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Research into the influence of the thin ore body occurrence elements and stope parameters on loss and dilution values

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Abstract

Purpose. Development of a methodology for determining the level of actual losses and dilution of ore based on the identified patterns of the influence of occurring ore body elements and the stope parameters when mining thin slope deposits using a system for delivering ore by the blasting force.

Methods. The task set is solved using an integrated approach, including the analysis of literary sources and the existing practical experience on the issues of losses and dilution, conducting experimental-industrial experiments in the conditions of the Akbakai deposit to assess the recommended method effectiveness for determining the values of excess losses and dilution of ore, geomechanical assessment of the mass using the methods of limit equilibrium, numerical and probabilistic analyses.

Findings. A methodology for determining losses and dilution when mining thin slope deposits using a system for delivering ore by the blasting force is proposed, which makes it possible to predict excess losses and dilution coefficients arising from stope roof caving and incomplete ore delivery. The actual losses and dilution of ores in the Akbakai deposit have been determined based on the proposed methodology and instrumental surveys. The developed methodology for determining the values of excess losses and dilution makes it possible to take preliminary measures to prevent the stope roof caving with the complete ore mass delivery using the blasting force.

Originality. New dependences have been revealed for the conditions of the Akbakai deposit: logarithmic – the value of dilution depending on the angle of the ore body occurrence; polynomial – the stope maximum span depending on the ore body dip angle; exponential – ore losses depending on the angle of the ore body occurrence.

Practical implications. The practical significance, confirmed in the course of pilot-experimental work, is in minimizing the percentage of loss and dilution of the useful component when mining thin slope ore deposits using a sublevel blast-hole stoping system with an open stope space, which makes it possible to reduce the cost of the produced mineral.

Keywords: *stability, rock caving, ore dilution, ore losses, stress-strain state, numerical analysis*

1. Introduction

The global increase in prices for non-ferrous, rare and precious metals had a positive impact on the dynamics of growing productivity in the mining-and-metallurgical sector of Kazakhstan. Based on the data of the Republican Agency for National Statistics, at the end of 2021 – beginning of 2022, the production of non-ferrous metal ores in Kazakhstan increased by 4.7% and is currently only growing. Corporation Kazakhmys TOO began to mine more copper ore by 6%. JSC AK Altynalmas increased mining of gold ore by 59%. Aluminium of Kazakhstan JSC for 10 months has mined bauxite by 11% more than in the same period last year. Moreover, against the backdrop of rising prices for metals, the above corporations have recently become intensively involved in mining of previously considered economically inefficient and difficult-to-develop deposits with extremely difficult mining-geological and mining-technical conditions [1]-[6].

Mining-geological and mining-technical factors that form the complexity of mining deposits are very different. They are expressed not only in the fracturing of ores and rocks, the diversity of their physical-mechanical properties, but also in the diversity and variability of the occurring elements. From this point of view, in our opinion, gold vein deposits are one of the most difficult geological objects to mine. Gold ore vein deposits have a number of peculiarities: complex geological structure and tectonic disturbance, extremely uneven distribution of reserves in the subsoil, limited dimensions of the mining space, morphological variability of the veins, a variety of conditions for the stability of ores and host rocks [7]-[10].

There are 14 main gold mining regions on the territory of the Republic of Kazakhstan, in which almost all gold reserves are concentrated. These are Mugadzharsk, Zhetygarinsk, Kokshetau, Akmola, Maikinsk, North Pribalkhash, Kalbinsk, Rudno-Altaysk, Yuzhno-Altaysk, Dzhungar, Zailisk, Akbakai, Karatau, Zhezkazgan. The deposits are represented by ten geological-industrial types: quartz-vein; stockwork; mineralized zones; weathering crusts; alluvial deposits; sulphide-polymetallic silver-gold; in complex with asso-

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ciated gold; sulphide-gold-silver-polymetallic; porphyritic-gold-copper and sulphide-gold-copper [11]-[13]. The main share of the above industrial regions is represented by deposits with thin veins. The most important geological-industrial types of deposits are ores of mineralized zones, which account for more than 20% of explored reserves (Bakyrchik, Bolshevik, etc.); quartz-vein ores, which account for about 16% of the reserves (Aksu, Zholymbet, Akbakai, etc.); stockwork ores 14% (Vasilkovskoye, Yubileinoye, etc.); gold-sulphide ores 10% (Maykain, Abyz, etc.). When mining thin gold ore veins, more than 40% of the ore is mined by underground method, while the share of mining slope deposits is 20-30%.

