Mining of Mineral Deposits
Volume 20 (2026), Issue 2, 75-88

https://doi.org/10.33271/mining20.02.075

Geomechanical justification of combined support for temporary workings during rib pillar extraction under dynamic loading

Yerdulla Serdaliyev [1], Yerkin Iskakov [1*], Koptileu Shukirbayev [2], Zhandos Kenessov [1], Serik Amangeldi [1]

[1] *Satbayev University, Almaty, Kazakhstan*
[2] *JSC TNK Kazchrome, Aktobe, Kazakhstan*
**Corresponding author: e-mail y.iskakov@satbayev.university*

Abstract

Purpose. To justify the parameters of a combined support system for temporary mine workings providing access to rib pillars during their re-mining under conditions of a technogenically disturbed rock mass at the Zhezkazgan deposit, taking into account the dynamic impact of drilling and blasting operations.

Methods. The study was carried out using an integrated approach that included analytical assessment of blast wave propagation in a rock mass containing mined-out spaces, numerical modeling of the stress-strain state of the rock mass in Rocscience RS2 under different support options, and pilot-scale industrial tests with instrumental seismometric measurements using a Sigma 4+ station under the conditions of the East Zhezkazgan mine.

Findings. It was established that the rock mass in re-mining areas is characterized by a limit stress-strain state, with a factor of safety of 0.60-0.90 in the absence of support. A combined support system was substantiated, consisting of 1.8 m long GFRP rock bolts installed at a spacing of 1.0-1.2 m and a 0.05 m thick fiber-reinforced shotcrete lining, which provides a factor of safety greater than 1.0 while utilizing up to 85-88% of the bolt load-bearing capacity. Pilot-scale industrial tests established that the safe distance required to preserve support functionality during blasting of fan-pattern blastholes with a charge of up to 320 kg is at least 8-10 m from the blast source, at (PPV ≤ 15 mm/s. A power-law dependence was also established for the attenuation of peak particle velocity with distance from the blast source and explosive charge mass.

Originality. For the conditions of rib pillar re-mining at the Zhezkazgan deposit, the parameters of combined support for temporary workings were established for the first time with consideration of dynamic loading caused by blasting operations. A power-law dependence of seismic wave attenuation was substantiated, three characteristic zones of blast impact on the bond between the support and the rock mass were identified, and the safe distance required to preserve the load-bearing capacity of the support system was determined.

Practical implications. The practical value of the study lies in improving mining safety during rib pillar re-mining through the application of an economically feasible combined support system that ensures the stability of temporary workings under intensive dynamic impact from drilling and blasting operations and preserves the bond between the fiber-reinforced shotcrete lining and the rock mass.

Keywords: *rib pillars; temporary workings; combined support; fiber-reinforced shotcrete; GFRP rock bolts; numerical modeling; seismic monitoring*

1. Introduction

Long-term exploitation of large underground ore deposits is inevitably accompanied by the gradual depletion of the most accessible and high-grade mineral reserves concentrated in the central parts of ore fields [1], [2]. As mining operations advance to deeper horizons and peripheral areas of deposits, the proportion of difficult-to-recover and previously abandoned reserves increases substantially [3]. Under these conditions, one of the promising approaches to improving the efficiency of mineral resource utilization is the re-mining of reserves concentrated in technological and protective pillars formed during the primary extraction of ore bodies [4]-[7]. When selecting such technological solutions, particular importance should be attached to justifying the efficiency of underground mining, ensuring the rational use of mineral resources, and maintaining the environmental safety of mining operations [8], [9]. In addition, modern mining systems increasingly require the use of algorithmic optimization approaches to improve the energy efficiency and operational reliability of auxiliary equipment and technological processes [10].

Over a prolonged period of operation, the use of room-and-pillar mining systems has resulted in a considerable amount of ore reserves being left underground, primarily within rib pillars and barrier pillars [11]. This issue is of

Received: 6 January 2026. Accepted: 30 May 2026. Available online: 30 June 2026

particular relevance for large underground copper ore deposits in Kazakhstan, especially the Zhezkazgan ore district, where underground mining has been carried out for several decades. According to expert estimates, the total volume of ore written off as losses over the entire period of operation of the Zhezkazgan deposit exceeds 100 million tonnes, with approximately 40% of these reserves concentrated specifically in numerous columnar rib pillars distributed across previously mined-out areas of the deposit [12], [13].

The presence of substantial volumes of residual reserves makes their involvement in re-mining necessary [14]. In this context, the extraction of rib pillars is considered not only as a reserve for increasing the mineral recovery ratio, but also as one of the means of normalizing the stress-strain state of the rock mass in areas of long-operated mines. The long-term preservation of pillars under conditions of high rock pressure and extensive mined-out voids leads to the gradual accumulation of damage within the rock mass. Under the influence of static loads, repeated blasting impacts, and long-term deformation processes, zones of failure and intense fracturing are formed in the surrounding rocks [15]-[17].

As a result of these processes, the strength characteristics of pillars decrease over time, which may lead to their partial or complete failure. This, in turn, is accompanied by spontaneous collapse of the roof and overlying strata into the mined-out space. Such phenomena pose a serious threat both to the safety of mining operations and to the stability of the ground surface above previously mined areas of deposits. Therefore, controlled extraction of rib pillars followed by managed roof caving is considered one of the effective methods for relieving the rock mass from accumulated elastic energy and stabilizing the geomechanical conditions.

The geomechanical conditions for mining ore bodies in areas with extensive mined-out spaces are characterized by a complex and non-uniform distribution of stresses within the rock mass. In the vicinity of previously mined stopes, stress redistribution zones are formed, accompanied by the concentration of abutment pressure on the remaining pillars and marginal parts of the ore bodies. Numerous studies indicate that the stability of such pillars is governed by the combined influence of the mechanical properties of ores and host rocks, the geometric parameters of the pillars, and the spatial configuration of previously formed mined-out spaces [18], [19]. A significant influence is also exerted by the ratio of pillar height to width, which determines its load-bearing capacity and stability under long-term exposure to rock pressure.

Over time, the rock mass in the areas of mined-out stopes is subjected to additional technogenic impacts associated with drilling and blasting operations carried out during primary ore extraction [20]. Repeated dynamic loading caused by these impacts on the rock mass promotes the development of microfracturing and a reduction in rock mass stiffness [21]. As a result, zones of weakened rock mass are formed around mined-out spaces, characterized by reduced strength and increased deformability of the rocks [22]. The presence of such zones substantially complicates the geomechanical conditions for the re-mining of rib pillars [23].

The technology for extracting rib pillars generally involves the development of special mine development workings that provide access to the pillar mass and enable drilling and blasting operations [24], [25]. The most common approach is the driving of field development drifts that connect active mine workings with the areas where the pillars are located [26], [27]. From these workings, drilling-and-loading drifts are formed, which are intended for the placement of drilling equipment, as well as for organizing the drawing and transportation of broken ore. The general conceptual scheme for rib pillar extraction through a system of field workings is shown in Figure 1.

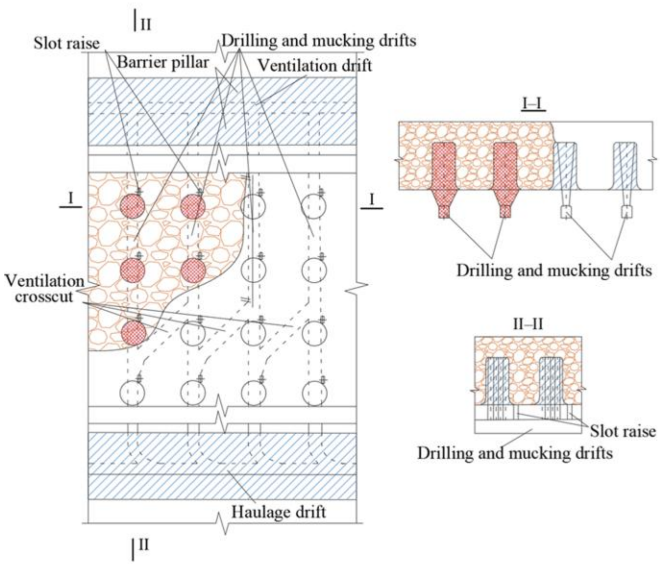


Figure 1. Conceptual scheme of rib pillar extraction

As follows from the presented scheme, the rib pillar is located within a rock mass surrounded by previously mined stopes. On several sides, the pillar is bounded by mined-out spaces formed as a result of primary ore extraction. Access to the pillar is provided through a system of field development workings, including drilling-and-loading drifts and ventilation drifts. The drilling-and-loading drifts are used for placing drilling equipment and carrying out drilling and blasting operations, while the ventilation workings provide the required air exchange and removal of blasting fumes from the face area.

Direct fragmentation of the pillar mass is generally carried out using a fan-pattern blasthole drilling scheme. The efficiency and safety of drilling operations are largely determined by the technical solutions adopted and the selected drilling parameters [28]-[32]. Drilling and blasting parameters have a significant effect on the efficiency of rock mass fragmentation and the level of dynamic impact on the surrounding mine workings [33], [34].

According to the technology, fan-shaped blastholes are drilled from the drilling-and-loading drift into the pillar mass, after which explosive charges are placed in them. Sequential blasting results in the fragmentation and breakage of the pillar mass, followed by the drawing of broken ore through the drilling-and-loading workings. The schematic arrangement of fan-pattern blastholes during rib pillar extraction is shown in Figure 2. The use of fan-pattern drilling schemes makes it possible to ensure efficient fragmentation of the pillar mass and improve the technological efficiency of stoping operations [35]. However, the presence of extensive mined-out spaces around the pillar has a considerable influence on the propagation of stresses and dynamic effects generated during blasting. During detonation of explosive charges, elastic waves propagate through the rock mass and interact with the boundaries of voids formed by previously mined stopes [36].

