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Geomechanical substantiation of the parameters for the mining system with ore shrinkage in the combined mining of steep-dipping ore bodies

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Abstract

Purpose. The research purpose is to substantiate a rational technology for mining steep-dipping ore bodies based on a complex of geomechanical studies in combined mining of deposits.

Methods. Analysis of existing constructive methods for calculating the optimal mining system parameters when mining under-open-pit ore reserves in the zone of influence of surface mining operations, taking into account the natural stress-strain state of the rock mass. Numerical modeling is used to study the geomechanical processes occurring in the mass during the mining of under-open-pit reserves of steep-dipping ore bodies in order to substantiate the mining system with ore shrinkage. The geotechnical mapping of mine workings is conducted directly in the face to determine the mass rating.

Findings. The calculation of the optimal parameters for the stope chamber, inter-chamber and inter-level pillars based on a complex of geomechanical studies has shown that the more intense horizontal stresses act in the bottom of the blocks and in the inter-block pillars, in which a large number of board gates have been driven.

Originality. For the first time, using high-precision programs and given the nonuniform distribution of horizontal and vertical stresses acting in the mass for the Abyz Mine conditions, it has been revealed that when mining an individual block, the maximum horizontal stresses around the mined-out space reach 10-15 MPa; when mining a group of blocks – 20-25 MPa.

Practical implications. The research results can be used in planning and mining with shrinkage of steep-dipping ore bodies during mining of under-open-pit reserves.

Keywords: *ore, mining system, stope space, inter-chamber pillars, numerical analysis, fracturing, rock mass*

1. Introduction

The use of a combined method for mining steep-dipping ore bodies has become widespread in the mining of minerals, especially at non-ferrous metal deposits, at the mines of Kazakhstan and neighboring countries [1]-[3].

Combined mining of deposits leads to the formation of a complex geomechanical system, a characteristic peculiarity of which is the repeated impact of loads on the same rock mass areas while simultaneously or sequentially conducting of surface and underground mining operations [4], [5]. The accumulated extensive practical experience of domestic and foreign mines confirms the possibility of mining ore by a surface method from the caving zone or from those areas that were previously mined by an underground method [6], [7]. It is well known the experience of mining the reserves by sequential surface and underground mining at the Abyz, Itauyz, Akzhal, Ridder-Sokolsky, Sayak-I, Tishinskoye and other deposits.

A combined mining system was used at the Prince Lyell Mine (Australia) [8], [9]. The mineralized zone, mined by the Prince Lyall mining company, was prepared for underground exploitation in 1972. The ore bodies, represented by a series of parallel lenses, contain 400 million tons of ore with an average copper content of 1%. Their length along the strike is 360 m, the average thickness is 61 m, and the dip angle is 70-80º. The host rocks are mainly represented by quartz-sericite crystalline schist with clearly observed extensive fracturing. The under-open-pit reserves are mined by a system of sublevel caving and a sublevel mining with open stoping. The main part of the reserves is mined by a system of sublevel caving with an end outlet, which eliminates the problem of mining the pillars and requires less volume of preparatory operations.

The Ridder-Sokolsky polymetallic ore deposit [10], located in East Kazakhstan, is confined to the tectonic block anticlinal folds. Ores and host rocks are mostly strong (hardness coefficient according to the Prof. M.M. Protodyakonov scale is 14-18) and stable. The deposit is mined using a combined method by three mines, including Leninogorsk deposit, located in the same vertical plane with the Andreevsky Quarry. Underground operations for mining high-grade ores be-

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gan in the 30-40[s] before the beginning of surface mining operations. Then, the inter-chamber pillars, flank zones, block bottoms and part of the underlying reserves were mined by a surface method, while conducting underground mining with caving of ore and host rocks. Part of the cave-in funnels during the release of near-wall reserves went into the open-pit non-working zone.

The Chambishi Mine (Zambia) [11], [12] mines a copper schist deposit 2.1 km long and 15-20 m thick by a sequential surface and underground method. The ore body dip angle in depth varies from 30 to 70°. The upper deposit part to a depth of up to 235 m is mined by a surface method. The final open-pit dimensions in plan are 1360×630 m. The total ore reserves within the open-pit contour are 18.9 million tons, overburden – 132.3 million tons, and under-open-pit reserves – 638.31 million tons.

