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Ore dilution control when mining low-thickness ore bodies using a system of sublevel drifts

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Abstract

Purpose. The research is aimed at substantiating the optimal parameters of blast-hole ore breaking to reduce the rock mass destruction when mining low-thickness vein deposits using the mining system of sublevel drifts (SD). The focus is on analyzing the impact of blasting on the out-contour rock mass during the blast-hole breaking process.

Methods. The three-dimensional block model is constructed using rating classifications of rocks based on studying their strength properties and structural peculiarities. The inelastic deformation zones around the stoping extraction are determined by numerical analysis using the finite element method in 2D formulation. Experimental blasts are assessed by varying the blast-hole drilling scheme depending on the stability rating.

Findings. During the experimental-industrial tests, rational blast-hole drilling schemes have been substantiated, contributing to maintaining the stability of the host rocks when mining low-thickness veins.

Originality. Effective methods for reducing the ore dilution have been substantiated, which take into account not only the strength properties and structural peculiarities of rocks, but also their seismic impact from blasting on the out-contour rock mass stability when mining low-thickness deposits using a system of sublevel drifts.

Practical implications. Practical significance is in the possibility of minimizing the percentage of mineral dilution when mining low-thickness ore bodies using a system of sublevel drifts, which can significantly reduce the cost of mined minerals by reducing ore losses caused by the rock mass destruction during mining operations.

Keywords: *ore, dilution, drilling and blasting operations, stope space, rocks, deposit*

1. Introduction

The difficult mining-geological conditions of occurrence of most vein deposits and it prediction are the reason for the variety of mining systems used, the complexity of technological operations for ore mining and the difficulty of selecting highly efficient mechanization tools for them [1], [2]. Open-stope space, ore shrinkage in blocks and backfill systems are mainly used when mining the low-thickness and thin veins [3], [4]. The disadvantages of the mining system with ore shrinkage are low labor productivity, increased labor intensity, poor mechanization, significant losses and dilution of ore during mining the pillars [5]-[8].

Mining systems with backfilling the stope space is one of the most effective ways to maintain the mined-out area, allowing for a complete ore mining and minimizing ore dilution [9], [10]. Due to the high labor intensity, the large volume of preparatory works, and the high cost of backfill operations, the considered system of mining vein deposits is not widely used. However, when mining ore bodies at deep horizons with increased rock pressure, the use of a mining system with backfilling the mined-out space is highly effective [11]-[14].

The mining of low-thickness vein deposits using the open-stope space systems is complicated with high ore dilution rates. The authors of the paper [15] analyze ore dilution in the underground deposits of Copper Cliff Mine (Canada), Dugald River (Australia), Zholymbet (Kazakhstan), Akbakai (Kazakhstan), Shaumyan (Armenia) and Cracow (Australia). All of the above fields mine low-thickness ore deposits using a sublevel drift system and all deposits have a high ore dilution percentage.

The advantage of the sublevel drift mining system is the high mining intensity due to the use of self-propelled machines in all mining processes and low labor intensity. The main disadvantage of the mining system, in particular for low-thickness ore bodies, is high ore dilution. To date, to reduce dilution when mining low-thickness veins using a system of sublevel drifts, the cable fastening is widely used to strengthen the stope space [16]-[18].

Foreign researchers in their studies [19]-[21] describe the effectiveness in the use of cable fastening to control ore dilution using open-stope space systems.

The experimental tests are conducted using cable fastening in the underground Akbakai deposit [15]. The tests

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are conducted on low-thickness ore deposits at ore thickness up to 1.0 m and at a dip angle of more than 32º, that is, in inclined and steep-dipping ore deposits. 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. 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, according to the test results it is noted that the effectiveness of the artificial maintenance of the mined-our area in inclined veins up to 40º is insignificant and does not cover the costs spent on drilling blast-holes and materials for fastening. In this regard, to reduce ore dilution, the use of cable fastening in the conditions of the Akbakai mine is not effective.

For a more detailed analysis of dilution rates, a graph is constructed based on the data of a survey of the stope space in the previously mined sublevel drifts of the 400-520 m horizons of the Akbakai mine. The results are shown in Figure 1. Actual ore dilution ranges within 51-68%, with a planned maximum of 38%.

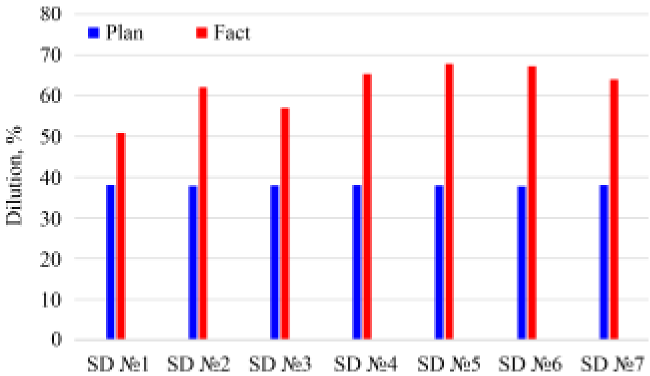


Figure 1. Comparative analysis of planned/actual ore dilution for sublevel drifts

Analysis and synthesis of literature data [22]-[24] make it possible to estimate that more than 90% of rocks during mining are separated from the mass by blasting. Despite the large volume of theoretical and experimental research, effective blast control is insufficiently studied, as there is currently no method for calculating the parameters for drilling and blasting operations (DBO) taking into account the structural, strength and deformation properties of the rock mass and the principal stresses acting in the rock mass. Therefore, the problem of increasing out-contour rock mass stability is very relevant and its solution is based on improving methods for controlling blast energy, which are based on reliable physical concepts of the processes of rock destruction by blasting.

Destruction by blasting of continuous stressed media and identification of its patterns are the subject of research of many scientists. A significant contribution to the research into the processes of rock destruction, the formation and propagation of stress waves in the rock mass during blasting of explosive charges has been made by a number of scientists, the latest results of which are presented in the studies [25]-[29].

