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Multifactorial analysis of a gateroad stability at goaf interface during longwall coal mining – A case study

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Abstract

Purpose. Creating a generalized algorithm to account for factors (coal seam thickness, enclosed rock mechanical properties, the dimension and bearing capacity of artificial support patterns) causing a gateroad state under the effect of longwall face and goaf.

Methods. The assessment of the gateroad stability is based on numerical simulation of the rock stress-strain state (SSS). The finite element method is used to find out the changes in the SSS of surrounding rocks at various stages of longwall mining. The elastic-plastic constitutive model and Hoek-Brown failure criterion implemented in codes RS2 and RS3 (Rocscience) are applied to determine rock displacements dependently on the coal seam thickness, enclosed rock strength, width and strength of artificial support (a packwall comprised of hardening mixture "BI-lining"). To specify the mechanical properties of the packwall material a series of experimental tests were conducted. A computational experiment dealing with 81 combinations of affecting factors was carried out to estimate the roof slag and floor heaving in the gateroad behind the longwall face. A group method of data handling (GMDH) is employed to generalize the relationships between rock displacements and affecting factors.

Findings. The roof-to-floor closure in the gateroad has been determined at the intersection with the longwall face and goaf dependently on the coal seam thickness, enclosed rock strength, width of the packwall, and strength of hardening material. It is revealed that the support material gains the strength value of 30 MPa on the 3rd day from its beginning to use which is fully corresponding to the requirements of protective element bearing capacity. The possibility of using untreated mine water to liquefy the mixture is proved, that allows simplifying and optimizing the solute mixing and pumping technology.

Originality. This study contributes to improving the understanding of the factors that influence the stability of underground mining operations and highlights the importance of utilizing numerical simulations in optimizing mining designs. The impact of each factor on the resulting variable (decrease in cross-section of gate road by height) based on the combinatorial algorithm of structural identification of the model is estimated as follows: the packwall width is 48%, the thickness of coal seam is 25%, the strength of enclosing rocks is 23%, and the strength of the packwall material is 4%.

Practical implications. The findings provide stakeholders with a technique to determine reasonable parameters for support and protective systems, and the predictive model developed can be used to mitigate potential instability issues in longwall mining excavations. The results have implications under similar geological settings and can be valuable for mine design and optimization in other regions.

Keywords: *longwall, gateroad, stability, numerical simulation, GMDH, predictive model*

1. Introduction

Underground coal mining mostly adopts the longwall mining method. It is considered to be one of the safest, most productive, and efficient ways to extract coal [1]. The analysis of longwall mining use during the period of more than two last decades is detailed in [2]. Authors gave a comprehensive insight into the research frame-works, testing schemes, and modeling methods used to study the rock failure when longwall mining. This method is supposed to provide a high degree of automation resulting in smooth advancement of shearers and roof supports thus keeping people away from possible accidents [3]-[5]. The researchers, on the one hand, emphasized that the safety of work should be a priority, but on the other hand, they demonstrated the benefits of highly effective machinery from top global manufacturers that was implemented in the Czech and Ukrainian coal mines. However, the increase of excavation rate can alter the mechanical processes in the rocks due to an increase in the rate of rock loading while fast face advancing [6], [7]. Various studies on the impact of the face advance rate on the ground control and strata movement point that when extracting coal seams the longwall face collapses and the sudden dynamic loading on shields are frequently observed in the mining stope [8]. Li et al. underlined existing many mining risk issues, including non-uniform stress in the overburden, flooding and hazardous gases accumulated in the gob [9]. The performance of some coal mines located in India and

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longwall mining applying has been analyzed by Shi and Zhang [10] and the challenge faced due to longwall method application has been emphasized. It is pointed out that heavy pressure on the shield associated with unfavorable geological conditions could cause a collapse of a rather powerful longwall face hydraulic support.

Successful longwall retreat significantly depends on the stability of the roadways providing an access to the face. The development of excavation damaged zones (EDZs) around a roadway under mining-induced pressure is considered to be responsible for the unstable state of underground structures while the mining process [11]. In situ observing the initiation and propagation of cracks in the damaged zones provide for capturing the characteristics of the EDZ. The authors highlighted the complexity of the rock failure mechanism where such factors as shear and tension are involved, and the gate road stability is affected. It is stated that mining activities cause a significant damage to the surrounding rocks and reduce roadway support. Yang et al. presented a comprehensive case study of the surrounding rock failure characteristics and the control techniques for roadways at the Huojitu coal mine [12]. They pointed out that the stress concentration caused by the "excavation effect" is the fundamental cause of rock damage and the most effective method for achieving the "excavation compensation effect" is to reduce the maximum principal stress in the rock mass and increase minimum principal stress.

Such indicators as mining depth, enclosed rock mass quality and mining support potential can be referred to a great variety of geotechnical and design factors stipulating the failure processes [13], [14]. However, ceteris paribus being a close gate road location to the face has the greatest impact on its stability. Reputedly, when the gate is beyond the excavation impact it can be considered as another roadway and its convergence is low until the longwall retreat starts [15]. The longwall face approach makes the bearing pressure increase, thus resulting in an intense deformation of the excavation and, sometimes, reducing cross-section sizes to unacceptable values [16]-[19]. Maintaining permissible dimensions of the gate road cross-section at an intersection with the gob and behind the face still poses serious challenges despite the use of new support systems and protective patterns [20]. The principal causes of this issue are mining depth increase and overlapping impact of former exploitation [21] while conditions of weak and heavily jointed rock mass can dramatically worse the situation. The problem is common for a central part of Donetsk Coal Basin (Ukraine). The reason can lie in assigning unreasonable parameters of protective patterns not suitable for the geological conditions of the mined site resulting in poor excavation conditions of served longwalls [22]-[24].

In spite of well-developed instrumental observation methods that are considered to be a required part of the situ survey, predicting rock behavior is not always possible due to altering geological conditions, for example, near geological faults and small amplitude disjunctives [25]-[27]. The lack of "in-situ" information can be compensated by mathematical modeling of the rock mass stress-strain state [28], [29]. Numerical methods of rock mechanics have served as an effective tool for a gate road layout design at longwall mining [30], [31]. Most of the works cited above represent the study of geomechanical processes based on numerical simulation. Song et al. used the two-dimensional finite element method (FEM) implemented in the RFPA2D codes to simulate the fracture and failure process in brittle rock around the roadway served the longwall [8]. Małkowski and Ostrowski evaluated the roadway stability based on a two-dimensional FEM-analysis involving an elasto-plastic deformation model and the Hoek-Brown failure criterion [15]. The specificity of the methodology developed lies in identifying the criterion parameters based on the measured convergence in the maingate. Wu et al. adopted a two-dimensional Universal Distinct Element Code (UDEC) to optimize parameters for the roadside backfill body based on studying the mechanism of crack expansion at various distances from the working face [16]. The discontinuous medium is represented as an assemblage of discrete blocks while the discontinuities are treated as boundary conditions between blocks. A UDEC Trigon model [32] is adapted to simulate real fractured coal-rock mass. The researchers proved that a reasonable aspect ratio of the roadside backfill body can greatly increase the area of the yield-bearing zone and reduce the damage degree.