The Helen Mine (Canada) [13], [14] mines an iron ore deposit with a thickness of 18 to 90 m with a dip angle of 70-80°. Mining is conducted by a surface method up to the level of 422 m, after which a gradual transition to underground mining of out-contour reserves is performed. The border reserves are mined by chambers open into an open-pit, the upper part of which is drilled from the open pit by downward wells. Thick inter-chamber pillars are left between the open chambers.

The Flin Flon Mine (Canada) [15], [16] mines a steep-dipping deposit, folded by hard massive sulphide ores. The upper part of the deposit to a depth of 84m is mined with an open pit. The open pit and the underground mine are separated by a horizontal ore barrier pillar 40m thick. Under-open-pit reserves are mined with an open chamber 60 m high with dimensions in plan of 30×66 m. In underground mining operations, chamber systems with sublevel ore breaking are used, the width of the chambers is 30 m, the length is up to 80 m, the height is 60 m. Vertical pillars 12-24 m thick are left between the chambers.

The Gunnar Mine (Canada) [17] mines a uranium deposit located on the Lake Athabasca shore in the Saskatchewan Province. The deposit is represented by syenites. Its dip angle relative to the lake surface is 45°. Near the earth's surface, the ore body is monolithic, in plan it has the shape of a circle with a diameter of about 135 m, and below the eighth horizon (345 m) it is lenticular. From 1955 to 1958, all ore was mined only by a surface mining method. In 1961, an underground mine was put into operation. An open chamber mining system with ore breaking in deep wells and backfilling the mined-out area was used in underground mining operations. In the process of mining the chambers, the broken ore was partially shrinked, and after complete mining, the formed cavities were filled with hydraulic backfill or dry overburden rock. This made it possible to perform selective mining of ores with a relatively high content of useful components and maintain the non-working open-pit wall in a stable state. In addition, the backfilling allowed mining of inter-chamber pillars. The interesting solution was to backfill the chambers with large rock, as well as to strengthen the latter with classified beneficiation tailings.

The Tishinskoye deposit (East Kazakhstan) [18], mined in parallel with an open pit and underground mine, is represented by steep-dipping polymetallic ore bodies with a thickness of 5 to 70 m. In 1976, mining operations in the open pit were completed. Ore bodies in the western open-pit wall are mined by a level-chamber, as well as cut and fill mining system using an uncemented rockfill. Preparatory workings are conducted from adits, driven from the open-pit benches.

In all the above-mentioned deposits, mining systems of sublevel caving with deep blast holes were used, which resulted in the caving of the host rocks due to the impact of the blast force on the rock mass [19].

At present, underground mining at the Abyz deposit is conducted using self-propelled machinery with a system of sublevel caving in the southern area directly under the walls and bottom of the mined-out open pit, previously flooded with water. Blast-hole breaking of low-thickness ore bodies leads to high dilution. In addition, when driving drilling-loading drifts of large section for self-propelled machinery, ore often falls out from the roof, as well as rocks of the walls along the main fracture systems oriented along the strike of the ore. It is especially difficult to keep the junctions of mine workings. Therefore, the decision of transition to a mining system with blast-hole breaking and ore shrinkage is absolutely expedient.

Underground mining operations conducted in the under-open-pit zone changes the stress-strain state of the near-wall mass, the strength characteristics of the host rocks, and, consequently, the current ratio of retention and shear forces in the wall mass, as well as the sliding surface position [20]-[24]. In this regard, the design of combined technologies and the selection of their rational parameters based on geomechanical research is an urgent task for both researchers and producers.

2. Research methods

The natural stress state of the Abyz Mine mass has not been measured before. Therefore, its assessment can only be performed using the data of the international World Stress Map (WSM) project. In the area adjacent to the Balkhash District, the natural stress state of the mass is characterized by the Strike-Slip (SS) – shift tectonic condition (Fig. 1) [25].