Although much work has been done and research has progressed in assessing the impact of mass stressed state on the resulting impact of blasting, there are differing views among researchers. To date, there is no definitive scientifically-based approach to determining the rational parameters of drilling and blasting operations when driving mine workings [30]. Improving the efficiency of drilling and blasting operations, taking into account the above factors, is an important practical and scientific challenge, the solution of which will reduce costs per unit of mined mineral.

Existing methods for selecting parameters for drilling and blasting operations do not fully take into account the principal stresses acting in the mass, structural and strength properties of rocks. The authors of the paper [31] believe that fracture formation along the line of the predicted split during blasting occurs due to the concentration of tensile stresses in the blast-hole exceeding the maximum rock strength values. Methods of mathematical theory of elasticity are widely used to study the stress-strain state of rocks surrounding the mine workings [32]-[35].

To date, ore dilution is an unsolved problem in all low-thickness deposits. The consequence of dilution is an increase in the cost of transporting and processing the ore, thereby increasing the cost of the mineral [36]. Thus, the problem of ore dilution during mining of low-thickness steep-dipping ore bodies using a system of sublevel drifts is an urgent scientific-practical task for both researchers and producers.

In order to control ore dilution at the Akbakai deposit, experimental tests have been conducted to determine effective drilling schemes for stope chambers when mining low-thickness ore bodies using a system of sublevel drifts. Reduction of ore dilution requires a comprehensive study of structural peculiarities, strength properties, stress-strain state of the rock mass and the seismic blast force impact on the out-contour rock mass.

This paper is aimed at substantiating the optimal parameters of blast-hole ore breaking to reduce dilution when mining low-thickness vein deposits using a system of sublevel drifts. To achieve the purpose set, the following objectives should be solved:

– perform geotechnical mapping of mine workings according to International Rock Mechanics Community (ISRM) standards on rating classifications;

– construct a geotechnical block model of a deposit divided into structural domains based on rating classifications;

– conduct computer modeling of the rock mass using the finite element method to determine possible zones of inelastic deformations around the mined-out space;

– develop improved technological schemes for drilling blast-holes depending on the rock stability rating;

– conduct experimental blasts using the developed drilling schemes to reduce ore dilution when mining low-thickness deposits using a system of sublevel drifts.

These objectives will not only achieve the purpose set, but also provide new methods and solutions for efficient operation in low-thickness deposits, which is of great importance for the mining industry.

2. Methods

At present, rating assessments of the state of rock masses are widespread when assessing the stability of mine workings. Rating rock mass classifications (systems) is a typology of the complexity of engineering-geological conditions of a field, based on a complex quantitative assessment of components of engineering-geological conditions, determining complexity and their quantitative characteristics, which allow a sufficiently clear identification of favorable and unfavorable rock mass areas [37]-[41]. The essence of the assessment is to assign a numerical value (rating) to a certain rock mass area, obtained as a result of studying the main engineering-geological characteristics – fracturing, strength, nature of the filler, groundwater action, rock weathering degree and a number of others. One of the main advantages of rating systems is the numerical representation of results, which allows constructing models of the required accuracy and detail.

The Akbakai mine conducts geotechnical orientated drilling to determine the mass stability rating. The drilling program is aimed at studying the physical-mechanical properties of rocks and determining the mass stability rating according to the standards of the international rock mechanics community. Drilling is conducted across the ore bedding in descending and ascending order. The number of positions is 8, the distance between positions is about 25 m. Drilling operations are planned to be conducted by the Levent-1001 machine. Drilling diameter is TT-48 mm. The total volume of drilling is 160 linear m, the number of blast-holes is 16 pcs. Figure 2 below demonstrates a sectional view of the geotechnical drilling project blast-holes.

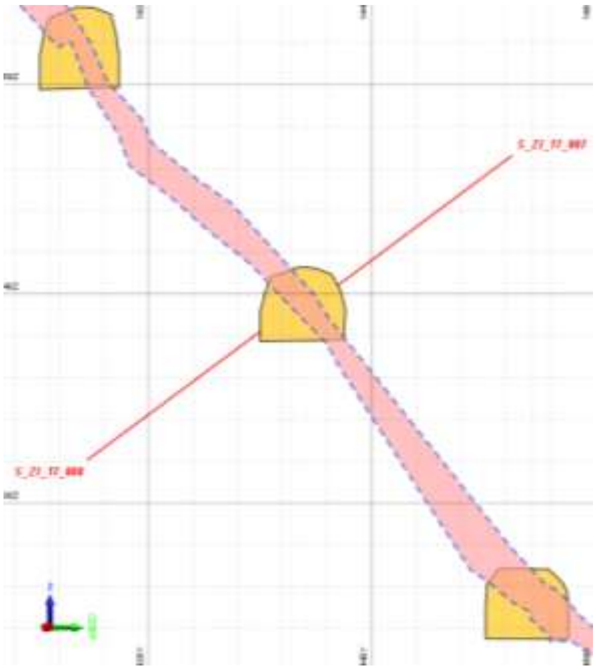


Figure 2. View of geotechnical blast-holes (sectional view)

The geotechnical program for mapping the walls of the mine workings and blast-holes is focused on mapping the Geological Strength Index (GSI) and the Rock Quality Index (RQD). Mapping is aimed at obtaining more structural data from active mine workings between the 520-580 m horizons for processing in kinematic analysis and further rock mass stability analysis. The mine working wall mapping program is conducted mainly on the eastern flank of the field, where the main geotechnical hazard can be identified.

Figure 3 below illustrates the percentage distribution of geotechnical intervals documented within geomechanical zone boundaries.

