Mining of Mineral Deposits
Volume 16 (2022), Issue 1, 43-51

https://doi.org/10.33271/mining16.01.043

Substantiation of mining-technical and environmental safety of underground mining of complex-structure ore deposits

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Abstract

Purpose. Substantiation of mining-technical and environmental safety of underground mining of complex-structure ore deposits based on the study of a rock mass stress-strain state (SSS), as well as determining the safe parameters of stopes for specific mining-geological conditions and patterns of rock pressure propagation in various environments.

Methods. To assess the improvement of mining-technical and environmental safety of mining operations, the research includes theoretical generalizations with the use of mathematical statistics, physical and mathematical modeling, performing calculations, as well as technical and economic feasibility, laboratory and full-scale experimental studies, industrial tests in mine conditions and on the earth's surface in the zone of blasting influence, based on the standard and new methods.

Findings. A functional interrelation between the rock mass SSS value and the number of impulses (destruction sounds) per minute, characterizing its structural (a) and strength (b) properties, is proposed, which is described by a curvilinear dependence of the type $y = ax^b$ and allows to quickly determine the stable parameters of stopes. Assessment and prediction are made for various forms of rock pressure manifestation, based on the stress concentration coefficient within $0.91 < K_v < 0.70$, taking into account the conditions of the elastic behavior of rocks. The value of $\varepsilon = 0.0002\text{-}0.0003$ is taken as permissible relative deformation, which ensures both mining-technical and environmental safety, as well as the rock mass stability during repeated blasting operations.

Originality. A classification of the rock mass SSS has been developed depending on the direction of the maximum stresses relative to the mine working location, the level of the rock mass stress state and the mechanism of rock pressure manifestation (η), as well as the category of rock-bump hazard. In particular, the rock mass with the values in the range of $0.12 < \eta \leq 0.2$; $0.2 < \eta \leq 0.3$; $0.3 < \eta \leq 0.5$ and $\eta > 0.5$ are classified as non-rock-bump hazardous and belong to III, II, and I hazard categories, respectively.

Practical implications. To substantiate the safe parameters of stopes based on the results of multi-year research into underground mining of complex-structure deposits, depending on the rock mass properties, a graphical-analytic method (nomographic charts and calculation formulas) is recommended. These parameters are determined for highly fractured, medium fractured and weakly fractured rock mass with its horizontal outcropping to the surface. Using this method, the Instruction on the Geomechanical Substantiation of the Safe Mining of the Reserves at the Skhidnyi Hirnycho-Zbahachuvalnyi Kombinat, DP (SE VostGOK), Ukraine, has been developed.

Keywords: *complex-structure deposits, underground mining, geomechanics, blasting operations, seismic, mining-technical and environmental safety*

1. Introduction

In case of technogenic disturbance of the subsoil, a common problem in mining the ore deposits is the interaction of natural and technical systems that ensure the geomechanical balance of natural and artificial masses in the area of subsoil development. At the same time, the possibility of monitoring the rock mass stress-strain state (SSS) for a long period of time should be provided [1], [2]. Under the influence of mining operations, a zone of displacements and deformations exceeding the permissible values is distinguished in the upper layers of lithosphere [3]-[6]. Therefore, to ensure the safety of mining operations and the life activity of the population living in mining regions, it is important to study the mechanism of the occurrence and redistribution of SSS factors in the rock mass. This is an important scientific, practical and social task that requires an urgent solution [7]-[9].

Mining enterprises in the course of their activities are in direct contact with industrial zones, residential areas, natural objects such as water and agricultural lands, having an adverse effect on them. Underground mining at a shallow depth under secure facilities are technologically complex and dangerous, accompanied by intensification of geodynamic processes over time [10]-[13]. Thus, over the two-century history of mining the Sadonskoye deposit (North Caucasus, the Republic of North Ossetia, Alania), up to 5 million m^3 of unfilled technogenic cavities have been accumulated. Such a

Received: 28 April 2021. Accepted: 20 December 2021. Available online: 30 March 2022

Mining of Mineral Deposits. ISSN 2415-3443 (Online) | ISSN 2415-3435 (Print)

volume of cavities affects the change in the geodynamic and seismic conditions in the earth's crust. There is a hypothesis that dynamic processes in cavities cause catastrophic phenomena similar to the rock ice slide of the Kolka Glacier in the Karmadon Gorge of North Ossetia and others [14], [15].

Analysis of technogenic cavities indicates that with an increase in the depth of mining the ore deposits and the service life of the stopes, the number of uncontrolled rock caving in them increases. Underestimation of the specified factors can lead to the earth's surface failure over large areas, air blasts in underground mine workings, as well as to social strain of residents living in the zone influenced by mining operations [16]. The earth's surface failure on an area of 16 hectares at the Ordzhonikidze Mine in Kryvbas (Ukraine) in 2010 is a case in point. In such a way, the analysis results lead to a conclusion that an important issue for solving the problem is the formation of technogenic cavities that affect the emergence and redistribution of the rock mass SSS. Their occurrence in the earth's crust provokes the influence of geomechanical and seismic phenomena, up to the earthquakes [17]-[20].

The research purpose is to substantiate the mining-technical safety during underground mining of complex-structure ore deposits based on the study of a rock mass stress-strain state (SSS), as well as determining the safe parameters for specific mining-geological conditions in various environments.

