

Perspective

Research into Impact of Leaving Waste Rocks in the Mined-Out Space on the Geomechanical State of the Rock Mass Surrounding the Longwall Face

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Abstract: Backfilling technology is not always used by mining enterprises, which is conditioned by technological and economic factors, such as the need for high mining rates and costs for the technological processes of transporting backfill materials from the daylight surface to the mined-out space. This concerns the underground mining of hard coal, which is a strategic energy resource, in the mines of Ukraine. This paper aims to study the effect of leaving the waste bottom rocks in the mined-out space of the longwall face without their drawing to the earth's surface on the geomechanical state of the rocks surrounding the longwall face. The geomechanical assessment of the stress state of the rock mass surrounding the longwall face, when leaving the waste rocks from the seam bottom rocks in the mined-out space, is performed by the finite element method using the Ansys software package. A geomechanical model has been developed and substantiated, which adequately reflects the mining-geological conditions for seam mining within the extraction site, the actual structure and properties of the coal-bearing rock stratum, the parameters of the longwall face and the modified powered support for the processes of leaving the rocks in the mined-out space. The values and patterns have been determined of the decrease in the stress intensity concentrations in the coal-bearing roof mass in the frontal bearing pressure zone and destressing zone with an increase in the ratio of the rock pack thickness to the extracting seam thickness. The relative indicators of the load on the powered support section and the lowering of its roof have been determined by the ratio of the thickness of the rock pack formed in the mined-out space to the extracting seam thickness. The proposed mining method is of significant commercial and research interest for owners of coal mines developing thin coal seams because environmental costs for placing waste on the surface are reduced, and the energy potential of coal is increased due to the separation of waste rocks from coal in underground conditions. The need for a cycle of beneficiation of mined mass is eliminated and the geomechanical conditions of coal mining processes are improved.

Keywords: coal seam; longwall face; selective mining; undercut rocks; rock pack; stress concentration; support loading



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1. Introduction

Hard coal is an important energy resource that contributes to the development of the economy in many countries [1–3]. However, the share of coal in the global energy balance has declined significantly over the past few years, which is primarily due to the decarbonization course taken by a number of developed countries and the development of alternative energy sources to improve the state of the natural environment [4,5]. In the light of recent military-political events in Ukraine and the world, a change in the global energy balance is expected, since countries that have completely or partially abandoned

Russian natural gas are forced to look for an alternative by importing energy resources from other countries and, first of all, developing their own fossil fuel reserves. In this regard, the relevance of coal mining will increase significantly. This applies primarily to European countries such as Poland, the Czech Republic and Germany [6,7]. Sufficiently high volumes of coal mining are observed in China, India, Australia and the USA [8]. For Ukraine, hard coal has been and remains a strategic energy resource [9–11].

As known, underground coal mining is accompanied by a significant negative impact on the state of the natural environment. Of the most important problems, the earth's surface subsidence and waterlogging of territories [12–14], accumulation of coal mining waste on the earth's surface [15–17], atmospheric pollution with mine gases [18,19] and pollution of water bodies by highly mineralized mine waters [20,21] should be noted. Today, many types of minerals are mined on the basis of “green mining” principles in order to minimize the harmful impact on nature [22–25].

At the planning stage of mining mineral deposits occurring at great depths, in order to assess the rock mass state, it is necessary to use geophysical research methods in order to select the optimal project solution [26,27]. Then, in order to make managerial decisions on leveling technological production in order to remove the negative impact on the natural environment, it is necessary to perform geomechanical process modelling, based on data obtained from geophysical surveys. This can ensure the safe mining of a mineral deposit by surface, underground or combined methods, without incurring additional financial costs.

A generally recognized effective method to reduce the negative impact on the geological environment and the earth's surface is the backfilling of the mined-out space in the mines [28–31].

In Ukraine, the conditions for the occurrence of coal seams are the most difficult among the countries where underground coal mining is developed. This is primarily conditioned by the low seam thickness (80% of coal reserves are in seams < 1.0 m thick), low strength and stability of rocks, as well as high gas content [32–34]. The main coal-mining capacities of Ukraine are concentrated in the Western Donbass, where eight mines produce almost 70% of the country's total coal. In this region, coal mining without backfilling has already led to serious environmental consequences: more than 100 million tons of waste rock and 300 million tons of beneficiation tailings have been accumulated in dumps, and the area of flooded and waterlogged territories due to surface subsidence is 17.0 km² [15,35,36].

As world experience indicates, basically all known methods of coal mining with backfilling are conducted from the surface by supplying the backfill mixture (rocks, tailings, ash, cement, water) through a network of pipelines laid in the mined-out space [37–39]. However, for the Western Donbass conditions, the big problem of backfilling the mined-out space when mining thin seams is the lack of effective technology and cheap backfill material. The technology of backfilling could significantly limit the technical and economic performance of mines. However, a contradiction arises, since in order to maintain a stable high productivity of mines, reliable operation of longwall faces equipped with mechanized complexes is necessary. Thus, in difficult mining-geological conditions, cases of landing sections of a powered support on a rigid base are quite frequent, which leads to a stoppage in coal mining [40–42]. In some cases, this leads to the complete loss of support sections in the longwall face and the cessation of mining the extraction pillar. This is caused by complex geomechanical transformations in the roof rocks, the pressure of which exceeds the load-bearing capacity of the support [43–45]. In addition, heavily diluted coal is mined (ash content is up to 40–50%), since the set extracting seam thickness is 1.05 m, and the geological seam thickness is in the range of 0.7–0.9 m [35]. This leads to the necessity of undercutting bottom rocks, thereby diluting the coal and the technological cycle of beneficiation.

Previous scientists have studied the geomechanical state of the roof rocks, mainly for thick coal seams with the formation of a solid backfill mass in the mined-out space of the longwall face [46–49]. However, the conditions for mining of precisely thin coal seams and the placement of mine rocks behind the longwall face over the area of the mined-out space

without their drawing to the surface, as well as their influence on the geomechanical state of the roof and powered support, have not been sufficiently determined.

