Mining of Mineral Deposits
Volume 18 (2024), Issue 2, 49-59

https://doi.org/10.33271/mining18.02.049

Control of blast parameters for high-quality breaking of thin slope ore bodies

Yerdulla Serdaliyev [1], Yerkin Iskakov [1*], Azamat Alibayev [1]

[1] *Satbayev University, Almaty, Kazakhstan*

**Corresponding author: e-mail y.iskakov@satbayev.university*

Abstract

Purpose. The research is aimed at substantiation of the effective method for mining thin slope ore bodies occurring in soft unstable host rocks by optimizing the breaking process, while determining the patterns of blast energy impact on the disturbed mass by explosive charges with controllable density, taking into account the geomechanical rock mass state.

Methods. The research uses a comprehensive approach, including analysis of literature sources, practical experience of mining the slope ore bodies in difficult mining-geological conditions, modeling of the energy characteristics of blasts and wave action on the mass using software, as well as conducting experimental-industrial tests in the Akbakai mine.

Findings. An innovative method for effective and safe ore mining from thin slope ore deposits in masses with weakened host rocks has been substantiated and developed. It implies the use of a new construction and location in the blast-holes of a charge consisting of mixed low-density explosives with widely controllable characteristics and with which the blast-holes are charged in two layers with different densities of explosives and detonated at different delay intervals. The optimum delay intervals have been determined, which improve the conditions for controlling the blast energy by changing the direction of the blast action vector towards the newly outcropped surfaces formed in the rock mass after the blasting the first stage charges. The main factors influencing the ore delivery range when mining thin slope ore bodies with blast delivery system have been revealed and methods for increasing this process efficiency are proposed.

Originality. New parameters of drilling and blasting operations have been determined for the conditions of mining thin slope ore bodies of the Akbakai deposit: a rational charge construction with controllable blast characteristics has been developed; the optimum range of blast-hole charging density with mixed low-density explosives and delay intervals have been substantiated; a new exponential dependence of the ore delivery range on the specific blasting agent consumption and the angle of the ore body occurrence has been revealed.

Practical implications. Practical significance is in increasing the efficiency of blast breaking of minerals, improving the quality of blast delivery of broken ore to loading sites while maintaining the host rock mass continuity and reducing the ore mass dilution, eliminating the formation of large-sized pieces that complicate the blast delivery of the broken ore.

Keywords: *deposit, mining, dilution, loss, blast construction, blast energy, blasting agent, charge density*

1. Introduction

In today's mining industry, the efficient mining of thin slope ore bodies occurring in soft and unstable host rocks is a challenging task that requires innovative approaches and technologies [1]-[4]. Insufficient mass stability, high probability of rock failure and mineral losses during mining are key challenges faced by mining enterprises [5]. In this context, optimizing the breaking process of slope ore bodies becomes a priority to improve mining efficiency and to ensure the safety of mine workers.

The complex geological structure of the mass with weakened host rocks, as well as their varying stability and geometry, including the inclination angle of ore deposits for gravity delivery, create significant obstacles and increase the cost of mining thin slope ore deposits [6], [7]. Under such conditions, at most mining enterprises in Kazakhstan, the efficiency of mining methods is significantly reduced due to high ore dilution resulting from waste rock spalling and ore losses in slope stopes, caused by the rock mass pile formation at the block bottom. These data are also confirmed by a number of researchers [8]-[10].

In the studies of foreign scientists, methods of controllable rock destruction using rock pressure are increasingly used [11]-[13]. The principle of self-caving system operation is based on the use of ore weight and potential energy of rocks. By undercutting the ore block at the base and then weakening its connection to the mass, pressure on the top of the block leads to its gradual caving [14], [15]. In cases where it is impossible to achieve self-caving of a block under the weight of overlying rocks, the method of controllable block destruction using rock pressure is used [16]. This is achieved by creating special conditions, including the placement of cut holes that weaken the block on its sides and cut in the middle. Different combinations of cut hole positions form critical stresses leading to block destruction.

Received: 26 February 2024. Accepted: 24 May 2024. Available online: 30 June 2024

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

In [17], the use of host rock self-caving is recommended to reduce the cost of explosives. Creating conditions under which stresses in rocks exceed their ultimate strength allows us to talk about the method of controllable self-caving. This method is based on the instantaneous destruction of supporting structures, resulting in the use of rock pressure energy to destroy rocks.

Rock pressure control significantly reduces both labor and material resources [18], [19]. Tests of blastless rock mining technology, including the preliminary formation of de-stressing cavities, demonstrate the controlled release of potential rock mass energy reserves [20]. Such control of the rock destruction process using rock pressure forces makes it possible to increase the rate of mining in hard rocks without the occurrence of dynamic phenomena in the latter and to reduce energy consumption for destruction [21], [22].

It is possible to conclude from the conducted analysis that it is possible to controllably use the initial stress field energy for brittle fracture of rocks [23]. It should be noted that blastless impact on technological processes is possible at low rock hardness, but in rocks of medium and high hardness it is impossible to do without the use of blasting technologies [24]. In this case, the principles of energy and resource saving should be taken into account when designing and implementing blasting operations.

Currently, in the studied eastern part of the Akbakai mine (Kazakhstan), when mining the Pologaya vein, there is a significant increase in ore dilution rate, the value of which reaches 77%, and there are frequent cases of oversized pieces of rock spalling (up to 5%), which prevent the ore mass from sliding to the lower sublevel drift (Fig. 1).

(a) (b)

Figure 1. Mining of the Pologaya vein between sublevel drifts No. 18 and No. 19 of the Akbakai mine

Slope (a = 35-50°) gold ore thin (m = 0.7-1.5 m) Akbakai deposits are mined using a sublevel blast-hole stoping by layer-by-layer breaking of ore with deep blast-holes and ore delivery by the blasting force (Fig. 2). With this mining system, the block preparation begins with driving of a haulage roadway, followed by inclines of exploratory undercutting drift (sublevel drift) with a cross-section passing through the ore body at horizon level to clarify the ore body contour within the block boundaries. In the hanging wall of exploratory undercutting drift, drive a niche for organizing an inclined cut-out raise.

Cut-out raises are driven to the overlying sublevel every 50 m and, in addition, this mine working serves as a compensating space (slot) when breaking and drawing ore from the sublevel drifts. The distance between sublevel drifts averages 10 m. Every 100 m along the strike of the vein, pillars 7 m wide are left.

(a)

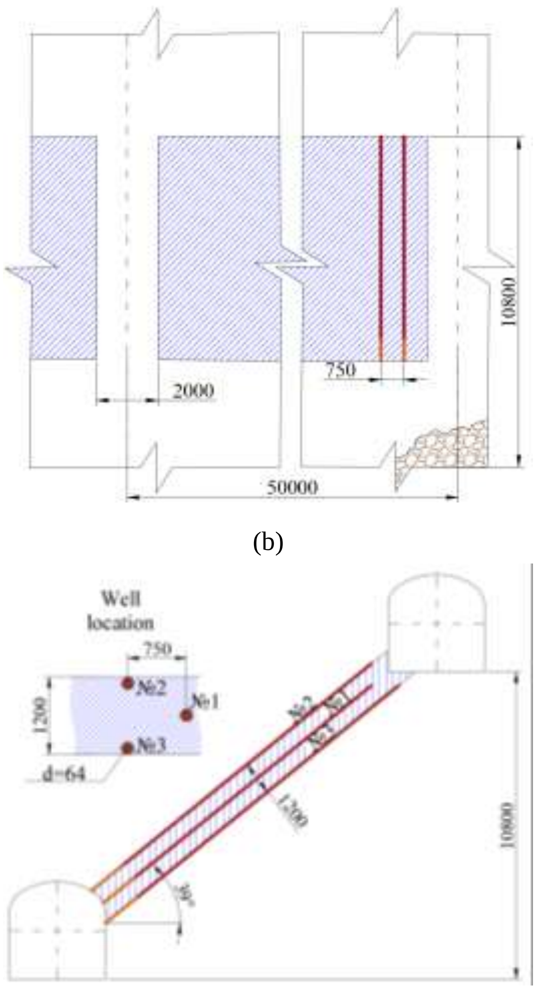


Figure 2. Sublevel blast-hole stoping system with ore delivery by the blasting force used for mining the Pologaya vein of the Akbakai mine: (a) main view; (b) cross-section side view

Stope operations begin with mining the upper sublevel stope and the ore is broken off by blasting parallel blast-holes into a cut-out raise (slot). Further operations on ore breaking with delivery by the blasting force are also carried out with sections of blast-holes drilled from sublevel drifts. The mine uses a parallel grid with a compensating blast-hole for ore breaking in thin slope veins (Fig. 2).