In order to achieve the set purpose, the following objectives are solved:

1. To propose a functional interrelation between the rock mass stress value and the number of impulses (destruction sounds) per minute, which ensure stable outcrop parameters.
2. To provide an assessment and prediction for various forms of rock pressure manifestation based on the stress concentration coefficient and the value of relative deformation, which guarantee mining safety and stability of rock masses during blasting operations.
3. To develop a graphical-analytic method for determining the safe parameters of stopes when mining the complex-structure ore deposits.
4. To recommend the parameters for stoping mining systems with backfilling for underground mining of complex-structure ore deposits.

2. Research methods

When conducting research, the data of literary sources and patent documentation in the area of technologies and technical facilities for underground mining of complex-structure ore deposits, as well as substantiation of safe technological parameters of production blocks, laboratory and production experiments, physical modeling are used.

The qualitative measurement of the main stresses acting in the rocks and the mass stability is performed using the sound-ranging method, the physical basis of which is that the majority of rocks, when the constrained equilibrium of the stress field is disturbed, generate sounds that arise in the process of local micro-fractures during rock deformation under the influence of stress redistribution. In this case, the "sound activity" of the rock appears long before micro-fracturing and serves as a warning about the destruction process development or an indication of the relative, at the moment, mass stability. For making a prediction of failures, as well as to assess the rock mass stability using the sound-ranging method, the dependence of the fracture frequency in the rock mass on the stresses acting in it is studied. The sound frequency for specific rocks is determined experimentally. The practice of observations in the mines of the Kryvyi Rih Basin has revealed that a change in the sound vibration frequency in the range of 1-20 impulses per minute characterizes, in most cases, the destruction process development without external manifestations. The frequency, varying in the range of 20-40 impulses per minute, is accompanied by individual cleavage and local rockfalls, and frequency exceeding 40-60 impulses per minute precedes the bulk rock failure.

The sound rock activity in the studied mass is determined by alternately placing a sonic detector in specially drilled blast-holes (wells) and by recording the sound vibration frequencies for three periods of 5-10 minutes each. The sound-ranging method of the rock mass is as follows. Sounds received by a sensor located in a rock mass (in a blast-hole or well) are amplified and converted by an amplifier, and then transmitted to headphones and an oscillograph, which record the sounds of the emerging processes of rock destruction in the rock mass. The low-frequency amplifier, made on semiconductors, consists of three amplification stages and a bridge circuit with a microammeter included in the diagonal. The amplifier is powered by a single battery for the pocket torch. The sensor is a piezoelectric microphone, made in the form of a hollow metal cylinder, 30-32 mm in diameter and 110 mm long. Piezoelectric element is fixed by a cantilever inside the cylinder with a special holder.

The stability of the rock mass and pillars for various purposes is studied using sound-ranging and mine surveying instruments, acoustic strain gauges, depth and ground benchmarks, optical instruments, electrical circuits, visual and indirect methods of changing the mineralization of mine water. It has been revealed that a functional interrelation between the value of the rock mass principal stress and the number of impulses (destruction sounds) per minute, characterizing its structural (a) and strength (b) properties, is described by a curvilinear dependence of the type $y = ax^b$, which allows with a probability of 0.8, to quickly determine the optimal and stable outcrop parameters. With the help of the authors, analytical studies, comparative analysis of theoretical and practical results have been performed using standard and new methods.

3. Research results

3.1. Geomechanical stability of the outcrops

During the ore mining process, the state of rock masses is corrected by the relaxation phenomenon. Rock mass transfer into a guaranteed-stable or unstable state, as well as restricting the convergence of mine working abutments is ensured by optimizing the load-bearing capacity of technogenic structures, comparing the strength of technogenic processes with geological processes in natural conditions, and optimizing the safety factor [21]-[23]. Reducing the stress level to a safe value is ensured by engineering measures that contribute to rock displacement without discontinuity.

At complex-structure ore deposits, mined by the Skhidnyi Hirnycho-Zbahachuvalnyi Kombinat, DP (SE VostGOK), Ukraine, two mining systems are used. They differ in the bottom of the block and the equipment used at the draw of ore – through the drawpoints, and then by vibrating feeders such as PVG-1.2/3.1 or PVG-1.3/7.0. An end ore drawing is also used or through the ore drawing workings by load-haul-dump machines of the MPDN-1M type (with manual and remote control).

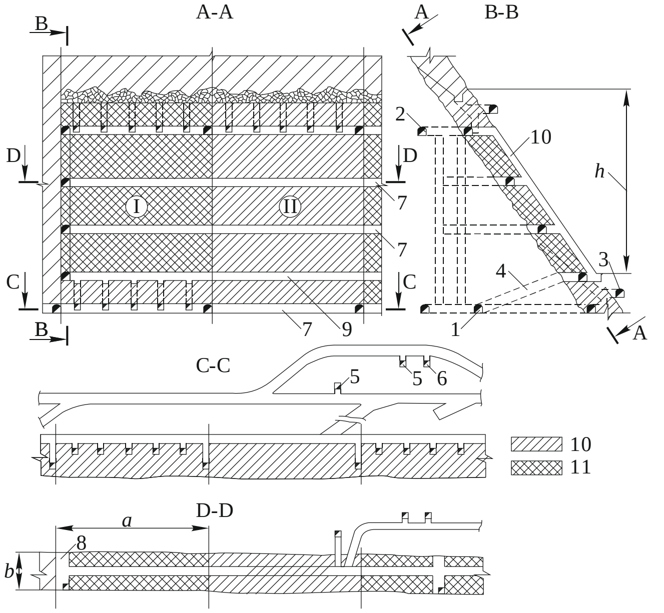

Figure 1. ***Sublevel stoping mining system with backfilling the mined-out area using cemented mixture; development workings: I – primary stopes; II – secondary stopes; a, b, h – length, width and height of the stopes; 1 – haulage drift; 2 – vent drift; 3 – collector; 4 – ramp; 5 – upraise block; 6 – ore pass; temporary working: 7 – sublevel drift; 8 – face cut; 9 – undercut drift; 10 – cut-out raise; 11 – reserves prepared for mining; 11 – reserves ready for mining***

