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Reduction of ore dilution when mining low-thickness ore bodies by means of artificial maintenance of the mined-out area

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Abstract

Purpose. The research purpose is to study the effectiveness of artificial maintenance of the mined-out space based on the use of cable bolts to reduce the dilution coefficient when mining low-thickness ore bodies.

Methods. Geotechnical mapping of the rock mass according to the Q, RMR, RQD and GSI rating classifications is conducted, as well as a linear survey of the fracture system in the hanging wall and footwall rocks is performed using a rock compass and the GEO ID application. Numerical analysis by the limit equilibrium method in the Unwedge software package is applied to determine the safety factor of a mass divided by fractures into wedges. Using a Schmidt test hammer, the uniaxial compressive strength of the mass rocks has been determined. The full-scale studies have been conducted using cable support in the conditions of the Akbakai deposit.

Findings. It has been revealed that the footwall rocks are in a stable state, while the safety factor of the hanging wall rocks is 0.98, which requires artificial maintenance using cable bolts. The cable support parameters are calculated taking into account nonuniform distribution of horizontal and vertical stresses in the rock mass. It has been determined that when strengthening the hanging wall with cable bolts in inclined veins with a dip angle of up to 40º, the average ore dilution is 66.1%, and that of previously mined without fastening is 68.7%. In similar experiments in steep-dipping veins with a dip angle of more than 60º, dilution decreases from 62.8 to 48.7%.

Originality. It has been revealed that in the conditions of the Akbakai deposit, cable fastening of the hanging wall rocks is effective at an ore deposit dip angle of more than 60º, at which the mined ore dilution coefficient decreases.

Practical implications. The research results can be used to increase the stability of hanging wall rocks when mining low-thickness ore bodies with a sublevel caving system.

Keywords: *dilution, ore, cable fastening, stope area, rocks, fracturing, dip angle*

1. Introduction

Kazakhstan is very rich in mineral resources. Oil, coal, various ore, and non-metallic deposits are the priceless treasure of the republic. Some of these mineral resources make Kazakhstan famous in the world. They include chrome iron ore deposits, polymetallic deposits, copper, tungsten, molybdenum and uranium ores [1]-[5]. At the same time tining enterprises need to develop deposits based on the efficient use of natural, material and labor resources. The practical implementation of this market economy provision depends to a large extent on solving technological problems conditioned by the need for continuous improvement and technical re-equipment of the mining complex [6]-[8].

Ore dilution leads to a loss in the mineral quality during the mining process, resulting in a decrease in the useful component content in the mined mineral mass compared to its content in the mined deposit [9]-[11]. This is usually caused by ore dilution with waste rocks or backfill mass caving [12]-[14]. The low-thickness ore bodies tend to have a complex structure with possible swells and twitches. When mining low-thickness ore deposits with a sublevel caving system with end ore drawing, the probability of a decrease in the mineral content increases [15], [16]. For example, the actual ore dilution at the Akbakai deposit of JSC AK Altynalmas reaches 70% or more. Table 1 summarizes the ore dilution coefficients of some gold mines mining low-thickness ore bodies.

Most of the low-thickness deposits are characterized by a complex morphology of ore bodies, the presence of parallel fractures, tectonic disturbances, and a variety of physical-mechanical properties of the ore and host rocks. The low ore body thickness, combined with difficult conditions of occurrence, causes significant ore dilution.

With low-performance mining systems (layered mining with backfilling of the mined-out area, mining with ore shrinkage), dilution is always lower than with high-performance systems with open stope space (level caving, mining with sublevel drifts, etc.) [17]-[19].

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Table 1. Ore dilution coefficients at some mines

Mine name	Ore body thickness, m	Stope space width, m	Ore dilution, %
Copper Cliff Mine (Canada)	1.24	3.13	62.6
Dugald River (Australia)	2.15	4.23	47.0
Zholymbet (Kazakhstan)	1.55	6.15	74.7
Akbakai (Kazakhstan)	1.0	2.85	64.9
Shaumyan, Armenia	0.7	2.05	60.9
Cracow, Australia	1.0	2.18	54.1

Naturally, when mining thick, homogeneous ore deposits, dilution is possible only at the contacts of the ore deposit with its host rocks, and in general, dilution in the ore body will be minimal. In low-thickness ore deposits, the value of dilution becomes higher due to the caving of the near-ore zone host rocks due to various factors [20], [21].