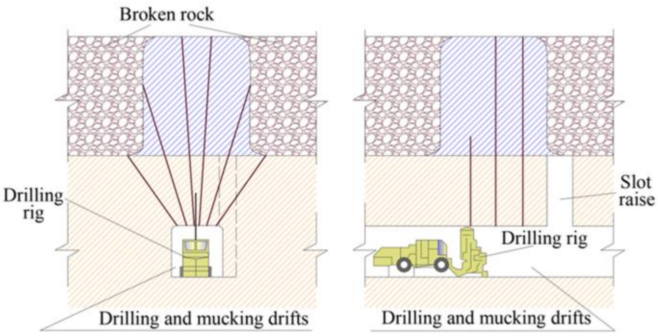


Figure 2. Fan-pattern blasthole drilling scheme for rib pillar extraction

As a result, reflection and interference of stress waves occur, which may lead to the formation of local zones of increased dynamic loading.

Particularly complex conditions are observed when pillar extraction is carried out beneath caving zones or within subsidence troughs formed as a result of long-term deposit exploitation. In such areas, the rock mass has already undergone significant deformation, manifested in the development of fracturing, the formation of local caving arches, and stress redistribution. Stable natural arches are often formed in the roof of mined-out spaces, bounded by relatively strong sandstone layers that act as specific "bridges" between sections of the rock mass [37]. At the same time, the adjacent rocks may be in a state of limit equilibrium and may exhibit increased sensitivity to additional dynamic impacts.

An additional challenge in rib pillar extraction is the need to drive temporary mine workings designed to provide access to the pillar mass and accommodate drilling equipment. Unlike permanent mine workings, the service life of such openings is generally limited and may amount to only several months. Despite their temporary use, these workings must remain stable throughout the entire period of drilling and blasting operations and ore drawing. At the same time, they are located in close proximity to the blasting zone and are subjected to dynamic loads intensified by the influence of the surrounding mined-out spaces. The interaction of blast waves with void boundaries may lead to the formation of tensile stress zones in the roof and sidewalls of the workings, the development of fracturing, weakening of the rock mass, and increased deformation in the near-contour zone [38], [39].

Ensuring the stability of temporary mine workings under these conditions is a complex engineering and geomechanical task. The use of massive and material-intensive support systems applied in permanent mine workings is often economically unjustified in this case due to the short service life of such workings. At the same time, an insufficient degree of support may lead to intense deformation of the roof and sidewalls, creating a threat to mining safety and potentially causing forced equipment downtime [40], [41].

The issue of mine working stability during the re-mining of rib pillars has been repeatedly addressed in studies devoted to the geomechanics of underground mining and rock mass condition control. A number of studies focus on the load-bearing capacity of rib pillars, the patterns of stress redistribution in the rock mass, and the conditions for caving initiation in mined-out spaces [42], [43]. Other studies examine the optimization of drilling and blasting parameters during the extraction of residual ore reserves and the control of rock mass failure processes during stoping operations [44], [45].

However, analysis of the existing scientific literature shows that researchers have focused primarily on the stability of the rib pillars themselves and on the technological parameters of their extraction. Considerably less attention has been paid to the stability of temporary mine workings driven to provide access to the pillar mass. Meanwhile, it is precisely these workings that are located under the most unfavorable geomechanical conditions, since they are situated in close proximity to blasting zones and, at the same time, within a rock mass weakened by the presence of extensive mined-out spaces.

An additional complication is that the rock mass in re-mining areas is characterized by technogenic disturbance formed as a result of the long-term impact of mining operations [46]. The presence of mined-out stopes, caving zones, and subsidence troughs leads to the formation of complex stress and deformation fields that differ significantly from the conditions of primary ore body extraction. Under such conditions, the propagation of dynamic waves generated by blasting operations is accompanied by their reflection from void boundaries, interference, and stress concentration in individual zones of the rock mass [47].

Previous studies indicate that, under re-mining conditions, the depth of failure and loosening zones in the roof and sidewalls of mine workings may reach 1.0-1.5 m or more [48]. This results in spalling and rock falls along the excavation contour, which significantly complicates stoping operations and increases the risk of emergency situations. Particularly unfavorable conditions are observed in areas where temporary workings are located in close proximity to mined-out spaces and are simultaneously exposed to repeated dynamic loading caused by drilling and blasting operations.

Under such conditions, traditional approaches to ensuring the stability of mine workings, based on the use of massive and material-intensive support systems, are not always rational. This is due to the fact that temporary workings used to access rib pillars have a limited service life and are often operated for a relatively short period of time. Therefore, when selecting support parameters, it is necessary to consider not only safety requirements, but also the technological and economic feasibility of the adopted solutions.

Thus, ensuring the stability of temporary mine workings during the re-mining of rib pillars under conditions of a technogenically disturbed rock mass and intensive blasting-induced dynamic impact represents a relevant scientific and technical challenge. Solving this problem requires a comprehensive investigation of the geomechanical processes occurring within the rock mass, as well as the development of rational engineering solutions for maintaining the stability of mine workings.

In this regard, the aim of the present study is to provide a geomechanical justification for the parameters of combined support systems for temporary mine workings that ensure safe access to rib pillars during their extraction under conditions of a technogenically disturbed rock mass and dynamic impact from drilling and blasting operations. The study examines the specific features of the stress-strain state of the technogenically altered rock mass, analyzes the influence of drilling and blasting operations on the stability of temporary workings, and substantiates rational parameters for their support.

2. Methods

The research methodology was developed with consideration of the need for a comprehensive assessment of the stability of temporary mine workings driven to provide access to rib pillars in a technogenically disturbed rock mass. The main factors determining the stress-strain state of the rock mass included the geometry of previously formed mined-out spaces, the parameters of rib pillars, the depth of mining operations, the physical and mechanical properties of ores and host rocks, and the dynamic impact of drilling and blasting operations.

The study comprised three consecutive stages. At the first stage, an analytical assessment was performed of the propagation of dynamic stresses from blasthole charges, taking into account wave attenuation and their reflection from the boundaries of mined-out stopes. At the second stage, numerical modeling of the stress-strain state of the rock mass was carried out in the Rocscience RS2 software package for various computational schemes and support options for the temporary working. At the third stage, the calculation results were compared with the conditions of pilot-scale industrial tests at the East Zhezkazgan mine, which made it possible to assess the applicability of the proposed combined support system under real mining and geological conditions.

To analyze the mechanism of stress propagation in a rib pillar during detonation of blasthole charges, a computational scheme was considered that describes the interaction of blast waves with the rock mass and the boundaries of mined-out stopes. In this scheme, the source of dynamic impact is represented by an explosive charge located in the central part of the rib pillar. Characteristic rock mass failure zones are formed around the charge: a crushing zone, a fracture formation zone, and an elastic deformation zone [49]. As the distance from the blast source increases, stresses in the rock mass decrease due to geometric energy dispersion and internal wave attenuation in the rock medium. A schematic representation of stress propagation and the formation of failure zones in a rib pillar during detonation of a blasthole charge is shown in Figure 3.

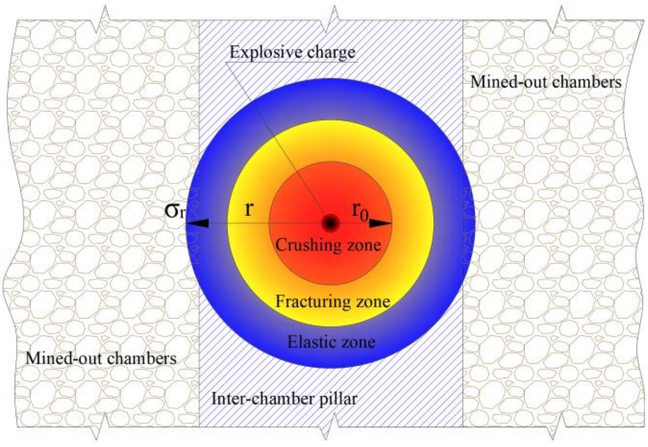


Figure 3. Schematic representation of stress propagation and failure zone formation in a rib pillar during detonation of a blasthole charge

Based on the presented scheme of stress propagation in the rock mass, the relationship describing the variation in radial stresses with distance from the source of dynamic impact can be expressed as follows:

$$\sigma_r = \sigma_0 \left(\frac{r_0}{r} \right)^n , \tag{1}$$

where:

σ_r – stress in the rock mass at a distance r from the source of impact, MPa;

σ_0 – stress near the loading source, MPa;

r_0 – radius of the initial impact zone, that is, the crushing zone radius, m;

r – distance from the charge center to the considered point of the rock mass, m;

n – dimensionless stress attenuation coefficient.

It follows from this expression that the stress intensity in the rock mass decreases rapidly with increasing distance from the loading source. However, in the presence of mined-out spaces, the pattern of stress propagation becomes considerably more complex due to the reflection and interference of elastic waves at void boundaries. In the case of rib pillars, mined-out stopes are located on both sides of the pillar mass and act as free surfaces. When the wave front reaches such surfaces, stress reflection occurs together with a change in stress sign, which may lead to the formation of an interference pattern of stresses.

Taking into account the dynamic nature of wave propagation, a more general expression for radial stress can be written as:

$$\sigma_r = (r,t) = \sigma_0 \left(\frac{r_0}{r} \right)^n e^{-ar} \cos(\omega t - kr), \tag{2}$$

where:

$\sigma_r (r, t)$ – radial stress in the rock mass at a distance r from the charge center at time t, MPa;

α – attenuation coefficient of elastic waves in the rock mass, m^{-1};

ω – angular frequency of oscillations, s^{-1};

k – wave number, m^{-1};

t – ime after charge detonation, s.

For the mining and geological conditions of the East Zhezkazgan mine, the following parameter values were adopted: attenuation coefficient $\alpha = 0.05$ m^{-1}, angular frequency $\omega = 400$ rad/s, and wave number $k = 0.11$ m^{-1}, which corresponds to a longitudinal wave propagation velocity of $C_p \approx 3500$ m/s.

For a qualitative assessment of the stress wave interference pattern at the boundaries of mined-out spaces, the reflection condition for a free surface was considered. For typical mining and geological conditions of the Zhezkazgan deposit, the reflection coefficient R was assumed to range from 0.6 to 0.8, which is characteristic of a rock-air boundary in the presence of mined-out stopes. As waves propagate from a blasthole charge toward the boundaries of mined-out stopes, wave reflection occurs. For a free surface, the reflection condition can be expressed as:

$$\sigma_r^{ref} = -R\sigma_r^{inc} , \tag{3}$$

where:

σ_r^{ref} – reflected stress, MPa;

σ_r^{inc} – incident stress, MPa;

R – dimensionless wave reflection coefficient from the free surface.

