Development of Technologies for Mining Ores with Instable Hanging Wall Rocks

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Abstract

Underground mines of Kryvyi Rih iron ore deposit apply room mining systems or systems with bulk caving of ore and overlying rocks in a ratio of 35% to 65%. Most mines prefer room mining systems with pillar caving due to high, technical and economic indicators. However, when mining certain areas, the problem arises of hanging wall rocks stability. Under the same mining and geological conditions of the deposit, stopes are stable in some areas, but in others waste rocks get in the stope from the side of the hanging wall when a slight exposure is created. Thus, in conditions of instable rocks of the hanging wall, development and improvement of the technology involving room mining is an urgent issue. Analysis of researchers reveals factors that significantly indluence stability of the hanging wall rocks and ore. The developed methods enable determining stability parameters and applying an improved option of room mining system in conditions of the instable hanging wall with the help of a protective ore pillar located at the instable hanging wall. Calculations performed demonstrate that application of the proposed mining system enables an increase in the iron content in the mined ore mass by 0.94%, the increased amount of the ore mass extracted and a profit of 18.73 thousand euros for the whole of a block.

Keywords: *mining system, stress, methods, pillar width, rock strength, losses, dilution*

1. INTRODUCTION

Kryvyi Rih iron ore deposit (Ukraine) is represented by narrow strips of metamorphic rocks stretching for about 100 km from south to north. In terms of stratigraphy, Kryvyi Rih series of rocks is divided into four formations: schist-amphibolite (K_0), lower arkose-phyllite (K_1), middle iron ore (K_2) and upper schist (K_3) ones [1–5]. The main productive thickness is the iron ore formation (K_2) which consists of seven pairs (ferruginous and schistous horizon) of rocks alternating with each other. The main ferruginous layers of the iron ore formation (K_2) mined by the underground method are: the fourth (PR_1Sx^{4f}), the fifth (PR_1Sx^{5f}) and the sixth (PR_1Sx^{6f}). Within these layers, deposits of rich ores are located, which are represented by a wide variety of shapes (seam-, lens-, pillar- and nest-like) and their combinations, [6–8].

Almost all rich iron ore deposits are composed of hematite and differ in physical and mechanical properties and the ratio of mineral varieties. Martite ores are characteristic of the northern group of mines. The ore is 2–3 points stronger than in the southern district. Stability of rocks depends on their physical and mechanical properties, the most important of which are: strength, mineral composition, fissuring, porosity, etc. Physical and mechanical properties and elements of occurrence can change dramatically within one mine block [9–11].

In Kryvyi Rih iron ore basin, iron ore is mined by room mining systems and systems with bulk caving of ore and country rocks [12–14].

The system with bulk ore caving is the most widely applied in mining the deposits. However, this system is characterized by significant ore dilution. [15–17]. In most cases, these mining systems are improved by optimizing structural elements of the block or by changing parameters of ore breaking and drawing [18–20]. Availability of a great number of options of the system with bulk caving enable their classification according to methods of breaking, transportation and location of the compensation room in the block [1, 8, 16]. It should be noted that the specific volume of subsidiary development systems with bulk caving is 4–7 m/1000 t, however, due to more intensive extraction, the cost of maintaining the workings is reduced by 15–20% as compared with room mining systems. However, this option of the mining system is characterized by the following disadvantages: the volume of pure ore mined does not exceed 35–55%; sharp fluctuations in the iron content in the mined ore mass lead to its sorting on the surface; significant ore losses on the footwall of the deposit; depression craters are formed on the surface [18–20].

When applying room mining systems, the iron content in the mined ore mass increases, but so do ore losses in pillars (up to 35–50%) and the volume of subsidiary workings (6–7 m/1000 t).