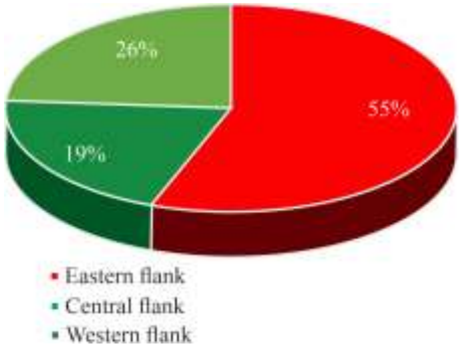


Figure 3. Percentage distribution of intervals (mine working wall mapping) documented within geomechanical zone boundaries

Structural documentation is prepared using the Trimble DeviCore BBT diameter HQ core orientation system. Orienting marks are placed on the end core surface, after which the structural elements (α/β angles) are measured from the lowest point of the disturbance ellipse. Sixteen geotechnical blast-holes are drilled with oriented core sampling using the DeviCore system, as this system has proven to be a reliable core orientation tool. After the core is extracted from the core receiver, all individual core pieces are joined until they fit tightly together. Then, an orientation line is drawn, starting from the end core surface mark.

Based on the results of structural documentation of blast-holes and geotechnical mapping of mine working walls, a database is created for further construction of a 3D block model. Based on the rating classification of the Akbakai deposit rock mass, a geotechnical block model is constructed using the RQD, Q, RMR and GSI rating classifications.

Figure 4 below demonstrates the geotechnical block model for the Pologaya vein of the Akbakai deposit.

The Akbakai deposit is divided into 3 geotechnical domains (areas) to form a 3D geotechnical model. Geotechnical domains are named as follows: Western, Eastern and Central. Domains are determined primarily by structural and strength properties, but fracture intensity (within the geological column), alteration and weathering degree are also considered.

The differences in rock properties between the domains of the western, eastern and central sites are taken into account when modeling the rock mass stability. There is some difference in the rock characteristics of these domains, so they are divided into distinct domains, but the difference is not sufficient reason to divide the mass into even more domains.

At the Akbakai deposit, above-plan ore dilution occurs during stope operations by breaking parallel blast-holes. Figure 5 shows a single drilling scheme, which is used in all veins with a thickness of up to 2.5 m.

For example, when blasting the drill rings on the western flank of the Pologaya vein using this scheme, ore dilution often does not exceed the planned rates, while when blasting the drill rings on the eastern flank, above-plan dilution occurs. It is worth assuming that the different effect from one drilling scheme is due to the variability of structural and strength mass properties.

The analysis presented in Figure 6 on the determination of actual dilution rates depending on the stability rating of the mass shows that the rock stability category directly affects the percentage of dilution.

