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Solutions to improve the mining technology diagram for ore bodies at Vi Kem Copper Mine, Lao Cai province, Vietnam

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Abstract. Designing and selecting a mining technology scheme for ore body conditions is complex because it depends on many factors. The technology schemes need to ensure efficiency and safety in the production process. Therefore, it is necessary to propose solutions to improve the mining technology scheme to exploit ore bodies to meet the current production requirements at underground mineral mines of the Vinacomin – Minerals Holding Corporation. Based on the current mining plan and mining technology design scheme of the Vi Kem Copper Mine, the authors conducted a survey, analysis and evaluation, thereby proposing solutions to improve the mining technology scheme for ore bodies at the Vi Kem Copper Mine and selected an area in the mine to conduct design calculations for the proposed solution. To achieve the research results presented in this article, the authors used methods such as data collection, analysis and synthesis, field surveys, analysis of results and evaluation, combined with theoretical approaches to calculate the experimental design area. The proposed options to improve the mining technology diagram are highly feasible when applied. There is a high possibility of applying mechanization in technological stages, increasing productivity and production efficiency of the mine, increasing ventilation capacity, and improving labor safety. The improved technology is basically the same as the preparation plan of the old mining technology diagram. However, each item will be optimized to increase the application of mechanization, bringing about production efficiency and labor safety. The research results show high feasibility when applying the improved technology diagram. Technical calculations show that the mining capacity of each chamber is 1.76 times higher, and labor productivity is nearly 3 times higher than the current technological scheme.

Keywords: *mining technology, solution, ore body, improvement, supplement, Vi Kem Copper Mine.*

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1. Introduction

The Vinacomin – Minerals Holding Corporation is the only national management unit assigned to manage and exploit major mineral mines (including all ore mines) in Vietnam. With the distribution characteristics and location of the ore bodies, the underground mining method has been applied in several mines, such as: Vi Kem Copper Mine in Lao Cai province; Cho Dien Zinc-lead mine in Thai Nguyen province; Lang Hit Zinc-lead Mine in Thai Nguyen province; Tay Nam Nui Phao Tin Mine and Cuc Duong Zinc-lead Mine in Thai Nguyen province. Vi Kem Copper Mine is one of the mines belonging to the Vinacomin – Minerals Holding Corporation. This is also one of the large underground mining ore mines in Vietnam.

Many mining technology diagrams and systems can be selected and applied to each specific ore body condition in different mines. However, to select a suitable mining technology scheme, it is necessary to analyze and evaluate the influencing factors, the main factors include thickness, slope angle of the ore body, stability of the ore body, mining depth, gas content in the mining area, the impact of mining on the surface works, as well as the conditions of the rock mass such as primary stress state, mechanical strength, layering, degree of cracking, properties of cracks.

The selection of an appropriate mining technology scheme is important because it affects the following issues:
– mineral recovery and ore impoverishment;
– the level of development of the necessary tunnels;
– capital requirements and operating costs;
– the type of necessary equipment selected and applied;
– cycle time and sequence of operations;
– annual mine production (tons/year);
– potential risks and occupational safety.

Actual mining operations at ore mines show that the collapse of rock blocks after mining ore blocks has caused deformation and tunnel collapse incidents, posing potential safety risks to people and equipment that may occur when mining ore bodies, especially at thick and steep ore bodies.

Today, many studies are related to designing mining technology diagrams, optimization, numerical modeling,

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assessments, and proposals for reasonable mining solutions for ore bodies in different mines and conditions. Some typical studies on development and solutions of ore mining technology [1, 2], studies related to forecasting models, the impact of mining technology on ore quality [3-6], improvement and development of ore mining technology [7-11], studies on ore mining technology in complex conditions, intelligent and automatic mining technology [12-17], forecasting blasting vibration and optimizing ore mining design [18-21], geological forecasting models of ore bodies [22-27], studies on assessing geological conditions and ore exploration [28-31], stability assessment of stopping operations and transportation equipment performance in ore mining [32-35].

In Vietnam, due to the conditions and characteristics of the distribution of ore types in different regions, the mines are all small, scattered, and not concentrated. Therefore, investment in research for mining ore mines using underground mining methods is still limited, mainly manual mining, drilling and blasting methods. A few projects are related to improving drilling and blasting for tunnels [36]; the Vietnam Institute of Mining Science and Technology implements some projects. In 2024 at the Vi Kem Copper Mine, ore bodies were exploited using two mining technology diagrams, including Ore shrinkage stoping and sublevel stoping mining technology diagrams. However, according to the assessment and analysis at the actual site, these mining technology diagrams have not yet brought about efficiency; some technological design steps are not reasonable, affecting labor productivity and mining output, potentially posing a risk to worker safety. In 2025, according to the exploitation plan, output still needs to be maintained and increased higher than in 2024, especially requiring increased output in the following years to meet the actual production situation of the Vi Kem Copper Mine. Thus, the research and improvement of the mining technology diagram for ore bodies at the ViKem Copper Mine is urgent for the current Vietnamese mineral mining industry. The selection of a suitable mining technology diagram for ore bodies is essential. It is the basis for the management agency and the mining mines to choose technology to ensure safety and improve production efficiency. This is a study of scientific and practical significance for the current Vietnamese mineral mining industry. The research results of the paper are applied to the Vi Kem Copper Mine of the Vinacomin – Minerals Holding Corporation to exploit ore bodies with medium to thick thickness to ensure safety and efficiency in the mining process.

2. Materials and methods

2.1. Study area

The research was conducted at the Vi Kem Copper Mine in Lao Cai province, Vietnam.

The surface boundary is defined in accordance with Mineral Exploitation License No. 1688/GP-BTNMT dated July 12, 2017, covering an area of 155 hectares. The boundary extends along the exploration axis from geological exploration line 65a (adjacent to the licensed iron ore mining area) to line 11d, with an average width of approximately 550 meters, ensuring that all mining works are encompassed within the licensed area.

The depth boundary of the mine is defined at the deepest reserve level of −90 m. The location of the study area is illustrated in Figure 1.

Figure 1. Location of Vi Kem Copper Mine (modified from [37])

The map (Figure 1) indicates the spatial position of the Vi Kem Mine in relation to nearby settlements and regional infrastructure, providing a geographical context for study.

2.2. Geological characteristics of the study area

2.2.1. Characteristics of lithological distribution

In the exploration area, the following main types of soil and rock are identified: cover layer, weathered layer, gneiss-biotite, granitogneiss, amphibolite, two-mica quartz schist, metamorphic rocks, and several other rocks interbedded with the above formations. The cover layer extends across the entire surface of the exploration area, with a thickness ranging from 5 to 15 m. It consists primarily of red-brown, gray-brown, and black-brown claystone and gravel. The layer is soft and friable, prone to landslides when saturated with water or disturbed by construction activities.

Gneiss-biotite rock interbedded with two-mica quartz schist occupies approximately 55% of the exploration area. The rock is gray-white to ash-gray in color, with a medium- to fine-grained texture, banded structure, and blocky composition comprising quartz, feldspar, and biotite.