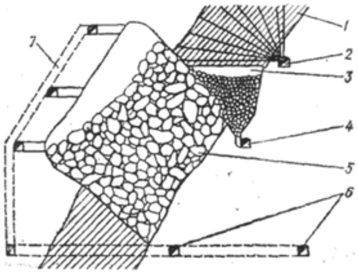

Fig. 1. Mining the deposit with the hanging wall rocks blasting: 1 – crown; 2 – drilling drift; 3 – undercutting space; 4 – collecting sublevel drift; 5 – caved hanging wall rocks filling the mined out room; 6 – workings of the transportation level; 7 – block raise from which special workings are created for vertical concentrated charges placement

Rys. 1. Eksploatacja złoża z odstrzałem skał wiszących: 1 – korona; 2 – sztolnia wiertnicza; 3 – przestrzeń podcięcia; 4 – zbieranie dryfu podpoziomowego; 5 – zawalone wiszące skały wypełniające wyrobisko; 6 – wyrobiska transportu poziomego; 7 – podwyższenie bloku, z którego tworzone są specjalne wyrobiska do pionowego umieszczania ładunków skoncentrowanych

ab

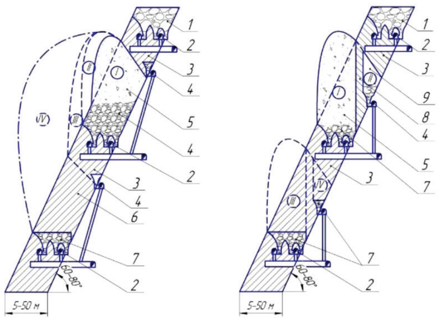

Fig. 2. Mining ore deposits in contact with clay rocks: a – with bulk blasting of the hanging wall; b – with local blasting of waste rocks of the hanging wall; I, II, III, IV – stages of mining; 1 – caved rocks; 2 – workings of the draw level; 3 – crown; 4 – auxiliary draw workings; 5 – sand-clay rocks; 6 – ore massif; 7 – caved ore; 8 – inter-room pillar; 9 – undercutting room for mining the "triangle" of the footwall and the crown

Rys. 2. Eksploatacja złóż rudy w kontakcie ze skałami ilastymi: a – z masowym wysadzaniem wiszącej ściany; b – z miejscowym wysadzeniem skał płonnych wiszącej ściany; I, II, III, IV – etapy wydobycia; 1 –skały zawału; 2 – wyrobiska wydobywcze; 3 – korona; 4 – wyrobiska pomocnicze; 5 – skały piaskowo-gliniaste; 6 – masyw rudy; 7 –urobiona ruda; 8 – filar międzykomorowy; 9 – pomieszczenie podrębowe do urabiania „trójkąta" spągu i korony

This mining system can only be applied in deposits represented by stable rocks of the hanging wall or with no less than medium ore stability and strength. While designing a block, optimal structural elements for this mining system are determined by the industry methodology developed by the Research Mining Institute (Kryvyi Rih, Ukraine).

According to practical data, mining of room reserves is 45–65%, which enables obtaining the maximum iron content in the mined ore mass. However, during mining of pillars, ore losses increase to 50%, and the iron content in the mined ore mass decreases by 3–15%.

Room mining systems have the following disadvantages: the two-stage extraction of ore, availability of a large volume of voids, significant ore losses and dilution during mining of pillars, high costs for maintaining mine workings, restricted conditions of application [12, 14].

It should be noted that in order to ensure stability, in practice, the volume of reserves in stopes with the instable hanging wall is reduced to 40–42%.

To increase efficiency of mining deposits with instable ores and rocks of the hanging wall, an option of combined mining is used, which involves simultaneous application of the room system and a system with bulk ore caving.

This option enables extraction of about 50–70% of pure ore from a block, but losses and maintenance costs for mine workings increase significantly [21–23].

2. ANALYSIS OF RESEARCHES AND PUBLICATIONS

The rock massif of Kryvyi Rih iron ore basin is not homogeneous. Application of room systems to mining rich ores causes, among others, a problem of providing conditions that exclude emergence of uncontrolled and randomly created areas of rock massif displacements.

To ensure stability of the rock massif, artificial structures are created, ore pillars are left unmined [12], or various methods of additional strengthening of instable local areas of the deposit are applied [24–26]. At the same time, a complex of mining operations is provided for that require significant labor and material costs, ultimately influencing economic efficiency and expediency of stoping.

