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Study and evaluation of the stability of underground mining method used in shallow-dip vein deposits hosted in poor quality rock

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Abstract

Purpose. This article proposes to analyze and determine the mining design for shallow-dip deposits hosted in poor quality rock.

Methods. We used the UBC tool to find the optimal exploitation method, the Rock mass rating (RMR) and Q-system (Q) to determine the optimal mining stope and the recommended rock support, the numerical modeling by RS2 software with a variety of geotechnical, geometrical, and technical conditions to analyze the evolution of the unstable zone width and the maximum total displacement around the stope after excavation.

Findings. The optimum mining method designated by the UBC tool for this type of deposit is the cut and fill. By projecting the obtained RMR and Q-system values on the design graph, it is concluded that the operating stope is located in the stable zone with a height of 3 m, and bolting support is recommended. The simulation by RS2 software reveals that the optimal mining design that can be used to mine shallow-dip vein deposits hosted in poor quality rocks consists of a 3 m high stope and a 75° dip with cemented backfill.

Originality. This work presents a study to choose the most suitable underground mining method and mine design for shallow-dip deposits hosted in poor quality rock.

Practical implications. In the mining industry, the success of operating an underground mine is conditioned by the selection of the appropriate method, of the mining design and dimensioning of a rock support adapted to the nature of the rock, and excavation geometry according to the type and nature of the deposit.

Keywords: *mining method, displacement, unstable zone, rock mass, poor quality, backfill*

1. Introduction

Mine design is the key to the success and long life of the operation of an underground mine [1]. Once the geological data is completed and shows that the deposit has resources that can generate positive profits, the process of mining engineering can begin with the design that includes the choice of the mining method and the geotechnical sizing.

The first process that corresponds to the selection of the mining method (MMS) is to choose the method that best meets the unique criteria of each deposit, such as geometric, geological, geotechnical, depth, and other data, such as economic, technological, and environmental [2]. It should be mentioned that this process is the one that will decide on the profitability and sustainability of a mine because once you apply the chosen operating method, it is complicated to change it [3]. For this, several authors have developed approaches and algorithms to select the optimal exploitation method. These methods can be classed into three categories [4]. The first category was a qualitative classification based on graphs to choose the producing mining method [5]-[11]. The second category uses the weighting of several parameters are related to the geometry of the minera-lization and the geotechnical conditions of the rock [12]-[14], the most used is developed by MILLER ET AL [15], which is the UBC (University of British Columbia) approach. The third category is a method that uses the Multicriteria Decision Making Technique (MCDM) by introducing financial data, technical data, and data related to the geometry of the mineralization and the geotechnical conditions of the rock, the most used are analytic hierarchy process (AHP) [16], [17], analytic network process (ANP), analytic hierarchy process and fuzzy method (FAHP) [18].

After choosing the mining method, the second step in the mine design process consists of analyzing the mine stability and dimensioning the support suitable for the excavation geometries imposed by the chosen mining method and rock conditions. This stability analysis proposes solutions to prevent rockfalls, one of the most severe security problems in

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the mining industry. These rockfalls linked to the intersection of the gallery surface and existing discontinuities in the rock mass [19], the state of the stresses (in situ stresses, and the stresses induced by the excavation of gallery) [20], [21], the convergence of land following the increase in depth [22], [23] and the interference between excavations [24].

The main objective of this study is to choose the optimal mining method for shallow-dip vein deposits hosted in poor quality rocks by involving the UBC method. Then, we played a stability analysis of the chosen mining method through empirical and numerical simulations to determine the geometric elements such as the height and dip of the mining stope, the type of backfill, and the support rock required to ensure the stability of the excavation.

2. Methods

In this article, our idea is to present the mining design for shallow-dip vein deposits hosted in poor quality rocks, because vein deposits:

– considered an important source of metals like silver, gold, tin, copper, lead, and zinc for years to come [25];

– located all over the world: in Europe, Australia, Canada, South America, and Africa, and when they have a power of less than 3 m, these deposits vary in terms of geometry, dip direction, and width [26];

– have geological characteristics, sometimes very complex, often irregular and erratic [27];

– present a challenge in rock mechanics for mining engineers at the time of operation, especially since a large part of the deposits that remain in the world are developed further at depth with an inclined dip, for example, in Kazakhstan, 40 to 50% of vein-type gold deposits are defined in deep inclined and steeply inclined zones with intense fracturing of the ore and host rocks and variable deposit elements [28].

This mining design is constituted of two stages.