It is known that one of the peculiarities of mining the vein-type deposits and predominantly thin deposits is their increased labor intensity compared to systems used for thick ore deposits [14]-[17]. This is explained by the fact that the conditions for mining thin deposits in a narrow stope space do not allow the use of high-performance drilling and load-haul-dump equipment, which would intensify mining processes. Moreover, due to the ore body low thickness, its mining is conducted with dinting of the host rocks, which leads to a large mined ore dilution (up to 65%) and difficulties in extracting gold during its processing. Since explosive charges, placed in a narrow face, work in a large clamp, resulting in the need to thicken the grid of their location. This, in turn, leads to a significantly reduction in the yield of broken ore and, as a result, an increase in labor costs for ore breaking, economic and ecological issues [18]-[23]. Especially often, the narrow width of the extraction space causes the difficult drawing of broken ore, as a result of which the labor costs for this operation reach 35-40% of the total labor intensity of the stope extraction, causing, in turn, increased losses of broken ore (up to 12%). Dilution and losses of ore are one of the main indicators of the efficiency of the applied mining systems, on which the economy of mines and gold extracting factories (GEF) largely depends.

In most cases, when mining vein deposits, the dilution value is determined through the ratio of the thickness of the broken waste rocks to the width of the stope space [24], using the Formula:

$$P = \frac{m_s - m_v}{m_s}, \tag{1}$$

where:

m_s – width of the stope space, m;

m_v – vein thickness, m.

There is also a known method [25], the essence of which is to determine the coefficient of actual dilution, equal to the ratio of the mined ore weight to the weight of the vein mass contained in it. The coefficient value of the broken ore real dilution is determined based on the vein thickness, the width of the mined-out space, the volume weight of the vein mass and the host rocks:

$$P_0 = \frac{(m_v - m_v)\gamma_r + m_v\gamma_v}{m_v\gamma_v}, \tag{2}$$

where:

γ_r – the volume weight of the host rocks, ton/m^3;

γ_v – the volume weight of the vein mass, ton/m^3.

Although these methods are more flexible and make it possible to determine the ore dilution separately for each block, but have their drawbacks. These methods do not fully take into account the ore body elements and geomechanical processes occurring in the mass after ore breaking. For example, the ore body dip angle, possible roof caving, stope bottom sliding, etc. are not taken into account.

The experience of mining thin slope ore deposits shows that one of the most common ore mining technologies is a sublevel blast-hole stoping using a system for delivering ore by the blasting force implemented in some deposits to mine more than 60% of the reserves [26]-[28]. Main advantage of this mining system is its simplicity, which makes it universal and allows mining ore deposits in various mining-geological conditions. With a sublevel blast-hole stoping system, ore losses are determined mainly by a variant-analytical method according to the criterion of maximum output from 1 ton of depleted balance mineral reserves based on a comparison of the economic consequences of mining the reserves along the bottom position contours along the stope. The value of the balance ore (K_l) loss factor or balance ore loss (L) is expressed by the Formulas:

$$K_n = 1 - \frac{D(a-b)}{B(c-b)}; \tag{3}$$

$$n = \left[1 - \frac{D(a-b)}{B(c-b)}\right] \cdot 100\%, \tag{4}$$

where:

D – the amount of mined ore mass, ton;

B – the value of depleted balance ore reserves, ton;

a – the useful component grade in the mined ore, %;

b – the useful component grade in the mixed rock, %;

c – the useful component grade in the deposit ore (mass), %.

Despite the large amount of research in this area [29]-[31], the existing technologies for mining ores from thin slope veins are characterized by high labor intensity, low technical and economic indicators, significant losses, and ore dilution.

At present, in the main gold mines of Kazakhstan, including mines of the Akbakai industrial zone [32], [33], a sublevel blast-hole stoping system of ore breaking with deep wells and delivery by the blasting force is used to mine thin slope veins. Since this system provides higher performance in terms of labor productivity, is the safest, and allows organizing the continuous ore mining. However, a system for delivering ore by the blasting force has high loss and dilution coefficients. For example, the actual losses and dilution coefficients in the mining of thin slope deposits in the mines of the Akbakai industrial zone have been increasing in recent years (Fig. 1). This is mainly due to the deepening of mining operations to the deposit depth, as well as the involvement in mining of previously unprofitable ore deposits with complex morphology. As a result of the data analysis on loss and dilution in the “Pologaya-1” vein, it has been found that the losses on sublevels 10-13 increase to 12%, when the planned level of loss in the vein is 8.6%, and dilution increases to 55% with the planned 37.8%, respectively.

Currently, ore losses and dilution of the Akbakai deposit are taken into account based on weighing and testing results, adjusted according to the data from gold extracting factories. However, this method does not provide an opportunity to assess the results of ore mining from stopes and blocks. Accordingly, it cannot indicate the main sources of losses and dilution, give an assessment of the applied vein mining technology, in order to make a decision on reducing losses and dilution.