Blast-holes are drilled using percussive-rotary drilling rig of the PHQ-3000 type. In this case, the blast-hole diameter is d = 64 mm, the least resistance line (LRL) is W = 0.7-0.9 m, and the blast-hole length is L = 14-15 m, depending on the inclination angle of the vein.

The advantage of this mining system is the use of blasting force to deliver the broken ore mass. This point is very important for mining areas of slope ore bodies, since with longer delivery range, some of the broken ore does not reach the bottom of the block and special methods are required to deliver it to the site of removal. Otherwise, its loss increases significantly.

As noted above, mining operations in the eastern part of the Akbaka mine have seen a marked increase in the number of rock inrushes during the breaking of ore bodies, particularly in the case of the Pologaya vein. However, the special conditions of ore delivery using blasting force (limitations due to the inability to directly observe the stope contour) made it difficult to monitor the state of the stope contour along the entire length of the ore body. In addition, the small width of the extraction stope did not allow for surveying and determining the boundaries of the host rock mass unstable areas. From a purely visual analysis of the condition of the stopes, it has been revealed that inrushes of oversized pieces mainly occur from the hanging wall of the stope and they are arc-shaped, which is most often localized in the middle stope areas.

It is known from practice that the host rock mass is exposed to more intense deformations during blasting of thin deposits due to the fact that the roof is located in the zone of shear and seismic wave impact [25], [26]. Consequently, it can be assumed that the parameters of spalling and inrushes are related in a certain way to the parameters of blasting operations, the mass structure, the degree of its fracturing, as well as the physical-mechanical properties of the rocks.

Currently, there is no universal calculation method in practice that allows at least an approximate form of the destruction surface, separation of the ore deposit from the mass. Nevertheless, for practical purposes, the most commonly used method for assessing destruction depth is the method known as break off force [27]. Break off force is the main parameter that determines the degree of the host rock destruction during breaking of thin ore bodies, and, consequently, the impact of a particular blast on them. According to the results of an analysis of typical cases of host rock destruction from the blast, the intensity of destruction may range from minor, barely noticeable failure to complete caving of ceilings and inrushes in stopes [28]. Thus, from a practical point of view, it is important and in most cases it is sufficient to have information on the break off force in order to further assess the studied mass stability.

In this regard, in previous studies, a numerical analysis of the mass stress parameters during blasting of groups of blast-hole charges was conducted, based on the theory of ore layer breaking off and rating mass classification for the purpose of preliminary prediction of possible zones and nature of the host rock mass caving during breaking of thin ore depo-sits [29]. The blast impact numerical model has been developed for the disturbed host rock mass of the Pologaya vein in the Akbakai mine, horizon 590, between sublevel drifts No. 18 and No. 19, using the Rocscience RS2 software product (Fig. 3).

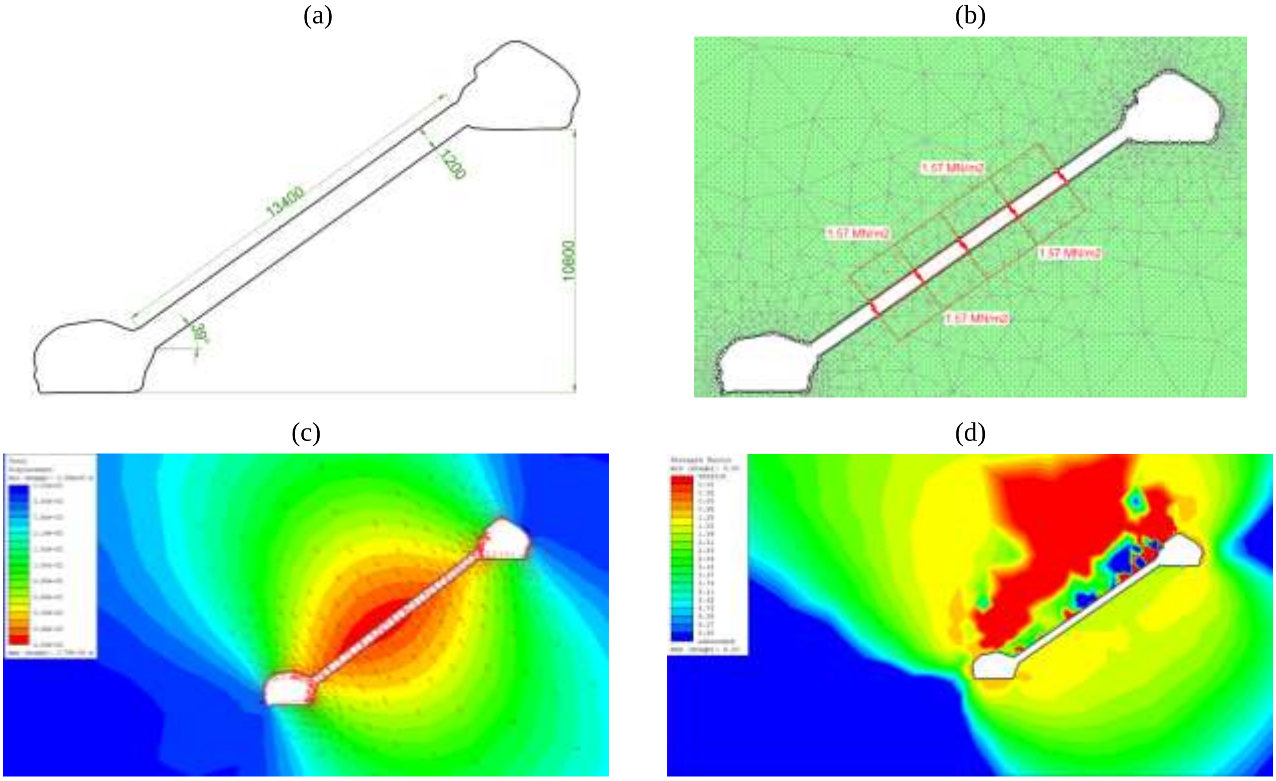


Figure 3. Numerical model of rock mass stress during breaking of the Polalogaya vein in the Akbakai mine [29]: 1 – initial geometry of the studied object; 2 – stress distribution modules in relation to the blasting force value; 3 – vectors of displacement propagation after the blast; 4 – the predicted zone and nature of host rock caving after the blast

In the process of numerical model development, based on the passport parameters of drilling and blasting operations (drill ring No. 57, Pologaya vein), the blasting force impact (1.57 MN/m^2) of explosive charges from ammonium nitrate and diesel fuel (AS-DT) within one drill ring, as well as construction and location of the charge in the blast-holes, have been calculated. Using these parameters, the blast impact has been simulated on the model. Based on the results, it has been determined that the mass caving may occur along unstable contacts between layers, along fractures and along low-strength rock interlayers. Caving will be in the form of an arc-shaped plate with parallel or near-parallel bases. The caving spread depth is within 1.0-1.5 m.

To solve this problem, a method for supporting the host rock mass on the stope roof side with rope bolts has been tested. But this method did not give positive results, as rope bolts do not provide effective mass stability when mining thin deposits [30], [31]. In this regard, it became necessary to conduct research on the development of flexible technologies for mining thin slope ore bodies in difficult mining-geological conditions, by conducting blasting operations that ensure the effective use of blast energy at the variability of destroyed rock characteristics, while maintaining the host rock continuity.

Based on the previously conducted research results, we have found that mixed low-density explosives make it possible to regulate the amount of blast energy per unit volume of a blast-hole by increasing or decreasing the charge energy concentration as the mass resistance or physical-mechanical properties of rocks and ores change, and increase the blast efficiency [32].

