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Substantiation and selection of parameters for supporting mine workings at deep levels

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Abstract

Purpose. The research aims to conduct a comprehensive study on substantiation and selection of optimal parameters for supporting mine workings at deep levels by analyzing the stress-strain state (SSS) of the rock mass, modeling geomechanical processes and developing effective strengthening technologies in difficult geological conditions.

Methods. The research includes modeling of the stress-strain state of the rock mass using ANSYS, Phase2 and Prorock software packages. To assess and predict the stress-strain state of the mass in the area of stope blocks, a series of numerical experiments have been conducted to analyze the near-contour rock mass stability. The physical-mechanical characteristics of ores and host rocks are used as input data for modeling.

Findings. The mass SSS analysis has confirmed that increasing the number of cable bolts up to four provides mine working stability, but is accompanied by intensive fracture formation beyond the contour, as well as convergence up to 14 cm, which requires the use of reinforcing mesh and reinforcing frame in two layers. The mining system elements have been found to provide the necessary safety factor for end ore drawing ($k_s \geq 3.0$) and slicing mining with hardening backfill ($k_s \geq 5.0$).

Originality. The parameters for supporting mine workings have been substantiated, which ensures safe mining of deep levels. It has been revealed that when backfilling, the height of the caving zone is 125 m and the fracture zone is 60 m. For mining systems with caving, these figures reach 280 and 470 m, respectively.

Practical implications. The technology has been developed for supporting capital mine workings with the use of reinforced combined support and roof bolts. The results obtained contribute to the improvement of reliability and efficiency of mining operations by providing accurate prediction of rock behavior under conditions of changing stresses and reduction of their strength characteristics.

Keywords: *stress state, mass, ore, rock, numerical analysis, supporting, mining system*

1. Introduction

The mining industry of Kazakhstan plays a key role in the country's economy, providing a significant share of its exports and being the most important source of raw materials for various industries [1], [2]. Geological diversity and rich natural resources make Kazakhstan one of the leading mineral producers in the world [3]. However, as the mining industry develops and moves to deeper levels, issues related to ensuring the safety, stability of mine workings and the efficient use of natural resources become increasingly relevant [4]-[6].

One of the most important tasks in mining is the selection and substantiation of optimal deposit mining systems taking into account difficult mining-geological conditions. This includes not only the safety of mining operations, but also ensuring the stability of mined-out spaces, which is particularly important when mining deep levels, where geostatic loads and the risk of accidents increase [7]-[11].

With the deterioration of the geomechanical environment and moving to deeper levels, the use of outdated mining systems focused on smaller sections and low loads becomes less effective [12]. Thus, to ensure safe and efficient deposit mining, especially in the context of difficult geomechanical situation, it is necessary to conduct research aimed at selecting and substantiating optimal parameters for supporting, as well as mining systems that take into account the specific conditions of certain objects [13]. To address these challenges, researchers have explored various approaches, including strategic goal setting for companies [13]-[16], success factors for entrepreneurship projects in mining and related industries [17], [18], and the realization of regional industrial projects [19]-[22].

Practice shows that for the same conditions, it is usually possible to use more than one mining system. Selection of mining systems suitable for mining-geological conditions is possible by any method, but the most widely used is the method of elimination, according to which systems that do

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not correspond to any mining-geological factor are excluded from all systems under consideration [23]-[26]. Long-term experience of underground mining operations at the Kazchrome Mine (Kazakhstan) has revealed the expediency of using mining systems based on induced or self-caving of ore and host rocks. At present, due to deterioration of mining-geological characteristics of ores and rocks, transition to deep levels and, as a consequence, complication of the geomechanical situation on site, the urgent task is to explore the possibility of switching to mining systems with backfilling of the mined-out space.

Analysis of research and development works, as well as project solutions for mining the field reserves, shows that the previously proposed mining systems are simple in design, which is quite understandable: small volume of commissioning works, small sectional sizes, etc. [27]-[30]. However, the use of mining systems with extended outcrop spans assumes the presence of stable ores and at least mean stability of the host rocks, which is not observed in the field. Along with the presence of people in the stope space, there is a high risk of conducting mining operations, uncontrolled failure of the walls and roof of the mined-out space (mine workings), in some cases, an uncontrolled process of self-localization of the chamber after the withdrawal of reserves [31]. Thus, the necessary task is to select the optimal design of the mining system, taking into account the mining-geological conditions, ensuring safety for personnel present in the zone of conducting mining operations, and substantiation of the parameters of constructive units.

The purpose of this research is to substantiate and select the optimal parameters for supporting mine workings at deep levels, taking into account mining-geological factors, which makes it possible to significantly improve the safety and efficiency of mining operations in conditions of difficult geomechanical and mining-geological characteristics.

2. Methods

This paper assumes the following mining systems:

– sub-level induced caving with end ore drawing;

– mining system using descending slicing with backfilling.

The main block parameters are panel width and sub-level height, so the block parameters are taken on their basis. There are many methodologies for determining ore drawing parameters for mining systems with caving [31]-[34], but there is no generally accepted and universal one. The most rational calculation is based on empirical dependences obtained by processing statistical data from a number of mines mining ores with different properties.

To assess and predict the stress-strain state of the mass in the area of stope blocks, models of the deposit areas have been developed. The models are based on geologic sections and horizon plans. Modeling is performed using the ANSYS software package, which is widely used for solving such research tasks [35]-[38].

The physical-mechanical properties of ores and host rocks composing the deposit are taken as input data. The stresses during modeling are calculated for the worst conditions, from the point of view of geomecha-nics: the stope extraction in the stope unit and after backfil-ling the chamber with hardening mixtures (backfill), if avai-lable. Figure 1 shows the initial models of the stope units when reserves are mined by the mining system.

(a)

(b) (c)

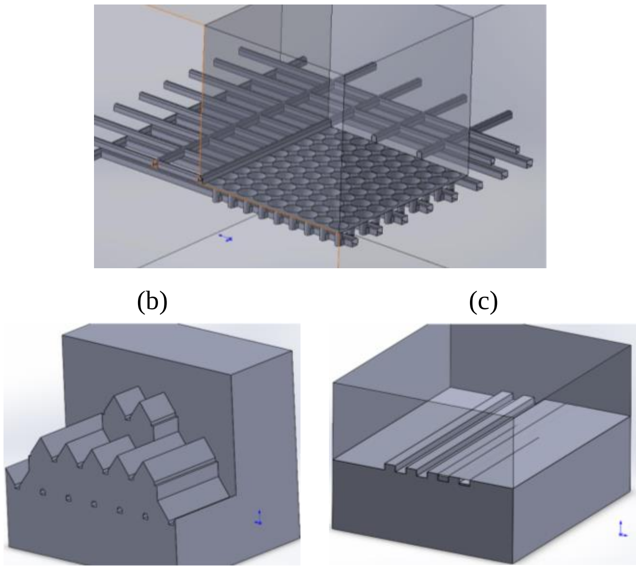


Figure 1. Initial model of the extraction unit: (a) caving with bottom ore drawing; (b) sub-level induced caving with end ore drawing; (c) descending slicing with hardening backfill

Since no geologic or lithologic model for the deposit has been provided, large-scale modeling is performed using available open-source data [39]. The ore body width in plan is assumed to be 220 m, the depth of mining operations – 600 m. The ore body thickness is assumed to be 80 m. It can be said that the geometry parameters of the mined-out space are taken such that in the general case (without taking into account local geometry peculiarities) the change in the natural stress field is maximum [40].

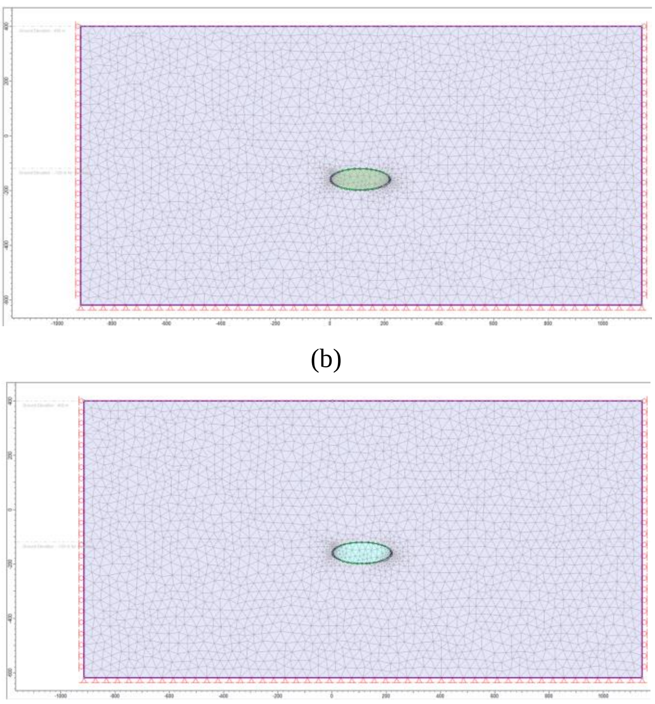
Large-scale modeling is performed in a planar setting on a schematic cross section. Modeling is performed under two scenarios of mining the reserves using different mining systems: with caving and backfilling of the mined-out space. Modeling is performed by finite element method in Rocscience Phase2 software.

Figures 2 and 3 show the models representing changes by stages. Figure 2 shows that the mining and subsequent backfilling of the mined-out space is specified by replacing the properties of the area corresponding to the ore deposit.

Figure 3 shows that the caved mass is not explicitly specified in the model, but is replaced by equivalent distributed loads. This is due to the fact that the mass failure processes in the finite element method are not explicitly reproduced, because of which, when modeling the deposit mining using caving systems, the changed stress-strain state cannot be realistically determined without any additional operations with the mass caving areas. The lateral pressure coefficient in the caved mass is assumed to be $\lambda = 1.0$, which is taken into account in the values of distributed loads applied to the vertical walls of the caved area. The fragmentation index for determining unit specific gravity of the caved mass is assumed to be equal to 1.2. For the calculations, it is assumed that the caving reaches the earth's surface at caving angles of 90°. For ease of data presentation, the modeling scheme described below will be referred to as Scheme 1.

The physical-mechanical properties of the backfill and caved mass specified in the calculation models are shown in Table 1. The strength properties (adhesion, tensile strength) are taken with a scale factor of 0.1.