Taking into account reflections from two opposite mined-out stopes, the total stress in the rib pillar can be represented as the superposition of incident and reflected waves:

$$\sigma_r^{tot}(r,t)=\sigma_r^{inc}(r,t)+\sum_{i=1}^{N}R_i\sigma_r^{inc}(r_i,t-\Delta t_i), \tag{4}$$

where:

σ_r^{tot} – total stress in the rock mass, MPa;

R_i – reflection coefficient at the i-th boundary of the mined-out space;

r_i – distance to the reflecting surface, m;

Δt_i – travel time of the reflected wave, s.

To describe the stress state of the rock mass near an underground opening, the equilibrium condition in a cylindrical coordinate system was used, which for an elastic medium can be expressed as:

$$\sigma_\theta=\sigma_h+\sigma_v-2(\sigma_h-\sigma_v)\cos 2\theta, \tag{5}$$

where:

σ_θ – tangential stress at the excavation contour, MPa;

σ_h – horizontal stress in the rock mass, MPa;

σ_v – vertical stress in the rock mass, MPa;

θ – angle relative to the direction of the principal stresses, rad.

In the presence of mined-out spaces, the values of the principal stresses may differ substantially from the initial geostatic values, resulting in significant shear stresses along the excavation contour.

The Mohr-Coulomb strength criterion was used as the rock failure criterion under the considered conditions:

$$\tau=c+\sigma_n\tan\varphi, \tag{6}$$

where:

τ – shear stress, MPa;

c – rock cohesion, MPa;

σ_n – normal stress, MPa;

φ – internal friction angle of the rocks, °.

When shear stresses exceed the rock strength limit, zones of plastic deformation develop, accompanied by fracturing and possible roof and sidewall falls in the working. Under conditions of rib pillar re-mining, these processes may be intensified by the dynamic impact of blasting operations.

The stability coefficient of the rock mass was used to assess the stability of the working:

$$K_s=\frac{\sigma_c}{\sigma_{\max}}, \tag{7}$$

where:

K_s – rock mass stability coefficient;

σ_c – uniaxial compressive strength of the rock mass, MPa;

$\sigma_{\max}$ – maximum acting stress, MPa.

If $K_s > 1$, the rock mass is considered stable. At $K_s < 1$ there is a probability of rock failure.

Under conditions of rib pillar extraction, the dynamic impact of drilling and blasting operations exerts an additional influence [50]. During detonation of explosive charges, a shock wave is generated in the rock mass. The detonation pressure can be estimated using the Chapman-Jouguet equation [51]:

$$P_{CJ}=\frac{\rho_0 D^2}{\gamma+1}, \tag{8}$$

where:

P_{CJ} – detonation pressure, Pa;

ρ_0 – explosive density, kg/m^3;

D – detonation velocity, m/s;

γ – adiabatic exponent of the detonation products; for the ammonium nitrate-based explosives used in this study, $\gamma = 3.0$.

The pressure on the blasthole wall is determined by:

$$P_0=\eta P_{CJ}, \tag{9}$$

where:

P_0 – pressure at the blasthole boundary, Pa;

η – coefficient of energy transfer from the blast into the rock mass.

The radii of failure zones around the blasthole can be estimated using the following relationships:

$$r_c=r_b\left(\frac{P_0}{\sigma_c}\right)^{1/2}; \tag{10}$$

$$r_f=r_b\left(\frac{P_0}{\sigma_t}\right)^{1/3}, \tag{11}$$

where:

r_c – radius of the crushing zone, m;

r_f – radius of the fracture formation zone, m;

r_b – blasthole radius, m;

σ_t – tensile strength of the rocks, MPa.

For the conditions of the East Zhezkazgan mine, with a blasthole diameter of 76 mm and a pressure on the blasthole wall of $P_0 = 1{,}8$ GPa, the radius of the crushing zone is $r_c \approx 0{,}19$ m, while the radius of the fracture formation zone is $r_f \approx 0{,}25$ m. In the presence of mined-out spaces, the interaction of shock waves with void boundaries is accompanied by wave reflection and local amplification in specific zones of the rock mass. This may lead to increased stresses in the roof and sidewalls of temporary workings and to an enlargement of the fracture formation zone in the near-contour area up to 1.0-1.5 m.

To assess the influence of mined-out spaces on rock mass stability, a computational scheme was developed that included a rib pillar, the surrounding mined-out stopes, and a temporary mine working. Numerical modeling was performed using the Rocscience RS2 software package.

Within the numerical modeling, two representative variants of mined-out space geometry were considered. In the first variant, a system of two stopes separated by a rib pillar was modeled. In the second variant, a computational scheme with an increased span of the mined-out space was considered, simulating the merging of adjacent stopes and the formation of a more extensive zone influenced by stoping operations.

Vertical stresses were determined as follows:

$$\sigma_v=\gamma H+p_t, \tag{12}$$

where:

γ – unit weight of the rocks, kN/m^3;

H – mining depth, m;

p_t – additional technogenic pressure, MPa.

Horizontal stresses were determined as:

$$\sigma_h=k\gamma H+\Delta\sigma, \tag{13}$$

where:

k – lateral pressure coefficient of the rock mass;

$\Delta\sigma$ – additional stress concentration caused by the presence of mined-out stopes, MPa.

For the conditions of the East Zhezkazgan mine, the following values were adopted: $\gamma = 27$ kN/m^3, $H = 320$ m, and $k = 0.35$. The additional technogenic pressure $p_t = 2.5$ MPa was determined from the results of numerical modeling, taking into account the presence of mined-out spaces. Stress concentration near the marginal parts of the stopes amounts to $\Delta\sigma = 8$-12 MPa, which corresponds to typical values for the mining systems used at the Zhezkazgan deposit.

To assess the stability of the development working, several computational variants were considered: an unsupported working; a working with a 0.05 m thick fiber-reinforced shotcrete lining; a working with a 0.10 m thick fiber-reinforced shotcrete lining; and a working with a combined support system consisting of rock bolts and a fiber-reinforced shotcrete lining.

In the computational model, the combined support system included 1.8 m long GFRP rock bolts and a 0.05 m thick fiber-reinforced shotcrete lining. The bolts were installed at a spacing of 1.0-1.2 m along the excavation perimeter, with a total of nine bolts in the cross-section. The fiber-reinforced shotcrete lining was modeled as a surface reinforcement layer providing stabilization of the near-surface rock mass layers and stress redistribution in the near-contour zone. The adopted support parameters correspond to typical solutions used for driving workings in fractured rocks of the Zhezkazgan region. The supported variants were modeled using the same geometric schemes of mined-out spaces as the unsupported variant: a scheme with two stopes separated by a rib pillar and a scheme with an increased span of the mined-out space. This ensured a correct comparison of the stress-strain state of the rock mass under different support options for the temporary working.

The selection of materials for the combined support system was determined by the specific conditions of rib pillar re-mining. GFRP rock bolts have several advantages over steel and friction-type bolts: during blast extraction of the pillar, they fail together with the rock mass without forming metallic fragments in the broken ore; they are chemically inert in the aggressive sulfate environment of the Zhezkazgan copper ores; and they provide a more uniform load distribution along the bolt length in a fractured rock mass due to their elastic modulus of 40-50 GPa. The use of fiber-reinforced shotcrete instead of conventional shotcrete is justified by the fact that shotcrete without dispersed reinforcement fails in a brittle manner under dynamic loading, with major cracks developing in the lining and causing its detachment from the rock surface. The combination of polymer and polypropylene fibers forms a spatial reinforcing framework that changes the failure mode from brittle to ductile, localizes crack formation, and preserves the integrity of the lining under repeated blasting impacts. The limited service life of temporary workings, which is typically 2-4 months, makes the use of massive support structures economically unjustified, whereas the proposed combined support system can provide the required level of stability with lower material consumption compared with massive support structures.

The mechanical interaction between the support system and the rock mass was taken into account in the model by introducing additional elements that simulated the performance of rock bolts and surface reinforcement. The bolt elements resisted tensile forces arising during rock mass deformation, while the fiber-reinforced shotcrete layer provided stress redistribution along the excavation contour and limited the development of fracturing.

The effectiveness of different support variants was assessed using the following parameters:

– stress distribution in the rock mass;
– displacement of the excavation contour;
– extent of plastic deformation zones;
– rock mass stability coefficient.

To verify the applicability of the proposed research methodology, the computational approaches were compared with the conditions of development working excavation at the operating East Zhezkazgan mine, mine No. 57, 320 m level, "Kresty 6-I" ore body, where access to rib pillars is provided through a system of development workings. Typical geometric parameters of the stopes are 10-15 m in width and 6-10 m in height, while the width of the rib pillars is 4-6 m. The computational model accounted for the actual geometric parameters of the stopes and temporary workings, the physical and mechanical properties of the host rocks, and the technological features of mining operations under conditions of already formed mined-out spaces.

Further verification of the applicability of the proposed combined support system was carried out under pilot-scale industrial testing conditions. The purpose of the tests was to evaluate the performance of the support system under real mining and geological conditions, as well as to verify the preservation of its load-bearing capacity and adhesion to the rock mass under dynamic loads generated by blasting fan-pattern blastholes during rib pillar extraction.

The tests were conducted in a drilling-and-loading drift at the 320 m level of the Annensk mine, located within the influence zone of previously mined-out spaces. The working, with a cross-section of 4.5 × 3.8 m, was driven through rocks with a Protodyakonov strength coefficient of $f = 8$-10. The geomechanical conditions of the test site were characterized by a disturbed rock mass, developed fracturing, and the presence of extensive mined-out stopes on both sides of the drilling-and-loading drift.