Results of conducted research and analysis of production processes of stope excavation and sublevel blast-hole stoping by layer-by-layer breaking of ore with deep blast-holes and ore delivery by the blasting force used when mining thin slope gold-bearing veins of Akbakai deposit show that further intensification of these processes should be aimed at improving quantitative and qualitative ore breaking indicators. This can be achieved through a rational relationship between the host rock mass state and the energy power of the blast, construction and parameters of the blast-hole charge for ores and rocks with changing physical-mechanical properties, as well as mining-geological conditions. Therefore, our further research is aimed at substantiating the optimum parameters of ore breaking by charges of mixed low-density explosives with controllable blast energy, providing during breaking the host rock mass continuity, minimizing ore losses and dilution during mining thin slope ore bodies, the results of which are presented in this paper.

2. Methods

The blastability of rocks, that is, their ability to resist destruction by blasting, correlates with their strength, toughness, density, and fracturing. If the other conditions are maintained, an increase in the strength of rocks leads to an increase in their resistance to external forces [33], [34].

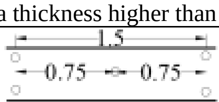
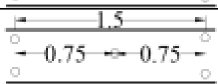
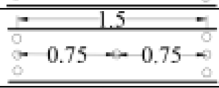
In the process of blasting, rocks exhibit their properties in various combinations, which requires taking into account the complex influence when assessing the resistance of a rock mass to destruction [35]. Due to the lack of a reasonable physical characteristic, the blastability of rocks is mainly assessed by the specific consumption of explosives (or energy) to destroy a given volume into pieces of a certain size under standard conditions [36]. Based on the blast theory, the energy used to crush rocks as a result of the blast impact is proportional to the area of new surfaces (f_c) formed during the rock destruction process [37], [38]. These surfaces, in turn, depend on the degree of crushing, which is determined by the ratio of the average linear particle size (d_e) in the initial mass to the particle size (d_k) after the blast:

$$q = f_c\left(\frac{d_e}{d_k}\right). \quad (1)$$

But this theory does not work effectively in the underground mining of thin slope ore deposits occurring in soft and disturbed host rocks. In such cases, special blast conditions are required to ensure comparability of blast results and to avoid the influence of additional factors on the rock blast difficulty characteristic [39], [40]. It is therefore necessary to assess the rock mass blastability using another characteristic of its resistance to destruction. The blast energy or the value of the nominal explosive flow rate, determined by modeling, can be considered as such a characteristic. This method of assessing the rock mass blastability, based on the blast energy, takes into account, in addition to the elastic, strength and structural characteristics of the mass, also charge parameters, blasting conditions and crushing quality.

In the studied Akbakai mine, in order to reduce ore loss and dilution by reducing the blasting force impact on the out-contour rock mass of the stope space, standard forms of blast-hole location are used depending on the mass rating (RMR/GSI) and the vein thickness (Table 1).

Table 1. Standard forms of blast-hole layout depending on the mass rating and vein thickness

RMR/GSI	Mass stability characteristic	Layout of blast-holes for veins with thickness from 1.0 to 3.0 m	Layout of blast-holes for veins with a thickness higher than 3.0 m
0-40	Unstable		
40-60	Stable with fastening		
>60	Stable		

According to standard forms, blast-holes within a single drill ring are located in parallel, in close proximity to the design contours of ore deposits. Research has found that in such conditions, in a drill ring of parallel blast-holes charges, the surface area of the explosive charge and the area of contact between it and the medium increases, the mass of charges per unit of lateral surface decreases, while combining the part of the brisant zones of parallel charges within the drill ring. When blasting blast-hole charges, the brisant forms of blast operation and the value of specific impulse are changed, while the conditions for transferring energy into the destroyed mass are improved. There is an interaction of blast waves and a change in the parameters of the resulting stress wave.

When blasting parallel charges in the conditions of thin deposits, two phases of the blast action mechanism in time can be distinguished: the first – until the moment the shock waves meet; the second – after the moment of meeting. The first phase largely predetermines the small fraction ore yield value. The absolute this fraction amount depends on the size of intense crushing zone (R_s), which is formed around each detonated charge and depends on the detonation characteristics of used explosives and the physical-mechanical properties of the destroyed mass. When simultaneously detonated charges converge, the overall dimensions of the intensive crushing zone become smaller than that of the same charges detonated separately.

As shown in the scheme of the action zone of charges (Fig. 4), the absolute amount of undesirable ore fraction yield when crushing per unit length of parallel charges (n), taking into account the zone angle (α), will be reduced by the value of:

$$\Lambda\delta = (n-1) R_S^2 \left(\frac{\pi\alpha}{180} - \sin\alpha \right). \tag{2}$$

It follows from the above expression that the absolute substandard fraction yield during the blasting of parallel charges decreases as the number of blast-holes in the drill ring, the radius of the intense crushing zone and the angle increase (Fig. 4). The latter, in turn, increases as the blast-holes converge in the drill ring and can be calculated using the Formula:

$$ctg\alpha = \frac{l_c}{2} \left(R_S^2 - \frac{l_c^2}{4} \right)^{1/2}, \tag{3}$$

where:

l_c – the distance between the centers of parallel blast-holes in the drill ring.

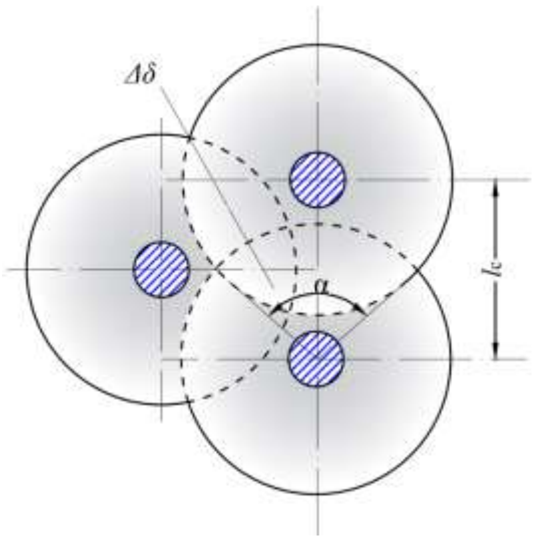


Figure 4. Dimensions of the blast impact zone of parallel charges

From practice it has been found that in such conditions, replacing parallel blast-holes with a single large-diameter blast-hole with mass-equivalent charge makes the organization of drilling operations somewhat easier, but significantly increases undesirable ore fraction yield (ore fines, substandard pieces). The blast impact on the host rocks beyond the design contours of the stope space leads to an intense fracturing in the mass, an increase in the oversize material yield and above-plan ore dilution.

Therefore, the main research purpose is to ensure high-quality crushing of the broken mass through efficient distribution of blast energy and optimization of the parameters of drilling and blasting operations.

To reduce the undesirable blast pulse impact on the blasted mass, causing the host rock caving, the detonation pressure (P_1) can be reduced. It depends on the charge density (ρ_c) and the square of the detonation velocity (D_c):

$$P_1 = \frac{1}{4} \rho D^2 . \tag{4}$$

If blast-hole charges (n) are concentrated into an equivalent large-diameter blast-hole charge, the lateral "charge-rock" contact surface per unit mass of explosives increases by ($n^{1/2}$) times. Accordingly, the time of pressure (P_1) impact on the barrier when reflecting the detonation wave increases, the brisant blast impact manifest itself to a greater extent, and the blast energy loss for undesirable ore fraction yield increases.

Thus, the use of parallel blast-holes when breaking ore in a narrow stoping face provides more opportunities for improving ore crushing quality in the stope space. Therefore, it is expedient to apply the existing standard forms of blast-hole layout for breaking of thin slope ore deposits in the Akbakai mine. Then, the idea of increasing the share of useful use of blast energy should be achieved by the use of rational charge construction and controlled-density composition of explosives, while determining the effective values of delay intervals between blast-holes in the drill ring. This includes the order of blasting within a single blast-hole, allowing a differentiated and rational approach to the blast-hole charge energy distribution.