Gneiss-biotite associated with migmatite typically surrounds and hosts the ore bodies. It is hard, brittle, fractured, and broken, with low engineering stability in fractured zones, exhibiting a compressive strength below 900 kg/cm^2. Rocks located farther from the ore bodies are generally compact, less fractured, and display a durable structural integrity.

Metamorphic rocks constitute the principal ore-bearing formations within the exploration area. Their thickness varies from 0.5 to 30 m. These rocks are typically gray-black or blue-black, containing copper mineralization filling fractures and cavities. Ore-bearing metamorphic rocks are dense, compact, and less fractured, indicating high compressive resistance and stable structural characteristics.

Granitogneiss covers about 20% of the exploration area. It is gray-white, with a granular texture and blocky structure, composed mainly of quartz and biotite. The formation is typically located along the margins of ore bodies.

Fractured sections of granitogneiss show a compressive strength below 863 kg/cm^2, whereas intact zones have a stable structure with compressive strengths reaching up to 2156 kg/cm^2 in some locations. Additionally, minor occurrences of amphibolite, granite, and recrystallized limestone are observed within the area. These formations are 0.5-5 m thick, exhibit a blocky structure and granular texture, and are generally dense and compact.

2.2.2. Stratigraphic characteristics and physical properties of soil and rock

Based on the mechanical strength and the results of the analysis of the physical properties of soil and rock, the relative stability with water, combined with the naked eye observation and the ore-bearing characteristics of soil and rock according to the geological perspective, the soil and rock of the mine are divided into the following layers.

Layer 1. Rock with poor geological structure.

This layer includes the entire weathered crust extending over the exploration area. The composition consists mainly of reddish-brown and gray clay, two-mica quartz schist, and gray-white to yellow-gray gneiss-biotite. Due to intense weathering, the material is soft, loose, and highly susceptible to collapse when exposed to water or subjected to dynamic construction loads. The layer's average thickness ranges from 5 to 15 m, reaching up to 52 m in some locations.

Layer 2. Rock layer above the ore body.

This layer represents a relatively stable geological structure, composed mainly of gneiss-biotite, granitogneiss, two-mica quartz schist, and other interbedded rocks. The thickness of the layers varies between 0.5 and 30 m. The rocks are slightly weathered, exhibiting cracking and fracturing in some sections – areas with increased fracturing show poorer structural stability. Groundwater may be present under slight pressure within fractures and cavities, but its quantity and impact on construction activities are negligible.

Layer 3. Ore-bearing rock layer.

This layer consists of metamorphic rocks and migmatized gneiss-biotite, which form the primary ore-hosting formation. The rocks are dense, compact, and slightly fractured, displaying granular and banded textures. Most layers are structurally solid and coherent, although some localized zones show moderate to intense fracturing.

Layer 4. Rock layer below the ore body.

This layer comprises gneiss-biotite, granitogneiss, amphibolite, and lithified limestone. The rocks are generally solid and compact, with few fractures and a durable geological structure, providing a stable foundation for construction and mining activities.

The average analytical results of physical and mechanical properties for these layers are presented in Table 1 [38].

Table 1. The average results of the analysis indicators

№	Indicator names	Unit	Value		
			Layer 2	Layer 3	Layer 4
1	Humidity:				
	Dry humidity, W_{kg}	%	0.163	0.163	0.131
	Moisture absorption, W_{hn}	%	0.43	0.296	0.295
2	Porosity, n	%	1.136	0.835	0.83
3	Density, ∇	g/cm^3	2.729	2.859	2.788
4	Weight of volume:				
	– dry wind, γ_{kg}	g/cm^3	2.701	2.836	2.766
	– saturation, γ_{bh}	g/cm^3	2.711	2.845	2.774
	– absolutely dry, γ_c	g/cm^3	2.699	2.835	2.764
5	Compressive strength:				
	– dry wind, δ_n	kg/cm^2	760	1008	1203
	– saturation, δ_{bh}	kg/cm^2	721.4	960	1150
6	Tensile strength, δ_k	kg/cm^2	75.82	94.85	109.2
7	Internal friction angle, ϕ	degree	36.55	37.20	37.31
8	Cohesion, C	kg/cm^2	138.5	177.8	211.6
9	Firmness coefficient, f		7.439	9.046	10.21
10	Soft deformation coefficient, k		0.946	0.95	0.951

2.3. Mine's field reserves

The underground mine comprises 18 identified copper ore bodies, designated as TQ1.1, TQ1.2, TQ2.1, TQ2.2, and TQ1a.1, TQ1a.2, TQ5.1, TQ7.1, TQ7.2, TQ7a.1, TQ9.1, TQ9.2, TQ9.3, TQ10.1, TQ11.1, and TQ12.1.

Among these, eight ore bodies – TQ1.1, TQ1.2, TQ2.1, TQ2.2, TQ1a.1, TQ1a.2, TQ5.1, and TQ7.1 – have been classified as industrial ore bodies with approved reserves. The remaining bodies have been identified only as copper ore resources.

The calculation of copper ore reserves and resources was conducted based on geological data presented in the Additional Exploration Report of Copper Ore and Minerals in the Vi Kem Area, Bat Xat District, Lao Cai Province.

The total copper ore reserves (sulfur ore type) amount to 5.154 million tons, of which 0.247 million tons belong to grade 121 and 4.908 million tons to grade 122. The detailed reserves of individual ore bodies at the Vi Kem Copper Mine are presented in Table 2 [38].

Table 2. The reserves of the ore bodies at the Vi Kem Copper Mine

№	Ore body name	Ore block area, m^2	Reserve thickness, m	Volume of the block, m^3	Dry ore reserves, t	Copper content, %	Metal reserves, t
1	TQ1.1	163.697	3.51	574.194	1900.583	0.66	12.631
2	TQ1.2	86.188	4.87	419.504	1388.559	1.01	14.050
3	TQ2.1	52.015	2.94	152.675	450.390	0.65	2.942
4	TQ2.2	47.555	3.74	177.765	524.405	0.56	2.957
5	TQ1a.1	18.954	3.21	60.867	201.468	0.91	1.843
6	TQ1a.2	24.554	3.65	89.740	297.038	0.94	2.792
7	TQ5.1	18.378	2.96	54.414	160.521	0.50	806
8	TQ7.1	26.367	2.77	72.948	231.245	0.47	1.083
Grade 121 Reserve		17.946	4.15	74.474	246.509	0.88	2.166
Grade 122 Reserve		419.762	3.64	1527.632	4907.701	0.75	36.938
Total		437.708	3.66	1602.106	5154.210	0.76	39.104

These results were approved by the Mineral Reserves Assessment Council under Decision No. 837/QĐ-HĐTLKS, dated November 4, 2011.

2.4. Assessment of the current status of the mining technology diagram of the ore bodies

2.4.1. Opening up the mine's field

The Vi Kem Copper Mine is developed through a pair of inclined shafts for ore transport, material haulage, and mine ventilation, combined with crosscuts connecting the working horizons. After the construction of the inclined shaft station system at level +30, a haulage crosscut is excavated at the same level toward the TQ1.1 ore body to establish the transport horizon for the mine. The central ventilation system is formed by driving a ventilation crosscut at level +150 toward the TQ1.1 ore body, then excavating a ventilation tunnel at the same level in the TQ2.1 ore body, thereby completing the mine’s ventilation horizon.