The working was supported according to the following technological sequence (Fig. 4). At the first stage, 20 mm diameter and 1.8 m long GFRP rock bolts were installed using resin capsule anchorage. Blast holes for bolt installation were drilled with an Epiroc drilling rig at an angle of 90° to the roof and sidewall surfaces. A total of nine bolts were installed in the cross-section of the working: five bolts in the roof at a spacing of 1.0-1.2 m and two bolts in each sidewall at a spacing of 1.2 m. After the bolts had been installed and a curing period of at least 2 h had elapsed to allow the anchoring compound to gain strength, the fiber-reinforced shotcrete layer was applied by spraying using a Normet Spraymec 1050 WPC unit. Spraying was carried out at a pressure of 5-6 atm, with a nozzle-to-surface distance of 1.0-1.2 m. The applied layer thickness was 0.05 m and was controlled using marker pins.

The fiber-reinforced concrete mixture consisted of M400 Portland cement, 5-10 mm aggregate, and water at a cement:aggregate:water ratio of 1:3:0.5. Glenium T803 plasticizer and MEYCO SA 162 hardening accelerator were used to improve the technological properties of the mixture. A combination of polymer fibers with dimensions $d \times l = 1.5 \times 30$ mm and a wavy profile, together with polypropylene fibers with dimensions $d \times l = 0.1 \times 12$ mm, was used as dispersed reinforcement.

(a)

(b)

(c)

Figure 4. Installation of the combined support system in the working: (a) installation of GFRP rock bolts; (b) preparation of the fiber-reinforced concrete mixture; (c) application of the fiber-reinforced shotcrete layer by spraying

To assess the stability of the support system under dynamic loading during blasting operations, seismometric measurements were performed directly in the mine working. Seismic impact parameters were recorded using a Sigma 4+ seismic monitoring station. Geophones were placed at several points in the working at different distances from the blast source (Fig. 5). The blasting parameters corresponded to production conditions, with the total explosive charge mass reaching up to 320 kg.

Figure 5. Seismometric measurements

Thus, the applied research methodology is based on a combination of analytical analysis of the stress-strain state of the rock mass, numerical modeling of different support options for mine workings, and pilot-scale industrial tests with instrumental seismometric measurements. This approach makes it possible to perform a comparative assessment of the effectiveness of the considered support systems and to verify the applicability of the proposed combined support system under the real mining and geological conditions of the East Zhezkazgan mine.

3. Results and discussion

Numerical modeling of the stress-strain state of the rock mass at the first stage was performed for the case of an unsupported working. This calculation was carried out to determine the initial geomechanical pattern of stress and deformation redistribution within the rock mass, as well as to identify potential zones of stability loss along the excavation contour. The model accounted for the combined influence of natural rock pressure and the additional dynamic load generated by blasting fan-pattern blastholes.

Within the numerical modeling, two representative variants of mined-out space geometry were considered. In the first variant, a system of two stopes separated by a rib pillar was modeled. In the second variant, a computational scheme with an increased span of the mined-out space was considered, simulating the merging of adjacent stopes and the formation of a more extensive zone affected by stoping operations.

The modeling results for the distribution of major principal stresses σ_1 showed that pronounced stress concentration zones are formed in the corner areas of the stopes and at the junctions between the mined-out space and the rib pillar mass (Fig. 6). In the first computational scheme, the maximum values of major principal stress reach 104.33 MPa, which is observed in the side corners of the stopes. These zones are the most heavily loaded elements of the system and are characterized by an increased probabi-lity of local failure development.

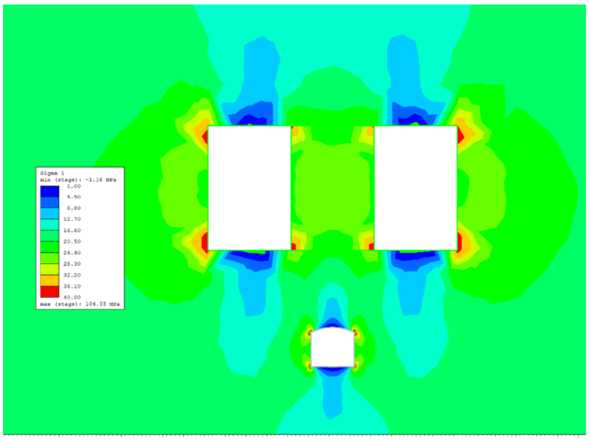
At the same time, a pronounced unloading zone is formed in the roof and sidewalls of the lower working, caused by stress redistribution resulting from the creation of the mined-out space. Such redistribution leads to the formation of a vertically oriented weakening zone extending from the stopes toward the lower working.

In the second computational scheme, the maximum stress values decrease to 85.10 MPa; however, the area affected by stress redistribution increases substantially. As a result, the unloaded zones extend over a larger portion of the rock mass, including the area above the lower working. This indicates the formation of a more extensive zone of geomechanical influence caused by the mined-out space.

Analysis of the distribution of the rock mass stability coefficient also confirms the unfavorable nature of the stress-strain state in the absence of support (Fig. 7). In the first computational scheme, zones where the stability coefficient decreases to 0.60 are observed near the contour of the lower working and in the corner areas of the stopes, which corresponds to a state close to the limit equilibrium condition. This indicates a high probability of sidewall and roof failure under further technological impacts.

In the second scheme, the stability coefficient values in the zone of the lower working range from 0.80 to 0.90; however, a significant part of the rock mass around the central pillar also remains in a state of reduced stability.

(a)



(b)

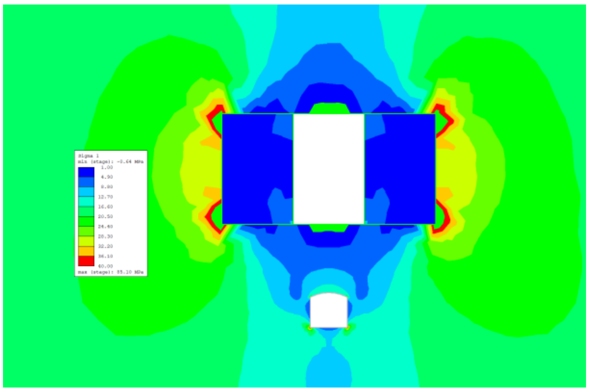


Figure 6. Distribution of major principal stresses σ_1 in the unsupported rock mass: (a) computational scheme of two stopes separated by a rib pillar; (b) computational scheme with an increased span of the mined-out space

This indicates stress redistribution onto the central pillar and the surrounding rock mass, which increases the likelihood of progressive deformation development.

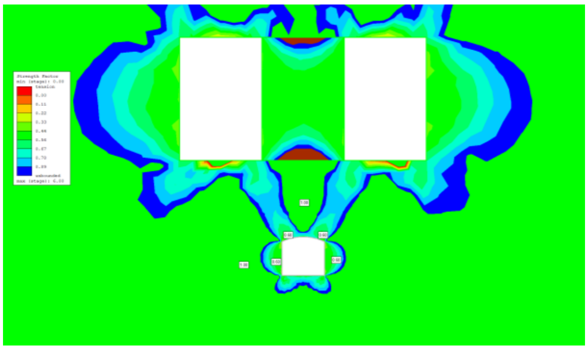
A generalized assessment of the modeling results shows that the overall factor of safety of the rock mass is about 0.80, while the residual strength reserve does not exceed 10% of the total rock mass strength. Under such conditions, the rock mass operates in a near-limit state, which creates a high risk of roof and sidewall failure in the working.

Thus, the numerical modeling results clearly indicate that driving the working without support under the considered mining and geological conditions cannot be regarded as a safe technological solution. The absence of support leads to the formation of zones with a limit stress-strain state and significantly increases the probability of excavation contour instability as shown in [52] and [53].

In view of these findings, which demonstrate unsatisfactory rock mass stability in the absence of support, the results of modeling for the working supported by the combined support system described in the Methods section were then analyzed. The purpose of this stage was to quantitatively assess the effect of support on stress redistribution, deformation reduction, and the increase in the rock mass stability coefficient under the combined action of natural rock pressure and dynamic loading generated by blasting fan-pattern blastholes.

The numerical modeling results showed that the application of support has a significant effect on the stress-strain state of the rock mass and makes it possible to improve excavation stability. Analysis of the distribution of major principal stresses σ_1 (Fig. 8) indicates a reduction in stress concentration levels in the near-contour zone and the formation of a more uniform stress field compared with the unsupported case. In the area of the lower working, a reduction in stresses is observed, indicating contour stabilization.

(a)



(b)

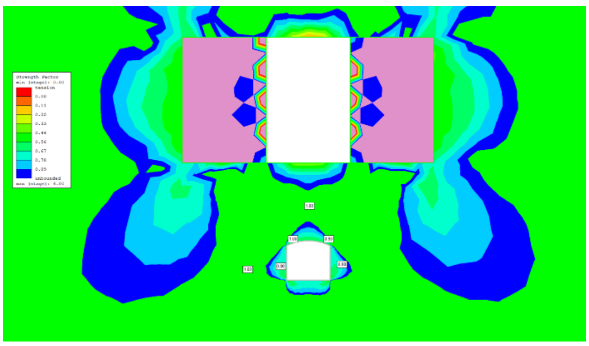
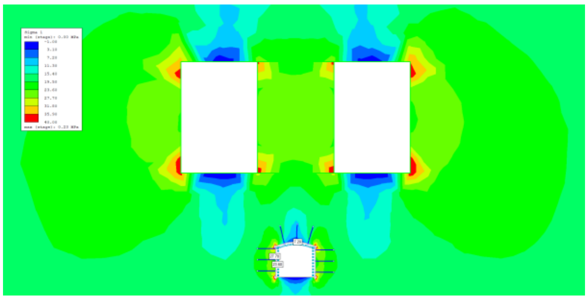


Figure 7. Distribution of the rock mass stability coefficient in the unsupported case: (a) computational scheme with two stopes; (b) scheme with a central pillar

(a)



(b)

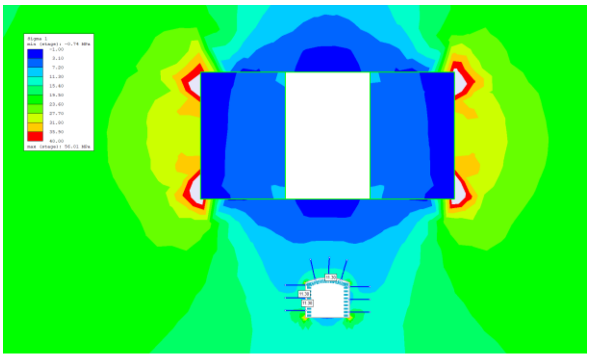
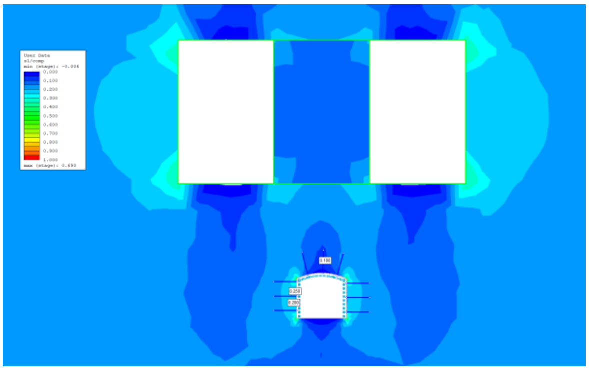


Figure 8. Distribution of major principal stresses σ_1 in the rock mass with the combined support system applied: (a) scheme with two stopes and a rib pillar; (b) scheme with an increased span of the mined-out space

Assessment of the rock mass stability coefficient (Fig. 9) confirms the positive effect of support, manifested in the reduction of limit-state zones and the increased stability of the roof and sidewalls of the working.