The first step is to apply the UBC method [15] to use to mine shallow-dip vein deposits hosted in poor quality rocks. This method is based on the Nicholas quantitative methods [14] it uses RMR for rock conditions instead of RQD, and it takes into account narrow deposits with a thickness than 3 m. UBC is a weighting method, which means that the mining method with the highest total represents the optimal mining method for the deposit studied [29].

The second step is to determine the mine design for the shallow-dip vein deposits hosted in poor quality rocks, using:

– the empirical methods that generally used in the mining industry [30], such as the Q-system method [31] and Rock mass rating (RMR) [32], to determine the stable geometry and the support necessary to ensure the stability of the excavation;

– finite Element Numerical Modeling (FEM), which frequently used to analyze geotechnical problems in the mine, as it can adapt to material heterogeneity, nonlinear deformability, complex boundary conditions, in situ and gravity constraints [33], [34].

2.1. Data collection

2.1.1. Geological and geometrical characteristic

In this article, the mining design is to be studied for vein deposits shallowly-dipping hosted in poor quality rocks. The geological data used in this study are summarized in Table 1.

Table 1. Geological and geometrical characteristic

Rock unit	Criteria	Description
Ore zone	Deposit shape	Irregular
	Deposit distribution	Erratic
	Ore thickness	3 m
	Ore dip	45°
	Depth	600 m
Hanging wall	poor quality rock mass	–
Foot wall	poor quality rock mass	–

2.1.2. Geomechanical and geotechnical characteristic

For the Geotechnical data, we used the shale data to simulate the Geotechnical parameters of the poor quality rock mass. This simulation was made by ORMAS V1.0 software: Online Rock Mass, Strength [35]. Table 2 summarizes the various Geotechnical parameters of the poor-quality rock mass.

Table 2. The characteristics geomechanical and mechanic for a poor quality rock mass

Description	Symbol	Value	Unit
Unit weight	δ	2600	kg/m^3
Intact rock strength	σ_{ci}	25	MPa
Hoek-Brown constant	mi	6	–
Geological Strength Index	GSI	33	–
Intact Elastic Modulus	Ei	5000	MPa
Disturbance Factor	D	0	–
Tunnel Depth	H	600	m
Friction angle	φ'	21.22	°
Cohesive strength	c'	0.7997	MPa
Rock mass compressive strength	σ_{cm}	0.5277	MPa
Rock mass tensile strength	σ_{tm}	-0.0267	MPa
Deformation modulus	Em	496	MPa
Poisson's ratio	v	0.3	–
The value of the Hoek-Brown constant m for the rock mass	mb	0.5482	–
Constants that depend upon the rock mass characteristics	s	0.0006	–
	a	0.5183	–

In this section, we also determine the parameters of geomechanical classification and empirical simulation such as Rock mass rating (RMR), Q-system (Q), vertical stress (σ_v), and Rock substance strength (RSS). These parameters are given by the equations, which are indicated in the following table.

Table 3. Rock mass classification ratings for the poor rock mass

Description	Symbol	Value	Unit	Correlation formula	Proposed by
Rock mass rating	RMR_{89}	50.78	–	(1)	[36]
Q-SYSTEM	Q	2.84	–	(2)	[37]
Vertical stresses	σ_v	15.60	MPa	(3)	[38]
Rock substance strength	RSS	1.60	–	(4)	[15]

$$RMR = 1.36GSI - 5.90; \tag{1}$$

$$RMR = 42.87Q^{0.162}; \tag{2}$$

$$\sigma v = \delta H; \tag{3}$$

$$RSS = \frac{\sigma ci}{\sigma v}. \tag{4}$$

2.2. UBC mining method selection

In this study, we used the UBC tool (a test version) developed by Mining Studio [39] to select the optimal mining method for shallow-dip vein deposits hosted in poor-quality rock. Figure 1 summarizes the results obtained. This tool shows that the cut and fill extraction method is the most suitable method for extracting this type of deposit.

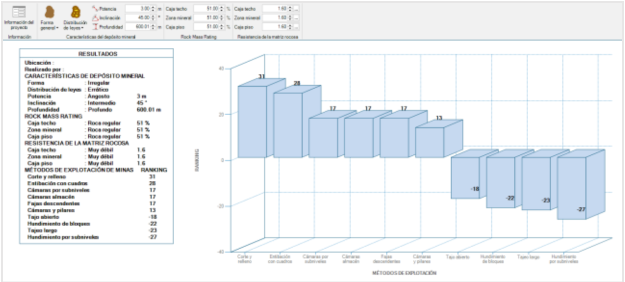


Figure 1. Ranking results obtained by the UBC tool [39]

2.3. Designs of the mining methods

2.3.1. Empirical method

The combination of the Benwinski (RMR) and Barton (Q-SYSTEM) geomechanical classification results with the stability graphs widely used to define the stable dimensions of the excavation (Fig. 2) [40] and the determination of the appropriate support for this excavation depending on the nature of the rock (Fig. 3) [41].

