3	5.2	16.33	17.16	6.3	5.2	16.33	17.16	
200	<u>6.2</u>	<u>4.6</u>	<u>9.97</u>	<u>13.36</u>	<u>5.8</u>	<u>4.4</u>	<u>12.23</u>	<u>14.64</u>	<u>5.3</u>	<u>4.2</u>	<u>15.26</u>	<u>15.41</u>	<u>5.3</u>	<u>4.2</u>	<u>15.26</u>	<u>15.41</u>	
	7.3	5.8	13.12	16.07	6.8	5.4	13.62	15.48	6.3	5.2	17.12	18.27	6.3	5.2	17.12	18.27	
"Zapadno-Donbasskaya"	100	<u>6.4</u>	<u>4.3</u>	<u>24.40</u>	<u>19.71</u>	<u>5.9</u>	<u>4.2</u>	<u>21.91</u>	<u>22.52</u>	<u>5.5</u>	<u>4.0</u>	<u>26.89</u>	<u>23.42</u>	<u>5.5</u>	<u>4.0</u>	<u>26.89</u>	<u>23.42</u>
		11.5	8.3	22.95	23.49	10.6	8.0	25.13	23.38	9.8	7.6	26.07	21.44	9.8	7.6	26.07	21.44
	150	<u>6.4</u>	<u>4.3</u>	<u>25.28</u>	<u>20.34</u>	<u>5.9</u>	<u>4.1</u>	<u>21.90</u>	<u>22.67</u>	<u>5.5</u>	<u>4.0</u>	<u>26.67</u>	<u>23.27</u>	<u>5.5</u>	<u>4.0</u>	<u>26.67</u>	<u>23.27</u>
11.5		8.2	23.33	24.49	10.6	7.9	27.43	25.52	9.8	7.5	31.57	26.21	9.8	7.5	31.57	26.21	
200	<u>6.4</u>	<u>4.3</u>	<u>24.06</u>	<u>19.41</u>	<u>5.9</u>	<u>4.1</u>	<u>28.13</u>	<u>20.23</u>	<u>5.5</u>	<u>4.0</u>	<u>25.93</u>	<u>22.70</u>	<u>5.5</u>	<u>4.0</u>	<u>25.93</u>	<u>22.70</u>	
	11.5	8.2	23.31	24.54	10.6	7.8	27.28	25.46	9.8	7.5	33.29	26.49	9.8	7.5	33.29	26.49	

\*stowing is in numerator, complete caving is in denominator

Settlings, displacements and horizontal deformations of rock layers over working area of longwall and its worked-out area are those key parameters (in addition to standard loads within index zone) characterizing stress and strain state of roof rocks. In no small degree main roof rock settlings determine such mining conditions as possibility for inrush, rock failure within face space, load on powered support etc.

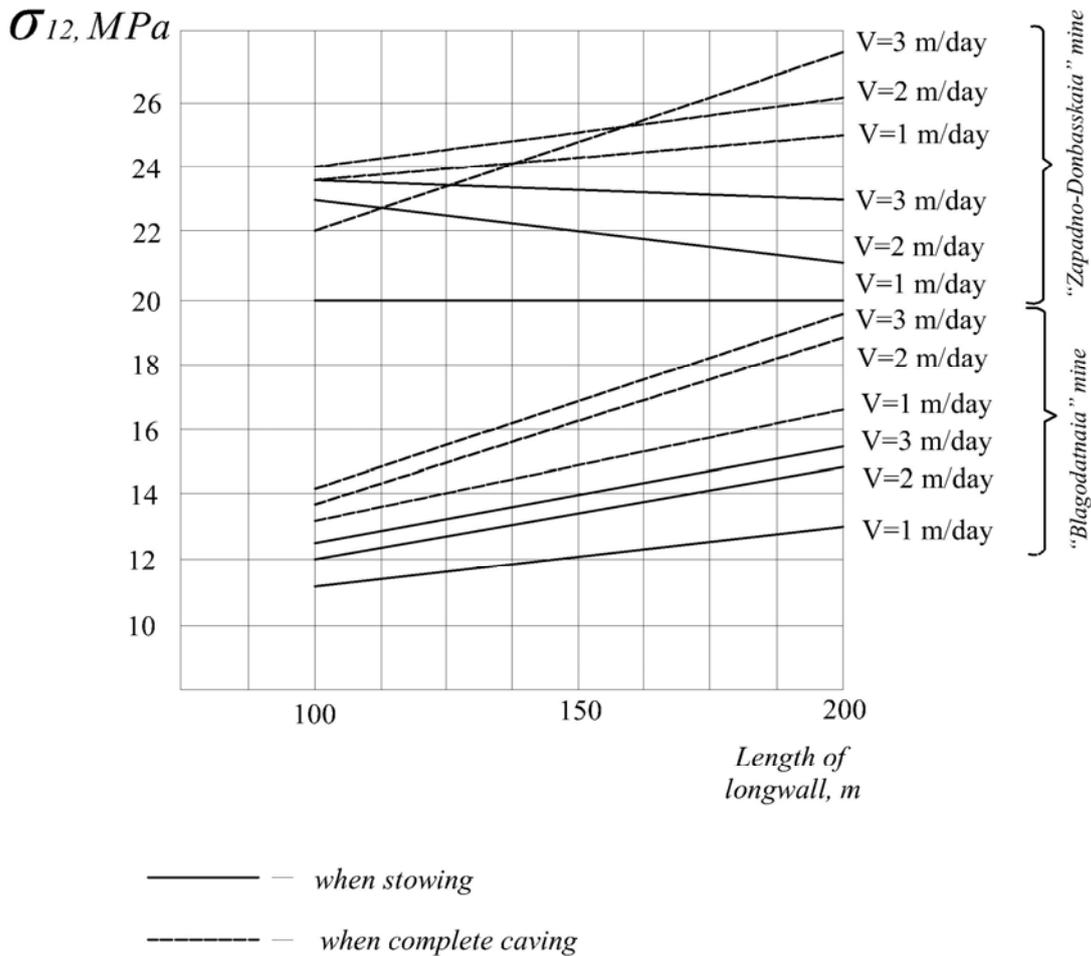


Fig.2.5 Dependence of maximum loads within main roof on the length of longwall and a stope advance

Subsidence ( $Y$ ), horizontal displacements ( $E$ ) and deformations ( $\Gamma$ ) have been determined using [74]:

$$Y = \frac{12L^3}{\pi^3 f(k)h^3} \sum_{k=1}^n \frac{B_k}{k^3} \left( \cos \frac{k\pi}{L} x - 1 \right); \quad (2.1)$$

$$E = \frac{12L^2}{\pi^2 f(k)h^2} \sum_{k=1}^n \frac{B_k}{k^2} \left( \sin \frac{k\pi}{L} x \right); \quad (2.2)$$

$$\Gamma = \frac{\varepsilon_2 - \varepsilon_1}{x_2 - x_1}; \quad (2.3)$$

where  $L$  is complete maximum half-span (half-span of rock layer when independent subsidence takes place), m;  $h$  is thickness of rock layer, m;  $B_k$  is coefficient depending on parameters of load diagram;  $f(\kappa)$  is deformation module varying in length.

Fig. 2.6 illustrates nature of changes in subsidence, deformations, and horizontal deformations within 1<sup>st</sup> rock layer depending upon distance to a stope, and roof control technique if longwall length is 150 m and stope advance is 2 m/day. It is accepted that in terms of “Blagodatnaia” mine mined seam thickness is 1.2 m, and 1m in terms of “Zapadno-Donbasskaia” mine.

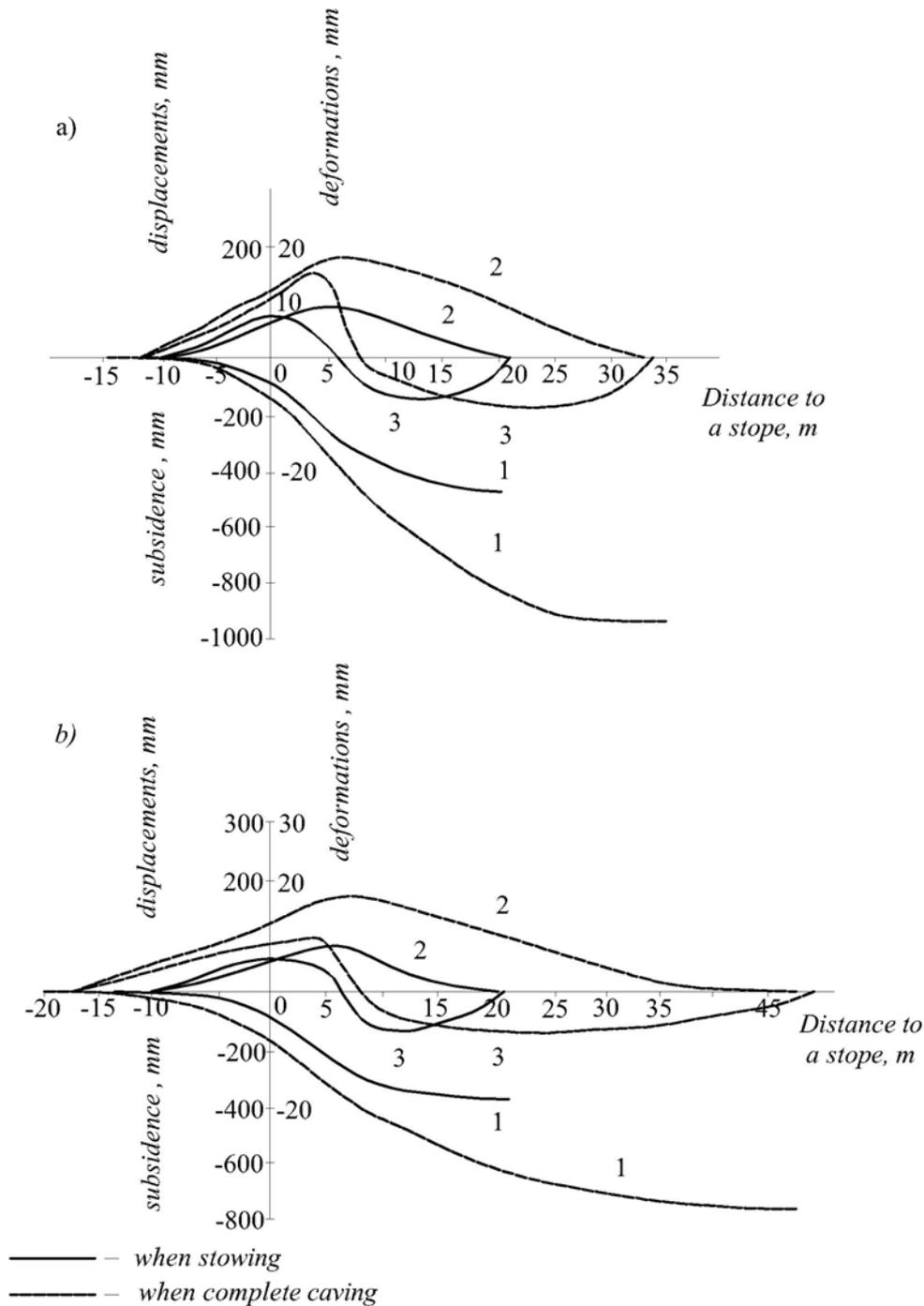


Fig. 2.6 Subsidence, displacements, and deformations of mine roof control in the context of “Blagodatnaia” mine (a) and “Zapadno-Donbasskaia” mine (b)

As it follows from Fig. 2.6 in the process of worked-out area stowing, subsidence (curve 1), horizontal displacements (curve 2), and deformations (curve 3) experience almost twofold decrease. Moreover, maximum half-span of layers also decreases as well as width of index zone in front of longwall face.

Value of horizontal deformation can help analyze probability of fracture initiation within a roof and, consequently, intrushes and roof rock failures.

Study of Fig. 2.6 (curve 3) shows that value of rock layer maximum deformations is almost 5-6 mm/m in terms of worked-out area stowing, and 10-17 mm/m in terms of complete caving.

The data enable to say that worked-out area stowing substantially reduce the possibility of roof discontinuity, and rock intrushes and failures in it.

Values of roof rocks subsidence and displacements both in the neighbourhood of face and at the boundary of a longwall working area are of the greatest interest. They determine power parameters and geometry of powered support making it possible to identify volume of worked-out area and other parameters required for stowing. Table 2.2 represents the values (regardless of support resistance) depending upon a stope advance; Figures 2.7 and 2.8 illustrate their graphs. Conventionally, in both longwalls under discussion, width of working area is 5m.

Table 2.2 data demonstrate that in terms of worked-out area stowing significant subsidence and horizontal displacements of roof rocks take place both in the neighbourhood of a face, and at the boundary of a longwall working space. Dependence of changes in calculated values of main roof subsidence on a longwall length is close to linear one; for this reason Figures 2.7 and 2.8 reduce them to straight line.

It follows from Fig. 2.7 that in the context of “Blagodatnaia” mine increase in longwall length factors into increase in main roof rock subsidence values in the neighbourhood of a face and at the boundary of working space; when a stope advance increase they experience their decrease. However, in terms of worked-out area increase in subsidence values is less intensive than in terms of complete caving. “Zapadno-Donbasskaia” mine demonstrates somewhat different pattern of changes in subsidence in the context of various operational parameters (Fig.2.8) where increase in longwall length at slow stope advance results in reduce of roof subsidence if longwall length increases. That can be explained by the fact that when half-span of a longwall exceeds maximum permissible half-spans of each seam; that is in terms of complete undermining of percarbonic mass, certain drop of stress may take place within index zone owing to redistribution of standard loads [75]. Consequently, values of main roof subsidence may also decrease. Such decrease is mainly seen if rock bridges are available within percarbonic mass. If so stresses in index zone goes down to 15-20 % and experience their increase in worked-out area.

Table 2.2 Values of roof rocks subsidence and displacements at various stope advance and longwall length

Mine	Stope advance, m/day												
	1				2				3				
	In the neighbourhood of a face		At the boundary of working space		In the neighbourhood of a face		At the boundary of working space		In the neighbourhood of a face		At the boundary of working space		
Longwall length, m	Subsidence, mm	Displacements, mm	Subsidence, mm	Displacements, mm	Subsidence, mm	Displacements, mm	Subsidence, mm	Displacements, mm	Subsidence, mm	Displacements, mm	Subsidence, mm	Displacements, mm	
10	97*	60	276	93	93	61	265	92	172	79	240	97	162
	136	94	365	132	98	98	361	171	229	95	267	132	172
15	97	61	281	97	64	277	98	182	182	86	262	103	176
	144	99	365	171	103	379	180	241	241	141	390	188	249
20	99	61	282	94	65	268	98	186	186	90	271	105	181
	156	106	414	180	110	410	187	259	259	152	427	202	275
10	110	61	288	73	57	252	81	163	163	83	243	85	160
	206	135	398	184	123	332	175	166	166	146	300	172	154
15	106	61	285	77	57	248	81	161	161	82	242	85	160
	191	129	374	179	126	341	178	170	170	148	305	175	157
20	103	60	279	77	61	267	82	172	172	81	239	84	158
	191	129	374	179	126	341	179	171	171	201	401	208	200

\*stowing is in numerator, complete caving is in denominator

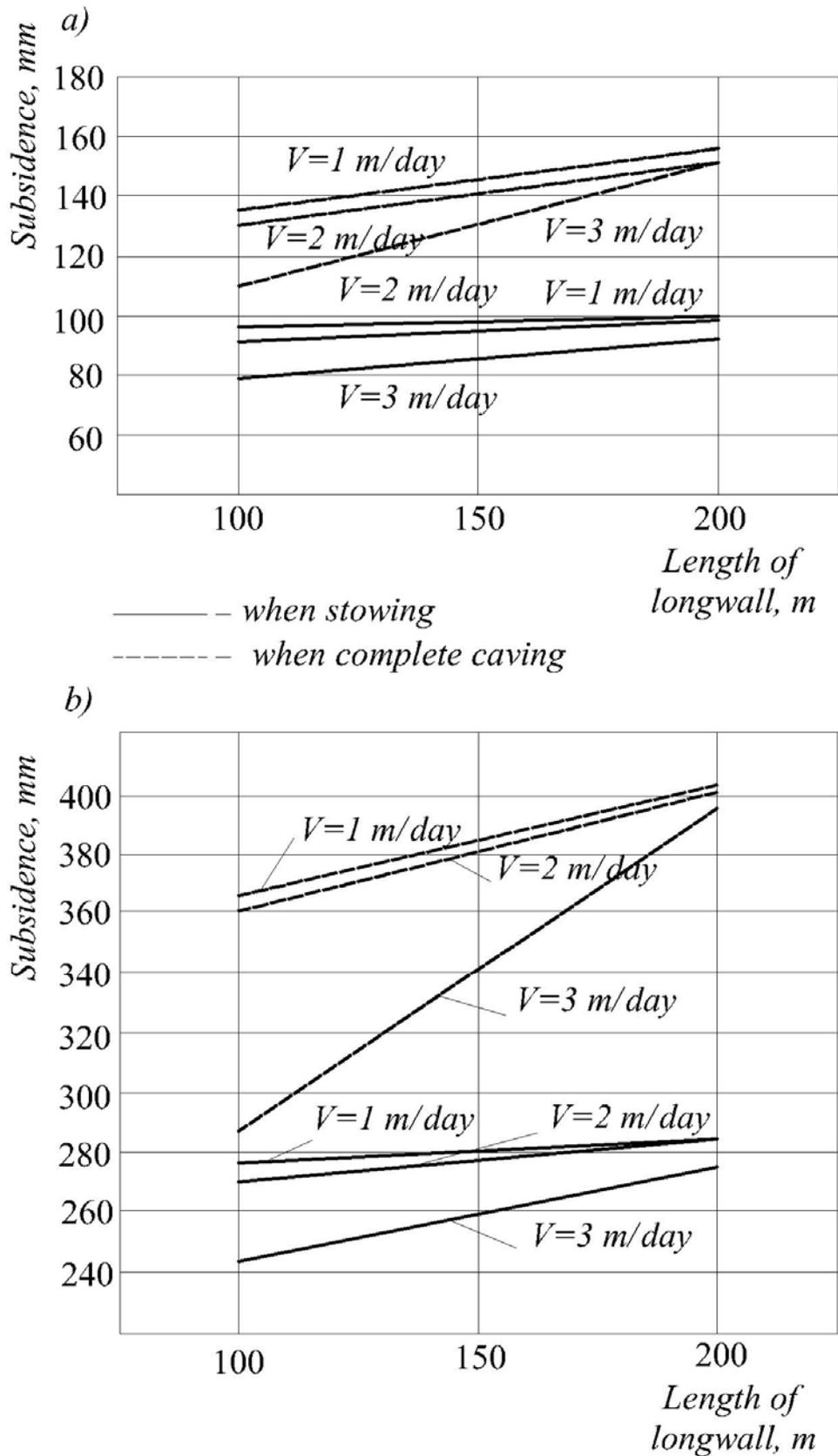


Fig. 2.7 Dependence main roof values subsidence on a longwall length and a stope advance in the context of “Blagodatnaia” mine: a) in the neighbourhood of a face; b) at the boundary of a longwall working space.

Such a situation is typical for “Zapadno-Donbasskaia” mine where sandstone which thickness is 22 m occurs in percarbonic mass; the sandstone is rock bridge. In the context of the mine changes in subsidence value resulting from a longwall length are less important in terms of stowing than in terms of complete caving (Fig. 2.8).

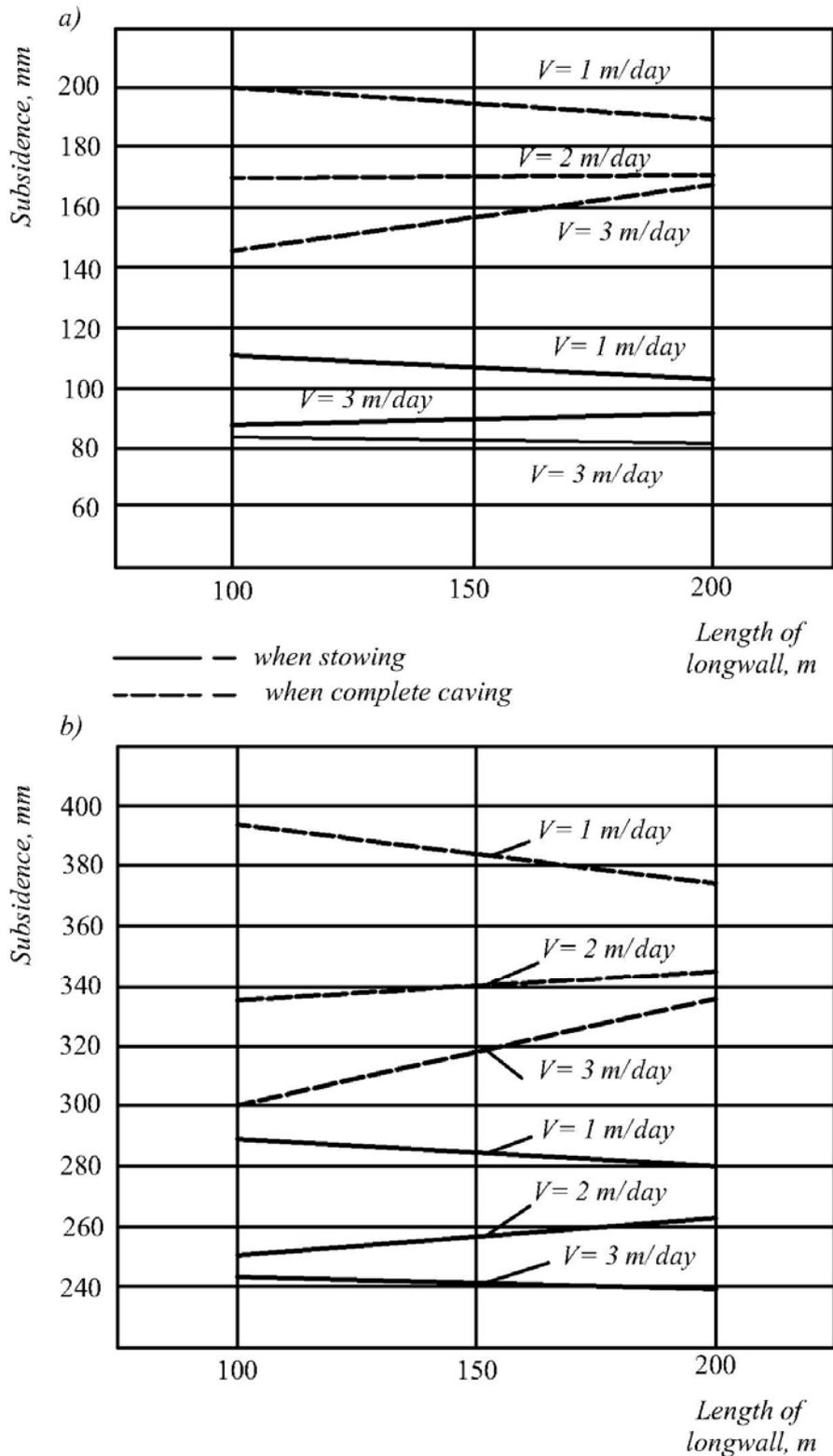


Fig. 2.8 Dependence of main roof subsidence value on a longwall length and a stope advance in the context of “Zapadno-Donbasskaia” mine: a) in the neighbourhood of a face; b) at the boundary of working space

Roof subsidence within working space of a longwall that is difference between subsidence at the boundary of working space and in the neighbourhood of a face while stowing (in the context of “Blagodatnaia” mine) is 162-186 mm to be by 22-29% less to compare with complete caving (Table 2.2); in the context of another mine under study they are 158-179 mm (to be by 5-7 % less to compare with complete caving). The difference can be explained by the fact that rock bridge is available within percarbonic mass of “Zapadno-Donbasskaia” mine. The rock bridge takes certain share of load from overlying rocks, and, to some extent, with the help of less mined seam thickness. With the increase in a stope advance percentage of roof subsidence within working space of a longwall experiences certain decrease and vice versa. Value of such subsidence changes (especially while stowing) is inappreciable being no more than 10% staying within accuracy of engineering evaluations.

It should also be noted that Table 2.2 demonstrates values of roof subsidence without notice of foot rock heaving. Numerous research performed for conditions of Western Donbass have determined [75] that usually value of ground uplift is 30-40% of roof rock subsidence value. Hence, one may assume that while stowing in the context of “Blagodatnaia” mine enclosing roof and floor convergence is 218-251 mm; it is 213-241 mm in the context of “Zapadno-Donbasskaia” mine.

Take a notice that resistance of powered support was not taken into consideration while determining roof subsidence and enclosing roof and floor convergence. However, it is known that to some extent powered support lowers a level of roof rock subsidence. To determine dependences of changes in subsidence values on powered support resistance using following expressions [75]:

$$M_p = \frac{2L}{\pi} \left(1 - \frac{Y_3}{Y_p}\right) \sum_{n=1}^m \left(\cos \frac{k\pi}{L} x - 1\right); \quad (2.4)$$

$$R_k = \frac{M_p}{x}, \quad (2.5)$$

where  $M_p$  is torque reaction, t/m;

$L$  is complete half-span of rock seam, m;

$B_k$  is coefficient depending upon parameters of load diagram;

$Y_3$  is required (specified) rock subsidence at the boundary of working space, m;

$Y_p$  is estimated rock subsidence where support effect is not involved, m (is determined using expression 2.1);

$x$  is current absciss equal to  $f_2 + M$ , m;

$f_2$  is distance from zero point to a stope (width of index zone), m;

$M$  is width of a longwall working space, m.

The calculation intends that when roof is controlled by means of complete caving, thin immediate roof occurs over worked-out area; due to formation of vertical fissures it is not connected with in-situ rocks and has not carrying capacity. Thus, support of a stope resists load only from main roof. It is accepted that width of working space in both longwalls under analysis (both in terms of worked-out area

stowing and in terms of complete caving) is 5m, and resulting force of support is applied at the boundary of working space of a longwall.

Values of specified subsidence were changed; then, using expressions (2.4) and (2.5) both torque reaction and total reaction of a support to be applied to provide specified subsidence. As a result, a dependence of roof subsidence value on powered support resistance has been identified (Fig. 2.9). The calculation has also determined that definable parameters has minor dependence on a longwall length; that is why Fig. 2.9 explains dependences of running values if longwall length is 150 m.

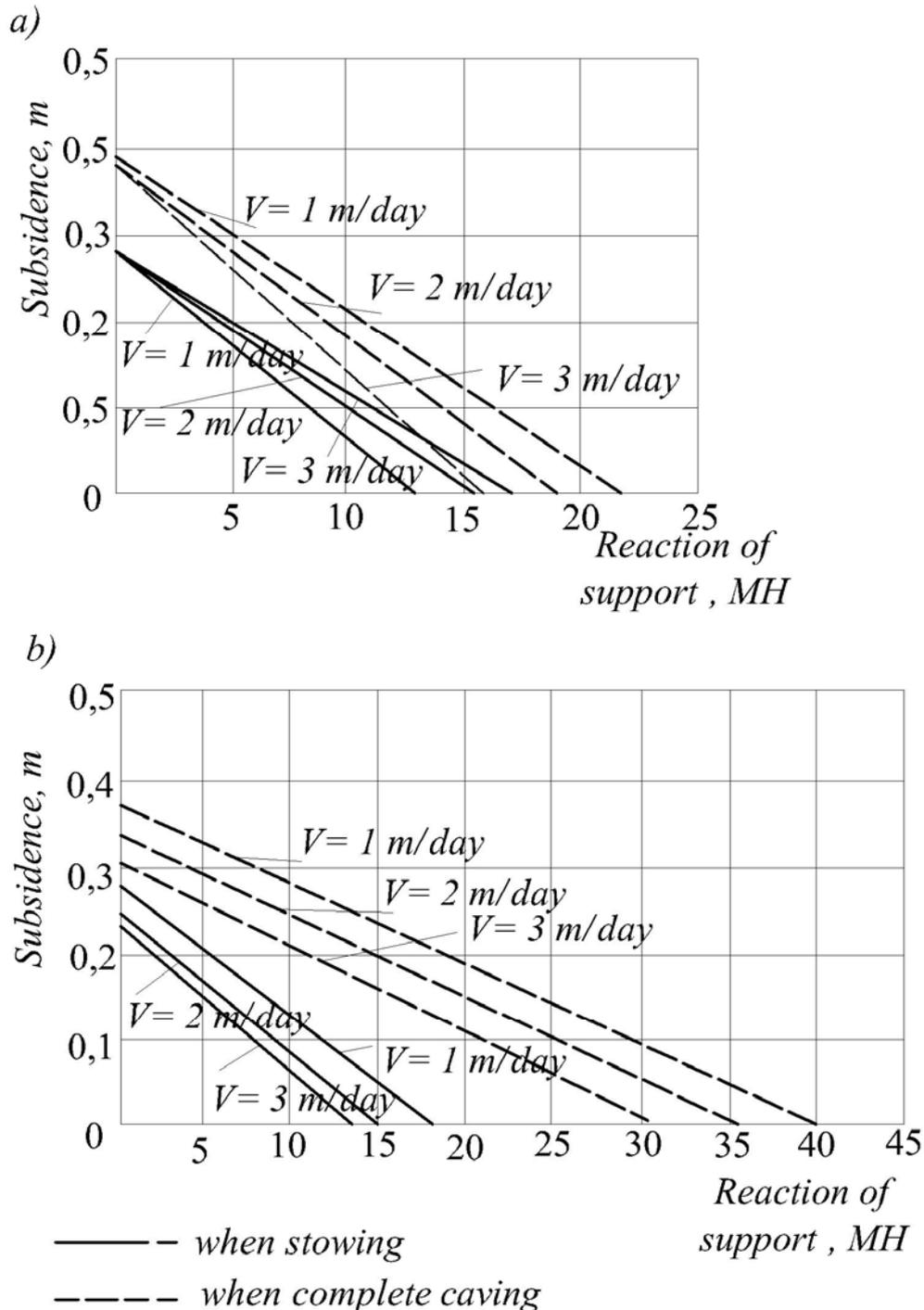


Fig. 2.9 Dependence of changes in roof rock subsidence on support resistance in the context of “Blagodatnaia” mine (a) and “Zapadno-Donbasskaia” mine (b)

Dependences of changes in roof subsidence on powered support resistance is of linear nature; moreover, to perform 10mm loss in subsidence value, worked-out area stowing requires 500-600 kN reaction per each running meter of a longwall. Total reaction of available powered supports to mine thin seams is 1500-2500 kN; that is why their application will help minify roof subsidence down to 30-40 mm if resulting force is applied at the boundary of working space.