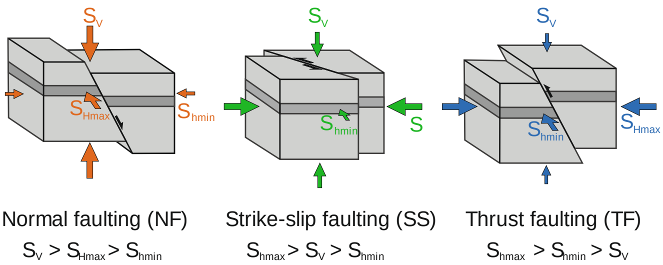

***Figure 1.** Three main tectonic conditions of the natural stress state of the mass, differing in the directions of acting principal stresses*

In a number of indirect signs at the Abyz Mine, the natural horizontal stress values in the mass σ_1 and σ_3 are close to the vertical γH, but slightly different from them. For further calculations, the following is taken:

– maximum tectonic stresses along the strike of ore bodies $\sigma_1 = 1.1\ \gamma H$;

– intermediate in value principal stresses are vertical (gravitational) $\sigma_2 = \gamma H$;

– minimum horizontal stresses across the strike of ore bodies $\sigma_3 = 0.9\gamma H$.

The specific weight of host rocks is taken as $\gamma = 2.7$ t/m^3.

The mass fracturing on the upper horizons of the Abyz deposit was studied in sufficient detail at the exploration stage in the underground mine workings. Using the Dips software [26], 4 fracture systems have been reliably identified. They are shown in Figure 2 and in Table 1.

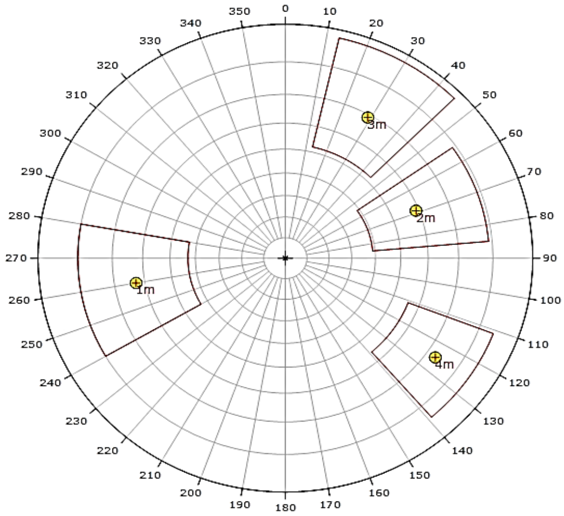

Figure 2. Fracture systems identified on the upper horizons at the stage of detailed exploration

Table 1. Characterization of fracture systems based on survey data

Fracture system	Dip angle, deg		Dip azimuth, deg		Average distance, cm
	Average	±	Average	±	
1	63	24	260	20	30
2	59	19	69	14	20
3	69	21	29	13	35
4	73	15	125	10	15

The fractured mass properties are determined by the Hoek-Brown failure criterion (25) using the geological strength index (*GSI*) [27] and the RocLab program [28].

The geological strength index (GSI) is calculated by the Formula:

$$GSI = \frac{52J_r / J_a}{\left(1 + J_r / J_a\right)} + \frac{RQD}{2}, \qquad (1)$$

where:

J_r, J_a – index of roughness and variability/filling of fractures according to N. Barton [29], [30]. For the conditions of the Abyz deposit, the following is taken: $J_r = 1.5$; $J_a = 2$. Hence, the geological strength index approximate value is $GSI = 47$.

Table 2 summarizes the mechanical properties of the mass. These rock properties are initial data for numerical modeling.

The Phase 2 program is used to perform numerical modeling [31]. Phase 2 is a program for 2D finite element ana-lysis of geotechnical structures at mining enterprises. It is applicable for both surface and underground mining operations.

In world practice, numerical modeling is widely used to determine the rock mass stress-strain state [32]. When modeling in the Phase 2 program, the structural characteristics and mechanical properties of rocks are taken into account, which increases the calculation accuracy.

Table 2. Mechanical properties of fractured masses

Geomechanical domains	Basalts and andesites	Lava-breccia puff-stone	Metasomatite	Gabbro-diorite diorite
Lithology codes	1	2 + 3	4	5
Rock properties in samples (intact properties)				
Unit specific gravity	2.7	2.7	2.7	2.7
Compressive strength, MPa	50	74	48	41
Rock type coefficient	25	19	10	
Elasticity modulus, GPa	66	63	60	61
Poison's ratio	0.27	0.23	0.2	0.27
Fractured rock mass properties (rock mass parameters)				
GSI	47	47	47	47
Disruption after blasting operations *D*	0	0	0	0
Tensile strength, MPa	-0.04	-0.07	-0.05	-0.03
Compressive strength, MPa	13	17	11	10

3. Results and discussion

3.1. Numerical modeling of the mass state when mining of under-open-pit reserves

Numerical modeling performed using the Phase 2 program [31] indicates that when mining steep-dipping ore bodies at a low depth (about 200 m) under the mined open pit, the values of the acting maximum stresses in the hanging wall of the mined blocks are small, amounting to 1.0-1.5 MPa (Fig. 3).