Yang et al. apply the finite difference method using codes FLAC^{3D} to determine the failure range and stress concentration degree in the rocks surrounding the roadway affected by two working faces [12]. The Mohr-Coulomb constitutive model is adopted as the failure criterion. Wu et al. also employ the FLAC^{3D} software to investigate numerically the stress-strain distribution and the failure of the rock mass surrounding the previous and current panel roadways during each stage of ‘‘twice excavation and mining’’ procedure [17]. The Lagrange algorithm and hybrid-discrete partition technique adopted in FLAC^{3D} is used to simulate the plastic failure of materials. Authors focused on the rock displacements as the most informative stability indicators, being aware that the floor heave significantly complicates the roadway functioning. Figuring out maximum values of the floor heaving in the roadway provides the correct technological regulations for the restoration of transport routes along the roadway. Authors showed that the gob-side pre-backfill maintained a good bearing capacity in all stages of mining and provided a strong support for the roadway stability.

The majority of simulations performed for various case studies in terms of the developed panel are dependent on a certain set of variables. The results obtained often do not have sufficient generalization and do not provide the basis to adopt urgent engineering solutions in response to complicated technical or geological situations.

This research is focused on design factors affecting the gate road stability in terms of Pokrovske PJSC conditions employing intensive coal mining in Central Donetsk Basin. A short-term goal of this study can be identified as finding reasonable parameters to adapt protective elements in terms of Pokrovske PJSC. Maintaining stable gate roads within the goaf during longwall retreat is crucial. The current panel headgate can be utilized for a next panel as a tailgate. It also can be employed for the site ventilation, and installation of gas exhaust equipment. Therefore, it is essential to ensure its stability. Creating a generalized algorithm to account all the factors causing the gate road state under the effect of longwall face and gob can be identified as a more capacious task to complete. To achieve this goal, the stress-strain state of the rock mass should be simulated by varying such combinations of geological factors as the mining depth, a rock mass structure and texture, a coal seam thickness, enclosed rock mechanical properties, the dimension and bearing capacity of artificial support patterns etc. A technique to adjust the design parameters should be developed after the rock stress-strain has been analyzed. Hence, a reliable method to develop a prognostic model of the excavation stability is supposed to be justified and adapted based on modern computational procedures.

2. Methods

2.1. Geological and geotechnical conditions

The coal seam d_4 mined by the company "Pokrovske" is characterized by thicknesses in the range from 0.8 to 2.7 m. The depth of the coal seam below the ground surface varies from 800 up to 900 m. The coal seam roof and floor are mudstones, siltstones, and sandstones. Mudstones and siltstones are jointed rocks with uniaxial compressive strength (UCS according to ASTM D7012-14e1) in the range from 20 to 65 MPa, the uniaxial compressive strength of sandstone varies from 57 to 90 MPa.

When exposed to water rocks lose 20-25% of their strength [33]. The coal formation is characterized by monoclinal bedding with the coal bed pitch (the dip angle of coal seam) of 6°. Numerous faults are located in the mined part of the deposit and the zones of heavily jointed rocks surrounding the faults. Coal mining is often accompanied by rockfalls while intersecting such structural features as bedding planes and joints. The height of the created opening varies from 1 m up to 4 m. Mining in fault zones can be accompanied by methane emissions and rock bursts.

Coal extraction is carried out by longwall system with roof caving (Fig. 1).

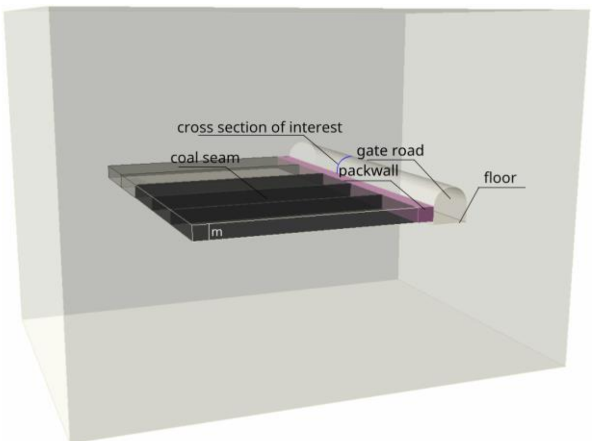


Figure 1. Graphical interpretation of the longwall mining process

Gate roads being excavated in soft and heavily jointed rocks are supported by steel yielding arches reinforced with rockbolts of the total length of 2.5 and 6 m long cable bolts installed to prevent abutment pressure influence. Such combined system provides sufficient stability of the gate road before the longwall approach but losses its efficiency after removing the face. Artificial structures erected in the mined-out area make it possible to provide support behind the longwall face without leaving coal pillars.

The most effective protective construction is a packwall (a cast strip) made of mineral bonders and cribs with high-bearing capacity to create a strong support pillar between the gate roadside and longwall gob [34], [35]. There is a wide range of dry mixtures intended for packwall molding [36]. General requirements for hardening mixtures are formulated as follows:

– strength and deformation characteristics of the material are supposed to provide the bearing capacity of the support structure;

– the components of the mixture must be non-toxic and non-combustible;

– the mixture should not be abrasive to provide adaptable operation of blowers;

– the packwall destruction under static load at any stage of hardening should occur smoothly, without any dynamic effects.

Dry mineral-cement mixture "BI-lining" being developed and produced in Ukraine (Patent of Ukraine No. 53569A) meets the above-mentioned requirements and is currently used in Pokrovske mine. Its composition consists of cement, fine sand, carbonate rock, and a considerable number of additives to accelerate the hardening process.

Even though the "BI-lining" manufacture provides the data dealing with the material strength properties, an additional analysis has been performed to clarify the hardening process.