Based on the results of previous studies [32], it has been determined that the solution to this problem could be achieved by using such an explosive and a mechanism that can generate a charge with a controlled-density charge and a corresponding detonation velocity. The conducted research has led to the development of a new charge construction made of mixed low-density explosives with widely controllable explosive characteristics. These explosive charges are based on a combination of granulated ammonium nitrate, granular polystyrene foam, diesel fuel and Na-CMC. In this case, the contour blast-holes in the drill ring consist of two layers of heterogeneous by density of explosives (Fig. 5) and charge activation delay, while the charge density is controlled by adding granular polystyrene foam to the composition of explosives.

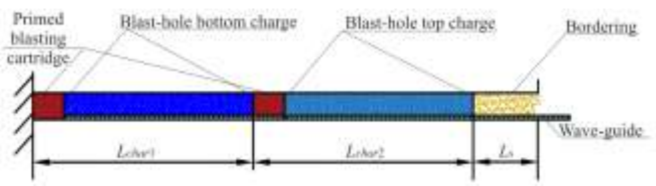


Figure 5. Scheme of charge construction in a blast-hole of mixed low-density explosives

The idea of this construction is that the charge, in addition to increasing the efficiency of blast breaking of minerals and maintaining the host rock mass continuity, is set in order to solve the problem of high-quality delivery of the blasted ore mass to the outlet. As noted above, when mining ore bodies with a dip angle of 35-50° with a long delivery distance, part of the broken rock mass does not reach the outlet working. This results in the formation of a rock pile at the stope bottom, which leads to ore loss and impedes the movement of the broken rock mass in subsequent layers.

Consequently, the purpose of subsequent research is to determine the optimal density of charges (or their parts) depending on the geomechanical state of the host rock mass, that is, to predict the possible depths of caving propagation and to determine the effective values of delay intervals between charges in a drill ring and within one blast-hole, providing the implementation of the proposed technology.

It is known that today there are no universal calculation methods that fully take into account the mechanical properties of the mass and the energy characteristics of explosives acting in the mass. The organization of series-produced experimental-industrial tests is a complex and resource-intensive process that requires significant material, financial and time expenditures. Therefore, in order to solve the above tasks it is expedient to use modeling based on special integrated packages and programs.

The global mining software market offers a wide range of products with two main categories: integrated packages and highly specialized programs for mining [41]-[43]. Although the division between them is conditional, since most packages include many specialized modules, and some highly specialized programs can provide a significant part of the functionality of integrated packages.

Taking into account functional peculiarities of software packages and the conditions for separating the ore from the mass, the SHOTPlus™ UNDERGROUND software product is used to substantiate the basic parameters of drilling and blasting operations. In the modeling process, the tear propagation strength is used as a breakout criterion. The tear propagation strength of the mass is linearly dependent on the rate of its deformation. Then, the dynamic tensile strength of rocks is represented in the following form:

$$\sigma_s = K_d \cdot \sigma_s^{st}, \tag{5}$$

where:

K_d – the dynamicity factor;

σ_s^{st} – the static tear propagation strength of rocks.

To model the behavior of detonation products in the explosion cavity in the case of 2D model, it is assumed that the cylindrical charge has such a small length that the effects of axial and radial motion of detonation products can be neglected. Therefore, to describe this behavior in the explosion cavity, it is enough to consider only the equations of state:

$$dE = P \cdot dV, \quad P = E \cdot (\gamma - 1), \tag{6}$$

where:

E, P – the specific energy and pressure in the explosion cavity;

V – the explosion cavity volume;

γ – the adiabatic index.

The short-delay blasting method with correctly selected delay intervals contributes not only to improving the rock mass crushing and reducing the yield of oversized pieces of ore, but also to reducing the volume of drilling and consumption of explosive materials, while reducing undesirable impacts on the out-contour rock mass [44], [45].

Despite the existing hypotheses about the mechanism of rock destruction with a short-delay blasting method, the optimal delay intervals are determined based on the results of experimental blasts with different deceleration levels, recording the rock mass yield and monitoring of the state of the host rock mass. However, due to the labor-intensive nature of these processes under production conditions, design standards recommend determining the delay interval based on following condition:

$$t_d = t_1 + t_2 + t_3, \tag{7}$$

where:

t_1 – the time of a direct compression or shock wave propagation from the charge to outcropped surface;

t_2 – the time of a direct compression or shock wave propagation from a fracture charge inside the blasting cone in the destructed mass or time of fracture formation with a length equal to the line of least resistance;

t_3 – the time for the mass to swell and shear to the extent that a new outcrop surface is formed, or the time for the blast-damaged mass to move.

According to the determined mechanism for the disturbed rock destruction by blasting, the time of mass destruction with outcropped surface can be considered as the beginning of the destruction process. In addition, the simultaneous rock mass destruction along the line of least resistance depending on both the explosive charge and the rock mass outcropped surface should be taken into account.

At known fracture propagation and expansion velocities, as well as taking into account the above factors, the optimal delay interval is recommended to be determined by the Formula:

$$\tau = \frac{W_a}{V_s} + \frac{b_t}{V_e}, \text{ s}, \tag{8}$$

where:

W – the least resistance line, m;

α – the coefficient to take into account the destruction volume from the stope roof side, depending on the depth of caving propagation (it is recommended to take 0.4 at a depth of caving propagation up to 1.0 m); 0.3 – when the depth of caving propagation is more than 1.0 m;

V_s – the fracture propagation velocity, m/s;

b_t – the average fracture width, m;

V_e – the fracture widening velocity, m/s.

Thus, with the increasing depth of field mining and the transition to forced level caving, where rock pressure plays an increasingly significant role in the process of rock destruction by blasting, the selection of blast-hole breaking parameters becomes key to increasing the efficiency of ore mining. Such a choice should take into account the varying blasting conditions of the broken mass sections, which will optimize the process and reduce mineral losses.

This model considers a single drill ring of blast-hole charges (Fig. 6), represented by a standard form (Table 1) used in the Akbakai mine, based on the following parameters: sublevel height is H = 10.8 m; ore body thickness is m = 1.2 m; inclination angle is α = 39°; blast-hole length is L = 13.4 m; blast-hole diameter is d = 64 mm; least resistance line is W = 0.75 m; total blast-hole charge length is L_{char} = 12.6 m; bottom charge length is L_{char1} = 6.3 m; top charge length is L_{char2} = 6.3 m; bordering length is L_{bord} = 0.8 m.

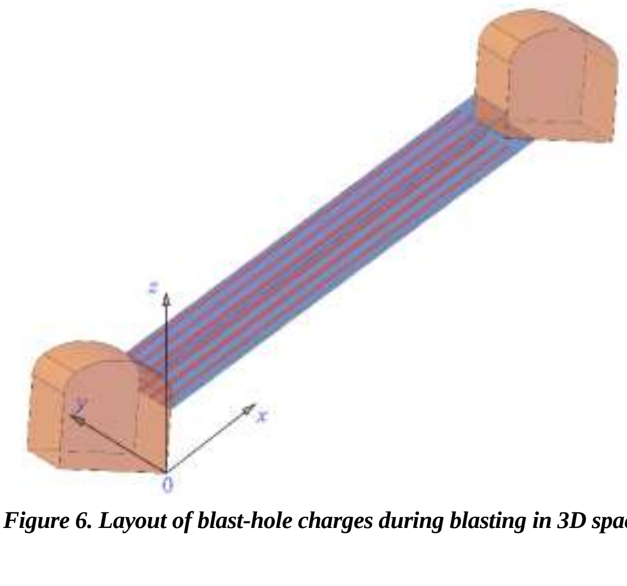


Figure 6. Layout of blast-hole charges during blasting in 3D space

In the SHOTPlus™ UNDERGROUND software product, the finite-difference equations of the model are formed as follows: the domain modeling the rock mass is divided by a grid into quadrangular cells. In a cylindrical coordinate system, each cell is toroidal-shaped with the same medium parameters for the whole cell (Fig. 7).