2.4.2. Mine's field preparation

The mine field is divided into two main areas, the Southeast and the Northwest, separated by the geological line T.53[a]. Within each area, preparation works are organized according to ore body groups by excavating transport and ventilation decline zones and longitudinal levels corresponding to the transport horizons. The initial preparation phase is conducted simultaneously at three ore bodies: TQ2.1, TQ2.2, and TQ1.1, comprising a total of six mining rooms: two at TQ2.1 (2.1-27-BLQ and 2.1-33-BLQ), two at TQ2.2 (2.2-4-BLQ and 2.2-10-DVPT), and two at TQ1.1 (1.1-32-BLQ and 1.1-35-BLQ). The rooms are developed according to the technological schemes of ore shrinkage stoping and sublevel stoping. Manual drilling and blasting are applied to ore shrinkage stopping, while mechanized extraction is used for sublevel stopping. The average mining capacity of each room using ore shrinkage stoping technology reaches about 45000 t/year, whereas mechanized sublevel stoping can achieve up to 125000 t/year.

2.4.3. Mining plan and order

The mining sequence of the mine is carried out on the following principle [39].

With the characteristics of the geotechnical conditions of the mine, the project mobilizes 08 ore bodies that are qualified to be included in the mining design, the reserves of the ore bodies are scattered and not concentrated, and the metal content of each ore body varies. To neutralize the metal content and ensure that the mining content is brought to the processing plant evenly in each period, 2 to 3 ore bodies will be exploited simultaneously in a year. Mobilize and exploit first areas with certain and favorable geological conditions (thickness, slope angle, metal content) and mobilize other areas later.

In a group of ore bodies for the same level, the upper ore body is exploited first, and the lower ore body is exploited later. However, depending on the geological structure and the distance between the ore bodies, the lower ore body can be exploited first if it does not affect the mining room of the upper ore body. In an ore body, the upper layer is exploited first, and the lower layer is exploited later. Reserve areas near the tunnel protection pillars and constructions are arranged to be exploited later.

2.4.4. General assessment of the mining technology of Vi Kem Copper Mine

Currently, the primary mining method applied at the Vi Kem Copper Mine is ore shrinkage stoping, with geological ore reserves extracted by this technology amounting to 3.185 million tons, representing 62% of the total mine reserves. The remaining 38% of the reserves are mined using the ore sublevel stoping method. After implementing these technologies in production, a comprehensive assessment of the mining technology applied at the Vi Kem Copper Mine has been conducted, summarized below [39].

– For the ore shrinkage stoping mining technology diagram: applied to ore bodies and blocks with a 1.2 to 3.5 m ore body thickness.

This is a manual mining technology, low output, very high risk of unsafety because the ore bodies at Vi Kem Copper Mine are a group of ore bodies and have a vein shape, many layers, especially the junction between the ore body and the roof rock, floor rock. In the ore shrinkage stoping mining technology, workers must stand and work directly under the ore block, the rock has been affected by drilling and blasting and cracked above. Implementing temporary support is relatively complex, and there are no specific studies and assessments, so there are many potential risks of falling, rocks, landslides, and unsafe labor.

The ore shrinkage stoping mining technology diagram is only applied to ore bodies with a thickness of less than 3.5 m (no calculation of the roof pressure to choose the appropriate room width). However, in reality, the thickness of the ore body at Vi Kem mine is unstable. Many locations form nests, lenses with local thickness greater than 3.5 m. Therefore, when working through these areas, the roof or floor of the ore body is suspended on the side of the storage room. Due to the separation of layers between the rock and the ore body, it is easy for the roof and side of the storage room to collapse, causing safety issues and loss of resources during the mining process.

The construction organization is challenging to carry out as designed, specifically: the size of the ore after blasting is uneven, the ore discharge work is difficult due to the manual design of the discharge door, so there is a risk of the discharge door being blocked; when removing the ore, the amount of ore is uneven, so the floor of the room is uneven, leading to difficulties in organizing the next construction cycle.

– For the ore sublevel stoping mining technology diagram: pplied to ore bodies and blocks with a thickness of 3.5 to 7.8 m, with a length in the mining block direction of a strike line greater than 300 m.

Manual reverse charge dynamite design calculations are not feasible (using DK52-64 reverse jumbo, drilling depth 12 m). In reality, there is no investment in jumbo, because the cross-section of the tunnels has been reduced from 8.4 to 6.5, so the appropriate drilling machine has not been selected to put into the tunnel. Thus, this technology cannot be applied to production. In addition, the design has not calculated or designed mining technology for ore bodies with a thickness greater than 7.8 m.

2.5. Proposing solutions to improve the mining technology diagram for ore bodies at the Vi Kem Copper Mine

Based on the analysis and evaluation of geological conditions, hydrogeology and especially the rock structure of the ore bodies. The selection of mining technology diagram, as well as the arrangement of excavation of tunnels depends on factors such as thickness, slope angle of the ore body, stability of the ore body, cross-section shape of the tunnel, rock mass conditions, primary stress state in the rock mass, mechanical strength, layering, degree of cracking, properties of cracks, water content of the rock mass, depth of the tunnel, impact of mining, surrounding underground works, dynamic loads (if any), etc. Therefore, analyzing and evaluating influencing factors will help select and improve the mining technology diagram more effectively.

Based on the current conditions and the actual operational situation of the mine, the authors propose improvements to the mining technology diagram of the ore bodies in two main directions: for the ore shrinkage stoping mining technological scheme and for the ore sublevel caving mining technological scheme.

2.5.1. Ore shrinkage stoping mining technological diagram

This mining technological scheme is appropriate for ore bodies characterized by steep dips exceeding 75° and small thickness. However, several improvements are required to enhance operational safety, efficiency, and mechanization. In particular, it is necessary to introduce temporary support systems, mechanize the drilling and floor-leveling processes, and improve the handling of hanging rock. The proposed modifications are as follows.

First, it is essential to supplement the calculation of rock pressure in the cutting chamber, including the pressure on the roof, floor, and working face. These calculations should be based on a cutting chamber height of less than 2 m and a width of less than 3.5 m, similar to the methodology used for tunnel pressure assessment. The results will be the basis for selecting an appropriate containment and support solution.

Second, a temporary support system must be incorporated into the continuous blasting shrinkage stoping process. During drilling, creating separation zones in the fractured and non-cohesive rock mass causes ore fragments to detach and fall freely toward the drilling area, posing significant safety risks. Therefore, it is necessary to design and install temporary reinforcement measures suitable for such conditions.