Figure 2 illustrate that the excavation for an RMR 89 value of 50.71 is located in the stable zone for a height of 3 m, while it is located in the transition zone for a height varying from 3 to 14 m, this excavation becomes unstable when the height of the stope exceeds 14 m.

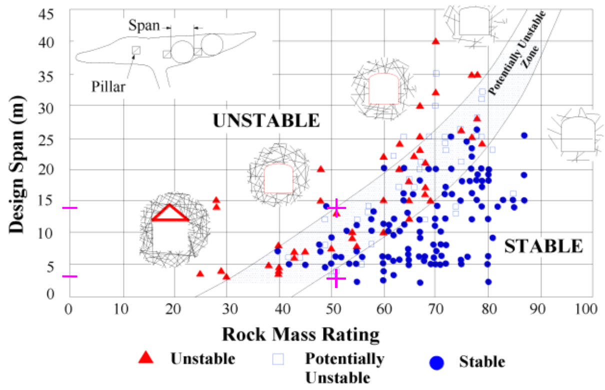


Figure 2. Stability study using the updated critical span curve [40]

The projection of the value *Q* of 2.84 on the graph of Figure 3 recommends support of the Systematic bolting type for an opening varies from 3 m and support of the Fiber reinforced shotcrete and bolting, 5-9 cm for a height varies from 6 to 12 m.

Table 4 summarizes the different types of support rock that can be used to stabilize the excavation for the different heights studied.

2.3.2. Numerical method

Analysis of the stability of mining excavation by numerical modeling is commonly used in the mining industry [24], [42]-[45].

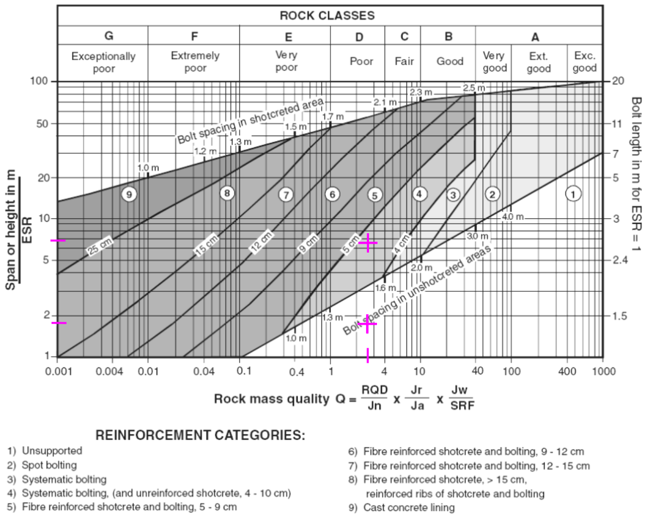


Figure 3. Estimated support categories based on Q [41]

Table 4. The different types of support recommended for each span

Zone	Stable	Potentially unstable	Unstable
Design span, m	< 3	> 3 and < 14	> 14
Rock support	Systi-matic bolting	Systematic bolting and Unreinforcced shotcrete (4 à 10 cm) for Span < 8 Fibre reinforced shotcrete and bolting, 5-9 cm for span > 8	Cable bolt and Fibre reinforced shotcrete and bolting, 9-12 cm for span

In this analysis, we used the RS2 software developed by RocScience Inc. [46] to analyze the displacements and the safety factor (SF) around the excavation generated by the overhand cut and wire operating method chosen as the extraction method optimal for exploiting shallow-dip vein deposits hosted in poor quality rocks.

Figure 4 shows the dimensions and geometry of the model used in this simulation, and to minimize the effects of the mining sequence the analysis will be done on a single stope, assuming that the rock mass is an elastic and isotropic material. A uniform mesh with 3 nodes was applied in the simulation. The HOEK BROWN criterion is applied as the failure criterion of the rock mass with gravitational stress to simulate the initial stresses in the rock mass over a depth of 600 m. Table 2 shows the geomechanical and mechanical parameters of the rock mass used.