The stopes are mined in sublevels of 10-15 m high. Ore blasting is performed with blastholes 57 and 65 mm in diameter, drilled with NT-2, PK-75 drilling machines and equipment of the UBSH-1GL, BU-85S, UBSH-201 type, developed by SS "Automation & Engineering" and manufactured by the Mechanical Repair Plant together with SE VostGOK (Ukraine). Parallel down-holes with a diameter of 85 and 105 mm are drilled with NKR-100M drilling rigs to form slots. As explosives, Gramonite 79/21, granulite AC-4, AC-4V, AC-8, AC-8V, rocky ammonite No. 1, etc. are used, which are charged by charging portable machines such as MZP-1, UZP-2A, UZP-3 and UTZ-2. Self-propelled load-haul-dump equipment from Atlas Copco Company (Boomer 281 drilling rig, ST 3.5 load-haul-dump machine), Tamrock Company (Minibur 1F drilling rig, TORO 151 load-haul-dump machine), etc. are also used.

To ensure the safety of mining operations in the zone influenced by cavities of the mined-out stopes, the following has been performed: predicting the rock mass SSS and assessing the conditions for the rock pressure dynamic manifestation; organizing a system for geomechanical and seismic monitoring of the rock mass SSS and the stable outcrops of the stopes [24], [25]. The main scientific and technical provisions for substantiating the safety parameters of the stopes should be developed on the basis of a long-term statistical base and using the results of mining similar ore reserves of the field, including:

– selecting a mining system with backfilling for individual deposits, taking into account their geomorphological and geomechanical peculiarities;

– determining safe geometrical parameters of production blocks, depending on the time of mining and spatial position in the enclosing mass, as well as on the structural state of the ore and enclosing masses, safe technological parameters of the cemented mixture for backfilling the mined-out area.

Moreover, a number of defining technological requirements should be taken into account:

– to the state of the surface above a mined-out deposit;

– mining-geological characteristics of deposits and conditions of their occurrence;

– geomechanical properties of ores and host rocks, as well as their stability;

– ensuring work safety and necessary sanitary-hygienic conditions;

– preventing excess losses and dilution of mineral resources;

– minimum production costs and maximum labor productivity in ore mining;

– ensuring the specified volumes of ore mining in terms of quantity and quality;

– possibility of averaging the quality of the ore mass during its processing.

Based on the results of multi-year comprehensive scientific research in the underground mining of complex-structure ore deposits in energetically disturbed masses, the authors propose a graphic-analytical method to substantiate the safe parameters of the stopes.

3.2. Graphic-analytical method for substantiating the rock mass outcrop and backfill stability

The calculation methodology can be shown using the example of substantiating the safe parameters of stopes with backfilling mined-out area with a cemented mixture of various composition and strength for complex-structure ore deposits.

Initial data: ore deposit dip – 60°; ore content – 0.85; degree of ore reserves exploration – C_1; ore body – from 24 to 38 m; ore body length is 120 m; according to the scale of Professor M.M. Protodiakonov, ores and host rocks are: hard, f = 16-18; fracturing of ore and host rock masses is medium; depth of mining is 420 m.

Solution: when mining an ore deposit across the strike, the safe geometrical parameters of the stopes are: the stope length (a, m), width (b, m) and height (h, m). The length of the stope (sublevel drift), as a rule, is equal to the horizontal thickness of the ore body, and its stability is estimated by the maximum permissible equivalent span of the horizontal outcrop according to the nomographic chart (Fig. 2).

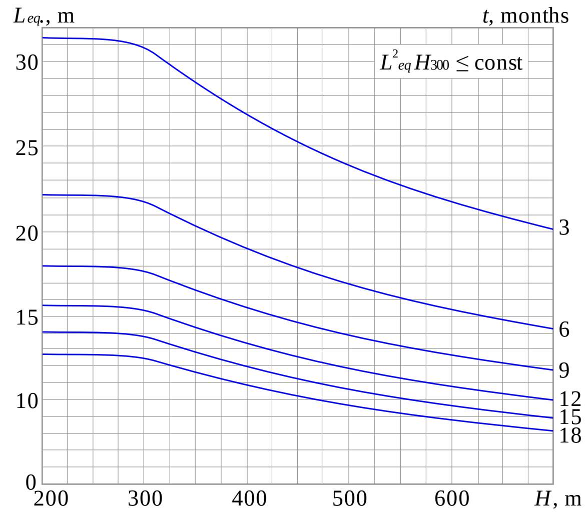

Figure 2. ***Nomographic chart for determining the maximum equivalent span of the outcrop in the edge area of the mass with medium fracturing ($L_{eq.}$, m) depending on the depth (H, m) and the lifetime (t, months) of outcrop***

Based on the stope lifetime, which is set by the local project for mining the production block (for example, 15 months), the depth of the stope from the surface (H = 420 m, the floor pillar level), the maximum permissible equivalent span can be found. When the stope width is 14 m, its length is determined from the nomographic chart, which is 45 m (Fig. 3).