Actually, resulting reaction of support is between 1<sup>st</sup> and 2<sup>nd</sup> rows of props; that is in terms of stowing it is 1.5-2 m closer to a stope. Thus, to minify roof subsidence by 10mm at the boundary of working space carrying capability of a support should be 650-750 kN, and powered support in use should be able to minify roof subsidence by 20-30mm to be almost 10% of unsupported subsidence value.

Changes in width of a longwall working space are more important for a value of roof rock subsidence. For example, when longwall width experience 5 down to 4m decrease, unsupported roof subsidence experiences 277 down to 252 mm decrease in the context of “Blagodatnaia” mine and 248 down to 277 mm in the context of “Zapadno-Donbasskaia” mine; if a longwall width is 6m, then subsidence is 301 and 267 mm respectively (Fig. 2.10). Practically similar values of changes in subsidence from working space width are observed when support resistance is 1500-2500 kN per running meter of a longwall.

Thus, for 10 mm reduce in roof subsidence it is quite enough to narrow working space of a longwall by 0.4-0.47 m to be structurally simpler than 500-600 kN increase in total support reaction.

It should be noted that there are many mathematical expressions enabling to determine convergence of enclosing roof and floor within working space of a longwall for certain mining and geological conditions. Thus, papers [16, 61] recommend following expression to determine rock convergence value:

$$\Delta h = \alpha m R , \quad (2.6)$$

where  $\alpha$  is coefficient taking into consideration specific rock convergence,  $m^{-1}$ ;  $m$  is mined seam thickness, m; and  $R$  is distance to a stope, m.

Moreover, expression developed by AURSI [52] involves powered support resistance as well as a period when roof is within a face area.

Our research described above help determine that in the context of stowing powered support resistance, stope advance, and longwall length have minor effect on changes in rock convergence value. Therefore, expression (2.6) can be used for approximate calculations for Western Donbass longwalls operating with worked-out area stowing.

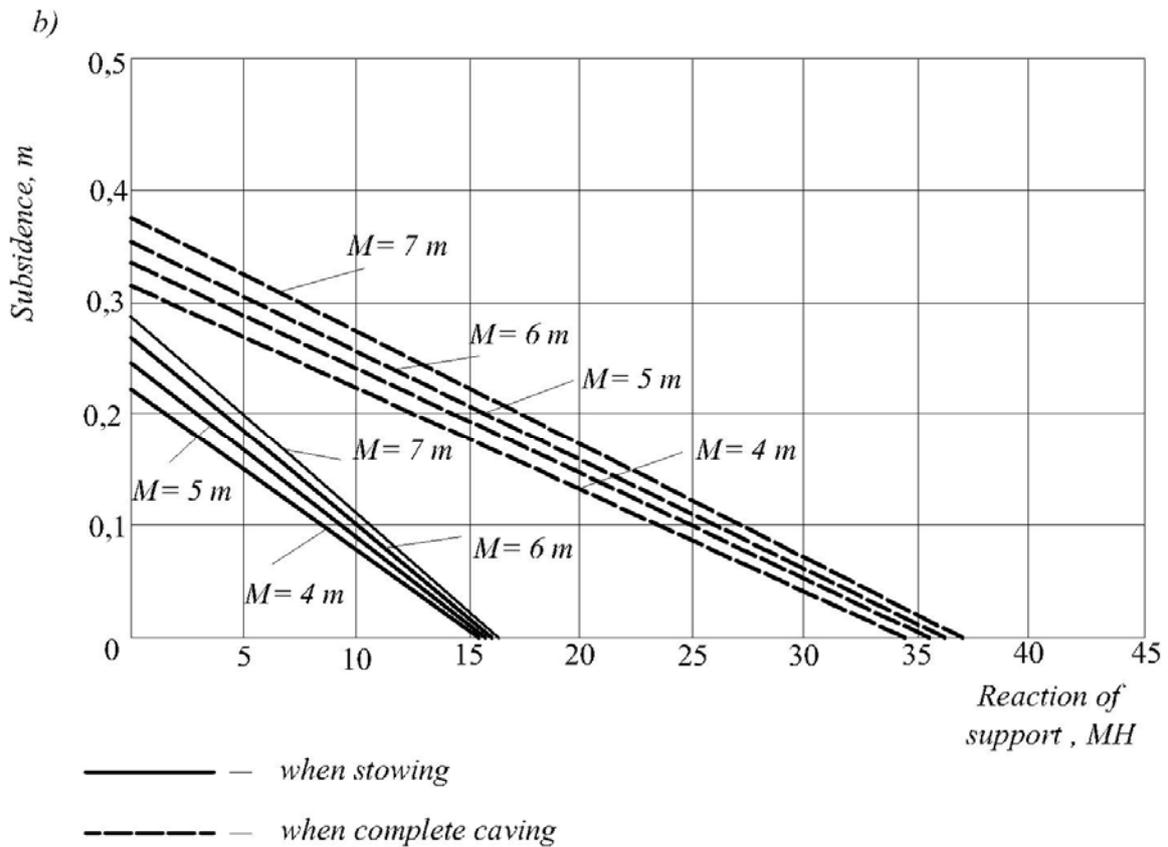
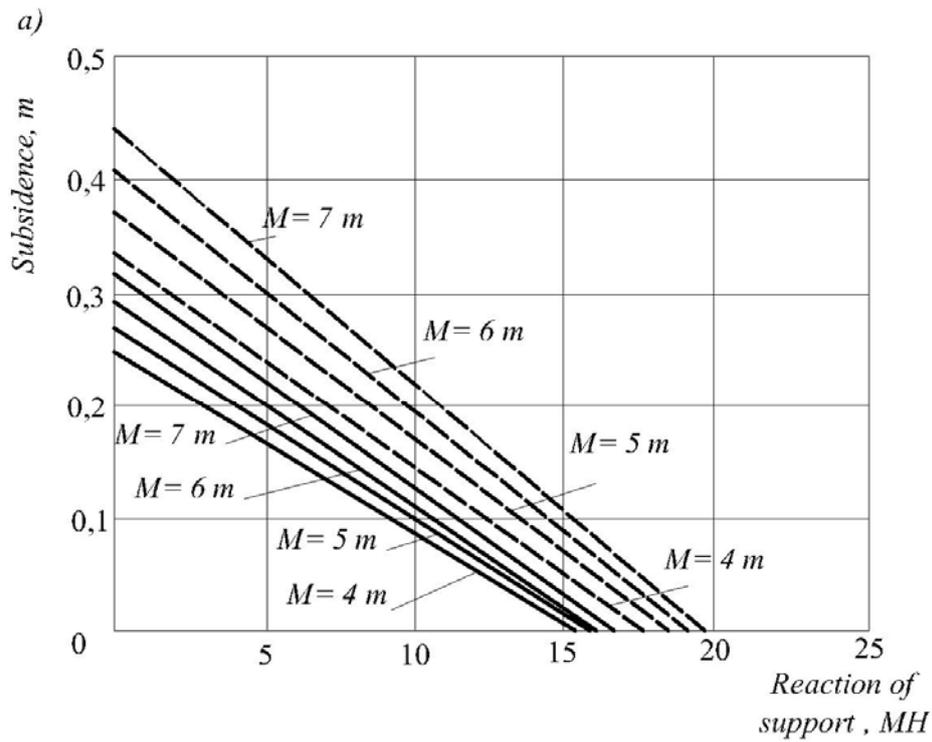


Fig. 2.10 Dependence of changes in roof rock subsidence on support resistance in terms of different width of a stope for "Blagodatnaia" mine (a) and "Zapadno-Donbasskaia" mine.

Coefficient  $\alpha$  being a part of the formula has been determined on earlier values of roof subsidence and enclosing roof and floor convergence within working space of a longwall. It should be considered as equal to 0.026-0.03 to determine roof rock subsidence and 0.036-0.04 to determine total value of enclosing roof and floor convergence. For more accurate determination of convergence values, it is required to perform detailed calculations of stress and strain state of mass.

To determine load upon stowing mass, values of surface subsidence and dimensions of displacement cavity a variant involving time passed after mining was over has been calculated for the two mines under consideration. It is assumed that 300 working days passed after mining was over; in this context advance of a stope was 200 m and longwall length was 150 m.

Results of the calculations show that if complete stowing of worked-out area takes place, the stowing mass will take up load equal to 2.65 MPa in terms of “Blagodatnaia” mine to be 63% of loads within virgin ground, and 7.46 MPa (78 % of  $\gamma H$ ) in terms of “Zapadno-Donbasskaia” mine.

In the context of complete caving load upon floor of seam will be 1.85 and 5.64 MPa respectively.

Thus, values of load on stowing mass at the end of 300 days after mining are not equal to loads within virgin rock; however their excess over load within worked-out area in terms of complete caving is 25-30%.

Moreover, load on 30 m width stowing track has been determined for “Blagodatnaia” mine in the context of partial stowing of worked-out area. The track reaction required to minimize roof subsidence from 875 mm down to 445 mm in its central part should be 48070 kN; maximum load in this point is 2.9 Pa which somewhat exceeds load upon stowing mass in terms of complete worked-out area stowing.

Table 2.3 illustrates rated parameters of earth surface displacement cavity in terms of various techniques of roof control and 150 m longwall length after 300 days of mining have passed.

Table 2.3 Parameters of earth surface displacement cavity in terms of various techniques of roof control

Mine	Roof control technique	Parameters			
		$a$ , m	$d_o$ , m	Complete half-span $L$ , mm	Maximum subsidence $\gamma$ , mm
“Blagodatnaia”	Stowing	39.6	21.6	119.5	453
	Complete caving	39.6	24.8	139.4	949
“Zapadno-Donbasskaia”	Stowing	41.3	25.9	134.9	290
	Complete caving	41.3	28.4	144.7	538

As data from Table 2.3 show, earth surface subsidence in the context of worked-out area stowing will be: 453 mm or 38% of mined seam thickness for “Blagodatnaia” mine and 290 mm (29 % of  $m_g$ ) for “Zapadno-Donbasskaia” mine to be respectively less by 52 % and 46 % to compare with complete stowing. Moreover, dimensions of displacement cavity reduce if stowing. Its effect extends beyond the

## 2.3. Geometrical parameters of operation schedules

### 2.3.1. Width of stowing track

As noted above, while thin and very thin seam mining it is expedient to use undercut rocks of a stope being a result of selective (coal and rock) technique application as stowing material. Such a stowing track width can be determined on the basis of equality of rock mass in pillar and in stowing mass, that is

$$lm_{np}r\gamma_{nop} = l_3r(m_g\Delta h)k_3\gamma_{3ak}, \quad (2.7)$$

where  $l$  is a longwall length, m;  $m_{np}$  is undercut rock thickness, m;  $r$  is web width of operating member of coal shearer, m;  $\gamma_{nop}$  is undercut rock density, t/m<sup>3</sup>;  $l_3$  is stowing track width, m;  $m_g$  is extracting seam thickness, m;  $\Delta h$  is enclosing roof and floor convergence within working space of a longwall, m;  $k_3$  is worked-out area volume efficiency;  $\gamma_{3ak}$  is stowing mass density. Therefore

$$l_3 = \frac{lm_{np}\gamma_{nop}}{(m_g - \Delta h)k_3\gamma_{3ak}}, \quad (2.8)$$

$\gamma_{3ak}/\gamma_{nop}$  ratio is substituted for a coefficient involving stowing mass density  $k_{n.3}$ ; in terms of pneumostowing it is 0.7 - 0.75 in the rough.

It is also clear that selective mining cannot be performed throughout the longwall length; it concerns pockets and those sites where undercutting with the help of coal shearer takes place.

It should also be considered that due to irregularity of loading units of winning machine certain amount of coal will fall within stowing mass; as a result stowing track experiences slight widening.

Underground investigations (Chapter 4) have determined that regardless of coal seam thickness stowing mass contains almost 16% of coal if coal shearer of 1K101 type is applied as winning machine, and up to 2% if 1K103 coal shearer is applied.

Based upon mentioned above expression (2.8) is

$$l_3 = \frac{(l - l_h - l_{3ap})m_{np}k_{h.n}}{(m_g - \Delta h)k_3k_{n.3}}, \quad (2.9)$$

where  $l_h$  is total length of pockets, m;  $l_{3ap}$  is length of that site of a longwall within which undercutting with the help of coal shearer takes place, m;  $k_{h.n}$  is coefficient involving incompleteness of coal loading. The coefficient is taken as 1.16 if 1K101 coal shearer is applied; 1.11 is taken if 1K103 coal shearer is applied.

Enclosing roof and rock convergence within working space of a longwall  $\Delta h$  is determined using either actual data or (2.6) expression with the help of identified coefficients  $\alpha$ .

Fig. 2.11 shows dependence of stowing track width calculated according to expression (2.9) on enclosing roof and floor cutting and coal seam thickness.

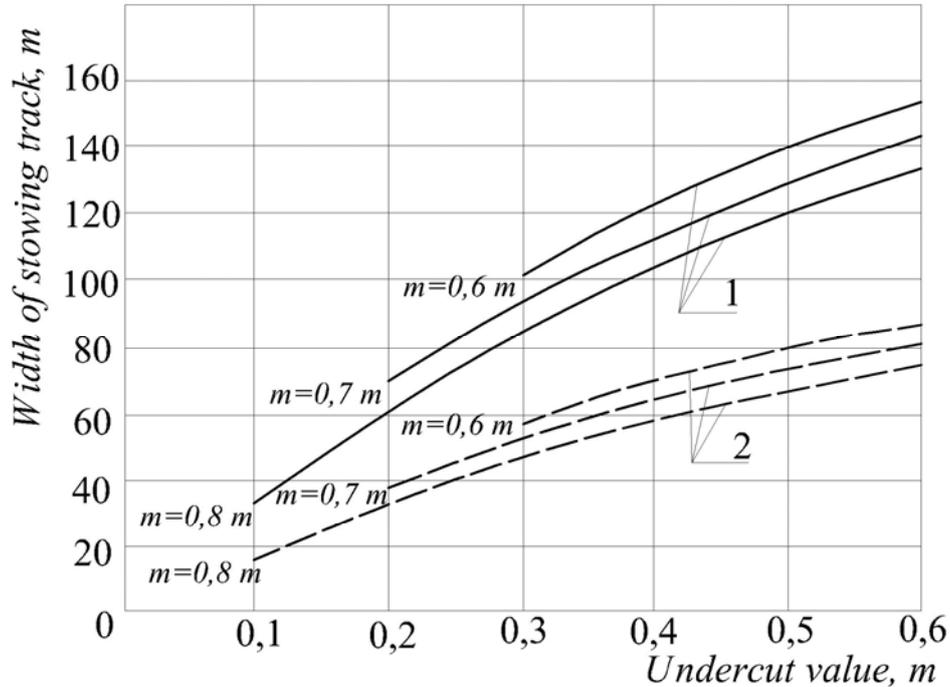


Fig. 2.11 Dependence of stowing track on a value of enclosing roof and floor and coal seam thickness: 1. If longwall length is 160 m; 2. If longwall length is 100 m.

As it follows from Fig. 2.11 practically all stopes in which enclosing roof and floor undercut is applied stowing track will be narrower to compare with longwall length; thus, complete stowing of worked-out area requires rock obtained from other sources. Extra rock volume for complete stowing of worked-out area in terms of a winning cycle can be determined by:

$$Q_H = (l - l_3)(m - \Delta h)k_{n3}k_3 \quad (2.10)$$

Lately combined systems and longwall ones have become popular; their application means that rock left after gate road construction has been completed is stowed into worked-out area of longwall. If development working follows a longwall then width of a pack is determined by the formula:

$$l_3 = \frac{S - S_y}{(m_g - \Delta h)k_3k_{n3}} \quad (2.11)$$

where  $S$  is driving section,  $m^2$ ;  $S_y$  is a part of a stope of cross-section of a drift,  $m^2$ .

Approximately stowing track width can be determined with the help of graphs (Fig. 2.12) obtained from (2.11) expression.

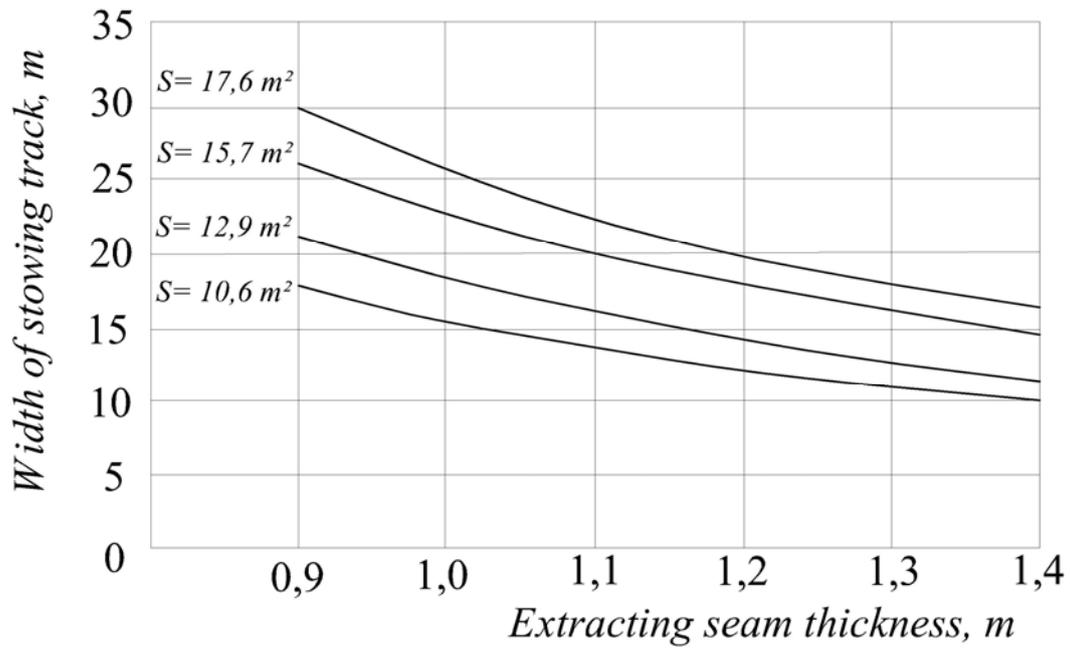


Fig. 2.12 dependence of stowing track width on mined seam thickness and a working being driven

If worked-out area is stowed using rock being a result undercut in a stope and construction of one of drifts then stowing track width is determined by means of (2.9) and (2.11) expressions summing up. That is

$$l_3 = \frac{S - S_y + (l - l_n - l_{zap})m_{np}k_{nn}}{(m_6 - \Delta h)k_3k_{n3}} \quad (2.12)$$

Analysis of the dependences (Figures 2.11 and 2.12) means that while mining seams with 0.7-0.8 m thickness and almost 0.2 m undercut, amount of rock being undercut is quite sufficient to stow up 35-45% of worked-out area; while applying combined mining method the value is almost 50 %. To perform complete stowing of worked-out area, deficiency rock amount (about 50-60 m<sup>3</sup> per each winning cycle) should be delivered from other sources; it may be underground crushing-and-sorting plant obtaining rock as a result of development workings construction and maintenance.

Thus, application of undercut rocks from a stope for complete stowing makes it possible to halve need for stowing material simplifying infrastructure of stowing facilities.

### 2.3.2. Length of a longwall

Analysis of rock mass stress and strain state has determined that changes in geomechanical parameters depending upon length of a stope are negligible and length of a longwall should correspond to economical value.

A number of works [9, 21, 28, 31, and 93] concern the problem of longwall length substantiation. Available optimization techniques of the parameter are of various approaches. Total costs [9, 93] or production cost of a ton of mined coal [21, 28, 31] are assumed as optimality criterion. Moreover, paper [21] takes into account economic damage due to loss of mining in the process of emergency maintenance.

We think that all available techniques have a number of disadvantages. For example, per face per day output is predetermined by constant value being independent of a longwall length. Besides, expenditures connected with mining are calculated with the help of simplified empiric expressions preventing from taking into consideration great variety of conditions and operation schedules of coal seam mining.

To close the gaps we have developed a technique considering minimum of total costs as optimality criterion. In this context, production cost involves those expenditures which experience essential modifications depending upon longwall length; that is:

- Construction of mine workings which maintain longwall;
- Haulage of general cargo and auxiliary one within mine workings which maintain longwall;
- Wages for a longwall work force;
- Appreciation of mining, hauling, and drifting equipment;
- Materials;
- Disassembly of equipment;
- Worked-out area stowing;
- Maintenance of mine workings neighbouring a stope. Total capital expenditures include expenditures connected with equipment acquisition and installation.

It should be noted that production cost of a ton of coal as well as total cost mentioned in the technique are not parameters of production unit as they don't involve a number of expenditures having minor effect on a parameter under determination (electricity bill, certain material types etc. They include certain expenditures involving in the process of general mine production cost and expenditures determination (construction of mine workings and their maintenance; cargo haulage etc.).

The technique takes into account effect of longwall length on a factor of longwall availability with the help of empiric factors obtained experimentally by V.N. Shabratski, DMI engineer, for various types of winning equipment.

The technique makes it possible to determine economically attractive length of stopes equipped with powered systems. The stopes apply both complete technique (inclusive of enclosing roof and floor undercut) and selective one in terms of long-pillar method, combined method or longwall method in terms of different methods of roof control.

Expenditures connected with construction of mine workings to maintain longwall are determined by:

$$C_1 = \frac{a_b(L + l_u) + a_k(L + l_u) + a_{m.k}l + a_{mn}}{m\gamma cL} \quad (2.12)$$

where  $a_b$ ,  $a_k$  is cost to construct a meter of airway working and belt roadway respectively, RUB/m;  $L$  is length of extraction pillar, m;  $l_u$  is total width of protective pillars between main haulage road, and installation/removal chambers, m;  $l$  is length of longwall, m;  $C$  is coal mining factor;  $a_{m.k}$  is cost to construct a meter of installation chamber, RUB/m;  $a_{mn}$  is cost to construct receiving sites, inclines, and chambers maintaining longwall, RUB;  $m$  is mining seam thickness, m;  $\gamma$  is coal density, t/m<sup>3</sup>. While mining complex-structure seam or seam with enclosing roof and floor undercut it is determined as weighted average value:

$$\gamma = \frac{\gamma_y m_y + \gamma_{nop} m_{nop}}{m} \quad (2.13)$$

where  $\gamma_y$ ,  $\gamma_{nop}$  are density of coal and undercut rocks respectively, t/m<sup>3</sup>;  $m_y$ ,  $m_{nop}$  are thickness of patch of coal and undercut rock respectively, m.

Expenditures connected with cargo haulage within belt roadway neighbouring a longwall are determined as follows:

$$C_2 = g\left(\frac{L}{2} + l_u\right) \quad (2.14)$$

where  $g$  is haulage cost of a ton of cargo for a meter, RUB.

For belt conveyors it can be roughly determined by [21] expression:

$$g = \left(b_1 + b_2\left(\frac{0.5L + l_u}{l_{л.к}}\right)\right) \frac{1}{A_c(0.5L + l_u)} + \frac{b_3}{A_c} + b_4 + b_5 \sin \alpha, \quad (2.15)$$

where  $b_1 \dots b_5$  are cost haulage parameters, RUB;  $\alpha$  is belt conveyor slope angle, degrees;  $A_c$  is daily output of a stope, t/days;  $l_{л.к}$  is length of belt conveyor, m.

To identify expenses connected with wages we propose the formula:

$$C_3 = \frac{R_1 + (n_m - 1)R_2)n}{N_s} + \frac{\sum P_i P_2 n}{rm\gamma c l} + \frac{lR_3}{l' A_c}, \quad (2.16)$$

where  $R_1$ ,  $R_2$ , and  $R_3$  are respectively full rates of coal-shearer operator, support worker of a stope, and machinery repairman involving all types of wage payments, RUB;  $n_m$  is normative labour intensity in terms of a system maintenance, manshifts;  $n$  is the number of coal-mining shifts per day;  $N_s$  is standard shift output per a system, t;  $\sum P_i$  is total labour intensity within connections of longwalls to perform a cycle, manshifts;  $l'$  is length of longwall site falling at machinery repairman, m. roughly it is  $l' = 50-70$  m [21].

Expenditures connected with amortization consist of sum of productions of value per machine per daily scale of amortization and reserve ratio. Finally, the expression has been developed:

$$C_4 = \frac{\sum_1^i n_{1i} U_{1i} \varphi_{1i} k_{p1i} + U_2 \varphi_2 k_{p2} l_{3ak} + \sum_1^i U_{3i} \varphi_{3i} k_{p3i} l}{A_c} \quad (2.17)$$

where  $U_{1i}$  is price per machine which cost is unaffected by longwall length, RUB. It includes cost of coal shearer, pump stations, face-end supports, heads of scraper, stowing machine etc.;  $U_2$  is cost of a meter of stowing pipeline, RUB;  $U_{3i}$  is cost of a meter of longwall equipment depending upon the longwall length. It consists of powered support, scraper conveyor, cable layer etc.;  $n_{1i}$  is the number of  $i^{\text{th}}$  production units which cost does not depend of longwall length;  $\varphi_{1i}$ ,  $\varphi_{2i}$ ,  $\varphi_{3i}$  are daily amortization of  $i^{\text{th}}$  equipment type, %;  $k_{p1}$ ,  $k_{p2}$ ,  $k_{p3}$  are reserve ratios of  $i^{\text{th}}$  equipment type;  $l_{3ak}$  is width of stowing track determined with the help of (2.9) and (2.11) expressions.

Expenses on materials involve only expenses on priority materials experiencing substantial changes when length of longwall varies: explosives, timber, and cutting tools of winning machine. They are determined by:

$$C_5 = \frac{a_{\text{ee}} \text{e}_{\text{ee}}}{m \gamma c l r} + \frac{a_{\text{л}} \text{e}_{\text{л}}}{m \gamma c l r} + \frac{a_3 \text{e}_3 (l - l_{\text{н}})}{l}, \quad (2.18)$$

where  $a_{\text{ee}}$  is consumption of explosives per a cycle of coal extraction, kg;  $a_{\text{л}}$  is consumption of timber per a cycle of coal extraction,  $\text{m}^3$ ;  $a_3$  is consumption of cutters per a ton of mined coal, pieces;  $\text{e}_{\text{ee}}$  is cost of a kilogram of explosives, RUB;  $\text{e}_{\text{л}}$  is cost of a cubic meter of timber, RUB;  $\text{e}_3$  is cost of a cutter, RUB;  $l_{\text{н}}$  total length of pockets, m.

Expenditures connected with dismantling activities are determined as follows:

$$C_6 = \frac{P_1 \text{e}_{\delta}}{m \gamma c l L} + \frac{P_2 \text{e}_{\delta}}{m \gamma c l}, \quad (2.19)$$

where  $P_1$  is mass of equipment in longwall and within its connections with a drift independent of the longwall length, t;  $P_2$  is mass of a meter of equipment in longwall depending upon the longwall length, t;  $\text{e}_{\delta}$  is average unit cost of the longwall equipment dismantling, RUB/t.

Expenditures connected with worked-out area stowing are determined using common empirical dependence [21]

$$C_7 = \frac{0.00083 A_{3ak} + 0,8 l_{3ak} + 88.4}{A_c}, \quad (2.20)$$

where  $A_{3ak}$  is daily output of stowing activities, t;  $l_{3ak}$  is stowing track width, m. It is determined with the help of (2.9) and (2.11) expressions.