Third, it is necessary to mechanize the leveling of the cutting chamber floor after each ore extraction and discharge cycle. The current manual leveling approach proposed in the design is inefficient and unsafe, as it requires significant labor and time while reducing productivity. To address this, the authors recommend using self-propelled drilling rigs and compact excavators equipped with bucket attachments, long chisels, or scraper blades to level the chamber floor and assist in removing hanging rock. Given that the geological conditions at the Vi Kem Mine differ from those at other sites, the traditional method of poking hanging rock with a drill bit has proven ineffective.

Finally, it is proposed to design a mechanized ore discharge system employing a hydraulic mechanism or an equivalent automated solution to replace the manual ore discharge method currently in use. This improvement will prevent frequent ore blockages during unloading operations, reduce steel consumption for hopper structures, lower manual labor intensity, and significantly enhance safety during ore handling.

2.5.2. Ore sublevel caving mining technological diagram

For the ore sublevel caving mining technological diagram, several key parameters are proposed to be adjusted and refined to improve safety, adaptability, and efficiency under the Vi Kem Copper Mine's specific geological and technical conditions.

First, it is recommended that the stratification height be recalculated and optimized, selecting a value smaller than that adopted in the current design. Reducing the stratification height will improve blasting control, ensure more uniform ore fragmentation, and enhance the stability of the remaining rock mass. To implement this, a compact and flexible self-propelled drilling rig with adjustable drilling angles is proposed, allowing greater precision and adaptability to the variable geometry of the ore body.

Second, it is necessary to replace the manual ore discharge method, which is unsuitable for the Vi Kem Copper Mine conditions. In the sublevel caving method, the blasted ore fragments are significantly larger than those obtained through shrinkage stoping, and manual handling of such material is unsafe and inefficient. Furthermore, after blasting, personnel are prohibited from entering the blasted voids to deal with oversized ore due to safety concerns. Therefore, a mechanized ore discharge system should be introduced, capable of handling large rock fragments and ensuring continuous, safe ore flow from the drawpoints without manual intervention.

3. Results and discussion

3.1. Justification and selection of design area

Based on the Vi Kem Copper Mine's current opening-up and preparation diagram, the selected design area includes ore blocks 13-122-1.1, 14-122-1.1, and 2-121-1.1 of the TQ1.1 ore body. These ore blocks are soon scheduled for extraction and have been accessed through the main haulage crosscut at level +30, extending from the shaft station to the TQ1.1 ore body. The corresponding ventilation system for this area is established through a ventilation crosscut at level +150 and a ventilation incline connecting levels +110 to +150.

The selected reserve blocks are distributed within the elevation range from –50 to +110, between the exploration lines T.47 and T.50-1. Therefore, the choice of these blocks for design calculation and trial implementation is entirely consistent with the current mine opening-up layout and development plan.

All selected ore blocks belong to steeply dipping ore bodies, with an average dip angle of about 75° and ore thickness ranging from 2.98 to 3.29 m, averaging 3.1 m. The main geological and structural characteristics of these ore blocks are presented in Table 3. In contrast, the longitudinal cross-section of the reserve block in the selected design area is shown in Figure 2.

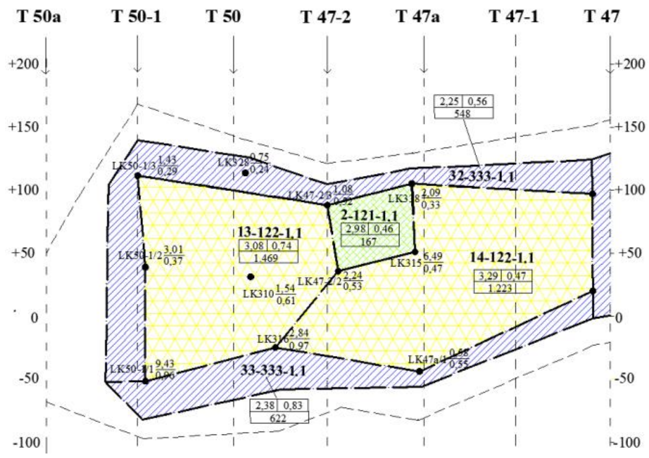


Figure 2. Longitudinal cross-section of the reserve block in the selected design area

The rock layer above the ore (roof rock). The rock block has a relatively stable geological structure, including Gneisbiotite, Granitogneis, 2-mica quartz schist, and other rocks interspersed. The layers are from 0.50m to 30m thick.

Table 3. Characteristics of the selected ore blocks for design area

№	Ore body name	Ore block number	Ore block area, m^2	Reserve thickness, m	Volume of the block, m^3	Dry ore reserves, t	Copper content, %	Metal reserves, t
1	TQ1.1	2-121-1.1	3.691	2.98	10.981	36.348	0.46	167
2	TQ1.1	13-122-1.1	19.373	3.08	59.697	197.599	0.74	1.469
3	TQ1.1	14-122-1.1	24.109	3.29	79.387	262.770	0.47	1.223
Total			47.173		150.066	496.717		2.859

The rock blocks are affected by weak weathering and are cracked and broken in some places. The cracked and broken places have a generally poor geological structure. The rock in this layer contains local pressure water in cracks and holes, but it is poor and has a negligible impact on the passing construction.

The ore-bearing rock layer includes metamorphic rocks and micmatized Gneisbiotite. The rock is compressed, has minor cracking, has a granular structure, scales and a strip-like structure, and is solid in the block, but has some places with strong cracking and breaking.

The rock layer under the ore (floor rock) includes Gneisbiotite, Granitogneis, Amphibol, calcified limestone, and others. The rocks in this layer are usually solid, with few cracks, and have a durable geological structure. The average results of the analytical indicators are shown in Table 1.

The design selection area is currently prepared with a haulage level at +70 level and a ventilation level at +110 level. Both of these tunnels are dug along the ore body. The connection between the haulage level of +70 and the main haulage level of +30 is by a central incline from +30 to +70. The connection between the +110 and the ventilation level +150 is by a ventilation incline from the +110 to the +150 levels. Thus, the boundary division diagram of the mining layer has been formed, which is very suitable for the mining preparation work. The vertical height of the mining layer is 40m. The length in the direction of a strike line of the design ore block area is 350 m. The haulage and ventilation levels are dug with a dome-shaped cross-section, supported by anchors or without support. The height of the tunnel is 2.5 m, the width is 3.5 m, and its area is 7.5 m2. The current status of the haulage and ventilation levels of the design area is shown in Figure 3.

Figure 3. Current status of the haulage and ventilation levels in the design area

The preparation of the mining room at these ore blocks has not been carried out (the room incline has not been dug yet). Therefore, the preparation according to the proposed system diagram and mining technology is favorable and is not affected by the old mining technology diagram (the old mining technology diagram applied to this condition is the mining system of the ore shrinkage stoping, the ore extraction by drilling and blasting, ore recovery through the discharge door onto the trolley at the haulage level of +70 level). The longitudinal cross-section of the current status diagram of the preparation tunnels and the plan of the preparation tunnels of the design area are shown in Figures 4 and 5.