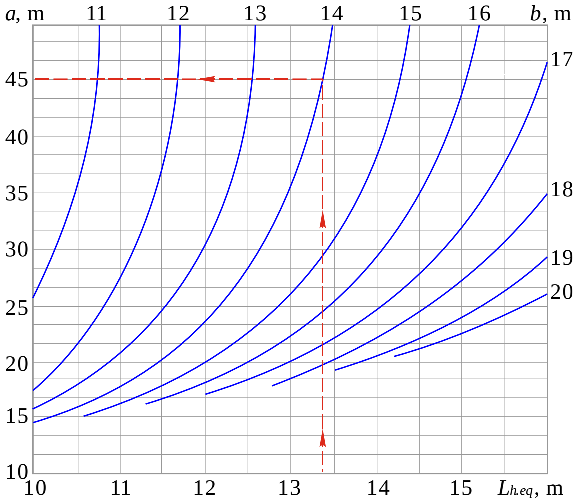

Figure 3. Nomographic chart for determining the stope length (a) depending on its width (b) and the maximum permissible equivalent span of horizontal outcrop (L^h_{eq}), m

The ore body thickness ranges from 24 to 38 m, which is lower than the calculated stope length according to the monographic chart. Consequently, there is a possibility of increasing the stope width to 14.4 m with a length of 38 m with this extremely stable equivalent span of the horizontal outcrop.

The stability of vertical outcrops with known parameters of the stopes located across the strike of the ore body through the maximum permissible equivalent span of vertical outcrop should be checked using the nomographic chart (Fig. 4), which with a stope length of 38 m is equal to $L_{v.eq}$ = 32 m, or by the Formula 1:

$$L_{v.eq} = \frac{h \cdot a}{\sqrt{h^2 + a^2}} = \frac{60 \cdot 38}{\sqrt{60^2 + 38^2}} = 32 \text{ m.} \quad (1)$$

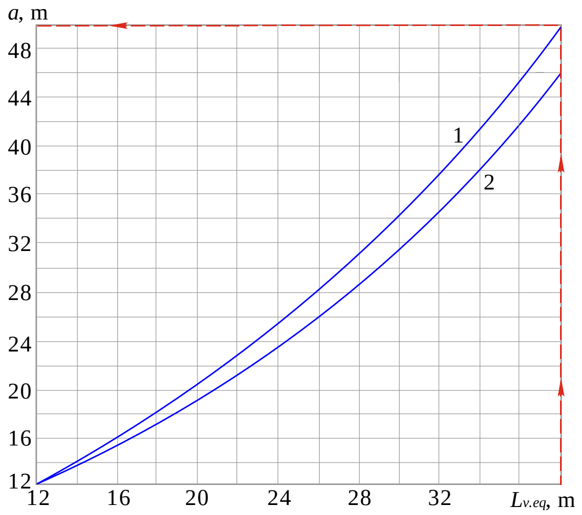

Figure 4. Nomographic chart for determining the stope length (a) depending on the maximum permissible equivalent span of vertical outcrop L^h_{eq}, m; 1, 2 – the stope height is 60 and 70 m respectively

In the future, the stope width can be adjusted with a decrease in the ore body thickness to 24 m (refer initial data), which is equal to 16 m Thus, it has been determined that the stope width varies depending on the ore body thickness from 16 to 14.4 m. This enables to place 8 production blocks along the strike (120 m) with a level height of 60 m [26], [27]. Systematized parameters of stope safety calculation are shown in Table 1.

Table 1. Safe parameters for stopes of complex-structure ore deposits

Parameter name	Calculation formula
Geomechanical safety of stope parameters	
Equivalent spans for outcropping, m (L_{eq})	
Regular-shaped	$L_{eq} = \dfrac{a \cdot \text{в}}{\sqrt{a^2 + \text{в}^2}}$
Irregular-shaped	$L_{eq} = \dfrac{2{,}5 \cdot S}{P_o}$
Outcrop stability criterion, m (L_{edop})	$L_{eq} = L_{edop} = \dfrac{L_{eo}}{1{,}1}$
Stable spans, m	
Horizontal ($L_{h.eq}$)	$L_{h.eq} = \dfrac{a \cdot \text{в}}{\sqrt{a^2 + \text{в}^2}}$
Vertical ($L_{v.eq}$)	$L_{v.eq} = \dfrac{a \cdot H}{\sqrt{a^2 + H^2}}$
Equivalent span (L_{eo}, m), taking into account the lifetime of the outcrops (t, month)	$L_{eo}^2 \cdot t = const = A$
Equivalent spans for backfilling, m	
Horizontal	$L_{h.eq} = \sqrt{\dfrac{2\sigma_{iz} \cdot h_{sl}}{\gamma_z \cdot K_z}}$
Vertical	$L_{v.eq} = \dfrac{2C_m}{\gamma_z \cdot K_z} ctg\left(45° + \dfrac{\rho}{2}\right)$
Stability of outcrops, m:	
Length (a)	$a = \dfrac{L_{h.eq} \cdot b}{\sqrt{b^2 + L_{h.eq}^2}}$
Width (b)	$b = \dfrac{L_{h.eq}^2 \cdot a}{\sqrt{a^2 + L_{h.eq}^2}}$
Height (H)	$H = \dfrac{L_{v.eq} \cdot a}{\sqrt{a^2 + L_{v.eq}^2}}$