(a)



(b)

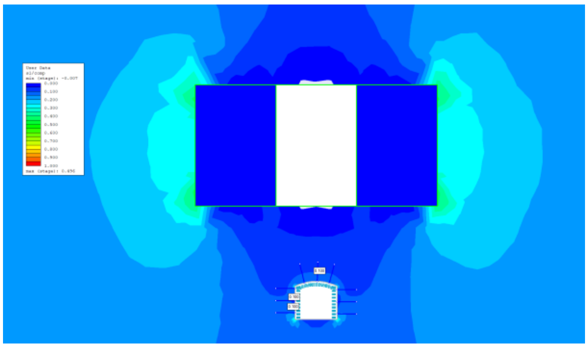


Figure 9. Distribution of the rock mass stability coefficient with the combined support system applied: (a) scheme with a rib pillar; (b) scheme with an increased span of the mined-out space

Analysis of the considered support options showed that the use of only a 0.05 m thick fiber-reinforced shotcrete lining does not provide the required load-bearing capacity and is associated with its overloading. Increasing the lining thickness to 0.10 m reduces the stress level and partially improves the operating conditions; however, it does not completely eliminate the zones in which the strength criteria are not satisfied.

A more detailed analysis of support element performance (Fig. 10) showed that the inclusion of rock bolts substantially changes the interaction pattern within the "support – rock mass" system. The calculations showed that axial forces in the most heavily loaded bolts reach 0.145-0.149 MN, with a bolt load-bearing capacity of 0.17 MN, which corresponds to the utilization of 85-88% of their strength capacity. At the same time, the factor of safety is 1.14-1.17, indicating that the bolting system remains serviceable despite the limited strength reserve of individual elements.

It was established that the combined application of rock bolting and a 0.05 m thick fiber-reinforced shotcrete lining provides load redistribution between the support elements, which ensures stabilization of the near-contour zone and satisfaction of the strength criteria in the rock mass, including under the action of dynamic loads generated by mass blasting of fan-pattern blastholes during rib pillar extraction.

Thus, the modeling results show that the required level of excavation stability is achieved not by increasing the thickness of the fiber-reinforced shotcrete lining, but by a rational combination of rock bolting and the fiber-reinforced shotcrete layer. The most effective option is the use of rock bolts together with a 0.05 m thick fiber-reinforced shotcrete lining, which ensures a stable state of the rock mass with minimal support design parameters, taking into account the combined action of static and dynamic loads.

For the purpose of quantitatively comparing the efficiency of the considered support options, a comparative analysis of the stress-strain parameters of the rock mass was carried out, the results of which are presented in Table 1.

It was established that the combined application of rock bolting and a 0.05 m thick fiber-reinforced shotcrete lining provides load redistribution between the support elements, which ensures stabilization of the near-contour zone and satisfaction of the strength criteria in the rock mass, including under the action of dynamic loads generated by mass blasting of fan-pattern blastholes during rib pillar extraction.

Thus, the modeling results show that the required level of excavation stability is achieved not by increasing the thickness of the fiber-reinforced shotcrete lining, but by a rational combination of rock bolting and the fiber-reinforced shotcrete layer. The most effective option is the use of rock bolts together with a 0.05 m thick fiber-reinforced shotcrete lining, which ensures a stable state of the rock mass with minimal support design parameters, taking into account the combined action of static and dynamic loads.

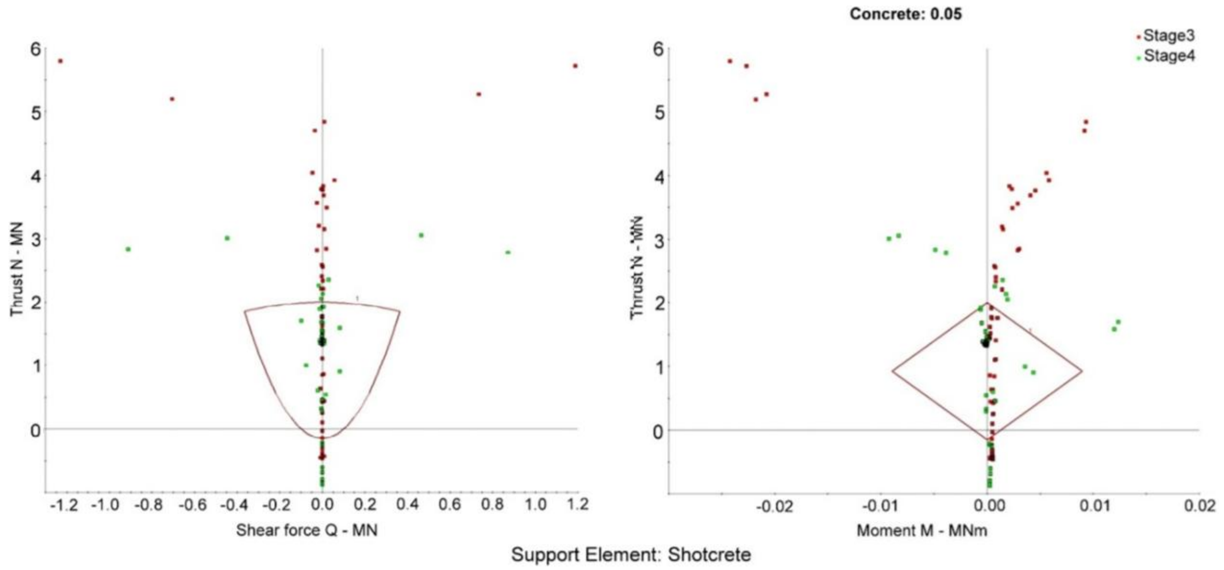
For the purpose of quantitatively comparing the efficiency of the considered support options, a comparative analysis of the stress-strain parameters of the rock mass was carried out, the results of which are presented in Table 1.

Comparison of the numerical modeling results (Table 1) shows that driving the working without support is accompanied by the formation of a limit stress-strain state in the rock mass. The use of only a fiber-reinforced shotcrete lining does not provide the required level of stability, even when its thickness is increased. At the same time, the combined support system, consisting of rock bolts and a 0.05 m thick fiber-reinforced shotcrete lining, ensures the transition of the rock mass to a stable state through load redistribution and restriction of deformations in the near-contour zone.

Based on the numerical modeling results, pilot-scale industrial tests of the proposed support system were carried out. The test conditions and seismometric measurement methodology are described in the Methods section. The results of instrumental measurements and the calculated attenuation curve are presented in Figure 11, while the measurement data are summarized in Table 2.

Analysis of the obtained results makes it possible to distinguish three characteristic zones of dynamic impact on the support system. In the zone of direct failure ($r \leq 3$ m), peak particle velocity ranged from 870 to 1780 mm/s, which corresponds to energy classes $K = 5.1$-5.8 according to the Zhezkazgan monitoring scale. Failure of support elements in this zone is technologically induced and is determined by the very nature of the pillar blasting process; therefore, it is not considered as a criterion for evaluating the proposed support system.

(a)



(b)

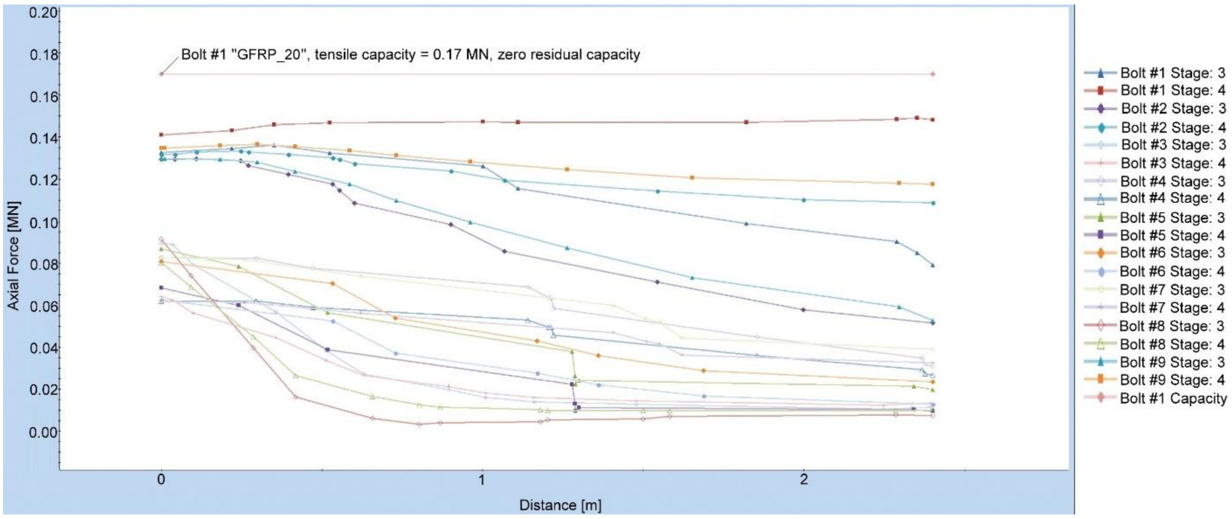


Figure 10. Performance assessment of the combined support system: (a) interaction diagram of stresses in the fiber-reinforced shotcrete lining; (b) distribution of axial forces in the rock bolts