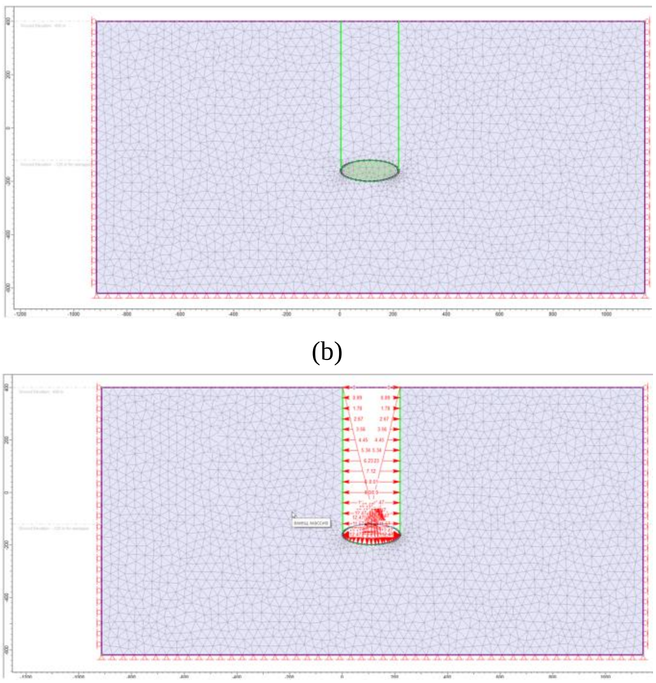
(a)



(b)

Figure 2. Graphical representation of the model with backfilling of the mined-out space (Scheme 1): (a) stage 1; (b) stage 2

(a)



(b)

Figure 3. Graphical representation of the model with caving of the host rocks (Scheme 1): (a) stage 1; (b) stage 2

Table 1. Physical-mechanical properties of the backfill and caved mass specified in large-scale models to assess artificial SSS

Mass	Backfill	Caved mass
Adhesion, MPa	2.00	0.02
Internal friction angle, °	22	35
Unit specific gravity, ton/m^3	500	–
Poison's ratio	0.25	–
Deformation modulus, MPa	500	–

This value is derived from the element dimensions (5-30 m), the deformation curve adopted in Phase2 and the parameters of fracturing.

The vertical component of the natural stress field is 23 and 28 MPa at 880 and 1040 m depths, respectively. The horizontal principal stress across the strike of the deposit is 20 and 21 MPa at the same depths. These values can be described by the dependences $\sigma_z = 1.03\,\gamma H$ and $\sigma_x = 0.83\,\gamma H$. These values are specified in all large-scale models. Note that the adopted dependence is slightly different from that proposed by Golder.

Due to the disadvantages of large-scale modeling, the decision is made to perform numerical modeling in two additional ways:

– FEM modeling with explicit specification of cave roofs and disintegrated mass within the boundaries of the roofs (hereinafter – Scheme 2);

– modeling of reserves mining in the finite-discrete element method (FDEM) (hereinafter – Scheme 3).

According to the first option, it is assumed that a cave roof is formed, the height of which in the case of flat-lying and horizontal bodies can be found as H_0 = 4 m, where m – is the thickness of the mined-out space. In the case of systems with backfilling, in the practice of rock shear calculations, it is assumed to use the effective thickness m_e instead of m, which depends on the value of underbackfilling and shrinkage. Thus, the above expression will be as follows $H_0 = 4m_e$. The effective thickness is assumed to be equal to m_e = 0.2 m.

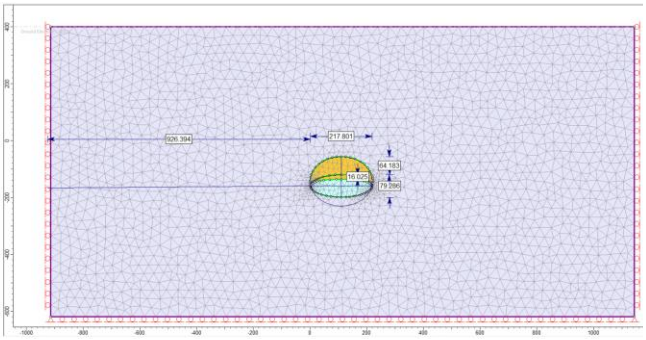
FDEM modeling does not require explicitly specifying the cave roof as in FEM. The cave roof will form naturally as one of the modeling results, which will depend on the size of the mined-out space, material properties and other parameters.

Figure 4 demonstrates the graphical representation of the models according to Scheme 2. Only stages 2 are shown, the first stages are identical to those adopted in Scheme 1 (Figs. 2, 3). Properties for calculations are assumed to be similar to those used previously (Table 1). Boundary conditions are assumed to be similar to stage 1 in the Scheme 1 of the model.

The modeling according to Scheme 3 by finite-discrete element method is performed in Prorock software. The constructed model is shown in Figure 5. The standard best practice for FDEM, used here as well, is to construct two zones in the model. The first one allows for caving, and it is built around the area of interest. The triangulation grid in it is not displayed in Figure 5. Around the first zone, a zone is constructed in which the mass is set to be elastic. The elastic part of the model ensures that the boundary conditions are set correctly. The grid in this zone is set high spacing for computational efficiency.

The differences in the models at the second stage are as follows. In mining with caving, the elements corresponding to the ore body are completely excluded from the calculation: thus, extraction is modeled. When mining with backfilling, only the upper part of the ore body, corresponding to underbackfilling, is excluded from the calculation, for the rest of the part physical-mechanical properties are replaced by eliminating the natural stress field. The properties of the ore are replaced by the properties of the backfill material. The transition from the first stage to the second stage is modeled gradually, through gradual changes in material properties.

(a)



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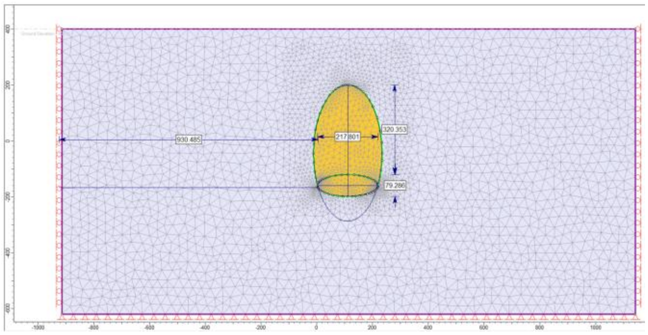


Figure 4. Graphical representation of stage 2 of models, constructed according to Scheme 2: (a) option of mining with backfilling of the mined-out space; (b) option with caving of the host rocks

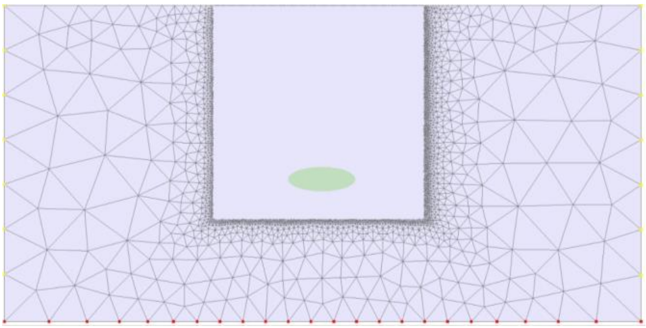


Figure 5. Graphical representation of the FDEM model (Scheme 3)

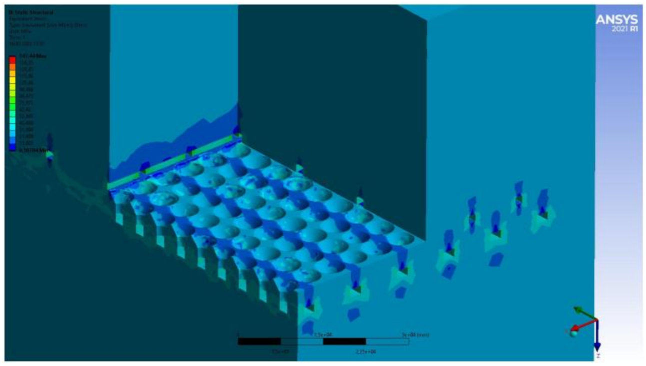
3. Results and discussion

Figure 6 shows the nature of stress distribution in the rock mass when the reserves are mined by the mining system based on caving with bottom ore drawing. Estimated safety factor values of constructive elements of extraction units when reserves are mined by the mining system based on caving with bottom ore drawing are presented in Table 2.

Table 2. Estimated safety factor values of constructive elements of extraction units when reserves are mined by the mining system based on caving with bottom ore drawing

Name of the constructive element of the extraction unit	Safety factor value, unit fraction
Pillar between drawing cones	1.17
Pillar between draw raises (delivery horizon level)	0.71
Scraper drift near-contour mass	0.91
Near-contour mass of undermining crossdrift	0.74

(a)



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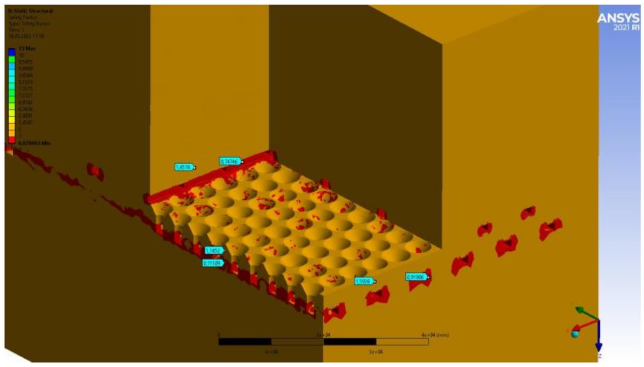


Figure 6. SSS of constructive elements of the mining system based on caving with bottom ore drawing: (a) equivalent stresses; (b) safety factor values

The SSS modeling data analysis of constructive elements of the mining system based on bottom ore drawing has shown that the safety factor values are less than 1. This fact, along with high fracturing of the ore and rock mass, high degree of the chamber bottom irregularity due to preparatory-cutting workings indicates the necessity of supporting all mine workings with heavy types of rock support. These types of supports (made of special concave profile, SCP) are currently used at the mine. However, the practice of their application shows the presence of extra-standard additional supporting of mine workings, increased material consumption compared to the planned one, which negatively influences the safety of mining operations, intensification and productivity of the mine.

In real conditions, due to the occurrence of rheological processes, the stress level may slightly differ, but it will have a high enough degree of convergence with the actual data due to the presence of triaxial compression zones in low-height pillars. The stress redistribution effect is observed in the form of cutter break formation, rock inrush of the mine working roof and walls [41], [42].

Thus, ensuring the block bottom stability can be achieved, in our opinion, by forming it from artificial mass (based on cementitious material) at the stage of preparing block reserves for stope extraction, which will negatively affect the feasibility study and is inexpedient at the present time.

Figure 7 shows the nature of stress distribution in the rock mass when reserves are mined by the mining system based on sub-level induced caving with end ore drawing. Calculated safety factor values of constructive elements of extraction units when reserves are mined using the mining system based on sub-level induced caving with end ore drawing are presented in Table 3.