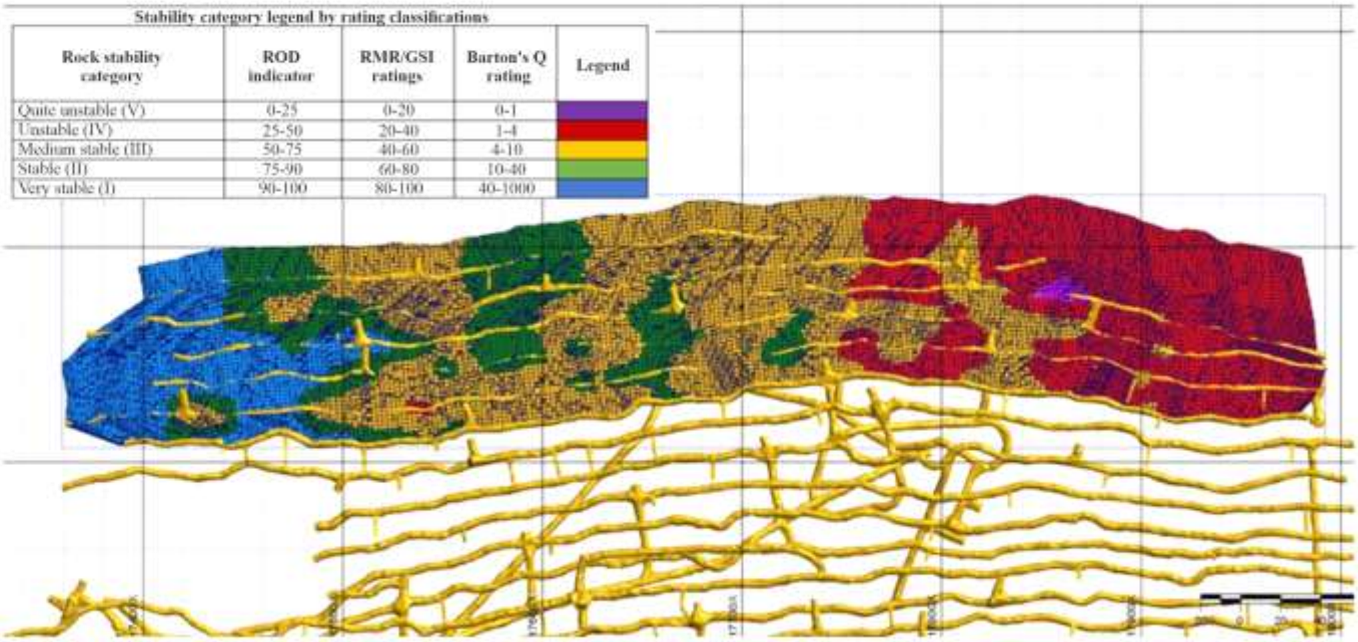


Figure 4. GSI-rated geotechnical block model

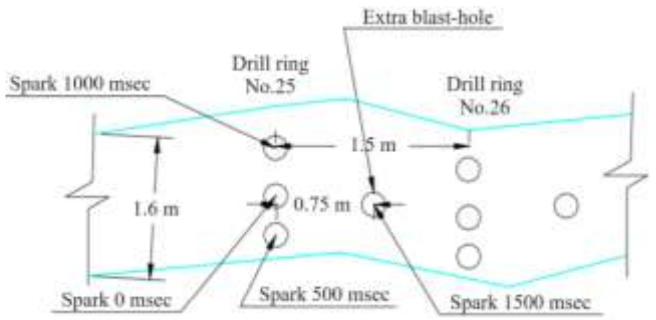


Figure 5. Drilling scheme applied before optimizing drilling parameters

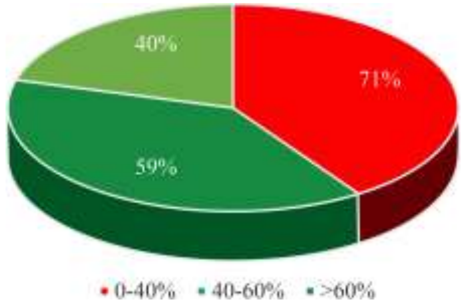


Figure 6. Actual dilution rates by GSI rating

Based on a set of geotechnical works, comparative, statistical and numerical analyses, a decision is made to conduct experimental blasts aimed at reducing ore dilution by changing blast-hole drilling schemes depending on the mass stability rating and vein thickness. The experience of the Cracow mine, located in Australia, is used to select optimal schemes for drilling and blasting operations. The geological and mining-engineering characteristics of the Cracow deposit are similar to those of the Akbakai deposit. The Cracow mine uses a "Zigzag" drilling scheme for veins up to 1.0 m thick. The drilling scheme is shown in Figure 7.

As a result, experimental blasts conducted on the eastern flank of the sublevel drift No. 11 of the Pologaya vein using this scheme, give unsatisfactory results.

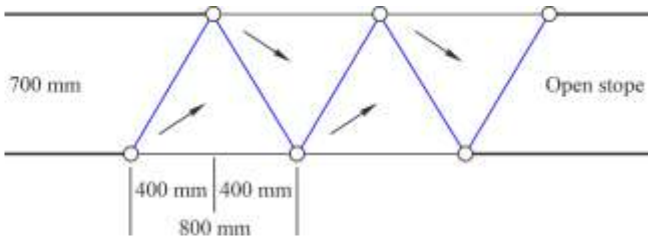


Figure 7. Blast-hole drilling scheme used at Cracow mine

It is assessed that the caving of the host rocks is influenced by the location of the blast-holes at the "ore-rock" contact, and in this regard the results do not have a positive effect.

Numerical analysis in RS2 software is performed to determine the optimum parameters of technological drilling schemes depending on the rock mass stability rating. The purpose of the numerical analysis is to determine the zones of inelastic deformation around the mined-out space, for predictive ore dilution assessment. The advantage of RS2 software compared to similar programs intended for numerical analysis is the parameter allowing determining the rock mass disturbance (D-factor) from the quality of blasting operations [42]. The D-factor analysis is the main indicator for substantiating technological schemes for drilling blast holes [43]. Numerical analysis is performed by finite element method using the generalized Hoek-Brown criterion [42]. Initial data for numerical modeling, obtained from a set of performed geotechnical works, are used to determine the stability rating, strength and structural properties of the mass by laboratory and field methods. Modeling is performed on mine working sites with GSI ratings between 20% and 100%, as well as D-factor between 0 and 0.8. After computer modeling, the results are compared with actual breaking results to determine the actual D-factor value.

3. Results and discussion

According to the sections presented in Figure 8 of the actual state of the mined-out space based on the survey data, it is evident that in the entire mine working length, ore dilution varies depending on the rock mass stability rating.

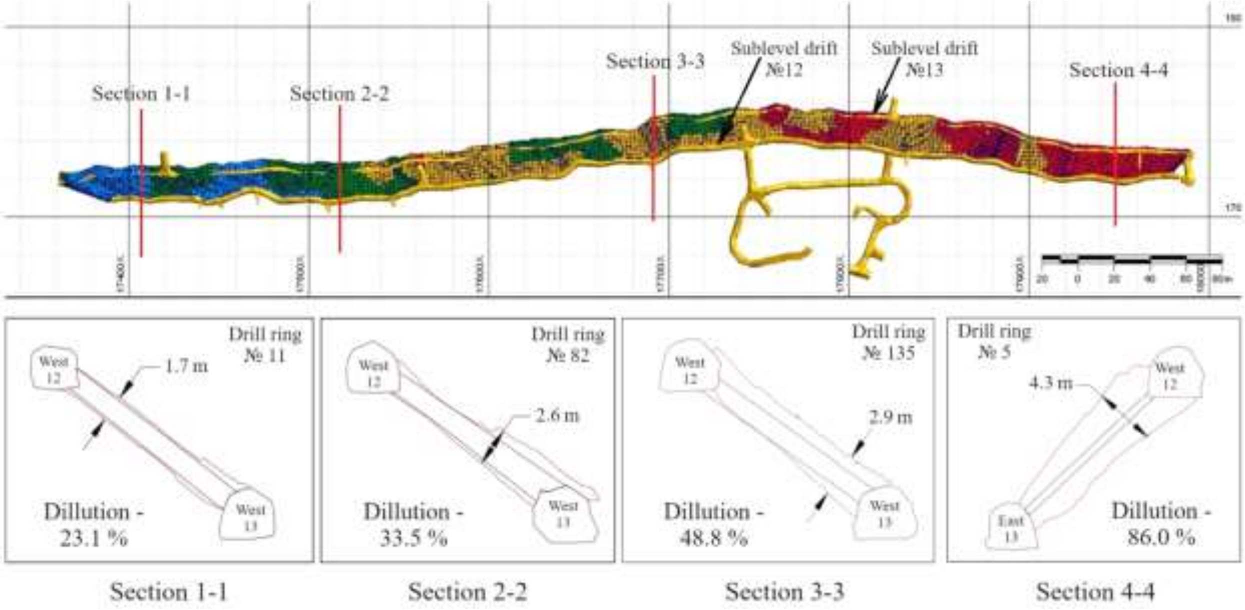


Figure 8. Comparison of actual results of breaking the stope chambers according to the rock mass stability rating

It should be noted that the drilling scheme presented in Figure 5 is used throughout the entire mine working length.

Figure 9 shows a graph of the change in ore dilution depending on the rock mass stability rating. The graph is constructed based on statistical analysis of previously mined-out sublevels 8, 9 and 10.