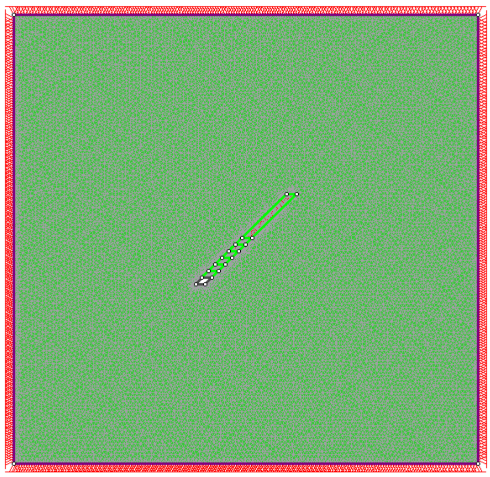


Figure 4. Geometric model used in analyses with RS2

The results of this analysis are presented according to the height of the mining chamber, the dip of the mining chamber, and the type of backfill used to fill the voids.

3. Results and discussion

3.1. Effect of the height of the stope

In this step of analysis, we will have analyzed the deformation and the extent of the unstable zone around the excavation to determine the optimum height of the design mine. The deformation will be represented by the total maximum displacement, while the extent of the unstable zone indicates the depth of the unstable zones, which have a safety factor less than 1 (SF < 1), this value is fixed at 1 view that the Mine openings are considered temporary openings [44]. This analysis will be done for different cases of the stress factor ratios K ($K = 1$, 1.5, 2). This factor represents the horizontal-vertical stress ratio and is determined by the following equation [38]:

$$K = \frac{horizontal\ stress}{vertical\ stress}. \tag{5}$$

The data published in the literature suggest that the stability of the rock mass depends on the height of the stope [47]. Figure 5 and 6 shows the evolution of the safety factor and the total maximum displacement at the turns of the excavation at three stress ratios.

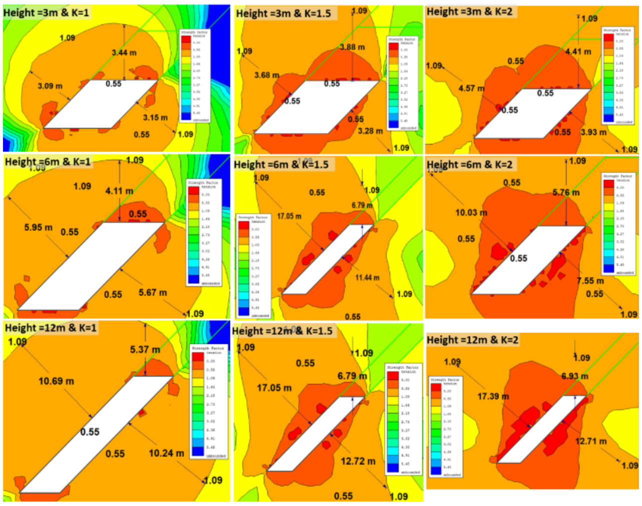


Figure 5. Numerical analysis by RS2 of the safety factor for several heights and stress ratio K

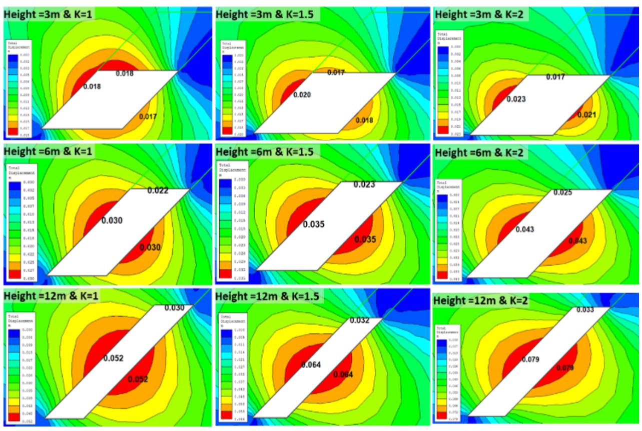


Figure 6. Numerical analysis by RS2 of the maximum total displacement for several heights and stress ratio K

The maximum width of the unstable zone (SF < 1) around the excavation is seen when the horizontal stress is two times the vertical stresses for all three types of openings, while the minimum width of this zone is seen when the stress horizontal equal to the vertical stress (Table 5). The total maximum displacement observed after excavation around the opening is recorded when the horizontal stress is two times the vertical stress for the three types of openings, while the minimum value of this parameter is recorded when the horizontal stress equal to the vertical stress (Table 5).