For the conditions of thin coal seams, a new method of selective coal mining with leaving the rocks in the mined-out space of the longwall face has been developed [50]. This method makes it possible to mine coal with low ash content (15–18%) and place the undercut rocks in the mined-out space of the longwall face without their drawing as part of the rock mass to the earth's surface, thereby eliminating the beneficiation cycle. This process cannot be called backfilling, since the rocks are simply stockpiled without the formation of a backfill mass with the desired properties. This mining method is a theoretical development protected by copyright patents. Unfortunately, backfilling of the mined-out space or leaving the waste rocks in the underground space in the conditions of the Western Donbass mines is not applicable for economic reasons. Nevertheless, preliminary assessments of the complex environmental and economic effect of using the new method are impressive. Therefore, this research is aimed at an evidence-based assessment of the improvement of the geomechanical environment around the longwall face—one of the many positive effects of using the new method. The new method makes it possible to involve into mining thinner coal seams (0.6–0.7 m), that is off-balance reserves, in order to extend the operation of coal mines. Using this method, it is of scientific interest how the broken rocks of the seam, which are placed in the mined-out space, affect the geomechanical situation around the longwall face.

Thus, the presented paper is intended to study the effect of leaving the waste rocks from mining the bottom rocks in the mined-out space of the longwall face without their drawing to the surface and using selective technology on the geomechanical state of the roof rocks. The authors' research results make an original contribution to the development and substantiation of the effectiveness of low-waste technologies for coal mining from thin coal seams.

2. Study Area

The mining-geological conditions of the Zakhidno-Donbaska mine, which develops steam coal reserves in the Western Donbass region, have been chosen for the research. At the moment, there are eight coal mines operating in the region, which are part of the PJSC DTEK Pavlohradvuhillya (Ukraine). The Zakhidno-Donbaska mine is one of the most promising mines in the region, with an estimated operation term of at least 50 years.

The mining-geological conditions of the 861 longwall face extraction pillar are considered for performing a geomechanical assessment of the state of the longwall face roof mass while leaving the seam bottom rocks in the mined-out space. The extraction pillar is located within the mine field of block No. 3 and is mined to the rise. The layout of the mine and 861 longwall face extraction pillar is shown in Figure 1. The choice of the test site for the research is conditioned by several factors. Firstly, within the mine field boundaries throughout the C_8^b seam, 14 million tons of coal is concentrated in the area of distribution with a seam thickness of 0.55–0.80 m. This is a huge reserve, with an average annual mine production of two million tons. In this case, the traditional method of mining reserves of 0.55–0.80 m is economically unprofitable, but with the new method of selective coal mining with leaving the rocks in the mined-out space of the longwall face, it is expedient [36]. Secondly, to the west of the central group of mine shafts is the floodplain of the Ternovka River, and the undermining of this territory in the future will cause a probable subsidence in the earth's surface and waterlogging of the territory. Thus, leaving the rocks in the mined-out space will reduce these negative phenomena.

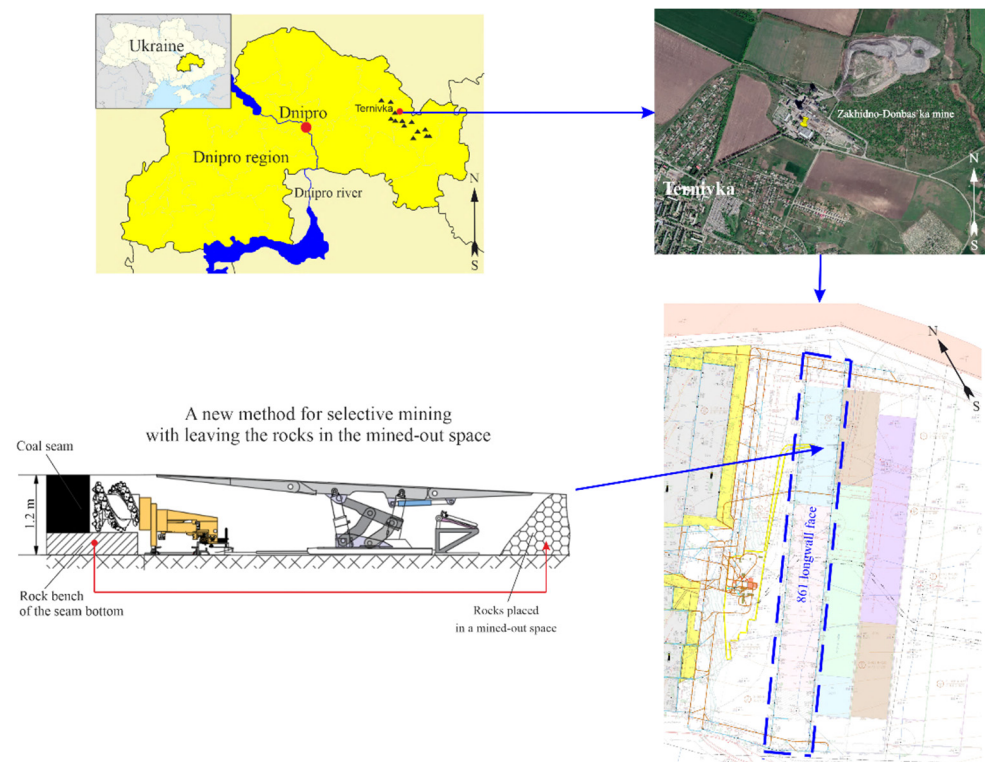


Figure 1. Layout of the mine and research object.