One of possible options for stoping is application of the technology involving partial destruction of the hanging wall rocks to ensure replacement of the volume of ore mass with rocks, Fig. 1. However, in this case, extraction indicators deteriorate sharply, especially at deposit dips of 45–55 degrees, [27–29].

This is due to the fact that the ore mass is replaced with a relatively limited layer of rocks near the plane of the hanging wall, which results in a significant part of the caved ore left on the footwall. Creation of auxiliary sublevel transportation workings dramatically increases the costs for preparatory and subsidiary workings, complexity of their maintenance, especially in instable fissured rocks, which results in increased ore mining costs.

The monograph [30] describes the technology of creating a rock "pillow", proposed by B.I. Rymarchuk, as one of

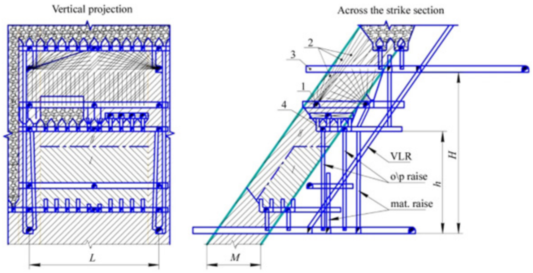

Fig. 3. System of sublevel bulk caving with transportation in drifts and a compacted layer of caved ore: 1 – drilling drift; 2 – long blasthole rings for creating an overcompacted ore layer; 3 – access ort of the upper level; 4 – scraper level

Rys. 3. System podpoziomowego zawału z transportem w sztolniach i zagęszczoną warstwą zawału rudy: 1 – sztolnia wiertnicza; 2 – długie pierścienie otworów strzałowych do tworzenia zagęszczonej warstwy rudy; 3 – wejście na górny poziom; 4 – poziom zgarniacza

the ways to combat clay rushes or uncontrolled destruction of rocks from the hanging wall side.

Fundamental flowsheets for mining the ore deposit, which allow reduction of the amount of additional mining of overlying rocks, are given in Fig. 2. These flowsheets differ in conditions of their application that depend primarily on the ore deposit dip, the ore deposit thickness, as well as location of instable rocks in the hanging wall.

The idea behind the technology presented in Fig. 2,a consists in the following: at the first stage, the ore reserve is mined out near the hanging wall I. During caved ore drawing 7, rocks of the hanging wall move simultaneously with the caved ore creating a caving zone.

After the upper sublevel is mined out, a crown 3 is drilled and blasted. With the help of the auxiliary draw working 4, the reserve of the crown is removed, while a displacement zone III is created. The disadvantage of this technology includes significant ore losses and dilution ranging from 16 to 25–45%.

The main feature of the flowsheet b, (see Fig. 2) is the local blasting of waste rocks of the hanging wall and their moving into the mined out stoping space. At the same time, between the caved rocks 5 and the "triangle" of the footwall 9, the ore massif (pillar) 8 is left.

Bulk ore caving and the use of a compacted layer, Fig. 3, is also a commonly used option. This technology enables reduction of caved ore losses and dilution in the instable area from the hanging wall side.

The idea behind this mining system is as follows. The mined block is divided into two technological areas; I – for bulk caving of the ore massif; II – for mining using a compacted ore layer.

On the contact of the caved ore with rocks, an additional drilling drift 1 is driven which is necessary for creating a compacted layer, by drilling the ore area of the hanging wall with long blasthole rings 2. To create a protective layer above the main reserve of the ore massif in the upper part of the block, previously driven workings of the above lying levels 3, 4 are used.

The ore massif of the main reserve is caved first. To do this, a drilled ore massif is broken on the previously created compensation room. Then, the long blasthole rings from the hanging wall and above the main block are exploded with delay. Consideration should be given to the fact that when breaking the massif onto a compressed medium, the optimal thickness of the broken ore layer is the main parameter.