Table 5. The values of the unstable zone and total maximum displacement for each height and stress ratio

		Depth of unstable zone, m (SF < 1.09)			Total max. displacement, m		
Stope height, m	Stress ratio (K)	left wall	roof	right wall	left wall	roof	right wall
Height stope 3 m	1	3.090	3.440	3.150	0.018	0.018	0.017
	1.5	3.680	3.880	3.280	0.020	0.017	0.018
	2	4.570	4.410	3.930	0.023	0.017	0.021
Height stope 6 m	1	5.950	4.110	5.670	0.030	0.022	0.030
	1.5	6.820	4.900	5.920	0.035	0.023	0.035
	2	10.030	5.760	7.550	0.043	0.025	0.043
Height stope 12 m	1	10.690	5.370	10.240	0.052	0.030	0.052
	1.5	17.050	6.790	11.440	0.064	0.032	0.064
	2	17.390	6.930	12.710	0.079	0.033	0.079

Figure 7 shows the evolution of the safety factor and the total maximum displacement around the excavation for three types of height. The width of the unstable zone (SF < 1) will be multiplied by 2 when going from a height of 3 to 6 m and by 3.6 when going from a height of 3 to 12 m for the right and left wall, then that for the roof, the width of the unstable zone (SF < 1) will be multiplied by 1.25 when going from a height of 3 to 6 m and by 1.63 when going from a height of 3 to 12 m. The value of the maximum total displacement will be multiplied by 1.8 for a height from 3 to 6 m, and by 3.2 for a height from 3 to 12 m for the right and left wall. While for the roof, the total maximum displacement value will be multiplied by 1.35 for a height from 3 to 6 m and by 1.83 for a height from of 3 to 12 m.

3.2. Effect of stope geometry

The effect of the inclination on the stability of the stope was analyzed (Fig. 8 and 9). Table 6 illustrates the variation in the width of the unstable zone (SF < 1) and the total maximum displacement at different inclinations and for an opening of 3 m and an in situ stress ratio $K = 1$. This analysis shows that the width of the unstable zone and the total maximum displacement decrease with the increase in the inclination in the right and left wall, while these two indices evolve positively by increasing the inclination at the level of the roof (Fig. 10).

3.3. Effect of the nature of the backfill

In the underground mining industry, backfill is used as a platform to reach upper section mineralization and to increase containment in the excavation and fill voids created by mining [48]. Usually, backfill is subdivided into cemented backfill (CB) and uncemented backfill (UB) [49]. The uncemented backfill is represented only by-products from the excavation of the galleries or the open pit [48], this type of backfill has the same geotechnical characteristics of the waste rock used with zero cohesion [50].

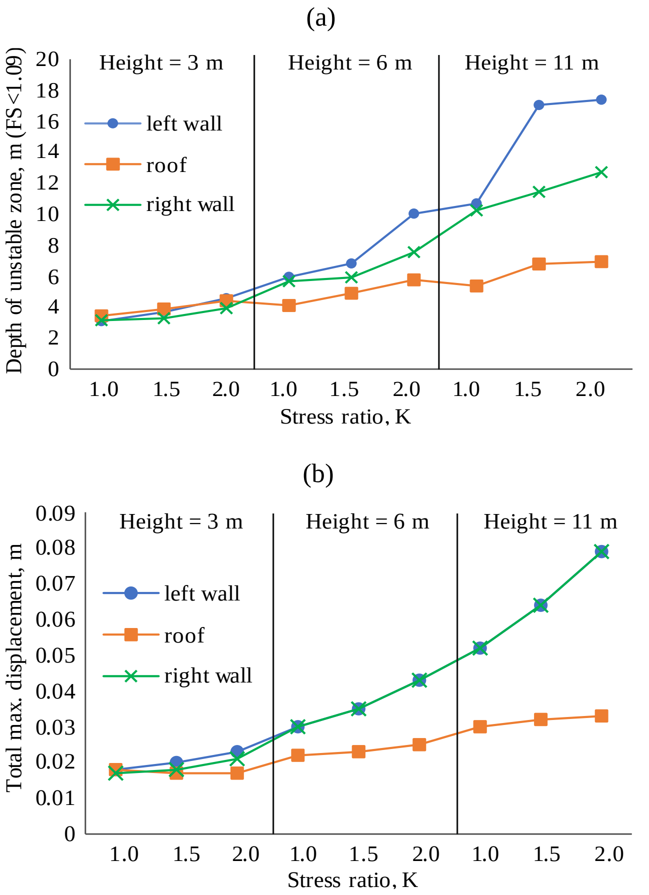


Figure 7. Evolution of the unstable zone and total maximum displacement: (a) the unstable zone around the excavation at different stress value materialized by the stress ratio K and at different heights of the stope; (b) total displacement around the excavation at different stress value materialized by the stress ratio K and at different heights of the stope

While cemented backfill involves mixing a material that can be waste rock or sand with the cement, the geotechnical characteristics of this type of backfill depend on several parameters such as the percentage and type of binder used, the nature and the particle size of the products used and the type and amount of water applied [51].