The C_8^b coal seam occurs with a dip towards the northeast at an angle of $3\text{--}5^\circ$. The seam thickness ranges from 0.75–1.0 m, and the average coal seam thickness is 0.85 m. The contacts of the coal seam with the host rocks are even, sharp, and the adhesion is weak. The main roof is represented by intercalation of argillites, siltstones, sandstones, coal seams and interlayers. The immediate roof is represented by horizontally stratified unstable argillite. The immediate bottom is represented by argillite of a lumpy texture, unstable and, when moistened, subject to losing its load-bearing capacity and swelling.

The primary main roof landing is expected at a distance of 30–45 m from the installation chamber (from the experience of mining similar longwall faces). The natural methane content of the C_8^b coal seam is $10\text{ m}^3/\text{t}$ on average. According to emission and rock burst factors, coal and host rocks are not hazardous. The longwall face length is 280 m, and the extraction pillar length is 2650 m. The average mining depth is 440 m.

To effectively use the new method of selective seam mining with leaving the rocks behind the longwall face, a new type of powered support with a reverse cantilever has been developed [48], and the specified effective extracting coal seam thickness is 1.2 m. At present, with traditional bulk coal seam mining (seam + bottom undercutting), the extracting seam thickness for the conditions of the Western Donbass mines is 1.05 m. Thus, with an extracting coal seam thickness of 1.2 m, according to the selective coal seam mining technology, depending on the change in the geological seam thickness along the extraction pillar length, the height of the rock bench mined in the longwall face will be 0.2–0.45 m. Broken rocks of the seam bottom bench are placed in the mined-out space of the longwall face using a horizontal-loop conveyor. In this case, the loosening coefficient of broken bottom rocks is 1.6–1.7 [51].

Technological peculiarities of the studied mining method operation are as follows:

- Coal mining process—in the initial position, the support and the conveyor are pushed to the face, the powered support sections are unfastened, the drive heads of the conveyor are moved, and the shearer is cut into the coal seam at the ventilation drift. The front auger in the direction of the shearer movement is set in the upper position and adjusted along the seam roof, while the rear auger is adjusted along the bottom

for mining the remaining coal pack. As the shearer moves from the ventilation drift to the conveyor drift, coal is mined without undercutting the bottom rocks. As the shearer moves towards the conveyor drift, the support sections are moved one by one, while the bottom-hole and backfill lines of the conveyor are not moved.

- The process of mining the rock and leaving it in the mined-out space—after mining the coal and rolling it into the conveyor drift, the shearer is reversed. Then, the rock bench is mined in the direction from the conveyor drift to the ventilation drift. The rock from the face is transported to the conveyor drift, then goes around the mine working and in the opposite direction is delivered to the backfill conveyor line of the horizontal-loop scraper conveyor.
- The process of backfilling with rocks—the backfill conveyor line is set at an angle to the seam bottom plane, and as the undercut rock moves along the inclined pans, it is self-unloaded into the mined-out space.

The rock pack placed in the mined-out space behind the longwall face contributes to the smooth lowering of the rock layers in the immediate and main roof areas, preventing their discontinuity. This leads to a redistribution of stresses in the rock mass surrounding the longwall face and makes it possible to significantly reduce the load on the powered support sections. However, the degree of this influence remains unknown.

Further on, studies are conducted on the effect of leaving the coal seam bottom rocks in the mined-out space on the geomechanical state of the roof rocks and the load on the powered support.

3. Theoretical Background

When studying the geomechanical state of the rock mass surrounding the longwall face, the initial prerequisites are used, based on the existing hypotheses of the mechanism of destruction and caving the rock mass, and the formation of rock pressure in longwall faces. Thus, with traditional stope technology, which provides for the complete rock caving behind the mechanized complex sections, according to the hypothesis of “hinge blocks” [52], the undermined rock mass is divided into separate blocks that interact with each other. The rock discontinuity above the stope working is characterized by the formation of an uncontrolled collapse zone of the immediate roof rocks and orderly displacement of the immediate roof rocks. In this case, the orderly displacement of the immediate roof rocks is created as a result of the fact that the rock layer blocks, separated from each other by fractures, do not lose adhesion to each other and together form a hinge system capable of withstanding loads that are transmitted from the side of the overlaying layers. According to existing ideas, for the Western Donbass conditions, the rock caving process is characterized by the formation of characteristic zones in the mined-out space: uncontrolled collapse zones from the side of the mined-out space, hinge-block displacement zones, and zones of smooth deflection of the layers without discontinuity, the parameters of which are studied in detail in these works [53,54]. In this case, the main factor influencing the formation of loads on powered support is the deformation modulus E of the rock layer [55]. When mining a coal seam with leaving the rocks in the mined-out space, the mechanism of the roof rock displacement above the mined-out space is divided into two zones, excluding the uncontrolled collapse zone [56]. The degree of filling of the mined-out space is of great importance when conducting backfilling operations, since this parameter determines the volume of rock that can be backfilled into the mined-out space, as well as the effectiveness of supporting the roof in the longwall face [57]. Along with this decisive factor influencing the deformation processes behind the longwall face and the stress–strain state formation in the rock mass surrounding the stope working, it is necessary to take into account the compression characteristics of crushed rocks, which are studied in the works [58–60].

Ideas about the hypothesis and mechanisms of deformations and displacements of the rocks surrounding the stope, as well as the interaction of roof rocks with the support, can serve as the basis for constructing a number of theoretical calculation schemes. The latter will take into account both the elements of the process kinematics and the force

characteristics of the deformation and destruction of rocks and support. Principal schemes of the rock displacements in the coal-overlying formation using traditional technology and selective seam mining with leaving the rocks in the mined-out space are shown in Figure 2a,b.