According to V.R. Imenitov, displacement of broken ore (rock) after explosion of the first set of blasholes in the compressed medium is 2.0–2.5 m and reaches 3.0 m when blasting 4–5 sets of blastholes [31–33]. With average ore strength and thick deposits, compaction of the massif is 25–30 m.

To create a compacted layer, parameters of drilling and blasting (the line of least resistance and the distance between hole toes) must be reduced by 15–20% compared to the mass breaking of the main massif. Excessive amount of explosives in the ore massif in the peripheral zone of the block inevitably leads to an additional compaction of the broken ore mass by 3–5 m. The broken ore layer thickness in the hanging wall and in the upper part of the caved massif depends on the ratio of primary loosening of the main massif and its compaction factor [34–36].

However, it should be noted that the ore layer II from the side of the hanging wall is caved last. Therefore, at a considerable pressure of hanging wall rocks, when the main massif I is destructed, its additional destruction occurs. Ill-timed destruction of the ore layer from the hanging wall side can result in significant losses and dilution during ore drawing from the main massif.

3. PURPOSE

The presented study aims to determine the minimum permissible thickness of rocks of the hanging wall, which ensures stability of the rocks depending on rock pressure. For this, the following tasks should be solved:

1. To investigate the influence of the initial field of massif stresses on stability parameters of the rock massif.
2. To develop methods for determining the minimum permissible thickness of rocks of the hanging wall at which it is advisable to apply a system with bulk ore caving and a compacted layer in conditions of instable rocks of the hanging wall.

4. METHODS

To determine stability parameters of structural elements in Kryvyi Rih iron ore basin, the method of calculated functional characteristics is used, numerical values of which depend on basic factors associated with dimensions of structural elements and their stress-strain state [37]. Geometrical parameters are determined by using two types of functional characteristics: equivalent exposure spans; dimensionless characteristics for determining the width of the inter-room and the thickness of the inter-floor pillars.

Actual values between the equivalent spans of exposures in the room and their dimensions are described by the formulas:

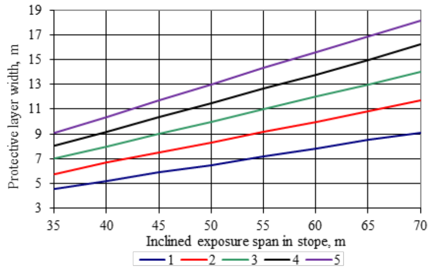

Fig. 4. Dependencies of the width of the protective layer on the exposure span in the stope and the ultimate compression strength of rocks: 1–5 – the ultimate strength of rocks of 140, 120, 100, 80 and 60 MPa respectively

Rys. 4. Zależności szerokości warstwy ochronnej od rozpiętości ekspozycji w przodku wybierkowym i ostateczna wytrzymałość skał na ściskanie: 1–5 – wytrzymałość skał odpowiednio 140, 120, 100, 80 i 60 MPa

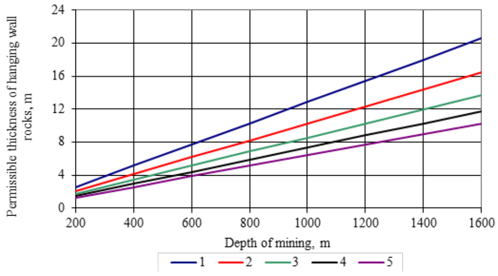

Fig. 5. Dependencies of the permissible thickness of the hanging wall rocks on the depth of mining operations and the ultimate strength of rocks at the stope width of 15 m and the deposit dip of 60 degrees: 1, 2, 3, 4 and 5 – the ultimate compression strength of 80, 100, 120, 140 and 160 MPa respectively

Rys.5. Zależności dopuszczalnej miąższości skał wiszących od głębokości eksploatacji i wytrzymałości skał przy szerokości przodka 15 m i upadzie złoża 60 stopni: 1, 2, 3, 4 i 5 – ściskanie graniczne odpowiednio 80, 100, 120, 140 i 160 MPa

- for horizontal exposures

$$m_h = \frac{aM_h}{\sqrt{a^2 + M_h^2}}, \text{ m;} \quad (1)$$

- for vertical exposures

$$m_v = \frac{bM_n}{\sqrt{b^2 + M_n^2}}, \text{ m;} \quad (2)$$

- for inclined exposures

$$l_n = \frac{ab}{\sqrt{a^2 + b^2}}, \text{ m,} \quad (3)$$

where a, b are the dimensions of rooms along the strike and downdip respectively; M_h, M_n are the horizontal and normal thickness of the deposit respectively.