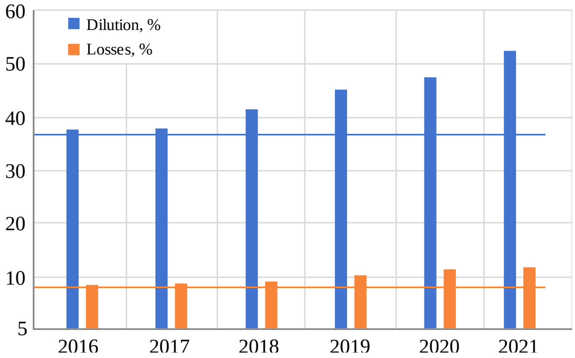

***Figure 1.** Comparative diagram of ore loss and dilution coefficients in the mining of thin slope Akbakai deposits (2016-2021): 1 – planned dilution level (37.8%); 2 – planned loss level (8.6%)*

Therefore, the issues of finding more efficient and flexible methods for determining losses and dilution in the mining of thin slope ore deposits, taking into account occurring ore body elements and the geomechanical processes of the mass, are relevant and do not lose their significance.

By properly accounting for losses and dilution, the causes can be identified and the most effective measures can be taken to reduce or eliminate them. It is very difficult to determine the boundaries where losses and dilution can be excluded. In this regard, in order to solve the above problem, the task set is to reveal and study the main causes affecting the value of losses and dilution when mining thin slope ore deposits.

2. Materials and methods

2.1. Study area

Mining of ore bodies in the Akbakai deposit using a 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 gold ore veins with dip angles of up to 35° and from 35 to 50°, with a thickness of 1.51 to 1.92 m. Currently, the following veins are mined by this system:

– "Yubileynaya" vein with a dip angle of 40-55° and an average thickness of 1.51 m;

– "Glubinnaya" vein with a dip angle of 35-55° and an average thickness of 1.61 m;

– "Pologaya-1" vein with a dip angle of 35-50° and an average thickness of 1.92 m;

– "Pologaya-6" vein with a dip angle of 30-35° and an average thickness of 1.74 m.

The main advantage of this system is that it eliminates the presence of workers and machines in the stope space, thereby increasing the safety of work, ensuring the independence of breaking and drawing operations, while having sufficient drilled and blocked-out ore reserves, as well as creating conditions for the continuous performance of work. In addition, this system makes it possible to mechanize all the technological cycle production operations as much as possible, and high-performance self-propelled equipment can be widely used. But as noted above, the system for delivering ore by the blasting force has high loss and dilution coefficients.

From the analysis of data on the deposit, it has been revealed that losses and dilution depend on a number of factors: losses of broken ore on the stope bottom due to incomplete ore delivery by the blasting force; ore losses on the contacts of ore bodies with host rocks; dilution of the ore due to the roof caving and stope bottom sliding.

At the same time, it should be noted that at present there is no reliable methodology for quantifying losses and dilution in a system for delivering ore by the blasting force. Therefore, there is a need to find a new methodology for determining losses and dilution, taking into account the patterns of occurring ore bodies, the geomechanical process, as well as the trajectory and range of the ore throw after breaking.

For ore breaking when mining slope veins with a thickness of up to 1.2 m at the Akbakai mine, a parallel grid is used with a compensation well of small diameter (Fig. 2).

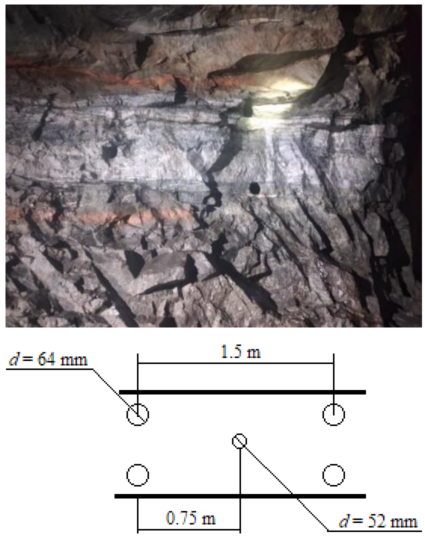

***Figure 2.** Well location grid (drill ring No. 187) along the "Pologaya-1" vein at the Akbakai mine*

The specific conditions for using a system for delivering ore by the blasting force (the impossibility of direct inspection of the roof and bottom of the stope) make it difficult to monitor the stope roof state. Existing mine surveying methods do not allow to perform these operations and, accordingly, to determine the places of concentration of losses and dilution. This problem has been solved by determining the actual contours of the stope using remote tacheometric survey and predicting the geomechanical process of the rock mass after blasting operations.

2.2. Calculation of the studied stope parameters

The study of any measures to reduce losses and dilution is associated with the need to obtain data on the number (n) and (P) within the specific stope under study. In this case, the losses (n) include the ore losses (n_{pt}) in the arches and unbroken crusts of the stope ceiling, ore losses in the pillars (n_c^+) when it overhangs the pillar contour limits, minus that part of the ore (n_c^-) penetrating from pillars during their destruction, ore loss (n_{ph}) on the stope bottom due to incomplete delivery, minus the rock mass (n_{ob}) penetrating from the hanging wall caving. Thus, ore losses within the stope can be determined by the Formula:

$$n = \frac{n_{pt} + \left(n_c^+ - n_c^-\right) + n_{ph}}{B}, \quad (5)$$

where:

B – balance ore reserves in the stope, ton.

$$n_{ph} = M - n_{ob}, \text{ ton}, \quad (6)$$

where:

M – the amount of rock mass bulked on the bottom, ton.