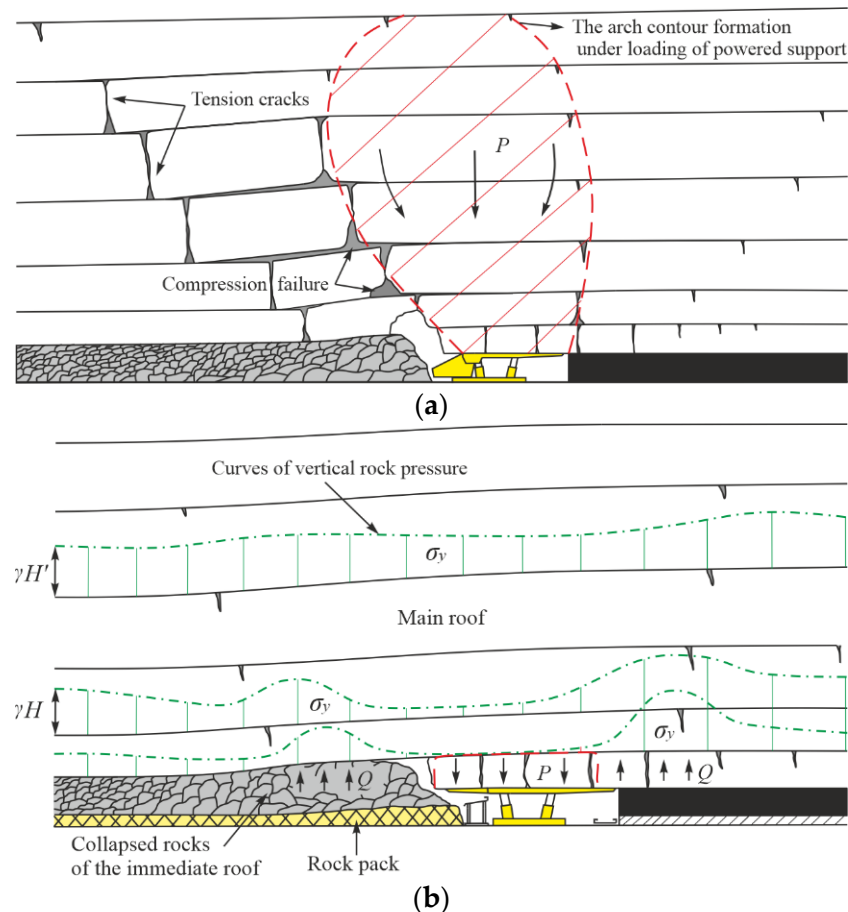


Figure 2. Rock displacements in the coal-overlying formation: (a) Known scheme of traditional technology with complete caving of the roof rocks; (b) Expected rock mass behavior at selective seam mining with leaving the undercut rocks in the mined-out space.

In the case of traditional technology with complete caving of the roof rocks (Figure 2a), the rock cantilevers of the main roof layers, hanging over the longwall face, form a volume of unstable rocks bounded by two surfaces: from the side of the longwall face, the area of pinching rock cantilevers, their destruction and the lowering of rock blocks onto a powered support; from the side of the mined-out space—areas of sign-changing curvature of the rock layers bending with destruction mainly due to horizontal tensile stresses. The resulting unstable mass volume creates a load (P) that exceeds the rock pillar weight directly along the length of the powered support section, since the caving rock cantilevers go beyond the dimensions of the longwall face working space.

During selective seam mining (Figure 2b), the rock pack in the mined-out space restricts the lowering of the main roof rock layers to the height of its thickness m_{rp} . Therefore, the development of the roof stratification slows down with displacement stabilization at a reduced distance behind the longwall face; the volume of unstable rocks and the load from their weight are reduced. When a rock pack with a thickness of at least 0.5 of the extracting seam thickness m_{ex} is formed in the mined-out space, the hanging rock cantilevers of the main roof cease caving, since the possibilities of smooth lowering of the layers (in the Western Donbass) without caving compensate for part of the extracting seam thickness m_{ex} , and the rock blocks receive a bearing surface from the side of the mined-out space with

the reaction Q ; the same bearing is located in the bottom-hole zone of the coal seam; with sufficient thickness of the first rock layer in the main roof, it does not collapse, but retains the integrity of the rock beam on two bearings.

With a small yielding property of the powered support section (up to 50 mm), it deviates from the load from the side of the main roof rock layers and takes up only a relatively small load (P) from the side of the immediate roof, which is 7–16% of the powered support load-bearing capacity.

The described mechanism of displacement in the coal-overlying formation, when leaving the undercut rocks in the mined-out space, has been confirmed by performing a set of computational experiments with different indicators Δm_{rp} —the relative rock pack thickness (the ratio of the rock pack thickness m_{rp} to the extracting seam thickness m_{ex}).

4. Research Methods

4.1. Numerical Modeling

To study the stress–strain state of a rock mass when mining mineral deposits, numerical modeling by the finite element method has become widespread, which allows real geomechanical processes to be reflected with a high degree of reliability [61–65]. The geomechanical assessment of the stress state of the rock mass surrounding the 861 longwall face, while leaving the waste rocks from undercutting the seam bottom rocks in the mined-out space, has been performed by the finite element method using the Ansys software package in the elastic–plastic formulation. The Ansys software package is a well-known, professional program available for use in the environment of geotechnical engineering calculations, which is based on the finite element method. This software product is quite often used for the numerical analysis of stress distribution in a rock mass during underground coal mining [66–69].

A calculation scheme for the geomechanical modeling of the rock mass surrounding the 861 longwall face, which consists of 14 rock layers, is presented in Figure 3. The model height (along the y coordinate) is 55 m, the dip/rise width is 60 m (x coordinate), and the seam dip angle is 3 degrees. The main physical–mechanical characteristics of the coal-bearing stratum taken for the modeling have been obtained from the mining–geological prediction data of the extraction pillar of the 861 longwall face and are given in the Table 1. For the Western Donbass mines, the physical–mechanical rock properties are determined in laboratory conditions at the M.S. Polyakov Institute of Geotechnical Mechanics (Dnipro, Ukraine). A load of 8.0 MPa is applied to the geomechanical model surface, which corresponds to the rock pressure value (γH) at a depth of 440 m, taking into account the average density of overlying rocks (2.0 t/m^3).