The influence of the stress-strain state of the massif on parameters of rooms and pillars is taken into account in the classification of deposits according to the proposed methodology, [37].

As a rule, permissible spans of exposures are established on the basis of experiments conducted according to a special method [38–40]. The use of calculation methods is limited by certain conditions, e.g. heterogeneity of roof rocks. However, the results obtained should be considered as preliminary, requiring further experimental clarification. The most likely are the results obtained by the methods of calculating roofs represented by undisturbed or weakly layered rocks. In this system, the mined-out space is supported by ore pillars.

Based on the analysis results, it is established that the advantage of the methods developed by S.G. Borisenko, in comparison with the NDGRI (Research Mining Institute of Kryvyi Rih National University) methods, consists in considering the weight of not only overlying rocks but also those of the hanging wall [41–43].

5. RESULTS

To obtain high indicators of ore mass extraction during mining reserves by room mining systems, it is necessary to ensure stability of exposures and pillars within the entire period of mining the stoping block. Therefore, when mining ore deposits by room systems, structural elements of the stoping block should be refined and the thickness of rocks in the hanging wall should be taken into account.

When applying a room mining system, various stresses (tensile or compressive) act on pillars.

From the strength of materials theory, it is known that if a sample is evenly loaded over time, normal stresses increase in it to the ultimate compression strength. As soon as normal stresses exceed the ultimate compression strength, either linear strains occur or the room pillar is destructed. Thus, in order to preserve integrity of the rocks of the inter-room pillar, the following condition must be met.

$$\begin{cases} \sigma \le \sigma_{kr} \cong [\sigma_{st}], \\ \varepsilon = 0, \end{cases} \quad (4)$$

where σ is normal stresses, MPa; σkr is critical stresses, MPa; [σst] is the ultimate compression strength of rocks, MPa; ε is linear strains.

If the pillar is under action of compressive and tensile stresses throughout its lifetime, normal stresses in it first increase and then decrease. Repeated stresses cause linear strains in the pillar that significantly reduce the ultimate compressive strength of rocks.

If the pillar undergoes increasing stresses, normal stresses increase, according to (4); if stresses decrease, normal stresses do not reach the ultimate strength of rocks, the pillar is not destructed and the following condition is met

$$\begin{cases} \sigma \le \sigma_{kr} \cong \sigma_v << [\sigma_{st}], \\ \varepsilon \ne 0. \end{cases} \quad (5)$$

where σv is the destructive pressure, MPa.

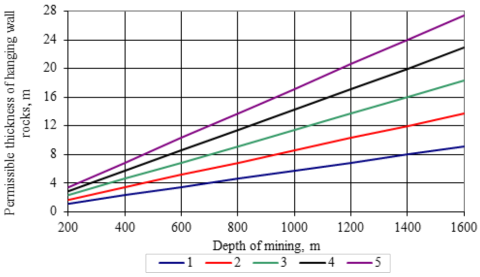

Fig. 6. Dependencies of the permissible thickness of the hanging wall rocks on the depth of mining operations and the stope width at the rock strength of 12 and the deposit dip of 60 degrees: 1, 2, 3, 4 and 5 – widths of the stope of 10, 15, 20, 25 and 30 m respectively

Rys. 6. Zależności dopuszczalnej miąższości skał wiszących od głębokości eksploatacji i szerokości przodka przy sile 12 i upadzie złoża 60 stopni: 1, 2, 3, 4 i 5 – szerokości przodka odpowiednio 10, 15, 20, 25 i 30 m

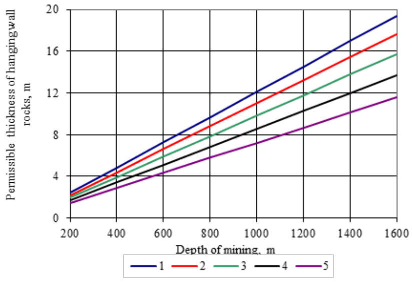

Fig. 7. Dependencies of the permissible thickness at the rock strength of 12 and the width of the stope of 15 m: 1, 2, 3, 4 and 5 – ore deposit dips of 45, 50, 55, 60 and 65 degrees respectively.