2.2. Experimental study of packwall material

The dry mixture "BI-lining" is delivered to the gate road in bags using a monorail system and is fed into a mixing machine to be mixed with water. A special chamber is constructed of wooden racks and steel beams in the gate road at a distance of 6 m from the longwall face to insert the tight bags made of industrial fibers. The mixture is prepared by a hydromechanical method and transported through a pipeline using compressed air to fill the bags. The hardening process begins right away, but the issue is getting enough strength to provide the appropriate support in the mined-out area between the gate road and the gob. It should be noted that the delivery of all mixture components is rather a complex logistic task, therefore, this process requires rationalization on the scale of the entire mine. Thus, the issue of mine water used to liquefy the hardening mixture should be addressed. Some laboratory tests have been carried out to determine whether water quality affects hardening material strength.

Two types of water were used to liquefy the "BI-lining" dry mix: the mine water, collected directly in the gateroad of Pokrovske mine, and the water obtained from municipal water supply. The prepared solute was poured into molds for hardening (Fig. 2) and shaping cubic samples with dimensions 7.07×7.07×7.07 cm.

Figure 2. Molds with the embedded solute

After unpacking, the samples were placed in a container where normal conditions of humidity and temperature were created. Every batch comprises 12 samples made corresponding to 6 series of tests at different hardening times: on days 1, 3, 7, 14, 21, and 28. The samples were tested for uniaxial compression using a hydraulic press type KL 200/CE, (TECNOTEST). The test results are presented in Figure 3.

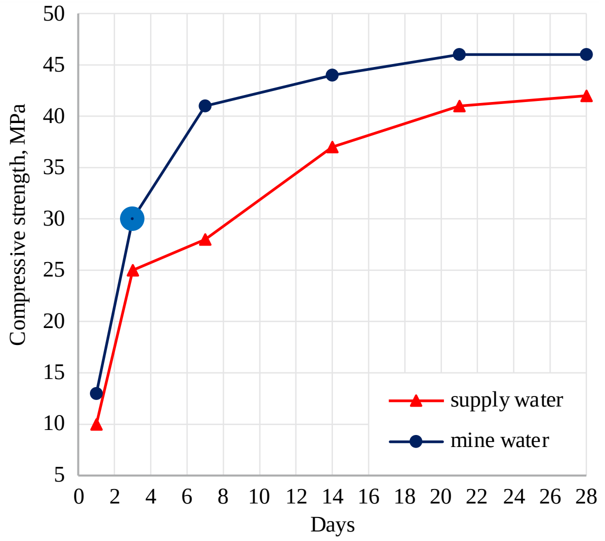


Figure 3. Influence of water quality on compressive strength of "BI-lining"

The test result analysis reveals that the mine water used to liquefy the dry mixture greatly increases the sufficient strength of the support material which is 15-20% higher than the values obtained after using the water supplied from the city network. These experimental results do not detail any physical basis of the phenomenon i.e., are the output of a "black box". However, it can be assumed that mine water, being a multicomponent system, contains a certain amount of calcium carbonate ($CaCO_3$). There is a chemical interaction between carbonates and hydrate cementing substances [37]. This provides good adhesion between the surfaces of the cement stone and the filler, and therefore, favorably affects the strength properties of hardened material.

According to Figure 3, the support material gains the strength value of 30 MPa on the 3rd day from its beginning to use which is fully corresponding to the requirements of protective element bearing capacity. Hence, the final strength of artificial material is determined and possibility of using untreated mine water to liquefy the mixture is proved, that allows simplifying and optimizing the solute mixing and pumping technology.

The obtained compressive strength of support patterns as well as the enclosed rock mechanical properties serve as input data for a multifactorial computational experiment based on a finite element analysis.

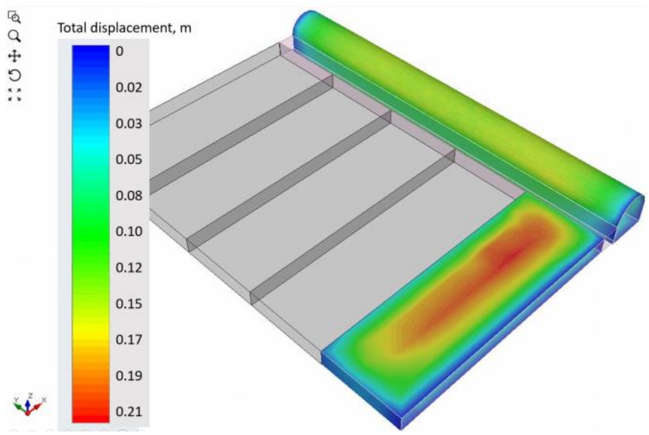
2.3. Numerical (FEM) Simulation

To meet the research goal both RS3 and RS2 FEM simulation software developed by Rocscience was applied. First, the 3D simulation (a large-scale design scheme) was employed to clarify the 3D effect of a longwall face on displacements around a gate road (Fig. 4).

The initial model of the 3D finite element analysis for the longwall system in underground mining was developed to simulate the behavior of the rock mass under geological conditions of Pokrovske mine. The model had dimensions of 50×50×50 m and was divided into tetrahedron elements.

The initial model included a set of parameters that defined the material properties of the rock mass, such as density, elastic modulus, and Poisson's ratio. In addition, the model accounted for the effects of rock mass strength, rock type, and the orientation of discontinuities within the rock mass using the generalized Hoek-Brown criterion.

(a)



(b)

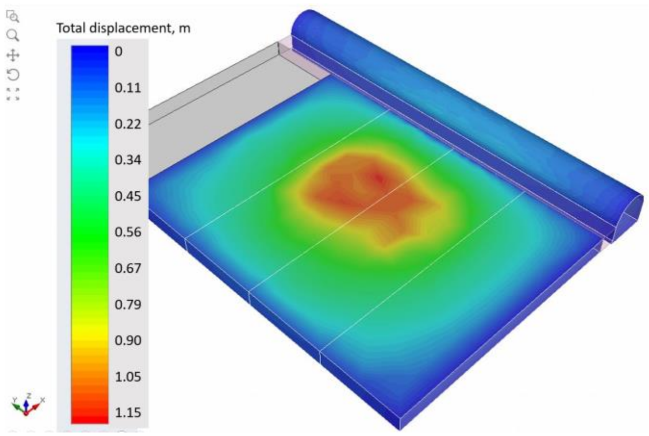


Figure 4. Rock displacements around a gate road: (a) before longwall face approaches the investigated section; (b) longwall face has passed the investigated section

This criterion is a widely used and accepted method for predicting the strength and deformation behavior of rock masses under various conditions.