The equivalent metal grade in the ore when host rocks penetrate into it is determined by the Formula:

$$a_{ek} = \frac{a_r n_{ph} + a_{pr} n_{pr}}{n_{ph} n_{pr}}, \tag{7}$$

where:

a_r – the metal grade in ore, g/ton;

a_{pr} – the metal grade in the host rocks, g/ton;

n_{pr} – the amount of diluting rock, ton.

Broken ore becomes substandard and is considered lost if the inequality is met:

$$a_{ek} < a_{min}, \tag{8}$$

where:

a_{min} – minimum cut-off metal grade in ore, g/ton.

From Equation 7 and Inequality 8, the boundary condition for the transition of broken ore into losses can be obtained:

$$n_{ph} < n_{pr} \left(\frac{a_{pr} + a_r \dfrac{n_{ph}}{n_{pr}}}{a_{min}} \right). \tag{9}$$

The methodology for determining ore losses on the bottom (n_{ph}) under the conditions of a system for delivering ore by the blasting force is as follows:

– it is determined by remote surveying the total rock mass (V_{nb}) bulked on the stope bottom;

– using a remote surveying and a preliminary prediction of the mass geomechanical state, the volume of caving rocks in the hanging wall (V_{ob}) and the volume of sliding rocks of the stope bottom are determined;

– the amount of lost ore on the stope bottom is calculated:

$$n_{ph} = \left(V_{nb} - V_{ob} K\right)\gamma, \text{ ton}, \tag{10}$$

where:

K – fragmentation index;

γ – volume weight of ore in loosened state, ton/m^3.

The volume of destruction in the stope bottom is determined quite simply by measuring the caving thickness of rocks in the foot wall and further calculating their volume (V_{ph}). Such measurements are made directly in the drill winze.

The ore dilution is determined for individual operational blocks by a direct method according to the ratio of the amount of diluting rock mass to the entire mined marketable ore:

$$P = \frac{G}{D} \cdot 100, \text{ \%}, \tag{11}$$

where:

G – the amount of diluting rock mass, ton.

$$G = G_1 + G_2, \text{ ton}, \tag{12}$$

where:

G_1 – the amount of diluting rock mass, ton;

G_2 – the amount of diluting off-balance ore, ton;

D – the amount of drawn rock mass, ton.

$$D = A + G, \text{ ton}, \tag{13}$$

where:

A – the amount of mined balance ore, ton.

Under the conditions of a system for delivering ore by the blasting force in a narrow stope, the amount of diluting rock mass (G) is determined as follows. Using a remote surveying, the volume of caved rocks of the hanging wall (V_{ob}) is determined, as well as the volume of sliding rocks of the foot wall and drawn together with the ore (V_{rz}). Then the amount of diluting rock mass is calculated by the Formula:

$$G = \left(V_{ob} + V_{rz}\right)\gamma, \text{ ton}, \tag{14}$$

where:

γ – the volume weight of the host rocks, ton/m^3.

In order to predict the overplanned ore dilution according to K. Matthews criteria, a methodology has been developed to determine the optimal parameters of stopes and pillars [34]-[36]. This methodology is based on the calculation and mapping of two values: stability index N, which characterizes the ability of the rock mass to maintain stability under given conditions of stress state, structural organization of mass disturbances and orientation of the stope space, and shape factor S or hydraulic radius HR, which account for the geometric interdependence of outcrop sizes.

The stability index N is calculated by the Formula:

$$N = \frac{RQD}{j_n} \cdot \frac{j_r}{j_a} \cdot A \cdot B \cdot C, \tag{15}$$

where:

RQD – rock quality index;

j_n – an index of the number of fracture systems;

j_r – fracture surface roughness index;

j_a – fracture adhesion index;

A – parameter characterizing the ratio of strength to the stress state of rocks;

B – parameter characterizing the fracture orientation;

C – parameter characterizing the outcrop dip angle (slope).

The correctness of accounting for dilution is controlled by determining dilution when drawing ore from the stopes according to the composition of the drawn ore. The dilution coefficient for the studied stope is found by comparing the total weighed sample material with the weight of the sorted rock fraction.

2.3. Numerical modeling

In order to find effective methods to eliminate losses and dilution, as well as to correctly select sites for the implementation of these methods, a quantitative analysis of the dependences of various types of losses and dilution on the ore body dip angle is necessary. To predict the mass stability from the side of the ore body hanging wall, a geomechanical model has been constructed along the drill ring No. 187 between sublevel drifts 17-18 based on the RQD and RMR rating classification of the "Pologaya-1" vein at the Akbakai deposit using the methodology of K. Matthews.