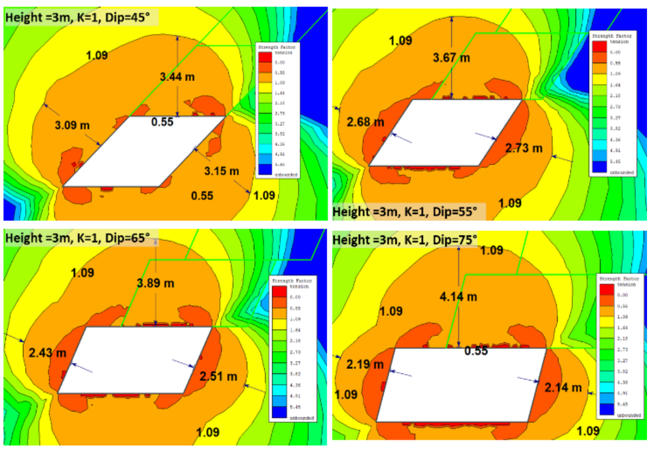


Figure 8. Numerical analysis by RS2 of the unstable zone for several inclinations

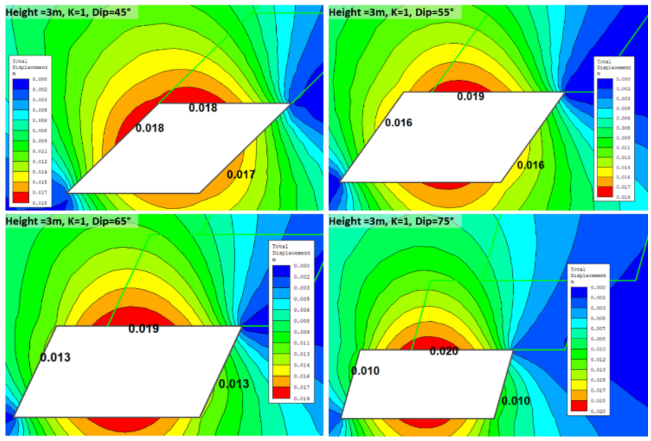


Figure 9. Numerical analysis by RS2 of the total maximum displacement for several inclinations

Table 6. The values of the unstable zone and total maximum displacement for several inclinations, with stope height = 3 m, stress ratio K = 1

Hanging-wall dip angle, °	Depth of unstable zone, m (SF < 1.09)			Total max. displacement, m		
	left wall	roof	right wall	left wall	roof	right wall
45	3.090	3.440	3.150	0.018	0.018	0.017
55	2.680	3.670	2.730	0.016	0.019	0.016
65	2.430	3.890	2.510	0.013	0.019	0.013
75	2.190	4.140	2.140	0.010	0.020	0.010

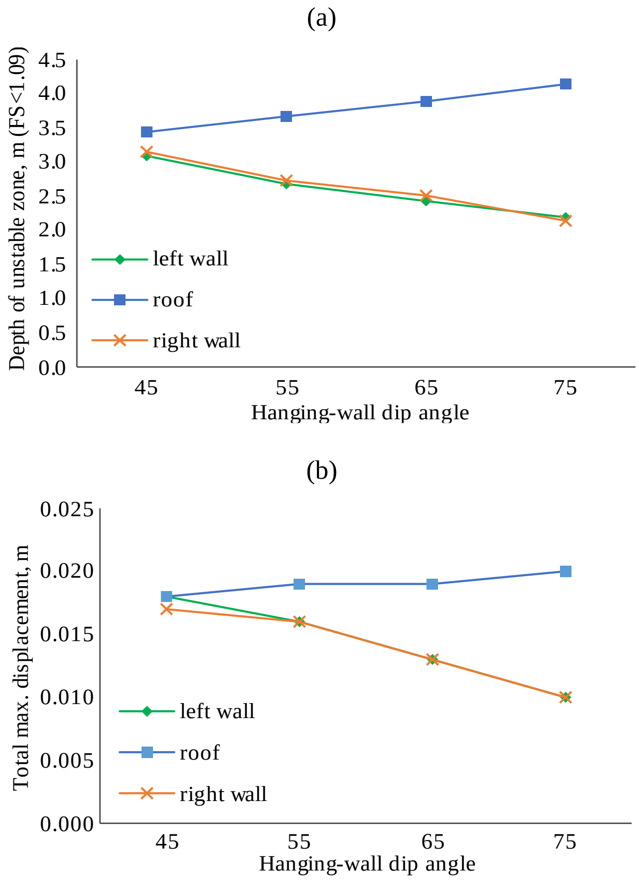


Figure 10. Evolution of the unstable zone and the total maximum displacement as a function of the inclination: (a) the unstable zone around the excavation at different inclination of the stope; (b) total displacement around the excavation at different inclination of the stope