It is confirmed that displacements around a gateroad increase dramatically as the longwall face approaches a section located initially outside of the face impact. The largest displacements occur behind the longwall depending on the gob size and the dimensions of the coal pillar left or the artificial structure (the packwall) erected instead of the pillar. Obviously, in terms of the designed support system (steel frames and rockbolts) packwall rigidity can be identified as a principle and controllable parameter to compensate rock displacements under given geological conditions. To substantiate this parameter mutual deformations of rock opening and an artificial structure should be considered. In addition, a gradual accumulation of rock strains during mining deve-lopment should be considered as well. Unfortunately, sufficient accuracy cannot be attained within the large-scale computational scheme while modeling this process, so RS2 within 2-dimensional scheme should be applied to analyze a rock stress-strain state around a gateroad in-depth.

The initial data for the simulation are as follows: the mining depth; the gateroad dimensions; the coal seam thickness; the structure of the enclosing rocks; the strength and deformation properties of the rock layers; data on the rock jointness; parameters of the steel lining and the data on the rockbolt location. The parameters of a packwall made of mineral binders to create a strong support pillar between the gateroad and mined-out space should be also taken in consideration.

The initial stress field is assumed to be hydrostatic $\sigma_x = \sigma_y = \sigma_z = \gamma H$, where H is the depth of mining and γ is the rock specific gravity (volumetric weight). At mining depth H = 800 m and average specific gravity γ = 25 kN/m^3 the initial stress field is $\sigma_x = \sigma_y = \sigma_z$ = 20 MPa. Properties of rock mass are represented in Table 1.

The state of a gate road section being perpendicular to longitudinal axis is simulated in terms of plane deformation hypothesis. The generalized Hoek-Brown failure criterion and elastic-plastic constitutive model are applied to determine the rock displacements and convergence in the gate road cross-section under the longwall effect.

The simulation process comprises 4 stages, where each one corresponds to a certain mutual arrangement of the gateroad and retreating longwall (Fig. 5).

Table 1. Rock mass and packwall properties

Property	Sandstone	Siltstone	Coal	"BI-lining"	Variable floor rock
Young's modulus, MPa	4000	3000	12000	5000	4000
Poisson' ratio	0.25	0.25	0.30	0.30	0.25
Compressive strength of intact rock, MPa	80	43	37.5	10-50	20-80
GSI	55	55	55	100	55
Hoek-Brown parameter, m_b	1.77	1.17	1.66	0.98	1.77
Hoek-Brown parameter, a	0.51	0.51	0.5	0.51	0.51
Hoek-Brown parameter, s	0.0016	0.0015	0.004	0.01	0.0016

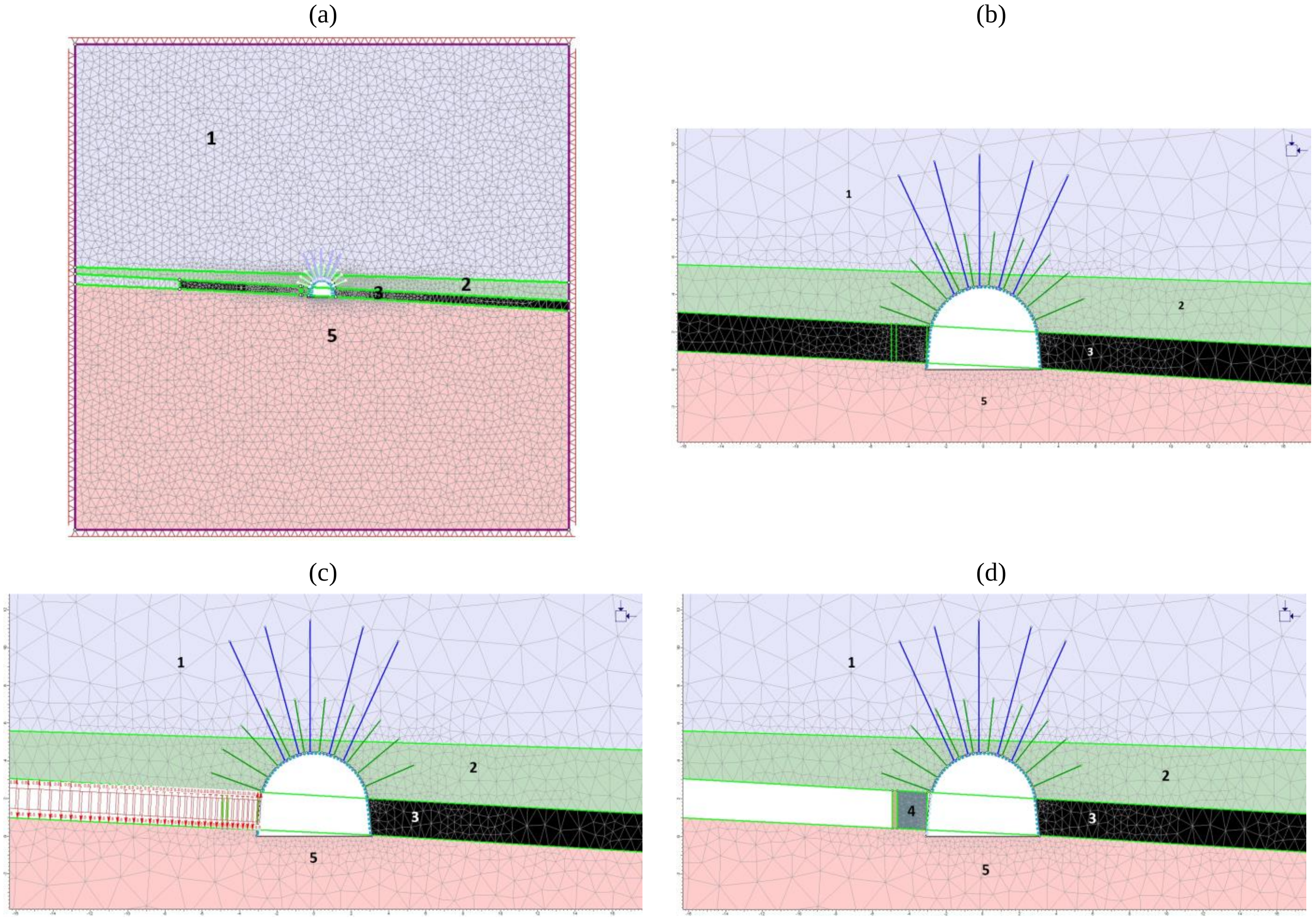


Figure 5. The stages of mutual arrangement of the longwall and gateroad at different mining stages: (a) Stage #1 (b) Stage #2; (c) Stage #3; (d) Stage #4; 1 – sandstone; 2 – siltstone; 3 – coal; 4 – packwall; 5 – immediate floor

Stage#1 is related to creating a single excavation (main gate) outside the longwall influence (Fig. 5a); stage #2 corresponds to installing the steel-polymer rockbolts and cable bolts (Fig. 5b); stage #3 means intersecting the gateroad and the longwall face with applying the distributed load to simulate hydraulic support (Fig. 5c); stage #4 is referred to erecting the packwall between the gateroad side and mined-out space behind the longwall face (Fig. 5d).