(a)

(b)

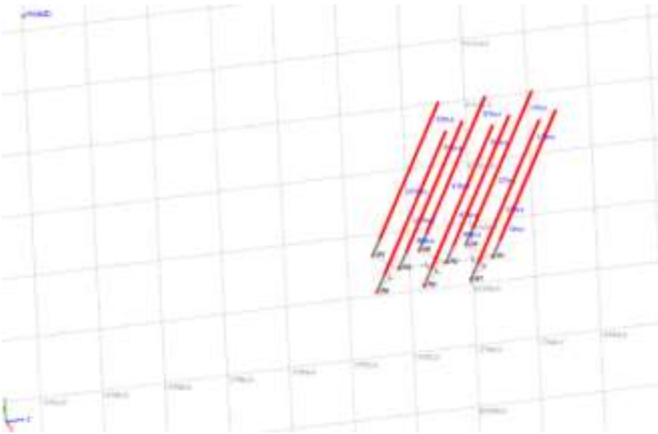


Figure 7. Interface and workspace of the SHOTPlus™ UNDERGROUND software for the block under study: (a), (b) geometric layout of blast-hole in the program workspace from different points

In the research process, the simulation of blasting underground blast-hole charges, located according to the passport data of drilling and blasting operations in the Pologaya vein of the Akbakai mine, horizon 590, between sublevel drifts No. 18 and No. 19 is used. Blast-hole charges are represented as a continuous series of elongated charges initiated sequentially at regular time intervals (τ). Contour blast-hole charges are initiated after the central blast-hole, while the charges within the same blast-hole have different densities and, accordingly, different initiation times.

3. Results and discussion

During the experiment, the dynamics of detonation transmission along the entire length of the charge and the nature of rock destruction during the blasting of a drill ring consisting of three blast-holes in accordance with a standard scheme in the presence of outcropped spaces on both sides around the ore body, that is, on one side – a compensating space (slot) and on the other side – a sublevel drift. Various charge construction variants with different explosive densities, schemes for their layout and blasting intervals have also been tested. In this case, the charges were detonated at intervals that excluded the impact of the wave from the neighboring charge detonation on the wave of the current charge. The blast-hole charge parameters specified during the research are given in Table 2.

Table 2. Specified blast-hole charge parameters during blast modeling

Experience series	Blast-holes	Charge construction	Charge density, kg/m^3 (bottom/top)	Delay interval, ms (bottom/top)
1	No. 1	Solid	700	0
	No. 2	Density-combined	500/350	35/25
	No. 3	Density-combined	500/350	35/25
2	No. 1	Solid	800	0
	No. 2	Density-combined	550/450	30/20
	No. 3	Density-combined	600/450	30/20
3	No. 1	Solid	850	0
	No. 2	Density-combined	650/500	25/20
	No. 3	Density-combined	700/500	25/20
4	No. 1	Solid	900	0
	No. 2	Density-combined	750/650	25/15
	No. 3	Density-combined	750/650	25/15
5	No. 1	Solid	1000	0
	No. 2	Density-combined	800/700	20/10
	No. 3	Density-combined	800/700	20/10

Note: Blast-hole numbers are taken according to the scheme shown in Figure 2

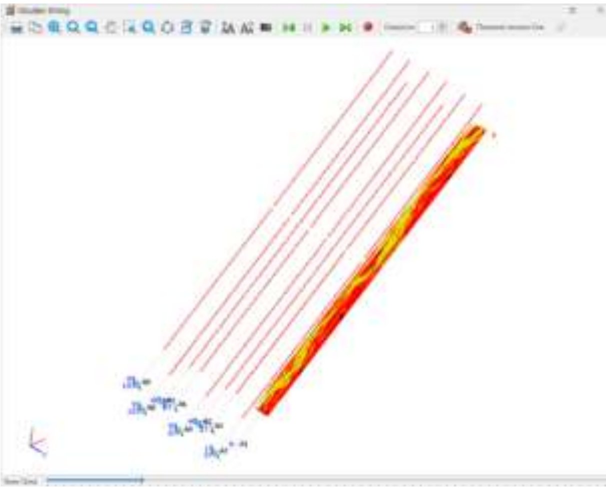
The tools of the used software product enable to configure specified intervals between blast-holes and charges in blast-holes, specify explosive parameters, charge densities, charge separation by density within one blast-hole and bordering in each blast-hole For better visibility, the characteristics of the blast impact have been captured by video and photo animations illustrating the charge detonation dynamics and the sequence of blasting the charges in the blast-holes (Fig. 8).

During the research process, in addition to different time of blast-hole charge initiation within a single drill ring, different initiation time is set for charges with different densities within one blast-hole: blast-hole No. 1 in the drill ring is detonated first, and then the other two blast-holes No. 2 and No. 3 are detonated simultaneously at intervals. In this case, within a single blast-hole, the charge part is initiated first near the hole collar, then the charge part at the blast-hole bottom is initiated at interval.

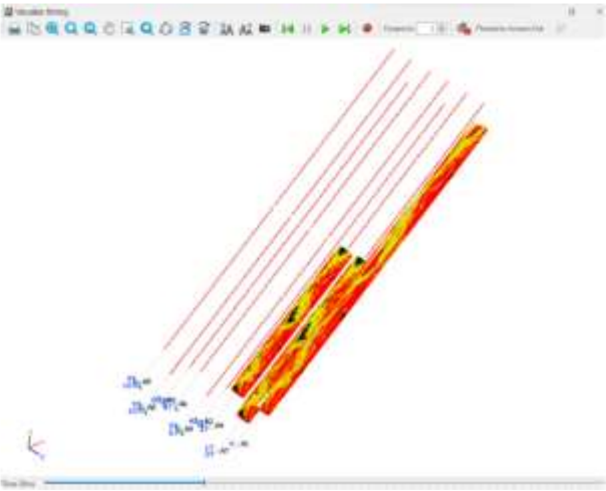
In addition to sequential detonation, the charges of contour blast-holes differ in density: blast-holes No. 2 and No. 3 are charged with two layers with different explosive densities, while blast-hole No. 1 has a solid charge construction. This blast-hole design has been developed to reduce ore loss and dilution, as well as to ensure efficient transfer of blast energy, taking into account the current geomechanical rock mass state, mining conditions and ore body occurrence characteristics.

As a result of the research, it has been revealed that when the charge density in contour blast-holes No. 2 and No. 3 is increased up to 800 kg/m³, in the contact zone between ore and rock, the formation of area of intense blast detonation impact on the mass is observed, the size of which significantly exceeds the similar zone around the charge of the central blast-hole No. 1.

(a)



(b)



(c)

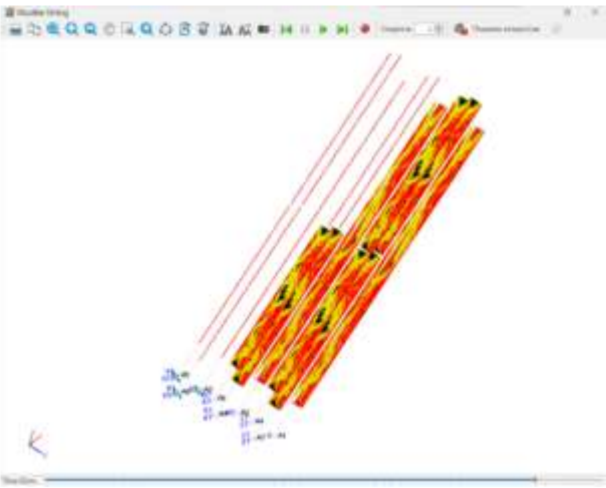


Figure 8. Visualization of blast-hole charge initiation to monitor detonation during blast simulation: (a), (b), (c) – sequence of 1, 2 and 3 blast-hole charge initiation, respectively

In addition, between the charges of contour blast-holes No. 2 and No. 3, there is a clear formation of two stress waves, which eventually merge into one common wave with a flat front, moving towards the outcropped surface. At low charge densities, from 350-450 kg/m³, very low detonation transmission along the charge length is observed.

The explosive mass distribution over different delay intervals results in changes in the activation of deformation waves in the blast-hole: both their decrease and increase can be observed. It has been revealed that a short-term pulse of 10 ms interval between charge layers within one blast-hole causes a dynamic detonation impact in the blast-holes, contributing to the detonation wave movement towards the outcropped surface from the bottom (the side of the sublevel drift).