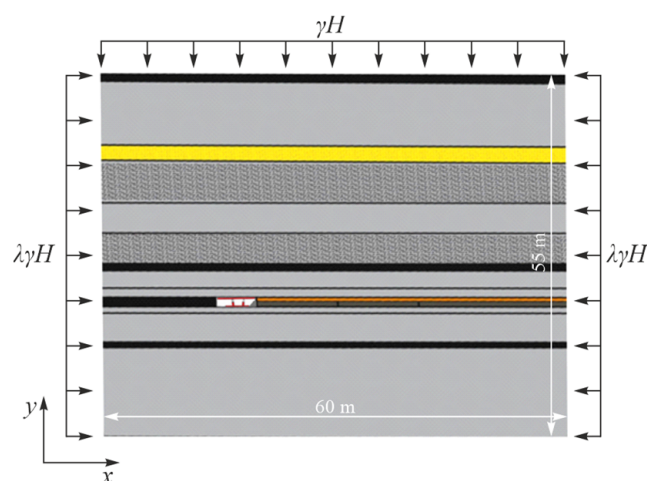


Figure 3. Calculation scheme for geomechanical assessment of the state of coal-bearing mass around the longwall face.

Table 1. Physical-mechanical characteristics of the coal-bearing rock stratum.

No	Rock Type	Parameter				
		Compressive Strength (σ_{comp}), MPa	Tensile Strength (σ_{tens}), MPa	Rock Density (γ), kg/m ³	Elasticity Modulus ($E \cdot 10^4$), MPa	Poisson's Ratio, (μ)
1	Coal	31.0	1.3	1.24	0.3	0.3
2	Argillite	12.0	0.7	2.5	0.8	0.4
3	Sandstone	40.0	2.7	2.6	2.0	0.35
4	Siltstone	18.0	2.2	2.33	0.9	0.38

Based on the analysis of the deformation properties of lump-type backfill materials from mine rocks [70,71], the following mechanical characteristics of the rock pack formed in the mined-out space, corresponding to the period of compressive loading, have been taken. The rock pack deformation properties after shrinkage are taken as follows: deformation modulus $E_{rp} = 50$ MPa, Poisson's ratio $\mu_{rp} = 0.4$. Depending on the calculation stage, the relative height of the rock pack formed in the mined-out space changes. As a result, five calculation stages are performed with a relative rock pack height from 10% to 50%.

To facilitate the calculation of the geomechanical system stress–strain state, powered support is taken with a uniform support reaction to the immediate roof and bottom rocks, which is achieved by replacing the real support section structure with a parallelepiped of a deformation modulus E_{sup} , which is calculated by the following expression:

$$E_{sup} = \frac{P_{max}}{\varepsilon_{sup}}, \quad (1)$$

where P_{max} is load-bearing capacity of the support section ($P_{max} = 350$ – 500 kPa for the basic technology of the seam mining; $P_{max} = 260$ – 370 kPa for selective technology with leaving the rocks in the mined-out space); $\varepsilon_{sup} = \frac{\Delta U}{m_{ex}}$ is the relative yielding property of the support hydraulic prop; m_{ex} is the extracting seam thickness, taken as 1.05 for the basic technology and 1.2 m for selective technology with leaving the rocks in the mined-out space.

The deformation modulus is $E_{sup} = 55$ MPa for a 1KD-90 support in the standard version (with basic technology) and $E_{sup} = 35$ MPa for a modernized 1KD-90 support with a reverse cantilever (with selective technology), respectively.

4.2. Determining Force and Deformation Parameters for the Powered Support Loading

The methodology for conducting research is as follows. On the same geomechanical model (already substantiated earlier), within the height of the extracting seam thickness, four planes are drawn in the mined-out space, separating the rock mass from the rocks of the uncontrolled collapse zone. Giving each of the obtained layers the mechanical properties of collapsed rocks or backfill mass makes it possible to obtain five values of Δm_{rp} within one model:

- $\Delta m_{rp} = 0\%$ —there is no rock pack—all the simulated layers are assigned the properties of collapsed rocks.
- $\Delta m_{rp} = 20\%$ —the layer closest to the bottom simulates a rock pack, and the rest simulate collapsed rocks.
- $\Delta m_{rp} = 30\%$ —the two lower layers simulate a rock pack, the rest (throughout the height of the mined-out space) simulate collapsed rocks.
- $\Delta m_{rp} = 40\%$ —the three lower layers are the rock pack, the two upper layers are collapsed rocks.
- $\Delta m_{rp} = 50\%$ —the four layers, beginning with the seam bottom, simulate a rock pack and one upper layer simulates collapsed rocks.

Using this technique in the technology of conducting computational experiments, five values of the relative P_{rel} indicators (the ratio of the load on the support section to its load-bearing capacity) and U_{rel} (the ratio of the roof lowering to the value of section separation displacement) have been obtained. The load on the support section is determined from the

curve of vertical stresses σ_y at the contact with the immediate roof; the roof lowering at the same contact is calculated based on the indicators of vertical immediate roof displacements U_y . As a result, according to the five values of the indicators, dependences $P_{rel}(\Delta)$ and $U_{rel}(\Delta)$ have been constructed.

In order to expand the area of reliable use of the specified dependences, two variants of mining-geological conditions, which are called “difficult” and “favorable”, are considered. Based on the analysis and systematization of the mechanical properties of the Western Donbass coal-bearing stratum, three groups of conditions have been identified according to the criterion of the estimated compressive strength of rocks with the participation of acting weakening factors. Two extreme groups of conditions have been chosen and two variants of the coal-bearing mass state in “difficult” and “favorable” mining-geological conditions have been modeled.