(a)

(b)

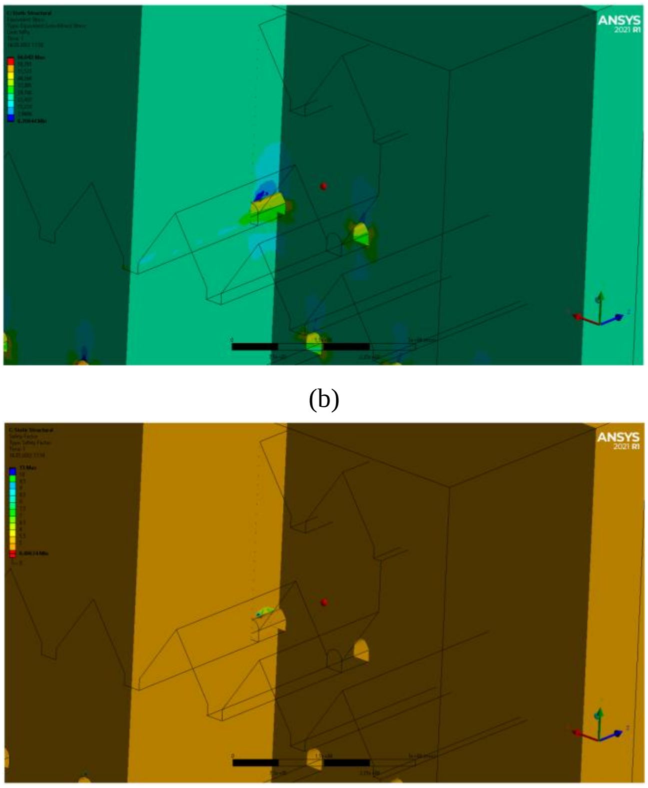


Figure 7. SSS of the constructive elements of the mining system based on sub-level induced caving with end ore drawing: (a) equivalent stresses; (b) safety factor values

(a)

(b)

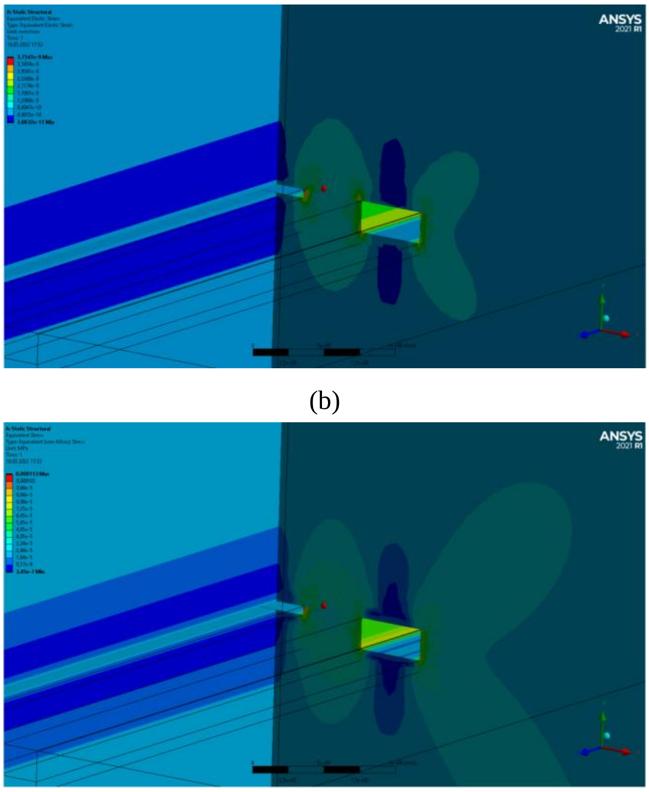


Figure 8. SSS of the constructive elements of the mining system using descending slicing with hardening backfill: (a) equivalent stresses; (b) safety factor values

Table 3. Estimated safety factor values of constructive elements of extraction units when reserves are mined by the mining system based on sub-level induced caving with end ore drawing

Extraction unit constructive element name	Safety factor value, unit fraction
Trench pillar	3.0-5.0
Near-contour rock mass of delivery drift in the bottom hole zone	3.0-5.0
Near-contour rock mass of delivery drift (5 m distance behind the face)	≥ 5.0

The modeling data SSS analysis of the constructive elements of the mining system based on sub-level induced caving with end ore drawing has shown that the safety factor values are not less than 3.0. Consequently, mining of reserves by this mining system ensures stability of constructive elements and safety of mining operations.

In real conditions, due to the occurrence of rheological processes, it may be necessary to specify the mining system parameters for the level using the results of experimental industrial tests. Thus, the use of the mining system based on sub-level induced caving with end ore drawing provides a reduction in labor costs, reduced volume of preparatory-cutting workings, and increases the reserve mining intensity.

Figure 8 shows the nature of stress distribution in the rock mass when the reserves are mined by the mining system using descending slicing with hardening backfill. Estimated safety factor values of constructive elements of extraction units when the reserves are mined by the mining system using descending slicing with hardening backfill are presented in Table 4.

Table 4. Estimated safety factor values of constructive elements of extraction units when reserves are mined by the mining system using descending slicing with hardening backfill

Extraction unit constructive element name	Safety factor value, unit fraction
Stope pass bottom	5.0-7.0
Beyond-contour mass (up to 0.5 m)	8.0-10.0
Beyond-contour mass (over 0.5 m)	≥ 10.0

The SSS modeling data analysis of the constructive elements of the mining system using descending slicing with hardening backfill has shown that the safety factor values are more than 5.0. Consequently, the mining of reserves using this mining system makes it possible to ensure the stability of constructive elements and the safety of mining operations to the greatest extent, provided that people are present in the stope space.

From the mathematical modeling data analysis, it has been revealed that the backfilling of the stope pass with hardening mixture changes the nature of stress distribution generated around the stope. In this case, the descending order of mining will make it possible to exclude the presence of contact with highly fractured rocks and ores in the roof and walls of the pass, thereby stabilizing the beyond-contour mass in terms of variation of its strength characteristics.

As a result of modeling, conducted according to the scheme described in the Phase2 software methodology, the principal stress distributions can be obtained (Fig. 9a-d). The figures show only the area around the mass section to be mined. Figure 9a, b demonstrate the stresses caused by mining the reserves with backfilling of the mined-out space, while Figure 9c, d show the stress state associated with mining with caving of the host rocks.

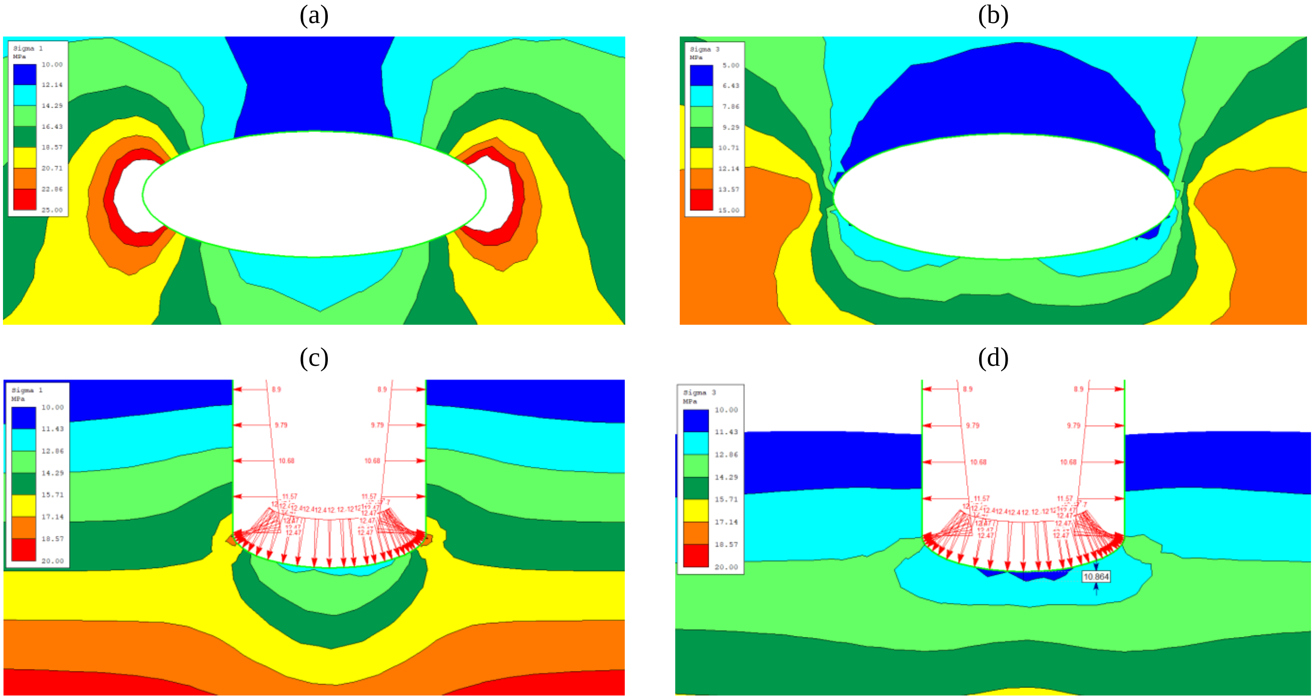


Figure 9. Modeling results of determining stresses in the mass by the Scheme 1: (a), (b) option of mining with backfilling of the mined-out space; (c), (d) option of mining with caving of host rocks

Figure 9a, b show that the principal stresses under the mined-out space during mining of the deposit with backfilling are σ_1 = 12-18 MPa, σ_3 = 6-8 MPa. In the stope space sides, the highest principal stresses are σ_1 = 25-30 MPa, σ_3 = 8-10 MPa.

The principal stresses in the mined-out space bottom during mining with caving (Fig. 9 c, d) are σ_1 = 13-15 MPa, σ_3 = 11-12 MPa. In the same conditions, the highest principal stressses in the stope space sides are σ_1 = 16-19 MPa, σ_3 = 13-14 MPa.

It should be noted that here and below in all models the stresses are identified in two zones:

– under the mined-out space with a thickness of ~20 m;

– in the mined-out space sides, in the zone with a width of ~20 m.

In these zones, the greatest difference between the principal stresses will be expected for the entire period of mining the deposit reserves. In addition, capital and preparatory workings can be expected to be located in these zones.

That is, the minimum principal stress value σ_3 in the case of the use of backfilling is lower than when mining with caving, and σ_1, on the contrary, is somewhat higher. Obviously, these results cannot be considered realistic, since it is known that the use of backfilling results in less influence of the mined-out space on the stress state. This discrepancy is caused by both the underestimation of the minimum principal stresses in the modeling with this scheme of backfilling and their overestimation in the modeling with caving.

The first is caused by the fact that the FEM underestimates the roof subsidence in the mined-out space due to the lack of the ability to explicitly model the mass ruptures. That is, above the backfilled volume due to underbackfilling, shrinkage and compression of the backfill, roof ruptures occur, and a cave roof is formed. The mass thus disintegrated exerts pressure on the backfill and the mass area under it. These processes cannot be reflected in finite element models when modeled according to the above scheme (Fig. 9 a, b).

As for the modeling of the mining system with caving, the inaccuracy in the given scheme is that the model does not reflect the possibility of formation of rock barricades, overhangs of rocks over the mined-out space, and locking of the caved mass against the lateral stable surrounding mass. This causes the entire weight of the caved rock to be placed on the bottom of the mined-out space, thus overestimating the minimum principal stress value.

Add to the description of Figure 9 a-d, that the principal stresses at the first stage (“green field”) at the same elevations are σ_1 = 16 MPa, σ_3 = 12 MPa. This confirms that the natural stress field setting is correct when the depth and unit specific gravity of the rocks are taken into account.

The calculations performed using the finite element method according to Scheme 2 resulted in the following results (Fig. 10 a, b and 10 c, d). The stresses occurring in the mass under the mined-out space have been determined. The principal stresses for the mining system with backfilling (Fig. 10 a, b) are as follows σ_1 = 14-19 MPa, σ_3 = 5-7 MPa. In the side of the stope space, the principal stresses are σ_1 = 22-26 MPa, σ_3 = 5-9 MPa.