Rys. 7. Zależności dopuszczalnej miąższości przy wytrzymałości skały 12 i szerokości przodka 15 m: 1, 2, 3, 4 i 5 – upały złoża rudy odpowiednio 45, 50, 55, 60 i 65 stopni

Taking into account the above, parameters of structural elements at different stages of block mining can be determined for the first option (the stoping block is mined from the foot- to the hanging wall).

At the first stage of block mining, parameters of the stope (the width of the stope along the strike, the height and width of the pillar, the length of the inclined exposure span, the height of the level (sublevel) are determined by the NDGRI methods [17].

Then, in the conventional way, certain technological operations are performed in the stoping block: preparation, drilling and blasting, selection of the mode of caved ore mass drawing and transportation. It should be noted that after caved ore drawing from the first stope, inter-room ore pillars and the crown are not caved at this stage. Therefore, when determining exposure and pillars lifetime, it is necessary to take into account the total time needed for mining the stoping block (taking into account the second stage).

Maximum permissible thickness of the protective ore layer is obtained by the formula

$$m = \frac{la\gamma K_z}{4[\sigma_{st}]K_{str.o}}, \tag{6}$$

where l is the maximum permissible exposure span, m; a is the width of the stope, m; γ is the volumetric weight of ore, N/m³; Kz is the factor of safety (accepted 1.5–2.0); Kstr.o is the factor of rock structural weakening by fissures (accepted from 0.65 to 0.95).

Based on (6), the dependencies of the width of the compacted layer on the exposure span in the stope and the ultimate compressive strength of rocks are built (Fig. 4).

Fig. 4 demonstrates that an increase in the exposure span from 35 to 70 m requires the increase in the width of the protective layer from 9 to 18 m at the rock compression strength of 140 MPa.

If longitudinal compressive forces acting along the inclined exposure plane of the pillar are withstood without its integrity failure, lateral stresses and strains in the direction of previously mined out stopes, the thickness of the inter-room pillar is determined by the formula

$$m_\sigma = \frac{H\gamma aK_dK_z \cos\alpha}{n[\sigma_{st}]K_{str.o}}. \tag{7}$$

where H is the depth of mining, m; α is the ore deposit dip, degrees; n is the number of pillars per stope.

The dependencies of the permissible thickness of waste rocks from the hanging wall side when applying room mining systems on the depth of mining, the width of the stope and the ore deposit dip are shown in Fig. 5–7.

The above dependencies demonstrate that the permissible thickness of waste rocks from the hanging wall side increases directly proportional to the increase in the mining depth, the width of the stope and inversely proportional to the increase in the ultimate compressive strength of rocks and the ore deposit dip.

Fig. 5 demonstrates that with an increase in the depth of mining from 200 to 1600 m, the permissible thickness of the hanging wall rocks, which enables maintaining their stability for the period of mining out the stope, must be at least 1.3–20.5 m at the decreased ultimate compressive strength of rocks from 160 to 80 MPa and the width of the stope of 15 m.