The initial data for revealing patterns of dilution from the ore body dip angle, depending on the stope span dimensions, are measurements of roof caving in the mined stopes of the Akbakai deposit slope veins. A typical form of roof caving is presented in Figure 3.

The index of dilution as a result of caved rocks is determined by the Formula:

$$P_{ob} = \frac{V_{ob}\gamma}{B}, \tag{16}$$

where:

V_{ob} – volume of caved rocks according to remote surveying data, m^3;

γ – the volume weight of the hanging wall rocks, ton/m^3.

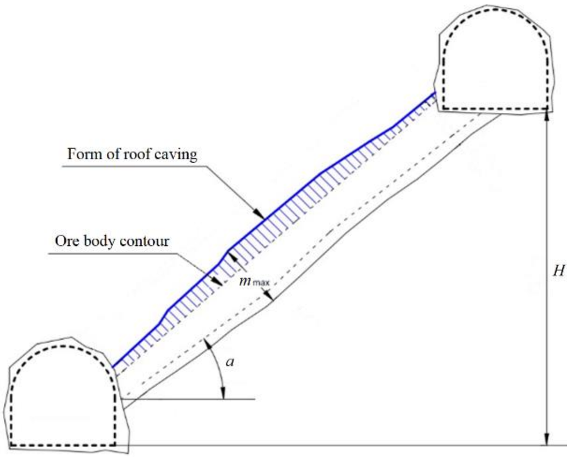

Figure 3. Typical form of the stope roof caving

A numerical geomechanical model of deposit mining is developed to predict excess dilution coefficients arising from possible stope roof caving and incomplete ore delivery by the blasting force using the Rocscience RS2 software product. This makes it possible to take into account a large number of factors affecting the rock mass state. The calculations take into account not only the physical-mechanical rock properties and the stresses acting in the mass, but also the structural characteristics of the mass, as well as the degree of technogenic impact. The use of the Rocscience RS2 software package, as a result of taking into account a large number of factors influencing the mass state, significantly increases the reliability of the mass stability assessment.

3. Results and discussion

The sublevel blast-hole stoping using a system for delivering ore by the blasting force is characterized by significant changes in the value of losses and dilution depending on the ore body dip angle. At dip angles less than 45°, there are losses of broken ore on the stope bottom due to incomplete delivery, and the value of such losses increases significantly with a decrease in the dip angle of the deposit. The dilution of the ore caused by the roof caving tends to decrease with increasing dip angle, but at the same time, the stope bottom often loses its stability.

To predict the mass stability from the side of the ore body hanging wall based on the RQD and RMR rating classification of the "Pologaya-1" vein at the "Akbakai" deposits, using the methodology of K. Matthews, a geomechanical model has been developed along the drill ring No. 187 between sublevel drifts 17-18 (Table 1).

Table 1. Results of the stope hanging wall mass stability

Hanging wall												
Hydraulic radius for various block geometrics												
Vertical	At an angle	Block length, m										
Stope height, m	Height, m	10	15	20	25	30	35	40	45	49	55	60
10	15.557	3.04	3.82	4.38	4.79	5.12	5.39	5.60	5.78	5.90	6.06	6.18
15	23.336	3.50	4.57	5.38	6.03	6.56	7.00	7.37	7.68	7.90	8.19	8.40
20	31.114	3.78	5.06	6.09	6.93	7.64	8.24	8.75	9.20	9.52	9.94	10.24
30	46.672	4.12	5.68	7.00	8.14	9.13	10.00	10.77	11.46	11.95	12.62	13.13
40	62.229	4.31	6.04	7.57	8.92	10.12	11.20	12.17	13.06	13.71	14.60	15.27
50	77.786	4.43	6.29	7.95	9.46	10.83	12.07	13.21	14.25	15.03	16.11	16.94
60	93.343	4.52	6.46	8.24	9.86	11.35	12.73	14.00	15.18	16.07	17.30	18.26
70	108.9	5.58	6.59	8.45	10.17	11.76	13.24	14.63	15.92	16.90	18.27	19.34

N	1.3	Hydraulic radius, m
	Constant	3.29
	Near-border without fastening	5.5
	Constant with fastening	8.4
	Near-border with fastening	10.4
	Variable	10.4

An analysis of the stability results of the mass along the drill ring No. 187 between sublevel drifts 17-18 has shown that there are unstable areas in the mass from the side of the hanging wall, prone to caving during ore breaking with the charges of the deep wells.

To predict excess dilution coefficients arising from possible stope roof caving and incomplete ore delivery by the blasting force, a numerical model of deposit mining has been developed based on the determined inelastic deformation zones around the mined-out space using the RS2 software product (Fig. 4).

