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Design and application of an efficient mining method for gentle-dipping narrow vein at Kafang Mine

F Gao *Central South University, China*

KP Zhou *Central South University, China*

HW Deng *Central South University, China*

NG Yang *Central South University, China*

JL Li *Central South University, China*

Abstract

It is recognised that a gently-dipping narrow vein is hard to be exploited due to the difficulties in ore drawing and operating large mining equipment in the restricted stoping space. Therefore, considerations are required to adopt an efficient mining method that is capable of dealing with the irregular boundary of this kind of orebody, resulting in a high recovery rate while minimising the ore dilution and the preparations prior to stoping. In the case of mining the low-grade ore cluster I-9 at the Kafang Mine of Yunnan Tin Industry Company, China, the requirements of mining at a large scale poses another challenge. For the ore cluster I-9, there are four layers of ore bodies, with an average thickness of 2.22 m and dip angles in the range of 15-20°. The average grade of the main commodity (Cu) is only 0.58%. After evaluating and comparing different mining methods based on a geotechnical investigation, rock mechanics tests and rock mass classification, the efficient breast stoping (overall mining method) was selected. Specifically, the stopes along the strike direction were arranged at an angle of 45°, with respect to the dip direction (termed apparent dip layout henceforth). This layout enhanced the operating ability of load-haul-dump (LHD) and other trackless equipment. With a Sandvik drilling jumbo DD210L and a TCY-2A diesel LHD, continuous mining of the full thickness of the orebody significantly increases production when compared with previous mining practice. The ore loss decreases from 14.6 to 8.2%. The extraction-to-cut ratio is down from 20.6-9.2 m/kt. The mining cost is reduced from CNY 158.88 to CNY 104.35 per tonne. This mining method has the potential to be widely used to exploit similar orebodies.

1 Introduction

Gently-dipping narrow veins are those with a dip angle less than 30° and a thickness between 0.8 and 4 m. Currently the main approaches used to extract the mineral resources in this type of deposit include the room-and-pillar method, breast stoping (overall mining method), the filling method, longwall caving. In order to improve mining efficiency, various mining methods have been proposed, and some of them have proved efficient in mining practice over the last decade (Yi & Wang 2007; Zhou et al. 2010; Yang & Yuan 2013; Guo et al. 2014).

Over the last 20 years, mining of gently-dipping narrow vein using traditional technologies in China suffered from a low level of mechanisation, high labour intensity, low stope production, high dilution and loss and sometimes faced unsafe working conditions. To tackle these challenges, mining researchers in China have developed many new combined mining methods, such as an open stope method in room-and-pillar within the overall ore block, a vertical striping mining method, an advancing breast stoping method and others, which significantly improved the mining efficiency. Examples of these new methods can be found in many mines in China, such as Hexi Gold Mine (Wang et al. 2003), Wangershan Gold Mine (Ding et al. 2014), Yangpingwan Gold Mine (Cheng 2008) and Tongkeng Mine (Zhou et al. 2012a).

Outside China, the traditional room-and-pillar method is applied in Tweefontein Chrome Mine of South Africa's Samancor region. The high level of mechanisation together with the use of underground trackless equipment brought high efficiency, but also high cost and dilution (Guo & Wang 2010). Longwall stoping is also used extensively in South Africa and to a less extent in other countries, such as Chiatura Mine of Georgia (Caruso et al. 2012), Moyeuvre Mine in France.

Based on the above, it can be concluded that there is no universal mining method that can be applied everywhere with the same operational procedure without considering the complexity of the local geology. The mining method must be varied for different specific deposits. In summary, there are three dominant techniques worldwide for mining gently-dipping narrow vein:

- *Longwall mining method*: this method is suitable for mining bedding deposit with a dip less than 35°, 3-4 m thickness and poor roof. The advantages of this method are simple layout, high labour productivity, good stope ventilation and low dilution and loss. The ore recovery is typically high, reaching up to 80-95%. However, it is very labour-intensive with roof supporting and complicated roof control. High material consumption of wooden or hydraulic support increases the total mining cost.
- *Room-and-pillar method*: this method is used particularly in mining in competent ore and surrounding rock and with a dip less than 15°. Development mining is largely in the orebody, and trackless equipment can be used to enhance the production capability. On the other hand, the necessity of leaving pillars leads to a low recovery ratio. Therefore, there is more ore loss when mining under high ground pressure because more and larger pillars are needed.
- *Breast stoping*: it is an open stoping method often used for mining gently-dipping narrow vein orebodies with thickness between 1.5 and 3 m and irregular floor. Ore and surrounding rock should be at a minimum, moderately competent. This method also has to leave pillar.