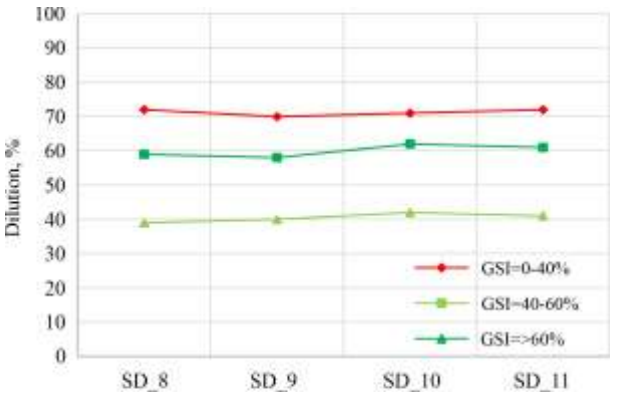


Figure 9. Graph of the change in ore dilution depending on the rock mass stability rating

Statistical analysis shows that above-plan dilution is directly influenced by the blast force in the process of breaking the stope chambers by blasting deep blast-holes. Consequently, correcting drilling schemes depending on the rock mass stability rating may be the key to controlling above-plan ore dilution during breaking of stope chambers.

Figure 10 graphically demonstrates the D-factor mass disturbance variation depending on the category of rock stability according to the GSI rating. The constructed dependency graph indicates that in order to reduce the host rock distur-bance, technological schemes for drilling blast-holes should be developed taking into account the rock stability category.

Based on this dependency graph, the following logarithmic equation can be formulated to determine the D-factor:

$$D = -0.463 \ln(GSI) + 0.0385. \quad (1)$$

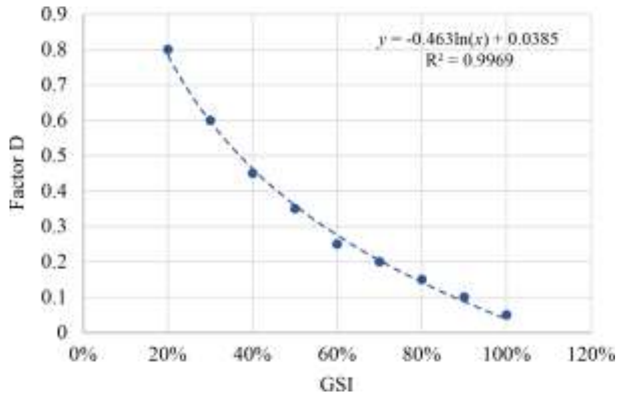


Figure 10. Dependency graph of the D-factor on the rock mass stability rating

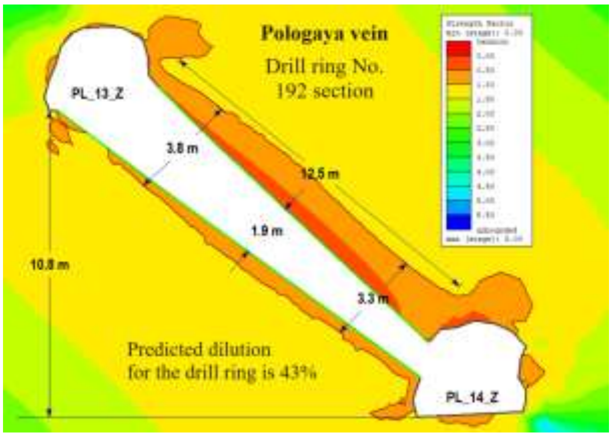
Based on this dependency graph, experimental blasting will be conducted, aimed at reducing the disturbance of out-contour rocks from the blast impact of by optimizing the passports of drilling and blasting operations.

Figure 11 shows a comparison of the computer modeling results of the predicted dilution estimate (43%), where $GSI = 50$ and $\text{D-factor} = 0$ with the actual dilution results (56.8%). From this comparison, it can be determined that the actual dilution exceeds the predicted figures by 13/8%, that is, the actual D-factor is 0/4. To improve the accuracy of dilution prediction, a more thorough examination of the factors influencing this process is needed, as well as improved modeling, possibly including additional variables or more accurate data.

To determine the dependence of the disturbance factor influence on ore dilution, a quantitative numerical analysis of large volumes of predicted dilution rates and a comparative analysis with actual results are performed. In addition, the dependency graph of ore dilution on the mass disturbance factor is plotted on the basis of inverse calculations.

The results of the analysis indicate a significant influence of the rock mass disturbance factor on the ore dilution process. The dependency graph allows a visual assessment of the nature of this dependence: perhaps it can be linear, quadratic or other.

(a)



(b)

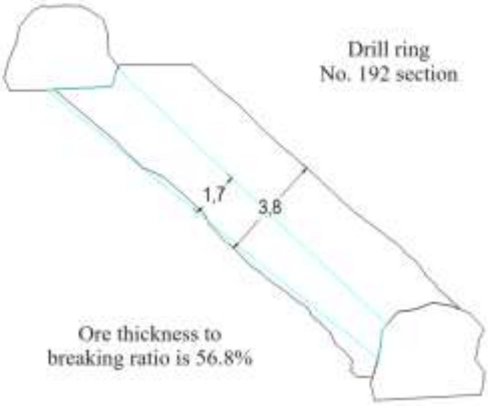


Figure 11. Comparison of predicted-actual ore dilution: (a) computer modeling results; (b) actual results

Understanding this dependence is important for predicting and optimizing the dilution process, as it enables more effective management of the factors that influence its results.

Consequently, it should be assumed that technogenic factors, such as the seismic blast force impact on the out-contour rock mass, lead to the destruction of the host rocks and directly to the above-plan dilution. Dependency graph in Figure 12 shows that the actual disturbance D-factor is 0.7. That is, the applied drilling scheme (Fig. 5) does not provide stability of the host rocks and in order to reduce ore dilution it is necessary to optimize drilling schemes depending on the rock stability rating.