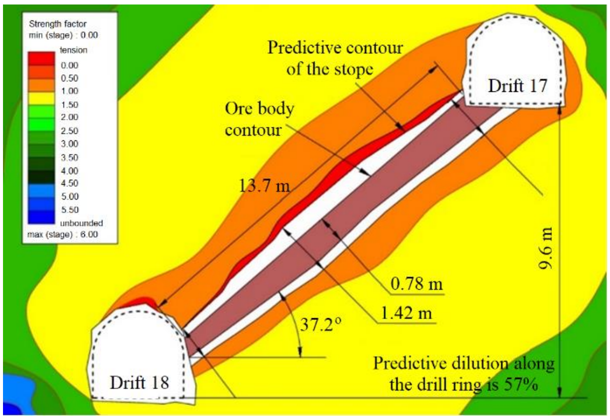

Figure 4. Numerical model of mining the "Pologaya-1" vein along the drill ring No. 187

As a result of numerical modeling, the change in the mass stress-strain state has been determined. On the basis of this, the surrounding mass stability coefficient, the values and directions of its displacements have been calculated, as well as the distribution zones of the mass destruction probability in the given intervals of boundary conditions have been determined. Based on performed numerical analysis, it has been revealed that the predicted dilution along the drill ring No. 187 is 57%. During mining of thin slope veins in the Akbakai deposit, the authors have studied the dependence of dilution on the ore body dip angle, depending on the stope span dimensions. Monitoring results are presented in Table 2 and shown in Figure 5.

Table 2. Results of monitoring the ore dilution at different angles of occurrence

Vein name	Sublevel	Dip angle, degree	Caving thickness, m	Dilution, %
Pologaya-6	5-6	30	0.71	68
	8-9	33	0.68	65
	13-14	33	0.65	63
Pologaya-1	10-11	35	0.64	62
	12-13	36	0.61	60
	17-18	38	0.60	59
Glubinnaya	11-12	40	0.60	59
	13-14	40	0.58	57
	15-16	44	0.55	55

As can be seen from the graph (Fig. 5), the dilution value increases at dip angles less than 45°. The difference in the dilution value depending on the angle of ore body occurrence, when mining thin slope ore deposits, is about 0.9% per 1°. These data make it possible to find effective methods that would reduce dilution with constant stope spans.

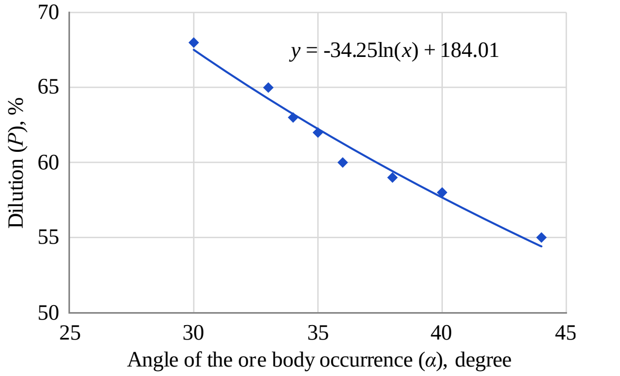

Figure 5. Dependency graph of the dilution value on the ore body occurrence angle

As a research result, it has been determined that with an increase in the angle of the ore body occurrence over 45°, the roof caving decreases and dilution occurs mainly due to the destruction and subsequent sliding of the bottom rocks in the stopes.

The bottom destruction is caused by the impact of dynamic stress waves during blasting of the lower row of wells. Bottom destruction with blasting is observed both at low and steep dip angles (Fig. 6). But under conditions of inclined ore body occurrence, the destroyed bottom, as a rule, does not penetrate into the ore, but remains in the lower part of the rock mass pile. In this case, the unfavorable effect of the bottom destruction is that the bottom rocks, loosening, increase the height of the pile, which makes it difficult to deliver broken ore by subsequent blasting operations.

Figure 6. State of the "Pologaya-1" vein stope bottom along the drill ring No. 187

At steep angles, bottom sliding is conditioned by the action of well blasting and, moreover, by the fact that the foot wall is undercut with a sublevel drift. In such cases, the dilution due to the sliding of the foot wall rocks averages 6-7%.

When mining the thin slope of the Akbakai ore deposit using the system for delivering ore by the blasting force, the outcrop of hanging wall rocks occurs as the ore is broken with deep wells. With an increase in the outcrop area, the roof caving begins.

As a result of a geomechanical study of the stope roof stability, it has been determined that the maximum span of outcrops at dip angles of 35-45° ranges within 1.6-1.9 m, and at dip angles of 50-55° it reaches 2.5 m. To determine the dependence of the maximum span and the dip angle, the values of equivalent spans are calculated by the Formula:

$$L_{ek} = \frac{ab}{\sqrt{a^2 + b^2}}, \qquad (17)$$

where:

a, b – the values of the long and short sides of the rectangular outcrop, respectively, m.

The dependence of the maximum span at different dip angles corresponds to the following Expression:

$$\frac{l_a}{l_0} = \frac{1}{\cos a'}, \qquad (18)$$

where:

l_0 – the maximum span at horizontal ore body occurrence, m;

l_a – the maximum span at inclined ore body occurrence, m.