When mining reserves using mining system with caving (Fig. 10 a, b), under the mined-out space, the principal stresses are σ_1 = 22-27 MPa, σ_3 = 6-8 MPa. Principal stresses obtained in the sides are σ_1 = 27-31 MPa, σ_3 = 7-11 MPa. Note again that the stresses are identified in zones of about 20 m in the side and under the mined-out space, where capital and preparatory mine workings can be expected to be located.

The modeling results of Scheme 2 seem to be more realistic compared to Scheme 1, as the principal stress values differ more when using system with caving than when using system with backfilling of the mined-out space. That is, these models are more in line with practice-based expectations. At the same time, it should be noted that the minimum principal stress value of σ_3 is smaller than expected. Change in model compared to Scheme 1 does not lead to the expected increase in σ_3.

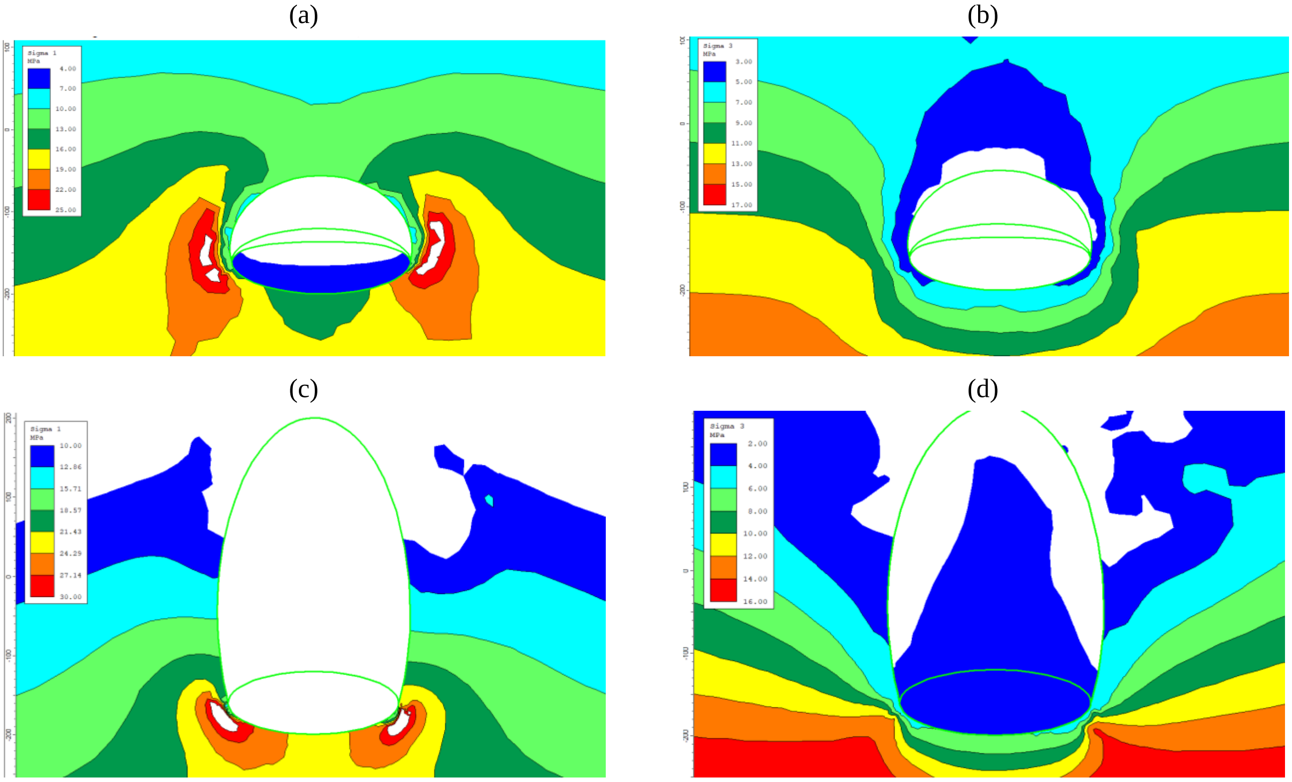


Figure 10. Modeling results by the Scheme 2: (a), (b) option of mining with backfilling of the mined-out space; (c), (d) option of mining with caving of host rocks

It can be stated that finite element modeling by two schemes, equally valid for application, has resulted in rather different results. This additionally shows that it is necessary to determine the stresses using another method that more accurately reflects the failure processes in the mass. This requirement is well met by the finite-discrete element method. Figure 11 demonstrates the results of simulation of mining an ore deposit with backfilling.

The areas with a large difference in principal stresses are most important, since it is the great differences in stress that pose a risk to the stability of mine workings. Special attention is also paid to the areas where the principal stress reversals are observed, as they may cause concentrations in other areas of the mine working.

The areas identified by the specified features are located in the mass in the side of the mined-out space and under it (Fig. 11). In the most part of the area under the extraction site, the highest principal stress during mining using the systems with backfilling is σ_1 = 17 MPa. In the same area, there are small sections (up to ~10 m in size) of both relaxation and stress concentration in the range of σ_1 = 10-30 MPa, but they will not be taken into account in further calculations as a special case. The lowest principal stress value in most of the considered area is within σ_3 = 4 MPa, at the same time varying in local sections in the interval of σ_3 = 0-10 MPa.

In the sides of the extraction site, there are areas with the highest stress value of σ_1 = 30 MPa, where the lowest principal stress value is σ_3 = 7-10 MPa. Figure 12 shows the predicted modeling consequences of reserve mining using mining systems with caving. Under the mined-out space, the highest principal stress value is approximately σ_1 = 17 MPa. Also in this area, there are sections (up to 10-20 m in size) of both relaxation and stress concentration σ_1 = 10-30 MPa, but they should be considered as a special case.

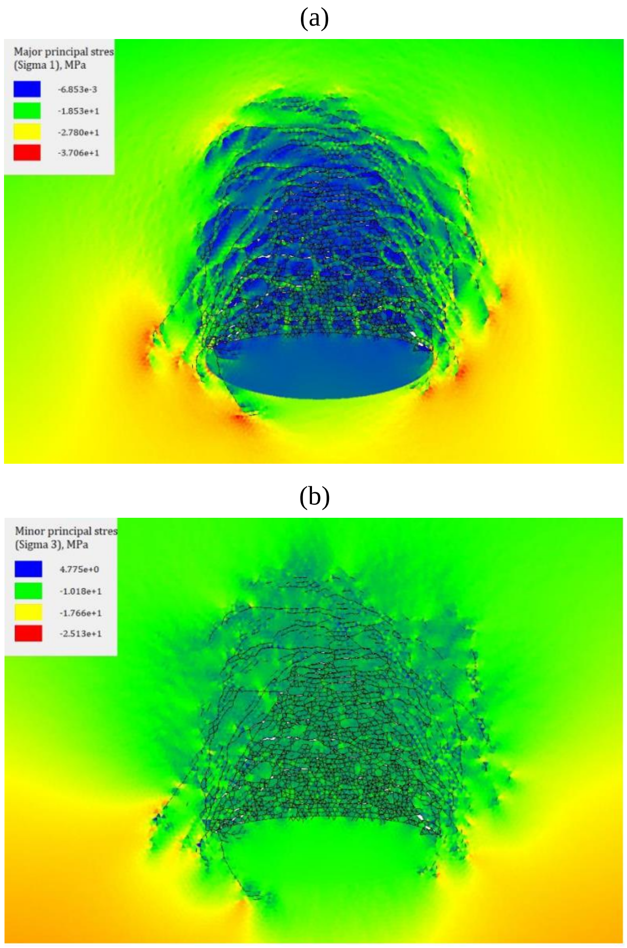


Figure 11. FDEM modeling results by the Scheme 3 (option of mining with backfilling of the mined-out space): (a) sigma 1; (b) sigma 3

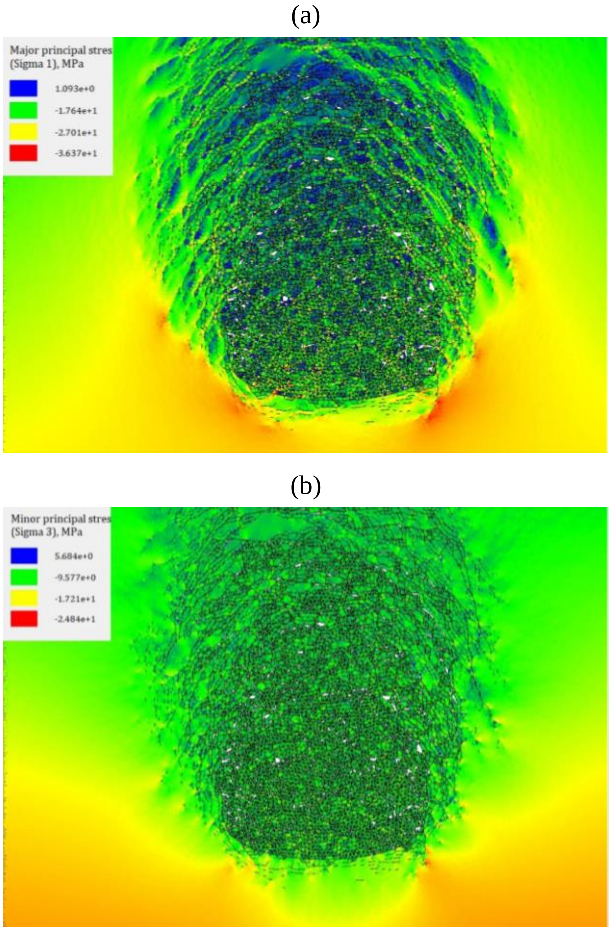


Figure 12. FDEM modeling results by the Scheme 3 (option of mining with caving of host rocks): (a) sigma 1; (b) sigma 3

The lowest principal stress in the most part of the considered area is in the range of σ_3 = 5-9 MPa, while varying in local sections in the range of σ_3 = 3-15 MPa. In the sides of the extraction site, the highest principal stress is on average σ_1 = 25-30 MPa. The minimum principal stress value in this area varies in the range of σ_3 = 9-15 MPa.

In addition to the above information, when analyzing the data obtained as a result of modeling, the following can be noted. The caving zone height during mining with backfilling is 125 m or 7.8 m_e, the fracture zone height is another 60 m. When mining with caving, the same parameters are 280 m (3.6 m) and 470 m, respectively. In this case, identifying the caving zone is much more difficult in the second case, since the degree of the mass disintegration changes smoothly enough from bottom to top (Fig. 13). The fracture zone almost reaches the earth's surface when using systems with caving of the host rocks.

The realistic nature of the results is confirmed by the subsidence values observed on the earth's surface. When modeling mining with caving, subsidence value is up to 2 m. Note that the highest surface subsidence of 0.6 m in FEM modeling has been obtained according to Scheme 2 with caving.

An interesting phenomenon is the formation of protrusion wedge near the surface, due to the formation of which in the central part of the shift trough, the subsidence is characterized by lower values than closer to the edges from the center (Fig. 14). A similar phenomenon is observed in the FEM modeling according to Scheme 2.