The simulation results are components of stresses, strains, and displacements, as well as the area of yielding zone around the gateroad.

A multivariate computational experiment was carried out involving four parameters (factors) varied in following ranges: the coal seam thickness from 1.4 to 2.0 m, the packwall width (w) from 60 to 80% of the coal seam thickness (m), the strength of immediate floor (R_c) from 20 to 80 MPa, and the strength of packwall from 10 to 50 MPa. Such values of each factor as minimum, maximum, and average ones were sequentially involved in the simulation. So, the matrix of full factorial analysis is formed by varying 4 factors at 3 levels with a total number of variants 3^4, which is 81 simulation models.

Maximum displacements of the gateroad roof and floor as well as the total loss of the cross-section along with the height and width are considered to be the output data of each simulation. The purpose of the computational experiment is to develop the target function (the roof to floor closure) depending on varied factors. The algorithm of multifactorial analysis will be discussed later. To verify the quality of designed mathematical model the particular cases are considered below.

3. Results and discussion

3.1. Case 1. Ensuring an admissible excavation state at unfavorable combination of natural factors

The worst geomechanical scenario in terms of Pokrovske mine corresponds to the thickest coal seam (m = 2.0 m) and the weakest floor rock (R_c = 20 MPa) ceteris paribus. It should be outlined that the greatest reasonable width (w) of the packwall with regard to the coal seam thickness under given conditions should be 80% of the coal seam thickness according to the original project. Based on this consideration, the pack wall width has been taken as 1.6 m in this modeling option (Fig. 6).

The simulation results demonstrate that if the packwall width is 1.6 m and the packwall material strength is 30 MPa, the roof sagging and floor heaving do not exceed 0.15 and 0.22 m respectively, the horizontal convergence is about 0.32 m at the stage #2 corresponding to the gateroad cross-section location outside the longwall effect (Fig. 6a). At the next stage #3 related to the gateroad intersection with the longwall, a certain increment of displacements occurs (Fig. 6b). The timely installation of support patterns provides the roof sag not exceeding 0.3 m (Fig. 6c), the floor heaving is not more than 0.45 m under conditions of weak rocks, and the horizontal convergence is 0.6 m.

This mining method requires the packwall to be installed before the forward longwall system movement. The strength increases within 3 days and reaches 25-35 MPa (according to obtained experimental data considered in the previous section). During this period a displacement increment can occur, but if a packwall has sufficient width (1.6 m), this increment will be insignificant (0.03-0.04 m in the roof and 0.05-0.06 m in the floor). Therefore, the installation of a Bi-lining packwall of 1.6 m wide (80% of coal seam thickness) under the conditions of weak rock results in the roof to floor closure not exceeding 0.75 m (17% of the gateroad height) and the horizontal convergence of 0.65 m (10.5% of the gateroad width). This state of the gateroad is acceptable both for the current operation as a maingate and for further reuse as a tailgate.

3.2. Case 2. Deterioration of the excavation state due to the inconsistency of support parameters with geological conditions

The combination of two affecting natural factors is still the most unfordable (excavated coal seam thickness is 2.0 m and compressive strength of the floor rocks (R_c) is 20 MPa) in this model. But the packwall width is taken as the minimum technologically affordable one (2.0 m times 0.6 m = 1.2 m). Besides, we assumed that decreasing stipulated time for BI-lining hardening resulted in not achieving the design strength. So, such parameter as 10 MPa of the BI-lining strength was taken as the input in this model option.

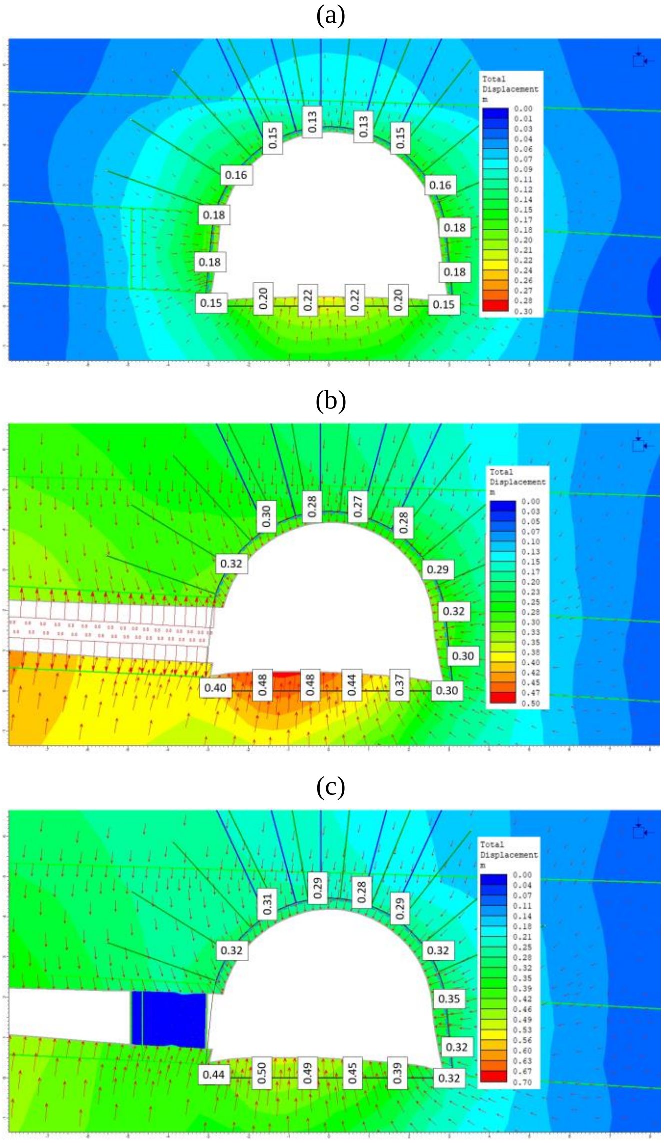


Figure 6. The displacement increase (Case 1: coal seam thickness is 2.0 m, floor rock strength is 20 MPa, packwall width is 1.6 m and packwall strength is 30 MPa); (a) stage #2 – the gateroad supported with rockbolts outside of longwall effect; (b) stage #3 – the gateroad and longwall intersection; (c) stage #4 – the packwall installation

The contour displacements at the initial three simulation stages are similar to Case #1. But as it was noted, the convergence at the 4th stage increases dramatically compared to the previous case (Fig. 7).