Table 1. Comparative assessment of the stress-strain state of the rock mass under different support options

Calculation variant	Maximum major principal stress σ_1, MPa	Factor of safety	Residual strength reserve, %	Rock mass condition
Unsupported, scheme (a) – two stopes with a rib pillar	104.33	~0.80	~10	Limit state, unstable
Unsupported, scheme (b) – increased span	85.10	0.80-0.90	~10	Unstable
Fiber-reinforced shotcrete lining, 0.05 m	–	< 1.0	–	Overloaded, strength criteria not satisfied
Fiber-reinforced shotcrete lining, 0.10 m	–	~1.0	–	Partial improvement, local instability zones remain
Rock bolts + 0.05 m fiber-reinforced shotcrete lining	–	> 1.0	–	Stable condition, strength criteria satisfied

In the transition zone (r = 5-7 m) at PPV = 158-340 mm/s (K = 3.8-4.4; M = 1.0-1.3) isolated hairline cracks were observed in the fiber-reinforced shotcrete lining; however, the bond between the lining and the rock mass was fully preserved, and the load-bearing capacity of the support was not impaired. The use of a combined system of polymer and polypropylene fibers ensured localization of crack formation without loss of lining integrity (Fig. 12). This range of impact intensity is in good agreement with the levels of seismic activity recorded by the ISSI system at the Annensk mine during periods of active panel caving, with magnitudes of M = 1.0-1.5. In the safe zone ($r \geq$ 10 m), where PPV decreases to 72 mm/s or less ($K \leq 3.1$; $M \leq 0,6$), the support system exhibited no visible damage, and the bond between the fiber-reinforced shotcrete lining and the rock surface remained intact.

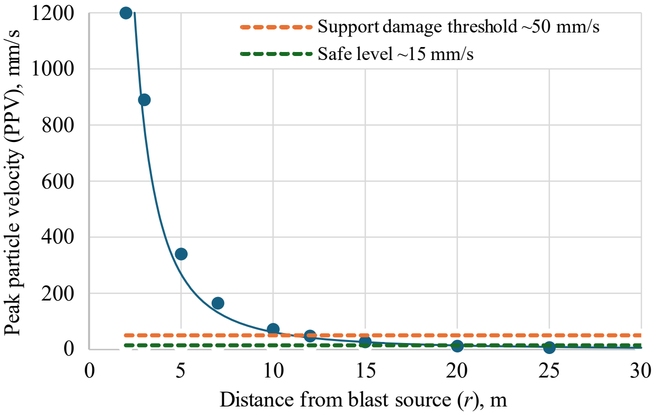


Figure 11. Attenuation of peak particle velocity (PPV) and dynamic pressure during blasting of fan-pattern blastholes (320 m level, Annensk mine, Q = 320 kg)

It should be noted that $K = 2$-3, corresponding to distances of $r = 10$-15 m, lies near the lower threshold of representative recording by the monitoring system at the Zhezkazgan mines ($K \geq 2.1$-2.8), which further confirms the validity of using the obtained data in the context of instrumental observations at this site.

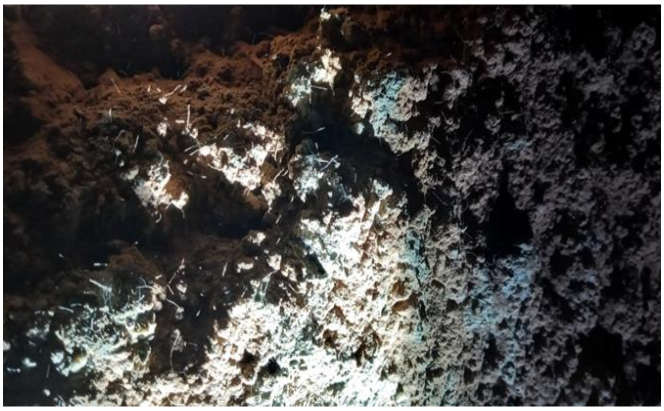


Figure 12. Condition of the fiber-reinforced shotcrete lining in the transition zone (r = 5-7 m) after blasting of fan-pattern blastholes: localization of crack formation without loss of bond to the rock mass

The calculated attenuation curve $PPV = 700 \cdot \left(Q^{1/3}/r\right)^{1.6}$, calibrated against seismological monitoring data from the Zhezkazgan deposit, provides a good fit to the experimental points over the entire measurement range.

Table 2. Results of seismometric measurements during blasting of fan-pattern blastholes

No.	r, m	PPV, mm/s	f, Hz	Energy class, K	Magnitude, M	Support condition
1	2	1820	80-120	5.8	2.1	Blasting zone; failure is technologically induced
2	3	890	70-100	5.1	1.7	
3	5	340	55-80	4.4	1.3	Isolated hairline cracks; bond preserved
4	7	165	45-65	3.8	1.0	Cracks in the fiber-reinforced shotcrete lining; load-bearing capacity not impaired
5	10	72	35-55	3.1	0.6	Support without visible damage; full bond maintained
6	12	48	30-48	2.8	0.4	Support without damage; full bond maintained
7	15	27	25-40	2.4	0.1	
8	20	12	20-32	1.9	-0.2	
9	25	6.5	15-25	1.5	-0.4	

The safe distance required to preserve the functionality of the combined support system during blasting of fan-pattern blastholes with a charge of up to 320 kg is at least 8-10 m from the blast source, where PPV does not exceed 15 mm/s. Beyond $r = 18$-20 m, the effect of blasting on the support system is practically negligible.

A detailed analysis of the performance of the rock bolts (Fig. 13) showed that the roof-row bolts were the most heavily loaded, with axial forces reaching 0.145-0.149 MN at a load-bearing capacity of 0.17 MN, which corresponds to 85-88% utilization of their strength capacity. The sidewall bolts operated within a range of 0.098-0.118 MN, or 58-69% of their load-bearing capacity. The factor of safety for the most heavily loaded roof bolts was 1.14-1.17, indicating that they remained serviceable despite a limited strength reserve. None of the support elements reached a limit state, either under static conditions or under the dynamic loading induced by blasting of fan-pattern blastholes.

The combined results of the pilot-scale industrial tests confirmed the numerical modeling data and established that the use of a combined support system consisting of 1.8 m long GFRP rock bolts and a 0.05 m thick fiber-reinforced shotcrete lining ensures the stable condition of temporary mine workings under conditions of rib pillar re-mining and dynamic loading.

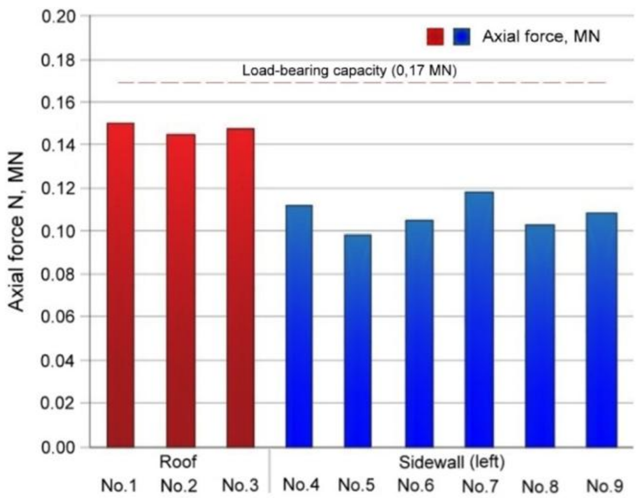


Figure 13. Axial forces in rock bolts and load-bearing capacity utilization factor

Blasting for pillar extraction have a significant effect on the bond between the support and the rock only in the immediate vicinity of the blast source ($r < 8$ m), at PPV > 50 mm/s. Beyond this zone, the combined support system effectively withstands both static and repeated dynamic loads without loss of load-bearing capacity or bond with the rock mass.

4. Conclusions

It was established that the rock mass in areas of rib pillar re-mining is characterized by a limit stress-strain state. In the absence of support, the factor of safety ranges from 0.60 to 0.90, while the residual strength reserve does not exceed 10% of the total rock mass strength. The main factors contributing to stability loss are the concentration of abutment pressure in the marginal parts of mined-out stopes, where the maximum major principal stresses reach 104 MPa, and the dynamic impact of blasting operations, which promotes the formation of tensile stress zones in the roof and sidewalls of temporary workings.

It was shown that the use of only a 0.05-0.10 m thick fiber-reinforced shotcrete lining does not provide the required level of stability for the temporary working. The most rational solution is a combined support system consisting of 1.8 m long GFRP rock bolts installed at a spacing of 1.0-1.2 m and a 0.05 m thick fiber-reinforced shotcrete lining. With this support system, the rock mass factor of safety exceeds 1.0, while axial forces in the most heavily loaded roof-row bolts amount to 0.145-0.149 MN at a load-bearing capacity of 0.17 MN, corresponding to a factor of safety of 1.14-1.17.

Pilot-scale industrial tests established that the zone of significant blast influence on the bond between the support and the rock mass is limited to a distance of less than 8 m from the blast source. In the transition zone, at r = 5-7 m and PPV = 158-340 mm/s, isolated hairline cracks were observed in the fiber-reinforced shotcrete lining; however, the bond between the lining and the rock mass, as well as the load-bearing capacity of the support system, was preserved. In the zone $r \geq 10$ m, at PPV ≤ 72 mm/s, the support system exhibited no visible damage. This makes it possible to adopt a safe distance of at least 8–10 m from the blast source to preserve the functionality of the combined support system under the considered drilling and blasting parameters.

It was established that attenuation of peak particle velocity during blasting of fan-pattern blastholes with a charge of up to 320 kg is described by the obtained relationship $PPV = 700 \cdot \left(Q^{1/3}/r\right)^{1.6}$. This relationship agrees well with the experimental data over the distance range of r = 2-25 m and can be used to predict the intensity of dynamic impact when designing support systems for temporary workings under similar mining and geological conditions.