As a result of modeling, a dependency graph of the change in maximum stresses in the rock mass on the increase in the mining depth has been constructed (Fig. 4).

Based on the data obtained, the change in maximum stresses during mining of under-open-pit reserves (Fig. 4), it can be stated that the stress intensity changes linearly depending on the open-pit depth, reaching a value of 13.6 MPa for vertical stresses and 10.0 for horizontal stresses.

Also, with depth, there is a uniform correlation of proportional change between the maximum vertical stresses and the maximum horizontal stresses, which ranges within 25-35%.

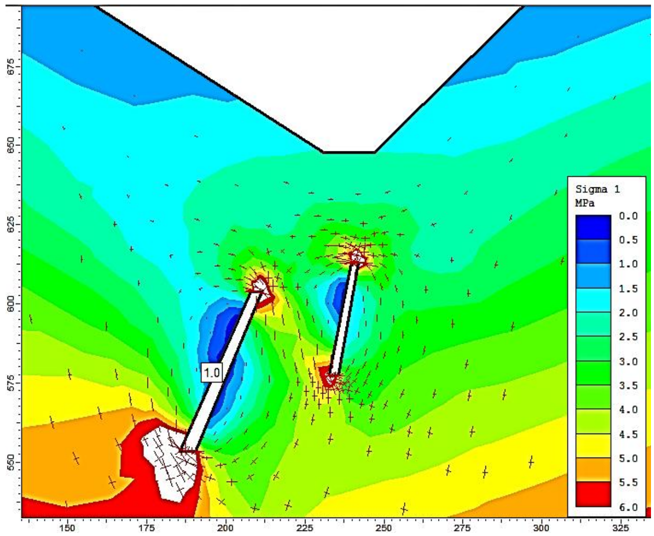

Figure 3. Distribution of maximum stresses during mining of under-open-pit reserves

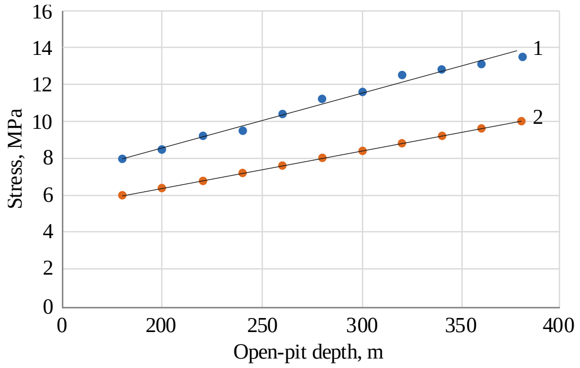

Figure 4. Graph of the change in maximum stresses during mining of under-open-pit reserves: 1 – maximum vertical stresses; 2 – maximum horizontal stresses

These values should be taken into account when calculating permissible parameters of the stope blocks, which are mined by the system with shrinkage.

3.2. Determining the parameters for the mining system with shrinkage

The Mathews-Potvin method [33]-[35] is used to calculate the permissible dimensions of stope blocks, based on the calculation of two parameters:

– hydraulic radius *HR*, which describes the size of outcrops (an analogue of *HR* is the equivalent half-span of an outcrop 0.5 L_e);

– stability index *N*, which reflects the ability of rock mass outcrops to maintain stability in a given stress state.

The hydraulic radius *HR*, which is defined as the ratio of the hanging wall (or roof) outcrop area to its perimeter, is determined from the Expression:

$$HR = \frac{w \cdot h}{2(w+h)}, \tag{2}$$

where:

h, *w* – block span to the dip and strike, m.

The stability index is calculated by the Formula:

$$N = \frac{RQD}{J_n} \cdot \frac{J_r}{J_a} \cdot A \cdot B \cdot C, \tag{3}$$

where:

RQD – index of rock mass destruction by fractures;

J_n – index of the number of fracture systems, considering the number of fracture systems;

J_r – fracture roughness index;

J_a – fracture surface change index;

coefficients that take into account:

A – rock strength and stress state;

B – orientation of fractures (disturbances) in relation to the hanging wall outcropping;

C – ore body dip angle *α* (hanging wall outcrop slope).