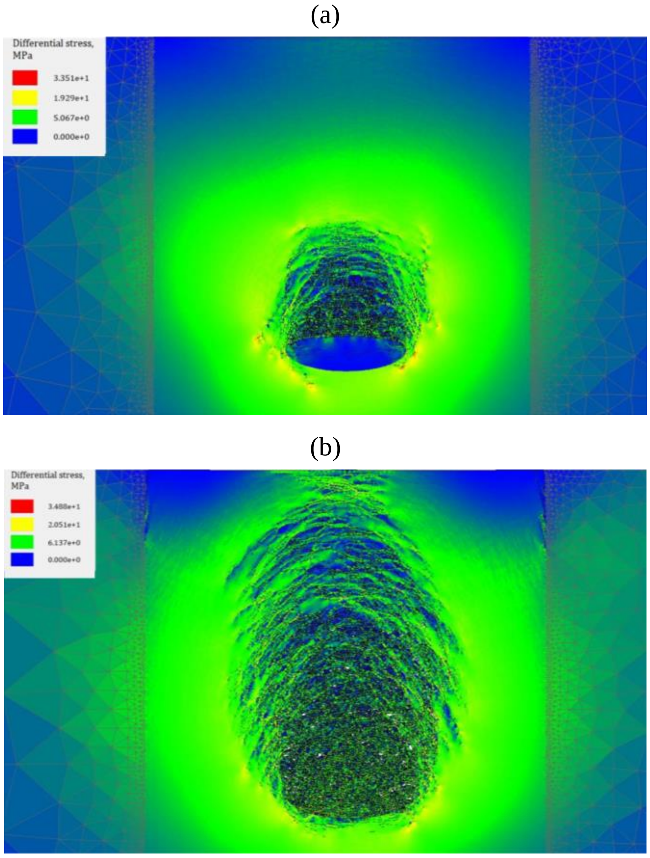


Figure 13. Differential stresses acting in the mass by end of reserve mining: (a) system with backfilling; (b) system with caving

In the FDEM models in the Prorock software, a quite large fracture zone is formed under the mined-out space, where the mass de-stressing occurs. Partially, the formation of this fracture zone is associated with the dynamic action of falling pieces of host rocks, which will not occur in reality, since the deposit mining does not occur at once for the entire thickness, as it happens in the simulation. Therefore, the formation of this fracture zone with a lower thickness should be expected in reality.

The areas of de-stressing and stress concentration formed under the mined-out space in the FDEM modeling show the possible variation of stress values in the case of in-situ stress measurements. That is, Figures 11-13 demonstrate the complexity of conducting stress measurements in the zone of influence of the mined-out space.

The final principal stress values, determined by modeling by the three schemes, are given in Table 5. Comparison of the stresses obtained by FDEM modeling with the results of FEM modeling shows that the stresses determined by the finite-discrete element method are within the similar values obtained by the finite element method. For this reason, and also considering the fundamental peculiarities, disadvantages and advantages of the methods, for further calculations assume the stresses determined according to Scheme 3. Based on the obtained data, it is possible to perform modeling of the mine working to assess its stability in the field of induced stresses.

The modeling is performed similarly to the one performed at the previous stage in the natural stress field. The only difference is in the stress field, which is now specified taking into account the influence of the stope space.

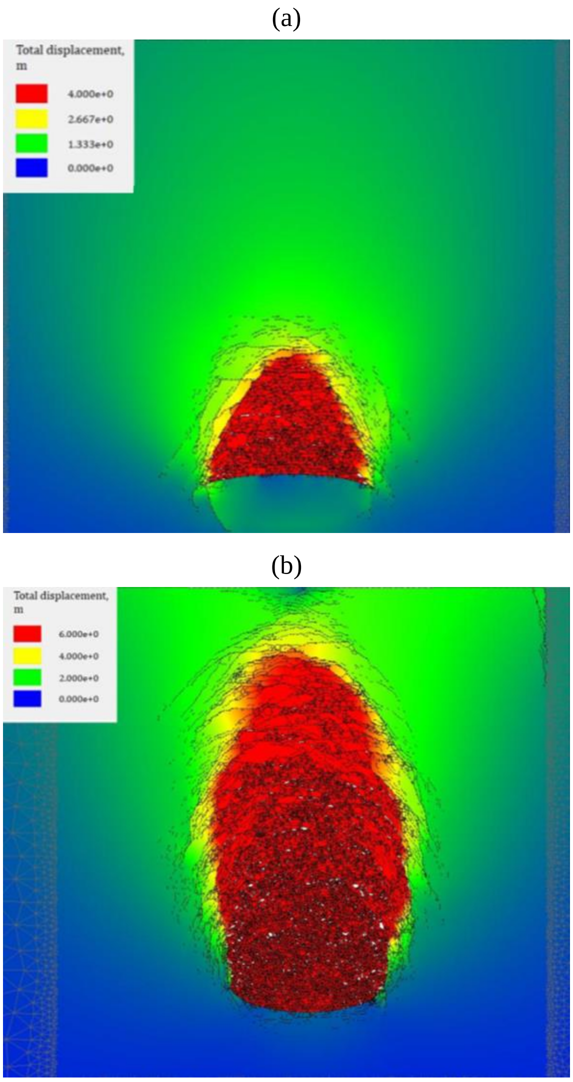


Figure 14. Mass displacements by the end of reserve mining: (a) system with backfilling; (b) system with caving

Table 5. Averaged induced stresses determined by numerical modeling

Mining system	Under the mined-out space		To the side of the mined-out space	
	σ_1, MPa	σ_3, MPa	σ_1, MPa	σ_3, MPa
Scheme 1 (FEM)				
With backfilling	15	7	27	9
With caving	14	11	18	13
Scheme 2 (FEM)				
With backfilling	17	6	24	7
With caving	25	7	29	9
Scheme 3 (FEM)				
With backfilling	17	6	27	10
With caving	17	7	27	10

Thus, the induced stress field is characterized by the following values and rotation angles (Table 6).

Table 6. Estimated parameters of the artificial stress field

Stress orientation	Option 1		Option 2	
	Value, MPa	Inclination angle, °	Value, MPa	Inclination angle, °
Maximum principal stres σ_1	17	0	27	90
Minimum principal stress σ_3	6	90	10	0

Table 6 values are specified in the FDEM models. The principal stresses in Option 1 may be characterized by some rotation (up to 20°) away from the center of the deposit, but the rotation value is rather insignificant and, in our opinion, may not be taken into account. A total of 4 models have been constructed for two options and in two types of rocks: in serpentinites and in ore. Physical-mechanical properties of rocks and ore are presented in Table 7. Properties are taken with mine working standing time of 6 hours before supporting. Figure 15 shows the results of mine working modeling in the induced stress field.

Table 7. Material properties taken for modeling

Rock name	Serpentinite	Ore
Adhesion, MPa	11	5.6
Internal friction angle, °	41	40
Tensile strength, MPa	3.5	1.9
Unit specific gravity, ton/m^3	2.55	4.01
Deformation modulus, MPa	39.8	13.8
Poison's ratio	0.25	0.24

From the modeling results (Fig. 15) it can be seen that the mine working stability with the taken supporting parameters is provided under the specified conditions. Figure 15 (Option 2, ore) demonstrates that the mine working stability cannot be ensured in the ore when the mine working is located in the side areas of the stope space. That is, in such circumstances, a different scheme for supporting should be considered.

A relatively higher inrush volume is observed in ore with worse strength properties (Fig. 15, Option 1, ore) and at higher stresses in the mass (Fig. 15, Option 1, serpentinite).

The inrush volume and intensity of fracture opening in the models (Fig. 15) are relatively the same as that observed in the models in the natural stress field. However, the inrush volume in the mine working driven in serpentinite in the side of the stope space is still somewhat larger and similar to the situation observed when analyzing the stability of mine working supported with violations of technology, namely with reduced load-bearing capacity of roof bolts.

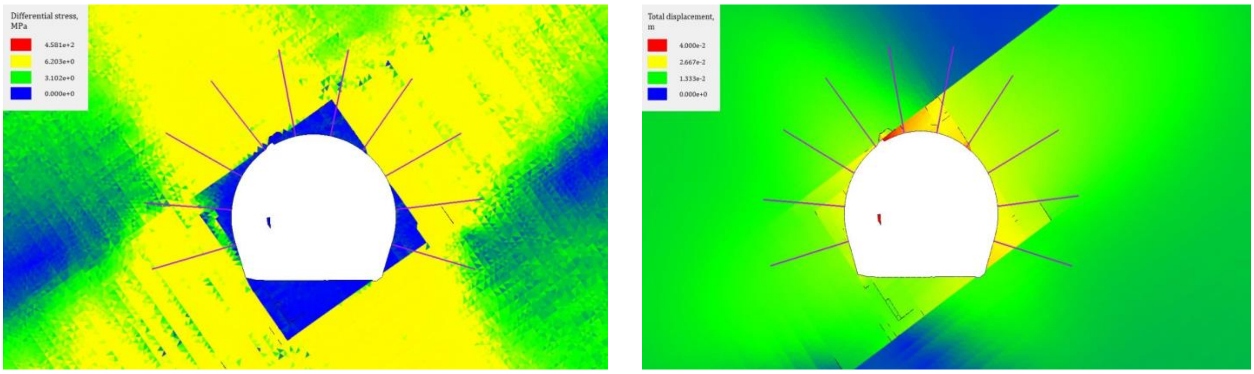
The value of the displacements on the mine working contour is interesting. As for mine working in serpentinite in Option 1 SSS, does not undergo significant displacements that barely reach 4 cm in the model. In Option 2 SSS, displacements exceed 5 cm. In ore (Option 1 SSS) displacements on the contour exceed 7 cm, which according to the experience of modeling at the previous stage, is a dangerous value.

Based on the displacement values, it can be said that the supporting of mine workings under the stope space in serpentinite can be somewhat optimized. At the same time, mine workings driven there through the ore are likely to have a low safety factor value.

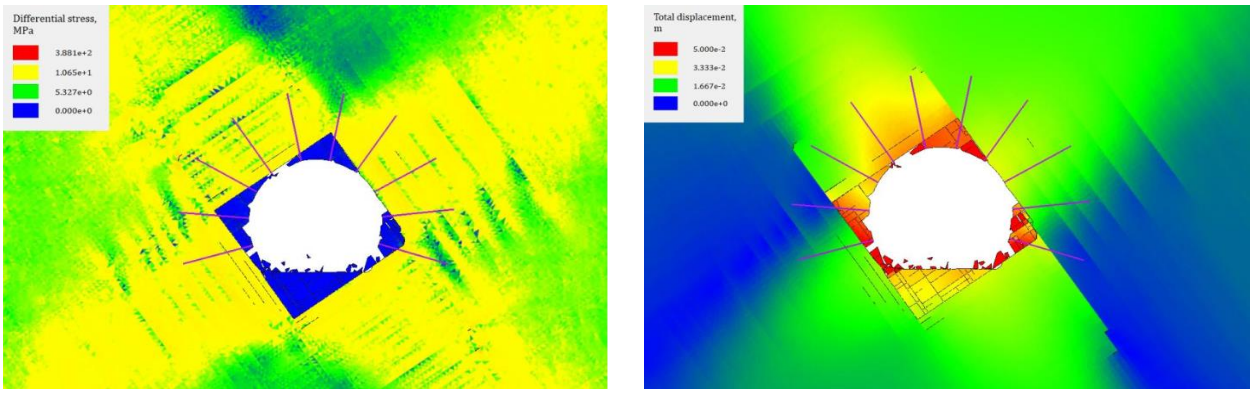
It is characteristic that the modeling results show an increase in the intensity of inrush and failure beyond the mine working contour in the areas of stress concentration. In Option 1 SSS (Fig. 15), concentration areas are located in the roof and bottom of the mine working, in Option 2 (Fig. 13) – in the sides. This indirectly indicates the correctness and realistic nature of the models, as well as the fact that FDEM models can be used for reverse analysis of the mass stress state based on the actual deformations of mine workings.