Figure 11 and 12 illustrate the simulation after the backfill of the excavation by the cemented and uncemented backfill, using the geomechanical properties of the waste rock without cohesion for the uncemented backfill and the classical geomechanical properties presented by [52].

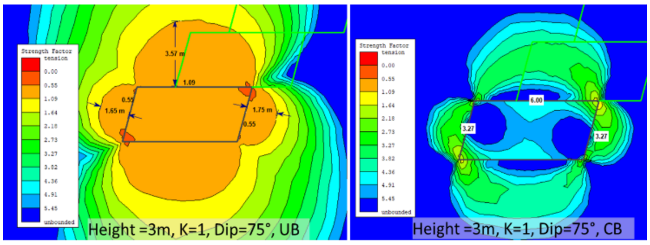


Figure 11. Numerical analysis by RS2 of the unstable zone as a function of the backfill

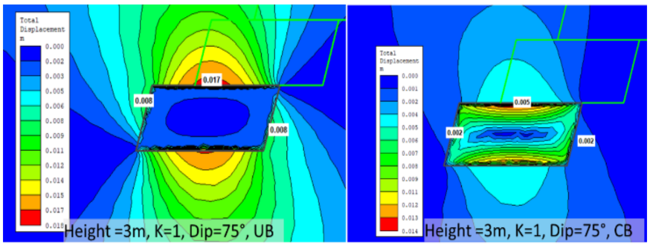


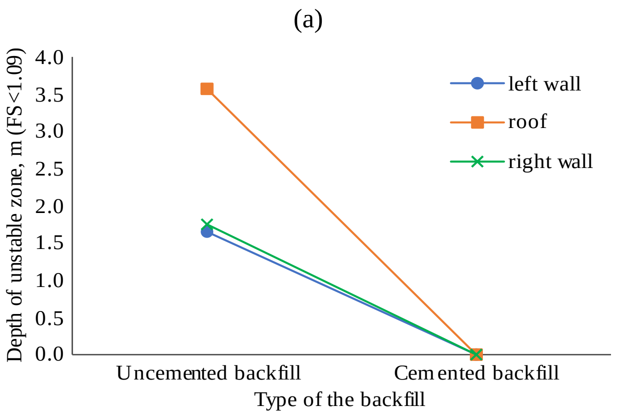
Figure 12. Numerical analysis by RS2 of the total maximum displacement as a function of the backfill

Table 7 summarizes the change in the width of the unstable zone (SF < 1) and the total maximum displacement depending on the type of backfill used.

This analysis shows the cemented backfill canceled out the unstable zone and reduced deformation by 25 to 29% around the excavation (Fig. 13).

Table 7. The values of the unstable zone and total maximum displacement for the backfill with Stope height = 3 m, stress ratio K = 1 and hanging-wall dip angle = 75°

	Depth of unstable zone, m (SF < 1.09)			Total max. displacement, m		
Type of backfill	left wall	roof	right wall	left wall	roof	right wall
Uncemented backfill	1.650	3.570	1.750	0.008	0.017	0.008
Cemented backfill	0.000	0.000	0.000	0.002	0.005	0.002



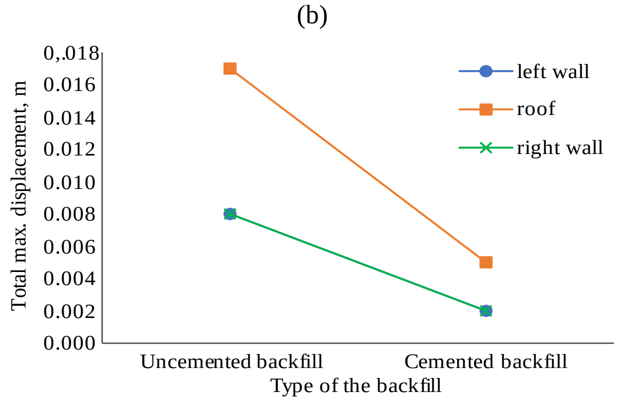


Figure 13. Evolution of the unstable zone and the total maximum displacement as a function of the backfill: (a) the unstable zone around the excavation as a function of the backfill; (b) total displacement around the excavation as a function of the backfill