The use of a mining system with shrinkage is allowed for the host rock stability ranging from medium to stable. Therefore, the dimensions of the stope blocks should be determined by calculation so that the hanging wall outcrop remains stable. Therefore, the permissible spans of the stope chambers to the dip and along the strike of ore bodies should be determined from the Expression [36], [37]:

$$HR_{per} = \frac{w \cdot h}{2(w+h)} \le 0.32N + 4.3, \tag{4}$$

For the conditions of the Abyz Mine, based on the collected geomechanical data and recommendations [38], it should be taken: $RQD = 47\%$; $J_n = 9$; $J_r = 1.5$; $J_a = 2$; $A = 1$; $B = 0.3$. The calculated values of stability index *N* and permissible values of HR_{per} depending on the dip angle *α* of ore bodies are given in Table 3.

Table 6. Permissible outcrops of the hanging wall chambers

α	60°	65°	70°	75°	80°	85°
C	5.0	5.5	5.9	6.4	7.0	7.5
N	5.9	6.5	6.9	7.5	8.2	8.8
HR_{per}	6.2	6.4	6.5	6.7	6.9	7.1

3.2.1. Determining the height of the inter-level pillar

Structural elements of a mining system with shrinkage are shown in Figure 5 with the parameters taken at the Belousovsk and Irtyshsky mines. Mining systems with shrinkage have been widely used in mining of vein deposits of rare metals and gold. It should be noted that in recent years the scope of these systems has been significantly expanded due to the creation of new types of fastening.

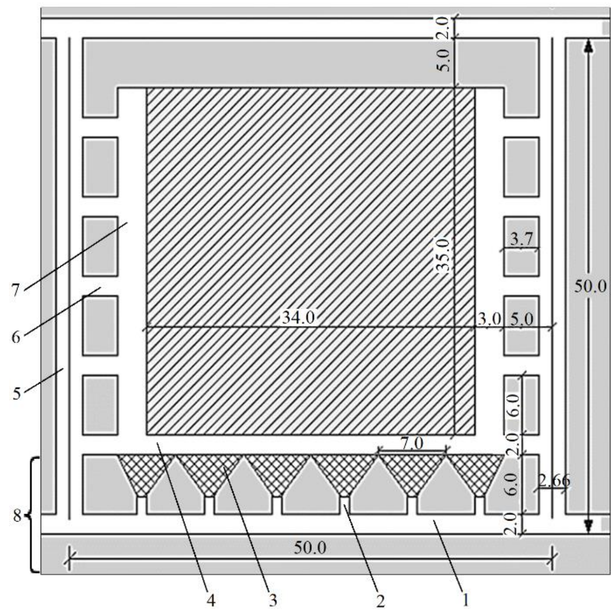

Figure 5. Mining system with ore shrinkage: 1 – scraper drift; 2 – boxholes; 3 – funnel; 4 – horizon of funnels; 5 – VHV; 6 – air passages; 7 – cut-out raise; 8 – the inter-level pillar

A distinguishing peculiarity of mining systems with ore shrinkage is filling of the mined-out space with broken ore, which serves to support the host rocks or is used as a kind of platform for workers. In all cases, after the block is mined, the broken ore is completely drawn. When using these systems, it should be borne in mind that the broken ore occupies a larger volume than in the mass. This should be taken into account when determining the required volume of compensatory space [39].

The inter-level pillar consists of the bottom of the overlying level chamber with a height h_{bott} ~ 6-7 m (the pillar above the drift) and the ceiling of the underlying level chamber (the pillar under the drift) h_{ceil} = 5 m. Taking into account the bottom weakening by the outlet mine workings, the inter-level pillar height, according to the experience of the Belousovsk and Irtyshsky mines, is recommended to take equal 11-12 m. The same ceiling is recommended to leave between the mined block and the bottom/wall of the mined-out open pit.

3.2.2. Determining the parameters for the inter-chamber pillars

When mining thin, steep-dipping deposits, the maximum stresses in the load-bearing elements act subhorizontally – along the normal to the dipping ore bodies. In order to determine the most stressed mining system elements, the distribution of stresses normal to the plane of ore bodies has been calculated.