Expenditures connected with maintenance of mine workings consist of expenditures connected with maintenance of air working and haulage road neighbouring the longwall. They are determined depending on agreed mining technique in accordance with below expressions being a result of mathematical transformations of known formulas [21]:

- For coal roads and boundary passways constructed with longwall advance:

$$C_{8e} = \frac{1}{m\gamma cl} \left( \frac{r_{1e} l_1 mcl}{305 A_c} + r_{2e} + r_{3e} + \frac{r_{4e} m\gamma cl L}{710 A_c} \right), \quad (2.21)$$

$$C_{8k} = \frac{1}{m\gamma cl} \left( \frac{r_{1k} l_1 mcl}{305 A_c} + r_{2k} + r_{3k} + \frac{r_{4k} m\gamma cl L}{710 A_c} \right);$$

- For the same mine workings driven subsequent to longwall:

$$C_{8e} = \frac{1}{m\gamma cl} \left( r_{3e} + \frac{r_{4e} m\gamma cl L}{710 A_c} \right), \quad (2.22)$$

$$C_{8k} = \frac{1}{m\gamma cl} \left( r_{3k} + \frac{r_{4k} m\gamma cl L}{710 A_c} \right);$$

- For coal roads and boundary passways being filled in terms of pillar mining:

$$C_{8e} = \frac{1}{m\gamma cl} \left( \frac{r_{1e} m\gamma cl}{305 A_c} + r_{2e} + r_{3e} \right), \quad (2.23)$$

$$C_{8k} = \frac{1}{m\gamma cl} \left( \frac{r_{1k} m\gamma cl}{305 A_c} + r_{2k} + r_{3k} \right);$$

- For the same mine workings being maintained for reutilization:

$$C_{8e} = \frac{1}{m\gamma cl} \left( \frac{r_{1e} m\gamma cl}{305 A_c} + r_{2e} + r_{3e} + \frac{r_{4e} m\gamma cl L}{710 A_c} \right),$$

$$C_{8k} = \frac{1}{m\gamma cl} \left( \frac{r_{1k} m\gamma cl}{305 A_c} + r_{2k} + r_{3k} + \frac{r_{4k} m\gamma cl L}{710 A_c} \right); \quad (2.24)$$

where  $l_l$  is minimum reserve of drift advance in front of longwall stope behind a zone of temporary reference pressure, m;  $r_{1e}$ ,  $r_{1k}$  are expenditures connected with maintenance of a meter of airway working and belt roadway in the context of coal mass and rock mass respectively, RUB;  $r_{2e}$ ,  $r_{2k}$  are expenditures connected with retimbering of a meter of airway working or belt roadway respectively in front of a stope within a zone of temporary reference pressure during the whole period of its location in the zone, RUB;  $r_{3e}$ ,  $r_{3k}$  are expenditures connected with retimbering of a meter of mine working behind a stope face within a zone of mining effect during the

whole period of its location in the zone, RUB;  $r_{4e}, r_{4k}$  are expenditures connected with annual maintenance of mine working meter within worked-out area out of zone of mining effect, RUB.

Total expenditures connected with maintenance of airway working as well as belt roadway are:

$$C_8 = C_{8e} + C_{8k} \quad (2.25)$$

Capital expenditures consist of expenditures connected with purchase and installation of transporting, heading, and mining facilities. They are determined by:

$$K = K_1 + K_2, \quad (2.26)$$

where  $K_1$  is expenditures connected with equipment purchasing, RUB.;  $K_2$  is expenditures connected with equipment erecting, RUB.

Expenditures connected with equipment purchasing are calculated as follows:

$$K_1 = \frac{1}{m\gamma c l L} \left( \sum_1^n n_{1i} U_{1i} + U_{3.m} l_{3ak} + \sum_1^m U_{2i} l \right) \quad (2.27)$$

where  $n_{1i}$  is the number of  $i^{\text{th}}$  type transporting, heading, and mining production units which cost is independent of longwall length, RUB;  $U_{1i}$  is a cost of  $i^{\text{th}}$  type transporting, heading, and mining production unit independent of longwall length, RUB;  $U_{3.m}$  is a cost of a meter of stowing pipeline, RUB;  $U_{2i}$  is cost of a meter of  $i^{\text{th}}$  type equipment which depends on longwall length, RUB.

To identify expenditures connected with equipment installation, following expression has been proposed:

$$K_2 = \frac{1}{m\gamma c l L} \left( P_1 \epsilon_m + P_n \epsilon_n + P_m \epsilon_m (L + l_y) + P_{3.m} \epsilon_m l_{3ak} + P_2 \epsilon_m l \right) \quad (2.28)$$

where  $P_n$  is mass of heading facilities, t;  $P_m$  is mass of transporting facilities in longwall neighbouring haulage roadway, t;  $\epsilon_m$  is specific cost of installation activities in terms of longwall equipment and its matching joints, RUB/t;  $\epsilon_n$  is specific cost of installation activities in terms of heading equipment, RUB/t;  $\epsilon_m$  is specific cost of activities connected with transportation facilities installation, RUB/t.

Determine total cost with the help of the known expression:

$$S_{np} = \sum_1^8 C + EK \quad (2.29)$$

where  $E$  is standard branch coefficient of investment efficiency.

Daily output of a stope being a part of (2.15) - (2.17) and (2.20) - (2.24) expressions are determined by the known dependence [59]:

$$A_c = nT_{cm}qk_mC, \quad (2.30)$$

where  $n$  is the number of coal mining shifts per day;  $T_{cm}$  is a shift period, min;  $C$  is coal extraction coefficient;  $q$  is a minute's output of a coal sharer efficiency, t/min. It is determined as follows:

$$q = m\gamma rV \quad (2.31)$$

where  $V$  is feed velocity of winning machine, m/min;  $K_M$  is machine time coefficient.

To identify machine time coefficient of a longwall, (2.39) expression is applied where equipment availability coefficient  $k_{z.o6}$  is determined by the formula:

$$k_{z.o6} = \frac{1}{k_1 + k_2l} \quad (2.32)$$

where  $k_1$  and  $k_2$  are empiric coefficients involving changes in a longwall equipment availability coefficient on its length in terms of various types of mining facilities. According to experimental results by Dnipropetrovsk Mining Institute engineer B.N. Shabratski obtained for Western Donbass mines  $k_1 = 2.02$  and  $k_2 = 0.0023$  for such systems as КД-80 and KM103, and  $k_1 = 2.67$  and  $k_2 = 0.0064$  for longwalls with "Donbass" KMK97, and KM87 systems.

Table 2.4 Initial data to calculate optimum longwall length

Parameters	Variations			
	1	2	3	4
Type of powered system	KM88	KM88	МКД3-90	МКД3-90
Type of winning machine	1K101Y	1K101Y	KA80	1KM103
Web width, m	0.63	0.63	0.8	0.8
Geological seam thickness, m	0.8	0.8	0.8	0.8
Extracting seam thickness, m	1.2	1.2	1.0	1.0
Mining technique	Long-pillar	Combined	Long-pillar	Combined
Seam mining technique	Complete	Selective	Complete	Selective
Finished section of airway working, m <sup>2</sup>	8.2	11.7	11.7	11.7
Finished section of haulage drift, m <sup>2</sup>	8.2	reuse	reuse	reuse
Roof control technique	Complete caving	Partial stowing	Complete stowing	Complete stowing
Type of stowing machine	-	"Titan-1"	ZS-240	ZS-240

A problem of determination of effective length of longwall is solved by means of calculation of total cost in terms of different lengths of longwall varying incrementally within the prescribed limits. That length of longwall is optimum when total cost is optimum.

Optimum length of longwall has been calculated for representative conditions of longwalls in Western Donbass. Four alternatives have been analyzed. They differ in production technique (either complete mining or selective one); a technique being applied; type of mining and stowing equipment; type of roof control and other parameters (Table 2.4).

Tables 2.5 and 2.6 cover calculation results in terms of every alternative. Fig. 2.13 explains graphically dependence of total cost in a stope on length of a longwall.

Analysis of the calculation results shows that changes in length of a longwall involve significant changes in output, operating cost, and investment; moreover, depending upon increase in length of longwall certain factors experience increase while others experience their decrease.

Daily average of output in terms of complete seam mining with the help of KM88 system reaches its maximum (596 tons) if length of longwall is 140-160 m. In the context of further increase in length of longwall, daily average of output drops due to loss in mining equipment reliability. In terms of selective mining maximum daily average output (222 tons) is achieved when length of longwall is more than 240 m. It can be explained by the fact that time consumed to mine out rock bench considered as idle time of a stope extends proportionally to length of longwall.

When length of longwall increases up to specified value, total cost experiences its decrease with following increase (Fig. 2.13). Inflection point of the dependence is adequate to that length of longwall when minimum production cost per a ton of coal is provided.

In terms of complete seam mining and pillar mining (alternative 1) minimum value of total cost being equal to RUB 4.53 per ton is adequate to 215 m length of longwall; in terms of selective mining and combined mining (alternative 2) minimum cost equal to RUB 8.50 per ton will be available if length of longwall is 230 m.

In the context of МКДЗ - 90 system application and complete stowing of worked-out area, optimum length of longwall in terms of complete mining (alternative 3) will be 175 m; if selective sea mining is applied (alternative 4) then the figure will be 200 m. Minimum total cost will be equal to RUB 5.97 per ton and RUB 8.94 per ton respectively. In the context of optimum length of longwall average output per stope will be 780 t in terms of complete mining and 523 t in terms of selective one.

Thus, application of МКДЗ - 90 system when complete stowing of worked-out area is performed longwall should be some shorter to compare with application of KM88 system. It depends on the fact that when МКДЗ - 90 system is applied

amortization cost experiences significant increase due to its high cost to compare with standard machine systems being less expensive.

Moreover, the results make it possible to state that in terms of selective seam mining technique application it is expedient to lengthen longwall by 10-15 % to compare with length of longwall where complete mining technique is applied.

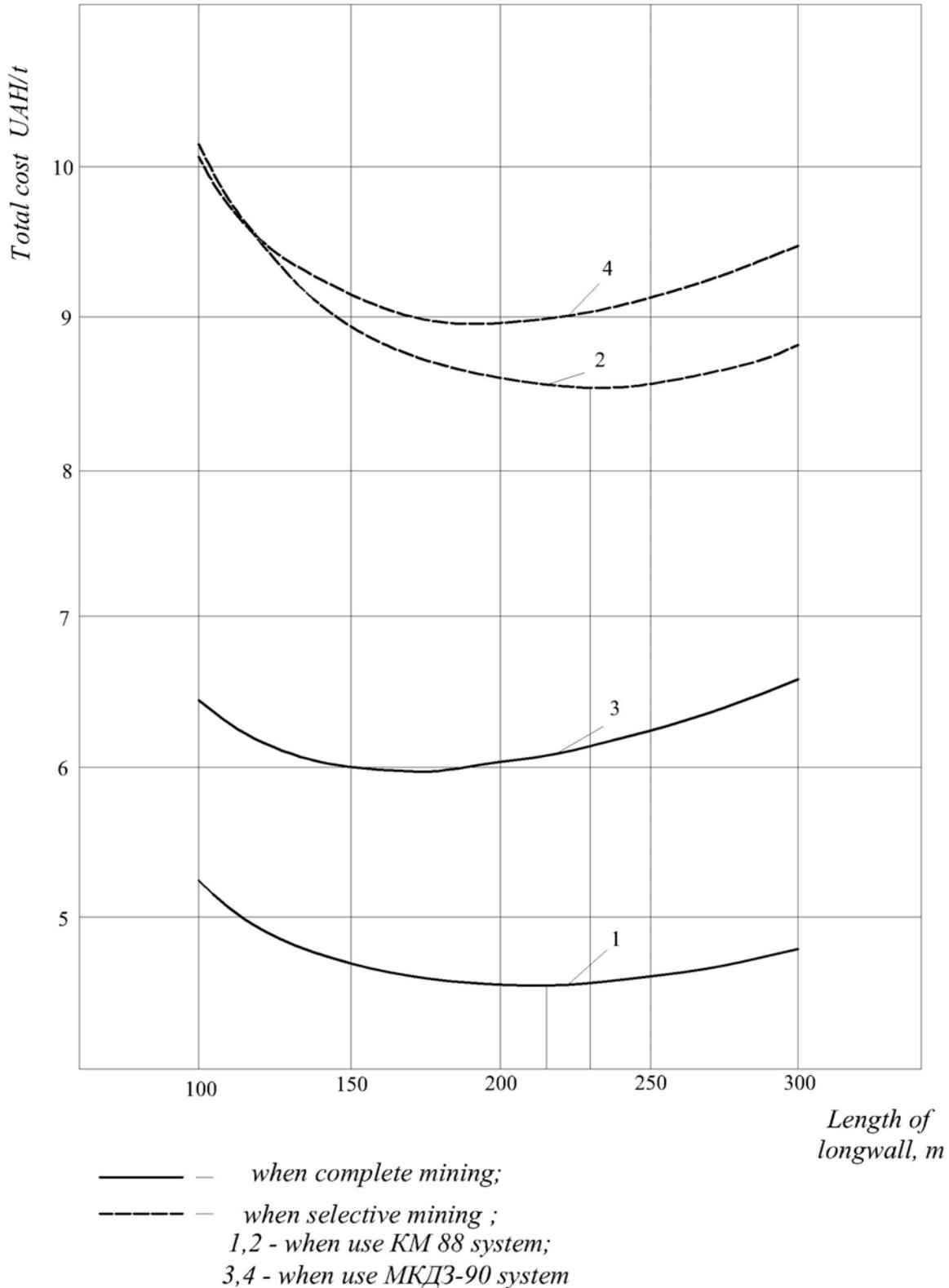


Fig. 2.13 Dependence of total cost of length of longwall

Table 2.5 Dependence of basic cost-performance ratios of a stope equipped with KM 88 system of length of longwall

Length of longwall, m	Daily output, t/day	Expenses, RUB/t								Investment, RUB/t	Total cost, RUB/t
		Construction of mine workings	Hauling	Salary	Amortization	Materials	Equipment disassembly	Stowing	Maintenance		
100	<u>575*</u>	<u>1.360</u>	<u>0.264</u>	<u>2.306</u>	<u>0.507</u>	<u>0.324</u>	<u>0.014</u>	=	<u>0.086</u>	<u>2.417</u>	<u>5.223</u>
	185	1.632	0.813	2.777	1.579	0.367	0.031	0.793	1.511	5.763	10.314
120	<u>589</u>	<u>1.137</u>	<u>0.258</u>	<u>2.240</u>	<u>0.559</u>	<u>0.320</u>	<u>0.014</u>	=	<u>0.076</u>	<u>2.134</u>	<u>4.931</u>
	196	1.368	0.756	2.668	1.678	0.357	0.030	0.695	1.274	5.032	9.590
140	<u>595</u>	<u>0.978</u>	<u>0.255</u>	<u>2.209</u>	<u>0.617</u>	<u>0.318</u>	<u>0.014</u>	=	<u>0.066</u>	<u>1.931</u>	<u>4.747</u>
	205	1.180	0.734	2.596	1.793	0.349	0.029	0.666	1.105	4.510	9.128
160	<u>595</u>	<u>0.859</u>	<u>0.255</u>	<u>2.160</u>	<u>0.680</u>	<u>0.316</u>	<u>0.013</u>	=	<u>0.060</u>	<u>1.778</u>	<u>4.633</u>
	211	1.038	0.712	2.548	1.919	0.344	0.028	0.647	0.979	4.119	8.833
180	<u>592</u>	<u>0.766</u>	<u>0.256</u>	<u>2.165</u>	<u>0.747</u>	<u>0.315</u>	<u>0.013</u>	=	<u>0.056</u>	<u>1.660</u>	<u>4.568</u>
	215	0.928	0.698	2.515	2.054	0.339	0.028	0.634	0.881	3.814	8.650
200	<u>587</u>	<u>0.692</u>	<u>0.259</u>	<u>2.153</u>	<u>0.818</u>	<u>0.314</u>	<u>0.013</u>	=	<u>0.052</u>	<u>1.565</u>	<u>4.536</u>
	218	0.840	0.688	2.494	2.198	0.336	0.027	0.625	0.803	3.571	8.547
220	<u>579</u>	<u>0.632</u>	<u>0.262</u>	<u>2.145</u>	<u>0.894</u>	<u>0.313</u>	<u>0.013</u>	=	<u>0.049</u>	<u>1.487</u>	<u>4.531</u>
	220	0.768	0.682	2.482	2.348	0.333	0.027	0.619	0.740	3.371	8.504
240	<u>571</u>	<u>0.581</u>	<u>0.266</u>	<u>2.141</u>	<u>0.973</u>	<u>0.312</u>	<u>0.013</u>	=	<u>0.047</u>	<u>1.423</u>	<u>4.546</u>
	222	0.708	0.678	2.476	2.505	0.330	0.027	0.615	0.687	3.205	8.507
260	<u>562</u>	<u>0.538</u>	<u>0.270</u>	<u>2.140</u>	<u>1.055</u>	<u>0.312</u>	<u>0.013</u>	=	<u>0.045</u>	<u>1.368</u>	<u>4.578</u>
	222	0.657	0.676	2.475	2.668	0.328	0.026	0.614	0.643	3.060	8.547
280	<u>552</u>	<u>0.502</u>	<u>0.274</u>	<u>2.141</u>	<u>1.142</u>	<u>0.311</u>	<u>0.013</u>	=	<u>0.044</u>	<u>1.321</u>	<u>4.624</u>
	222	0.614	0.676	2.478	2.836	0.327	0.026	0.614	0.605	2.944	8.617
300	<u>543</u>	<u>0.470</u>	<u>0.279</u>	<u>2.144</u>	<u>1.231</u>	<u>0.311</u>	<u>0.013</u>	=	<u>0.043</u>	<u>1.281</u>	<u>4.683</u>
	222	0.576	0.677	2.484	3.010	0.325	0.026	0.614	0.572	2.840	8.712

\* - complete mining is in numerator (alternative1), selective mining is in denominator (alternative 2)

Table 2.6 Dependence of basic cost-performance ratios of a stope equipped with МКДЗ - 90 system on length of longwall

Length of longwall, m	Daily output, t/day	Expenses, RUB/t										Investment, RUB/t	Total cost, RUB/t
		Construction of mine workings	Haulage	Salary	Amortization	Materials	Equipment disassembly	Stowing	Maintenance				
100	<u>627*</u> 375	<u>1.113</u>	<u>0.242</u>	<u>2.001</u>	<u>1.399</u>	<u>0.143</u>	<u>0.035</u>	<u>0.269</u>	<u>0.136</u>	<u>7.215</u>	<u>6.421</u>		
		1.632	0.403	2.151	2.350	0.337	0.052	0.450	1.213	10.594	10.178		
120	<u>681</u> 412	<u>1.933</u>	<u>0.223</u>	<u>2.963</u>	<u>1.500</u>	<u>0.139</u>	<u>0.035</u>	<u>0.271</u>	<u>0.117</u>	<u>6.596</u>	<u>6.171</u>		
		1.368	0.366	2.095	2.484	0.332	0.051	0.448	1.016	9.685	9.612		
140	<u>724</u> 443	<u>0.805</u>	<u>0.210</u>	<u>2.937</u>	<u>1.611</u>	<u>0.136</u>	<u>0.034</u>	<u>0.277</u>	<u>0.104</u>	<u>6.154</u>	<u>6.037</u>		
		1.180	0.341	2.056	2.634	0.328	0.050	0.453	0.875	9.035	9.272		
160	<u>757</u> 469	<u>0.708</u>	<u>0.201</u>	<u>1.919</u>	<u>1.730</u>	<u>0.134</u>	<u>0.034</u>	<u>0.286</u>	<u>0.094</u>	<u>5.823</u>	<u>5.979</u>		
		1.038	0.322	2.029	2.796	0.325	0.050	0.462	0.770	8.547	9.074		
180	<u>783</u> 491	<u>0.633</u>	<u>0.195</u>	<u>1.906</u>	<u>1.855</u>	<u>0.133</u>	<u>0.034</u>	<u>0.297</u>	<u>0.086</u>	<u>5.575</u>	<u>5.972</u>		
		0.928	0.309	2.010	2.967	0.322	0.049	0.474	0.688	8.169	8.973		
200	<u>804</u> 508	<u>0.573</u>	<u>0.190</u>	<u>1.189</u>	<u>1.986</u>	<u>0.131</u>	<u>0.033</u>	<u>0.306</u>	<u>0.080</u>	<u>5.359</u>	<u>6.003</u>		
		0.840	0.298	1.996	3.146	0.321	0.049	0.489	0.622	7.866	8.941		
220	<u>820</u> 523	<u>0.524</u>	<u>0.186</u>	<u>1.890</u>	<u>2.122</u>	<u>0.130</u>	<u>0.033</u>	<u>0.323</u>	<u>0.075</u>	<u>5.190</u>	<u>6.061</u>		
		0.768	0.290	1.986	3.332	0.319	0.049	0.506	0.569	7.617	8.962		
240	<u>833</u> 535	<u>0.483</u>	<u>0.183</u>	<u>1.886</u>	<u>2.263</u>	<u>0.129</u>	<u>0.033</u>	<u>0.337</u>	<u>0.070</u>	<u>5.050</u>	<u>6.143</u>		
		0.708	0.283	1.979	3.524	0.318	0.049	0.524	0.525	7.410	9.022		
260	<u>842</u> 546	<u>0.448</u>	<u>0.181</u>	<u>1.883</u>	<u>2.409</u>	<u>0.129</u>	<u>0.033</u>	<u>0.352</u>	<u>0.067</u>	<u>4.931</u>	<u>6.242</u>		
		0.657	0.278	1.975	3.723	0.317	0.048	0.544	0.487	7.236	9.114		
280	<u>849</u> 554	<u>0.419</u>	<u>0.180</u>	<u>1.882</u>	<u>2.559</u>	<u>0.128</u>	<u>0.033</u>	<u>0.368</u>	<u>0.064</u>	<u>4.829</u>	<u>6.357</u>		
		0.614	0.274	1.972	3.926	0.316	0.048	0.565	0.455	7.085	9.233		
300	<u>854</u> 561	<u>0.393</u>	<u>0.179</u>	<u>1.881</u>	<u>2.713</u>	<u>0.128</u>	<u>0.033</u>	<u>0.385</u>	<u>0.062</u>	<u>7.740</u>	<u>6.485</u>		
		0.576	0.270	1.971	3.135	0.315	0.048	0.586	0.427	6.956	9.373		

\* - complete mining is in numerator (alternative 3), selective mining is in denominator (alternative 4)

## 2.4. Operating conditions of the technique

Stowing activities exercise a significant influence on operating conditions of a stope. Feed velocity of winning machine (especially in terms of selective coal and rock mining), coefficient of machine time of a stope as well as its efficiency are among them. It depends on the fact that processes of mining and stowing are time and space integrated system; that is why changes in one parameter factor into changes in another one.

Measurements of coal-shearer feed velocities performed by us under various conditions show that following empiric dependence is the most reliable to describe the parameter [6]:

$$V = \frac{Pt_{pe3}}{mrA} - 0.2V_{pe3} , \quad (2.33)$$

where  $P$  is total power consumed by a coal shearer, kW;  $t_{pe3}$  is a pitch between operating cutting lines, cm;  $m$  is mined seam thickness, m;  $r$  is web width, m;  $A$  is resistance of a seam to breaking down while cutting, kN/m;  $V_{pe3}$  is cutting velocity, m/s.

The expression can be recommended to calculate feed velocity in terms of complete (combined) coal and rock mining as well as terms of selective coal mining technique. In the context of complete mining with enclosed roof and floor undercut coal seam resistance to cutting  $A$  is determined as weighted mean:

$$A = \frac{A_y m_y + A_{nop} m_{nop}}{m} , \quad (2.34)$$

where  $A_y$  is coal seam resistance to cutting, kN/m;  $A_{nop}$  resistance of undercut rock to cutting, kN/m;  $m_y$  is coal seam thickness, m;  $m_{nop}$  is enclosed roof and floor undercut value, m.

To calculate feed velocity while patch of coal mining in terms of selective technique, thickness of coal patch  $m_y$  is substituted into (2.33) expression instead of mined seam thickness  $m$ .

To calculate feed velocity while rock mining, paper [37] recommends using expression involving rock bench damping at the expense of advance coal mining:

$$V_n = \frac{Pt_{pe3}}{m_{nop} r A_{nop} k_{ocл}^n} - 0.2V_{pe3} , \quad (2.35)$$

where  $k_{ocл}^n$  is rock damping factor.

The same work mentions that while using selective seam mining in terms of Western Donbass, feed velocity slightly depends of undercut thickness being able to achieve its maximum.

In the context of selective coal and rock mining when the latter is stowed momentary efficiency of coal shearer in terms of rock cannot be more than momentary efficiency of crusher or stowing machine; i.e.

$$Q_{3.0} \geq m_{nop} V_{nop}^{max} r k_{h.n} k_{p.n} , \quad (2.36)$$

where  $Q_{3.o}$  is momentary efficiency of stowing equipment,  $m^3/\text{min}$ ;  $k_{h.n}$  is coefficient involving incompleteness of coal loading; and  $k_{p.n}$  is coefficient of rock loosening.

Hence determine maximum feed velocity of coal shearer in terms of rock bench mining:

$$V_{nop}^{\max} = \frac{Q_{3.o}}{m_{nop} r k_{h.n} k_{p.n}} \quad (2.37)$$

Finally, in terms of rock mining feed velocity is:

$$V_n = \min \left\{ \frac{Pt_{pez}}{m_{nop} r A_{nop} k_{ocn}^n} - 0.2V_{pez}; \frac{Q_{o.3}}{m_{nop} r k_{h.n} k_{p.n}}; V_{pa6}^{\max} \right\}, \quad (2.38)$$

where  $V_{pa6}^{\max}$  is maximum operational speed of coal shearer,  $m/\text{min}$ .

Calculation performed for mines in Western Donbass helps determine that feed velocities of 1K103 and 1K101 coal shearers for rock mining are not limited by their capacity achieving maximum operation speed if ZS-240, ПЗБ-200 and ПЗБ-250 stowing machines are applied. Feed velocity of coal shearers is limited if only low-production stowing equipment is used. It concerns crushing and stowing system "Titan-1" when undercut thickness is more than 0.14-0.19 m (Fig. 2.13). If thickness of undercut rocks is in excess of the value, then feed velocity of winning machine decreases in accordance with hyperbolic law.

A number of papers [44, 88] propose expressions to determine operating conditions of a stope on "stowing operations" factor. The proposed techniques don't involve details of undercut rocks stowing into worked-out area.

A.G. Koshka represents details of selective seam mining in terms of machine time coefficient determination in the following expression [37]:

$$k_M = \frac{l}{\frac{l}{k_{pa6}} + (T_{k.o} + T_{3.o})V + \frac{lV}{V_n}}, \quad (2.39)$$

where  $l$  is length of longwall,  $m$ ;  $k_{2.o6}$  is equipment availability;  $T_{ko}$  is time consumed for final operations,  $\text{min}$ ;  $T_{3.o}$  is time to eliminate operational problems (downtime) unrelated to coal shearer operation,  $\text{min}$ ;  $V$  is feed velocity of coal shearer in terms of coal mining,  $m/\text{min}$ ;  $V_n$  is feed velocity of coal shearer in terms of rock mining,  $m/\text{min}$ .

Application of selective seam mining along with worked-out area stowing in (2.39) expression should involve time consumed for side rock caving or dismantlement of stowing pipe sections in terms of sublevel rock caving.

Thus, the expression will be:

$$k_M = \frac{1}{\frac{1}{k_{2.o6}} + (T_{k.o} + T_{3.o} + \frac{l}{k_{2.3}V_n} + \frac{t_{M.m}l_3}{l_m})\frac{V}{l}}, \quad (2.40)$$

where  $k_{2,3}$  is stowing equipment availability;  $t_{mm}$  is time consumed to control side caving or to dismantle a section of stowing pipe, min;  $l_3$  is width of stowing track, m;  $l_T$  is distance between rock caving, m.

Availability of mining equipment and stowing equipment can be determined on the known expressions [59]:

$$k_{2,06} = \frac{1}{1 + \sum_{i=1}^{n_{04}} \left( \frac{1}{M_{04,i}} - 1 \right)}; \quad k_{2,3} = \frac{1}{1 + \sum_{i=1}^{n_3} \left( \frac{1}{M_{3,i}} - 1 \right)}; \quad (2.41)$$

where  $n_{04}$  and  $n_3$  is the number of mining equipment and stowing equipment, respectively;  $M_{04,i}$  and  $M_{3,i}$  are coefficients of availability of  $i^{\text{th}}$  type of mining equipment and stowing equipment, respectively.