At an increase in the width of the stope from 10 to 30 m and the depth of mining from 200 to 1600 m, the thickness of the hanging wall rocks at the rock ultimate compressive strength of 120 MPa and the deposit dip of 60 degrees must be at least 1.1–27.4 m, Fig. 6. Thus, at the increased mining depth

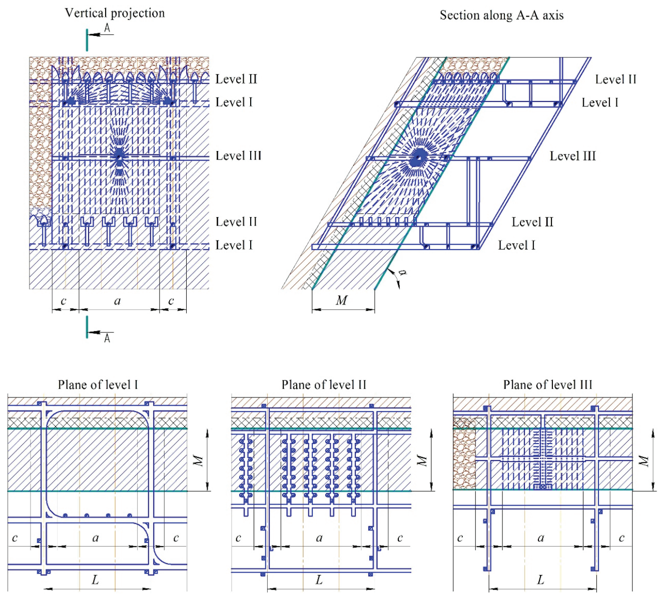

Fig. 8. The proposed option of the room system with a protective layer and subsequent pillar caving
Rys. 8. Proponowany wariant układu pomieszczeń z warstwą ochronną i późniejszym zawałem filarów

and the ultimate compression strength of 120 MPa, to ensure its stability, the width of the stope should be: at the depth of 700 m – ak = 30 m; at the depth of 850 m – ak = 25 m; at the depth of 1050 m – ak = 20 m; at the depth is 1400 m – ak = 15 and at the depth of 1600 m – ak = 10 m.

The dependencies shown in Fig. 7 demonstrate that with an increase in the deposit dip from 45 to 65 degrees, the permissible thickness of the hanging wall rocks must be at least 15–8 m at the depth of stoping works of 1200 m.

Thus, application of room mining systems is limited by the ultimate compressive strength of rocks, the thickness of the hanging wall rocks, as well as the height of the caved layer, the deposit dip and the rock stability factor.

Based on the results of the presented study, it is established that application of the room system to mining ore deposits is reasonable if the thickness of the hanging rocks exceeds 6 m.

According to the study conducted, an option of the room mining system is developed, Fig. 8.

The idea behind the proposed mining system consists in a certain order of mining operations depending on mining and geological conditions of the deposit.

At the first stage, a stope of the reduced by 10–15 m width is created across the strike leaving the ore thickness in the hanging wall. The parameters of the stope at the first stage are determined by the NDGRI methods [17].

The thickness of the protective layer from the hanging wall side is determined by the above described methods. Calculations by (6) and (7) for the conditions of the HVARDIISKA underground mine of the JSC KRYVBASZALIZRUDKOM enable obtaining the minimum width of the ore pillar based on the rock thickness in the hanging wall:

$$m = \frac{90 \times 50 \times 3.6 \times 2.0}{4 \times 100 \times 12 \times 0.65} \approx 11 \text{ m;}$$

$$m_{\sigma} = \frac{1260 \times 3.6 \times 50 \times 1.8 \times 2 \times \cos 60}{1000 \times 2 \times 14 \times 0.65} \approx 23 \text{ m.}$$

Thus, if the thickness of the rocks of the hanging wall does not exceed 23 m, an 11 m thick ore pillar is left. Therefore, dimensions of the stope are as follows: the length along the strike is 50 m, the width is M–m=35-11=24 m.

The mine block is drilled by circular long blasthole rings from the drilling drift to the full height of the level. The vertical compensation room is created in the middle part of the block, and it should be noted that the slot raise is located at the footwall.

After caving the stope reserve, the crown and the inclined pillar at the hanging wall are drilled. The crown is caved first, and then the pillar is caved on the compressed medium.

Caved pillars are drawn in direction from the hanging to the footwall. According to G.M. Malakhov theory of drawing, ore moves in a 8–12 m wide stream parallel to the angle of the hanging wall to the drawpoint. The inter-room pillar is mined out after the caved ore is drawn.