Given that when mining thin veins using a system for delivering ore by the blasting force, the presence of people in the stope space is excluded, in connection with this, it is proposed to take the stope width equal to approximately 90% of the maximum span value.

As a result of monitoring the roof state, the caving thickness values for different stope spans have been obtained (Fig. 7).

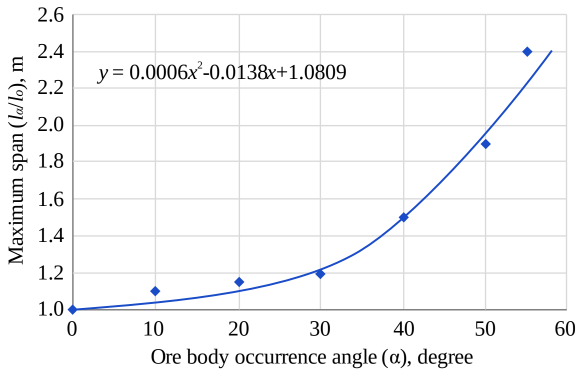

Figure 7. Dependence of the stope maximum span on the ore body dip angle

Analyzing the results of experimental observations on dilution depending on the stope width, it has been revealed that with an increase in the stope width, dilution increases, and ore losses decrease.

As noted above, specifics of the system used for mining thin deposits is the use of the blasting force to deliver the broken rock mass to the block bottom. This moment is very important for areas of deposits with a dip angle of 35-50°, since with a large delivery length of the broken ore, part of it does not reach the block bottom and a rock mass pile is formed (Fig. 8), resulting in ore losses.

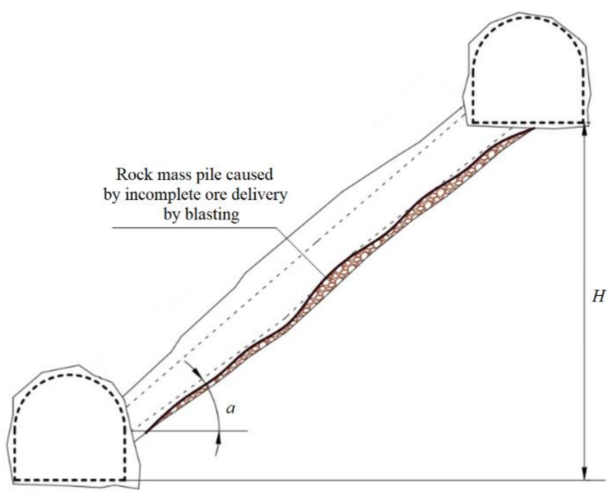

Figure 8. Broken ore mass pile in the stope at the end of its mining

Observing the pile shape, it can be noted that the ore throw initial velocity vector with the usual, normal location of the wells to the stope bottom plane is parallel to the bottom, and if the shear resistance of the lower layer part is taken into account, then the vector (V_0) is directed directly to the stope bottom. Therefore, the studied pile can have a significant braking impact on the lower part of the broken layer. It is clear that under such conditions it is difficult to provide the required ore throwing distance, as a result of which ore losses increase.

In order to determine the most effective position of the lower half-plane focusing the location of the wells, it is necessary to find the dependence of the ore delivery length on the dip angle of this half-plane to the stope bottom at different ore body dip angles. The laws of exterior ballistics are used to find this dependence.

The distance of an elementary body throw from an inclined plane with an initial velocity (V_0) is determined by the Formula:

$$L = \frac{2V_0^2 \sin a}{g}, \text{ m} \tag{19}$$

where:

g – the free fall acceleration, m/s^2;

a – the bottom inclination angle, degree.

If this body is raised relative to the inclined plane to a height (h), then the throw distance (L) will have the following form:

$$L = \frac{2V_0^2 \sin a}{g} + htga, \text{ m.} \tag{20}$$

When the drill ring plane deviates by an angle (φ), the distance of throwing increases due to an increased initial velocity vector (V_0) by value of $\cos^2(a - \varphi)$, and also due to the fact that each elementary volume of the broken layer increases relative to its previous position by $\sin\varphi$. Therefore, the distance of throwing, when the blasting plane is displaced by an angle (φ), can be determined by the Formula:

$$L = \frac{2V_0^2 tga}{g \cos a} \cos^2(a - \varphi) + htga + h \sin\varphi, \text{ m.} \tag{21}$$

The loss of broken ore on the stope bottom due to incomplete delivery by the blasting force is determined based on Formulas 5-10. The results of instrumental survey along the drill ring No. 187 of the "Pologaya-1" vein are presented in Table 3 and shown in Figure 9.