Note: Designations in the formulas: S and P_o – outcrop area and perimeter, m^2 and m, respectively; A – constant, the value of which depends on the rock mass properties and is determined from experiment, units (varies from 26.8 thousand for a highly-fractured mass to 22.0 thousand for a weakly-fractured mass with its horizontal outcrop); σ_f – backfill ultimate bending strength, t/m^2; h_{lay} – thickness of the low continuous backfill layer (h_{lay} = 4 m is taken in calculations); γ_1 – reduced volumetric backfill weight, t/m^3; k_s – safety factor (k_s = 3 is taken in calculations); C_b, C_o – the adhesion coefficients of the backfill mass and ore mass, respectively, t/m^2; ρ – internal friction angle (ρ = 32º is taken in calculations).

The sequence of mining eight production blocks in a deposit with an average stope width of 15 m should be conducted in four stages, in two sections of four blocks, which can protect the cemented rockfill outcrops from the negative influence of seismic vibrations caused by blasting operations when mining the reserves in the production block.

With such a sequence of mining the deposits, only the fourth block of the 1st section is in contact with two vertical outcrops of the cemented backfill. With a different sequence, the numbers of vertical outcrops of the cemented rockfill on both sides of the mined block increases [28], [29].

The geometric parameters of the stopes, such as length (a), width (b) and height (H), as well as the values of the equivalent horizontal and vertical spans, ensuring the backfill mass stability, are found from the known dependences [30], [31]. In the primary stopes, the permissible equivalent span of outcrops depends on the strength of the host ore rocks and the depth of mining (Fig. 5).

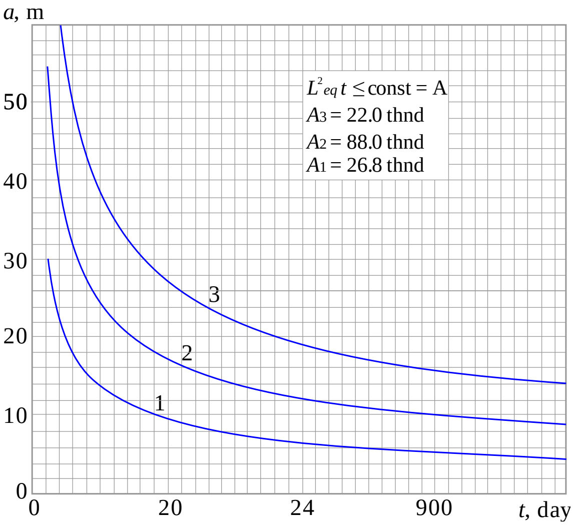

Figure 5.** **Nomographic chart for determining the maximum permissible equivalent span of the floor pillar outcropping for a rock mass: 1 – highly-fractured; 2 – medium-fractured; 3 – weakly-fractured

An equivalent horizontal outcrop span with a thickness of ore bodies up to 3 m and backfill compressive strength of more than 3.0 MPa can be of unlimited length.

The geometrical parameters for sublevel stoping mining system for ore deposits with a horizontal thickness of up to 15 m, when the length of the stope along the strike by 2-3 times or more exceeds the width (thickness), have been determined.

3.3. Strength characteristics of the cemented rockfill

Remote systems for sound-ranging control of stopes and the rock mass as a whole have revealed that the stresses redistributed in the mass occurs within 15 minutes after a massive explosion and then does not exceed 0-10 imp/min. Sound-ranging observations in 14 stopes out of 23 open stopes have confirmed the stable state of the mass (impulse intensity is 0-17 imp/min). It has been determined that with a stable size of the outcrops in the open stopes for 2-2.5 years, their further state remains stable. These stopes can be excluded from the list of stopes subject to sound-ranging control. By means of instrumental measurements using the sound-ranging devices, the degree of rock mass stability has also been determined at an impulse intensity of 0-3 imp/min. This is confirmed by a visual inspection of the outcrops in the open stopes of 10 production blocks at the levels of 280-520 m of the Smolinsk mine, SE VostGOK. The surveyed stopes and the rock mass as a whole are in a stable state, there is no danger of rock failures [32], [33]. The stability of horizontal and vertical backfill outcrops is directly dependent on the quality of the backfill mixture, hardening time and solidity (Fig. 6).

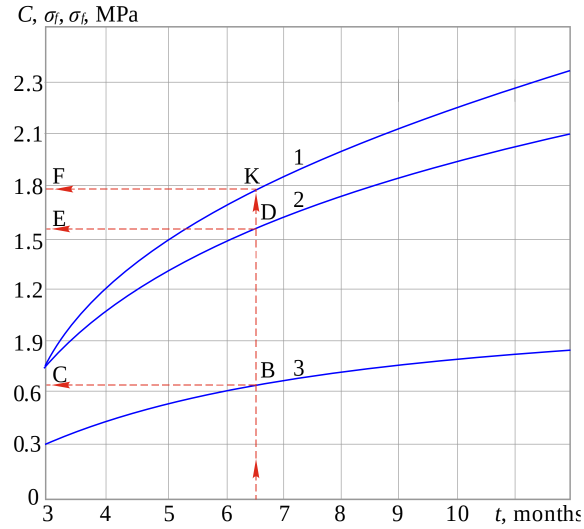

Figure 6.** **Strength characteristics of backfill depending on the hardening time (t): 1, 2, 3 – adhesion coefficient (C), ultimate bending strength (σ_f), ultimate tensile strength (σ_s), MPa respectively; key: A-B-C; A-D-E; A-K-F

The rock-backfill contact bending and tensile strengths are equal to the backfill strength in terms of these parameters. The internal friction angle (ρ) is taken to be 32°. The backfill solidity depends on the degree of the mixture stratification during the breaks when backfilling the stopes. A change in the ratio of the constituent components in the stratified part of the backfill reduces the calculated strength properties by more than 1/3.