In case the mine working is located in the side of the stope space in the ore, its destruction occurs, therefore, the decision is made to review the supporting scheme.

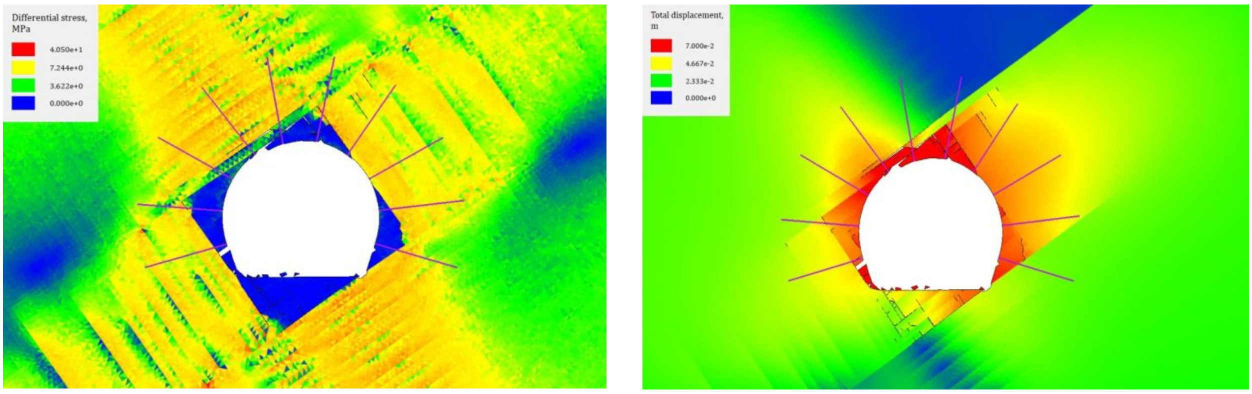
Option 1, serpentinite



Option 2, serpentinite



Option 1, ore



Option 2, ore

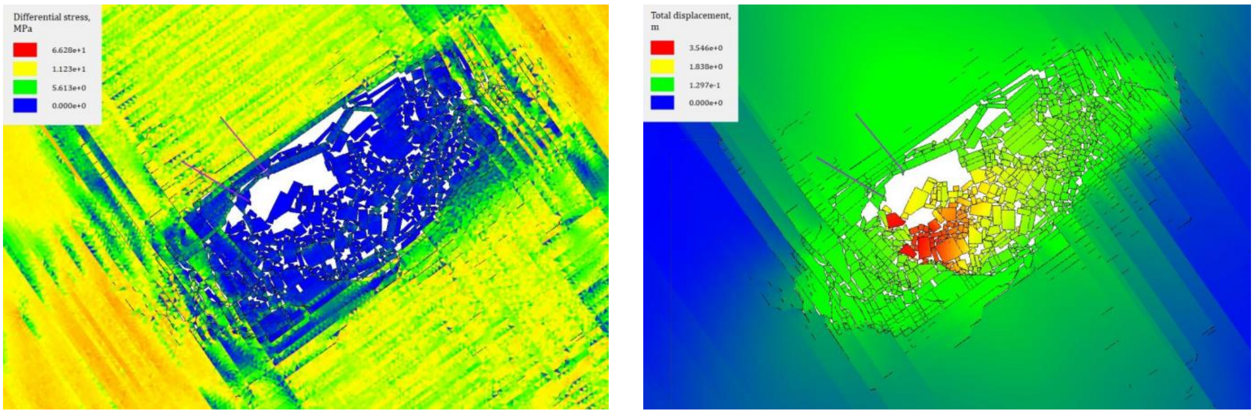


Figure 15. Results of mine working modeling in the induced stress field

Figure 16 shows the zones where intense formation of rupture and shear fractures in the rock occurs, that is, the rock barricades are destroyed. Behind these zones in the mass there are stable zones that do not undergo displacements. In order to ensure the mine working stability, it is necessary to fix the isolated sections of the mass relative to the stable areas. This can be achieved by using cable bolts, which have a long length and high load-bearing capacity.

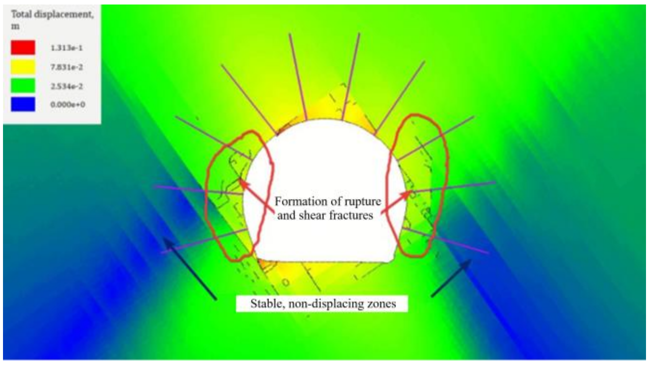


Figure 16. Displacements and fracturing behind the mine working contour before caving (Option 2 SSS, ore)

Figure 17 also shows the mechanism of mine working caving, where it is shown that the mine working sides cave first. Some displacement of failure epicenters in the diagonal direction occurs due to patterns of fracturing, but conditions of fracturing may be variable, and the accepted systemic fracturing of the mass cannot be taken as a general case. Therefore, this displacement is not taken into account.

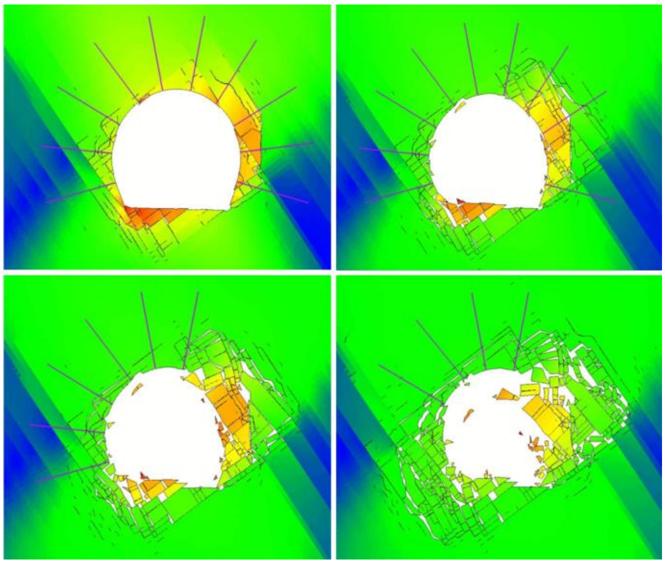


Figure 17. Mechanism of mine working failure (Option 2 SSS, ore)

As a result, three possible supporting schemes have been selected, as follows (Fig. 18). In the first one, the roof bolts are set in the same way as in the previous model, while the mine working sides are additionally fastened with cable bolts. In the second scheme, two cable bolts are added to strengthen the mine working walls and the number of roof bolts is increased along the perimeter. In this case, the distance between roof bolts is left equal to 0.9 m, but the distance from the bottom to the first roof bolt is reduced to 0.45 m. In the third scheme, the bottom is additionally fastened with roof bolts and the sides with cable bolts.

The load-bearing capacity of the cable bolts is assumed to be 210 kN. The deformation characteristics of cables are assumed to be similar to roof bolts. Figures 19 and 20 show the modeling results that illustrate the changes in the mine working stability, when fastened with additional two and four cable bolts, respectively. Analysis of the obtained data makes it possible to assess the effectiveness of various supporting schemes under conditions of a specified SSS.

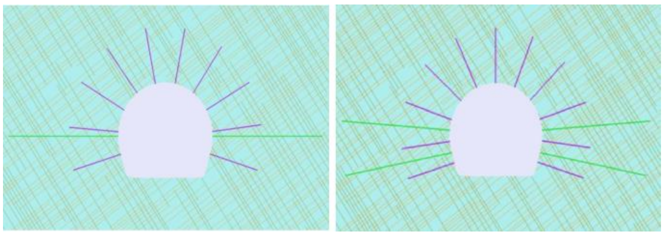


Figure 18. Schemes for supporting an unstable mine working: options with two, four and six cable bolts

Figure 19 shows the results of deformation modeling of the mine working fastened according to the first option, that is, with two additional cable bolts. From this figure it can be seen that the mine working experiences deformations that can hardly be considered acceptable. That is, the first adopted option of additional supporting can be considered unsuccessful. It should be noted that the effect of the cables is clearly visible and consists in increasing the stability of the mine working walls. However, this causes unacceptable deformation of the sides in the lower part, closer to the mine working bottom.

It can be seen from Figure 20 of the second supporting option that the mine working stability is ensured with the adopted supporting scheme. However, there is a very intense fracturing behind the contour, which is accompanied by a noticeable mine working convergence and significant stretching of the roof bolts. Displacements of the mine working contour reach more than 14 cm from the moment of driving the mine working, and the bottom is displaced to an even greater extent. Such deformations of the mine working will result in the need for additional supporting (at least using shotcrete) and cleaning-up of the bottom.

Based on the modeling results, the following supporting schemes are recommended for use:

– strengthening of the combined roof-bolt-and-shotcrete support with crown runners (headboard) over the outcrop area by adding second-order roof bolts: up to 6-8 m deep. In this case, the distance between the roof bolts of the first order is not more than 0.9 m, the distance from the bottom to the first roof bolt is 0.45 m.

– the bottom is additionally fastened with roof bolts and the sides are fastened with cables.

Prospects for further research include in-depth analysis of the stability of mine workings, taking into account more complex geomechanical conditions. Additional research is needed to optimize the supporting parameters depending on the type of ores and rocks, taking into account their strength characteristics and fracturing. This will make it possible to develop more accurate recommendations on the selection of supporting systems for different geological conditions and improve the safety of mining operations. In addition, further research should focus on the development of new types of roof bolts with improved characteristics, such as increased tensile strength.

It is also worth studying the influence of factors such as seismic activity and water saturation on the stability of mine workings. Implementation of monitoring systems to control the state of supporting and mass deformations in real time will make it possible to react quickly to changes, ensuring higher safety and efficiency of mining operations at deep levels.