The dilution coefficient value of low-thickness ore deposits mainly depends on the mining system used, the structural and strength properties of the mass, the impact of the blasting force on the host rocks, the use of artificial maintenance of the mined-out space, the natural rock mass stress field, the geometric characteristics of the vein, the thickness and dip angle of the ore body [22]-[26].

Many domestic and foreign authors have studied the problem of ore dilution. Despite a large amount of theoretical and experimental research, there is still no scientifically-based approach to manage the ore dilution when mining low-thickness ore deposits with a system of sublevel caving with end drawing of ore. Reducing the ore dilution requires a comprehensive study of the structural, strength properties and the stress-strain state of the rock mass, drilling and blasting operations, leaving protecting pillars, maintaining near-ore rocks with a support [27], [28].

Despite certain successes in creating new mining systems and improving their individual structural elements, the technology of stoping extraction of low-thickness ore deposits is characterized by a low level of mechanization of the main technological mining processes, as well as high material and labor costs for their implementation. The relative importance of each process in terms of labor intensity and production cost depends on the mining system used in the relevant mining-geological conditions, and varies widely [29], [30].

The breaking of ore deposits to a limited free surface is accompanied by additional edge destruction of the mass rocks, which has a significant impact on the formation of the mined ore quality [31]. Therefore, the study of ore dilution when mining low-thickness deposits is an important link in the chain of tasks to be solved on improvement of mining systems.

In overseas mines of Australia (MMG Limited Dugald River) and New Zealand (OceanaGold Waihi), cable bolts are widely used to support hanging wall and footwall in order to reduce over-planning ore dilution. Foreign researchers N.R. Barton, R. Lien, S.D. Nickson, R. Hassell in their works [32], [33] discuss the effectiveness of using cable fastening to manage ore dilution in systems with open stope space.

The authors in the studies [34], [35] argue that, when adhered to the fastening technology, cable bolts are an effective method to manage ore dilution when mining low-thickness ore deposits by mining systems with an open stope space. D.R. Chinnasane, M. Knutson and A. Watt in their studies [36] state that at the Copper Cliff Mine (Canada), the cable bolts are only used to temporarily maintain the mined-out space, and then, during mining, the hanging walls should be additionally supported by backfilling. Foreign researchers Hutchinson and Diederichs [37] provide an overview of how and where cable bolts can be used to support, strengthen or hold rock mass around the majority of mine workings in an underground mine.

Depending on the tasks set for studying or improving the technology for underground mining of ore deposits, the authors of the conducted research distinguish certain of its most significant peculiarities, based on which it possible to study in detail the individual elements of the constituent links of the general technological scheme for ore mining. Moreover, the entire complex of mining production technological processes has been improved, which, combined with the rational organization of labor at mining enterprises, gives a great technical and economic effect.

Based on the results of foreign researchers, it should be assumed that cable fastenings are one of the most effective methods for reducing ore dilution in underground mining of low-thickness deposits by a sublevel caving system. Thus, the analysis of literature sources confirms the effectiveness of using the cable bolts in order to maintain the stope chamber out-contour mass in a stable state.

To date, ore dilution is an unresolved problem in almost deposits. The consequences of dilution lead to an increase in the cost of transporting and processing of ore, thereby increasing the cost of minerals [38], [39]. Therefore, reduction of ore dilution when mining low-thickness ore bodies by means of artificial maintenance of the mined-out space is an urgent task requiring a complex of research and practical work.

2. Research methods

The low-thickness Akbakai deposit has been selected for research and experimental-industrial testing.

An underground mining method with horizons of 60 m and inclined ramps was used at the deposit.

The thickness of the veins is low and ranges from 0.5 to 3.0 m. The average thickness is about 1.7 m. By the dip angles, the veins are conventionally divided into two groups:

– steep-dipping with a dip angle of 55-85º;

– inclined, with a dip angle of 30-50º.

For these conditions, the most suitable systems are those with an open stope space and a continuous mining, as well as sublevel drifts with end ore drawing. The best technical and economic indicators are achieved when using a sublevel blast-hole stoping system of mining with an end ore drawing.

With this system, the vein is divided along the strike and to the dip into blocks with the following parameters:

– block length along the strike of the vein is 100 m;

– the block height along the strike to the entire level height to the dip of the ore body, broken down into sublevels (stopes) L – 10.0 m.

The Akbakai deposit reserves are characterized by low-thickness veins (0.5-2.0 m) and dip angles from 32º. The sublevel blast-hole stoping system by layer-by-layer breaking of ore with deep wells and delivery by the blasting force provides for the mining of 2 types of veins according to the following dip angles:

– inclined from 32 to 55º;

– steep-dipping from 55º.