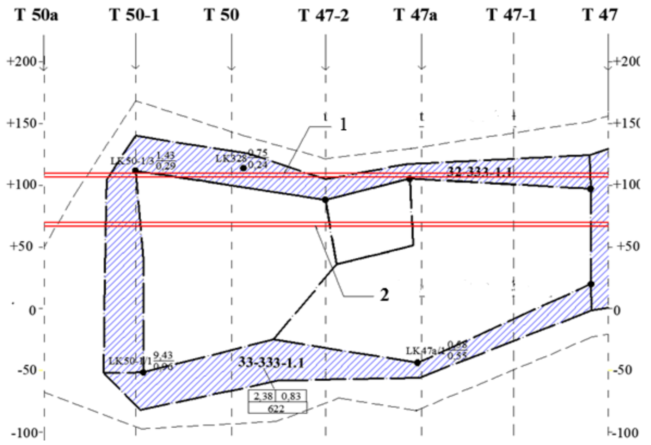


Figure 4. Longitudinal cross-section of the current status diagram of the design area: 1 – ventilation level at +110; 2 – haulage level at +70

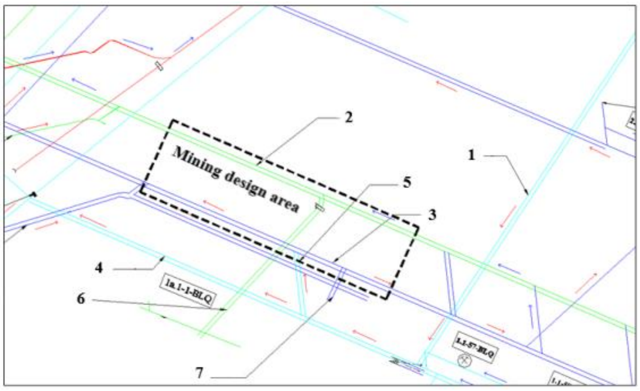


Figure 5. Plan of the preparation tunnels in the design area: 1 – main haulage crosscut at +30 level to the TQ1.1 ore body; 2 – ventilation level at +110 of the TQ1.1 ore body; 3 – haulage level at +70 of the TQ1.1 ore body; 4 – haulage level at +30 of the TQ1.1 ore body; 5 – central haulage brake incline; 6 – crosscut at +110 level from the TQ1.1 to the TQ1a.1 ore body; 7 – crosscut at +70 level from the TQ1.1 to the TQ1a.1 ore body

3.2. Selecting a mining technology diagram suitable for the conditions of the design area

As analyzed above, the system diagram and sublevel caving are selected in the design area to improve the mining technology diagram.

According to the distribution characteristics of the ore blocks in the design area, the length in the direction of a strike line is 350 m. To ensure the complete ore exploitation and not need to leave pillars to protect the tunnels, at level +110, a ventilation level is dug in the floor rock. At the same time, it is parallel and 10m away from the longitudinal level dug in the ore body. At level +70, a haulage level is also dug in the floor rock, about 10 m away from the ore body. The connection between the longitudinal level of the rock and the longitudinal level of the ore body is made by a cross-measure drift. In the center of the actual design area, an inclined tunnel is dug from level +70 to +110, connecting the longitudinal level of the rock at level +70 to the longitudinal level of the rock at level +110 to serve the transportation of equipment between the stratified longitudinal levels. The inclined tunnel is dug at a slope angle of 150 to facilitate equipment transportation. This tunnel is dug according to each stratified height. The height of each mining horizon is selected to correspond to the drilling equipment, expected to be 10-12 m.

To prepare the mining sublevel, from the +70/+110 inclined tunnel, the cross-measure drifts are dug into the ore body. The cross-measure drifts are dug at the corresponding high levels of the mining sublevel in the design area. When the cross-measure drifts meet the ore body, the longitudinal level of the mining sublevel is dug out towards the border of the design area. At the +70 longitudinal level of rock and the longitudinal level of the last mining sublevel, excavate the ore chute and cross-measure drifts from the longitudinal level of the ore body at level +70 to prepare for ore discharge during the mining process. The mining technology diagram is shown in Figure 6.

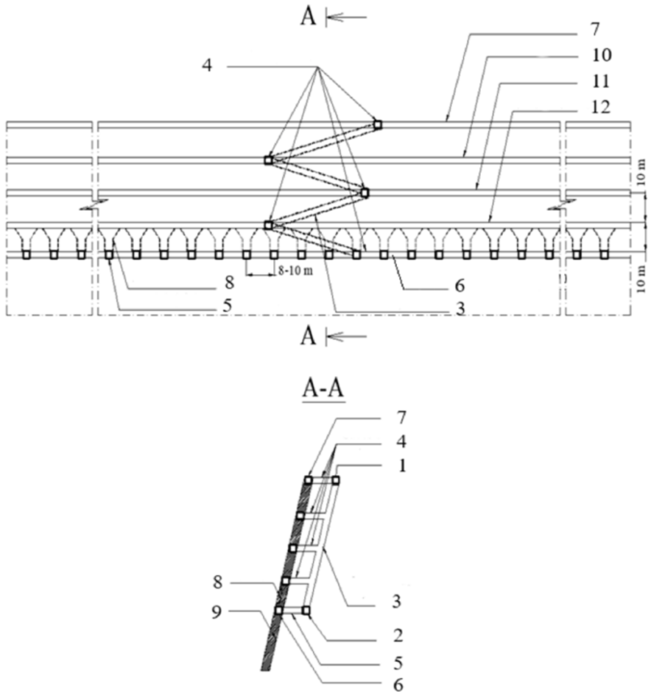


Figure 6. Diagram of the preparation of the mining room in the design area: 1 – ventilation level at +110 (in floor); 2 – haulage level at +70 (in floor); 3 – inclined tunnel connecting +70 to +110 levels; 4 – cross-measure drifts; 5 – cross-measure drift (ore discharge); 6 – haulage level at +70 (in ore body); 7 – ventilation level at +110 (in ore body); 8 – ore chute; 9 – ore body; 10 – stratified longitudinal level at +100; 11 – stratified longitudinal level at +90; 12 – stratified longitudinal level at +80

3.3. Calculation of the main parameters of the selected mining technology diagram

According to the adopted mining technology diagram, the ore body's geological characteristics and the mining area's corresponding technical parameters are determined as follows. The average ore body thickness is 3.12 m, and the average length of a cutting room is 10.4 m. The height of the mining stratification is set at 10 m, while the ore hardness ranges from f = 10 to 16, indicating a complex to tough rock type. The fundamental parameters of the ore drilling and blasting passport developed for the selected design area are presented in Table 4.

The preparation tunnels in the design area include inclined tunnels for transporting equipment from level +70 to +110, and longitudinal levels in the rock at level +70 and +110. The tunnels are stratified into longitudinal levels, cross-measure drifts, and the ore chute. In practice, the rock and ore in the mine area are excellent and stable.

After excavation, the tunnels do not need to be supported. In particular, only a few tunnel sections excavated through faults and weak geology need to be reinforced with steel or wood.