The research results have shown that optimal parameters for activating deformation waves of blasting blast-hole charges of a given construction are achieved under the following conditions: central well No. 1 is charged with a solid charge with a density of 800-900 kg/m^3 and is initiated first, while contour blast-holes No. 2 and No. 3 are charged with two layers along the length and detonated 15-17 ms after the central one. In this case, the charge density in different parts in both contour blast-holes differs equally: the bottom part of the charge has a density of 650-700 kg/m^3, and the hole collar part – 500-550 kg/m^3. In addition, these charge layers are initiated at 10 ms interval: first the hole collar part is detonated, then, after 10 ms, the bottom part of the charge is detonated. The intervals determined during the research not only improve the dynamics of deformation waves, but also prevent damage to the initiation means from the scattering of rock pieces.

The proposed method for ore breaking with the above parameters has been tested in industrial conditions in the Akbakai mine when mining of Polalogaya and Glubinnaya veins with a dip angle of 35-55°, at an average thickness of 1.3 m, using a sublevel blast-hole stoping by layer-by-layer breaking of ore with deep blast-holes and ore delivery by the blasting force.

Compositions of mixed low-density explosives were produced in mine conditions using a UI-2 [46] mixing plant based on ammonium nitrate, diesel fuel, and granular polystyrene foam; sodium salt carboxymethylcellulose. The blast energy was controlled by varying the charge density in the range of 500 to 900 kg/m³ depending on the polystyrene foam content of the charge, which ranged from 45 to 60% by volume.

As industrial tests have shown, when breaking ore with low-density explosives, the broken rock mass layer is separated from the mass as a result of the action of shear stresses under compressive forces along the ore body contour with a small grab (up to 30%) of waste rock on the side of the hanging wall of the stope (Fig. 9).

(a) (b)

Figure 9. Experimental-industrial tests of breaking thin slope ore bodies in the Akbakai mine using charges with controllable blast characteristics

Due to the fact that the broken layer experiences tensile stress, the kinetic energy of the flying pieces increases and the throw distance increases accordingly, the impact of blasts on the mass is significantly reduced and ore crushing is improved. During the industrial test, the values of throw distance and ore crushing quality have been determined depending on the specific consumption of explosives, taking into account the ore body dip angle (Tables 3 and 4, Fig. 10).

Table 3. Results of the research of the ore throw distance depending on the specific consumption of explosives

Specific consumption of explosives, kg/m^3	Ore throw distance taking into account the body dip angle, m				
	35°	40°	45°	50°	55°
1.0	7-8	8-10	11-12	12-14	15-16
1.2	9-10	9-11	12-14	14-15	17-19
1.4	10-11	11-12	14-15	15-17	19-22
1.6	12-13	13-14	15-16	17-18	–
1.8	13-14	14-15	16-17	18-20	–
2.0	15-16	16-18	18-19	–	–

Table 4. Granulometric composition of blasted ore mass depending on specific consumption of explosives

Grain-size class, mm	Granulometric composition (%) depending on specific consumption of explosives (kg/m^3)					
	1.0	1.2	1.4	1.6	1.8	2.0
0…–100	9	10	10	12	14	15
+100…–200	10	11	12	14	16	18
+200…–300	17	18	20	20	22	23
+300…–400	22	22	23	24	25	27
+400…–500	26	26	24	22	18	14
+500	16	13	11	8	5	3

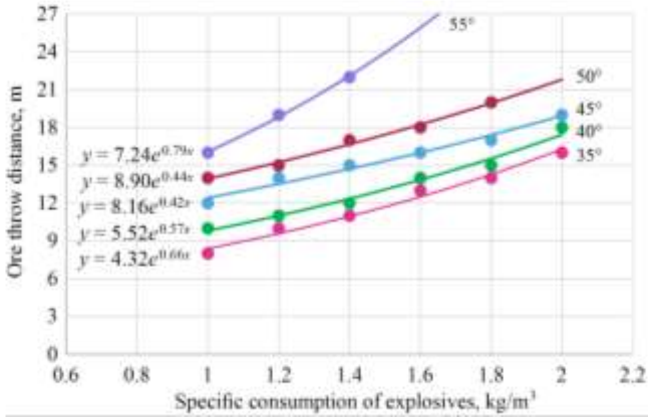


Figure 10. Ore throw distance depending on the specific consumption of explosives (35-55 – dip angles)

In addition to improving the throw distance and the ore crushing quality, the proposed method for breaking thin slope ore bodies with charge construction with controllable blast energy reduces the level of losses and dilution in the conditions of a system with ore delivery by the blasting force. Dilution of blasted ore is reduced by avoiding waste rock blasting, while losses are reduced due to the high-quality delivery of the blasted ore mass to the loading sites (Fig. 11).

Analysis of the obtained results shows that the proposed method for breaking thin slope ore bodies, occurring in soft and disturbed host rocks, due to the effective use of blast energy, ensures the completeness of breaking, improves the crushing quality, increases the ore delivery range up to 25%, reduces losses up to 20% and dilution up to 27%.

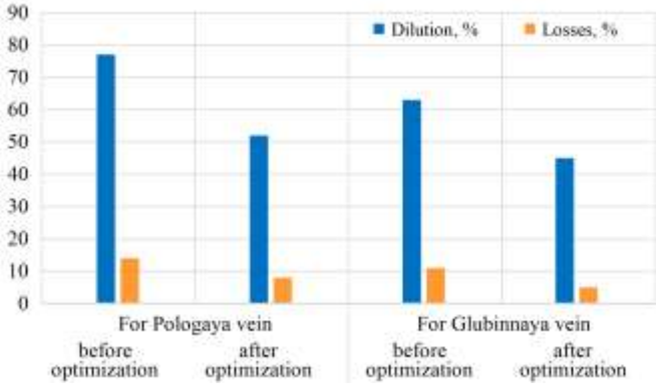


Figure 11. Comparative diagram of ore losses and dilution during mining ore deposits in the Pologaya and Glubinnaya veins of the Akbakai deposit before and after optimization of drilling and blasting operation parameters

Theoretical generalizations and analysis of the practice of mining thin slope ore bodies in difficult mining-geological conditions have shown that the damage from ore loss and dilution in most mines reaches up to 45% of the total cost of metals, which is the main reason for the low efficiency of use of natural resources in such deposits. To improve the efficiency of the technology for mining thin slope ore bodies occurring in unstable host rocks, it is necessary to optimize the breaking process, taking into account the identification of patterns in the blast energy impact in the context of the geomechanical mass state.

The economic and technological effect from the implementation of the proposed technology is achieved by maintaining the host rock mass continuity (reducing the broken ore dilution by avoiding waste rock blasting and reducing the ore loss due to high-quality delivery of the blasted ore mass) by applying a gentle mode of blast impact on the host rock mass and effective blast impact on the ore mass delivery to the loading sites.

4. Conclusions

Since the tear propagation strength of the mass depends on the rate of its deformation, it is required to ensure the use of energy through the rational charge construction and controlled-density composition of explosives, while determining the effective values of delay intervals between blast-holes in the drill ring, including within one blast-hole.

A method for mining thin slope ore deposits, occurring in soft unstable host rocks, has been developed, using a new charge construction made of mixed low-density explosives with widely controllable characteristics. In this method, blast-holes are charged in two layers with different densities of explosives (with densities ranging from 500 to 900 kg/m^3) and detonation velocities (from 2500 to 4000 m/s) and are detonated at different delay intervals. Moreover, the central blast-hole in the drill ring is detonated first, and then, with an interval of 15-17 ms, the contour blast-holes are simultaneously detonated, and within one blast-hole, the charge part located near the hole collar is first initiated, and then, at an interval of 10 ms, the charge part at the blast-hole bottom is initiated.

It has been determined that these delay intervals improve the conditions for controlling the blast energy by changing the direction of the blast action vector towards the newly outcropped surfaces formed in the rock mass after the blasting the first stage charges. Accordingly, after the blasting of the first series of explosive charges, the vectors of the second series of charges change their direction in accordance with the newly formed outcrops in the rock mass. This, in turn, allows for high-quality delivery of the blasted ore mass to the outlet working without forming rock piles at the stope bottom.