When breaking ore, ore dilution of more than 60% occurs with a design dilution of no more than 38%. Dilution occurs due to rock ingress, mainly from the hanging wall. Thus, fractures in the rock mass form bedding planes. They are generally sub-parallel with respect to the occurrence of the ore body, prone to cleavage and caving during mining of mineral reserves.

In order to reduce the ore dilution at the Akbakai deposit, it has been decided to conduct industrial tests to determine the effectiveness of cable bolts. For conducting the experiments, a local area of 30 meters in length on the No. 13 sublevel drift of the Pologaya vein has been selected, the sketch in plan is shown in Figure 1, in section it is shown in Figure 2. The vein dip angle is not more than 40º.

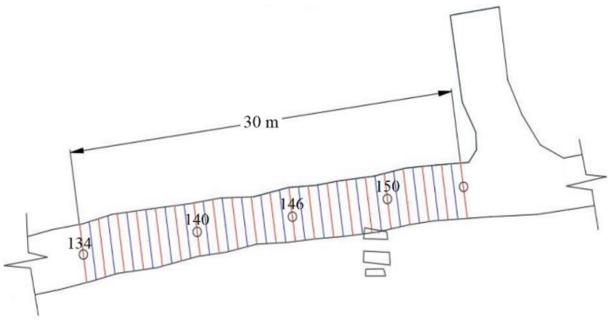

Figure 1. Cable fastening test area

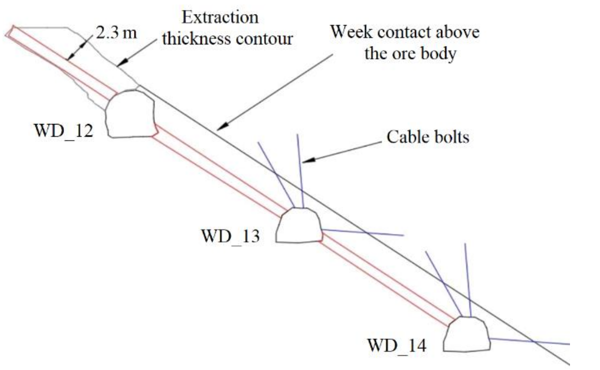

Figure 2. View of an area for testing on cable fastening in section

At the experimental-industrial test site, the rock mass geotechnical mapping has been conducted according to the Q, RMR, RQD and GSI rating classifications, as well as a linear survey of fractures has been performed using a rock compass and the GEO ID application [40]. Using a Schmidt test hammer, the uniaxial compressive strength of the rocks directly in the mass has been determined. To substantiate the cable fastening parameters, the natural rock mass stress state has been measured [41].

When planning underground mining operations, the rock mass stress-strain state is of paramount importance for ensuring safety and predicting the stability of mine workings [42], [43]. There are a number of methods that make it possible to conduct in-situ measurements of the rock mass geomechanical parameters. At the same time, in many cases, the validity of the stress values obtained remains an open question due to the incorrect formulation of the inverse problem, in which one or another method finds a theoretical substantiation [44].

The analysis of literature sources [45] shows that the most effective method is the hydraulic well fracturing for determining the values of the principal stresses acting in the mass. Classical analysis of the hydraulic seam fracturing is based on the Kirsch solution for stress distribution around a circular hole in a homogeneous, isotropic, elastic material exposed to high compressive stresses [46]. The main research objective when using the hydraulic fracturing method is to determine the value and directions of the principal stresses and the natural stress field acting in the rock mass for further use in planning [47]-[48].

In order to measure the stress, 2 wells are drilled to a depth of about 100 m with a diameter of 96 mm. The depth of mining operations is 460 m at the S_ZL70_15_GM1 well location and 340 m at the S_FR76_28_GM2 well location.

As mentioned above, the classical analysis of the hydraulic fracturing method is based on the Kirsch solution for stress distribution around a circular hole in a homogeneous, isotropic, elastic material exposed to high compressive stresses. In the case of a vertical well, the Kirsch solution uses the Hubbert and Willis formula (1957) [49] for the critical pressure *PC* at the time of beginning destruction (1):

$$PC = 3 \cdot Sh - SH + T - Pp, \quad (1)$$

where:

Sh and *SH* – horizontal principal stresses;

T – hydraulic tensile strength of rock in a mass;

Pp – pore pressure in the rock mass.

The fracture opening stress is assumed to be the principal stress, the rock is homogeneous, isotropic and initially impermeable, and the induced fracture is oriented perpendicular to the minimum horizontal principal stress *Sh*. The last assumption gives the following Expression (2):

$$Sh = Psi, \quad (2)$$

where:

Psi – the shut-off pressure that simply keeps the fracture open after the pressurization system is closed.