Unfavorable conditions: weak, water-saturated rocks at the lower limit of compressive strength: coal is 20 MPa, claystone is 5 MPa, siltstone is 10 MPa, sandstone is 20 MPa. Favorable conditions: water-free rocks at the upper limit of compressive strength: coal is 40 MPa, argillite is 20 MPa, siltstone is 25 MPa, sandstone is 60 MPa. Both variants in the form of functions $P_{rel}(\Delta)$ and $U_{rel}(\Delta)$ are displayed graphically and are criteria for determining the formation of the expedient thickness of the rock pack Δm_{rp} in the mined-out space.

5. Results and Discussion

5.1. Results of Numerical Modeling around the Longwall Face

Given the research peculiarities, special attention should be paid to the influence of the thickness m_{rp} of the rock pack erected behind the powered support sections in the mined-out space on the stress–strain state parameters of the rock mass surrounding the longwall face. For greater visibility, vertical sections are drawn along the model in the most important zones: in the zone of acting frontal bearing pressure ahead of the longwall face (section I), directly within the area of the longwall face (section II), and in the destressing zone behind the longwall face (sections III and IV) with the corresponding distances between them (Figure 4).

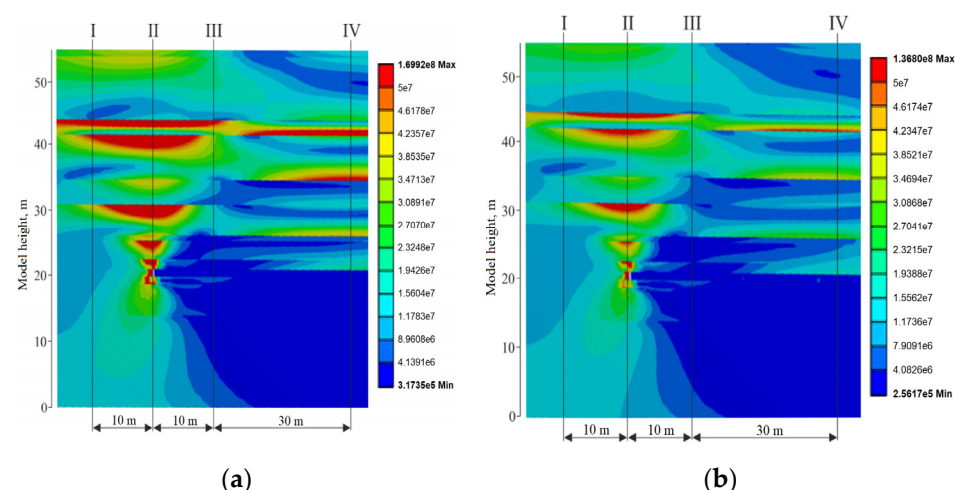


Figure 4. Curves of stress intensity σ around the longwall face: (a) Using traditional technology with complete caving of the roof rocks; (b) Using selective technology with leaving the rocks in the mined-out space ($\Delta m_{rp} = 0.5$); I–IV are location of sections along the model.

This approach makes it possible to obtain changes in stress intensity in the form of graphs, which are presented in Figure 5. Here, as well as for other stress components, already at the stage of visual assessment, it is possible to state the tendencies of reduction in the mass stress using the selective technology of coal seam mining with leaving the rocks in the mined-out space.

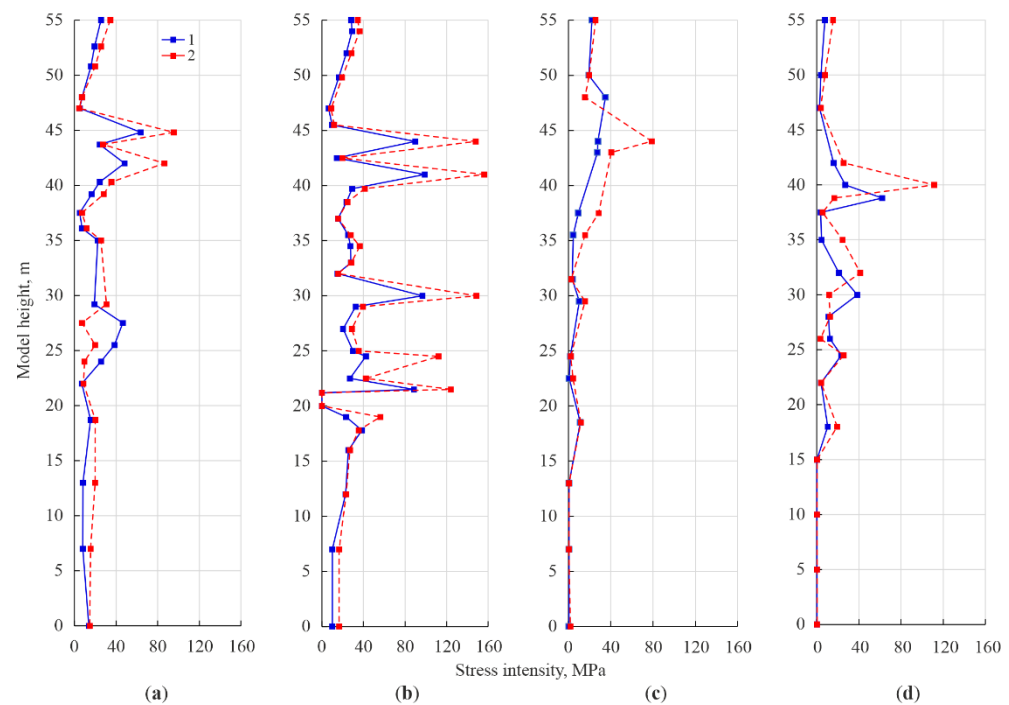


Figure 5. Changes in the stress intensity in the coal-bearing mass sections at different distances from the face: (a) Section I (10 m ahead of the longwall face); (b) Section II (directly within the zone of the longwall face); (c) Section III (10 m behind the longwall face); (d) Section IV (30 m behind the longwall face); 1—using selective technology with leaving the rocks in the mined-out space ($m_{rp} = 0.5$); 2—using traditional technology with complete caving of the roof rocks.