Table 3. Results of monitoring the ore dilution at different angles of occurrence

Vein name	Sublevel	Dip angle, degree	Delivery length, m	Rock mass volume on the stope bottom, m^3	Loss, %
Pologaya-6	5-6	30	14.2	3.2	9.3
	8-9	33	13.8	2.7	8.8
	13-14	34	14.0	29	9.1
Pologaya-1	10-11	35	13.1	2.5	8.4
	12-13	36	12.5	2.2	8.1
	17-18	38	13.7	2.6	8.7
Glubinnaya	11-12	40	12.7	2.0	8.0
	13-14	40	12.2	2.0	7.7
	15-16	44	13.0	2.3	8.2

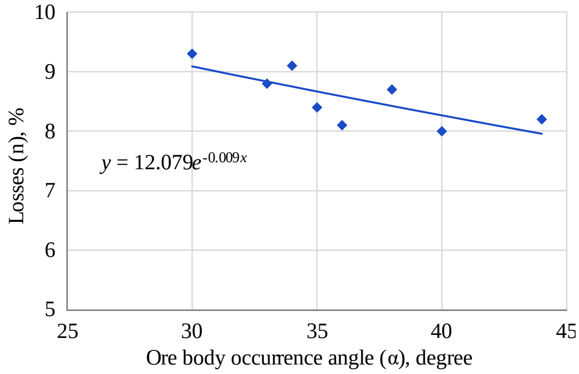

Figure 9. Dependence of ore losses on the ore body occurrence angle

The research results confirm that when mining thin slope ore deposits, the values of losses and dilution directly depend on the elements of the ore body occurrence. The possible ore loss value, depending on the vein dip angle, ranges from 4 to 10%.

4. Conclusions

The system for delivering ore with the blasting force when mining thin slope deposits is characterized by high labor productivity, work safety and relatively low cost of ore mining. However, it has high loss and dilution rates.

The actual ore loss when using the above system in the mines of the Akbakai industrial zone is 5-7% under conditions of complete broken ore delivery, which increases at an angle of less than 40° to 10% due to the broken ore losses in the stope, caused by incomplete ore mass delivery by the blasting force. When mining stopes with small spans and inclined occurrence, dilution is caused by dinting of host rocks and roof caving, and in some places averages 55%, sometimes reaching 70%.

The developed methodology for determining losses and dilution when using a system for delivering ore by the blasting force makes it possible, on the one hand, to predict excess losses and dilution resulting from a possible stope roof caving and incomplete ore delivery. On the other hand, on its basis, preliminary measures to eliminate them can be taken.

Losses from incomplete broken ore delivery to the outlet workings increase at dip angles of less than 40° and can be found by the function $n = 12.079e^{-0.009a}$.

The dilution caused by the stope roof caving increases with a decrease in the angle of the ore body occurrence and can be determined by the function $P = -34.25\ln(a) + 184.01$, in deposits with similar conditions.

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Дослідження впливу елементів залягання малопотужного рудного покладу та параметрів камер на величину втрати і збіднення

Є. Сердалієв, Є. Іскаков, Б. Бахрамов, Д. Аманжолов

Мета. Розробка методики визначення рівня фактичних втрат і збіднення руди на основі встановлення закономірностей впливу елементів залягання рудного тіла та параметрів камер при відпрацюванні малопотужних, похилих покладів системою доставки руди силою вибуху.

Методика. Поставлена задача вирішена використанням комплексного підходу, що включає аналіз літературних джерел і наявного на сьогоднішній день практичного досвіду з питань втрат і розбіжності, проведення дослідно-промислових експериментів в умовах родовища "Акбакай" для встановлення ефективності рекомендованого методу визначення величини наднормативних втрат і разу, геомеханічної оцінки масиву із використанням методів граничної рівноваги, чисельних та ймовірнісних аналізів.

Результати. Запропоновано методику визначення втрат і розбіжності при відпрацюванні малопотужних, похилих покладів системою з доставкою руди силою вибуху, яка дає можливість спрогнозувати наднормативні показники втрат і розбіжності, що виникають через обвал покрівлі камер і неповноту доставки руди. Визначено фактичні втрати й збіднення руд по Акбакайському родовищу на основі запропонованої методики та інструментальних зйомок. Розроблена методика визначення величин наднормативних втрат і збіднення дає можливість попередньо вживати заходів щодо запобігання обваленням покрівлі камер та повноти доставки рудної маси силою вибуху.

Наукова новизна. Для умов Акбакайського родовища виявлено нові залежності: логарифмічна – величини розбіжності від кута залягання рудного тіла; поліноміальна – граничного прольоту камери від кута падіння рудного тіла; експоненційна – втрата руди від кута залягання рудного тіла.

Практична значимість. Практична, підтверджена в ході дослідно-експериментальних робіт, значущість полягає в мінімізації відсотка втрат та збіднення корисного компонента при відпрацюванні малопотужних, похилих рудних покладів підповерхово-камерною системою з відкритим очисним простором, що дозволяє знизити собівартість корисного матеріалу.

Ключові слова: *стійкість, обвалення порід, кут падіння, руйнування руди, втрати руди, напружено-деформований стан, чисельний аналіз*