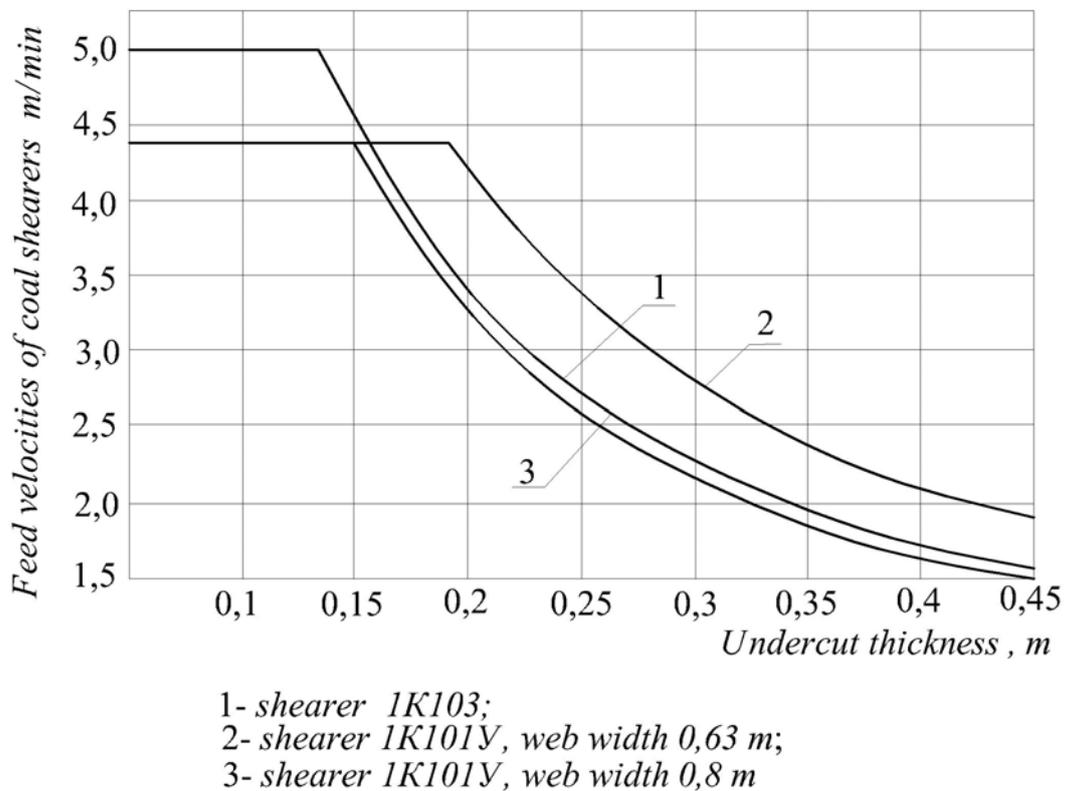


Fig. 2.14 Dependence of feed velocities of coal shearers on undercut thickness in terms of undercut rock stowing with the help of "Titan-1" system

$V_n$  and  $l_3$  values being a part of (2.40) expression depend on undercut rock thickness; therefore, after all machine time coefficient also depends on the parameter. Fig. 2.14 demonstrates calculated dependence of machine time coefficient on undercut rock thickness when various mining and stowing facilities are used.

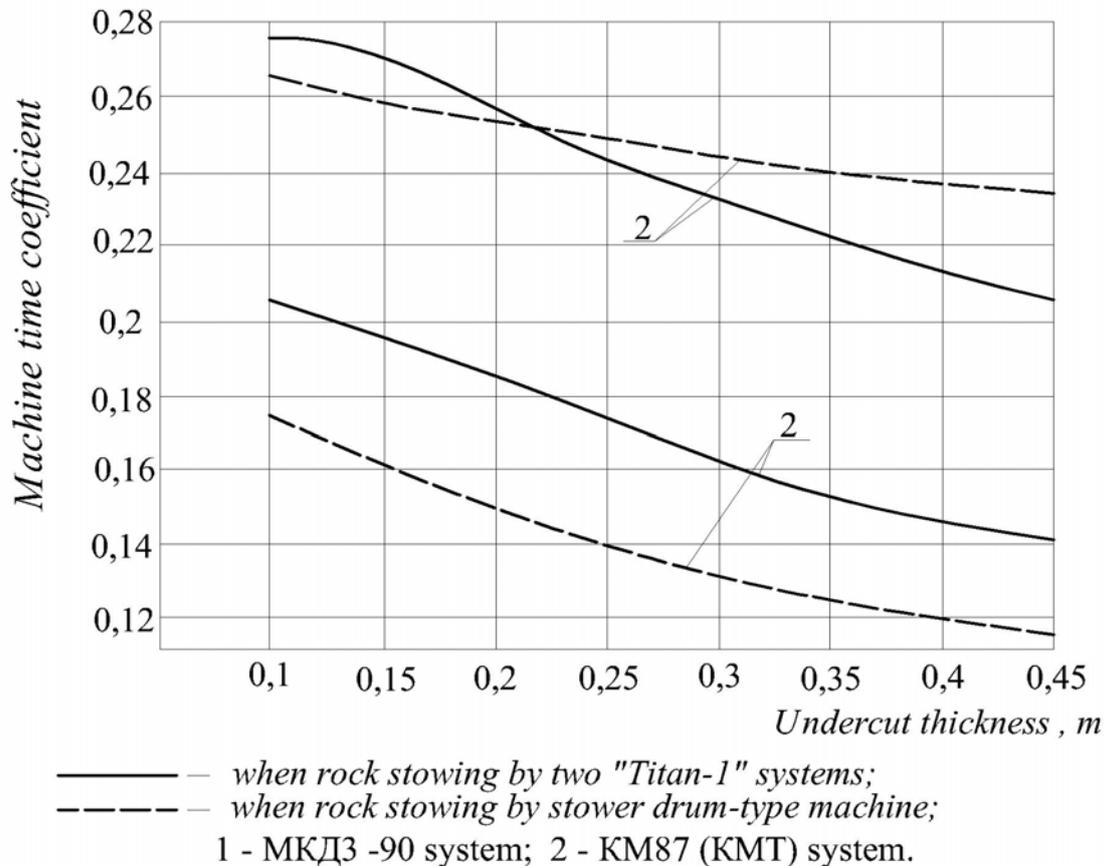


Fig. 2.15 Dependence of machine time coefficient on undercut thickness

It follows from Figures 2.14 and 2.15 that increase in undercut rock thickness factors into decrease in machine time coefficient owing to increase in stowing operations volume and decrease in feed velocity of coal-shearer in terms of rock mining. It should be noted that in the context of MKД3 - 90 system application the parameter is higher to compare with KM-87 (88) and KMT systems.

## Conclusions

- Analytical research of rock mass stress and strain state helps determine that:
  - Worked-out area stowing results in significant narrowing (down to 20-40 %) of index zone to compare with complete caving and decrease in bearing load on coal seam;
  - In terms of worked-out area stowing subsidence, horizontal displacements and deformations of rock stratum one experience almost double decrease; as a result, probability of roof discontinuity as well as formation of rock caving in it reduces;
  - While stowing with longwall extension standard loads, subsidence and horizontal displacements experience less intensive increase to compare with complete roof caving; if it is required length of longwall may be extended up to cost-effective;

- Load on stowing mass at the distance of 200 m after 300 days of mining activities completion is almost 70 % of loads within virgin rock;

- On the context of worked-out area stowing earth's subsidence experiences almost 50% drop; moreover, displacement through size also reduces.

2. Dependence of width of pack in terms of undercut rock stowing into worked-out area on enclosed roof and floor undercut thickness as well as on mined seam thickness has been determined; in contrast to common ones it involves increase in stowing material volume at the expense of coal availability in it as a result of selective seam mining. That makes it possible to make more accurate calculation of geometrical parameters of operation schedules involving use of selective mining technique.

3. The dependence of changes in total cost on length of longwall helps conclude that when mining and stowing system МКДЗ - 90 performing complete stowing of worked-out area is applied length of longwall should be about 175 m in terms of complete mining, and 200 m in terms of selective technique.

## **CHAPTER 3. UNDERGROUND INVESTIGATIONS OF PARAMETERS OF THIN SEAM MINING TECHNIQUE WITH WORKED-OUT AREA STOWING**

### **3.1. General remarks**

To validate basic theoretical aspects and determine actual parameters to extract thin flat seams with stowing in terms of different mining methods, a number of full-scale investigations were carried out in “Blagodatnaia” mine. Moreover, the underground investigations were aimed at performance verification of the proposed technique and determination of its basic cost/performance ratios.

The research followed the demands of specially developed technique involving specifications and ground rules of known industrial practices.

### **3.2. State of conditions and test site of the technique**

The technique was tested in “Blagodatnaia” mine. Two its longwalls (746-bis and 719) of  $C_7$  seam were involved.

Sites of mine field with extremely unfavourable mining and geological conditions were taken. Within working areas  $C_7$  seam was of double-patch structure (Fig. 3.1); in addition it was separated with interlayer of rather unstable argillite. Thickness of that parting was 0.7-1.2 m in 746-bis longwall and 2.0-2.8 m in 719 longwall. Within both mine sections thickness of upper coal patch ( $C_7^B$  seam) was 0.4-0.5 m and 0.7 -0.78 m ( $C_7^H$  seam) of lower one. Only  $C_7^H$  seam was mined with floor rock undercut; that is why lower layer of rock seam between coal patches with 0.1-0.5 thickness was false roof caved after pass of coal shearer. That complicated both mining activities and stowing ones. Main roof consists of free-caving argillite ( $f = 1.2-1.5$ ) and lens-shaped sandstone of mean hardness ( $f = 3-4$ ). Immediate roof of the seam is unstable argillite ( $f = 0.8-1.0$ ); humidification results in its slaking and intensive heaving.

In addition to unstable enclosed roof and floor stoping was complicated by availability of high rock pressure zones as mining took place within overlying seams.

### **3.3. Characteristic of subject of research**

Technique of thin flat seam  $C_7$  mining with partial stowing of worked out area has been a subject of the research. Both 746-bis and 719 longwalls were involved.

Powered system 1KM88 and 1K101Y coal shearer performed mining of the both longwalls reserves. Crushing and stowing system “Titan-1” performed stowing operations.

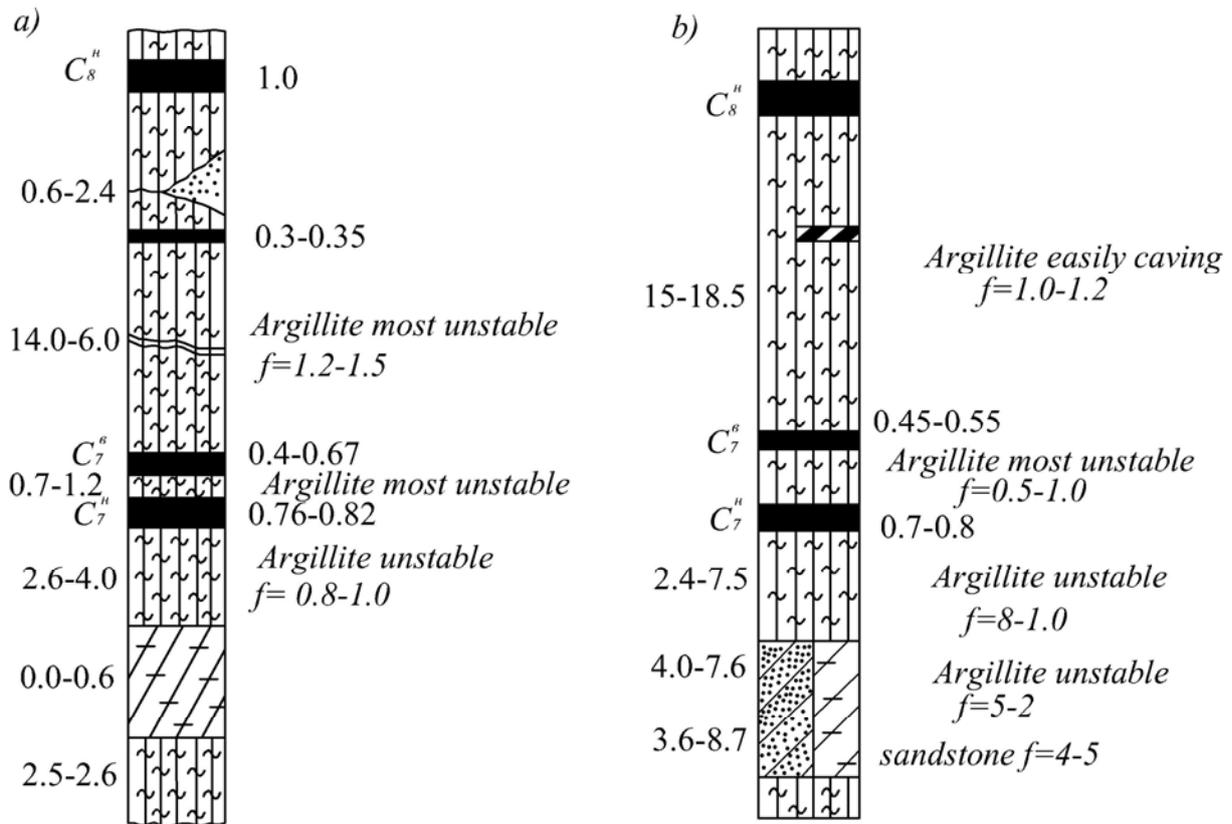


Fig.3.1 Stratigraphic columns of  $C_7$  seam within 746 – bis (a) and 719 (b) longwall blocks

To perform stowing operations from goaf side overhanging canopy was rigidly fastened to the floors of powered system sections. It has been done because of unstable roof to protect stowing mass construction site against caving of direct roof rocks. Length of overhanging canopy in longwall 746-bis was 0.9 m; in longwall 719 it was lengthened up to 1.2 m.

Mine section of 746-bis longwall was mined using pillar system. Selective seam mining was applied. Pass one of a coal shearer from belt entry to boundary one mined coal seam with 0.75-0.78 m thickness. Sections of powered support simultaneously followed the coal shearer. In terms of back pass (from boundary entry to belt one) rock bench of 0.4-0.45 m depth left in the seam floor was mined. Undercut floor rocks were used as initial material for pneumatic stowing. Longwall conveyor operated in reversing mode haulaged rocks to a hopper of “Titan-1” system through chain feeder. Then they were stowed into worked-out area of the longwall.

Width of stowing track was limited by standard value of haulage of the crushing and stowing system; that is why width of pack was not more than 30 m.

Mounting/dismounting scheme for stowing pipeline developed by members of “Dneprogiproshakht” Institute excluded continuous presence of staff within worked-out area while stowing; that closed out injuries as a result of direct roof rock caving. Pipeline was placed on a seam floor under back overhangs of powered system. While worked-out area stowing the pipeline shortened as one of pipes in boundary entry was dismantled. Special wrench hoisted away left sections of pipeline to entry. There the sections were connected to a pipe angle of main pipeline. It followed by worked-out area stowing at the distance equal to one pipe length.

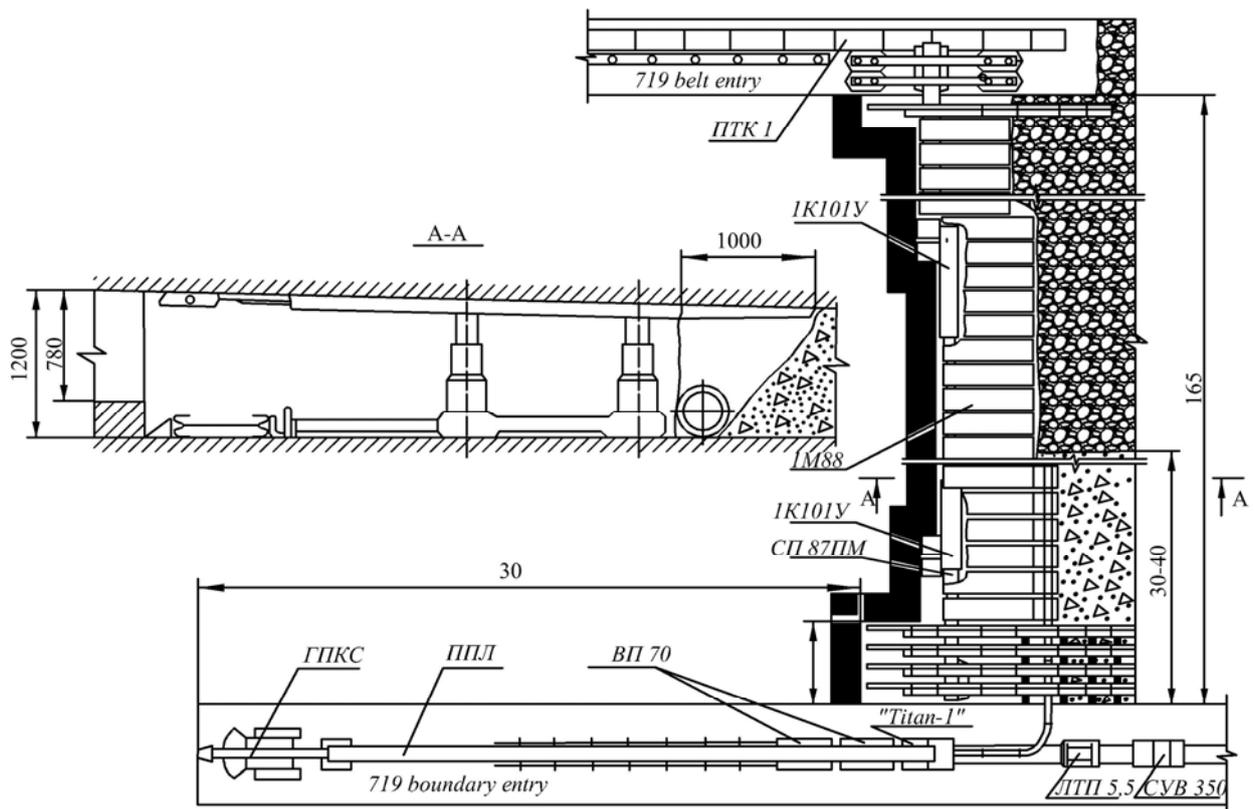
Stowing pipeline was mounted before stowing activities by means of displacement of the pipeline alongside a longwall with gradual pipe connection within entry.

To carry out functional test of the technique in terms of combined mining technique, rock leaving within worked-out area, and substantiation of its reuse possibility, the technique was evaluated in longwall 719. Boundary entry with 14.3 m cross-section was constructed together with longwall but with 30 m advance. Rock left after mining as well as rock left after rock undercut in longwall was stowed into worked-out area. Fig. 3.2 demonstrates operation schedule applied in longwall 719.

To increase air pressure in stowing pipeline, and therefore to extend range of raw material transportation and to densify stowing mass two paralleled wind blowers BII-70 were used as air supply sources. That made it possible to widen pack up to 45-50 m.

To lighten activities connected with stowing mass construction in longwall 719 pipeline by Skochinski IM design was applied. The pipeline involved lateral discharge of rock which was remotely controlled from working space of longwall.

Test of described above operation schedules analyzed actual parameters of coal and rock mining, parameters of stowing mass construction, rock pressure manifestation within working space of longwall, stowing mass, and neighbouring mine workings. Partial stowing of worked-out area in pilot longwalls helped compare parameters obtained in a zone of stowing mass construction with parameters of traditional technique application in terms of seam mining in Western Donbass where immediate roof rock complete caving was involved.



*Fig. 3.2 Operation schedule for 719 longwall*

### 3.4. Principles of the research

Methods of underground investigations take into account specifications regulating organizational and procedural as well as normative-technical basis of official test of product; nature of powered support and lateral rocks interaction; immediate roof rock inrush formation; changes in values of lateral rock displacement within working space of longwall as well as within stowing mass; and actual resistance of hydraulic props. All above parameters measured in a zone of goaf pack construction were compared with similar parameters in that part of longwall where complete caving was used.

Keeping watch over inrush of roof rocks was of periodic nature; it took part during the whole time of the research. In this context both depth and volume of the inrushes was registered while measuring their geometry.

Determination of enclosing roof and floor approaching within working area was performed by means of gage posts СУИ-II as follows: distances between benchmarks forced into roof and floor of a longwall were measured. To determine approaches of enclosing roof and floor within stowing mass specific gage rheostatic posts were placed between the benchmarks; their readings were entered in neighbouring mine working.

In terms of the parameters measuring, sampling frequency depended on enclosing roof and floor displacement as well as a stope advance.

Recording manometers M66A connected into work cavities of front and back posts measured actual resistance of hydraulic props of M88 powered support sections from initial thrust up to resistance before unloading process started. Measuring points were available both within sections with extended ceilings (that is within an area of goaf pack building) and to be compared in standard sections within a zone of complete caving. Volume of the measuring was not less than one hundred winning cycles or sixty-five meters of a stope advance according to “Methodological instructive regulations on study of rock pressure in terms of coal mines and shaly mines” by AURSI and “Temporary provision on the development of program and techniques for governmental tests of mining equipment with powered supports” by A.A. Skochinski IM and “Giprouglesh” approved by Engineering Office of Ministry of Coal Industry of the USSR.

Associativity and efficiency of networking of mining equipment, stowing equipment in addition to heading equipment in longwall 719 was estimated by means of comparison of simultaneously operating equipment, advance rates of a stope and drifting face in the context of combined mining method. Nonproductive time of one equipment when another equipment operated (for example, coal shearer in longwall was out of action while rock was mining in drifting face and stowing into pack) was registered.

In the context of stowing operations stop-watch readings were carried out according to a cycle components: full-length extension of pipeline; stowing of worked-out area with rock at length of one section of pipeline; shortening of pipeline by one section; and shortening of main pipeline within a drive.

Such mining and geological conditions of a seam mining as its thickness and formation dip, and physical and mechanical properties of coal and enclosing roof and floor were analyzed in detail. Special attention was paid to composition and structure of both roof and immediate floor. The conditions were analyzed according to the mine papers and by means of direct observations.

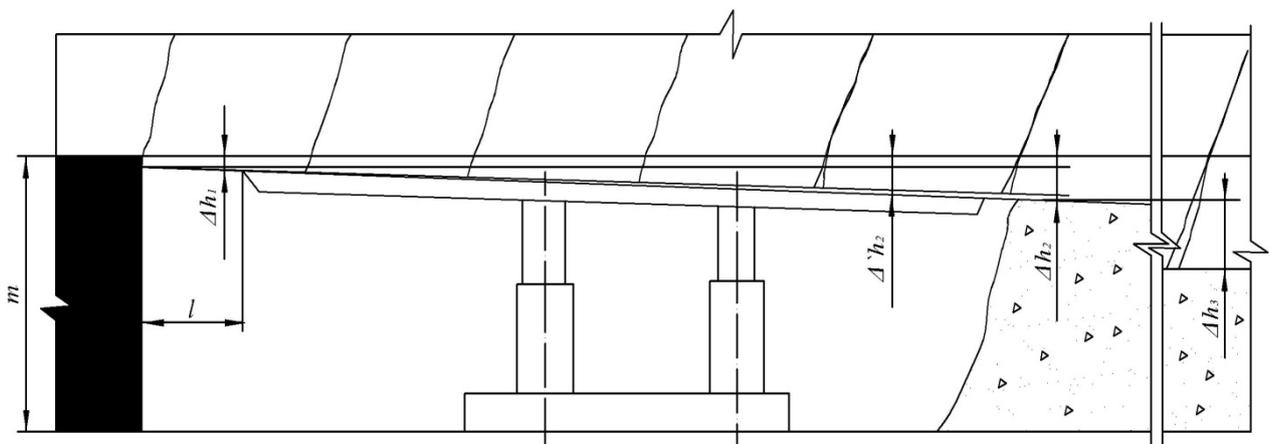
Analysis of rock pressure manifestations in terms of working area of both longwall and stowing mass involved field studies.

Enclosing roof and floor displacements in adjoining mine workings were measured by means of determination of distances between two pairs of contoured benchmarks; one of them was forced into floor, another one – into walls of mine working. Benchmark stations were equipped in front of a stope keeping out a zone of mining. When joint construction of a mine working with longwall took place they were equipped in the neighbourhood of drifting face. Reading periodicity depended on a stope advance (five to seven days). It was two to three days when a longwall approached benchmark stations.

### 3.5. Study of rock pressure manifestations both in working area of a longwall and in stowing mass

#### 3.5.1. Enclosing roof and floor displacement

Studies of enclosing roof and floor involved field studies of condition of roof rocks and floor rocks, measurements of their approach values both in working area of a longwall and in stowing mass. Fig. 3.4, Table 3.1, and graph in Fig. 3.7 demonstrate values of displacements determined in the process of the studies.



*Fig 3.4 Values to be measured while studying enclosing roof and floor displacements*

Analysis of rock condition within face zone of working area of longwall shows that in the both pilot longwalls, roof breaks had been formed throughout the length of a stope. As a result, the seam roof was divided into blocks equal to web width of final elements of coal shearer (Fig. 3.5). In this context, displacement of a block neighbouring a stope took place. Average value of the displacement within goaf pack building was 10mm (746-bis longwall) and 15 mm (719 mm) to be 25-40 % less to compare with remaining part of the longwall. In the process of a stope advance cracks between blocks experienced their openings, and blocks disintegrated into smaller blocks. In 746-bis longwall where mining and geological conditions were simpler owing to less thickness of rather unstable partings, blocks did not experience any disintegration and fell on stowing mass (Fig. 3.6).

As it follows from the research results represented in Table 3.1 enclosing roof and floor stowing results in 13-14 % decrease of enclosing roof and floor convergence within working area of longwall. That has a beneficial effect on enclosing roof and floor state, makes it possible to decrease mined seam thickness and, as a result, to minimize



*Fig. 3.5 Nature of roof rocks failure in the context of experimental-industrial longwall 719*



*Fig. 3.6 Nature of roof rocks in the context of stowing area of longwall 746-bis*

Table 3.1 Parameter values of mine pressure manifestation

Parameters	Symbols	Measuring units	Values			
			Longwall 746-bis		Longwall 719	
			Within an area of stowing	Within an area of complete caving	Within an area of stowing	Within an area of complete caving
Mined seam thickness	$m$	m	$\frac{1.11-1.29^*}{1.22}$	$\frac{1.10-1.31}{1.22}$	$\frac{1.16-1.37}{1.26}$	$\frac{1.17-1.37}{1.26}$
Distance from a stope to a ceiling end	$l$	m	$\frac{0.30-0.52}{0.46}$	$\frac{0.30-0.50}{0.45}$	$\frac{0.26-0.61}{0.46}$	$\frac{0.25-0.55}{0.44}$
Value of vertical displacement of block one of a roof	$\Delta h_1$	mm	$\frac{0-15}{10}$	$\frac{10-25}{18}$	$\frac{8-20}{15}$	$\frac{12-23}{20}$
Enclosing roof and floor convergence at the level of goaf end of a ceiling	$\Delta h_2'$	mm	$\frac{181-248}{213}$	$\frac{190-280}{245}$	$\frac{185-245}{208}$	$\frac{230-275}{242}$
Enclosing roof and floor convergence at the level of goaf end of back cantilever	$\Delta h_2$	mm	$\frac{203-268}{233}$	-	$\frac{216-265}{235}$	-
Stowing mass subsidence	$\Delta h_3$	mm	355	-	360	-
Total convergence of enclosing roof and floor	$\Delta h$	mm	598	-	610	-

\* Numerator shows minimums and maximums; denominator shows average value

undercut value while thin and very thin seams mining. Value of pack compression was 355-360 mm to be almost 35% of its initial thickness. Analysis of the value dependence on distance to longwall face (Fig. 3.7) helps say that within stowing mass a process of enclosing roof and floor approach experiences its practical termination at the distance of 22-24 m from a stope. Increase in approach speed of enclosing roof and floor takes place within stowing mass at the distance of 12-16 m from a stope (longwall 746-bis) and 9-12 m in the context of longwall 719. It depends on increase in load on a pack due to subsidence of main roof.