The stress intensity distribution at the relative rock pack thickness $\Delta m_{rp} = 50\%$ (Figure 5) was analyzed. Destructive concentrations (for all lithotypes, including sandstone) act in different areas of the coal-overlying formation: in the bearing pressure zone and above the longwall face, the concentrations σ are located in the upper part of each layer thickness; in the mined-out space—maxima.

The most stressed layer is sandstone due to its increased strength and rigidity; it is quite probable that it is the sandstone that represents the upper limit of the hinge-block displacement zone, above which there is a zone of smooth deflection of the layers without discontinuity.

In this case, during selective seam mining, the penetration of destruction in sandstone is 10–25% of its thickness, and the probability of a more stable thrust-block system formation increases. The same tendencies can be noted in the lower thick layer of siltstone, where the areas of acting maxima are reduced (with selective technology) by 1.8–2.15 times above the longwall face (Figure 5b), and above the mined-out space they completely disappear with the corresponding sharp drop in the concentration value (Figure 5c).

In the closer main roof layers, there is a significant reduction in the areas of distributing concentrations above the longwall face and their disappearance above the mined-out space. The total reduction in areas of acting destructive concentrations ranges within 40–80%. With the basic technology, more intense rock pressure acts and sandstone is prone to destruction to a height of at least half of its thickness: with such destruction, the creation of a sufficiently stable thrust system from sandstone rock blocks is unlikely.

Consider the change in the maximum stress intensity concentrations at different relative thickness of the rock pack Δm_{rp} (from 10% to 50%) and in its absence in the mined-out space in the most characteristic zones of rock pressure manifestation—the frontal bearing pressure zone (Figure 6) and the destressing zone (Figure 7) of the most stressed rock layers.

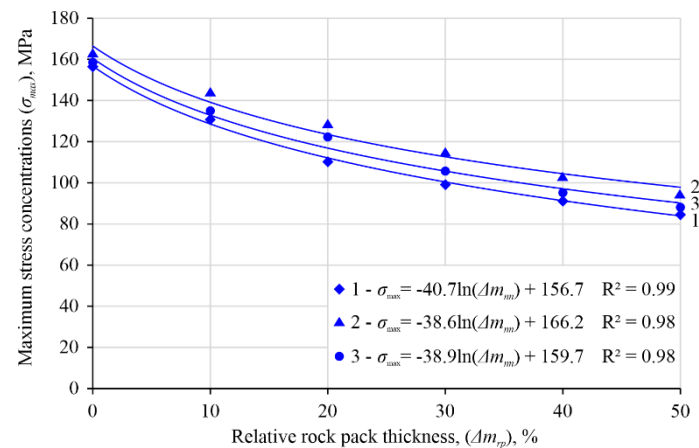


Figure 6. Dependence of a change in the maximum stress concentrations of rock layers in the frontal bearing pressure zone on the relative rock pack thickness: 1—sandstone ($y = 45$ m); 2—siltstone ($y = 42$ m); 3—argillite ($y = 29$ m).

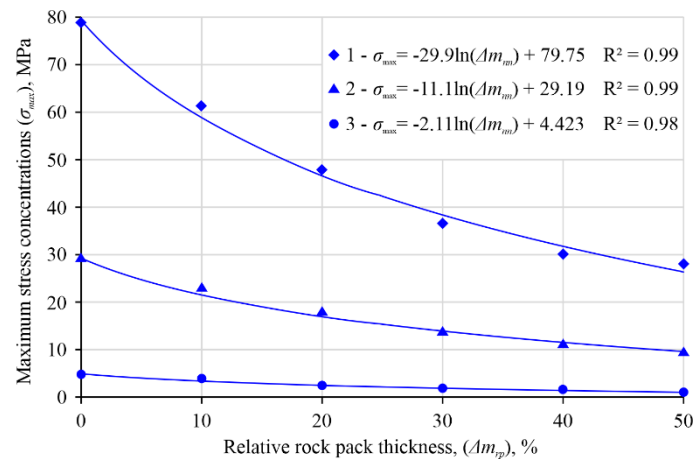


Figure 7. Dependence of a change in the maximum stress concentrations of rock layers in the destressing zone on the relative rock pack thickness: 1—sandstone ($y = 45$ m); 2—siltstone ($y = 42$ m); 3—argillite ($y = 29$ m).

It can be seen from the graphs in Figure 6 that in all studied variants, with an increase in the rock pack thickness in the mined-out space, the concentrations of maximum stresses decrease. For example, with an increase in the relative rock pack thickness Δm_{rp} from 10% to 50%, the absolute maximum stresses acting in sandstone decrease from 158 MPa to 78 MPa, that is, by 50.6% (Figure 6, curve 1). In the layers below the sandstone, the maximum acting stresses decrease by 42.6% and 43.9% for siltstone (Figure 6, curve 2) and argillite (Figure 6, curve 3), respectively.

Similarly, the influence of the rock pack thickness in the mined-out space on the change in the maximum stress concentrations in the destressing zone has been analyzed (Figure 7). An increase in the rock pack thickness has caused a significant decrease in the roof rock stresses in relation to the coal seam. Stresses at a distance of 10 m behind the longwall face reach 75–80 MPa in the absence of a rock pack, and 25–30 MPa with a relative rock pack thickness $\Delta m_{rp} = 50\%$, that is, due to the artificially created bearing, the maximum stress concentrations decrease by 62.5% (Figure 7, curve 1).

The distribution of maximum stress concentrations in other rock layers is as follows (Figure 7, curve 2 and 3). As a result of the increase in the rock pack thickness Δm_{rp} , the stresses in siltstone decrease from 80 MPa to 9 MPa and in argillite from 4.4 MPa to 0.6 MPa,

that is, there is a sharp decrease in the level of their stress–strain state due to leaving the rocks in the mined-out space.