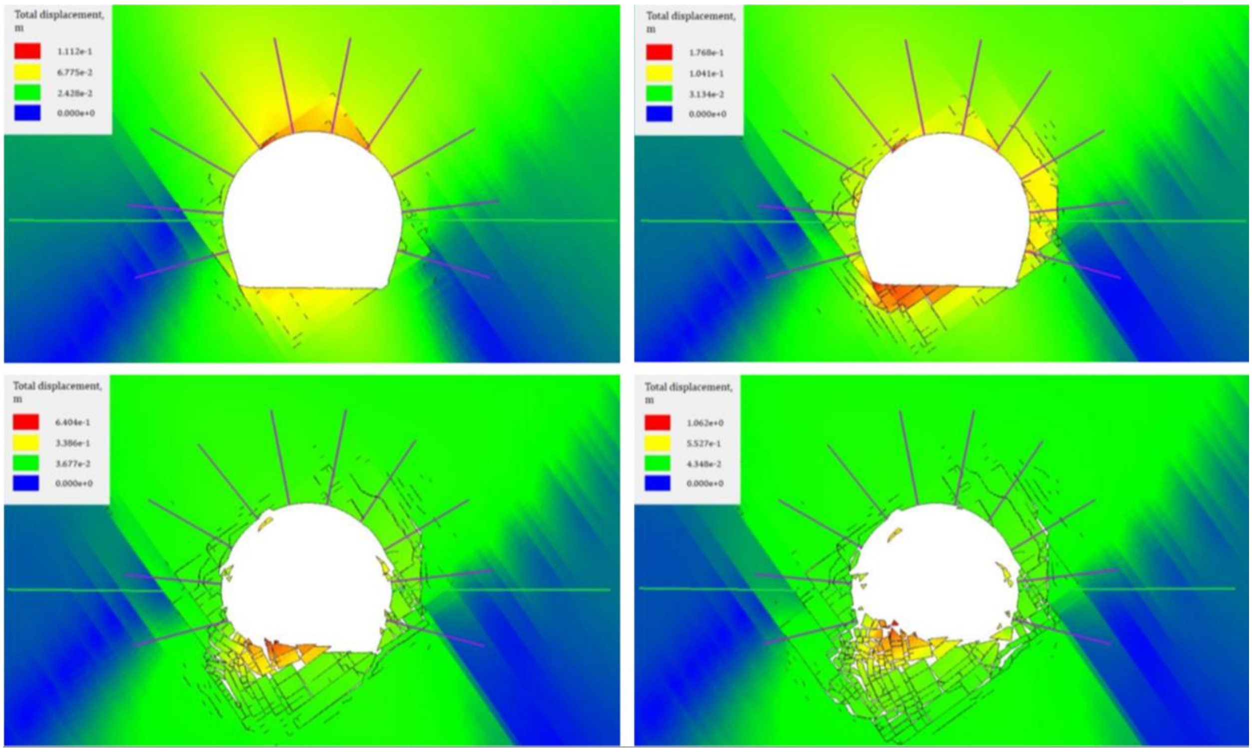


Figure 19. Simulation result of mine working in ore for Option 2 SSS with two additional cable bolts

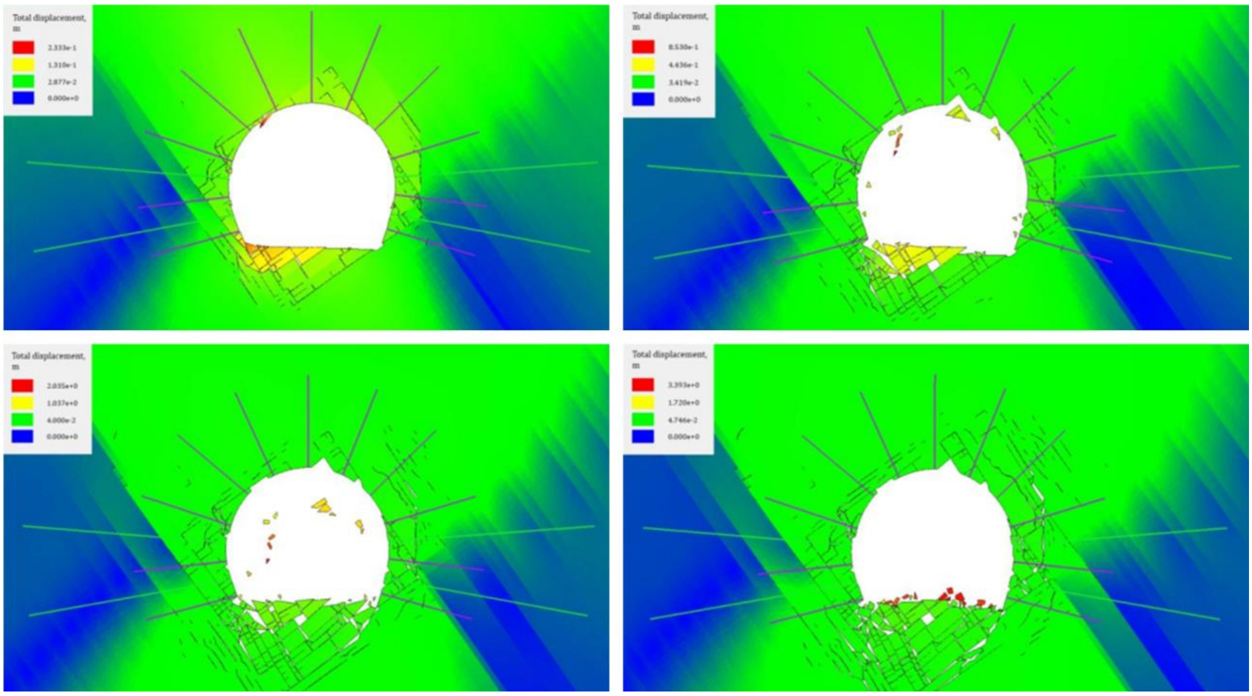


Figure 20. Simulation result of mine working in ore for Option 2 SSS with four additional cable bolts

4. Conclusions

The rocks and ores of the deposit are characterized by low strength properties (σ_c =10.7-97 MPa), which together with a high degree of disturbance of the mass by fractures, characterizes the mass composed by them as unstable (very unstable).

Based on the obtained data, modeling of mine workings has been performed to assess stability in the field of induced stresses. The analysis of models has shown that mine working stability with the adopted parameters of supporting is provided under the specified conditions for serpentinites and ore according to the Option 1.

It should be noted that as a result of the mass SSS modeling around the supported mine working, the effect of the cables is clearly visible and consists in increasing the stability of the mine working walls. However, in this case, unacceptable deformations in the lower part of the sides and bottom of the mine working are to be expected.

Increasing the number of cable bolts to 4 ensures the mine working stability. However, due to the action of stresses, there is a very intense fracturing behind the contour, which can be accompanied by a noticeable convergence (up to 14 cm) of the contours. The second-order roof bolts should have significant tensile strength. Such deformation of the mine working will result in the need to provide additional supporting (at least shotcrete restoration). In this case, it is necessary to have the perimeter of the mine working completely covered with reinforcing mesh and reinforcing frame in two layers.

Author contributions

Conceptualization: BU, ZS; Data curation: AM; Formal analysis: DK, GI; Funding acquisition: ZS; Investigation: AM, MI; Methodology: AM, RO; Project administration: BU, GI; Resources: DK; Software: AM, ZS; Supervision: RO; Validation: BU, ZS; Visualization: AM, GI; Writing – original draft: DK; Writing – review & editing: AM, MI, RO. All authors have read and agreed to the published version of the manuscript.

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Conflicts of interests

The authors declare no conflict of interest.

Data availability statement

The original contributions presented in the study are included in the article, further inquiries can be directed to the corresponding author.

References

[1] Tazhibekova, K., Shametova, A., Urazbekov, A., Akhmetzhanov, B., Akenov, S., & Tulupova, S. (2020). Enhancing eco-economic efficiency of mineral deposit exploration to achieve sustainable development in the mining industry of Kazakhstan. *Progress in Industrial Ecology, an International Journal*, *14*(3-4), 212-228. https://doi.org/10.1504/PIE.2020.113425

[2] Aubakirova, G., Rudko, G., & Isataeva, F. (2021). Assessment of metallurgical enterprises' activities in Kazakhstan in the context of international trends. *Economic Annals-XXI*, *187*. https://doi.org/10.21003/ea.V187-12

[3] Issatayeva, F.M., Aubakirova, G.M., Maussymbayeva, A.D., Togaibayeva, L.I., Biryukov, V.V., & Vechkinzova, E. (2023). Fuel and energy complex of Kazakhstan: Geological and economic assessment of enterprises in the context of digital transformation. *Energies*, *16*(16), 6002. https://doi.org/10.3390/en16166002

[4] Sadeghi, S., Soltanmohammadlou, N., & Nasirzadeh, F. (2022). Applications of wireless sensor networks to improve occupational safety and health in underground mines. *Journal of Safety Research*, *83*, 8-25. https://doi.org/10.1016/j.jsr.2022.07.016

[5] Chen, C., Zhou, J., Zhou, T., & Yong, W. (2021). Evaluation of vertical shaft stability in underground mines: comparison of three weight methods with uncertainty theory. *Natural Hazards*, *109*(2), 1457-1479. https://doi.org/10.1007/s11069-021-04885-5

[6] Yousufi, A., Ahmadi, H., Bekbotayeva, A., Arshamov, Y., Baisalova, A., Omarova, G., & Pekkan, E. (2023). Integration of remote sensing and field data in ophiolite investigations: A case study of Logar ophiolite complex, SE Afghanistan. *Minerals*, *13*(2), 234. https://doi.org/10.3390/min13020234

[7] Wu, M., Hu, N., Ye, Y., Wang, Q., & Wang, X. (2022). Multi-hazard risk characterization and collaborative control oriented to space in non-coal underground mines. *Scientific Reports*, *12*(1), 16452. https://doi.org/10.1038/s41598-022-20437-8

[8] Bai, C., Yun, M. X., & Wang, J. M. (2020). Hazards of environmental disruption in mine goafs and stability evaluation in Gaofeng Mining Area. *Nature Environment & Pollution Technology*, *19*(3), 949-953. https://doi.org/10.46488/NEPT.2020.v19i03.005

[9] Jia, H., Yan, B., Guan, K., Liu, H., Wu, Q., Yin, Y., & Liu, H. (2022). Stability analysis of shallow goaf based on field monitoring and numerical simulation: A case study at an Open-Pit Iron Mine, China. *Frontiers in Earth Science*, *10*, 897779. https://doi.org/10.3389/feart.2022.897779

[10] He, R., Liu, H., Ren, F., Li, G., Zhang, J., & Zhou, Y. (2022). Comprehensive evaluation and decision for goaf based on fuzzy theory in underground metal mine. *Advances in Civil Engineering*, *2022*(1), 3104961. https://doi.org/10.1155/2022/3104961

[11] Jiang, L., Yang, C., & Jiao, H. (2020). Ultimately exposed roof area prediction of bauxite deposit goaf based on macro joint damage. *International Journal of Mining Science and Technology*, *30*(5), 699-704. https://doi.org/10.1016/j.ijmst.2020.06.005