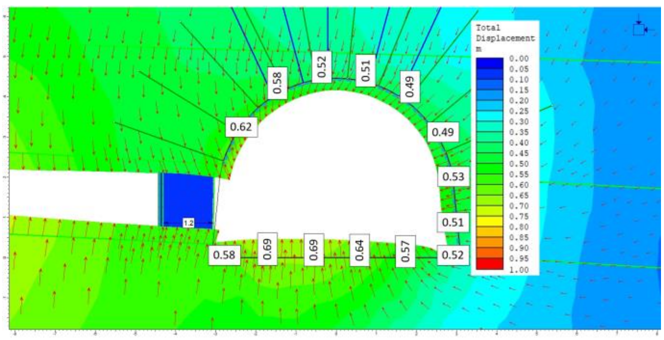


Figure 7. The increasing of contour displacement: stage #5 – packwall installation (coal seam thickness is 2.0 m, floor rock strength is 20 MPa, packwall width is 1.2 m)

Obviously, decreasing packwall width to 1.2 m and lining strength to 10 MPa results in increasing roof displacement up to 0.55 m and floor heaving up to 0.67 m correspondingly. The roof to floor closure is 1.22 m and horizontal convergence is 1.08 m (that is 27.3 and 18% respectively).

Reducing the excavation height by almost one third greatly decreases the possibility of its successful reuse. Moreover, additional costs should be projected for the support reinforcement and maintenance. Unpredictable overspending of funds and materials is supposed to be considered and modeling other possible concurrence of geological and technological factors as well as output data generalization is required.

According to the simulation results, there was a substantial degree of agreement with the field observations carried out at the Pokrovske mine with regards to the displacement of contour rocks. The obtained values for the roof to floor closure, as determined by the simulation, were found to be as 0.8 m when the packwall width is 0.8 × coal seam thickness = 1.6 m. This indicates that the simulation model used was able to accurately capture the relevant physical phenomena underlying the behavior of rocks in the mine, and thus provides a reliable tool for predicting and analyzing such behavior.

3.3. Generalization of simulation results

As it was mentioned above, this designed computational experiment was dealing with about 81 cases where a particular combination of affecting factors was simulated for each case. The tendency for roof and floor displacement depending on a packwall width is represented in Figure 8.

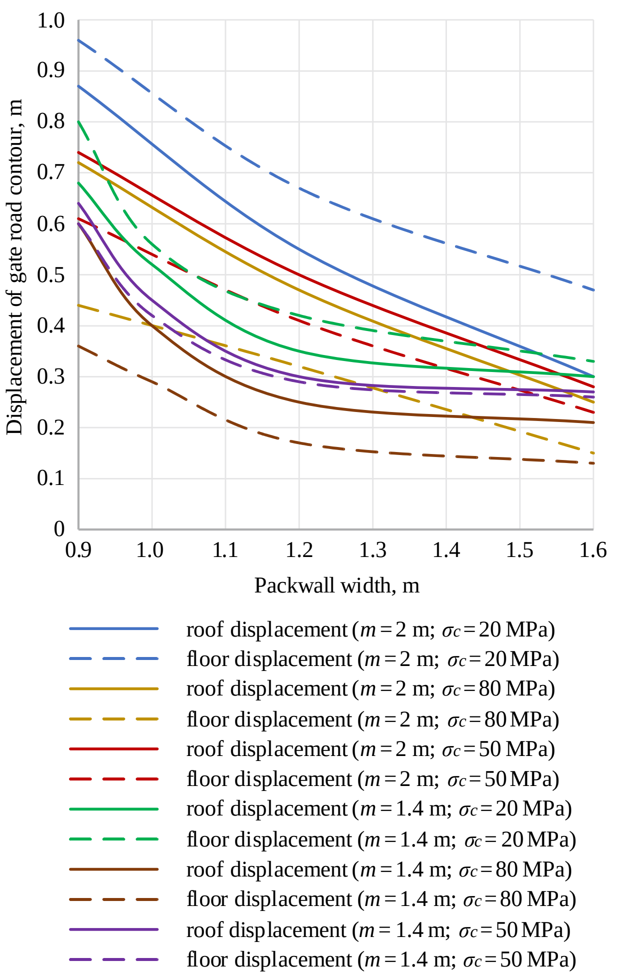


Figure 8. Roof and floor displacement change depending on packwall width

For practical purposes, establishing vertical convergence trends of the gate road contour in accordance with packwall width (Fig. 9a) and the strength (Fig. 9b) is highly recommended.

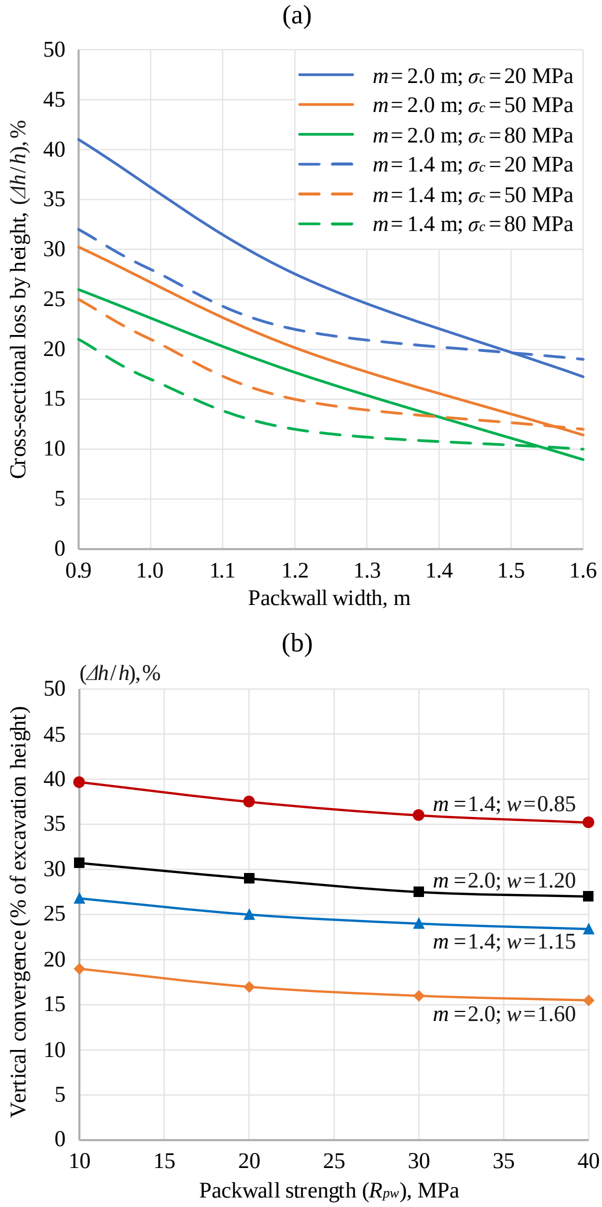


Figure 9. Roof to floor closure in percent depending on packwall: (a) width; (b) strength

The resulting relationships are the fitting sources to estimate opening stability and design support. However, a predictive model combining the effect of all factors and providing a direct calculation of the resulting characteristic, in particular, the loss of a cross-section area would be a more useful tool. To develop a prognosis mathematical model, the Group method of data handling (GMDH) is used [38], [39].