As a result of performing a series of computational experiments, the dependence of a change in the rock mass stress–strain state on the influence of the rock pack thickness Δm_{rp} , which is formed in the mined-out space during selective seam mining, has been revealed. The use of a rock pack, erected behind the powered support sections, the material for which is undercut rocks from the longwall face, can significantly reduce the stress intensity and increase the rock mass stability both around the longwall face and around extraction workings.

5.2. Study of Force and Deformation Parameters for Powered Support Loading

According to the computational experiment results, a stable tendency of reducing the load on the powered support has been determined with an increase in the rock pack thickness, depending on the degree of complexity of the mining-geological conditions for mining the coal seams. The pattern of a change in the force and deformation parameters of powered support loading depending on the rock pack thickness is shown in Figure 8.

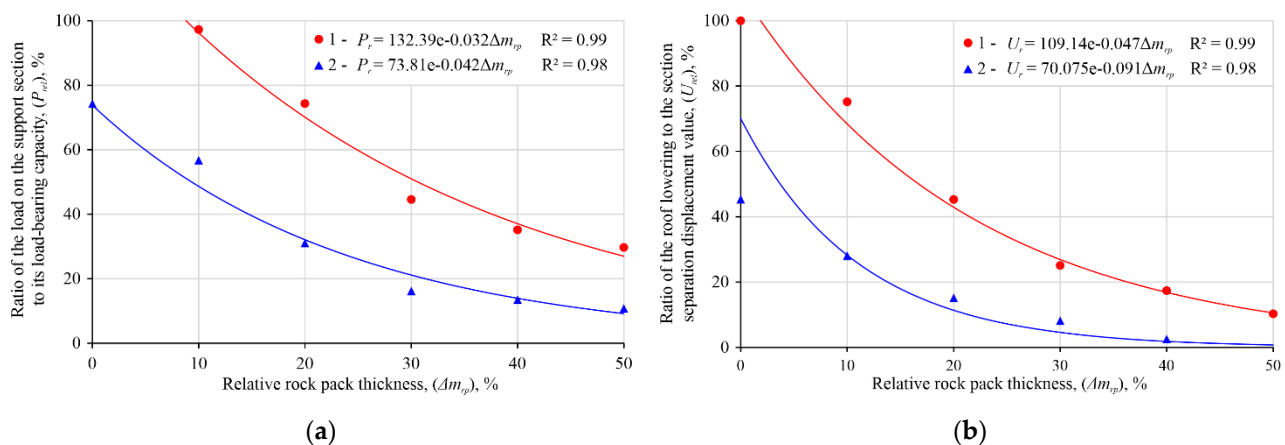


Figure 8. Dependence of a change in the maximum stress concentrations of rock layers in the destressing zone on the relative rock pack thickness: (a) Force characteristics of powered support; (b) Deformation characteristics of powered support; 1—unfavorable conditions; 2—favorable conditions.

When mining very thin seams, the loading of powered support is predicted to be up to 24% (in unfavorable conditions) and up to 10% (in favorable conditions) of the load-bearing capacity, with a relative rock pack thickness $\Delta m_{rp} = 50\%$ (Figure 8a). When mining seams with a thickness of 0.65–0.7 m with undercutting of rocks with a size of 0.55–0.6 m and their subsequent leaving in the mined-out space, the loading on the powered support section stabilizes at the level of 7–8% to 16–17.5% of its load-bearing capacity. In addition, a pattern has been revealed in the decrease in the value of the lowering (necessary yielding property) of the support roof with an increase in the relative rock pack thickness Δm_{rp} (Figure 8b).

In favorable mining-geological conditions, during support section operation, the displacement of hydraulic props and, accordingly, the operation of safety valves is not predicted. The absolute values of lowering are reduced by 37–45% in the range of changing $\Delta m_{rp} = 35$ –50%. In difficult mining-geological conditions, up to 50 mm (8.5% of the section separation displacement) is sufficient for maximum destressing of the support section, which is provided at $\Delta m_{rp} = 22$ –25%.

The given predictive data testify not only to the reliable and trouble-free operation of the support, but also to an increase in its MTBF (mean time between failures) due to the improvement of its operating modes.

6. Conclusions

The present research provides a geomechanical assessment of the effect of leaving the waste rocks from undercutting of the coal seam on a rock mass surrounding the longwall face using a new method of selective mining of thin coal seams. The obtained scientific results are as follows:

1. A geomechanical model has been substantiated and developed that adequately reflects the mining-geological conditions for seam mining within the extraction site, the real structure and properties of the rocks within the coal-overlying formation, the parameters of the longwall face and the modified powered support for the processes of leaving the rocks in the mined-out space. This makes it possible to predict completely objective results when conducting a series of computational experiments.
2. It has been determined that the stress intensity concentrations in the coal-bearing mass in the zone of frontal bearing pressure change according to a logarithmic dependence on the rock pack thickness value, which reduces their maxima by 1.7–2.0 times with an increase in the ratio of the rock pack thickness to the extracting seam thickness from 0.35 to 0.5. Given this dependence, it is possible to predict a decrease in rock pressure in the longwall face and extraction drifts in the case of selective mining of coal with leaving the undercut rocks in the mined-out space.
3. It has been revealed that the relative indicators of the load on the powered support section and the lowering of its roof are exponentially dependent on the ratio of thickness of the rock pack formed in the mined-out space to the extracting seam thickness. The found pattern substantiates the choice of the thickness of the rock pack formation in the mined-out space, which ensures trouble-free operation of the longwall complex.

Creating the possibility of leaving waste rocks in the mined-out space of coal mines opens a few positive aspects for achieving sustainable and “green” coal mining, which can directly serve as the subject of further scientific research. This is an assessment of the increase in the energy value of coal, a study of the reduction of costs for the transportation of waste rock in the underground and surface complex of the mine, an increase in the stability of excavation site workings, a study of subsidence of the earth’s surface, a study of the reduction of the movement of the flow of waste mine rock to the daylight surface.

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