In total, 18 stress field measurements have been performed at the Akbakai deposit.

If to take into account the nonuniform distribution of horizontal and vertical stresses, it is possible to improve the safety of conducting mining operations and increase the reliability of the results obtained by numerical analysis methods based on rock mass modeling using high-precision programs [50].

The following physical-mechanical parameters have been determined for accuracy of calculations:

– uniaxial compressive strength (UCS);

– Young's modulus and Poisson's ratio (UCS + YP);

– uniaxial tensile strength of rocks (UTS-Brazilian test);

– triaxial compressive strength (TXT) with construction of a strength passport and calculation of adhesion and internal friction angle;

– ultimate strength for direct shear along a natural fracture (SOJ – three-stage shear) with the construction of a strength passport, as well as the calculation of adhesion and the internal friction angle.

Strength and deformation characteristics are determined according to ISRM "Suggested Methods for Determining the Uniaxial Compressive Strength and Deformability of Rock Materials" and GOST 28985-91 "Mine Rocks. Method for determining deformation characteristics in uniaxial compression" [51]. The essence of the method consists in measuring the compressive force applied to the ends of a cylindrical sample, its longitudinal and transverse deformations caused by this force.

The ultimate strength values are determined by Formula (3):

$$\sigma_1 = \frac{F}{S} + \sigma_3 \cdot \left(1 - \frac{S_s}{S_p}\right), \tag{3}$$

where:

F – vertical load, kN;

σ_3 – confined pressure in the stope, MPa;

S_s – cross-sectional area of the sample, cm^2;

S_p – cross-sectional area of the piston, cm^2.

The characteristics of adhesion and the internal friction angle as parameters of a linear dependence (Coulomb-Mohr criterion) are determined by the Formula (4):

$$\tau = \sigma \cdot tg\varphi + C, \tag{4}$$

where:

τ – shearing stress, kPa;

σ – normal stress;

φ – internal friction angle;

C – adhesion, kPa.

The GSI value is determined as a result of the well data analysis and geotechnical mapping of mine workings. The Barton-Bendis criterion is used to determine the mass rock shear strength (5):

$$\tau = \sigma_n \tan\left[\varphi_b + JRC \log_{10}\left(\frac{JCS}{\sigma_n}\right)\right], \tag{5}$$

where:

σ_n – normal stress, MPa;

φ_b – internal friction angle;

JRC – fracture roughness coefficient;

JCS – fracture wall compression strength;

3. Results and discussion

According to the data obtained from the S_ZL70_15_GM1 well, at a depth of 521.3 ± 34.0 m, the vertical principal stress value is $\sigma_1 = 13.8 \pm 0.9$ MPa (Sv), the minimum horizontal stress value is $\sigma_3 = 7.0 \pm 0.7$ MPa (Sh), the maximum horizontal stress value is $\sigma_2 = 13.2 \pm 2.3$ MPa (SH). Azimuth of maximum horizontal stress is $\sigma_2 - N$ 114 ± 20º (θSH).

According to the data obtained from the S_FR76_28_GM2 well, at elevations of 403.4 ± 29.1 m, the vertical stress value is $\sigma_2 = 10.7 \pm 0.8$ MPa (Sv), minimum horizontal stress value is $\sigma_3 = 7.0 \pm 0.7$ MPa (Sh), the maximum horizontal stress value is $\sigma_1 = 11.5 \pm 2.5$ MPa, the maximum horizontal stress direction is $\sigma_1 - N$ 141 ± 20º (θSH).

According to the data obtained, the stress field can be characterized by the ratio of the principal stresses $\sigma_1 \approx \sigma_2 > \sigma_3$. The direction of the maximum principal stress action is northwest-southeast.

As a result of laboratory research, the strength properties of the following rocks have been determined: beresites, granodiorites, diorites and lamprophyres. The following are the values of the physical-mechanical properties of the indicated lithological varieties in the mass. With the help of RocData software, the mass rock strength passports have been constructed. Figure 3 shows the constructed rock strength passport.