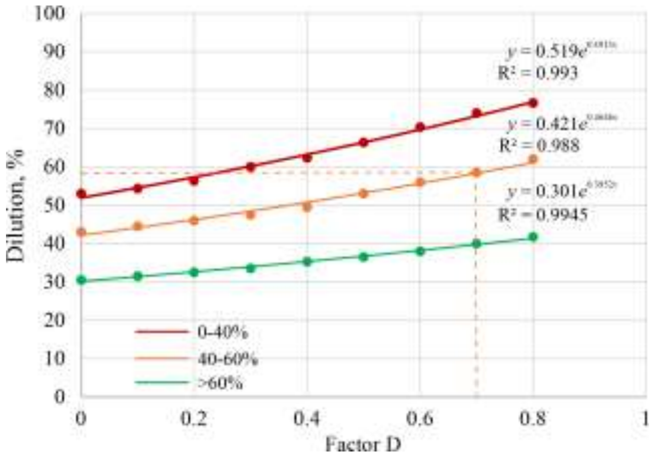


Figure 12. Dependency graph of ore dilution on D-factor

Further, on the lower sublevels No. 12 and 13, the blast-hole drilling scheme are corrected, as shown in Figure 13 with the blast-holes located in the middle of the ore vein.

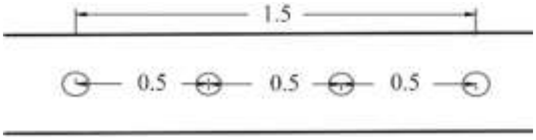


Figure 13. Scheme for drilling experimental blasts

Experimental blasts performed on No. 12 sublevel of the Pologaya vein between drill rings No. 17-37, 30.0 m long, have a positive effect and contribute to reducing ore dilution compared to the overlying sublevels. In order to confirm the results of experimental blasts conducted on the eastern flank of sublevel drift No. 12, it is decided to repeat the experimental blasts on the eastern flank of sublevel drift No. 13. Figure 14 shows in red the areas of the experimental tests.

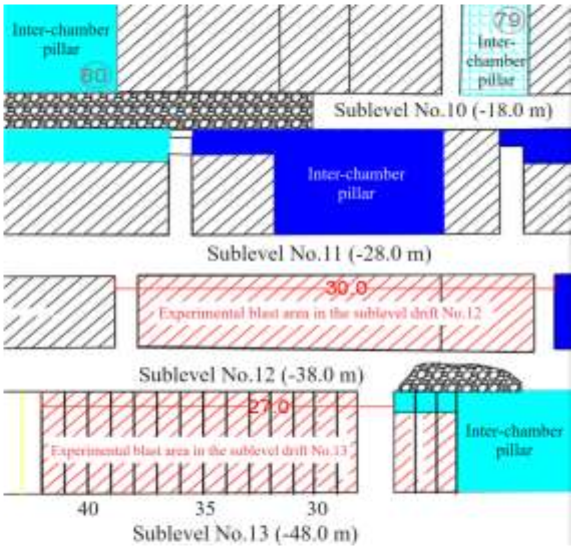


Figure 14. Experimental blast sites

Figure 15 below clearly shows is a sectional view of the actual results of the stope chamber minable width in the sublevel drifts No. 10-13 on drill ring No. 36. The remaining results of experimental blasts between drill rings No. 26-42 with a total length of 27 m are summarized in a comparative diagram presented in Figure 16.

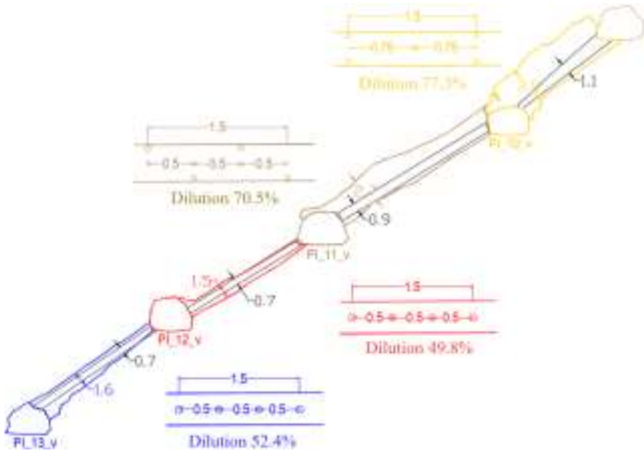


Figure 15. Ore dilution on drill ring No. 36

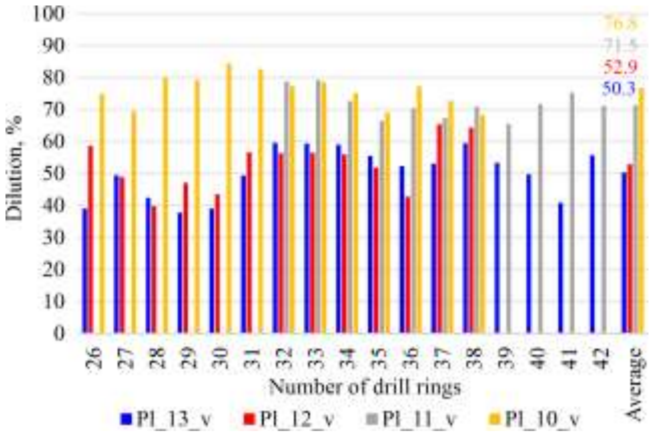


Figure 16. Comparative ore dilution diagram

Based on the comparative analysis results shown in Figure 16, it follows that the experimental blasts performed on sublevel drifts No. 12 and 13 had a positive effect and contributed to reducing ore dilution by about 20%. However, it is worth noting the fact that ore dilution is also influenced by vein thickness in addition to actual minable width. For example, the average vein thickness is 0.84 m in the sublevel drift No. 11 and 0.92 m in the sublevel drift No. 10, while in the sublevel drifts No. 12 and 13 the average thickness of the vein is 0.66 m.

By comparing minable widths, potential areas for optimizing the ore mining process can be identified. For example, if the results of blasting overlying sublevels differ significantly from those of the experimental blasts, this may indicate the need to correct the parameters or methods for breaking.