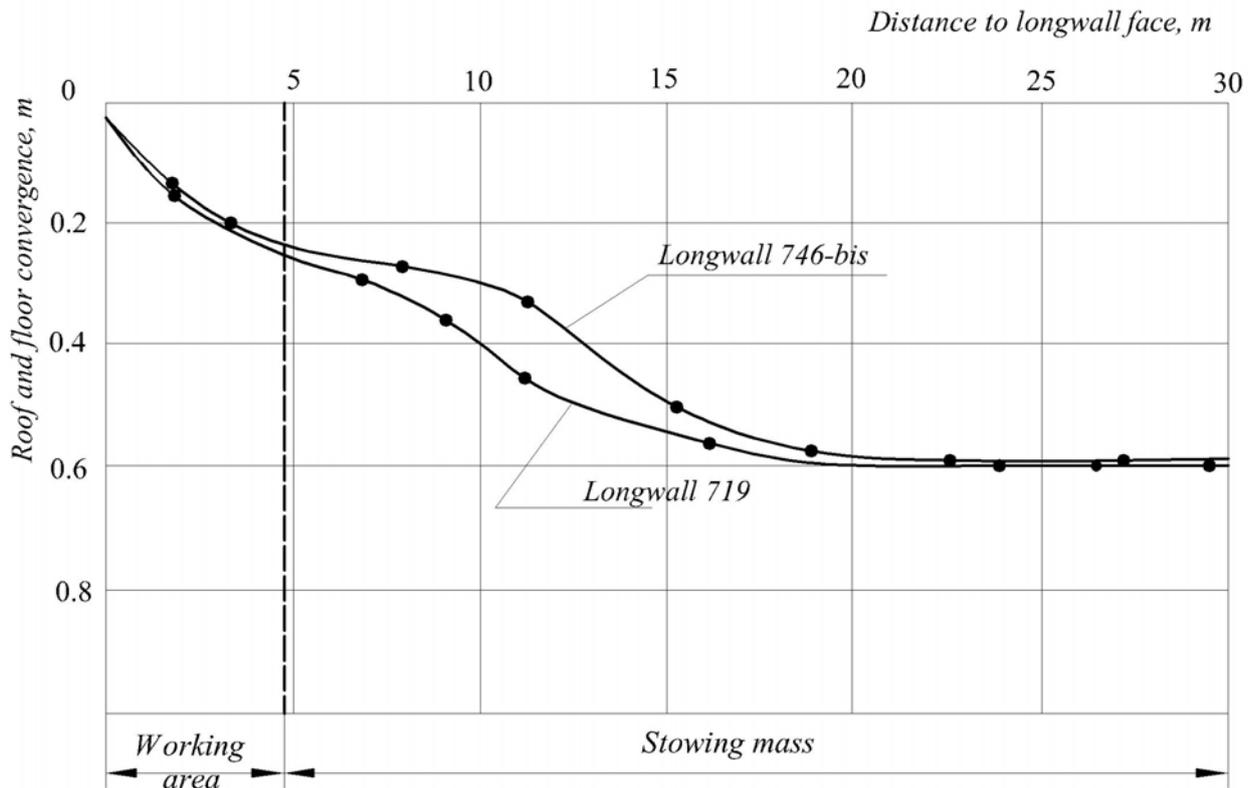


Fig. 3.7 Enclosing roof and floor convergence both in working area of longwall and in stowing mass

Total value of enclosing roof and rock approaches is a sum of the closest to a stope block subsidence ( $\Delta h_1$ ), and enclosing roof and floor convergence both in working area of longwall ( $\Delta h_2$ ) and in stowing mass ( $\Delta h_3$ ); that is

$$\Delta h = \Delta h_1 + \Delta h_2 + \Delta h_3 \quad (3.1)$$

Thus, total value of rock approach in longwall 746-bis was 598 mm; for longwall 719 it was 610 mm. In terms of percentage points there were 49.0 % and 48.4 % respectively.

As it is known, worked-out area stowing makes it possible to reduce value of earth's surface subsidence. It was not realistic to make rather accurate determination

of the subsidence actual value. The matter is that in one case pilot longwall was situated under water basin; in another case it was situated under the area of extensive agriculture as well as comparative narrowness of stowing track. However, results of the research concerning enclosing roof and floor make it possible to suggest that worked-out area stowing will enable to reduce the earth's surface subsidence at least by half.

### 3.5.2. Inrush within working area of longwall

For qualitative estimation of inrush in a stope both national and world scientists have developed a number of criteria [10, 36, and 103]. Depth, average area and inrush volumes within face area, mean value of roof tendency to inrushes, inrush frequency with more than 30cm depth, bench periodicity in terms of stepped roof fault etc. are basic ones. Each of the criteria is expedient to be used to estimate efficiency of roof control, its stability, and assessment of correctness of powered support parameters.

However, in our case qualitative estimation of inrush within various sites of longwall is quite sufficient. It concerns a zone of stowing mass construction and for comparison a zone where roof is controlled by means of complete caving. To do that it is not expedient to use above criteria since estimation can be performed with the help of several simplified factors.

Observations of roof rocks inrush within pilot longwalls were of periodical nature during the whole period of the research. Geometric measurements of inrush piles within face zone of a stope being a result of previous mining cycle were also involved. Observation periodicity was two to three times a month.

Average height of inrush within each section of powered support was taken as criterion one. The factor was a result of summing up of inrush height on each measurement with subsequent dividing of the sum by the number of measurements; that is:

$$H_{cp.i} = \frac{\sum_{i=1}^n H_i}{n}, \quad (3.2)$$

where  $H_{cp.i}$  is average inrush height within  $i^{th}$  section of powered support, m;  $H_i$  is inrush height within  $i^{th}$  section of powered support according to one of measurements, m; and  $n$  is the number of measurements.

The assumed criterion makes it possible to determine distribution of roof rock inrushes along the length of the longwall; to identify sites with the most intensive inrush as well as sites where inrush is either minor or nonavailable.

Figures 3.8 and 3.9 show distribution of roof rock inrushes in terms of longwalls 746-bis and 719, respectively.

Analysis of the distribution help conclude that within a zone of goaf pack building inrush height is negligible, and inrush is not available in the context of the majority of sites.

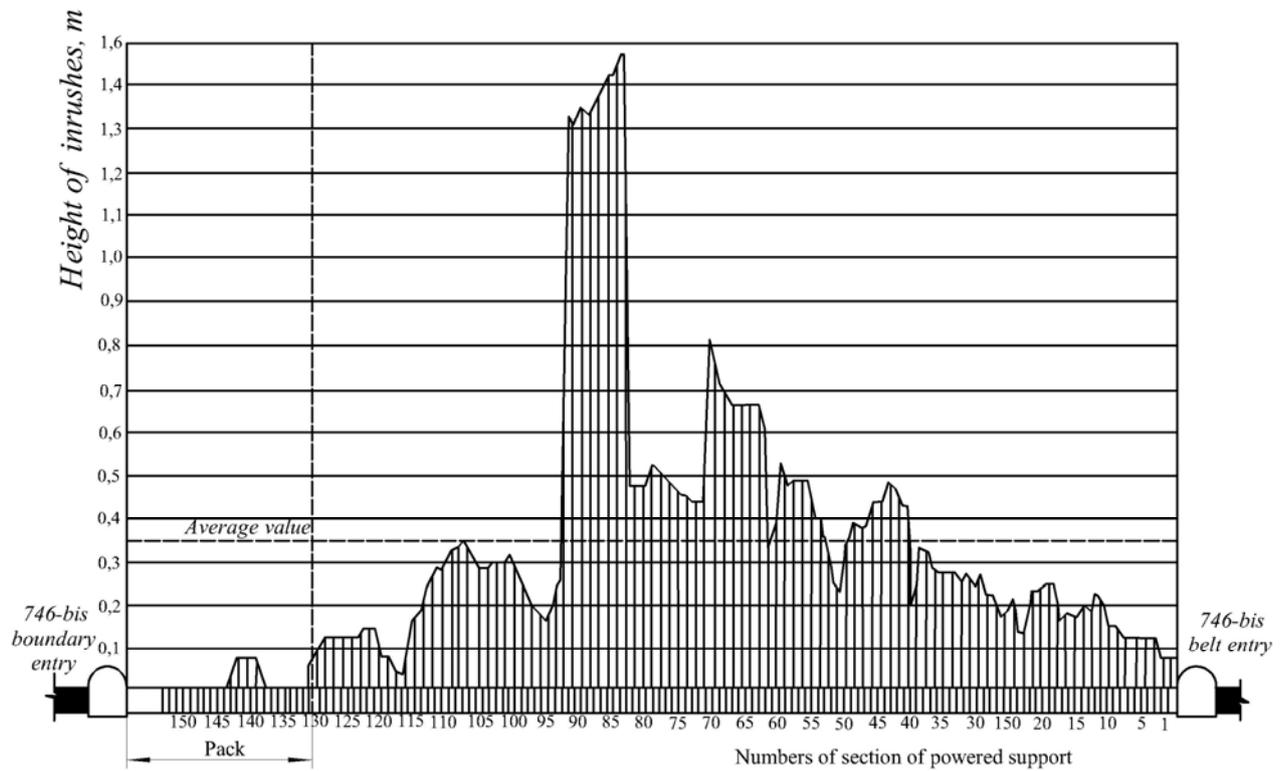


Fig 3.8 Distribution of roof rock intrushes in terms of longwall 746-bis length

Central part of the longwall demonstrates the highest level of intrushes and their frequency. It can be explained by the fact that the both pilot longwalls were located in pillar; at the distance being shorter than values of half-spans of seams, roof subsidence could not reach its maximum.

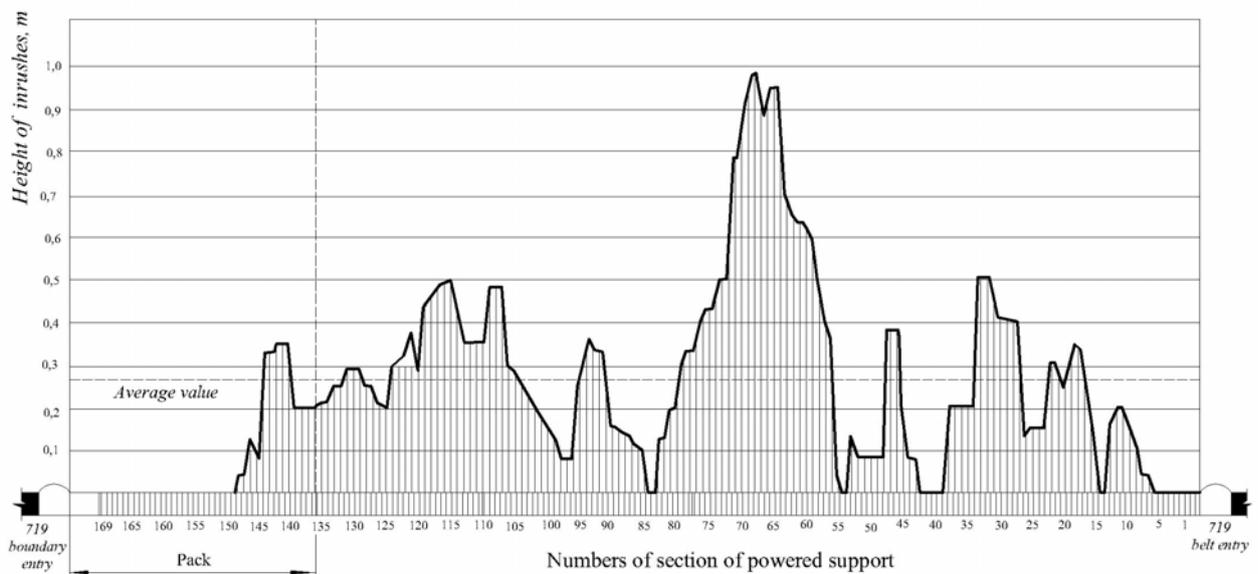


Fig. 3.9 Distribution of roof rock intrushes in terms of longwall 719 length

Specific inrush of roof rock became criterion two which helped to perform qualitative comparison of roof rock state within different sites of longwall. Principle of the criterion is that total amount of inrushes formed during previous mining cycle within certain area of longwall should be divided by the site length. In other words, specific inrush is average inrush amount referred to a running meter of a stope. The factor is determined as follows:

$$V_{cp} = \frac{\sum_{i=1}^{l_{yu}} V_{cp.i}}{l_{yu}}, \quad (3.3)$$

where  $V_{cp}$  is specific inrush,  $m^3/m$ ;  $V_{cp.i}$  is average amount of inrushes in terms of  $i^{\text{th}}$  section of powered support,  $m^3$ ; and  $l_{yu}$  is dimension of the involved site, m.

Three typical sites shown in Figures 3.10 and 3.11 have been chosen for comparative analysis of specific inrush in the context of pilot longwalls:

- Site one being a portion of longwall where pack was built;
- Site two being a portion of longwall at the neighbourhood of mine roadway which length is equal to length of site one;
- Site three being remaining portion of the longwall.

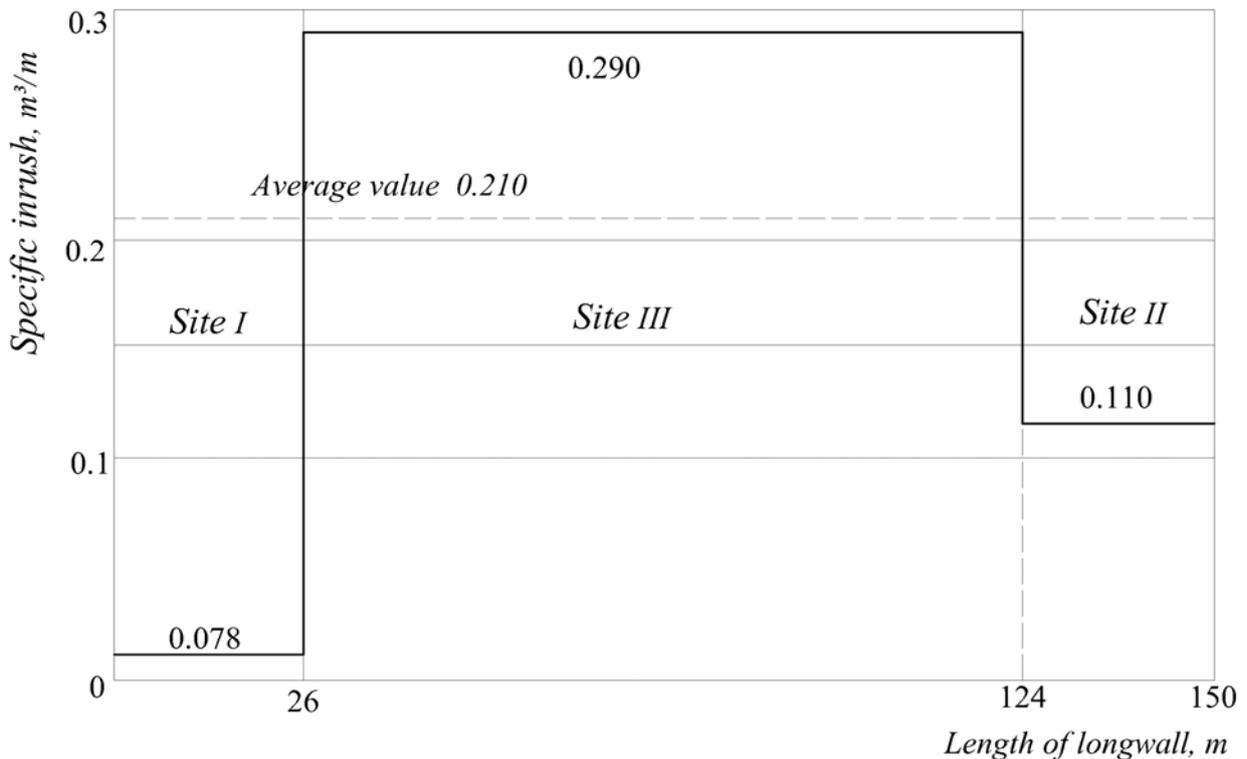
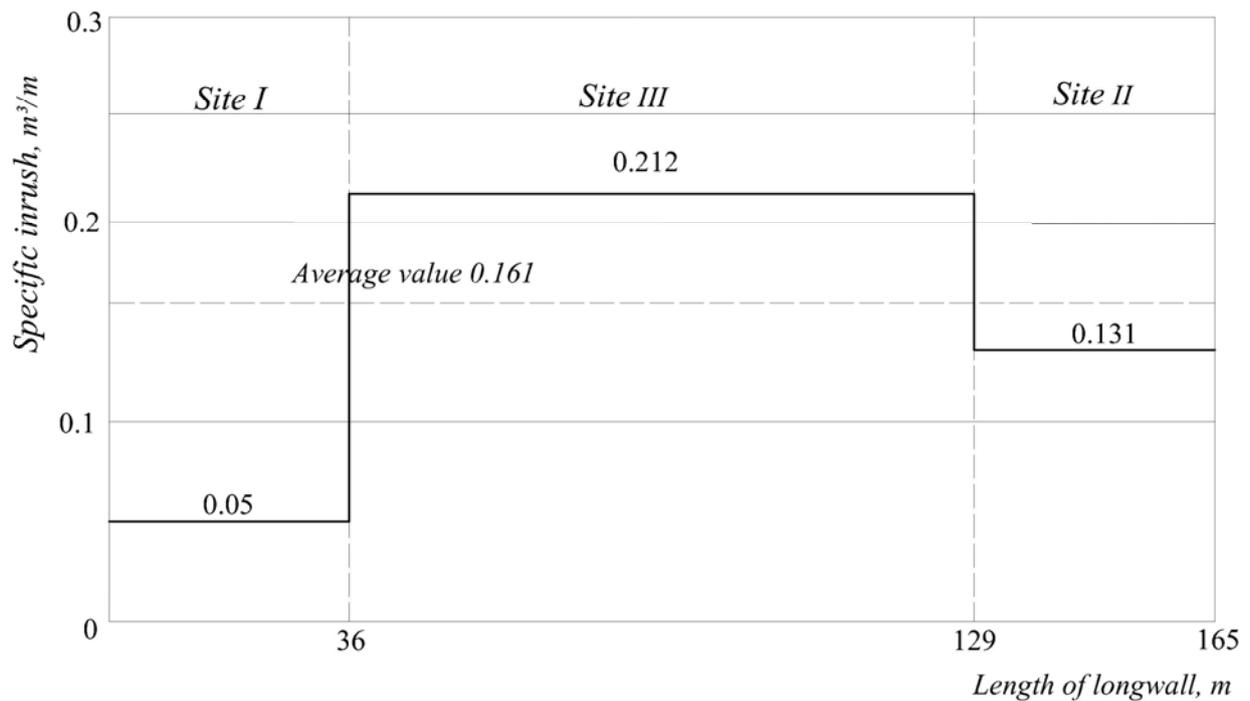


Fig. 3.10 Specific inrush within typical sites of longwall 746-bis



*Puc 3.11 – Specific inrush within typical sites of longwall 719*

Analysis of Figures 3.10 and 3.11 helps say that construction of stowing mass factors into considerable reduce in inrush within working area of longwall. In terms of longwall 746-bis decrease in specific inrush was more than 14 times; in terms of longwall 719 where quality of pack was worse the decrease was 2.6 times.

Thus, results of underground investigations confirm theoretical thesis that worked-out area stowing significantly reduce inrush within working area of longwall.

### **3.5.3. Power parameters of powered support**

Actual resistance of hydraulic props of powered support was measured with the help of recording manometers M66A positioned at measuring points. One of the points was in that portion of longwall where a pack was being constructed; another one was located in a zone with complete roof rocks caving. The recording manometers were engaged in permanent record of pressure changes in working spaces of hydraulic props of back row and front row of powered support.

Fig. 3.12 explains samples of changes in pressure values.

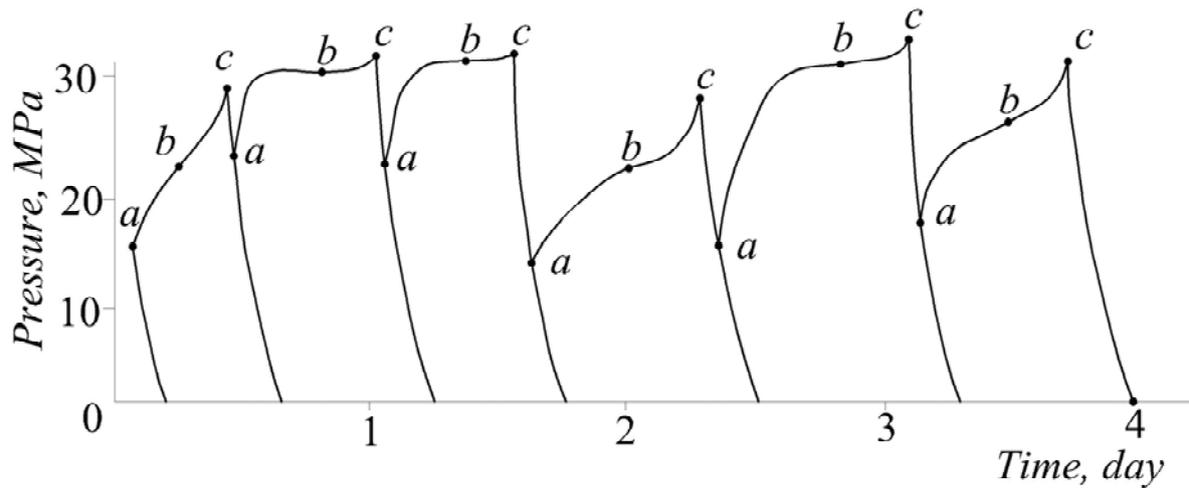


Fig. 3.12 Samples by recording manometer M66A

The dependence helps analyze nature of load disturbance of power support; determine loads on hydraulic props and, then, its total resistance. In Fig. 3.12 point "a" corresponds to initial thrust of hydraulic prop; "b" corresponds to its working resistance; and "c" corresponds to resistance before powered support pressure decrease.

The research results help determine that both within stowing zone and within a zone of complete roof caving 1M88 supports mainly operated in a mode of incremental resistance. Several cases of relief valve action have been recorded.

Table 3.2 shows values of support response obtained in the context of the two pilot longwalls; in the form of histogram Figures 3.13 and 3.14 explain interval distribution of working resistance values of support response of front row and back row.

Analysis of the histograms and Table 3.2 data demonstrates that within a zone of complete roof caving, load on front row was greater (19-20 %) to compare with back row as in the context of the majority of longwalls in Western Donbass where roof is controlled with the help of complete caving [10].

Within stowing zone, load disturbance of modernized sections (i.e. sections with back goaf consoles) differed. Load on back row was greater (18-22 %) to compare with that on front row.

On the average, working resistance of back row was 418 kN for longwall 746-bis, and 375 kN for longwall 719 to be 53 % and 48 % of certified values respectively.

Resistance of support specific resistance value being resistance in terms of 1 m<sup>2</sup> of supported roof (Table 3.2) helps determine that the value experience its decrease in the process of worked-out area stowing. In longwall 746-bis specific resistance experienced its 35 % decrease; in longwall 719 with lower quality of stowing mass construction increase it was only 2 %.

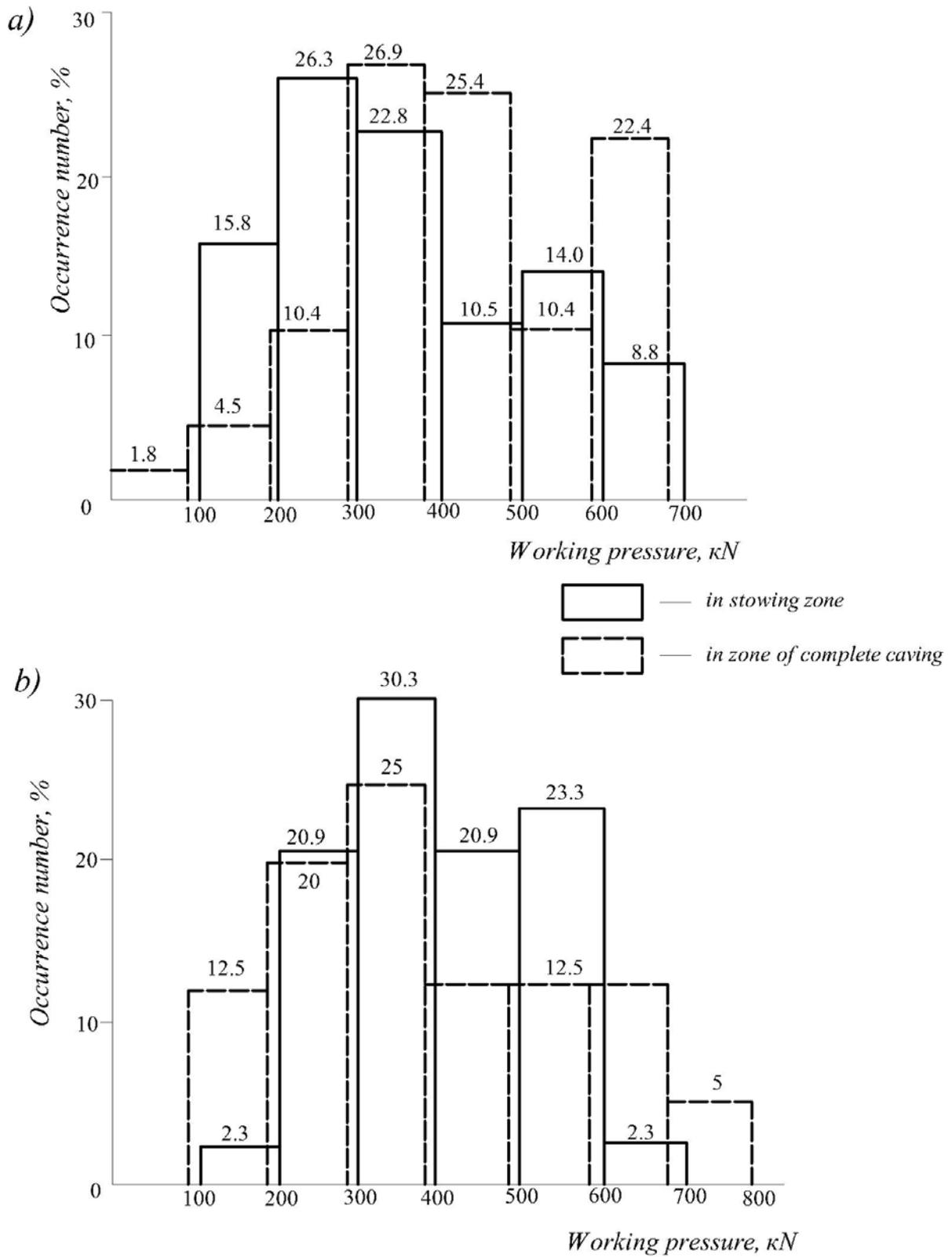
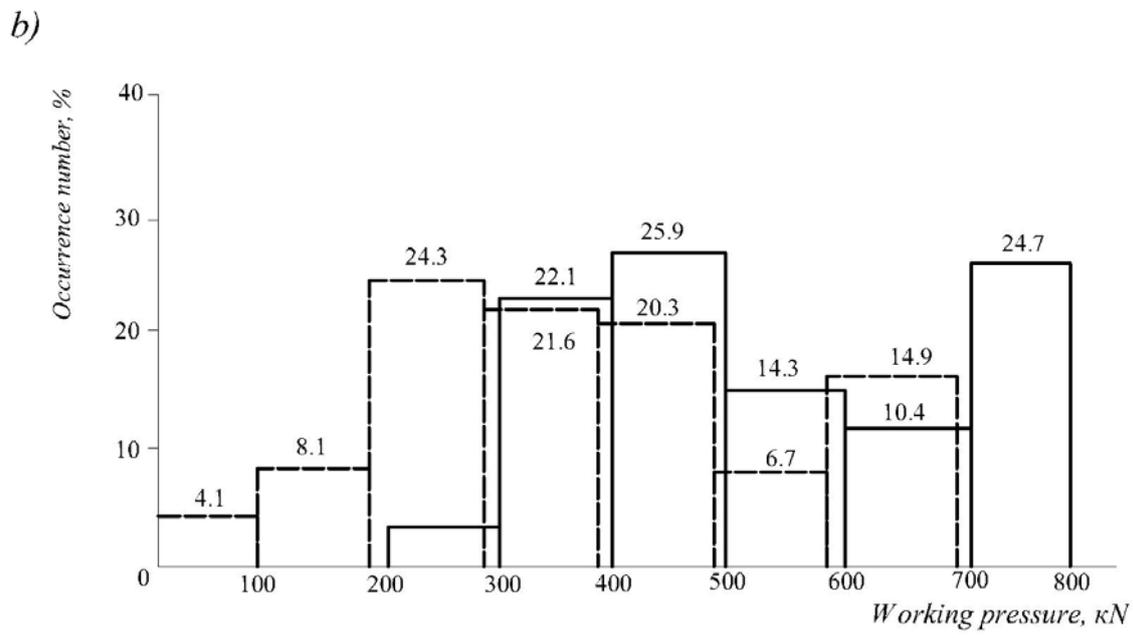
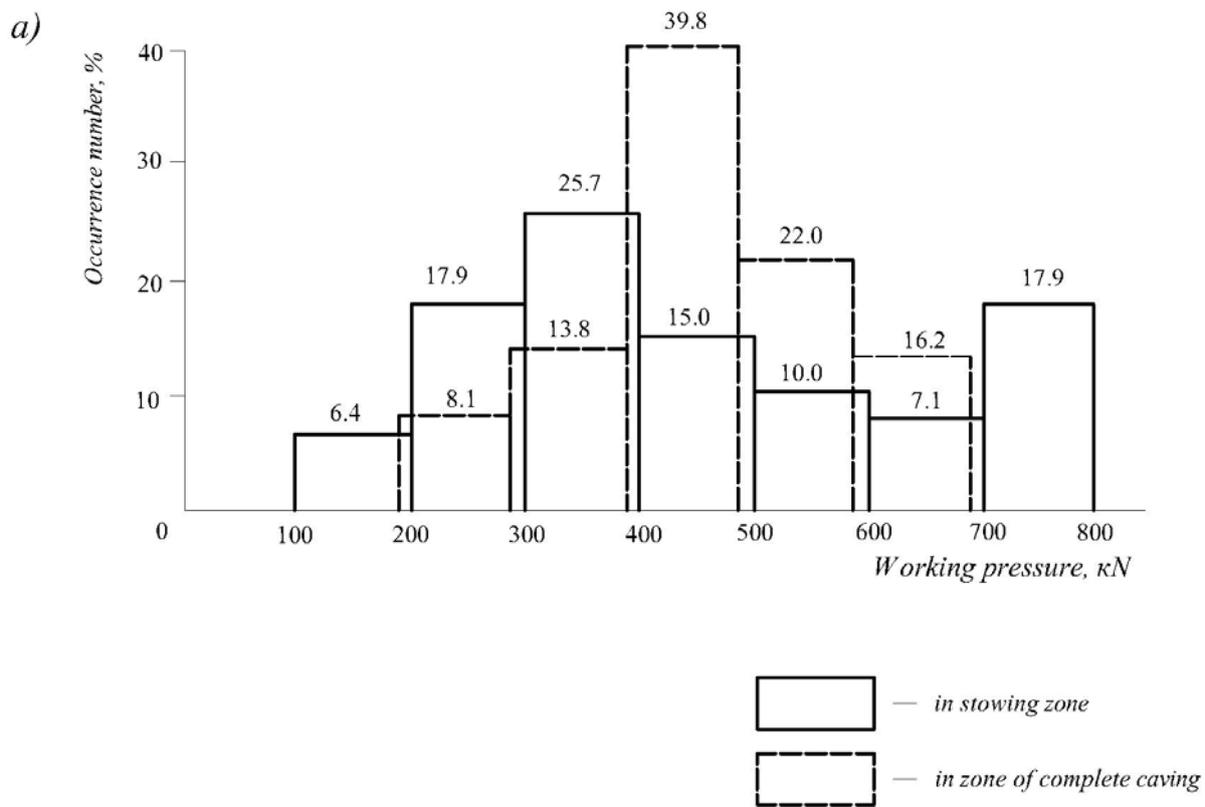


Fig. 3.13 Histograms of working pressure distribution for hydraulic props of support in longwall 746-bis



*Puc 3.14 Histograms of working pressure distribution for hydraulic props of support in longwall 719: a) front row; b) back row*

Table 3.2 Actual power parameters of powered support M88

Parameters	Longwall	Values							
		Within stowing zone				Within a zone of complete caving			
		Row one of supports	Row two of supports	Per section	Per 1 m <sup>2</sup> of supported roof	Row one of supports	Row two of supports	Per section	Per 1 m <sup>2</sup> of supported roof
Initial thrust, kN	746-bis	88-290 143	100-490 290	188-780 433	100	112-434 263	66-469 209	178-903 472	135
	719	86-569 290	145-771 319	231-1340 609	132	68-557 281	56-485 236	124-1042 516	148
Working resistance, kN	746-bis	136-660 235	278-660 417	414-1320 742	172	144-670 517	188-670 418	332-1340 935	267
	719	161-754 443	282-780 544	443-1534 987	214	231-688 471	69-681 375	300-1369 846	242
Resistance before pressure decrease, kN	746-bis	244-660 458	312-672 471	556-1332 929	215	290-736 600	252-760 490	542-1496 1090	311
	719	198-780 489	295-780 583	493-1332 1072	233	257-726 537	93-681 453	350-1407 990	283

\* Numerator shows minimums and maximums; denominator shows average value

### **3.6. Analysis of rock pressure manifestations in mine workings adjoining a longwall**

Enclosing roof and floor approach in terms of drift border, load on mine working support, and deformation of drift support are basic parameters of rock pressure manifestation within development workings.