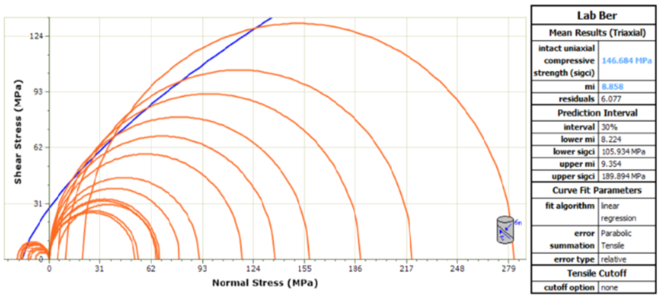

Figure 3. Mass rock strength passport

As a result of the performed calculations, the following values of the mass rock strength parameters have been obtained (Table 2). The results of calculating the rock mass shear strength properties are shown in Table 3.

The calculation results have shown that the uniaxial compressive strength of the beresitized granodiorite rocks decreases approximately by 90%, diorites by 92%, granodiorites by 92%, and lamprophyre dikes by 95%. The global strength of the beresitized granodiorite mass is 20%, diorites 17%, granodiorites 21%, lamprophyre dike 17% of the strength in the sample. The adhesion coefficient along a natural fracture is: beresitized granodiorites – 0.13 MPa, for diorites 0.29 MPa, for granodiorites – 0.13 MPa, for lamprophyres – 0.11 MPa.

Based on the linear survey data of fractures and its processing in the Dips software, a kinematic analysis of the mass fracturing has been performed, the results of which are shown in Figure 4. Three fracture systems have been identified.

Based on the kinematic analysis results, it should be assumed that the main fracture system is system No. 2, the orientation of which is parallel to the ore body contour. Along the fractures, the rock mass cleavage from the hanging wall under the action of its own weight is possible, while the cleavage from the footwall is unlikely. For certain fracture systems, analysis is performed in Unwedge software using the limit equilibrium method in order to determine the stability factor of wedges (Fig. 5).

Table 2. Mass rock strength properties

Rock type	Ultimate compressive strength in sample, MPa	Geological strength index	Elasticity modulus, MPa	Hoek-Brown strength criterion, mb	Internal friction angle, deg	Adhesion coefficient, MPa	Tensile strength, MPa	Compressive strength, MPa	Global strength, MPa	Deformation modulus, MPa
Beresites (Ber)	146.68	58	43560	1.98	31.82	8.06	0.7	14.01	28.98	20676.64
Diorites (Diorite)	196.35	55	62250	1.41	44.68	3.45	0.94	15.79	32.62	25415.47
Granodiorites (GRD)	190.79	56	51320	2.43	49.41	3.51	0.59	16.25	40.84	22071.68
Lamprophyres (LPH)	204.01	48	68670	1.61	46.61	2.9	0.39	10.93	34.54	18640.27

Table 3. Rock mass shear strength properties

Rock type	Internal friction angle in sample	Fracture roughness coefficient	Fracture wall compression strength, MPa	Normal stress, MPa	Internal friction angle in the mass	Adhesion coefficient in the mass
Beresites (Ber)	12.28	6.477	8.323	7.36	12.13	0.13
Diorites (Diorite)	13	13.22	8.78	7.74	12.64	0.29
Granodiorites (GRD)	12	6.63	8.3	7.33	11.85	0.13
Lamprophyres (LPH)	10	5.74	8.85	7.27	10.06	0.11