When determining the maximum equivalent span of the horizontal outcrop, the height of the solid backfill layer, under which the underlying stopes directly located, must be at least 4 m. The stopes should be backfilled to this height without interruptions in the operation of the backfill complex, giving the backfill in its lower part a flat or arched shape. The surface of the broken ore before backfilling the stope should be leveled by regulating the ore drawing.

3.4. Providing the safe parameters for the cemented rockfill outcrops

According to the nomographic chart (Fig. 7) with the maximum permissible equivalent span of vertical outcrop, $L_{v.eq}$ = 32 m. Stable outcropping of the cemented rockfill of the backfilled primary stopes is possible when its compressive strength is ≈ 7.0 MPa, which is achieved after 8 months from the end of the stope filling.

To intensify operations when mining the sections of production blocks in the deposit, it is expedient to reduce the time required to achieve the standard compressive strength by using the required formulation, taking into account the mass of the cementitious material (granular slag – 400 kg/m^3 of the cemented mixture). To ensure the stability of horizontal outcrops of the cemented rockfill according to the maximum permissible equivalent span $L_{h.eq}$ = 13.4 m (Fig. 7), 7-7.5 months of hardening and 6 MPa compressive strength are sufficient.

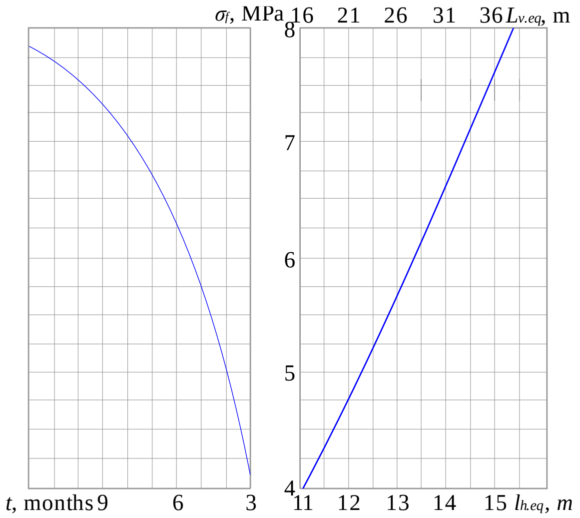

Figure 7. Nomographic chart for determining the period of outcropping the cemented rockfill (t, months) by its compressive strength (σ_{compr}, MPa) depending on the maximum permissible equivalent spans of the horizontal ($L_{v.eq}$) and vertical ($L_{v.eq}$) outcrops of the primary stopes, m

3.5. Implementation results

Various forms of the rock pressure manifestation are predicted according to the stress concentration coefficient within $0.91 < K_v < 0.70$. Based on the conditions of the elastic behavior of rocks, the relative deformation value $\varepsilon = 0.0002$-0.0003 is also taken, which ensures the mining-technical safety and stability of rock masses during repeated blasting operations (Table 2).

Table 2. Permissible relative rock mass deformations

Characteristics of the rock mass stress state	Permissible relative deformations, ε
Non-rock-bump hazardous ($K_v < 0.7$)	0.00030
Almost non-rock-bump hazardous, with a low probability of the rock-bump threat ($K_v \geq 0.71$-0.90)	0.00025
Prone to rock pressure dynamic manifestation, where the threat of rock bumps of various forms is possible (intense cutter break formation, exfoliation, spalling) ($K_v > 0.91$)	0.00020

Note: relative deformation ε is proportional to the compressive stress σ_{compr}

Based on the results of assessing the mass stress state level and the mechanism of rock pressure manifestation according to visual signs, the authors have developed a classification of the rock mass stress state, depending on the direction of the maximum stresses relative to the mine working location (Table 3). The methodology for substantiating the efficiency of mining operations is also extended to other complex-structure deposits with a stoping system of mining and blas-ting of reserves from sublevel drifts with cemented back-filling the mined-out area. In order to expand the area of basic standard distribution, research has updated the mining system parameters for mining-geological and mining-technical conditions of deposits at Stepnogorsk Mining and Chemical Plant LLP, Northern Kazakhstan [34], [35].