4. Conclusions

In this study, several analytical methods such as mining method selection, empirical and numerical were used to determine the optimal mine design for shallow-dip vein deposits hosted in poor quality rocks. Based on geotechnical and geological considerations, the optimum mining method designated by the UBC tool for this type of deposit is the cut and fill. The geomechanical characteristics of the rock mass used as input data for the empirical and numerical analysis were determined from the ORMAS V1.0: Online tool and from the correlation equations between GSI and RM on the one hand and from somewhere else between RMR and Q-system. By projecting the obtained RMR and Q-system values on the design graph, it is concluded that the operating stope is located in the stable zone with a height of 3 m, and bolting support is recommended.

Then a series of numerical modeling was done with a variety of geotechnical, geometric, and technical conditions to analyze the evolution of the width of the unstable zone (SF < 1) and the maximum total displacement around the opening after the excavation. This simulation reveals that the optimal mining design that can be used to mine shallow-dip vein deposits hosted in poor quality rocks consists of a 3 m high stope and a 75° dip with cemented backfill.

Acknowledgements

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Дослідження та оцінка стійкості методу підземної розробки пологозалягаючих жильних родовищ у породах низької якості

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Мета. Вибір оптимального методу підземної розробки пологозалягаючих жильних родовищ, що залягають у породах низької якості, за допомогою емпіричного і чисельного моделювання для визначення стійких геометричні елементів гірничої виробки.

Методика. Для вибору оптимального способу розробки був використаний метод Університету Британської Колумбії (UBC), для визначення оптимальної гірничої виробки і рекомендованого типу кріплення – системи класифікації породних масивів RMR і Q, для аналізу зміни ширини нестабільної зони та загального максимального зсуву навколо виробки після виймання використовувалося чисельне моделювання в середовищі RS2 для різних геотехнічних, геометричних і технічних умов.

Результати. Шляхом проектування отриманих значень RMR і Q-системи на розрахунковий графік проведення гірничих робіт стало очевидно, що очисна виробка знаходиться у стабільній зоні висотою 3 м, і рекомендований тип кріплення – анкерний. Моделювання в середовищі RS2 підтвердило, що оптимальна схема розробки пологозалягаючих жильних родовищ у породах низької якості включає вибій висотою 3 м з кутом нахилу 75° і цементованою закладкою.

Наукова новизна. Науково обґрунтовані стійкі геометричні параметри підземного видобутку та схеми розробки пологозалягаючих жильних пластів у породах низької якості.

Практична значимість. Ефективна робота шахти залежить від вибору відповідного способу розробки, схеми гірничих робіт і параметрів кріплення, адаптованої до типу породи, а також від геометрії виймання з урахуванням типу і природи родовища.

Ключові слова: *спосіб розробки, зрушення, нестабільна зона, породний масив, низька якість, закладка*

Исследование и оценка устойчивости метода подземной разработки пологозалегающих жильных месторождений в породах низкого качества

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Цель. Выбор оптимального метода подземной разработки пологозалегающих жильных месторождений, залегающих в породах низкого качества, с помощью эмпирического и численного моделирования для определения устойчивых геометрические элементов горной выработки.

Методика. Для выбора оптимального способа разработки был использован метод Университета Британской Колумбии (UBC), для определения оптимальной горной выработки и рекомендованного типа крепи – системы классификации породных массивов RMR и Q, для анализа изменения ширины нестабильной зоны и общего максимального сдвига вокруг выработки после выемки использовалось численное моделирование в среде RS2 для различных геотехнических, геометрических и технических условий.

Результаты. Путем проецирования полученных значений RMR и Q-системы на расчетный график проведения горных работ стало очевидно, что очистная выработка находится в стабильной зоне высотой 3 м, и рекомендованный тип крепи – анкерный. Моделирование в среде RS2 подтвердило, что оптимальная схема разработки пологозалегающих жильных месторождений в породах низкого качества включает забой высотой 3 м с углом наклона 75° и цементированной закладкой.

Научная новизна. Научно обоснованы устойчивые геометрические параметры подземной добычи и схемы разработки пологозалегающих жильных пластов в породах низкого качества.

Практическая значимость. Эффективная работа шахты зависит от выбора соответствующего способа разработки, схемы горных работ и параметров крепи, адаптированной к типу породы, а также от геометрии выемки с учетом типа и природы месторождения.

Ключевые слова: *способ разработки, сдвиг, нестабильная зона, породный массив, низкое качество, закладка*