Table 4. Basic parameters of the ore drilling and blasting passport

№	Names of basic parameters	Unit	Value
1	Mining room width	m	3
2	Mining room height	m	10
3	Length of one blasting in the cutting room	m	10.4
4	Unit explosive index	kg/m^3	1.61
5	Distance between rows of drill holes	m	1.9
6	Number of rows of holes drilled per blasting	row	04
7	Number of holes drilled in the cutting face	holes	6
8	Number of holes drilled in a blasting section	holes	24
9	Amount of explosives per explosion	kg	502

The results of calculating the number of holes drilled on the prepared tunnel face are shown in Table 5. The drilling and blasting passport are shown in Figure 7, and some economic and technical indicators of the prepared tunneling technology are shown in Table 6.

Table 5. Number of holes drilled on the prepared tunnel face

№	Names of basic parameters	Symbol	Unit	Value
1	Average excavation cross-sectional area	S_d	m^2	6.25-8.4
2	Rock strength coefficient	f	-	8-10
3	Unit explosive index	q	kg/m^3	2.35
4	Tunnel width	B	mm	2.640
5	The coefficient depends on the shape of the tunnel	C	-	3.86
6	Distance between border boreholes	r_b	mm	45
7	Explosive cost per meter boreholes	γ_0	kg/m	0.20
8	Explosive cost per meter of boreholes	γ	kg/m	0.45
9	Explosive density	Δ	kg/m^3	1100
10	Volume of explosives in 1m boreholes	V	m^3	0.0008
11	Explosive charge coefficient	a	-	0.60
12	Explosive compaction coefficient in the boreholes	b	-	1.00
13	Diameter of explosive stick	d	m	0.032
14	Total number of calculated drill holes	N^{n}_{g}	holes	37
-	Number of holes drilled at the side of the tunnel	N_b	holes	17
-	Number of holes drilled, cut, and the floor of the tunnel	$N_{r,f}$	holes	20
15	Total number of drilled holes arranged on the tunnel face	N_g	holes	38
-	Number of holes drilled at the side of the tunnel	N_b	holes	14
-	Number of empty boreholes	N_{tr}	holes	1
-	Number of drilled cut holes	N_r	holes	6
-	The number of holes drilled to break the floor and create water channels	$N_{f,n}$	holes	17

Table 6. Summary of technical and economic indicators for tunnel preparation

№	Names of basic parameters	Unit	Value
1	Cross-sectional area of tunnel excavation	m^2	6.25-8.4
2	Usable cross-sectional area of the tunnel	m^2	5.28
3	Perimeter of tunnel	m	6.1
4	Tunnelling step	m/cycle	1.53
5	Number of shifts per day and night	shift	3
6	Number of shifts completing a cycle	shift	1.5
7	Cycle completion coefficient	-	0.85
8	Tunnel digging speed per day and night	m/d-n	2.6
9	Tunnel-digging speed per month	m/mon	68
10	Number of people working day and night	worker	15
11	Direct labour productivity	m/w-sh	0.17

The design calculations show that 38 blast holes are arranged on the tunnel face, including 14 side, 6 cut, 17 floor, and 1 empty hole. The tunnel's cross-sectional area ranges from 6.25 to 8.4 m^2, with a rock strength coefficient of 8-10 and a unit explosive index of 2.35 kg/m^3. The explosive density is 1100 kg/m³, and the average explosive consumption per meter of borehole is 0.45 kg/m.

The usable tunnel area reaches 5.28 m^2, and the excavation perimeter is 6.1 m. Each drilling-and-blasting cycle advances the face by 1.53 m, taking about 1.5 shifts, with a completion coefficient of 0.85. With three shifts per day, the tunneling rate is 2.6 m/day or 68 m/month, achieved by 15 workers with a productivity of 0.17 m per worker-shift.

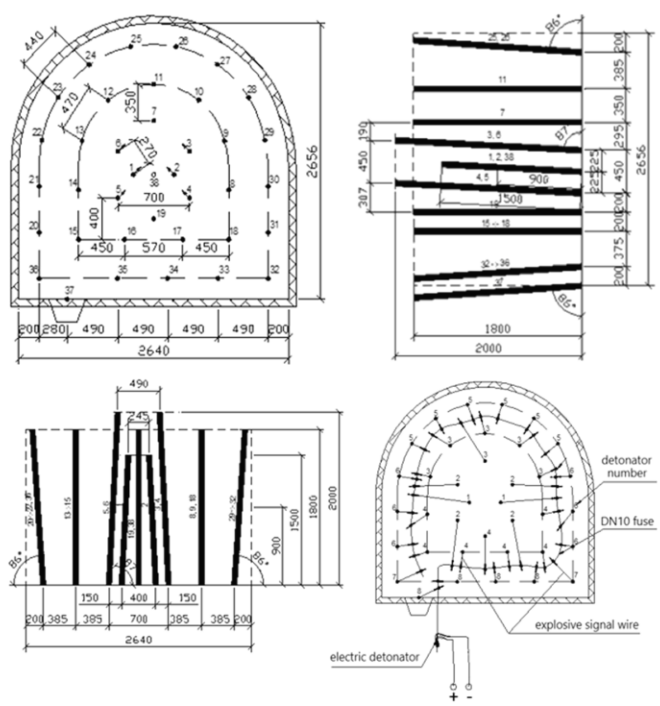


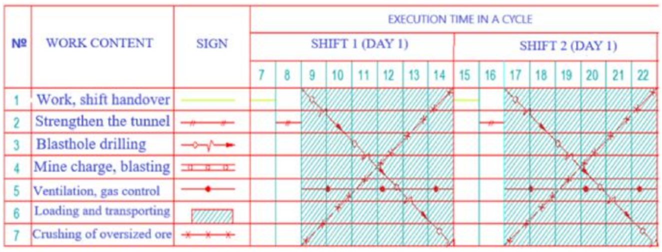
Figure 7. Diagram of the arrangement of drilled holes on the face and diagram of the detonator cap connection

The diagram illustrates the layout of the blast holes on the tunnel face and the corresponding connection scheme of detonator caps. The figure shows the numbering of boreholes, spacing parameters, and sequence of detonation, ensuring controlled rock fragmentation and uniform advance of the mining face.

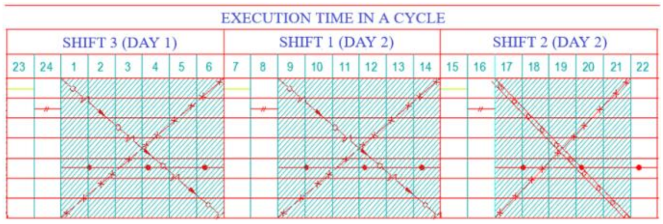
3.4. Production organization in the ore mining technology diagram

The production process in the selected ore mining technology diagram is organized in three working shifts daily, each lasting eight hours. The primary operations performed within one complete production cycle include: inspection and reinforcement of the stratified longitudinal level; breaking of oversized ore fragments; drilling of blast holes; loading and blasting operations followed by ventilation; loading of ore onto haulage trucks; ore transportation; and rock processing to create additional mining space.