3.4. Group method of data handling

GMDH algorithm gives the possibility to define a model structure according to observed data and allows to estimate a target variable depending on a set of input factors in case of lacking information related to input variables. The principle of self-organization can be formulated as follows: when a model complexity gradually increases, certain "external" criterion passes through a minimum. Achieving a global minimum can indicate of an optimum complexity model existence [40].

First of all, a self-organization approach requires choosing the class of functions where the best model should be sought. Usually, the general relationship between output and input variables is built in the discrete form of Kolmogorov – Gabor polynomial:

$$f\left(x_j, a_q\right) = \sum_{q=1}^{s} a_q \cdot \prod_{j=1}^{m} x_j^{\alpha}, \quad (1)$$

where:

$f(x_j, a_q)$ – a target variable;

q – a member number, $q = 1, 2, \ldots, s$;

s – a total number of terms in a model;

a_q – a coefficient at the q-th member;

x_j – the j-th input variable, $j = 1, 2, \ldots, m$;

m – a number of input variable;

α – an exponent where j-th input variable enters into the q-th term.

To provide a good predictive property of the model (resistance to new data), its sensitivity must be checked by entering new data. So, the entire set of values should be split into two parts. The first data set (T) will be used to build a model and is identified as training data. The second data set (C) will be considered as new data and its purpose is to verify a created model quality. This set is identified as checking data.

Let W be the set of input data represented by matrix X ($N \times m$) and vector Y ($N \times 1$). We randomly divide W by two subsets in a proportion: 70% of data is a training dataset (T), and 30% is a checking data (C).

Two types of criteria (internal and external) are used by the given technique [41]. At each iteration h, the internal criterion is applied to the training data set to define k candidates for the best model according to subset T. We used the residual sum of squares (RSS) as an internal criterion:

$$RSS\left(h\right) = \sqrt{\sum_{i=1}^{N}\left[y_i - f_i\left(x_{ij}, a_j\left(h\right)\right)\right]^2}, \quad (2)$$

where:

h – a number of the iteration.

Then, the external criterion is applied to the candidate models. We use checking subset data with parameter estimates being obtained on the training subset:

$$MSEF\left(h\right) = \left\|Y_C - f\left(X_C, a_T\left(h\right)\right)\right\|^2, \quad (3)$$

where:

Y_C, X_C are Y, and X values in subset C;

$a_T(h)$ – the parameter estimate for subset T.

The external criterion involves values of separated parts of the sample and represents the mean square error of forecasting.

The steps comprised in the algorithm are as follows [42]:

1. Define a class of models of increasing complexity.

2. Split the data (Experimental data = Training Data + Checking Data) [43].

3. Estimate parameters of the model using the training data set and applying the internal criterion at a given iteration (2).

4. Test the model on the checking data set by applying the external criterion (3).

5. If the external criterion reaches a minimum, the best model is found, otherwise, we increase the complexity of the model and go to step 3.

Generation, comparison, and selection of probable models are based on the "step by step" scheme [44].

A step number defines the maximum possible quantity of members in the model. The first stage involves the analysis of all probable monomials, the second stage deals with binomials creation etc. At the subsequent stages of the algorithm, the structure and complexity of generated models depend on the best model structure being obtained at the previous stage (selection principle). The number of selection levels increases while the external criterion decreases "a stopping rule" [45], [46].

4. Discussion of results

The developed algorithm was applied to determine the gateroad contour convergence (Δh) depending on the set of geological and technical characteristics, such as coal seam thickness (m), the strength of enclosing rocks (R_c), packwall width (w), and packwall strength (R_{pw}). The input data for the analysis are represented in Table 2.

Table 2. Input data for GMDH analysis (data extract)

Strength of the enclosing rocks σ_c, MPa	Coal seam thickness m, m	Width of the packwall w, m	Strength of the packwall R_{pw}, MPa	Roof displacement, m	Floor displacement, m	Contour convergence, Δh, m
20	1.4	1.15	50	0.47	0.53	1.00
20	1.4	1.60	50	0.24	0.37	0.61
20	2.0	0.90	50	0.87	0.96	1.83
20	2.0	1.20	50	0.55	0.67	1.22
80	1.4	0.85	50	0.61	0.36	0.97
80	1.4	1.15	50	0.27	0.32	0.59
20	2.0	1.20	25	0.60	0.72	1.32
80	1.4	0.85	25	0.67	0.38	1.05
80	1.4	1.15	25	0.29	0.34	0.64
20	2.0	1.60	25	0.33	0.50	0.83
80	1.4	1.60	25	0.25	0.15	0.40
80	2.0	0.90	25	0.79	0.47	1.26
80	2.0	1.20	25	0.51	0.34	0.86
80	2.0	1.60	25	0.27	0.16	0.43
50	2.0	0.90	25	0.81	0.65	1.47
80	2.0	1.60	15	0.28	0.16	0.45
50	2.0	0.90	15	0.85	0.68	1.53
...	...	...	...	...	...	...

Hence, the set of functions is acquired, and the GMDH least error model has been developed. The predictive model of the gateroad cross-sectional decrease along with the height (Δh) depending on mentioned factors has the following form:

$$\Delta h = 0.88 \cdot \frac{m}{w} - 0.0047 \cdot \sigma_c \cdot \left(m - \frac{1}{\sqrt[3]{R_{pw}}} \right). \qquad (4)$$

The coefficient of determination for the model is 0.97 for the training sample, and 0.99 for the checking sample. The average coefficient of determination is R^2 = 0.98. The maximum negative and positive deviations for the sample do not exceed (-0.08) and 0.06 m, respectively.

To calculate the importance of variables we replace variables in the model with their mean value consequently (one by one) and measure the root mean squared error (RMSE) of a "new" model. An original model error is considered to be a zero percent impact on RMSE and 100% impact is a case where all variables are replaced with their mean:

$$IMP = \frac{R_{\text{var}} - R_o}{R_{all} - R_o} \cdot 100\%, \qquad (5)$$

where:

R_{var} – RMSE of the variable we consider;

R_0 – zero-impact RMSE;

R_{all} – RMSE of a model where all variables are replaced with mean.