Rock pressure manifestations were analyzed within development workings protected with a pack in terms of both pilot sites; they were also compared with results obtained in mine workings supported with wood chocks.

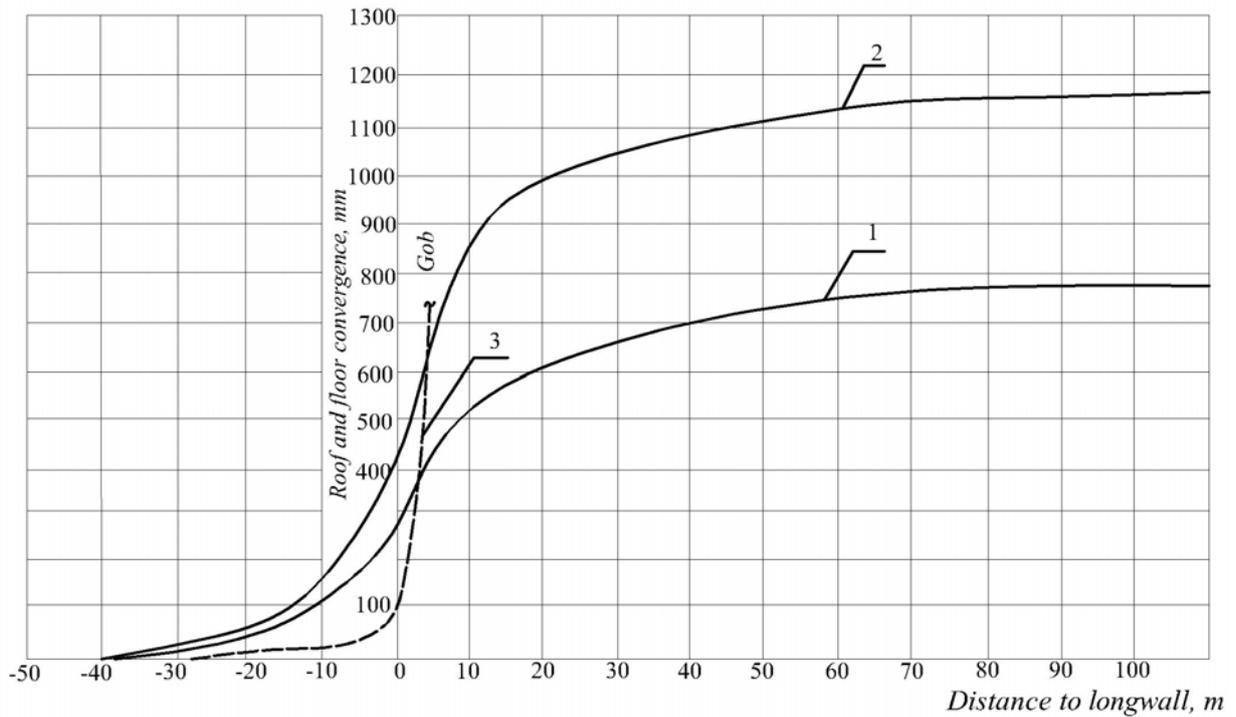
It should be noted that several benchmark stations within boundary drift of longwall 746-bis were located in a zone of increase rock pressure (IRP). In longwall 719 boundary drift was entirely placed under rib pillar formed as a result of mining within overlying seam C<sub>8</sub>; that is in a zone of IRP.

Benchmark stations to observe rock pressure manifestations within boundary drift 719 were located depending upon different support variations. In terms of variation one distance between support arches was 0.8m; special thrust bearings had been welded up to support bottom to lessen spin of the arch into the floor. Variation two involved 0.5m timbering pitch; in this context thrust bearings were not available.

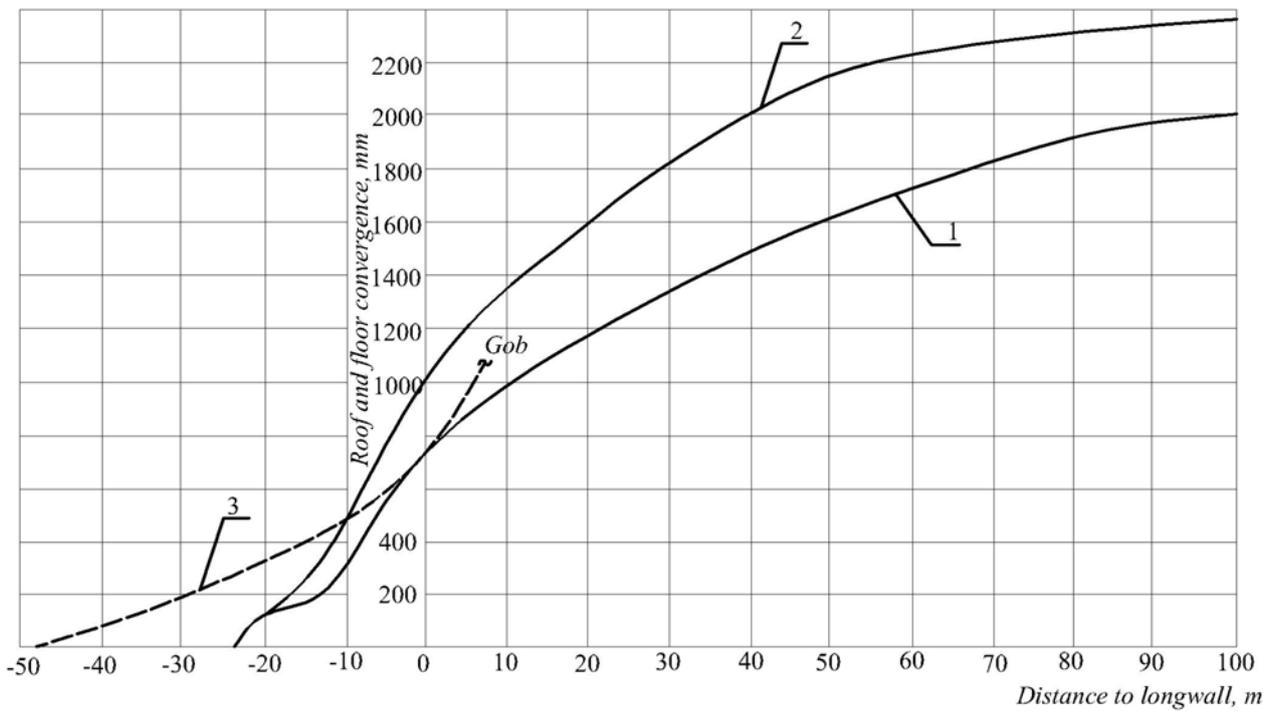
Figures 3.15 and 3.16 demonstrate values of enclosing roof and floor absolute convergency in the context of a variety of distances from a longwall face.

Result of the research conducted in mine workings adjoining longwall 746-bis helps to determine that effect of stope within border drift of longwall 746-bis protected by means of a pack and coal mass starts its manifestation at the distance of 38-42 m in front of the longwall face. When longwall approaches benchmark stations loss of height of mine working was 200 - 330 mm being 270 mm at an average (curve 1 in Fig. 3.15); it was 360 - 450 mm within IRP zone (curve 2). When longwall pass is over, value of rock approach experiences its 125 - 170 mm increase at the distance of 4m from stope. When distance is longer than 40 - 50 m at the back of longwall approach velocity of enclosing roof and floor experiences its significant reduce. Total value of rock approach at the distance of 100 m from longwall varied within 670-830 mm to be 750 mm at an average. Within a zone of increase rock pressure total convergency of enclosing roof and floor was 1130 at average in the context of boundary drift.

Similar research was conducted in belt roadway of longwall 746-bis where nature of rock deformation differed from boundary drift deformations (curve 3 in Fig. 3.15). Thus, increased rock pressure zone resulted from effect of stope spread over distance of 25-29 m in front of stope to be less to compare with that within boundary drift. However, when longwall approached benchmark stations sharp increase in velocity and rock convergency value took place. Total value of rock approach within belt roadway of longwall 746-bis before its failure at the distance of 5m at the back of longwall was 650 mm to be 35 % more to compare with that in the context of boundary drift if distance to longwall is similar.



*Fig. 3.15 Changes in enclosing roof and floor convergency at different distances in terms of mine workings adjoining longwall 746-bis*



*Fig.3.16 Changes in enclosing roof and floor convergency at different distances in terms of mine workings adjoining longwall 719*

Horizontal convergency of a drift was measured until stope approached benchmark station as longwall had already undercut boundary benchmark. Its value within boundary drift was 45mm at an average; it was almost 70 mm within belt roadway.

Graphs in Fig. 3.16 explain values of absolute convergency within mine workings adjoining longwall 719 at different distances from stope. Analysis of the dependences helps conclude that variation one of longwall 719 boundary drift timbering is more preferable since it provides less values of enclosing roof and floor approaches (curve 1) to compare with variation two (curve 2).

It follows from Fig. 3.16 that application of timbering variation one just after benchmarks were placed in the neighbourhood of drifting face resulted in significant increase of enclosing roof and floor approach. Thus, advance of drifting face for 5.5 m and advance of stope for 2 m resulted in 100 mm value of rock approach. That depends on availability of cavities between girder of arch support and drift roof which factors into roof rocks subsidence on support in terms of minor advance of drifting face. When complete subsidence of roof rocks on arch girder took place certain deceleration of rock approach was observed. However, at the distance of 12 - 13 m from longwall sharp increase in rock convergency occurred. When longwall moved toward benchmark station, value of enclosing roof and floor approach was 675 mm. At the distance of 45-50 m from stope convergency intensity decreased; then it became constant. Total approach of rocks in drift at the distance of 100 m from longwall was 1920 mm or 50% of mine working height.

Another component of enclosing roof and floor approach in addition to roof rock subsidence was floor heave making 25-30% of total convergency.

Significant load on support in boundary drift of longwall 719 resulted in its serious deformations. It was mainly seen in girder and support feet bending, and in increase of yield units on the part of longwall. Thus, when stope approached, overlap value within joint for components of a support from mined-out area experienced 480 mm increase (120%); the value was 190 mm (48%) in terms of pillar. 50 m behind a longwall the values were 1005 mm (250%) and 200 mm (50%) respectively. Such drastic changes in overlap value from mined-out area and minor ones from virgin ground resulted in bar displacement to worked-out area and breakage of arch support feet from pillar (Fig. 3.17).

To reduce support deformation, variation 2 to support boundary entry 719 was tested. The technique involved tighter mounting of arch support as well as smaller area of the arch base. However, in the context of the variation, support didn't experience drastic changes while spinning into the floor; as a result enclosing roof and floor convergence increased.

a)



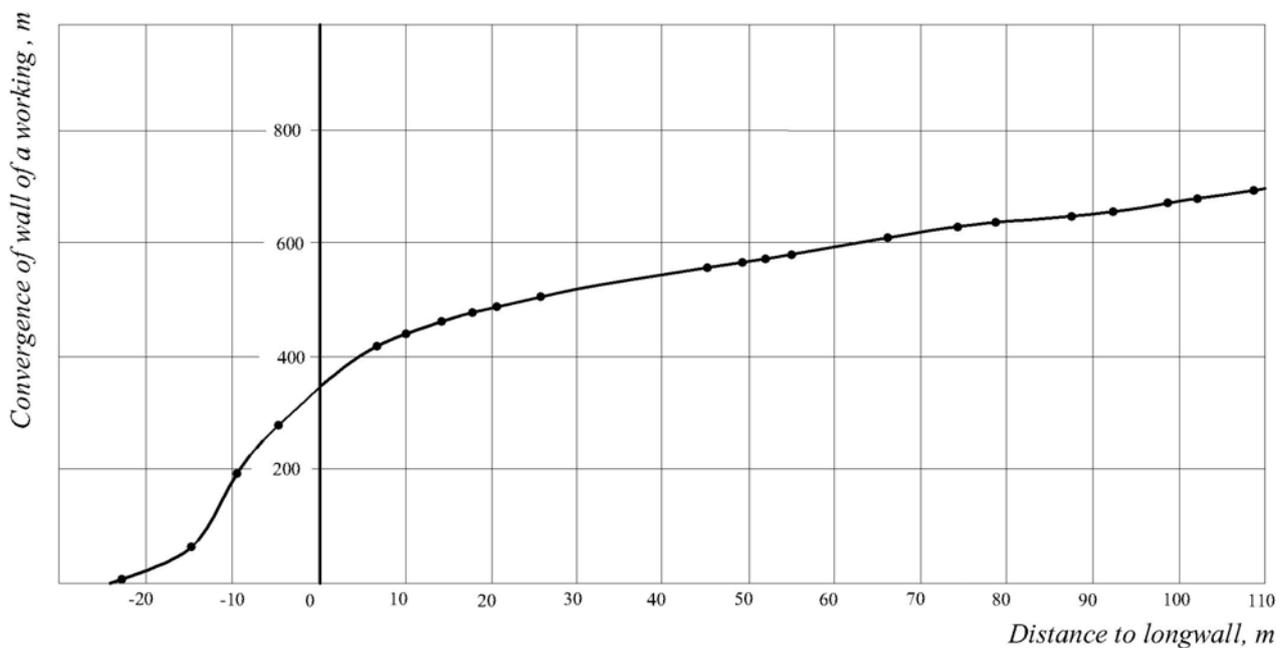
b)



*Fig. 3.17 Deformations of boundary entry 719 support: a) from worked-out area; b) from pillar*

When the stope approached benchmark station, loss in mine working height was 990 mm to be 32 % more to compare with variation one. 100 m behind the longwall face enclosing roof and floor convergence was 2370 mm to be 450 mm (19 %) more to compare with supporting technique one. Total loss of mine working height was 60 % of its initial height.

Convergence of mine working sides that is changes in mine working width was not very significant (700 mm at the distance of 107 m behind the longwall) to be 21 % of initial width of the entry (Fig. 3.18).



*Fig. 3.18 Values of convergence of wall of a working in boundary entry 719 in terms of different distances from a longwall*

To determine strength of mine working protection by a pack, results of analysis of enclosing roof and floor convergence in boundary entry 719 were compared with results of similar analysis in terms of mother entry 719. Absolute values of the mother entry rock displacement before its gobbing slightly differed from values of boundary entry rock displacement in terms of similar area. Thus, at the distance of 5m behind the longwall, the value was 900 m upon average. In the context of boundary entry within the same distance, average value was 830 mm in terms of supporting variation one, and 1170 mm in terms of supporting variation two. However, absolute displacement values in terms of both entries are hardly comparable due to different sections of the mine workings. The values can be comparable in terms of their percentage of initial height of mine working. In this

case, loss of mother entry height was 32 %; in terms of similar distance within boundary entry it was 21-29 % of its initial height.

As noted above, along the whole its length, boundary entry 719 has always been within a zone of high rock pressure; hence, at standard conditions of its construction less values of enclosing roof and floor convergence are possible.

Thus, results of enclosing roof and floor convergence analysis in the context of development workings help conclude that availability of a pack has a beneficial effect on a mine working maintenance. In some cases it will help reuse neighbouring mine workings.

It should also be noted that subsidence in front of a stope is almost 30 % of a value of enclosing roof and floor total convergence within development workings. Thus we may suppose that when construction of mine workings follows a longwall it is possible to decrease enclosing roof and floor convergence by the same value making the technique of mine working construction more preferable.

### 3.7. Operating conditions and economic performance of the technique

Values of basic operating conditions resulting from the research and generalized data concerning the two pilot longwalls are listed in Table 3.3.

Table 3.3 Operating conditions of the technique

Parameters	Complete mining	Selective mining	Selective mining with stowing
Feed velocity of coal shearer, m/min:	$\frac{1.6 - 3.6^*}{2.3}$	-	-
- in terms of coal	-	$\frac{2.0 - 4.0}{2.8}$	$\frac{2.0 - 4.0}{2.8}$
- in terms of rock	-	$\frac{2.0 - 5.8}{4.2}$	$\frac{2.0 - 2.8}{2.4}$
Net time for mining, min:			
- in terms of coal	-	$\frac{47 - 81}{57}$	$\frac{47 - 81}{57}$
- in terms of rock	-	$\frac{25 - 53}{35}$	$\frac{50 - 72}{60}$
- in terms of a seam	$\frac{42 - 100}{65}$	$\frac{77 - 144}{92}$	$\frac{90 - 134}{117}$

\*Numerator shows minimums and maximums; denominator shows average value

Analysis of Table 3.3 explains that in the process of rock bench mining efficiency of crushing-and-stowing machine restricts efficiency of coal shearer. That factors into almost 70 % feed velocity decrease of coal shearer in terms of rock as well as time increase in terms of rock bench mining. For that reason, seam mining period experiences 27 % increase (92 up to 117 min.).

It should be noted that operations to shorten stowing pipeline are responsible for winning machine shutdown causing 50 min. extension of mining period. Thus, operations connected with undercut rock stowing into worked-out area result in 80% cycle extension to compare with separate seam mining without worked-out area stowing; it is 2.5 times more to compare with complete mining technique.

Table 3.4 demonstrates basic economic performance of experimental longwall operation in “Blagodatnaia” mine while carrying out the research. As it follows from the Table data, average daily production was within 192-360 t (255 t at average) in terms of longwall 746-bis, and 143-245 t (204 at average) in terms of longwall 719. On certain days, daily production in the longwalls was 500 to 550 t. Average monthly stope advance was 32.3 m and 22 m respectively; cost price of a ton of coal was RUB 10.1 and RUB 15.4 respectively. Greater cost price and lesser underground output per man-shift in longwall 719 can be explained by extra material as well as labour costs to construct boundary entry 719 which were involved in economic performance of production unit.

It should be specified that implementation of selective mining technique apparently decreased (44 % down to 18-22 %) ash-content of mined coal. In certain sense it improved economic performance of the mine in whole.

Comparatively low economic performance of experimental longwalls can be explained by unfavourable mining and geological conditions, considerable volume of research and experimental works in the period of the longwall operations as well as a number of organizational factors. If the factors are not available, more effective operation of stopes is predicted.

Table 3.4 Economic performance of experimental longwalls in “Blagodatnaia” mine

The name of parameters	Measuring Units	Months of 1988/1990										Average value
		1 <sup>st</sup>	2 <sup>nd</sup>	3 <sup>rd</sup>	4 <sup>th</sup>	5 <sup>th</sup>	6 <sup>th</sup>	7 <sup>th</sup>	8 <sup>th</sup>	9 <sup>th</sup>	10 <sup>th</sup>	
Monthly output	t/m	$\frac{6358^*}{6033}$	$\frac{5575}{6207}$	$\frac{6978}{6249}$	$\frac{6718}{7352}$	$\frac{3230}{6768}$	$\frac{3515}{6341}$	$\frac{9196}{5642}$	$\frac{9632}{5470}$	$\frac{8206}{4296}$	$\frac{10786}{-}$	$\frac{7019}{6040}$
Daily average output	t/d	$\frac{212}{201}$	$\frac{192}{222}$	$\frac{233}{208}$	$\frac{224}{245}$	$\frac{215}{242}$	$\frac{234}{211}$	$\frac{297}{182}$	$\frac{311}{182}$	$\frac{273}{143}$	$\frac{360}{-}$	$\frac{255}{204}$
Monthly advance of a stope	m/m	$\frac{33}{26}$	$\frac{26}{26}$	$\frac{36}{21}$	$\frac{30}{25}$	$\frac{15}{24}$	$\frac{16}{23}$	$\frac{41}{21}$	$\frac{42}{20}$	$\frac{36.5}{15}$	$\frac{47}{-}$	$\frac{32.3}{22}$
Labour productivity of: - support worker of a stope; - worker of a longwall	t/m t/m	$\frac{113.5}{81.5}$	$\frac{97.3}{80.6}$	$\frac{118.3}{79.6}$	$\frac{108.4}{90.8}$	$\frac{50.5}{88.4}$	$\frac{55.8}{84.3}$	$\frac{146.0}{76.2}$	$\frac{140.5}{77.0}$	$\frac{123.9}{61.4}$	$\frac{160.7}{-}$	$\frac{111.5}{80.0}$
Cost price of a ton of coal extraction	RUB/t	$\frac{9.87}{11.31}$	$\frac{10.8}{15.98}$	$\frac{8.99}{16.13}$	$\frac{9.55}{13.81}$	$\frac{18.27}{15.58}$	$\frac{16.8}{16.57}$	$\frac{7.14}{17.08}$	$\frac{7.18}{16.73}$	$\frac{6.79}{-}$	$\frac{5.77}{-}$	$\frac{10.1}{15.4}$

\* Numerator shows efficiency of longwall 746-bis; denominator shows efficiency of longwall 719

### 3.8. Analysis and estimated results of the research

Actual parameters of technique for thin flat seam mining with worked-out area stowing specified as a result of underground investigations have been compared with theoretical ones and adequately estimated.

Table 3.5 explains design parameters and actual ones as well as a value of their deviation.

The comparison has been performed in accordance with geodynamic and operating conditions of a technique applied to mine out C<sub>7</sub> seam in “Blagodatnaia” mine.

Table 3.5 Design parameters and actual parameters of the technique

Parameters	Values				
	Design values	Actual values in terms of longwall 746-bis	Deviation, %	Actual values in terms of longwall 719	Deviation, %
<u>Geodynamic parameters:</u>					
Convergence of enclosing roof and floor within working area of longwall, mm:					
- within stowing area	215	233	8.4	235	9.3
- within caving area	292	306	4.8	303	3.8
Main roof caving within stowing zone, m	20	22	10	21	5
Total convergence of enclosing roof and floor, mm	527	598	13.4	610	15.7
<u>Operational parameters:</u>					
Feed velocity of coal shearer while rock mining	2.2	2.4	9.1	2.4	9.1
Daily average output of a stope	218	255	10.6	204	6.3

Table 3.5 shows design values of enclosing roof and floor convergence within working area of longwall as well as enclosing roof and floor total convergence involving rock heaving soil and powered support response (refer to Part 2.2.3).

While calculating enclosing roof and floor convergence within working area of longwall it was assumed that the zone width is 5m; that very time the value was 4m

in the context of full-scale conditions. Thus, to make the comparison comfortable, value of enclosing roof and floor convergence has been proportionally increased according to the longwall length.

The analytical research has helped determine that worked-out area stowing results in decrease of enclosing roof and floor convergence. The prerequisite has been practically confirmed; in addition, difference between design values and actual ones is not more than 10 %.

Table 3.5 explains that design values of enclosing roof and floor total convergence differ from actual ones 13.4 - 15.7 % to be some more to compare with that in engineering evaluations. That can be explained by the fact that actual mined thickness was greater than specified one used while calculating. Moreover, for a number of organizational and technical reasons, quality of the pack construction was lower to compare with required one factoring into increase in pack compression and, hence, increase in enclosing roof and floor convergence.

In addition, underground investigations have verified theoretical assumption that worked-out area stowing results in significant decrease of roof inrush within working area of longwall to be another justification for calculations and analytical approach.

It follows from Table 3.5 that precision of design and actual values of operating conditions of the technique is quite satisfactory. Therefore, expressions 2.37 and 2.39 as well as values involved by them are adequately accurate to exhibit actual values of the parameters.

Hence, estimated results of underground investigations as for operational parameters to mine out thin flat seams with worked-out area stowing help conclude that their basic theoretical assumptions have been justified and procedural value has been proved. For the most part, precision of the research results is within  $\pm 10$  %; it is acceptable to calculate parameters of coal-extraction technique.

## **Conclusions**

1. Underground investigations of proposed technique have proved its efficiency both in the context of long-pillar and combined mining technique. Daily average output of longwall has become 204-255 t; sometimes it has become 500-550 t.

2. Owing to 13-14% worked-out area stowing, enclosing roof and floor convergence experiences its decrease within working area of longwall. That favours state of enclosing roof and floor making it possible to decrease mined thickness of a seam as well as decrease value of rock undercut while mining thin seams and very thin ones.

3. Value of pack compression is almost 35% of its initial thickness and total convergence of enclosing roof and floor is 48-49 % of mined thickness of a seam. That makes it possible to reduce surface subsidence in the context of worked-out area complete stowing.

4. Worked-out area stowing results in significant inrush decrease within a stope which creates more favourable and safe labour conditions in longwall.

5. Worked-out area stowing factors into 35% decrease in specific resistance of support; moreover, load on back row is higher to compare with load on front row.

6. Protection of mine workings neighbouring longwall results in their reduced deformation; in some cases that makes it possible to reuse them.

7. Selective mining applied in experimental longwalls enabled to reduce substantially ash-content of mined coal (44 down to 18-20 %).

8. Due to application of selective seam mining and worked-out area stowing, mining period in experimental longwalls experienced 2.5 times increase. The time can be cut owing to application of more efficient stowing equipment, availability of accumulating tank in front of stowing equipment, or use of pipeline with minimized time to change place for stowing material discharge.

9. Actual values of thin flat seam mining with worked-out area stowing are rather accurately described by analytically obtained dependences. Deviation of actual values from design ones is not more than 16 %.

## **CHAPTER 4. PROCESS FLOW SCHEMES TO MINE OUT THIN SEAMS WITH WORKED-OUT AREA STOWING**

### **4.1. Limitations for the technique application**

Application of the technique is limited by following factors:

- Properties of rocks used as stowing material;
- Features of mining equipment being applied;
- Economic expedience.

By now demands placed on stowing material have been explicated [44, 66, 81, 88]. Following demands are basic ones:

- Stowing material should not involve rock pieces which dimensions are more than 0.3 of internal diameter of pipeline; woodchips; metallic objects and other ones which can give rise to pipeline blockage;
- Percentage of 0-6 mm rock grades if material moisture content is up to 5 % should not be more than 20 %;
- Percentage of fuel ingredients in stowing material should not be more than 20 %.

As experts from DonCI [44] believe above demands (except for last-mentioned one) cannot be critical in terms of processing factors. They have been determined on the basis of optimized operation of pneumatic transport.