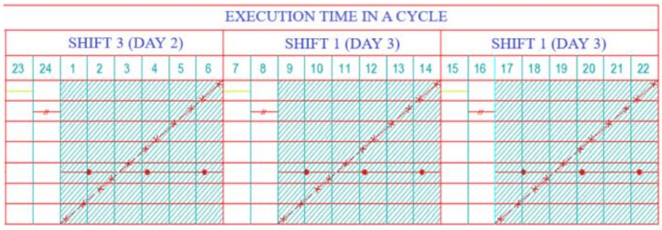
The production cycle is organized over eight working shifts based on the total workload and operational sequence. The design of the production cycle organization and the corresponding labor and equipment arrangement are illustrated in Figures 8 and 9.



(a)

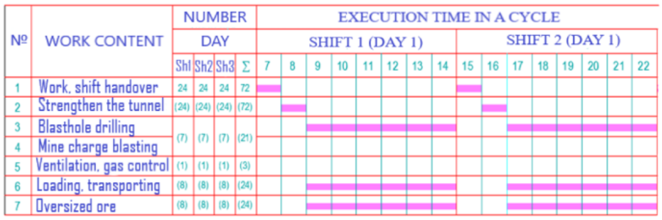


(b)

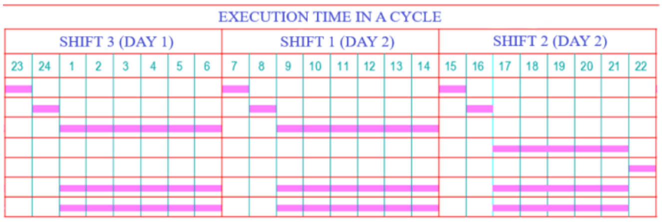


(c)

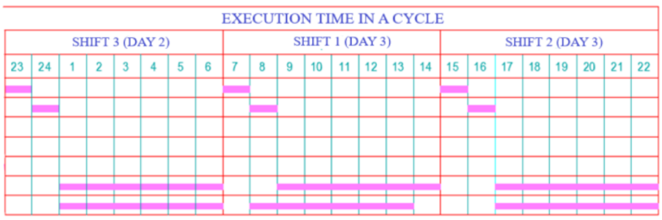
Figure 8. Production cycle organization chart in the mining face: (a) – tasks performed during shifts 1 and 2 of day 1; (b) – tasks performed during shift 3 of day 1 and shifts 1 and 2 of day 2; (c) – tasks performed during shift 3 of day 2 and shifts 1 and 2 of day 3



(a)



(b)



(c)

Figure 9. Human resource arrangement chart in the mining face: (a) – number of personnel assigned during shifts 1 and 2 of day 1; (b) – number of personnel assigned during shift 3 of day 1 and shifts 1 and 2 of day 2; (c) – number of personnel assigned during shift 3 of day 2 and shifts 1 and 2 of day 3

3.5. Calculation of economic and technical indicators of the mining technology diagram

3.5.1. Ore output in a blasting cycle

The ore output in a single blasting cycle is determined using the following formula:

$$Q_c = B \cdot r \cdot h \cdot \gamma_t \cdot k_{rec} \cdot k_{conv}, \quad (1)$$

where B is width of the mining room (3.0 m); r is depth of one blasting round (10.4 m); h is height of the mining room (10 m); γ_t is volumetric (bulk) weight of ore (3.3 t/m^3); k_{rec} is ore recovery coefficient, reflecting extraction efficiency (0.9); k_{conv} is conversion coefficient accounting for losses and dilution; since the total loss and dilution ratio is 5% (1.0).

Substituting these values into Equation (1), the ore output per blasting cycle is 926.6 tons.

The daily ore output is determined as:

$$Q_d = \frac{Q_{ck}}{n_{cyc}} \cdot n_{shift} \cdot k_{cyc}, \quad (2)$$

where: n_{cyc} is the number of shifts required to complete one full cycle (8 shifts); nshift is the number of shifts operated per day (3 shifts); kcyc is the cycle completion coefficient, representing the effective completion rate (0.8).

Substituting the values into Equation (2), the ore output per day and night equals 278 tons.

Monthly ore production is calculated by:

$$Q_m = Q_d \cdot n_{work}, \quad (3)$$

where: n_{work} is the number of working days per month (25 days).

Substituting the values into Equation (3), the monthly ore output is 6.95 thousand tons.

The annual production capacity of the design area is calculated using:

$$Q_y = Q_m \cdot n_{month} \cdot k_{tr}, \quad (4)$$

where: n_{month} is the number of working months in a year (12 months); ktr is the coefficient accounting for time spent on transition between stopes or mining areas (0.95).

Substituting the values into Equation (4), the annual ore output of the design area is approximately 79.2 thousand tons per year. The economic and technical indicators of the applied mining technology diagram in the selected design area are summarized in Table 7.

Table 7. Leading economic and technical indicators

№	Name of indicators	Unit	Value
1	Width of mining room	m	3
2	Height of the mining room	m	10
3	Volumetric weight of ore	ton/m^3	3.3
4	Depth of one blasting of a cycle	m/ck	10.4
5	Number of shifts operating a day and night	shift	3
6	Number of shifts completing a cycle	shift	8
7	Working time during the year	day	300
8	Time utilization coefficient in the year	-	0.85
9	Ore recovery coefficient in the mining room	-	0.95
10	Ore output in a blasting cycle	ton	926.6
11	Ore mining coefficient	-	0.90
12	The largest annual ore output	ton/year	79.230
13	Number of workers directly employed in the mining area	worker	72
14	Direct labor productivity	ton/worker	12.8
15	Loss and impoverishment rates due to drilling and blasting technology	%	5.0

3.6. Discussion

This research proposes improvements to the mining technological diagram to enhance the efficiency of ore extraction at the Vi Kem Copper Mine.

For the existing ore mining technology diagram, the study suggests optimizing the ore recovery system by using ore chutes and branch tunnels, and introducing wheel loaders to improve loading efficiency, labor safety, and the handling of oversized ore. This approach eliminates the current manual ore discharge method through small discharge doors, thereby addressing safety risks and preventing blockages caused by large rock fragments. As a result, it reduces the volume of oversized ore that must be crushed in the stope and shortens the preparation time for subsequent cutting cycles.

The study proposes adopting an ore sublevel caving mining technological diagram for medium-thick ore bodies and unprepared tunnel sections, where applying the shrinkage stoping method is difficult. In this system, the workers' positions are located safely within the stratified longitudinal level, enabling the mechanization of drilling operations, facilitating the movement of equipment between sublevels, improving ventilation efficiency, and increasing ore discharge capacity while reducing the overall volume of inclined tunnels.

For thick ore bodies, it is recommended to apply the layered sublevel caving method for lens-shaped ore blocks, with ore transportation directly along stratified longitudinal levels. For ore bodies of significant strike length, the sublevel caving diagram should be further optimized by incorporating inclined tunnels for equipment transfer, minimizing the excavation of inclined tunnels, and improving ore recovery in the cutting rooms through the use of ore chutes and mechanized excavator loading. Compact jumbos for drilling blast holes from upper to lower sublevels ensure high operational flexibility and safety.