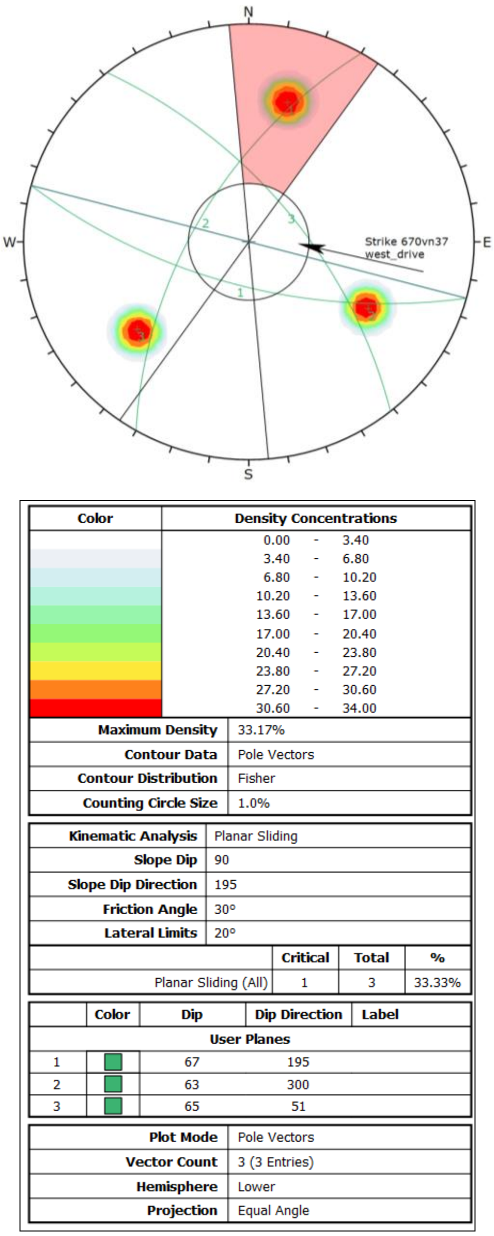

Figure 4. Kinematic analysis of mass fracturing

Unwedge software is designed for stability analysis and subsequent visualization of the results in 3D. Calculations are made for underground mine workings, within which the geological structure is distinguished by the presence of a large number of discontinuities in the structure (Fig. 5).

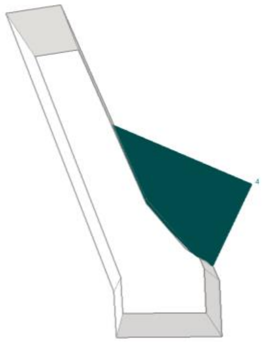

Figure 5. Limit equilibrium analysis in Unwedge software

The stability factor is calculated for a potentially unstable area. Unwedge can be used to quickly create models, perform stability factor analysis, select locations for strengthening structures and interpret results. The graphical data interpreter includes a rich set of tools, including 3D-animation, for conveniently displaying delineations around the test site.

The analysis by the limit equilibrium method has shown that the footwall rocks are in a stable state, while the safety factor of the hanging wall rocks is 0.98, according to the Unwedge program criteria. If the safety factor is less than 1, then the rocks are in an unstable state, from which it should be assumed that the hanging wall requires artificial maintenance by fastening using cable bolts.

Rope bolts are set in the wells, which are represented by geometrically modified cables, the minimum tensile strength of each cable is 250 kN at a length of 6 or 9 m. All rope bolts with double cables should have a diameter of 15.2 mm. Drilling blast-holes for setting cable bolts and setting itself are performed by the PHQ universal drilling machine. When the cables are set, the blast-hole walls between the rope bolts should be filled with cement mortar. The minimum dimensions of the rope bolt plates should be 300×300 mm with a thickness of 10 mm. The plates are set after 12 hours from the moment of filling the wells with cement mortar, in which the rope bolts are mounted.

The procedure for setting cable bolts:

– a cable bolt is inserted into the well drilled round, assembled with an air outlet pipe, so that the remaining part of the bolt about 200 mm long is left at the wellhead;

– a pipe is inserted into the well to a depth of 1.0 m, connected to a hose for supplying cement mortar;

– the wellhead with a set bolt and a cement mortar supply pipe is compacted and rammed with a closing plug (paper, rags, etc.);

– cement mortar is pumped using a pneumatic pump;

– after 12 hours of thickening, a support washer with conical nut is installed.

After the aggregate is made in the form of a cement mortar, samples are taken to determine the compressive strength of the cement. The term for the complete thickening of cement mortar is 28 days [52]. Figure 6 below shows a graph for determining the cement mortar strength by day.

When cement is reacted with water, the mixture sets, leading to the obtaining of strength properties and structure by the material. After the mortar has set, the thickening process begins. It is characterized by the beginning of hydration, or, in other words, by the interaction of cement and water. The strength of the resulting mixture and the thickening time are interrelated with each other. Water in the mixture must be present until the cement reacts with it.

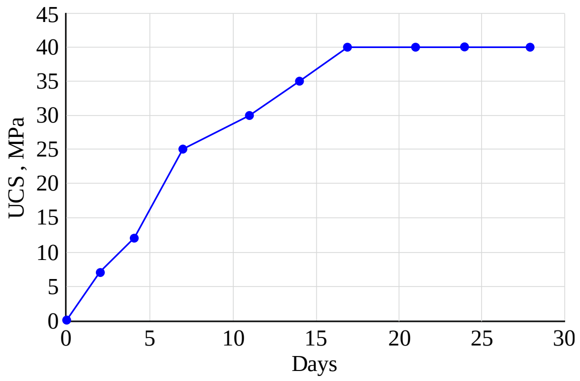

Figure 6. Compressive strength of cement mortar samples

Analyzing Figure 6, it can be stated that the first 17 days are characterized by a linear increase in UCS and reaches a value of 40 MPa. For the next 11 days, the compressive strength of the cement mortar samples does not change. Thus, complete thickening occurs much faster than the generally accepted drying time (28 days) to achieve its design strength.