Controlling the energy of mixed explosives per unit volume of a blast-hole by increasing or decreasing the charge density as the mass resistance changes and initiating the charge at different times increases the blast efficiency, thereby increasing the ore delivery range up to 25%, reduces the blasting of the host rocks along the ore body contour and due to this reduces losses up to 20% and dilution up to 27%.

It has been determined that the main factors influencing the ore delivery range by the blasting force are the dip angle of the deposit and the specific consumption of explosives. When mining slope ore bodies of the Akbakai mine with a dip angle of 39°, this parameter can be determined by the function $y = 5.52\ e^{0.57\,x}$.

The conducted research has shown that the proposed approaches to improving the use of blast energy and controlling its parameters provide an opportunity to improve the technology of drilling and blasting operations in deposits with other characteristics of ore bodies and host rocks. This should be the subject of further research.

Author contributions

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

Funding

This paper was published based on the results of research work performed within the framework of the AP 14871266 project "Development of innovative methods for efficient and safe underground mining of thin slope ore deposits", with grant funding from Ministry of Science and Higher Education of the Republic of Kazakhstan.

Acknowledgements

We would like to express our heartfelt appreciation to our colleagues and reviewers for their unwavering support, insightful feedback, and encouragement throughout the completion of this research.

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] Hou, D., Xu, M., Li, X., Wang, J., Wang, M., & Li, S. (2023). Optimization of mining methods for deep orebody of large phosphate mines. *Frontiers in Built Environment*, *9*, 1282684. https://doi.org/10.3389/fbuil.2023.1282684

[2] Bekbassarov, S., Soltabaeva, S., Daurenbekova, A., & Ormanbekova, A. (2015). "Green" economy in mining. *New Developments in Mining Engineering 2015: Theoretical and Practical Solutions of Mineral Resources Mining*, 431-434. https://doi.org/10.1201/b19901-75

[3] Turegeldinova, A.Z. (2014). Analysis of the effectiveness of benefit package structure. *Actual Problems of Economics*, *151*, 383.

[4] Vishwakarma, A.K., Himanshu, V.K., & Dey, K. (2024). Evaluation of optimum burden for the excavation of narrow vein ore deposits using numerical simulation. *Rock Mechanics and Rock Engineering*, *57*(2), 945-960. https://doi.org/10.1007/s00603-023-03596-6

[5] Abdellah, W.R.E., Hefni, M.A., & Ahmed, H.M. (2020). Factors influencing stope hanging wall stability and ore dilution in narrow-vein deposits: Part 1. *Geotechnical and Geological Engineering*, *38*, 1451-1470. https://doi.org/10.1007/s10706-019-01102-w

[6] Kassymkanova, K.K., Rysbekov, K.B., Nurpeissova, M.B., Kyrgizbayeva, G.M., Amralinova, B.B., Soltabaeva, S.T., Salkynov, A & Jangulova, G. (2023). Geophysical studies of rock distortion in mining operations in complex geological conditions. *The International Archives of the Photogrammetry, Remote Sensing and Spatial Information Sciences*, *48*, 57-62. https://doi.org/10.5194/isprs-archives-XLVIII-5-W2-2023-57-2023

[7] Abdellah, W. (2022). Effects of a pillarless, center-out stoping pattern on haulage drift performance and ore tonnage at risk. *Rudarsko Geolosko Naftni Zbornik*, *37*(2), 87-96. https://doi.org/10.17794/rgn.2022.2.8

[8] Begalinov, A. (2022). *Razrabotka strukturnoy modeli i tekhniko-tekhnologicheskikh sposobov podderzhaniya geosistemy "massiv-tekhnologiya-podzemnoe sooruzhenie" pri podzemnoy razrabotke zolotorudnykh mestorozhdeniy (Akbakay, Bakyrchik, Maykain) i kompleksnoy mekhanizatsii vedeniya gornykh rabot pri osvoenii grupp mestorozhdeniy zolota (Akbakay)*. Otchet o NIR #0112RK02709. Almaty, Kazakhstan, 158 s.

[9] 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

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

[11] Huo, X., Shi, X., Qiu, X., Zhou, J., Gou, Y., Yu, Z., & Ke, W. (2020). Rock damage control for large-diameter-hole lateral blasting excavation based on charge structure optimization. *Tunnelling and Underground Space Technology*, *106*, 103569. https://doi.org/10.1016/j.tust.2020.103569

[12] Lu, A., Dou, L., Bai, J., Chai, Y., Zhou, K., Kan, J., & Song, S. (2021). Mechanism of hard-roof rock burst control by the deep-hole blasting: numerical study based on particle flow. *Shock and Vibration*, *2021*(1), 9527956. https://doi.org/10.1155/2021/9527956

[13] Liu, Z., Ma, Z., Liu, K., Zhao, S., & Wang, Y. (2023). Coupled CEL-FDEM modeling of rock failure induced by high-pressure water jet. *Engineering Fracture Mechanics*, *277*, 108958. https://doi.org/10.1016/j.engfracmech.2022.108958

[14] Petlovanyi, M. (2016). Influence of configuration chambers on the formation of stress in multi-modulus mass. *Mining of Mineral Deposits*, *10*(2), 48-54. https://doi.org/10.15407/mining10.02.048

[15] 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(51)), 103-112. https://doi.org/10.29227/IM-2023-01-13

[16] Kononenko, M., & Khomenko, O. (2010). Technology of support of workings near to extraction chambers. *New Techniques and Technologies in Mining*, 193-197. https://doi.org/10.1201/b11329-31

[17] Pysmennyi, S., Chukharev, S., Peremetchyk, A., Shvaher, N., Fedorenko, S., & Tien, V. T. (2023, October). Enhancement of the technology of caved ore drawing from the ore deposit footwall "triangle". *IOP Conference Series: Earth and Environmental Science*, *1254*(1), 012065. https://doi.org/10.1088/1755-1315/1254/1/012065

[18] Gao, R., Kuang, T., Meng, X., & Huo, B. (2021). Effects of ground fracturing with horizontal fracture plane on rock breakage characteristics and mine pressure control. *Rock Mechanics and Rock Engineering*, *54*(6), 3229-3243. https://doi.org/10.1007/s00603-020-02294-x

[19] Zhu, Q., Li, T., Lou, Q., Liu, Y., Li, C., & Chen, J. (2023). Stability mechanisms of soft rock mining roadways through roof cutting and pressure relief: An exploratory model experiment. *Frontiers in Ecology and Evolution*, *11*, 1237894. https://doi.org/10.3389/fevo.2023.1237894

[20] Moruzi, G.A., & Pasieka, A.R. (1964). Evaluation of a blasting technique for destressing ground subject to rockbursting. *ARMA US Rock Mechanics/Geomechanics Symposium*, ARMA-64.

[21] Pysmennyi, S., Chukharev, S., Peremetchy, A., Fedorenko, S., & Matsui, A. (2023). Study of stress concentration on the contour of underground mine workings. *Inżynieria Mineralna*, *1*(1(51)), 69-78. https://doi.org/10.29227/IM-2023-01-08

[22] Khomyakov, V.A., Iskakov, E.E., & Serdaliev, E.T. (2013). Investigation of gravelly soil during underground construction in Almaty. *Soil Mechanics and Foundation Engineering*, *50*(4), 171-178. https://doi.org/10.1007/s11204-013-9230-z

[23] Chen, J., Shi, K., Pu, Y., Apel, D. B., Zhang, C., Zuo, Y., & Song, L. (2023). Study on instability fracture and simulation of surrounding rock induced by fault activation under mining influence. *Rock Mechanics Bulletin*, *2*(2), 100037. https://doi.org/10.1016/j.rockmb.2023.100037

[24] 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

[25] Ercins, S. (2024). Vibration prediction with a method based on the absorption property of blast-induced seismic waves: A case study. *Open Geosciences*, *16*(1), 20220633. https://doi.org/10.1515/geo-2022-0633

[26] Rysbekov, K.B., Toktarov, A.A., & Kalybekov, T. (2021). Technique for justifying the amount of the redundant developed reserves considering the content of metal in the mining ore. *IOP Conference Series: Earth and Environmental Science*, *666*(3), 032076. https://doi.org/10.1088/1755-1315/666/3/032076

[27] 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

[28] Serdaliyev, Y.T., Iskakov, Y.E., Bakhramov, B.A., & Amanzholov, D.B. (2024). Eksperimentalnoe issledovanie vozdeystviya vzryvnykh rabot na ustoychivost gornykh vyrabotok. *Gornyy Zurnal Kazahstana*, *1*, 4-10.