To correctly compare the results of experimental blasts with the results of breaking overlying sublevels, a comparative diagram by minable width is constructed, which is shown in Figure 17.

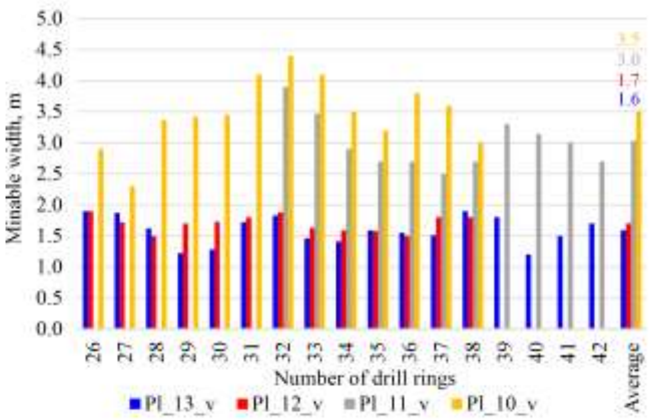


Figure 17. Comparative diagram by minable widths

From the comparative analysis data, it should be assumed that the minable widths in experimental sublevels No. 12 and 13 decrease approximately by 50% in comparison with the overlying sublevels. Based on the results of the experimental blasts, it should be stated that the test results are positive and this scheme is optimal for breaking stope chambers in rocks with GSI rating up to 40% and with vein thickness up to 1.2 m.

The drilling scheme used in experimental blasts will only be used in low-rated rocks (less than 40%) and no more than 1.2 m thick. In areas with average stability rating (40-60%) and in stable areas with rating above 60%, based on experimental blasts, optimal technological schemes for drilling blast-holes, contributing to the reduction of ore dilution, are substantiated.

Figure 18 shows a section along the line of drill ring No. 77 to visualize the results of the experimental blasts performed in medium stability rocks, and the rest of the results are summarized in the comparison diagram presented in Figure 19.

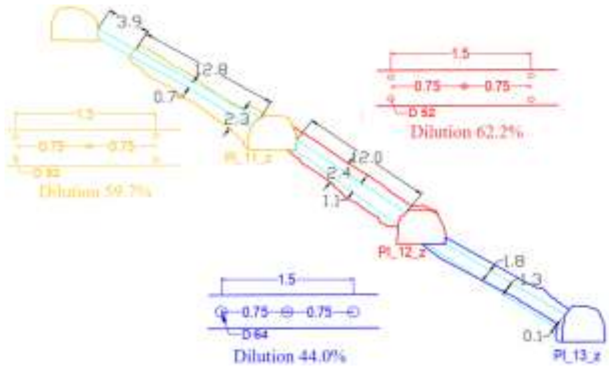


Figure 18. Ore dilution on drill ring No. 77

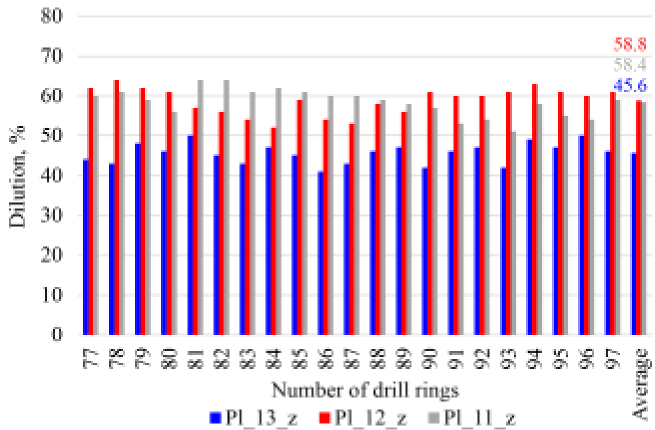


Figure 19. Comparative analysis based on the results of experimental blasts on medium stability rocks (GSI = 40-60%)

Recommendations for selecting drilling schemes are presented in Figure 20. These results are substantiated on the basis of a set of geomechanical studies and experimental-industrial tests. When choosing the optimal drilling schemes for blast drill rings, the RMR/GSI stability rating of the mass and the ore deposit thickness are taken as the basis.

These recommendations are intended to select drilling schemes to reduce ore dilution depending on the ore body thickness and the rock mass rating. Drilling schemes are developed on the basis of conducted experimental blasts, as well as visual, numerical, analytical and comparative analyses of the minable width actual results of the stope chambers with the predicted results of computer modeling of rocks. Table 1 provides the economic efficiency calculations comparing the results "before"/"after".