Figure 7 shows the results of experimental tests on the use of cable fastening to artificially maintain the mined-out space and to reduce ore dilution.

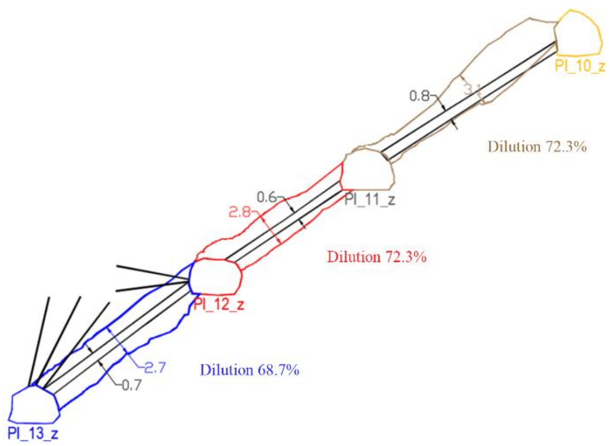

Figure 7. Results of experimental tests of cable fastening, sectional view along a drill ring 140

As shown in Figure 7, the dilution varies from 68.7% to 72.3% with an intermediate value of 71.7%. The comparative analysis results with the overlying sublevels are summarized in a diagram (Fig. 8). From a comparative analysis of the ore dilution, it can be seen that the cable bolts do not prevent the cleavage of the hanging wall rocks, and, accordingly, the ore dilution does not decrease. Therefore, the use of cable bolts in unstable, highly fractured rocks with a dip angle of not more than 40º does not give a positive result.

The maximum ore dilution value in inclined veins with a dip angle of not more than 40º is 72.5% within the Pl_11_z vein along the drill ring 142, while along the same drill ring within the Pl_13_z vein, the ore dilution is 68.0%, which is 6.3%. The minimum ore dilution value in inclined veins with a dip angle of not more than 40º is 61.0% within the Pl_13_z vein along the drill ring 139, while along the same drill ring within the Pl_11_z vein, the ore dilution is 66.0%, which is 7.6%. The average dilution value for all drill rings is 66.1% within the Pl_13_z vein, 68.7% within the Pl_12_z vein, and 68.6% within the Pl_11_z vein.

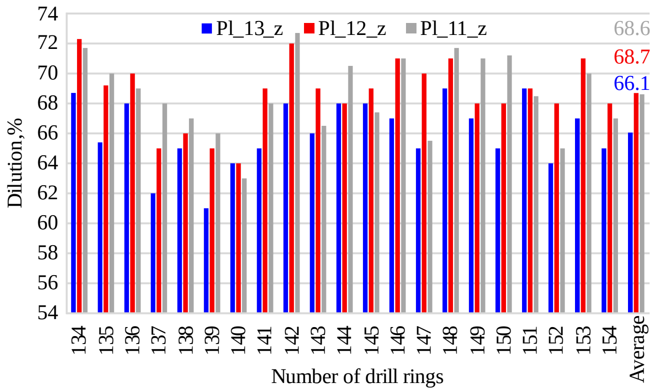

Figure 8. Comparative analysis of the actual ore dilution results in inclined veins with a dip angle of not more than 40º

Similar experiments are conducted in steep-dipping veins. For conducting the experiments, a local area of the Frolovskaya vein has been selected (Fr_18_v), with a vein dip angle of more than 60º. Figure 9 shows the results of experimental tests on steep-dipping veins with a dip angle of more than 60º.

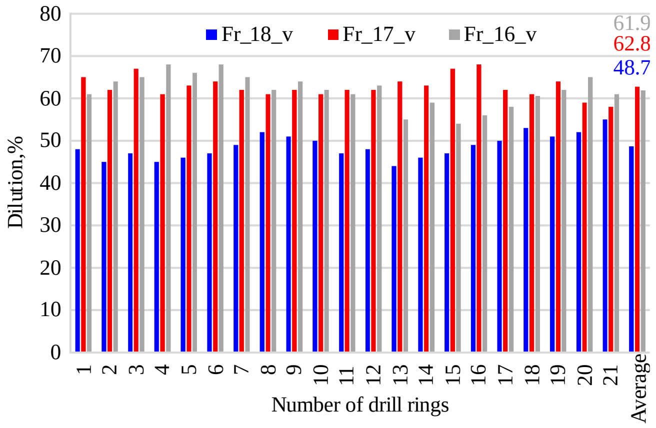

Figure 9. Comparative analysis of the actual ore dilution results in steep-dipping veins with a dip angle of more than 60º