The research results confirm the applicability of the proposed approach to the geomechanical justification of combined support parameters for temporary workings during rib pillar extraction. Further studies should be directed toward refining the physical and mechanical properties of the fiber-reinforced shotcrete lining and its bond with the rock surface under repeated cycles of dynamic loading.

Author contributions

Conceptualization: YS; Data curation: YS; Formal analysis: YS; Funding acquisition: YI; Investigation: YI; Methodology: YS; Project administration: YI; Resources: ZK; Software: KS; Supervision: YS; Validation: KS; Visualization: SA; Writing – original draft: YI, ZK, SA; Writing – review & editing: YS, KS, ZK, SA. All authors have read and agreed to the published version of the manuscript.

Funding

This research is funded by the Science Committee of the Ministry of Science and Higher Education of the Republic of Kazakhstan (Grant No. AP26196937 "Developing innovative methods to improve mine stability during underground redevelopment by examining the impact of explosive loads on rock mass").

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] Wang, Y., Chen, Q., Dai, B., & Wang, D. (2024). Guidance and review: Advancing mining technology for enhanced production and supply of strategic minerals in China. *Green and Smart Mining Engineering*, *1*, 2-11. https://doi.org/10.1016/j.gsme.2024.03.005

[2] Li, S., Su, W., Yin, T., Dan, Z., & Peng, K. (2025). Research progress and typical case of open-pit to underground mining in China. *Applied Sciences*, *15*(15), 8530. https://doi.org/10.3390/app15158530

[3] Miroshnykov, I., Cichoń, D., Shyrin, L., Dybrin, S., & Dychkovskyi, R. (2025). Ensuring the environmental sustainability of molybdenum ore mining. *IOP Conference Series: Earth and Environmental Science*, *1457*(1), 012014. https://doi.org/10.1088/1755-1315/1457/1/012014

[4] Kalybekov, T., Rysbekov, K., Nauryzbayeva, D., Toktarov, A., & Zhakypbek, Y. (2020). Substantiation of averaging the content of mined ores with account of their readiness for mining. *E3S Web of Conferences*, *201*, 01039. https://doi.org/10.1051/e3sconf/202020101039

[5] Kosenko, A., Khomenko, O., Kononenko, M., Polyanska, A., Buketov, V., Dychkovskyi, R., Polański, J., Howaniec, N., & Smolinski, A. (2025). Sustainable management of iron ore extraction processes using methods of borehole hydro technology. *International Journal of Mining and Mineral Engineering*, *16*(1), 92-112. https://doi.org/10.1504/ijmme.2025.145592

[6] Stupnik, M., Kalinichenko, V., & Pismennyi, S. (2013). Pillars sizing at magnetite quartzites room-work. *Annual Scientific-Technical Collection – Mining of Mineral Deposits*, 11-15. https://doi.org/10.1201/b16354-3

[7] Rysbekov, K., Toktarov, A., Kalybekov, T., Moldabayev, S., Yessezhulov, T., & Bakhmagambetova, G. (2020). Mine planning subject to prepared ore reserves rationing. *E3S Web of Conference*, *168*, 00016. https://doi.org/10.1051/e3sconf/202016800016

[8] Lyashenko, V.I., Dudar, T.V., Oliinyk, T.A., Stus, V.P., Shapovalov, V.A., & Garanenko, T.R. (2025). Justification of the efficiency of ore deposits underground development technology by mathematical modeling methods. *Mineral Resources of Ukraine*, *4*, 73-79. https://doi.org/10.31996/mru.2025.4.73-79

[9] Lyashenko, V.I., Dudar, T.V., Oliinyk, T.A., & Shapovalov, V.A. (2024). Justification of the effectiveness of subsoil protection and environmental safety during the development of near-surface reserves of ore deposits. *Mineral Resources of Ukraine*, *3*, 78-85. https://doi.org/10.31996/mru.2024.3.78-85

[10] Yelemessov, K., Baskanbayeva, D., Sabirova, L., Martyushev, N.V., Malozyomov, B.V., Zhanar, T., & Golik, V.I. (2025). Algorithmic optimal control of screw compressors for energy-efficient operation in smart power systems. *Algorithms*, *18*(9), 583. https://doi.org/10.3390/a18090583

[11] Skrzypkowski, K. (2020). Decreasing mining losses for the room and pillar method by replacing the inter-room pillars by the construction of wooden cribs filled with waste rocks. *Energies*, *13*(14), 3564. https://doi.org/10.3390/en13143564

[12] Ryilnikova, M.V., Yun, A.B., & Terenteva, I.V. (2015). Perspektivy i strategiya osvoeniya Zhezkazganskogo mestorozhdeniya. *Gornyiy zhurnal*, *5*, 44-49.

[13] Issayeva, L., Istekova, S., Tolybaeva, D., Togizov, K., Saurykov, Z., & Issagaliyeva, A. (2025). Petrophysical characteristics of geological formations of the Zhezkazgan Ore District (Kazakhstan) and their relationship with mineralization. *Minerals*, *15*(11), 1106. https://doi.org/10.3390/min15111106

[14] Nurpeisova, M.B., Salkynov, A.T., Soltabayeva, S.T., & Miletenko, N.A. (2024). Patterns of development of geomechanical processes during hybrid open pit/underground mineral mining. *Eurasian Mining*, *41*(1), 7-11. https://doi.org/10.17580/em.2024.01.02

[15] Nurpeisova, M.B., Meirambek, G., Fedotenko, N.A., & Anetov, B.T. (2024). A new approach to monitoring deformation of engineering structures in seismically active regions. *Eurasian Mining*, *2*, 43-45. https://doi.org/10.17580/em.2024.01.09

[16] Aitkazinova, S., Golovko, Y., Sdvyzhkova, O., Imansakipova, B., Babets, D., & Kirgizbayeva, D. (2025). Modeling of elastic oscillations in a fractured rock mass ahead of an underground roadway face. *Engineered Science*, *35*, 1582. https://doi.org/10.30919/es1582

[17] Ismagulova, A.Z., Begentayev, M.M., & Tileuberdi, N. (2025). Studies of influence of lithological composition of overburden rock on colmation process in open-type infiltration basins. *ES Materials and Manufacturing*, *28*, 1473. https://doi.org/10.30919/mm1473

[18] Serdaliyev, Y., Iskakov, Y., Bakhramov, B., & Amanzholov, D. (2022). Research into the influence of the thin ore body occurrence elements and stope parameters on loss and dilution values. *Mining of Mineral Deposits*, *16*(4), 56-64. https://doi.org/10.33271/mining16.04.056

[19] Petlovanyi, M., Saik, P., Lozynskyi, V., Sai, K., & Cherniaiev, O. (2023). Substantiating and assessing the stability of the underground system parameters for the sawn limestone mining: Case study of the Nova Odesa deposit, Ukraine. *Inżynieria Mineralna*, *1*(1), 79-89. https://doi.org/10.29227/IM-2023-01-10

[20] Lyashenko, V.I., Khomenko, O.E., & Golik, V.I. (2020). Development of environment-friendly and resource-saving methods of underground ore mining in disturbed rock masses. *Mining Science and Technology*, *5*(2), 104-118. https://doi.org/10.17073/2500-0632-2020-2-104-118

[21] Pashchenko, O., Khomenko, V., Ishkov, V., Koroviaka, Y., Kirin, R., & Shypunov, S. (2024). Protection of drilling equipment against vibrations during drilling. *IOP Conference Series: Earth and Environmental Science*, *1348*(1), 012004. https://doi.org/10.1088/1755-1315/1348/1/012004

[22] Buktukov, N.S., Gumennikov, E.S., & Mashataeva, G.A. (2019). In-situ gasification of steeply dipping coal beds with production hole making by supersonic hydraulic jets. *Mining Informational and Analytical Bulletin*, *9*, 30-40. https://doi.org/10.25018/0236-1493-2019-09-0-30-40

[23] Begalinov, A.B., Serdaliev, E.T., Iskakov, E.E., & Amanzholov, D.B. (2013). Shock blasting of ore stockpiles by low-density explosive charges. *Journal of Mining Science*, *49*(6), 926-931. https://doi.org/10.1134/s1062739149060129

[24] Akhmetkanov, D.K. (2023). New variants for wide orebodies high-capacity mining systems with controlled and continuous in-line stoping. *News of the National Academy of Sciences of the Republic of Kazakhstan, Series of Geology and Technical Sciences*, *3*(459), 6-21. https://doi.org/10.32014/2023.2518-170X.295

[25] Pysmennyi, S., Fedko, M., Chukharev, S., Rysbekov, K., Kyelgyenbai, K., & Anastasov, D. (2022). Technology for mining of complex-structured bodies of stable and unstable ores. *IOP Conference Series: Earth and Environmental Science*, *970*(1), 012040. https://doi.org/10.1088/1755-1315/970/1/012040

[26] *Issledovanie sootvetstviya opredeleniya parametrov i sistemyi otrabotki v usloviyah muldyi sdvizheniya zalezhey shahtyi Annenskaya*. (2022). Otchet po NIR. Almaty, Kazakhstan: TOO "KazNIItsvetmet", 342 p.

[27] Yun, A.B., Akhanov, T.M., Urumov, V.A., Ungitbayev, M.Zh., & Aymyshev, B.Sh. (2005). Extraction of pillars from field workings. *Gornyi Zhurnal Kazakhstana*, *3*, 6-7.