Paper [81] mentions that if moisture content of material is more that 4-6% it is desirable to screen out extra particles with 0-3 mm size when their percentage is more than 15 %.

Basically enclosing roof and floor in Western Donbass mines are argillites and aleurites prone to soaking. In the process of rock moisturing they become very sticky; in the process of pneumatic conveying they may stick to internal surface of pipeline provoking its blockage.

Moreover, characteristics of improperly treated undercut rocks supplied to stowing machine may differ greatly from established norms.

Therefore, properties of the rocks have been analyzed to substantiate their application for pneumostowing.

The technique can hardly be implemented while using certain types of standard mining equipment without major changes. For example, it is rather inconveniently to apply “Donbas M” system for selective mining in the context of unstable roofs. Operation of the system on “uncharged” schedule made it impossible to relocate sections of powered support right after coal had been mined out. Thus, face area of longwall should be supported with the help of extendable arm to be extremely inefficient while mining seams with unstable roofs.

KA80 and MK67 coal shearers can not mine coal and rock separately; that is why they should not be applied for a technique providing worked-out area stowing.

Application of KД-80 and KM103 systems complicates access to stowing pipeline without substantial changes in a design of powered support making it impossible to use them to mine seams with worked-out area stowing.

Efficiency of the operational schedules application depends on a number of factors: type of equipment, organization of activities, sources of rock and air supply, undercut thickness etc. That is why each specific case should involve substantiation of the efficiency in the context of new technique implementation.

## **4.2. Substantiation of possibility to use enclosing roof and floor being undercut as stowing material**

### **4.2.1. Mechanical analysis of undercut rock**

Properties of stowing material represented by undercut rocks of a stope have been analyzed in the context of  $C_7^H$  seam (“Blagodatnaia” mine). Mechanical analysis of rocks getting into crushing and stowing system “Titan-1” and leaving its pipeline has been carried out in accordance with GOST 2093-69. Rock screen sizing was performed using grading screens where diameters of holes were 1, 3, 6, 13, 25, and 50 mm. In the first instance samples were taken from a conveyer supplying rock to a bunker of a crusher; in the latter case they were taken from a pack.

Histograms in Fig. 4.1 explain results of rock mass screen sizing.

As results of the research show, when rock bench is broken with the help of 1K101Y coal shearer, rock yield of 0-1 mm grade is 16.7 %, and 1-3 mm – 8.6 %; rock yield of 3-25 mm grade is 32.0 %, and rock yield of 25-50 mm is 15.1 %. Rocks of grades of more than 50 mm were 26.7 %. It should be noted that size of maximum rock pieces yielded with the help of coal shearer was not more than 100-120 mm.

Screening of rocks leaving stowing pipeline demonstrates that diametric size of the largest pieces was 60-65 mm. Content of coarse fractions (more than 50 mm) experienced sharp decrease down to 12.5 %. In this connection, percentage of 25-50 mm rock grade increased (24.5 %) as well as the content of medium rocks.

It should be noted that content of less than 1mm rock grade decreased by 4.5 %. That can be explained by the fact that due to non-availability of efficient means of dust control, rocks of the grade were carried away by airflow into working area of longwall; their small volume remained in stowing pipeline in the form of adhered layer.

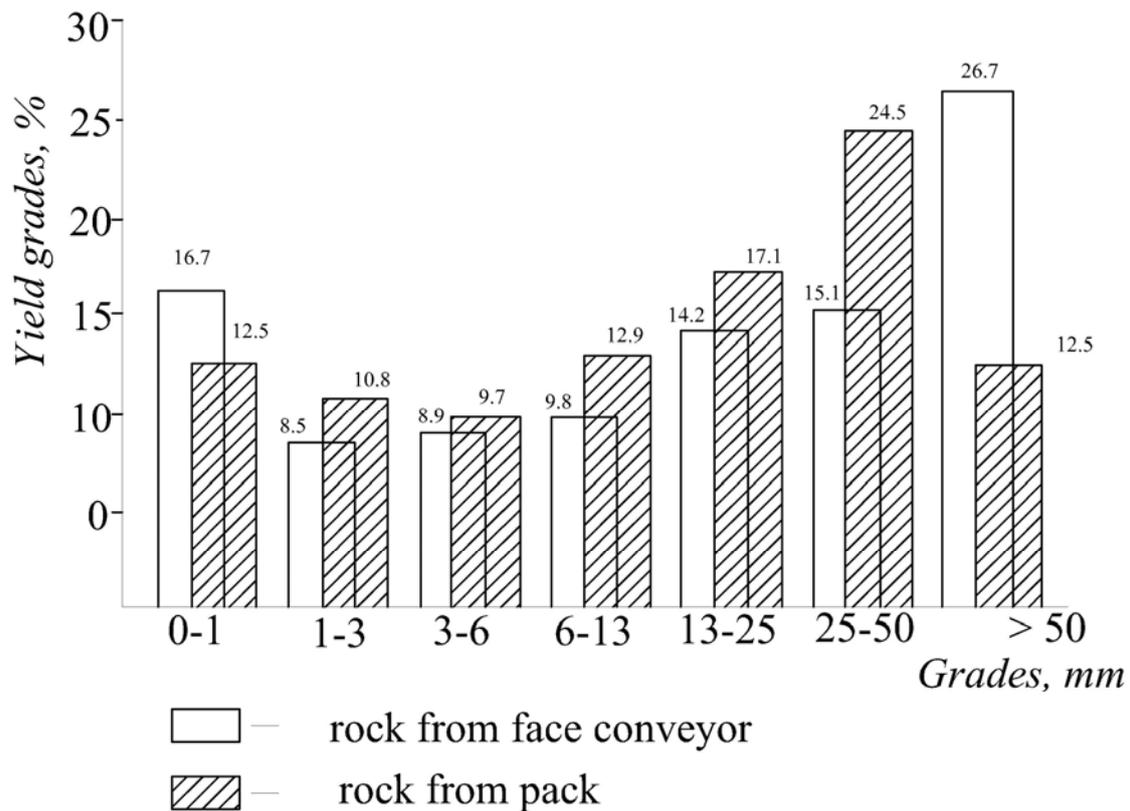


Fig. 4.1 Histogram of stowing material screen sizing

Thus, results of rock mass screen sizing help determine that almost 80 % of loose rock being the outcome of selective seam mining is less than 60 mm in size. Hence, they need not preliminary crushing before pneumatic conveying. Therefore, one-stage rock crushing is satisfactory while using such a source of stowing material; then, crushers of ДО type are reasonable.

It has also been determined that percentage of fine refuse (0-6 mm grade) in stowing material is almost 35 % to be a source of significant dusting within longwall. Moreover, much fine refuse creates conditions for stowing material to adhere to pipeline walls.

#### 4.2.2. Adhesive behaviour of stowing material

As it has been noted above, rocks in Western Donbass are of great adhesiveness; in the context of their substantial moistening they may provoke stowing pipeline blockage. Thus, to identify optimum sprinkling parameters, adhesive behaviour of stowing material has been analyzed in terms of its various moisture degrees.

Adhesion characterizes surface bonding of dissimilar bodies in the context of their interaction. Adhesiveness is basic operational characteristic of adhesion energy of rock mass. It is evaluated quantitatively by means of strength required to separate hard plate of unit area from rock mass [12].

According to S.A. Goncharov, best-attested results are obtained when rock mass adhesiveness is determined by means of separation as in this case effect of friction is eliminated. The idea provides the basis for the operation of Okhotin device [98] and its various modifications.

ИПГ-1 [85] device has been used in the process of our research. Its operating principle is to force steel plate to loosened rock placed into specific frame and measure breakaway effort. The research has been carried out according to a technique described in paper [85]. The device differs from Okhotin device as the former is equipped with driving washer placed between plate and a frame which putting down eliminates the plate jamming.

Result of mechanical analysis of a layer adhered in stowing pipeline helps conclude that the layer consists of rock fractions with up to 1 mm size; thus that very fraction has been used to analyze adhesion properties. To do that rock has been previously screened; bore diameter was 1 mm.

Property of rock adhesiveness is necessity to characterize it not with the help of one figure but with the help of adhesiveness curve being dependence graph of rock adhesiveness on its humidity.

While analyzing rock adhesiveness of  $C_7^H$  seam floor (“Blagodatnaia” mine) such a dependence has been developed and graphically specified (Fig. 4.2).

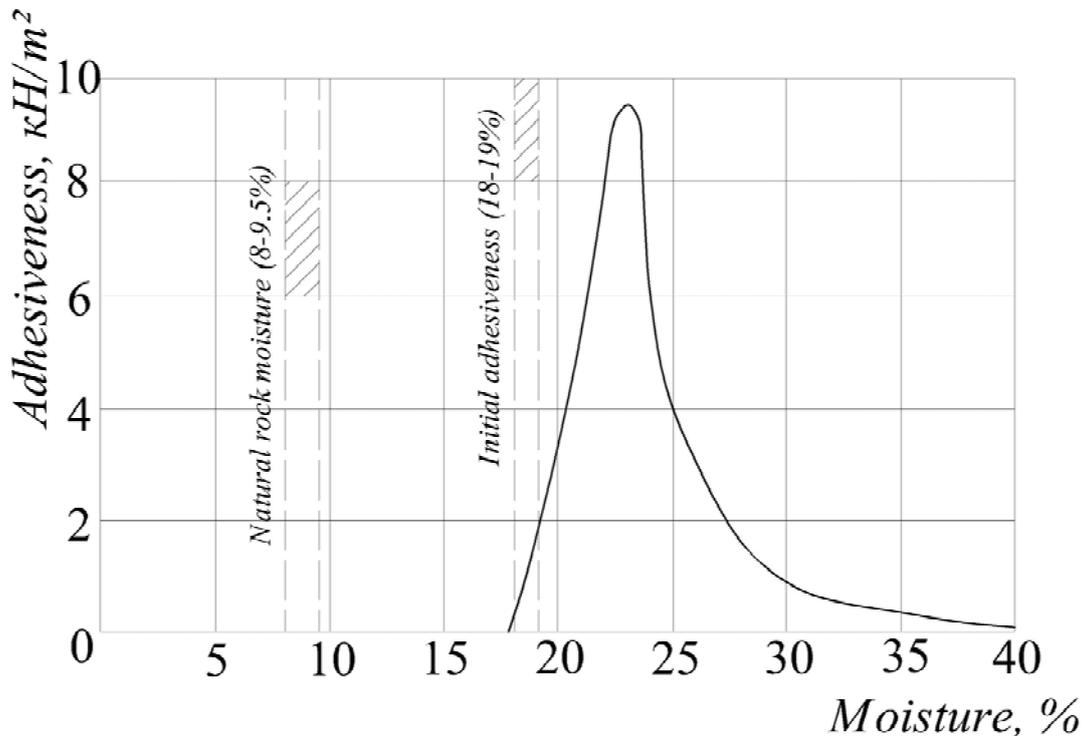


Fig. 4.2 Dependence of rock adhesiveness on their moisture content

It follows from the dependence that in terms of natural rock moisture (8-9.5 %) its adhesion properties are not displayed. 18-19 % moisture content corresponds to

initial rock adhesiveness; in terms of 22-23% moisture content rock adhesiveness reaches its maximum level. In proportion to further water saturation, decrease in adhesion takes place (Fig. 4.2).

Difference between rock moisture content adequate to maximum adhesiveness and initial one is minor (3-5 %); thus for efficient pneumatic conveying, natural moisture content should not be more than values of initial adhesiveness. In other words, rock can be used for pneumatic conveying if its natural moisture content is beyond the adhesive curve.

Difference between initial rock adhesiveness and natural moisture content makes it possible to determine quantity of water to be supplied to stowing material for dust control. In this context it should be taken into consideration that different rock fractions differently participate in freezing process. Thus, fine rock fractions as well as silt ones consume more water to compare with coarse fractions; hence, they may reach initial adhesiveness before stowing material on the whole. Results of the research have demonstrated that fine rock of 0-3 mm grade consumes almost total volume of supplied water (80-95 %); hence, that very grade should be taken into consideration while determining volume of water delivered for dust control. In view of abovementioned following expression is proposed to identify the parameter:

$$Q_e = \frac{10^{-4} q_{m.\phi} (W_{h.n} - W_e)}{k_{h.n}}, \quad (4.1)$$

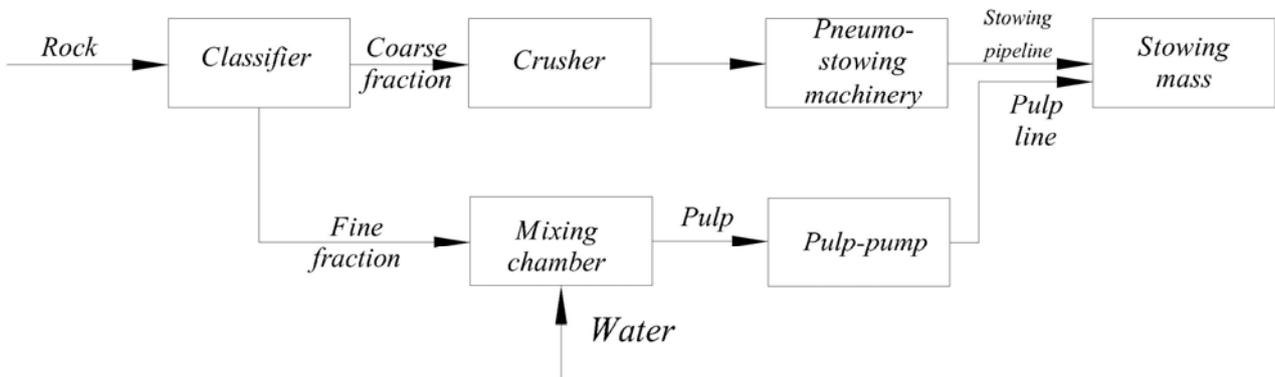
where  $Q_e$  is maximum quantity of water to be supplied to stowing pipeline,  $m^3/t$ ;  $q_{m.\phi}$  is content of fine fraction (0-3 mm) in stowing material, %;  $W_{h.n}$  is initial adhesiveness of rock (rock moisture when adhesion properties start demonstrating themselves), %;  $W_e$  is natural rock moisture, %; and  $k_{h.n}$  is stowing material cyclic variation coefficient. It is determined as ratio between maximum volume of supplied stowing material in unit time and average one, that is  $k_{h.n} = \frac{Q_{max}}{Q_{cp}}$ .

(4.1) expression and Figures 4.1 and 4.2 make it possible to say that in the context of longwalls of  $C_7^H$  seam (“Blagodatnaia” mine) extra  $0.016m^3$  (16l) of water per a ton of rock will not result in rock freezing within pipeline; that will help maintain its transporting capability as well as reduce dust release while conducting stowing operations.

Thus, analysis of stowing material adhesion properties has shown that rocks in Western Donbass represented by argillites are suitable for pneumatic conveying if natural moisture is up to 18%.

If undercut rock is of significant moisture content it is possible to reduce initial material adhesion in pipeline at the expense of fine fraction previous screening. Then fine refuse is mixed with water (rock moisture should be more than 35-40%) with

following stowing into worked-out are with the help of pulp pump. To do that extra pulp pump has been constructed in parallel with basic stowing pipeline (Fig. 4.3).



*Fig. 4.3 Schematic of stowing operations in longwalls with considerable inflow of water*

Leaving pipeline, pulp is mixed with stowing material making it possible to reduce dust release and increase stowing mass density.

#### **4.2.3. Content of combustibles within stowing material**

Content of combustibles within stowing material while using undercut rocks for stowing depends on loading capability of coal shearer. In turn, loading capability of coal shearers of 1K101 type depends upon direction of coal and rock mining. as paper [37] mentions and our research confirms, loading capability of 1K101 coal shearer is two-three times higher if it moves in the direction of its effectors location and vice versa.

In longwalls 746-bis and 719 (“Blagodatnaia” mine) coal seam was mined when coal shearer moved from conveying drift to boundary one; that is its direction was opposite to its effectors location. In this context the coal shearer was equipped with specific cowl to bare rock bench. Owing to that loading capacity of coal shearer experienced significant increase and only 50 to 60 kg of loading coal remained on the rock bench (Fig. 4.4 (a)). Rock bench was treated in the opposite direction without a cowl; after that a layer of broken rock with 0.06 to 0.1 m thickness remained Fig. 4.4 (b).

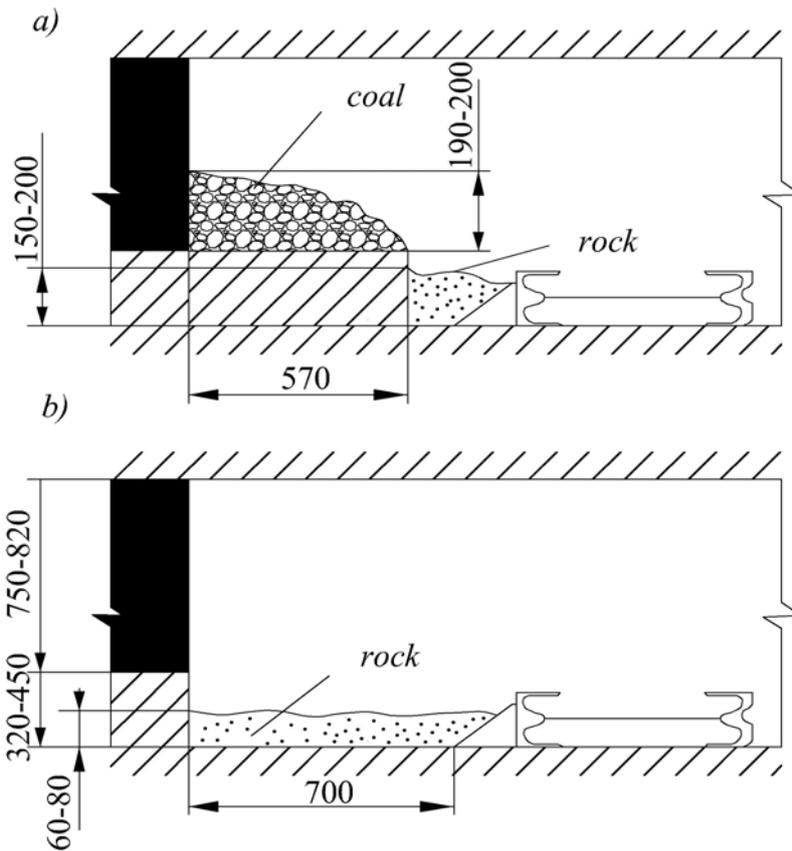


Fig. 4.4 Shape of a stope in terms of selective mining of coal (a) and rock (b) using 1K101Y coal shearer

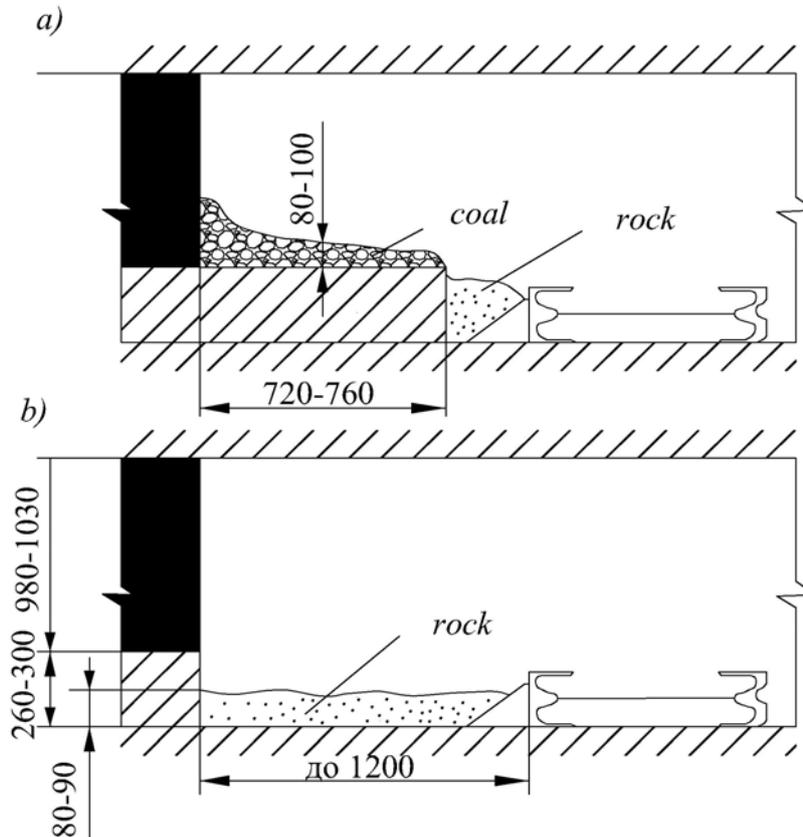


Fig. 4.5 Shape of a stope in terms of selective mining of coal (a) and rock (b) using 1K103 coal shearer

Loading capability of 1K103 coal shearer in terms of separate seam mining was analyzed in longwalls 1137 and 1162 of Geroev Kosmosa mine, and in longwall 906 of “Zapadno-Donbasskaia” mine. Within the mines separate seam mining was performed with the help of screws of 710mm diameter; moreover, coal shearers were not equipped with clearing colters.

Table 4.1 explains characteristics of conditions while researching.

Loading capacity of effectors while using 1K103 coal shearers in longwalls under study differs from that of 1K101 coal shearer. After coal shearer performed coal mining on rock bench, 5 to 10 % coal remained unloaded (Fig.4.5, a); after pass two height of rock layer on longwall floor was 0.08-0.09 m (Fig. 4.5, b).

Actual coal content in rock which can be used as stowing material has been determined by rock sampling and ash-content identification in accordance with GOST 10742-71.

Table 4.1 contains operating conditions of the longwalls while sampling.

Table 4.1 operational characteristics of the longwalls while sampling

A mine	Longwall	Type of coal shearer	Mined seam thickness, m	Geological thickness of seam, m	Undercut thickness of	
					floor	roof
“Blagodatnaia”	746-bis	1K101Y	1.2	0.76	0.44	-
Named after Geroev Kosmosa	1137	1K103	1.01	0.72	0.29	-
	1162	1K103	0.86	0.79	0.07	-
“Zapadno-Donbasskaia”	906	1K103	0.99	0.69	0.23	0.07

Coal dilution of undercut rock was determined by the expression:

$$Q_y = 100(1-k_3) \quad (4.2)$$

where  $Q_y$  is coal content in stowing material, %;  $k_3$  is dilution coefficient to be calculated on actual indices of ash-content [94]:

$$k_3 = \frac{A^d - A_{nl}^d}{A_{\sigma.n}^d - A_{nl}^d}, \quad (4.3)$$

where  $A^d$  is operational ash-content, %;  $A_{\sigma.n}^d$  is ash-content of enclosing roof and floor, %;  $A_{nl}^d$  is ash-content of a seam, %.

Table 4.2 involves data required to determine coal percentage in rock as well as calculation results.

Table 4.2 Results of research concerning combustibles contained in stowing material

Longwall	Ash-content of coal, %	Ash-content of rocks, %		Operational ash-content of undercut rock, %	Percentage of coal in rock, %
		in floor	in roof		
746-bis	12.1	88.4	-	76.1	16.1
1137	21.8	89.9	89.4	87.1	4.1
1162	22.7	89.9	90.1	82.4	11.1
906	8.7	89.9	91.0	85.8	5.2

Table 4.2 data prove that if 1K101Y coal shearer is applied then almost 16 % of coal is in undercut rock; in the context of 1K103 coal shearer the value varies within 4-11% meeting the demands placed to stowing materials.

To decrease coal percentage in undercut rock as well as rock percentage in mined coal, “Initial data for complete and selective mining of thin and very thin flat seams on the basis of 1K103 coal shearer as an integral part of МКДЗ 90 system or another one” have been developed by us. According to them 1K103 coal shearer should be completed with two loading devices for active forcing against floor; ranging arms should be lengthened in such a way to make distance between frame of coal shearer and screw sufficient for the loading devices. Frame of coal shearer should be elevated relative to lower part of conveyor clearing colter at the height of up to 300-320 mm.

We believe that such improvements for 1K103 coal shearer will make it possible to make 5 % coal saturation of stowing material.

### **4.3. Demands placed on operation schedules to mine seams in Western Donbass using stowing and their implementation techniques**

Basic factors generating a need for worked-out area stowing in Western Donbass mines are as follows:

- Protection of land, buildings and structures against undermining;
- Improving stability of enclosing roof and floor within working area of stopes;
- Improving stability of developments workings for their following reuse;
- Recycling of rock being a result of development working construction;
- Recycling of rocks being undercut within stopes in the context of selective seam mining.

The necessity to apply mining with stowing may depend on one of above factors or on several ones.

Depending upon problems to be solved while stowing, specific demands are placed to operation scheduled in addition to general ones (Fig. 4.6).

For example, while stowing worked-out area to prevent dangerous ground subsidence, primary objective is maximum density of stowing mass. The requirement is of prime importance for Western Donbass mines where ground subsidence is huge. Increase in stowing mass density may be a result of applying stowing materials with high compressive features (for instance, in terms of 25-30 % of sand is added to rock; use of stowing pipeline with sublevel caving or lateral one with minor trajectory deviation of stowing material movement from pipeline axis; correct distance between rock caving; optimum sprinkling parameters). Stowing mass should not have any hollows; besides, if possible it should fill the whole worked-out space between roof and floor.

Decrease in enclosing roof and floor convergence within working area of longwall is also important for ground subsidence minimization. According to our analysis, value of enclosing roof and floor is proportional to width of working area; thus, it is required to perform maximum approach of stowing mass formation area to stope as well as its construction promptness.

The research has showed that when roof is controlled by means of complete stowing, increase in enclosing roof and floor stability within working area of longwall takes place; in many cases, load upon powered support decreases. Hence, in the context of Western Donbass mines stowing is expedient while mining seams under complicated mining and geological conditions.

Most commonly roof rock caving takes place within end sections of longwall. To prevent the caving as well timber consumption saving, seams should be mined out using goaf pack building where stope-development workings junction takes place. Protection of drifts with the help of packs will make it possible to decrease values of development working roof and floor by 30% to compare with protection of drifts with the help of chokes; that creates conditions for their reuse.

Plans of Western Donbass mines did not involve construction of stowing systems; that is why delivery and transportation of stowing materials to stopes without reconstruction of mines is very difficult procedure. Moreover, refinement tailings contain much water which prevents them from being used as stowing material without their preliminary drying.

We believe that practices of home coal mining enterprises show that underground district crushing and stowing plants as well as application of materials left after development workings construction and undercut of rocks within longwalls is expedient for Western Donbass mines.

In the context of area under study, developments workings are driven on coal with enclosing roof and floor undercut. Thus, to use rocks as stowing material it is required to separate them from mineral. That can be achieved by means of separate coal conveying to the surface, and rocks to the crushing and stowing plant; besides, it is required to apply longwall systems as well as combined ones while driving developments workings after longwall.

As practices of “Blagodatnaia” mine demonstrate, undercut rock stowing into worked-out area when separate coal and rock mining is applied make following demands: smooth coordination between processes of mining and stowing; and minimum coal content in stowing material. To do that stowing equipment efficiency should correspond to mining system efficiency; moreover, downtime of each component should have minimum effect on efficiency of another component.

In the context of Western Donbass, seams are mined out within unstable enclosing roof and floor; that is why worked-out area stowing should be protected by means of back overhand fixed to support canopy from goaf side.

While selecting operation schedules it is also required to develop normalized schedule to mine seams without dangerous surface subsidence; to reuse development workings; and to effect low-waste production.

Fig. 4.6 demonstrates general picture of factors stipulating the necessity to apply worked-out area stowing; demands placed on operation schedules with stowing; and implementation techniques.

#### **4.4. Substantiation of basic constructive features of mining and stowing team of equipment**

Mining and stowing powered system (MSPS) is designed for integrated mechanization of mining and stowing operations in the context of thin and very thin flat seams in the process of longwall technique, long-pillar technique, and combined one.

MSPS is multipurpose system; that is why it should solve following production problems as well:

- Engagement in both complete and separate mining as well as in coal and undercut rock conveying within longwalls with thin and very thin seams and soft enclosing roof and floor;

- Construction of tight stowing mass both in the context of the whole length of longwall and within its end sections;

- Possibility to stow undercut rocks, rocks left after development workings neighbouring the longwall as well as rocks conveyed from central crushing and grading plant or sectional one into worked-out area.