The maximum ore dilution value in inclined veins with a dip angle of not more than 60º is 67.5% within the Fr_17_v vein along the drill ring 16, while along the same drill ring within the Fr_18_v vein, the ore dilution is 49.0%, which is 27.5%. The minimum ore dilution value in inclined veins with a dip angle of not more than 60º is 44.0% within the Fr_18_v vein along the drill ring 13, while along the same drill ring within the Fr_17_v vein, the ore dilution is 63.5%, which is 30.8%. The average dilution value for all drill rings is 48.7% within the Fr_18_v vein, 61.9% within the Fr_17_v vein, and 61.9% within the Fr_16_v vein.

Thus, based on the results of experimental tests, it should be noted that in steep-dipping veins, cable fastening can reduce the ore dilution from 62.8 to 48.7%, that is, in comparison with the overlying sublevels, the ore dilution has decreased by about 14%.

4. Conclusions

As a result of the research, a kinematic analysis has been performed in the Dips software based on the linear survey data of fractures, which determines the fracture systems that form wedges in the footwall and hanging wall of the stope space.

Based on the revealed fracture systems, a numerical analysis has been performed using the limit equilibrium method in the Unwedge software to determine the safety factor of the formed wedges.

The natural stress field has been measured, which is characterized by the ratio of the principal stresses $\sigma_1 \approx \sigma_2 > \sigma_3$. The direction of the maximum principal stress action is northwest-southeast.

Numerical analysis by the limit equilibrium method has shown that the footwall rocks are in a stable state, while the safety factor of the hanging wall rocks is 0.98, from which it should be assumed that the hanging wall requires artificial maintenance by fastening using cable bolts.

According to the results of experiments, by artificially maintaining the mined-out space in inclined veins with a dip angle of up to 40º, the average ore dilution is 66.1%, while the ore dilution in the previously mined sublevels without fastening is 68.7%. In similar experiments in steep-dipping veins with a dip angle of more than 60º, dilution decreases from 62.8 to 48.7%, that is by 14%.

Thus, the effectiveness of the artificial maintenance of the mined-our area in inclined veins is insignificant and does not cover the costs spent on drilling wells, equipment and materials for fastening. However, in steep-dipping veins with a dip angle of more than 60º, the use of cable fastening can significantly reduce ore dilution and is cost effective.

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Зниження збіднення руди при відпрацюванні малопотужних рудних тіл шляхом штучної підтримки виробленого простору

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Мета. Дослідження ефективності штучного підтримання виробленого простору на основі застосування тросових анкерів для зниження показника збіднення при відпрацюванні малопотужних рудних тіл.

Методика. Проведено геотехнічне картування масиву гірських порід за рейтинговими класифікаціями Q, RMR, RQD і GSI, виконано лінійну зйомку системи тріщин порід висячого та лежачого боків із застосуванням гірського компасу та додатку GEO ID. Застосовано чисельний аналіз методом граничної рівноваги у програмному пакеті Unwedge для визначення коефіцієнта запасу міцності масиву, що містить клини з тріщин. За допомогою молотка Шмідта було визначено межу міцності порід на одновісне стискання у масиві. Проведено натурні експерименти із застосуванням тросового кріплення в умовах родовища Акбакай.

Результати. Виявлено, що породи лежачого боку перебувають у стійкому стані, тоді як запас міцності порід висячого боку дорівнює 0,98, що потребує штучної підтримки шляхом кріплення із застосуванням тросових анкерів. Розраховано параметри тросового кріплення з урахуванням нерівномірності розподілу горизонтальних і вертикальних напружень у масиві. Встановлено, що при зміцненні висячого боку тросовими анкерами в похилих жилах з кутом падіння до 40° середнє збіднення руди становило 66,1%, а відпрацьованих раніше без кріплення – 68,7%. В аналогічних експериментах на крутопадаючих жилах із кутом падіння більше 60° збіднення знизилось з 62,8 до 48,7%.

Наукова новизна. Виявлено, що в умовах родовища Акбакай тросове закріплення порід висячого боку є ефективним при куті падіння рудного покладу понад 60°, при якому знижується показник збіднення видобутої руди.

Практична значимість. Результати досліджень можуть бути застосовані для підвищення стійкості порід висячого боку під час відпрацювання малопотужних рудних тіл системою підповерхового обвалення.

Ключові слова: *збіднення, руда, тросове кріплення, очисний простір, породи, тріщинуватість, кут падіння*