[29] Serdaliyev, Y.T., Iskakov, Y.E., Bakhramov, B.A., & Amanzholov, D.B. (2023) Issledovanie seysmicheskogo vozdeystviya vzryva na massiv pri otrabotke malomoshchnykh rudnykh zalezhey. *Gornyy Zurnal Kazahstana*, *9*, 8-10.

[30] Serdaliyev, Y.T., Iskakov, Y.E., Bakhramov, B.A., & Amanzholov, D.B. (2023). Obosnovanie parametrov krepleniya krovli kamer otrabatyvaemykh malomoshchnykh zalezhey kanatnymi ankerami. *Gornyy Zurnal Kazahstana*, *1*, 48-52.

[31] Bondarenko, V., Simanovich, G., Laguta, A., & Cherednychenko, Y. (2011). Mechanism of force interaction of "rock bolt-rocks" system. *Technical and Geoinformational Systems in Mining*, 7-12. https://doi.org/10.1201/b11586-4

[32] Serdaliyev, Y., Iskakov, Y., & Amanzholov, D. (2023). Selection of the optimal composition and analysis of the detonating characteristics of low-density mixed explosives applied to break thin ore bodies. *Mining of Mineral Deposits*, *17*(4), 53-60. https://doi.org/10.33271/mining17.04.053

[33] Latham, J.P., & Lu, P. (1999). Development of an assessment system for the blastability of rock masses. *International Journal of Rock Mechanics and Mining Sciences*, *36*(1), 41-55. https://doi.org/10.1016/S0148-9062(98)00175-2

[34] Navarro, J., Seidl, T., Hartlieb, P., Sanchidrián, J. A., Segarra, P., Couceiro, P., & Godoy, C. (2021). Blastability and ore grade assessment from drill monitoring for open pit applications. *Rock Mechanics and Rock Engineering*, *54*, 3209-3228. https://doi.org/10.1007/s00603-020-02354-2

[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] Azimi, Y., Osanloo, M., Aakbarpour-Shirazi, M., & Bazzazi, A.A. (2010). Prediction of the blastability designation of rock masses using fuzzy sets. *International Journal of Rock Mechanics and Mining Sciences*, *47*(7), 1126-1140. https://doi.org/10.1016/j.ijrmms.2010.06.016

[37] Rakishev, B., Rakisheva, Z.B., Auezova, A.M., & Orynbay, A.A. (2020). Digital hierarchical model of lumpiness of blasted rock mass. *Mining Technology*, *129*(4), 228-237. https://doi.org/10.1080/25726668.2020.1838775

[38] Rakishev, B.R., Shampikova, A.H., & Kazangapov, A.E. (2016). Geometric features of different parts lost in the collapse of the blasted rock. *International Journal of Applied Engineering Research*, *11*(21), 10447-10453.

[39] Yussupov, K., Myrzakhmetov, S., Aben, K., Nehrii, S., & Nehrii, T. (2021). Optimization of the drilling-and-blasting process to improve fragmentation by creating of a preliminary stress in a block. *E3S Web of Conferences*, *280*, 08015. https://doi.org/10.1051/e3sconf/202128008015

[40] Kabetenov, T. (2014). *Sovershenstvovanie skvazhinnoy otboyki pri razrabotke naklonnykh i krutopadayushchikh rudnykh tel maloy i sredney moshchnosti*. Almaty, Kazakhstan: Ekonomika, 192 s.

[41] Sotoudeh, F., Nehring, M., Kizil, M., Knights, P., & Mousavi, A. (2020). Production scheduling optimisation for sublevel stoping mines using mathematical programming: A review of literature and future directions. *Resources Policy*, *68*, 101809. https://doi.org/10.1016/j.resourpol.2020.101809

[42] Choi, Y., Baek, J., & Park, S. (2020). Review of GIS-based applications for mining: Planning, operation, and environmental management. *Applied Sciences*, *10*(7), 2266. https://doi.org/10.3390/app10072266

[43] Jung, D., & Choi, Y. (2021). Systematic review of machine learning applications in mining: Exploration, exploitation, and reclamation. *Minerals*, *11*(2), 148. https://doi.org/10.3390/min11020148

[44] Tambiev, P.G. (2017). *Razvitie vzryvnogo dela v Respublike Kazakhstan*. Almaty, Kazakhstan: ART DO, 424 s.

[45] Nifadyev, V.I., & Kalinina, N.M. (1988). *Nizkoplotnye i sverkhnizkoplotnye vzryvchatye smesi*. Bishkek, Kyrgyzstan: Ilim, 188 s.

[46] Tambiev, P.G. (2015). *Izgotovlenie vzryvchatykh veshchestv iz nevzryvchatykh komponentov i kompleksnaya mekhanizatsiya vzryvnykh rabot*. Almaty, Kazakhstan: KITs TOO, 392 s.

Управління параметрами вибуху для якісного відбивання похилих рудних тіл малої потужності

Є. Сердалієв, Є. Іскаков, А. Алібаєв

Мета. Обґрунтування ефективного способу розробки похилих рудних тіл малої потужності, що залягають у слабких нестійких вміщуючих породах, на основі оптимізації процесу відбивання з урахуванням встановлення закономірностей впливу енергії вибуху на порушений масив зарядами вибухових речовин із регульованою щільністю з урахуванням геомеханічного стану масиву.

Методика. В роботі був використаний комплексний підхід, що включає аналіз літературних джерел, практичний досвід щодо розробки похилих рудних тіл у складних гірничо-геологічних умовах, моделювання енергетичних характеристик вибухів та хвильового впливу на масив із використанням програмного забезпечення, а також проведення дослідно-промислових експериментів на руднику "Акбакай".

Результати. Обґрунтований і розроблений інноваційний метод ефективного та безпечного видобування руди з малопотужних похилих рудних покладів у масивах з ослабленими вміщуючими породами, із використанням нової конструкції зарядами з сумішевих низькощільних вибухових речовин з регульованими у широких межах вибуховими характеристиками, в якому свердловини заряджають і підривають із різним уповільненням. Визначено оптимальні часові інтервали уповільнення, які покращують умови управління енергією вибуху, що здійснюються за рахунок зміни напрямку вектора дії вибуху у бік поверхонь, що знову відкриваються та утворюються в гірському масиві після вибуху зарядів першої черги. Встановлено основні фактори, що впливають на дальність доставки руди при експлуатації похилих малопотужних рудних тіл із системою вибухової доставки та запропоновано методи підвищення ефективності цього процесу.

Наукова новизна. Для умов відпрацювання малопотужних похилих рудних тіл Акбакайського родовища встановлено нові параметри буропідривних робіт, а саме: розроблено раціональну конструкцію заряду з керованими характеристиками вибуху; обґрунтовано оптимальний діапазон щільності заряджання свердловин сумішевими низькощільними вибуховими речовинами та часові інтервали уповільнення; виявлено нову експоненційну залежність дальності доставки руди від питомої витрати вибухової речовини та кута залягання рудного тіла.

Практична значимість полягає в підвищенні ефективності вибухового відбивання корисних копалин, поліпшенні якості вибуходоставки відбитої руди до місць навантаження при збереженні цілісності масиву вміщуючих порід і зниженні збіднення ними рудної маси, усуненні утворення великогабаритних шматків, що ускладнюють вибуходоставку.

Ключові слова: *поклад, видобування, збіднення, втрати, конструкція заряду, енергія вибуху, вибухова речовина, щільність заряду*

Publisher's note