Table 3. Classification of the rock mass stress state

The point of disturbance manifested in the mine working	Form and intensity of the disturbance	Principal stress direction, σ_{max}	Mass stress state level, η	Category of rock-bump hazard
There are no disturbances along the entire perimeter	Mine working delineation along natural surfaces of weakening	Vertically	$\eta \leq 0.12$	Non-rock-bump hazardous
In the springs and in the walls from both sides	Exfoliation in depressions of the fracture joint tops of natural parting	Vertically	$0.12 < \eta \leq 0.2$	
	Local stratification throughout structural blocks		$0.2 < \eta \leq 0.3$	III
	Cleavage throughout mine workings, bulging. Dynamic cutter break formation, buckling of non-rock-bump hazardous rocks		$0.3 < \eta \leq 0.5$	II
	Rock bumps		$\eta > 0.5$	I
In the middle of the arch	Exfoliation in depressions of the fracture joint tops of natural parting	Horizontally	$0.12 < \eta \leq 0.2$	Non-rock-bump hazardous
	Local stratification throughout structural blocks		$0.2 < \eta \leq 0.3$	III
	Cleavage throughout mine workings, bulging. Dynamic cutter break formation, buckling of non-rock-bump hazardous rocks		$0.3 < \eta \leq 0.5$	II
	Rock bumps		$\eta > 0.5$	I
Displacement of the disturbance seat from the arch keystone	Exfoliation in depressions of the fracture joint tops of natural parting	At a tangent to the surface of disturbance	$0.12 < \eta \leq 0.2$	Non-rock-bump hazardous
	Local stratification throughout structural blocks		$0.2 < \eta \leq 0.3$	III
	Cleavage throughout mine workings, bulging. Dynamic cutter break formation, buckling of non-rock-bump hazardous rocks		$0.3 < \eta \leq 0.5$	II
	Rock bumps		$\eta > 0.5$	I

As a result of multi-year research, the modern technical level of the implemented mining system has been determined. In addition, its elements and technological processes have been standardized. This standard is "The sublevel stoping mining system with backfilling the mined-out area using a cemented mixture". The parameters and dimensions are regulated depending on mining-geological conditions. These include the length, width and height of the primary stopes, secondary stopes and third stages of stope mining, as well as their maximum permissible values, taking into account the change in the backfill strength properties during its hardening and the depth of mining, the sublevel height and the block bottom, the distance between the drawing load-haul-dump workings (drawpoints), depending on the ore body thickness and the level height, the shape and dimensions of the cross-sections of the main side workings [36], [37].

Within the framework of research work (No. GR0107U005388), the Instruction on the Geomechanical Substantiation of the Safe Mining the Reserves at the SE Directorate Mine has been developed, which represents the main provisions, geomechanical characteristic of ore deposits and determines the criteria for assessing the rock mass SSS. In addition, it describes the conditions for the safe conduct of mining operations, the procedure for determining the size of hazardous zones and brittle destruction of rocks prone to the dynamic manifestation of various rock pressure forms. It also presents the substantiation of the safe procedure for mining the ore deposits within the mine field and shows the safety of mining operations in adjacent ore bodies and ore deposits, as well as when mining the rock masses under the dynamic action of blasting. Measures are presented to safely conduct mining operations and reduce the dynamic manifestation of various rock pressure forms, including seismic-safe charge masses and their influence on the safe performance of mining operations when mining the ore deposits using stoping system with cemented backfilling the mined-out space at SE VostGOK mines, Ukraine [38], [39].

3.6. Advanced research directions

The authors note advanced research into predicting the vibration velocity of rock mass and cemented rockfill according to the reduced charge mass per deceleration stage during ore blasting for conditions of complex-structure deposits, which depend on the seismic-acoustic rock mass properties and blasting conditions, as well as the value of the permissible velocity of soil displacement at the base of the protected objects. Transition to combined technologies for mining complex-structure ore deposits using geotechnologies, as well as mining ore deposits in horizontal seams with a cemented backfilling. These technologies meet the requirements of efficient use of nature and resources by increasing the completeness of subsoil use (loss of 4-5%) and the quality of the extracted raw materials (dilution of 10-15%), as well as reducing the volume of waste rock dumped at the daylight surface and its processing at ore-processing plant, which has a positive effect on the hydrometallurgical process and the ecology of the mining region [40]-[44].

The rock mass structural disturbance by tectonic fractures, the lifetime of the outcrops, the gradient angle of the outcrop to the horizon, the sequence of block (stope) mining and the depth of mining operations have a decisive influence on the stability of the outcrops. Rock and ore masses of complex-structure deposits are classified according to the parameters of fracturing and blocky structure. An equivalent span is taken to assess the horizontal and vertical outcrop stability. The geometric parameters of the stopes and the vertical and horizontal outcrop stability depend on the intensity of mining and backfilling of the stopes. The more intensive the mining of the stope, the more outcrops can be expected. Mining-technical safety of mining the complex-structure deposits is achieved through the implementation of integrated methods and technical means for measuring the rock mass stress state on the basis of geomechanical monitoring [45]-[47].

Thus, the main negative impact of mining technology on the environment and humans is the high cost for preserving the daylight surface and ensuring the life activity of the population living in the zone of influence of mining facilities, as well as the allocation of large plots of land, especially fertile, etc. Therefore, it is necessary to provide funds for the following measures:

– subsurface processing of technogenic waste (tailings) distinguished by a wide variety of mineral forms in comparison with conventional ores;

– reclamation of the area of industrial sites and the adjacent territory after decommissioning;

– landscaping of the reclaimed area with grass and shrub vegetation;

– constant monitoring for environmental components, including hydrogeology, especially when implementing the geotechnological methods for leaching metals from ores in the subsoil and in the zone of influence of mining facilities.

4. Conclusions

1. A functional interrelation between the rock mass SSS value and the number of impulses (destruction sounds) per minute, characterizing its structural (a) and strength (b) properties, is proposed, which is described by a curvilinear dependence of the type $y = ax^b$ and allows with a probability of 0.8 to quickly determine the stable parameters of stopes.