[12] Matayev, A.K., Kainazarova, A.S., Arystan, I.D., Abeuov, Ye., Kainazarov, A.S., Baizbayev, M.B., Demin, V.F., & Sultanov, M.G. (2021). Research into rock mass geomechanical situation in the zone of stope operations influence at the 10th Anniversary of Kazakhstan's Independence mine. *Mining of Mineral Deposits*, *15*(1), 1-10. https://doi.org/10.33271/mining15.01.042

[13] Sultanov, M.G., Mataev, A.K., Kaumetova, D.S., Abdrashev, R.M., Kuantay, A.S., & Orynbayev, B.M. (2020). Development of the choice of types of support parameters and technologies for their construction at the Voskhod field. *Ugol*, *10*, 17-21. https://doi.org/10.18796/0041-5790-2020-10-17-21

[14] Teplická, K., Khouri, S., Beer, M., & Rybárová, J. (2021). Evaluation of the performance of mining processes after the strategic innovation for sustainable development. *Processes*, *9*(8), 1374. https://doi.org/10.3390/pr9081374

[15] Satybaldiyeva, D., Mukhanova, G., Tymbayeva, Z., Tyshkanbayeva, M., & Bolatkyzy, S. (2023). Applying the expert method to determine a company's strategic goals. *Transport Problems*, *18*(2), 123-132. https://doi.org/10.20858/tp.2023.18.2.11

[16] Kalpeyeva, Z., Kassymova, A., Umarov, T., Mustafina, A., & Mukazhanov, N. (2020,). The structure and composition of the business process model. *ACM International Conference Proceeding Series*, 1-6. https://doi.org/10.1145/3410352.3410783

[17] Urban, B., & Oosthuizen, J.H. (2009). An empirical analysis of corporate entrepreneurship in the South African mining industry. *Journal of Contemporary Management*, *6*(1), 170-192.

[18] Aitzhanova, D., Begentayev, M., Mukhanova, G., Antoni, A., Csiszárik-Kocsir, Á., Orazymbetova, A., & Kiss, F. (2024). Success factors of youth entrepreneurship projects based on fuzzy analytic hierarchy method on the example of Kazakhstan. *Acta Polytechnica Hungarica*, *21*(11), 135-154. https://doi.org/10.12700/APH.21.11.2024.11.8

[19] Mononen, T., Sihvonen, J., Sairinen, R., & Tiainen, H. (2023). Local governance of the mining industry – five Finnish examples. *Resources Policy*, *82*, 103478. https://doi.org/10.1016/j.resourpol.2023.103478

[20] Badakhshan, N., Shahriar, K., Afraei, S., & Bakhtavar, E. (2023). Determining the environmental costs of mining projects: A comprehensive quantitative assessment. *Resources Policy*, *82*, 103561. https://doi.org/10.1016/j.resourpol.2023.103561

[21] Shaldarbekov, K.B., Mukhanova, G.S., Dossova, S.N., Mussaeva, G.K., Nurmukhambetova, Z.S., & Shaldarbekova, K.B. (2018). Problems of Regional Industrial Projects Realization. *Journal of Advanced Research in Law and Economics*, *9*(6(36)), 2119-2128. https://doi.org/10.14505/jarle.v9.6(36).27

[22] Dou, S., Xu, D., & Keenan, R. J. (2023). Effect of income, industry structure and environmental regulation on the ecological impacts of mining: An analysis for Guangxi Province in China. *Journal of Cleaner Production*, *400*, 136654. https://doi.org/10.1016/j.jclepro.2023.136654

[23] Arystan, I.D., Nemova, N.A., Baizbaev, M.B., & Mataev, A.K. (2021). Efficiency of modified concrete in lining in underground structures. *IOP Conference Series. Earth and Environmental Science*, *773*(1), 012063. https://doi.org/10.1088/1755-1315/773/1/012063

[24] Moldagozhina, M.K., Krupnik, L., Koptileuovich, Y.K., Mukhtar, E., & Roza, A. (2016). The system is "roof bolting-mountain". *International Journal of Applied Engineering Research*, *11*(21), 10454-10457.

[25] Vladyko, O., Kononenko, M., & Khomenko, O. (2012). Imitating modeling stability of mine workings. *Geomechanical Processes During Underground Mining*, 147-150. https://doi.org/10.1201/b13157-26

[26] Petlovanyi, M., Medianyk, V., Sai, K., Malashkevych, D., & Popovych, V. (2021). Geomechanical substantiation of the parameters for coal auger mining in the protecting pillars of mine workings during thin seams development. *ARPN Journal of Engineering and Applied Sciences*, *16*(15), 1572-1582.

[27] Bekbergenov, D., Jangulova, G., Kassymkanova, K.K., & Bektur, B. (2020). Mine technical system with repeated geotechnology within new frames of sustainable development of underground mining of caved deposits of the Zhezkazgan field. *Geodesy and Cartography*, *46*(4), 182-187. https://doi.org/10.3846/gac.2020.10571

[28] Kononenko, M., & Khomenko, O. (2010). Technology of support of workings near to extraction chambers. *New Techniques and Technologies in Mining – Proceedings of the School of Underground Mining*, 193-197. https://doi.org/10.1201/b11329-31
[29] Ratov, B. (2023). Calculations on enhancement of polycrystalline diamond bits through addition of superhard diamond-reinforced elements. *Machines*, *11*(4), 453. https://doi.org/10.3390/machines11040453
[30] Pysmennyi, S., Chukharev, S., Peremetchy, A., Fedorenko, S., & Matsui, A. (2023). Study of stress concentration on the contour of underground mine workings. *Inżynieria Mineralna*, *1*(1(51)), 69-78. https://doi.org/10.29227/IM-2023-01-08
[31] Arystan, I.D., Baizbaev, M.B., Mataev, A.K., Abdieva, L.M., & Bogzhanova, Z.K. (2020). Selection and justification of technology for fixing preparatory workings in unstable massifs on the example of the mine "10 years of Independence of Kazakhstan. *Ugol*, *6*, 10-14. https://doi.org/10.18796/0041-5790-2020-6-10-14
[32] Dorador, L., Eberhardt, E., & Elmo, D. (2021). Procedure for estimating broken ore density distribution within a draw column during block caving. *Mining Technology*, *130*(3), 131-145. https://doi.org/10.1080/25726668.2021.188654
[33] Board, M., & Pierce, M.E. (2009). A review of recent experience in modeling of caving. *Proceedings of the International Workshop on Numerical Modeling for Underground Mine Excavation Design*, *9512*, 19-28.
[34] Ladinig, T., Wimmer, M., & Wagner, H. (2022). Raise caving: a novel mining method for (deep) mass mining. *Caving*, 651-666. https://doi.org/10.36487/ACG_repo/2205_45
[35] Han, U.C., Kang, I.Y., Pak, K.I., Hong, K.U., Ri, C., Choe, S.G., & Ri, Y.I. (2023). Numerical analysis of pillar stability for safe mining of a combined mine based on three-dimensional solid model. *ISRM Congress*, 15.
[36] Yang, C., Zhou, K., Gao, R., & Xiong, X. (2021). Numerical investigation of the dynamic response of a preconditioned roof in an underground mine: a case study of mining environment regeneration. *Soil Dynamics and Earthquake Engineering*, *140*, 106457. https://doi.org/10.1016/j.soildyn.2020.106457
[37] Li, Y., Ma, K., Li, H., Tao, Y., & Liu, G. (2022). Numerical simulation research of excavation based enhanced geothermal system. *Journal of Physics: Conference Series*, *2254*, 012013. https://doi.org/10.1088/1742-6596/2254/1/012013
[38] Chitombo, G.P. (2010). Cave mining: 16 years after Laubscher's 1994 paper "Cave mining – state of the art". *Mining Technology*, *119*(3), 132-141. https://doi.org/10.1179/174328610X12820409992255
[39] Zeitinova, S., Imashev, A., Bakhtybayev, N., Matayev, A., Mussin, A., & Yeskenova, G. (2024). Numerical modeling the rock mass stress-strain state near vertical excavations in combined mining. *Civil Engineering Journal*, *10*(9), 2919-2934. https://doi.org/10.28991/CEJ-2024-010-09-010
[40] Imashev, A.Zh., Sudarikov, A.E., Musin, A.A., Suimbayeva, A.M., & Asan, S.Yu. (2021). Improving the quality of blasting indicators by studying the natural stress field and the impact of the blast force on the rock mass. *News of the National Academy of Sciences of the Republic of Kazakhstan*, *4*(448), 30-35. https://doi.org/10.32014/2021.2518-170x.78
[41] Diomin, V.F., Khalikova, E.R., Diomina, T.V., & Zhurov, V.V. (2019). Studying coal seam bedding tectonic breach impact on supporting parameters of mine workings with roof bolting. *Naukovyi Visnyk Natsionalnoho Hirnychoho Universytetu*, *5*, 16-21. https://doi.org/10.29202/nvngu/2019-5/5
[42] Sdvizhkova, Ye.A., Babets, D.V., & Smirnov, A.V. (2014). Support loading of assembly chamber in terms of Western Donbas plough longwall. *Naukovyi Visnyk Natsionalnoho Hirnychoho Universytetu*, *5*, 26-32.

Обґрунтування та вибір параметрів кріплення гірничих виробок на глибоких горизонтах

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Мета. Провести комплексне дослідження з обґрунтування та вибору оптимальних параметрів кріплення гірничих виробок на глибоких горизонтах на основі аналізу напружено-деформаційного стану масиву гірських порід, моделювання геомеханічних процесів та розробки ефективних технологій зміцнення у складних геологічних умовах.

Методика. Дослідження включає моделювання напружено-деформованого стану масиву гірських порід із використанням програмних комплексів ANSYS, Phase2 та Prorock. Для оцінки та прогнозування напружено-деформованого стану масиву в районі очисних блоків проведено серію чисельних експериментів для оцінки стійкості приконтурного масиву. Вихідними даними для моделювання слугували фізико-механічні характеристики руд і вміщуючих порід.

Результати. Аналіз НДС масиву підтвердив, що збільшення кількості тросових анкерів до чотирьох забезпечує стійкість виробки, проте, супроводжується інтенсивним тріщиноутворенням за контуром і конвергенцією до 14 см, що вимагає застосування армуючої сітки та армокаркасу у два шари. Встановлено, що елементи системи розробки забезпечують необхідний запас міцності при торцевому випуску руди ($k_s \geq 3.0$) та шаровому вийманні з твердіючим закладанням ($k_s \geq 5.0$).

Наукова новизна. Обґрунтовано параметри кріплення гірничих виробок, що забезпечує безпечне освоєння глибоких горизонтів. Виявлено, що при закладанні висота зони обвалення становить 125 м, а зони тріщин – 60 м. При системах з обваленням ці показники досягають 280 і 470 м відповідно.

Практична значимість. Розроблено технологію кріплення гірничо-капітальних виробок із застосуванням посиленого комбінованого кріплення та анкерів. Отримані результати сприяють підвищенню надійності та ефективності гірничих робіт, забезпечуючи точне прогнозування поведінки порід в умовах напружень, що змінюються, і зниження їх міцнісних характеристик.

Ключові слова: *напружений стан, масив, руда, порода, чисельний аналіз, кріплення, система розробки*

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