Since the extraction thickness m of the stope chambers is much less than their dimensions to the dip h and strike w ($m << h \sim w$), they can be modeled with thin slots lying approximately in the same plane. Therefore, to describe the distribution of stresses normal to the plane of ore bodies, it is possible to use analytical solutions implemented in the engineering calculation method – the method of successive cycles [40]. Its essence is in the distribution of initial loads from the mined-out mass areas (stope chambers) to the left pillars, backfilling (shrinked ore) and the surrounding mass (Fig. 6).

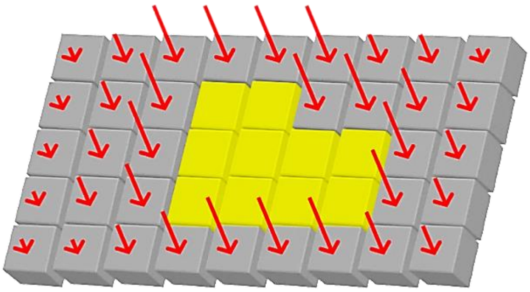

Figure 6. Calculation scheme for the method of successive cycles

The equivalent elasticity modulus of granular medium is used in the mined-out block area to calculate the shrinked ore repulse reaction [41]. In this paper, the equivalent elasticity modulus for the shrinked ore, based on the conclusions from the research on granular media by K.V. Ruppeneit amounts to ~100 MPa, which is ~2% of the ore mass elasticity modulus.

The loading of the mining system elements with shrinkage is calculated for the Abyz Mine conditions: depth – 200 m; specific weight of rocks – 2.7 t/m^3; coefficient of lateral pressure across the strike of ore bodies – 0.9; dip angle of ore bodies – 80°. The calculation results for an individual stope block are shown in Figure 7.

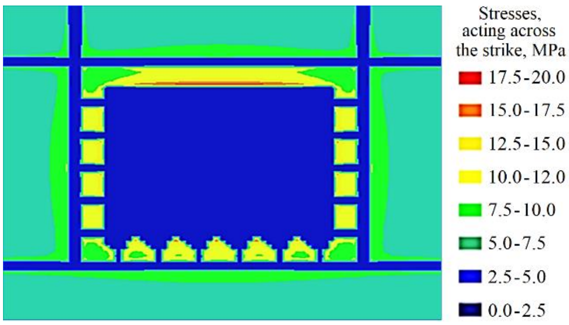

Figure 7. Distribution of stresses for mining systems with ore shrinkage

According to the calculation results, the more intense horizontal stresses act in the bottom of the blocks and in the inter-block pillars, in which a large number of board gates (boxholes, outlet funnels, material-manway raises, ventilation-manway cross slits from raises to blocks) have been driven. When mining an individual block, the maximum horizontal stresses around the mined-out space reach 10-15 MPa; when mining a group of blocks – 20-25 MPa. The thrust of the shrinked ore in the block reaches 0.3-0.5 MPa (30-50 t/m^2). Due to this thrust, the host rocks are kept from falling out.

The ore mass strength is assessed using the Hoek-Brown failure criterion and the geological strength index GSI. The ore strength in the samples is taken as σ_o = 120 MPa, GSI = 70, m = 25, then the strength of the fractured ore mass is σ_m = 50 MPa.

For the verification calculation of the safety factor of inter-block pillars, the stresses σ_d acting in them are determined by the Formula:

$$\sigma_d = \gamma H \cdot \left(\cos^2\alpha + \lambda \sin^2\alpha\right) \cdot \frac{1}{K_0} \cdot \frac{w+x}{x}. \tag{5}$$

where:

x – inter-block pillar width;

K_o – coefficient of the pillar weakening by board gates: raise and ventilation-manway cross slits [42]:

$$K_o = \frac{\left(x - x_b - 0.4\right)\cdot\left(h_{pr} - h_r - 0.4\right)}{x \cdot h_{pr}}. \tag{6}$$

where:

x – design width of the pillar;

x_r = 2.65 m – the pillar raise width;

h_{pr} = 8 m – distance in height between the axes of cross slits driven from the raise;

h_r = 2.0 m – height of cross slit in the pillar;

0.4 m – depth of the ore mass destruction during shallow blast-hole blasting.

With a pillar width of x = 10 m, coefficient of its weakening with board gates is equal to K_o = 0.48; at x = 15 m – K_o = 0.6.