The impact of each factor on the resulting variable (decrease in cross-section of gate road by height, Δh, m) based on the combinatorial algorithm of structural identification of the model is estimated as follows: the width of the cast strip (w) is 48%, the thickness of coal seam (m) is 25%, the strength of enclosing rocks (σ_c) is 23%, and the strength of the packwall (R_{pw}) is 4%.

The model analysis enables us to conclude that the width of the protective element (packwall) is the most significant parameter in the model structure. The effect of material strength on the resulting variable occurs not significant.

In practice, the maximum permitted convergence (Δh) is usually defined in terms of a given mining enterprise based on the requirements of air supply, equipment placement, or other factors should be taken into consideration. Stated differently, when creating the support design, a technological need that must be met is limiting value of convergence. Thus, the task is reduced to determine such a support (packwall) width that would guarantee the compliance with the technological requirements.

To meet the design goal, we resolve the Equation (4) for the width (w):

$$w = \frac{0.88 \cdot m}{\Delta h + 0.0047 \cdot \sigma_c \cdot \left(m - \frac{1}{\sqrt[3]{R_{pw}}} \right)}. \qquad (6)$$

An application of (6) for the new site in Pokrovske mine (input data are: coal seam thickness m = 1.8 m; strength of the enclosing rocks is σ_c = 50 MPa; roof to floor closure allowed taking into account the equipment overall dimensions and the minimum clearances is Δh = 0.9 m; and packwall strength is R_{pw} = 30 MPa) gives a design width of the packwall w = 1.27 m.

The technologically permitted contour convergence (Δh) should be taken from the experience of gate road exploitation in similar conditions. The specific value of Δh is based on expert assessment of opportunities to provide further mining operations and possibility to keep on gate road reuse for the next longwall systems.

The roof to floor closure fixed in the headgate #11 of the longwall #11 in the southern field of Pokrovske mine reached values in the range from 0.8 to 1.05 m behind the face. The gap between values calculated considering the rock strength in the floor of 20 MPa and those that were observed in situ ranges from 11 to 16%.

5. Conclusions

A series of experimental tests were conducted and the actual physical and mechanical properties of the packwall material "BI-lining" were defined. It was found that a dry construction mixture of BI-lining blended with mine water at the early stages up to 14 days has higher indicators of compressive strength (5-15 MPa) compared to other set of mixture qualities resulted from tap water adding. The experimental study allowed us to develop a plan of computational experiments and finite element simulation.

Applying the finite element method as the base for making analysis of the stress-strain state of the rock mass around the conjugation of longwall and gate road gave the possibility to create a general picture of changes at different stages of mining. A nonlinear increase in the displacements of the road contour is shown.

Multivariate modeling of the gate road condition stipulated by the width and strength of the packwall, and physical and mechanical properties of the enclosing rocks as well, demonstrates that reducing packwall width from the maximum design value ranging from 1.6 to 1.2 m, results in increasing the total cross-sectional loss in height in the range from 50 to 90%. Therefore, it can be concluded that protective element width plays a significant role as it was evidenced by the magnitude of the gate road contour displacement. The result of multivariate modeling was accumulating sufficient numerical data on the geomechanical system "gate road – protective element – longwall", which gave the possibility to apply the group method of data handling to establish the regularity between the studied factors.

To conclude, it should be noted that under the conditions of Pokrovske mine permissible cross-sectional dimensions can be provided with a strength of packwall material up to 20-30 MPa.

Based on this study, the following issues can be pointed out and considered: the lack of necessity in high material strength causes the delivery of a liquid mixture from the mine surface to the longwall face through a pipeline. Thus, we can avoid the procedure of preparing the mixture directly at the gateroad by placing additional equipment (a mixer and a pump) to clutter the gate road and provide a simultaneous water discharge in great amounts into the mined-out space to intensify the floor heaving.

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Багатофакторний аналіз стійкості штреку на перетинанні з виробленим простором при видобутку вугілля лавами

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Мета. Створення узагальненого алгоритму урахування факторів (потужності вугільного пласта, механічних властивостей порід, розмірів і несучої здатності штучних кріплень), що зумовлюють стан підготовчої виробки під дією лави та виробленого простору.

Методика. Оцінка стійкості базується на чисельному моделюванні напружено-деформованого стану (НДС) гірських порід. Метод скінчених елементів використовується для з'ясування змін НДС на різних стадіях розробки лавами. Пружно-пластична модель і критерій руйнування Хока-Брауна, реалізовані в кодах RS2 і RS3 (Rocscience), застосовуються для визначення переміщень гірських порід залежно від потужності вугільного пласта, міцності порід, ширини і міцності штучного кріплення (твердої полоси з суміші "БІ-кріплення"). Для уточнення механічних властивостей матеріалу, що твердіє, проведено серію лабораторних випробувань. Проведено обчислювальний експеримент з 81 комбінацією факторів, що впливають на стійкість. Метод групового врахування аргументів (МГУА) використовується для узагальнення взаємозв'язків між переміщеннями гірських порід та факторами, що впливають на стійкість виробки.

Результати. Змикання покрівлі та підошви штреку визначено у місці перетину з виробленим простором залежно від потужності вугільного пласта, міцності порід, ширини штучної полоси і міцності матеріалу. Доведена можливість використання шахтної води для виготовлення розчину, що твердіє. Це дозволяє спростити та оптимізувати технологію змішування та перекачування розчину.

Наукова новизна. Дослідження сприяє кращому розумінню факторів, що впливають на стійкість підземних виробок, і наголошує на важливості використання чисельного моделювання для оптимізації гірничих робіт. Вплив кожного фактору на результуючу змінну (зменшення поперечного перерізу штреку по висоті) на основі комбінаторного алгоритму структурної ідентифікації моделі оцінюється наступним чином: ширина штучної полоси 48%, потужність вугільного пласта 25%, міцність порід 23%, міцність матеріалу штучної полоси 4%.

Практична значимість. Отримані результати надають метод визначення раціональних параметрів для систем кріплення та охорони виробок, а розроблена прогностична модель може використовуватися для зниження ризику втрати стійкості виробки при перетинанні з виробленим простором. Результати мають значення в подібних геологічних умовах і можуть бути корисними для проектування та оптимізації гірничих робіт в інших регіонах.

Ключові слова: *лава, штрек, стійкість, чисельне моделювання, МГУА, прогностична модель*