2 Geology and technical conditions of mining at Kafang Mine

2.1 Geological outline

The deposit of Dabaiyan mining area in the middle of Kafang Mine of Yunnan Tin Group (Holdings) Co., Ltd, is located in the south of Yunnan Province, China. It is predominantly a copper (Cu) resource with by-products such as tungsten (W), gold (Au), silver (Ag), bismuth (Bi), sulphur (S) and fluorite elements. It is a genesis of basic volcanic rock-type copper deposit in the Yuannan-Guizhou Plateau. The types of ore mainly comprise mono-copper sulphide with dense massive structure, and disseminated and veinlet ore (Yang et al. 2010). Pyrite and pyrrhotite minerals constitute more than 90% of the ore.

Ore cluster I-9 in the Xinshan ore section of Dabaiyan lies in the elevation of 1,800-1,950 m and exists between no. 2-1 and no. 13 exploratory lines. It is divided into four parts, namely I, II, III, and IV, according to ore-controlling factors and the layer characteristic of the ore deposit. It covers an area of 1,067 × 500 m along strike and dips southeasterly. The horizontal projection area is 0.96 km^2 and a geological resources estimate of 4.2 million tonnes. The average thickness of the orebodies is 2.22 m, and most of them are gently-dipping. The geological parameters are presented in Table 1 and Figure 1.

Table 1 Geologic features of ore cluster I-9

	Section I	Section II	Section III	Section IV
Exploratory line	8-1 to 13	9-1 to 12-1	7-1 to 13	8 to 10
Elevation (m)	1,800-1,900	1,800-1,974	1,800-1,920	1,800-1,870
Strike	NE 28-36°	NE 40-52°	NE 50-70°	NE 35-55°
Length along strike (m)	230-420	317	559	155
Length along dip (m)	230	230	45-568	53-111
Dip angle	15-20°	15-20°	15-18°	5-28°
Average thickness (m)	1.40	1.90	3.57	2.58
Average ore grade	0.678%	0.579%	0.48%	0.402%

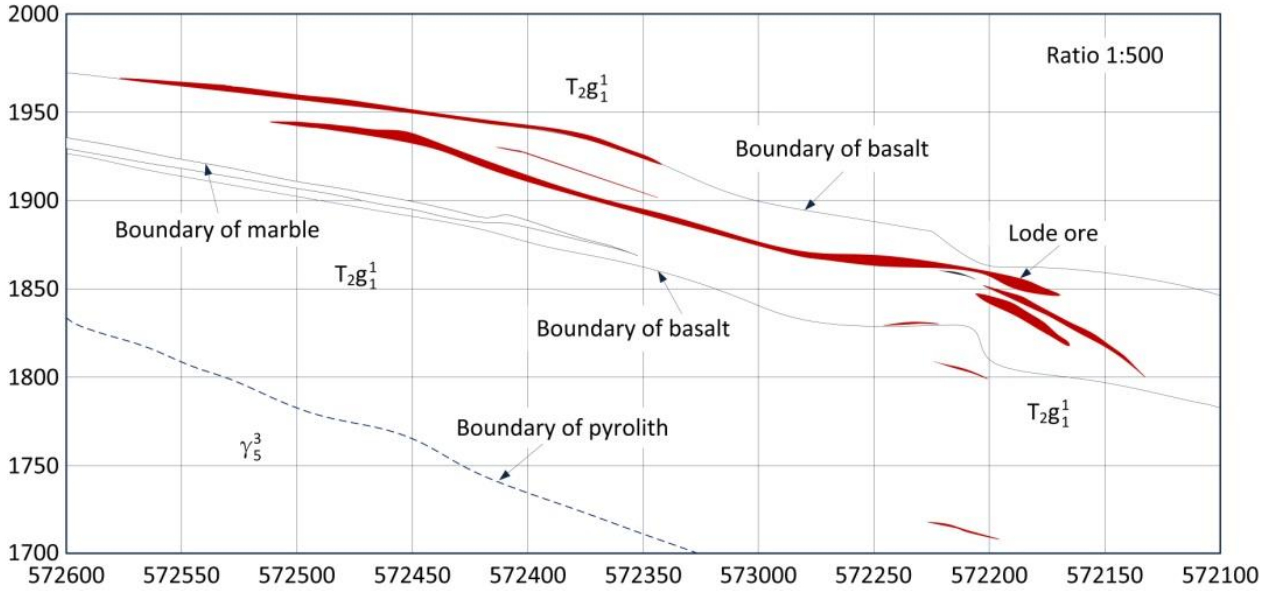

Figure 1 Longitudinal section of no. 10 exploration line of ore cluster I-9

2.2 Technical conditions of mining

2.2.1 Stability of rock mass

Hanging wall and footwall of ore cluster I-9 mostly consists of metal-basic basalt and marble, with a few calcite dolomite and skarn locally. They are competent, with a Protodyakonov coefficient f = 10-15. The orebody generally consists of sulphide ore with fine structure (hard and compact) and of good quality as well, with f = 10.