The proposed improvements are technically and economically feasible. They enable more mechanization across mining stages, increasing productivity, mine efficiency, ventilation capacity, and labor safety. Considering the current operational conditions at the Vi Kem Copper Mine, the modernization of the mining technological system is both timely and necessary. The improved technological diagram remains consistent with the fundamental layout of the existing system but optimizes each component to maximize mechanization and operational efficiency.

The analysis confirms the high feasibility of the proposed technological improvements. Calculations show that the mining capacity per cutting room can be increased by 1.76 times, while labor productivity can be enhanced nearly threefold compared with the current system. The new approach eliminates the need for ore discharge through steep ore chutes, which often causes production delays and safety hazards. Drilling is performed by self-propelled rigs in a top-down direction, ensuring operational convenience and safety. Since the working face is confined within the stratified longitudinal level, temporary supports, manual ore crushing, and manual leveling of the stope floor are no longer required. During mining, the ore naturally accumulates in the shrinkage chamber. It is gradually discharged through an improved ore chute system, without requiring manual discharge doors, and loaded by an excavator onto transport vehicles, resolving the significant challenges in the ore handling and transportation stages.

4. Conclusions

The analysis of the current mining system at the Vi Kem Copper Mine showed that the existing technological diagrams, mainly ore shrinkage stoping and ore sublevel caving, require modernization to increase efficiency, safety, and mechanization under the mine's specific geological and technical conditions.

The proposed improvement of the ore shrinkage stoping technological diagram includes the introduction of mechanized drilling and floor-leveling systems, temporary support solutions, and hydraulic ore discharge mechanisms. These modifications enhance operational safety, reduce manual labor intensity, and minimize ore blockages and oversized rock formations. A sublevel caving mining diagram is proposed for medium-thick and unprepared ore bodies. This system ensures safe working conditions for personnel, allows for mechanized drilling and flexible equipment movement between sublevels, improves ventilation, and reduces the excavation volume of inclined tunnels.

A layered sublevel caving scheme is recommended for thick and lens-shaped ore bodies, integrating inclined tunnels for equipment transfer and direct ore transportation along longitudinal levels. This approach improves ore recovery, enables the use of small jumbos for drilling from upper to lower sublevels, and further increases mechanization.

Technical calculations confirm the feasibility of the improved technological diagrams. The mining capacity per cutting room increases by 1.76 times, and labor productivity rises nearly threefold compared with the current system. Additionally, the proposed solutions enhance ventilation, reduce preparation time, and significantly improve labor safety. Implementing the improved mining technological diagrams at the Vi Kem Copper Mine is technically feasible, economically efficient, and timely. The proposed system maintains compatibility with the existing infrastructure while optimizing key operational elements to achieve higher productivity, safety, and sustainability of underground copper ore extraction.

Author contributions

Conceptualization: TTV, PQL; Data curation: DTL, DTTV; Formal analysis: DTL, PQL; Investigation: DTTV, TTV; Methodology: TTV; Project administration: TTV, PQL; Resources: DTL, DTTV; Supervision: TTV; Validation: PQL, DTL; Visualization: DTTV; Writing – original draft: TTV; Writing – review & editing: TTV, PQL. All authors have read and agreed to the published version of the manuscript.

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Conflicts of interest

The authors declare no conflict of interest.

Data availability statement

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

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Вьетнам, Лаокай провинциясы, Ви Кем мыс кенішінде кен денелерін игерудің технологиялық схемасын жетілдіру бойынша шешімдер

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Аңдатпа. Кен денелерінің пайда болу жағдайларына байланысты технологиялық даму схемасын жобалау және таңдау қиын міндет болып табылады, өйткені ол көптеген факторларға байланысты. Мұндай схемалар өндіріс процесінің тиімділігі мен қауіпсіздігін қамтамасыз етуі керек. Сондықтан Vinacomin-Minerals Holding Corporation корпорациясының жерасты кеніштерінің заманауи өндірістік талаптарын қанағаттандыру үшін кен денелерін өндірудің технологиялық схемасын жетілдіруге бағытталған шешімдерді ұсыну қажет. Тау-кен жұмыстарының қолданыстағы жоспары мен мыс кенішінің жобалық технологиялық схемасы негізінде авторлар зерттеу, талдау және бағалау жүргізді, олардың нәтижелері бойынша кен денелерін игерудің жетілдірілген схемасын ұсынды және жобалық есептеулер жүргізу үшін кеніштің учаскесін таңдады. Жұмыста деректерді жинау, талдау және синхрондау әдістері, далалық зерттеулер және эксперименттік учаскенің теориялық есептеулері қолданылады. Ұсынылған шешімдердің іске асырылу деңгейі жоғары: технологиялық операцияларды механикаландыру, өндірістің өнімділігі мен тиімділігін арттыру, желдетуді жақсарту және еңбек қауіпсіздігін арттыру мүмкіндігі қамтамасыз етіледі. Жетілдірілген технология тұтастай алғанда бұрынғы схеманың құрылымын сақтайды, бірақ әрбір элемент механикаландыруды қолдануды кеңейту үшін оңтайландырылған, бұл өндірістік тиімділік пен қауіпсіздіктің өсуін қамтамасыз етеді. Техникалық есептеулер көрсеткендей, әр камераның өндірістік қуаты 1.76 есе, ал жұмыс өнімділігі қолданыстағы даму схемасымен салыстырғанда үш есе артады.

Негізгі сөздер: *тау-кен технологиялық схемасы, жетілдіру, кен өндіру, кен денесі, Ви Кем, Вьетнам.*

Решения по совершенствованию технологической схемы разработки рудных тел на медном руднике Ви Кем, провинция Лаокай, Вьетнам

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Аннотация. Проектирование и выбор технологической схемы разработки в зависимости от условий залегания рудных тел представляет собой сложную задачу, поскольку она зависит от множества факторов. Такие схемы должны обеспечивать эффективность и безопасность производственного процесса. Поэтому необходимо предложить решения, направленные на совершенствование технологической схемы добычи рудных тел, чтобы удовлетворить современные производственные требования подземных рудников корпорации Vinacomin – Minerals Holding Corporation. На основе действующего плана горных работ и проектной технологической схемы медного рудника Ви Кем авторы провели обследование, анализ и оценку, по результатам которых предложили усовершенствованную схему разработки рудных тел и выбрали участок рудника для проведения проектных расчётов. В работе применены методы сбора, анализа и синтеза данных, полевые обследования и теоретические расчёты экспериментального участка. Предложенные решения обладают высокой степенью реализуемости: обеспечивается возможность механизации технологических операций, повышение производительности и эффективности добычи, улучшение вентиляции и повышение безопасности труда. Усовершенствованная технология в целом сохраняет структуру прежней схемы, однако каждый элемент оптимизирован для расширения применения механизации, что обеспечивает рост производственной эффективности и безопасности. Технические расчёты показывают, что производственная мощность каждой камеры увеличивается в 1.76 раза, а производительность труда – почти в три раза по сравнению с действующей схемой разработки.

Ключевые слова: *горнотехнологическая схема, совершенствование, добыча руды, рудное тело, Ви Кем, Вьетнам.*