2. Assessment and prediction are made for various forms of rock pressure manifestation, based on the stress concentration coefficient within $0.91 < K_v < 0.70$, taking into account the conditions of the elastic behavior of rocks. The value of $\varepsilon = 0.0002\text{-}0.0003$ is taken as permissible relative deformation, which ensures both mining-technical safety and the rock mass stability during repeated blasting operations.

3. A classification of the rock mass SSS has been developed depending on the direction of the maximum stresses relative to the mine working location, the level of the rock mass stress state and the mechanism of rock pressure manifestation (η), as well as the category of rock-bump hazard. In particular, the rock mass with the values in the range of $0.12 < \eta \leq 0.2$; $0.2 < \eta \leq 0.3$; $0.3 < \eta \leq 0.5$ and $\eta > 0.5$ are classified as non-rock-bump hazardous and belong to III, II, and I hazard categories, respectively.

4. To substantiate the safe parameters of stopes based on the results of multi-year research into underground mining of complex-structure deposits, depending on the rock mass properties, a graphical-analytic method (nomographic chart and calculation formulas) is recommended. These parameters are determined for highly fractured, medium fractured and weakly fractured rock mass with its horizontal outcropping to the surface. Using this method, the Instruction on the Geomechanical Substantiation of the Safe Mining the Reserves at the SE VostGOK (Ukraine) has been developed.

Acknowledgements

The creation, improvement and implementation of scientific developments are organized with the support and assistance of the following specialists: State Enterprise "Ukrainian Research and Design Institute for Industrial Technology" (SE Ukr R&D Institute for IndTech) and Skhidnyy Hirnycho-Zbahachuvalnyy Kombinat, DP (SE VostGOK), Zhovti Vody, Ukraine; Dnipro University of Technology, Dnipro, Ukraine; Institute of Geotechnical Mechanics named by N.

Poljakov of National Academy of Sciences of Ukraine, Dnipro, Ukraine; Ukrainian State Scientific and Research Mining Institute, Kryvyi Rih, Ukraine; Kryvyi Rih National University, Kryvyi Rih, Ukraine; Design & Survey and Research & Development Institute of Industrial Technology, JSC, Moscow and Research Institute of Mining Geomechanics and Markshading – Inter-Branch Scientific Center, JSC, St. Petersburg, Russian Federation.Stepnogorsk Mining and Chemical Plant, LLP, Nor-thern Kazakhstan.

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Обґрунтування гірничотехнічної та екологічної безпеки підземної розробки руд складноструктурного родовища

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Мета. Обґрунтування гірничотехнічної та екологічної безпеки підземної розробки руд складноструктурного родовища на основі дослідження напружено-деформованого стану (НДС) масиву гірських порід, а також визначення безпечних параметрів камер для конкретних гірничо-геологічних умов і закономірностей поширення гірського тиску в різних середовищах.

Методика. В роботі для оцінки підвищення гірничотехнічної та екологічної безпеки гірничих робіт використовувалися теоретичні узагальнення із застосуванням математичної статистики, фізичне та математичне моделювання, виконання розрахунків і техніко-економічних обґрунтувань, лабораторні й натурні експериментальні дослідження, промислові випробування в шахтних умовах і на земній поверхні у зоні впливу вибухів за стандартними і новим методиками.

Результати. Запропоновано функціональну взаємозв'язок між величиною НДС гірського масиву і кількістю імпульсів (звуків руйнування) у хвилину, що характеризують його структурні (a) і міцності (b) властивості, описується криволінійної залежністю виду $y = ax^b$, що дозволяє оперативно встановлювати стійкі параметри камер. Виконано оцінку та зроблено прогноз різних форм прояву гірського тиску за коефіцієнтом концентрації напружень у межах $0.91 < K_v < 0.70$ із урахуванням умов пружного поведінки гірських порід. Прийнято величину допустимої відносної деформації $\varepsilon = 0.0002$-0.0003, що забезпечує гірничотехнічну та екологічну безпеку і стійкість гірських масивів при багаторазовому виконанні вибухових робіт.

Наукова новизна. Розроблено класифікацію НДС гірського масиву в залежності від орієнтації максимальних напружень щодо вироблення, рівня напруженого стану масиву і механізму прояву гірського тиску (η), а також категорії ударонебезпеки. Зокрема, до неударонебезпечних, III-ї, II-ї та I-ї категорій небезпеки віднесені гірські масиви при значеннях у межах відповідно $0.12 < \eta \leq 0.2$; $0.2 < \eta \leq 0.3$; $0.3 < \eta \leq 0.5$ та $\eta > 0.5$

Практична значимість. Рекомендований графоаналітичний метод (номограми і розрахункові формули) щодо обґрунтування безпечних параметрів камер на підставі результатів багаторічних досліджень при підземній розробці родовищ складної структури залежно від властивостей гірського масиву і визначається для сильно-середньо- і слаботріщинуватого гірського масиву при горизонтальному його відслоненні. На основі методу складена "Інструкція щодо геомеханічного обґрунтування безпечного відпрацювання запасів на шахтах ДП "Схід ГЗК", Україна.

Ключові слова: *складноструктурні родовища, підземна розробка, геомеханіка, вибухові роботи, сейсміка, гірничотехнічна та екологічна безпека*