[28] Ratov, B.T., Khomenko, V.L., Kuttybayev, A.E., Togizov, K.S., & Utepov, Z.G. (2024). Innovative drill bit to improve the efficiency of drilling operations at uranium deposits in Kazakhstan. *News of the National Academy of Sciences of the Republic of Kazakhstan, Series of Geology and Technical Sciences*, *4*(465), 224-236. https://doi.org/10.32014/2024.2518-170X.437

[29] Toshov, J.B., Fozilov, D.M., Yelemessov, K.K., Ruziev, U.N., Abdullayev, D.N., Baskanbayeva, D.D., & Bekirova, L. (2024). Increasing the durability of drill bit teeth by changing its manufacturing technology. *Metal Working and Material Science*, *26*(4), 112-124. https://doi.org/10.17212/1994-6309-2024-26.4-112-124

[30] Ratov, B.T., Mechnik, V.A., Bondarenko, N.A., Kolodnitsky, V.N., Khomenko, V.L., Sundetova, P.S., Korostyshevsky, D.L., Bayamirova, R.U., & Makyzhanova, A.T. (2024). Increasing the durability of an impregnated diamond core bit for drilling hard rocks. *SOCAR Proceedings*, *1*, 24-31. https://doi.org/10.5510/OGP20240100936

[31] Karmanov, T., Tussupbayev, N., Kaliyev, B., Zhautikov, B., & Mauletbekova, A.B. (2024). Optimizing separation of waste drilling muds through ultraflocculation and flocculant selection. *Acta Montanistica Slovaca*, *29*(3), 618-629. https://doi.org/10.46544/ams.v29i3.09

[32] Muratova, S., Pashchenko, O., Khomenko, V., & Zhailiev, A. (2025). Application of machine learning for wellbore stability assessment. *Engineering for Rural Development*, *24*, 505-511. https://doi.org/10.22616/ERDev.2025.24.TF109

[33] Kononenko, M., Khomenko, O., Cabana, E., Mirek, A., Dyczko, A., Prostański, D., & Dychkovskyi, R. (2023). Using the methods to calculate parameters of drilling and blasting operations for emulsion explosives. *Acta Montanistica Slovaca*, *28*(3), 655-667. https://doi.org/10.46544/ams.v28i3.10

[34] Serdaliyev, Y., Iskakov, Y., & Alibayev, A. (2024). Control of blast parameters for high-quality breaking of thin slope ore bodies. *Mining of Mineral Deposits*, *18*(2), 49-59. https://doi.org/10.33271/mining18.02.049

[35] Lozynskyi, V., Yussupov, K., Rysbekov, K., Rustemov, S., & Bazaluk, O. (2024). Using sectional blasting to improve the efficiency of making cut cavities in underground mine workings. *Frontiers in Earth Science*, *12*, 1366901. https://doi.org/10.3389/feart.2024.1366901

[36] Kononenko, M., Khomenko, O., Kovalenko, I., Kosenko, A., Zahorodnii, R., & Dychkovskyi, R. (2023). Determining the performance of explosives for blasting management. *Rudarsko-Geolosko-Naftni Zbornik*, *38*(3), 19-28. https://doi.org/10.17794/rgn.2023.3.2

[37] Pysmennyi, S., Chukharev, S., Kourouma, I.K., Kalinichenko, V., & Matsui, A. (2023). Development of technologies for mining ores with instable hanging wall rocks. *Inżynieria Mineralna*, *1*(1), 103-112. https://doi.org/10.29227/IM-2023-01-13

[38] 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), 69-78. https://doi.org/10.29227/IM-2023-01-08

[39] Yang, J.H., Lu, W.B., Chen, M., Yan, P., & Zhou, C.B. (2018). Dynamic stress adjustment and rock damage during blasting excavation in a deep-buried circular tunnel. *Tunnelling and Underground Space Technology*, *71*, 591-604. https://doi.org/10.1016/j.tust.2017.10.005

[40] Liu, S., Liu, H., Xie, K., Shan, L., Xiao, F., Wang, B., & Wang, Y. (2025). Stability analysis and support optimization for a coal mine roadway subjected to high horizontal stress. *Applied Sciences*, *15*(5), 2276. https://doi.org/10.3390/app15052276

[41] Wang, D., Cheng, J., Li, R., Qu, Y., Jiang, H., Shen, Y., Zhao, S., & Wu, M. (2024). Research on the twin-link stepping temporary support mechanism and the repeatable supportability of the roof in the underground excavation roadway of coal mine. *Applied Sciences*, *14*(15), 6659. https://doi.org/10.3390/app14156659

[42] Yun, A.B., Bekturganov, N.S., & Stolpovskikh, V.V. (2011). *Geomechanical substantiation of interchamber pillar extraction under conditions of open mined-out space*. Almaty, Kazakhstan: National Center for Complex Processing of Mineral Raw Materials of the Republic of Kazakhstan, 248 p.

[43] Skrzypkowski, K. (2019). The influence of room and pillar method geometry on the deposit utilization rate and rock bolt load. *Energies*, *12*(24), 4770. https://doi.org/10.3390/en12244770

[44] Iskakov, E., Amanzholov, D., & Kenessov, Z. (2025). Innovative approaches to stabilizing underground mine workings in unstable rock masses by creating a preliminary protective shield. *Mining of Mineral Deposits*, *19*(2), 65-74. https://doi.org/10.33271/mining19.02.065

[45] Volkov, A.P., Buktukov, N.S., & Kuanyshbaiuly, S. (2022). Safe and effective methods for mining thin tilt and steeply dipping deposits with ore drawing via mud flow. *Gornyi Zhurnal*, *4*, 86-91. https://doi.org/10.17580/gzh.2022.04.13

[46] Kuzmenko, O., Dychkovskyi, R., Petlovanyi, M., Buketov, V., Howaniec, N., & Smolinski, A. (2023). Mechanism of interaction of backfill mixtures with natural rock fractures within the zone of their intense manifestation while developing steep ore deposits. *Sustainability*, *15*(6), 4889. https://doi.org/10.3390/su15064889

[47] Kyelgyenbai, K., Pysmennyi, S., Chukharev, S., Purev, B., & Jambaa, I. (2021). Modelling for degreasing the mining equipment downtime by optimizing blasting period at Erdenet surface mine. *E3S Web of Conferences*, *280*, 08001. https://doi.org/10.1051/e3sconf/202128008001

[48] *Tekhnologicheskaya instruktsiya po povtornoy razrabotke Zhezkazganskogo mestorozhdeniya podzemnym sposobom*. (1998). Almaty, Kazakhstan Korporatsiya "Kazakhmys"; Zhezkazgan NIP-Itsvetmet; NTs KPMS RK; IGD im. D. A. Kunayeva: 56 s.

[49] Serdaliyev, Y., & Iskakov, Y. (2024). Research into mass stress and failure zone parameters during blasting of fractured high benches using blasthole charges. *Mining of Mineral Deposits*, *18*(4), 98-108. https://doi.org/10.33271/mining18.04.098

[50] Begalinov, A., Serdaliyev, Y., Abshayakov, E., Bakhramov, B., & Baigenzhenov, O. (2015). Extraction technology of fine vein gold ores. *Metallurgical & Mining Industry*, *7*(4), 312-320.

[51] Cooper, P.W. (1996). *Explosives engineering*. Danvers, United States: John Wiley & Sons, 480 p.

[52] Bazaluk, O., Petlovanyi, M., Zubko, S., Lozynskyi, V., & Sai, K. (2021). Instability assessment of hanging wall rocks during underground mining of iron ores. *Minerals*, *11*(8), 858. https://doi.org/10.3390/min11080858

[53] Saadat, M., Khishvand, M., & Seccombe, A. (2025). FLAC3D-IMASS modelling of rock mass damage in unsupported underground mining excavations: A safety factor-based framework. *Mining*, *5*(4), 60. https://doi.org/10.3390/mining5040060

Геомеханічне обґрунтування комбінованого кріплення тимчасових виробок під час відпрацювання міжкамерних ціликів в умовах динамічного навантаження

Є. Сердалієв, Є. Іскаков, К. Шукірбаєв, Ж. Кенесов, С. Амангелді

Мета. Обґрунтування параметрів комбінованого кріплення тимчасових гірничих виробок, що забезпечують доступ до міжкамерних ціликів під час їх повторного відпрацювання в умовах техногенно порушеного масиву Жезказганського родовища з урахуванням динамічного впливу буропідривних робіт.

Методика. Дослідження виконано із застосуванням комплексного підходу, який передбачав аналітичну оцінку поширення вибухових хвиль у масиві з виробленими просторами, чисельне моделювання напружено-деформованого стану масиву в програмному комплексі Rocscience RS2 при різних варіантах кріплення, а також дослідно-промислові випробування з інструментальними сейсмометричними вимірюваннями за допомогою станції Sigma 4+ в умовах Східно-Жезказганського рудника.

Результати. Встановлено, що масив гірських порід у районах повторного відпрацювання характеризується граничним напружено-деформованим станом, а коефіцієнт запасу стійкості без кріплення становить 0.60-0.90. Обґрунтовано комбіновану систему кріплення, що включає склопластикові анкери GFRP довжиною 1.8 м із кроком 1.0-1.2 м та фібробетонне покриття товщиною 0.05 м. Така система забезпечує коефіцієнт запасу стійкості понад 1.0 за використання несучої здатності анкерів до 85-88%. Дослідно-промисловими випробуваннями встановлено, що безпечна відстань для збереження функціональності кріплення під час підривання віялових свердловин із зарядом до 320 кг становить не менше 8-10 м від джерела вибуху при PPV $\leq$ 15 мм/с. Також встановлено степеневу залежність затухання пікової швидкості коливань від відстані до джерела вибуху та маси заряду вибухової речовини.

Наукова новизна. Для умов повторного відпрацювання міжкамерних ціликів Жезказганського родовища вперше встановлено параметри комбінованого кріплення тимчасових виробок з урахуванням динамічного навантаження від буропідривних робіт. Обґрунтовано степеневу залежність затухання сейсмічних хвиль, виявлено три характерні зони впливу вибуху на зчеплення кріплення з масивом та визначено безпечну відстань для збереження несучої здатності кріплення.

Практична значимість. Полягає у підвищенні безпеки ведення гірничих робіт під час повторного відпрацювання міжкамерних ціликів завдяки застосуванню економічно доцільної комбінованої системи кріплення, яка забезпечує стійкість тимчасових виробок в умовах інтенсивного динамічного впливу буропідривних робіт і збереження зчеплення фібробетонного покриття з породним масивом.

Ключові слова: *міжкамерні цілики; тимчасові виробки; комбіноване кріплення; фібробетон; склопластикові анкери GFRP; чисельне моделювання; сейсмомоніторинг*

Publisher's note