The following values are used in the verification calculation: γ = 2.7 t/m^3; H = 200 m; α = 80°; λ = 0.9; σ_m = 50 MPa. If the span of the stope chambers in the clear is taken as w = 40 m, then an acceptable safety factor for the pillars $n = \sigma_m / \sigma_d$ = 1.7 is obtained only with a pillar width of x = 15 m.

For ore bodies with dip angles α = 60-65°, the permissible equivalent half-span of the hanging wall outcrop is HR_{per} = 6.2-6.4 m. With a chamber height in the clear (dip span) h = 40 m, the permissible span of the chambers along the strike can be taken as w = 20 m. Then the safety factor n = 1.6 is provided with a width of the inter-block pillars x = 10 m.

In the future, for a more accurate calculation of the parameters for the mining system with shrinkage, it is necessary to conduct a complex of geomechanical studies on the zoning of the deposit depending on the stability rating, thereby determining in advance more accurate parameters for the mining system used.

4. Conclusions

A detailed analysis of using combined mining system in domestic and foreign deposits has been performed. The disadvantages are noted of a mining system of sublevel caving with drilling of deep wells when mining underground reserves.

According to the kinematic analysis results in the Dips software and from the linear survey of fractures, 4 main fracture systems have been identified that have a dip angle ranging within 59-73 degrees, a dip azimuth ranging within 29-260 degrees and an average distance ranging within 15-35 cm.

A dependency graph of the change in the maximum horizontal and vertical stresses with an increase in the mining depth from 120 to 380 m has been constructed, based on the results of the rock mass numerical modeling when mining under-open-pit reserves.

On the basis of the performed studies and using the Mathews-Potvin method, the parameters for mining systems with shrinkage of stope blocks, inter-chamber and inter-level pillars have been substantiated. When mining an individual block, the maximum horizontal stresses around the mined-out space reach 10-15 MPa; when mining a group of blocks – 20-25 MPa. An acceptable safety factor for the pillars $n = \sigma_m / \sigma_d = 1.7$ is obtained only with a pillar width of x = 15 m and the span of the stope chambers in the clear of w = 40 m.

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Геомеханічне обґрунтування параметрів системи розробки з магазинуванням руди при комбінованому відпрацюванні крутопадаючих рудних тіл

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Мета. Обґрунтування раціональної технології відпрацювання крутопадаючих рудних тіл, що круто падають, на основі комплексу геомеханічних досліджень при комбінованому відпрацюванні родовищ.

Методика. Зроблено аналіз існуючих конструктивних методів розрахунку оптимальних параметрів системи розробки при відпрацюванні підкар'єрних запасів руди у зоні впливу відкритих гірничих робіт з урахуванням природного напружено-деформаційного стану масиву гірських порід. Дослідження геомеханічних процесів, що діють у масиві, проводилося із використанням чисельного моделювання (Phase 2) при відпрацюванні підкар'єрних запасів крутопадаючих рудних тіл для обґрунтування системи розробки з магазинуванням руди. Проведення геотехнічного картування гірничих виробок здійснювалося безпосередньо у вибої для визначення рейтингу масиву.

Результати. Розрахунок оптимальних параметрів очисної камери, міжкамерних і міжповерхових ціликів на основі комплексу геомеханічних досліджень показав, що найбільші горизонтальні напруження діють у днищі блоків і в міжблокових ціликах, в яких пройдено велику кількість нарізних виробок. Обґрунтовано параметри систем розробки з магазинуванням очисних блоків, міжкамерних та міжповерхових ціликів на основі виконаних досліджень і використання методики Метьюза-Потвіна.

Наукова новизна. Вперше із застосуванням високоточних програм і врахуванням нерівномірності розподілу горизонтальних і вертикальних напружень діють у масиві для умов рудника Абиз встановлено, що при відпрацюванні окремого блоку максимальні горизонтальні напруження навколо виробленого простору досягають 10-15 МПа; при відпрацюванні групи блоків – 20-25 МПа.

Практична значимість. Результати досліджень можуть бути використані при плануванні та відпрацюванні крутопадаючих рудних тіл із магазинуванням при відпрацюванні підкар'єрних запасів.

Ключові слова: *руда, система розробки, очисний простір, міжкамерні цілики, напруження, чисельний аналіз, тріщинуватість, гірський масив*