Advantages: a smaller volume of drilling workings; fewer redeployments of the drilling rig, as all holes in the ring are drilled from the same drilling level; additional fragmentation of ore due to collision of ore pieces caused by exploding opposite long blasthole rings; a significant amount of pure ore extracted.

Disadvantages: uneven fragmentation of the massif (excessive fragmentation of ore at the start of long blastholes and increased oversized yield at the toes); increased costs for drilling long holes; application in strong and stable ores and rocks of the hanging wall.

General technical and economic indicators are given in Tab. 1.

Analysi of the calculation results show that application of room mining systems with a protective layer on the side of the instable hanging wall enables reduction of ore losses from 16.3 to 12.4%, an increase in the iron content in the mined ore mass from 58.44 to 59.38% and an increase in profits from 105.27 to 111.54 euros/t.

6. CONCLUSIONS

As a result of the study, the following conclusions can be drawn:

1. Based on the study conducted, it is established that

Tab. 1. Technical and economic indicators

Tab. 1. Wskaźniki techniczne i ekonomiczne

No.	Indicators	u/m data	proposed option
1	Balance reserve of ore in block, kt	604.80	604.80
2	Ore reserve by elements, kt		
	- in stope	264.60	181.44
	- in inter-room pillars	151.20	151.20
	- in crown	113.40	113.40
	- in pillar of protective layer of hanging wall	0	106.92
	- in compensation room	75.60	51.84
3	Useful component content, %		
	- in massif	63	63
	- in rocks	38	38
4	Average useful component content in mined ore, %	58.44	59.38
5	Losses, %	16.30	12.40
6	Dilution, %	18.25	14.50
7	Ore mass extracted, kt	619.23	619.65
8	Mining cost, euro/t	150	170
9	Income, thousand euros	198.15	216.88
10	Profit, euro/t	105.27	111.54

the method for determining structural elements does not take into account the thickness of rocks that significantly influences stability.

2. The methods are proposed for determining the stability parameters of the structural elements of the mining system that enable the use of a room mining system.
3. The technology of the room mining system with the use of a protective layer of the ore massif from the side of the instable hanging wall is developed that ensures stability of the stope and an increase in the iron content in the mined ore mass by 0.94%, resulting in an estimated profit growth by 6.27 euros/t.

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Rozwój technologii wydobywania rudy z niestabilnymi wiszącymi skałami

Kopalnie podziemne złoża rudy żelaza w Krzywym Rogu stosują systemy urabiania komorowego lub systemy z zawałem rudy i nadległych skał w stosunku 35% do 65%. Większość kopalń preferuje systemy eksploracji komorowej z zawałem filarowym ze względu na wysokie wskaźniki techniczne i ekonomiczne. Jednak podczas eksploatacji niektórych obszarów pojawia się problem ze stabilnością wiszących skał. W takich samych warunkach górniczo-geologicznych złoża stopnie na niektórych obszarach są stabilne, ale na innych skały płonne dostają się do stopu od strony wiszącej ściany, gdy powstaje niewielkie odsłonięcie. Dlatego też w warunkach niestabilnych skał wiszącej ściany pilnym zagadnieniem jest rozwój i doskonalenie technologii eksploatacji komorowej. Analiza badań ujawnia czynniki, które znacząco wpływają na stabilność wiszących skał i rudy. Opracowane metody umożliwiają wyznaczenie parametrów statecznościowych oraz zastosowanie udoskonalonego wariantu systemu eksploracji pomieszczenia w warunkach niestabilnej ściany wiszącej za pomocą filaru ochronnego rudy, znajdującego się przy niestabilnej ścianie wiszącej. Z przeprowadzonych obliczeń wynika, że zastosowanie proponowanego systemu urabiania umożliwia zwiększenie zawartości żelaza w wydobywanej masie rudy o 0,94%, zwiększenie ilości wydobywanej masy rudy oraz zysk w wysokości 18,73 tys. euro za cały blok.

Słowa kluczowe: *system wydobywczy, naprężenia, metody, szerokość filaru, wytrzymałość skały, straty, rozcieńczenie*