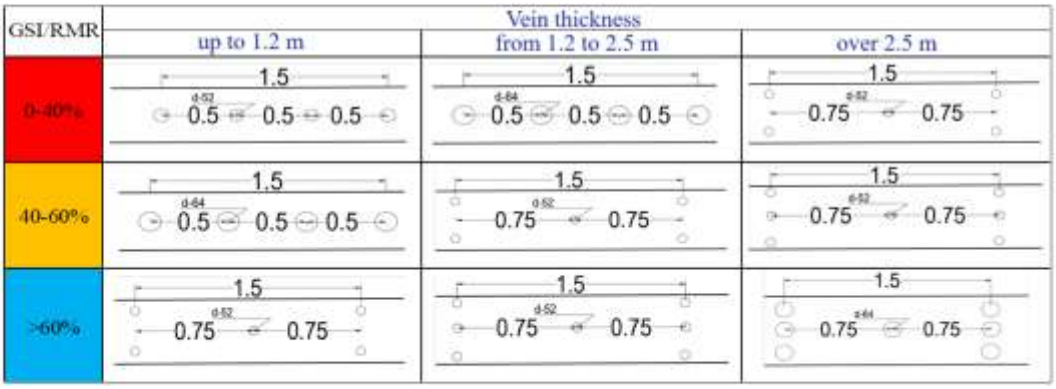


Figure 20. Optimal drilling schemes for the stope chamber

Table 1. Comparative analysis of economic efficiency

Cost name	Before / after	Deviation ± / %	Actual before / after, $/ton
Drilling	0.74/0.59	0.15/20	3.6/2.9
Blasting	0.74/0.59	0.15/20	2.2/1.8
Rock mass removal	3145/1622	1523/48	2.4/1.2
Transportation	3145/1622	1523/48	2.3/1.2
Fastening	–	–	0.1/0.1
Communications	–	–	0.1/0.1
Variable costs	–	–	10.7/7.3
Staff costs	–	–	12.6/12.6
Spare parts, service, repair	3145/1622	1523/48	3.2/1.7
Electricity	–	–	1.5/1.5
Fixed costs	–	–	17.3/15.8
Laboratory analyses	–	–	0.6/0.6
Security staff	–	–	1.0/1.0
Land transportations	–	–	2.0/2.0
Support services	–	–	4.5/4.5
Overhead costs	–	–	8.1/8.1
Total Akbakai output	–	–	36.1/31.1

The economic efficiency of technological developments is analyzed by comparing the actual variable cost indicators of the above sublevels No. 10-11 with the experimental sublevels No. 12-13.

The following economic indicators have been obtained from the comparative analysis:

– drilling costs is reduced by 20%;

– the cost of specific explosive substance consumption decreases by 20%;

– the cost of removing and transporting rock mass is reduced by 48%.

During the experimental blasts, the cost of ore mined has decreased by approximately 5%, which is a very positive result. Based on the above, it can be summarized that in the course of the research work at the Akbakai deposit, a reduction in ore dilution has been achieved through optimization of drilling and blasting operations when mining low-thickness ore bodies using the mining system of sublevel drifts.

There are a number of prospects for further research in the field of optimization of blast-hole ore breaking using a system of sublevel drifts. Firstly, an in-depth study of the mechanisms of the impact of drilling parameters on the structural and strength rock characteristics is possible, which will make it possible to more accurately determine the optimal parameters for minimizing mining damage. In addition, in the future it is important to pay attention to the development of mathematical models that can predict the rock mass behavior under conditions of blast-hole breaking and blast action. Another important area of research is assessing the effectiveness and practical applicability of methods for optimizing the mining process.

4. Conclusions

Based on a set of geotechnical studies, a 3D geotechnical block model of the deposit has been constructed according to the rock mass rating classifications. Based on the constructed block model, it is possible to divide the deposit into 3 structural domains (Eastern, Central and Western) according to the rock mass stability rating. Structural domains are fundamental indicators to substantiate the optimal parameters for blast-hole breaking.

Numerical modeling of rock mass by the finite element method has been performed for the predictive assessment of ore dilution during the mining of stope chambers. The conducted comparative analysis of the predictive assessment with actual dilution indicators has revealed that actual dilution rates exceed the predicted data, which indicates the influence of technogenic factors on the host rock stability. As a result of comparative and statistical analyses, a graph of ore dilution dependence on the mass disturbance factor (D-factor) has been plotted, which proves the necessity of correcting the blast-hole drilling schemes depending on the rock stability rating in order to maintain the host rock stability.

Technological schemes for drilling blast-holes have been developed based on the constructed dependency graph of dilution and rock stability rating. Experimental blasts have been conducted using the developed drilling schemes, the results of which are compared with the results of previously mined-out overlying horizons. Based on the results of experimental blasts, the optimum technological drilling schemes depending on the rock mass stability rating and the ore deposit thickness have been substantiated, which allows reducing ore dilution in the course of stope operations.

The economic efficiency of the conducted experiments has been substantiated by a comparative analysis of the actual results of production cost for mining operations prior to the execution of the work with the results obtained in the course of the conducted research. The cost of mining 1 ton of ore is expected to decrease by 13.8%, which determines the economic efficiency of technological developments obtained as a result of the research performed.

Author contributions

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

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Conflicts of interests

The authors declare no conflict of interest.

Data availability statement

The original contributions presented in the study are included in the article, further inquiries can be directed to the corresponding author.

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Управління збідненням руди при відпрацюванні малопотужних рудних тіл системою підповерхових штреків

А. Мусін, А. Кидрашов, Ж. Асанова, Є. Абдрахман, Д. Іваділінова

Мета. Дослідження спрямоване на обґрунтування оптимальних параметрів свердловинного відбивання руди для зниження руйнування породного масиву при розробці малопотужних жильних родовищ із використанням системи розробки підповерховими штреками (SD). Основна увага приділяється аналізу впливу вибуху на законтурний масив у процесі свердловинного відбивання.

Методика. Побудова тривимірної блокової моделі здійснювалася із використанням рейтингових класифікацій гірських порід на основі вивчення їх міцнісних властивостей та структурних особливостей. Визначення зон непружних деформацій навколо очисної виїмки проводилося за допомогою чисельного аналізу методом скінченних елементів у двомірній постановці. Оцінка дослідних вибухів проводилася з урахуванням зміни схем буріння вибухових свердловин залежно від рейтингу стійкості.

Результати. У процесі дослідно-промислових випробувань були обґрунтовані раціональні схеми буріння вибухових свердловин, що сприяють збереженню стійкості вміщуючих порід при розробці малопотужних жил.

Наукова новизна. Обґрунтовано ефективні методи зниження збіднення руди, які враховують не лише міцнісні властивості та структурні особливості гірських порід, але також їх сейсмічну дію під час вибуху на стійкість законтурного масиву при відпрацюванні малопотужних покладів із використанням системи підповерхових штреків.

Практична значимість. Полягає у можливості мінімізації відсотка збіднення корисної копалини при відпрацюванні малопотужних рудних тіл із використанням системи підповерхових штреків, що дозволить значно знизити собівартість руди, що видобувається, за рахунок скорочення її втрат, викликаних руйнуванням породного масиву в процесі видобування.

Ключові слова: *руда, збіднення, буропідривні роботи, очисний простір, породи, родовище*

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