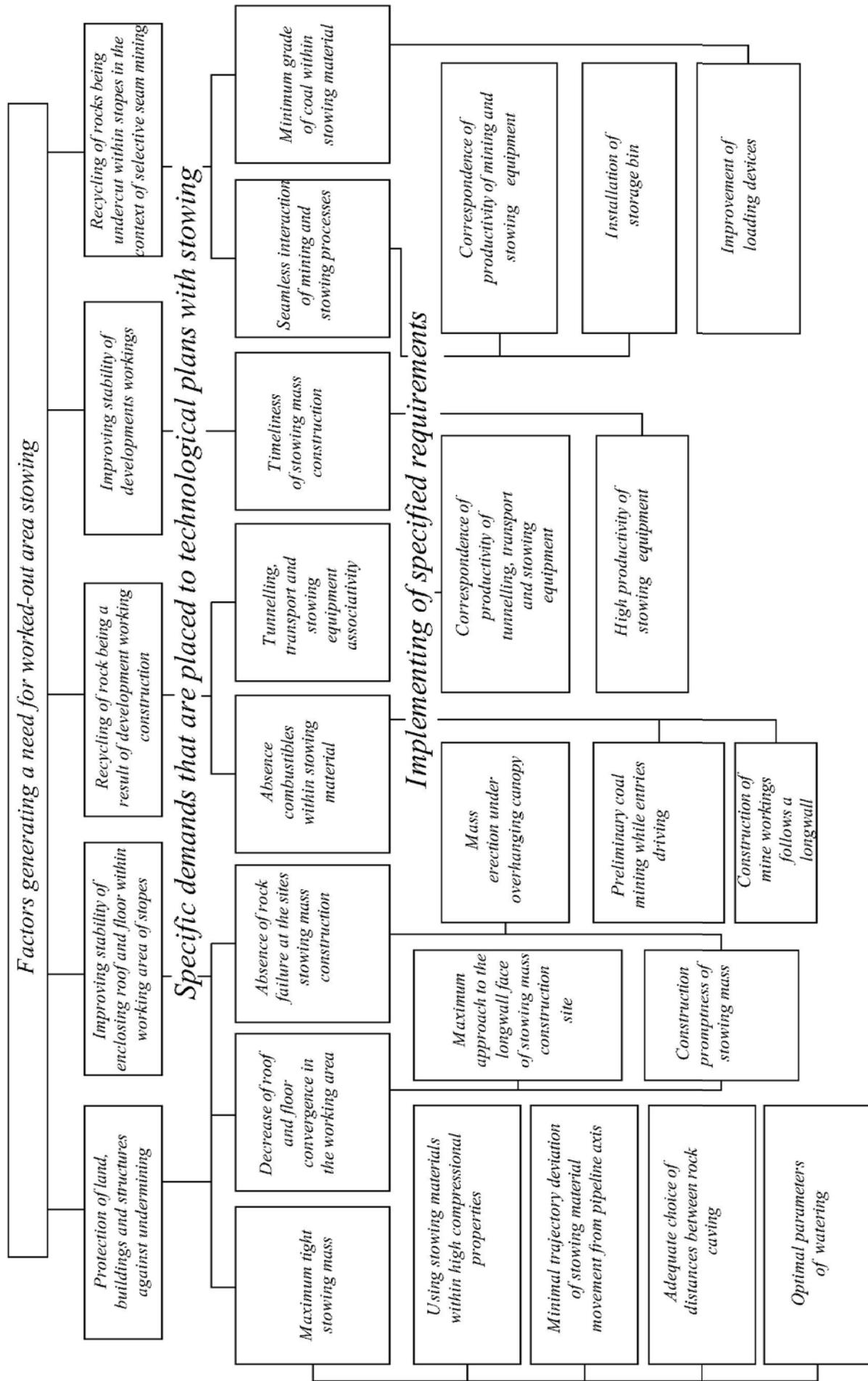


Fig. 4.6 The demands placed on operation schedules with stowing and their implementation techniques

To perform basic functions and solve the problems, demands placed on components of mining and stowing systems have been formulated. The demands and their implementation techniques are generalized in Table 4.3.

Table 4.3 Demands placed on MSPS components as well as implementation techniques

Demands placed on MSPS components	Procedures to implement the demands
1. Coal shearer	
Possibility to perform both complete and separate mining of coal and undercut rocks.	Use of coal shearers with selective-effect end organs. Possibility to apply small-diameter screws (0.56; 0.63 m).
High loading capacity.	Possibility to lift up either cowl or loading colter of coal shearer on rock bench. Rock bench scraping with the help of a frame of coal shearer. Adequate choice for coal and rock mining direction
Potential for efficient operation of coal shearer in terms of shuttle scheme.	Screw spacing along the length of coal shearer. Opposite rotational direction of screws.
2. Longwall conveyor	
Potential for regular operation in a reverse mode.	Two drive heads.
High loading capacity in terms of haulage	Clearing colter for conveyor.
3. Powered support	
Protection of stowing mass construction site against potential roof rock caving	Back overhang from goaf side of canopy.
Haulage of the support sections in terms of stowing operations and in terms of their nonavailability	Independent connection of the support sections with stowing pipeline. Back overhang from goaf side of canopy.
Safe protection of face area against dispersion of stowing material fragments.	Protection of working area of longwall. Drop-holes within goaf walls.
Access of workers to stowing equipment	Distance between supports along the length of longwall should not be less than 0.7 m.

High coefficient of roof lagging	Minimum intersectional gaps between canopies. Section haulage right after pass by coal shearer.
High stability of supports	Four-support design. Extensive area of the support section basement. Back goaf-side overhang.
Narrowing of working area of longwall	Approximation of first row of supports to stope.
4. Stowing equipment	
Maximum tightness of stowing mass	Either sublevel rock caving or side one with narrow deflection angle. Stowing pipeline minimum bending. Minimum leaks of compressed air within pipeline. Optimum selection of construction pitch for stowing mass and distance between rock caving.
Minimum labour intensity for stowing pipeline maintenance.	Hydraulic remote connection/disconnection of stowing pipeline. Remote control of side rock caving. Hydraulic haulage of pipeline.
Minimum dust release while stowing	Stowing material watering either within a bin of crusher or SS. Supply of water to stowing pipeline. Availability of water curtains within areas of intensive dusting.

All components of mining and stowing powered system should be synthesized into integrated mechanism.

Basic demands placed on associativity of MSS components are correspondence of their efficiency and dimensions.

Mining and stowing system МКДЗ - 90 designed by “Dongiprouglemash” is best matched to the demands. According to our recommendations, coal shearers of УКД type are applicable to use МКДЗ - 90 system for selective seam mining. In addition, application of such coal shearers will make it possible to narrow working area by 0.7 m; that will result in decrease of enclosing roof and floor convergence and, hence, decrease of surface deformations.

While using such equipment, MSS involves following components:

- Coal shearer of УКД type with end organs of selective action;
- Powered support ДЗ-90;
- Face-end supports КСД-90;
- Stowing device of ПЗБ-200 or ZS-240 type, or crushing and stowing systems "Titan-1" and "Titan-1M";
- Crushing and conveying machinery; and

- Heading equipment if longwall mining technique or combined one is applied. Such a MSS alternation can be used if only mining and stowing system МКДЗ-90 is in production chain.

Another MSS is available before design activates are completed as stowing is in acute need for Western Donbass mines. The system is based upon application of standard equipment. In this context mining and stowing system involves:

- Coal shearer of 1K101УКД, 1K103, УКД 200-250 type;
- Powered supports equipped with back overhangs used in mines of the region;
- Standard crushing and stowing equipment, and conveying one;
- Heading equipment to mine a seam in terms of longwall mining technique or combined one.

If such an MSS alternative is applied then powered support is equipped with rigid back overhang or back tightening one; if stowing track is rather wide, then stowing pipe is equipped with hydraulic docking devices.

In addition to above MSS alternatives, mining and stowing system based on КМК-97М (Д) system is also applicable for Western Donbass mines. Design of the system belongs to DonCI experts [44]. However, functional area of such a system is limited by seams where bottom is hard at least.

#### **4.5. Basic techniques to mine thin flat seams with worked-out area stowing**

Seams with 0 to 2-3° slope angles occur in Western Donbass. Enclosing rocks have a tendency to slaking; while humidifying, they loose up to 50 % of their strength. That results in caving and intensive heaving. Thus, hydraulic stowing is inefficient in the context of the Donbass region. Mechanical equipment for efficient stowing of large worked-out areas is not available today. In accordance with our opinion as well as opinion of other experts, pneumatic stowing is the most expedient technique for the conditions under discussion.

A number of operation schedules involving pneumatic stowing of worked-out area may be applied to mine out seams in Western Donbass. First, operation technique depends on the problems to be solved in the process of stowing as well as geological, mining, and technological conditions.

Operation techniques appropriate for Western Donbass conditions are grouped together in Fig. 4.7. In contrast to available operation schedules developed ones meant for seams mined with enclosing roof and floor undercut involve use of undercut rocks as stowing material; moreover, in some cases they are combined with other sources of the latter. The grouping together is based on the six chief features: mining technique, stowing type, rock source, type of mining equipment in use, location of stowing equipment, and its air supply source.

Following operation schedule is expedient to be applied for complete stowing of worked-out area with 0.65 up to 0.8 m thickness. Grouping from Fig. 4.7 makes it possible to say that the scheme consists of 1.3(1.3.1)-2.1-3.4-4.1-5.1-6.2 components. In other words, the technique involves application of combined mining with boundary entry in advance of a stope (Fig. 4.8). Reuse of the boundary entry as conveyor one while neighbouring stook mining is possible if cross-section of boundary entry is 15 to 18 m while driving.

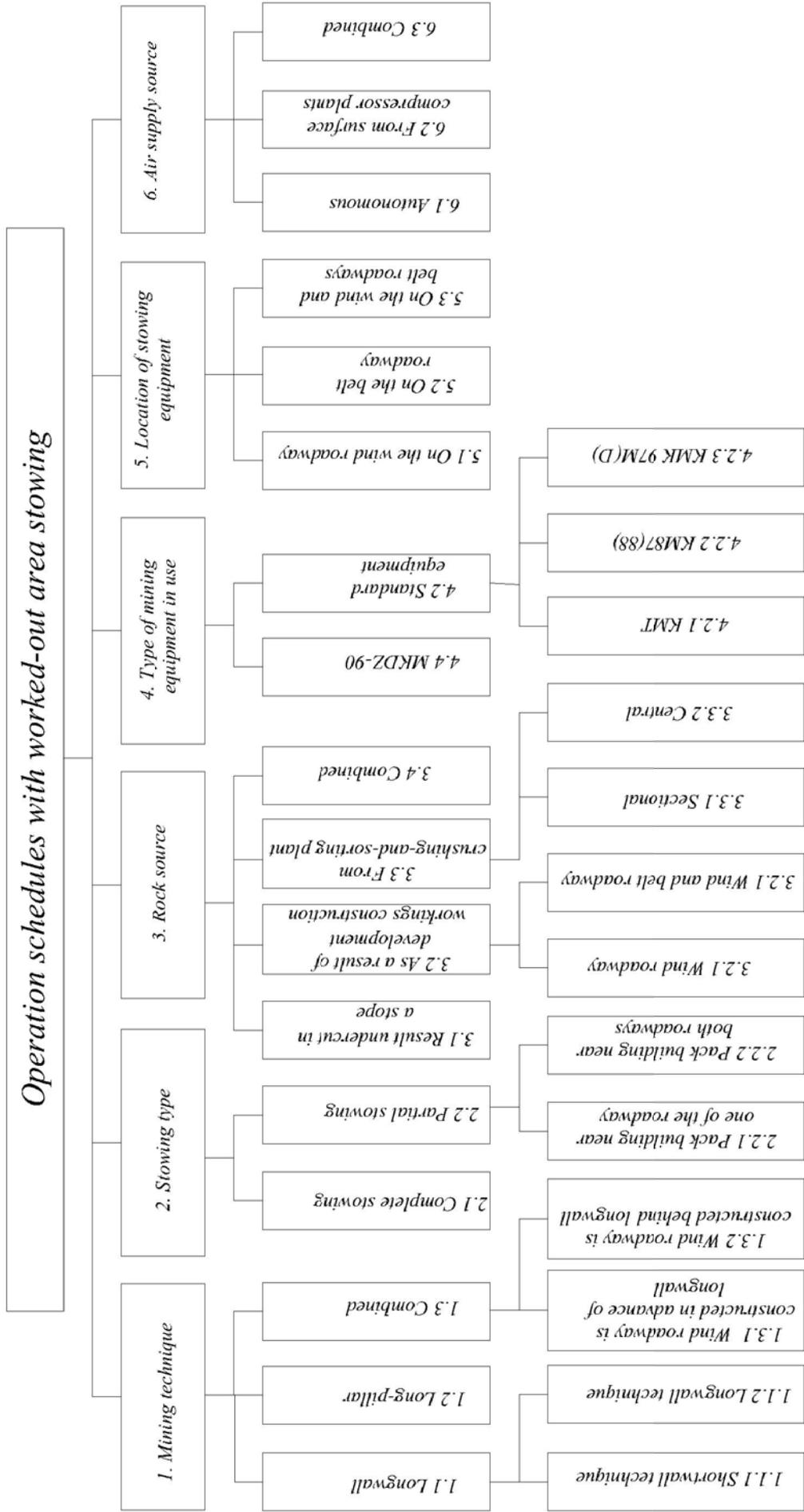


Fig.4.7 Grouping of proposed operation schedules

Mining machines of ГПКС, 4ПП2М, and 4ПП5 type are applicable.

Seam is mined with the help of mining and stowing system МКДЗ - 90, and coal shearer 1К103М. Stowing machine of ПЗБ or ZS-240 type is used. It should be noted that while mining rock, longwall conveyor operates in reverse mode.

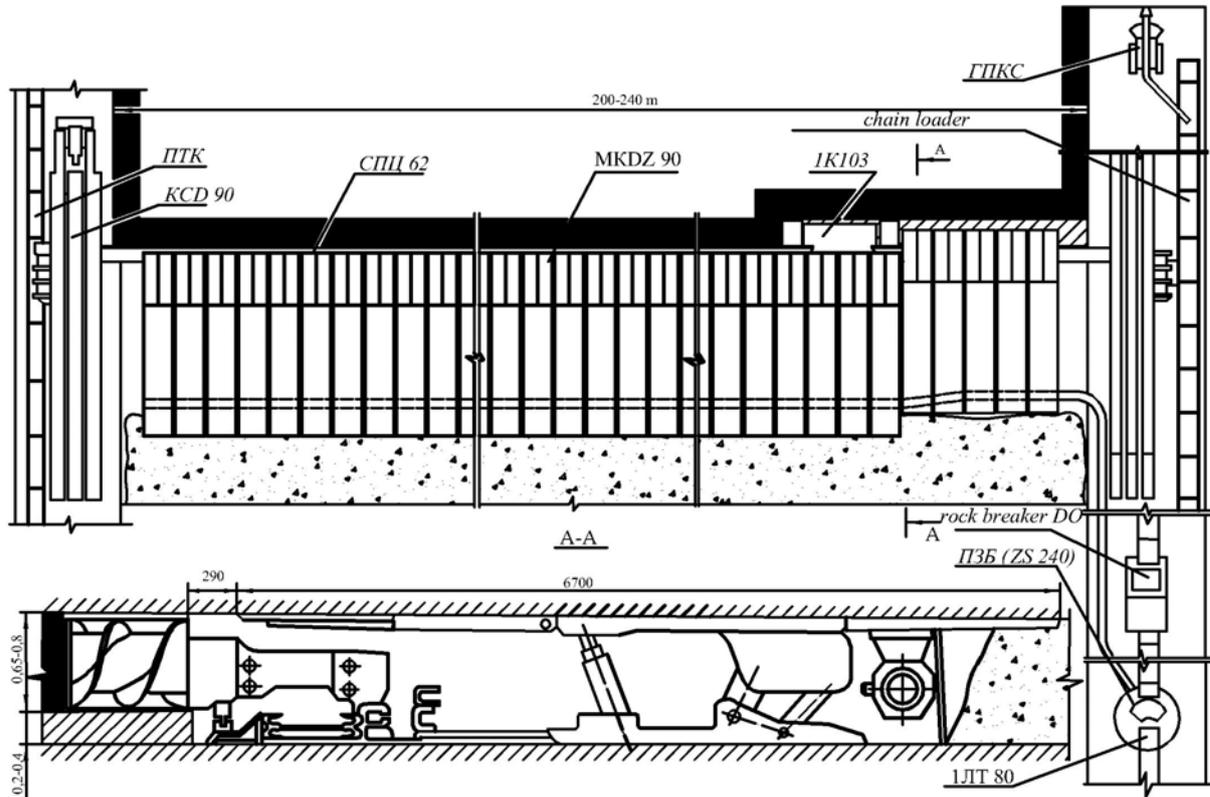


Fig. 4.8 Operation schedule for selective seam mining and undercut floor rocks

Rock required for complete stowing is conveyed to stowing machine from crushing-and-sorting plant; the rock is a result of construction of workings in other sections or seams where stowing is not applicable.

To crush rock from a stope and drifting face neighbouring longwall, a crusher of ДО type is applied.

If complete stowing of worked-out area is not required, and stowing is applied for rock disposal there is no necessity for extra volume of initial material from crushing-and-sorting plant; thus, only undercut rock as well as rock left after drift construction are stowed into worked-out area. In this context, operation schedule of stowing is simplified. If rock volume to be stowed is small, then crushing-and-stowing system "Titan-1" (or "Titan-1M") is used instead of stowing machines ПЗБ or ZS-240. The matter is that former goes without air pipe-lines from surface compressor plant to stowing machine.

The best condition of boundary entry is possible if it is constructed after longwall; that is why such an operation schedule is also feasible. However, in this case, unit of rock reload from longwall conveyor to chain feeder within boundary entry experiences its complication.

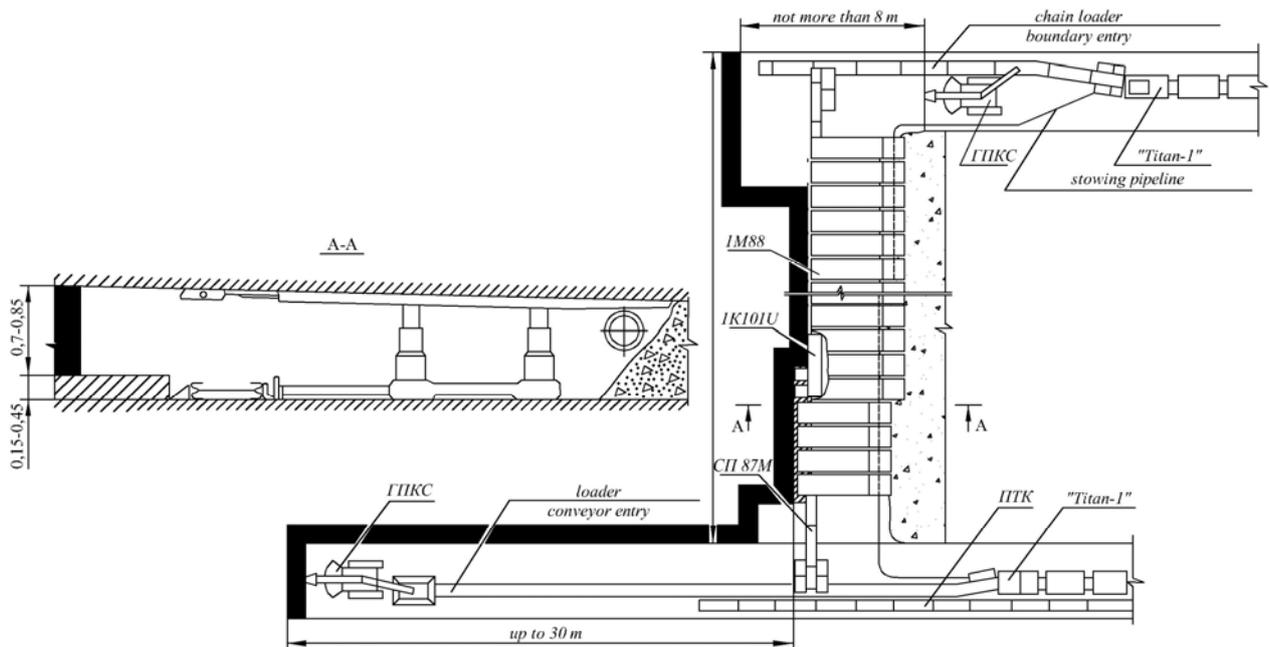
Above operation schedule may be applicable in the context of longwall system. If so, boundary entry is constructed after longwall, and conveyor one is constructed in its advance. Rock left after conveyor entry is stowed into worked-out area with the help of crushing-and-stowing system "Titan-1" mounted within the same entry. Proposed grouping makes it possible to express operation schedule as follows: 1.2(1.1.2)-2.1 (or 2.2)-3.4-4.1-5.3-6.3.

All proposed operation schedules are based upon mining-and-stowing system МКДЗ - 90 which full production has not been started yet. Moreover, the majority of the schedules involve air supply of stowing facilities from surface compressor plants not available in the most of Western Donbass mines.

That is why before full production of МКДЗ - 90 system and construction of compressor plants start, worked-out area stowing may be performed with the help of standard mining equipment and crushing-and-stowing system "Titan - 1" (or "Titan - 1M") as their air supply is effected from own, independent source.

In the context of such equipment use, longwall length is controlled by maximum distance of rock transportation by means of stowing machines "Titan-1" or "Titan-1M" being 80 and 100 m respectively. Thus, complete stowing of worked-out area requires two systems mounted within different entries. As pipeline in the neighbourhood of stowing machine should have 15 up to 20 m acceleration area, in terms of two systems application, longwall length should not be more than 120m for stowing machines "Titan-1" and 160m for "Titan-1M".

In the context of longwall method complete stowing of worked-out area may be provided with rocks left after undercut as well as after construction of ventilation entry and conveyor one; however, in that case longwall length may be less than that determined on maximum distance of rock transportation. Fig. 4.9 demonstrates operation schedule with the use of KM88 system and two crushing-and-stowing "Titan-1" systems in the context of longwall method. With the help of classification (Fig. 4.7) the operation schedule may be described as follows: 1.1(1.1.1)-2.1.-3.4-4.2(4.2.3)-5.3-6.1. If the schedule is applied, then two-pass selective coal and rock mining is involved. Undercut rock is stowed into worked-out area with the help of "Titan-1" system. Moreover, rocks left after construction of boundary entry and conveyor one are also stowed into worked-out area. In this context conveyor entry is constructed in advance of longwall; boundary entry is constructed behind it. Rock being a result of mine working construction is conveyed to stowing machines by means of chain feeders mounted within each drift; then it is stowed into worked-out area.



*Fig. 4.9 Operation schedule of complete stowing in the context of seam mining by means of shortwall technique*

Such a technique requires similar volumes of rock delivery to both stowing machines; that's why undercut rock is transported both to boundary entry and conveyor one. While transporting rock to boundary entry, longwall conveyor reverses. Specific valve is mounted within conveyor entry where rock is transferred to prevent rock from getting on coal chain-and-flight conveyor.

If thickness of coal seam is 0.7 m, undercut thickness is 0.4 m, and rock section of both entries is  $15.7 \text{ m}^2$  then longwall length should be almost 100 m to provide complete stowing of worked-out area.

The technique based on standard mining equipment application is applicable both for combined mining and pillar mining to utilize rock left after development working construction and coal-cutting. Such operation schedules have been tested in experimental longwalls of "Blagodatnaia" mine. Figures 3.2 and 3.3 demonstrate them.

It should be noted that operation schedule has to be implemented with minimum material expenses. It means that implementation of any operation schedules should be preceded by thorough economic study of each possible alternative under specific mining and geological conditions to determine conclusively.

#### **4.6. Efficiency of recommended technique**

A number of papers (f.i, [37, 91]) substantiate efficiency of selective seam mining. Western Donbass confirms to the conditions. The papers point out that selective coal and undercut rock mining when the latter are hoisted on the surface is expedient if undercut thickness is more than 0.1-0.15 m. Annually 5 percent decrease

of mined coal ash-content in average produces 2150-2700 thousand RUB profit (at 1990 values) [91].

While applying techniques with undercut rock stowing into worked-out area economic effect is possible owing to slowdown in expenditures connected with rock transportation within underground workings, its hoisting on the surface, and slowdown in expenditures connected with maintenance of development workings. While reusing drifts, economic effect is also achieved at the expense of their construction cheapening.

Certainly, economic effect is only possible if above expenditures are higher than expenditures connected with worked-out area stowing.

It should be noted that a number of mines have not any feasibility to achieve selective rock hoisting from stopes on the surface due to shortage of transport facilities, low capability of rock grade etc. making the technique of selective seam mining with worked-out area stowing non-alternative.

When the technique is applied in the context of combined mining and pillar one increase in mine profits is possible owing to lowering cost for transportation and rock from development faces winding. Taking into consideration the fact that the majority of drifting faces in Western Donbass mines involve rock transportation with the help of coal conveying facilities its stowing into worked-out area will make it possible to upgrade mined coal and cut expenditures connected with rock mass preparation.

Complete worked-out area stowing involves crushing-and-grading facilities as well as compressor plant equipment. That will require extra capital expenditures in the amount of almost RUB 6mln [63]. Moreover, total operation cost to stow worked-out area taking into consideration expenditures connected with preparation of stowing material, its conveying, cost of compressed air, maintenance of pipelines, and wages may reach 9-11 RUB at 1991 values [63]. Thus, in some cases expenses connected with complete worked-out area stowing will excess profit by cost saving for rock conveying, and maintenance of mine workings and their construction. However, in this situation indirect economy will take place owing to conservation of certain share of underworked land or cost saving for disturbed land site restoration. In recent years, land values experience constant increase; larger and large sums are consumed by reparation of damage connected with loss of farm products which could be raised etc. That is why social importance of techniques involving worked-out area stowing will increase year on year.

## **Conclusions**

1. Following factors restrict application of the technique:

- Properties of rocks used as stowing material;
- Design of mining equipment in use; and
- Economic feasibility.

2. Undercut rock of stopes is applicable to be stowing material if only their moisture content is up to 18 %.

3. Stowing material involves up to 35 % of fine refuse of 0 to 6mm grade to be a source of considerable dusting in longwall.
4. Almost 80 % of rock undercut as a result of selective mining is suited for pneumatic conveying without extra crushing.
5. An expression has been developed to determine water amount for stowing material to control dust. The expression takes into consideration adhesive properties of rock and content of fines in it.
6. Stowing material being undercut floor rock contains almost 16 % of coal if 1K101 coal shearer operates, and 4-11 % if 1K103 coal shearer is applied. That meets the requirements for stowing materials.
7. Basic design features of mining and stowing system have been substantiated; the system can be based on standard winning equipment and on МКДЗ - 90 system being developed. It has been determined that modern coal shearers of УКД type may be used as winning machine as part of МКДЗ - 90 system. That will help successfully use the system for selective seam mining.
8. Operation schedules to mine thin flat seams with worked-out area stowing have been developed. They involve use of undercut rocks of stopes as extra source of stowing material.

## OVERALL CONCLUSION

The monograph is a result of completed research. It has solved topical theoretical and practical problem concerning substantiation of basic parameters of technique to mine thin flat seams with worked-out area stowing in the context of Western Donbass mines.

Following the most important results have been obtained as part of the study.

1. Analysis of rock mass stress-strain state has helped determine that:

- Worked-out area stowing results in 20-40 % index zone narrowing to compare with complete caving and decrease in bearing load on coal seam;
- While worked-out area stowing, main roof deformations are 1.7-2 times less to compare with complete caving; they are not more than 5-6 mm/m preventing fall within working area of longwall and area where stowing mass is constructed;
- Changes in longwall length while worked-out area stowing within complete undercut of percarbonic mass results in minor changes (no more than 10 %) of roof rock subsidence and deformation in terms of working area of longwall making it possible to increase longwall length up to economically feasible value.

2. Dependences of width of stowing track, feed velocity of coal shearer while mining, machine time coefficient of longwall on enclosing roof and floor undercut thickness in terms of different mining and stowing facilities have been determined. The dependences take into account peculiarities of undercut rocks of a stope stowed into worked-out area. That makes it possible to improve accuracy of parameters of a technique involving seam mining with stowing.

3. The dependence of changes in total expenditures on longwall length helps conclude that in terms of mining and stowing system МКДЗ - 90 application with complete stowing of worked-out area, 10-15 % longwall lengthening is expedient while complete mining.

4. Underground investigations have determined that:

- In the context of complete stowing enclosing roof and floor convergence within working area of longwall is 13-14 % less to compare with complete caving;
- Support load decreases by almost 35 % if worked-out area stowing is applied;
- Compression of a pack being undercut rocks of a stope is almost 35%, and total convergence of enclosing roof and floor is 48-49 % of mined thickness of a seam.

5. Laboratory tests and underground investigations have shown that undercut rocks of a stope are applicable as stowing material if moisture content is less than 18 %. Dependence of water amount delivered to water stowing material on its natural moisture content. The dependence helps control dust discharge and improve efficiency of stowing material pneumatic conveying.

6. Basic design features of mining-and-stowing systems which can use both standard equipment and available МКДЗ - 90 system have been substantiated. It has been determined that modern coal shearers of УКД type are applicable as mining machines as a part of the system. That will help use the system for selective seam mining.

7. Principal operation schedules have been developed to mine thin flat seams with worked-out area stowing. The operation schedules involve use of undercut rocks of a stope as stowing material. Conclusion has been made that to mine seams with 0.65-0.8 m the most expedient idea is to apply operation procedure involving combined mining technique and use of undercut stope rocks as well as rocks left after air working construction.

Both design and development of МКДЗ - 90 system take into consideration results of the research.

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