2.2.2 Mechanical properties of the rock

According to the standard testing method of the International Society for Rock Mechanics, uniaxial compressive strength test (Figure 2) and Brazilian test (Figure 3) were conducted on the three typical rock types (marble, basalt and ore-bearing basalt), by the Instron-1346 electro-hydraulic servo compression machine. Table 2 lists the basic properties of the rocks.

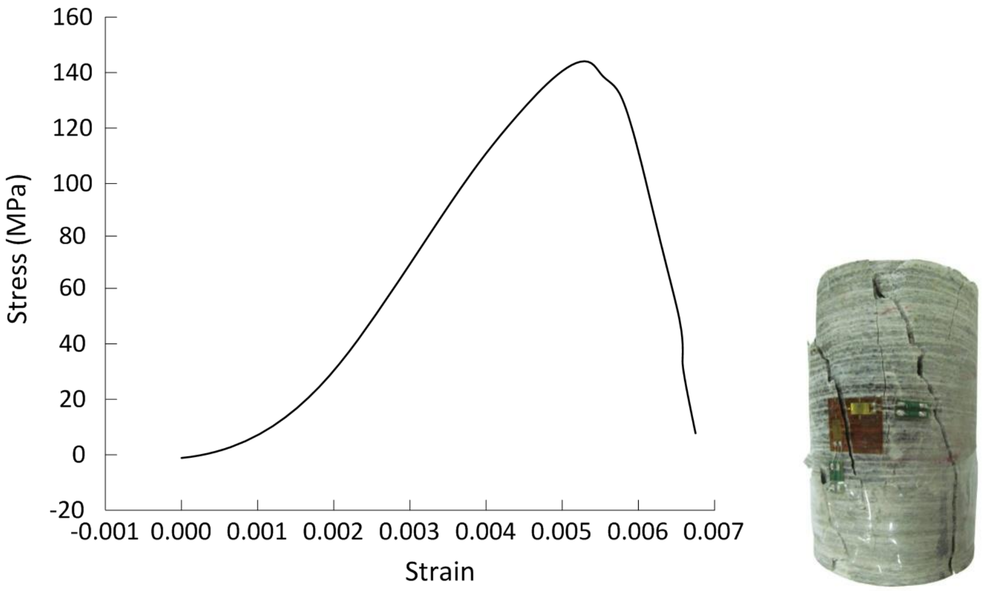

Figure 2 Complete stress-strain curve and failure form of rock sample (marble)

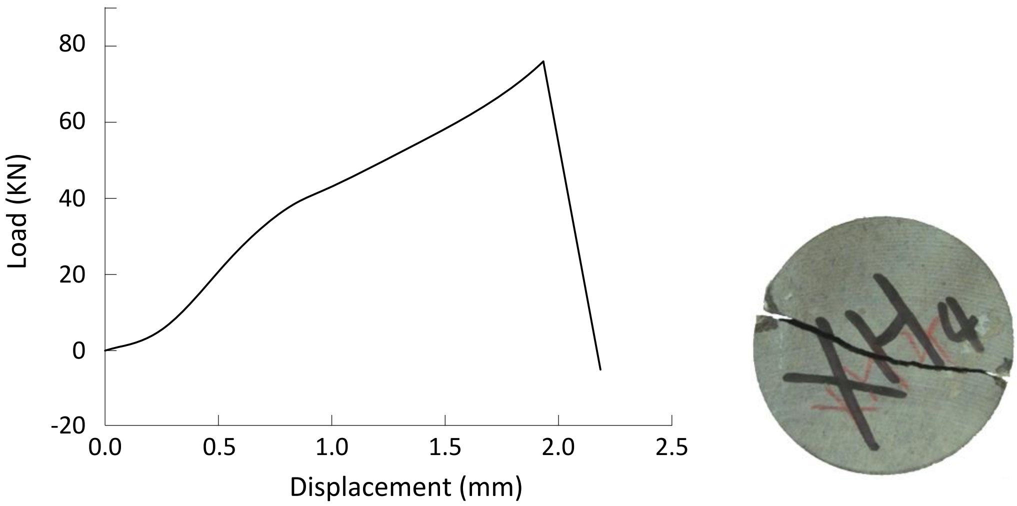

Figure 3 Load-displacement curve and failure form of rock sample (ore-bearing basalt)

Table 2 Average mechanical properties of rock samples

Rock type	Uniaxial compressive strength (MPa)	Tensile strength (MPa)	Young's modulus (GPa)	Poisson's ratio	Cohesion (MPa)	Internal friction angle (°)
Marble	132.3	6.3	43.0	0.25	14.3	72.5
Basalt	159.1	13.9	57.9	0.25	23.5	63.4
Ore-bearing basalt	151.3	8.9	51.5	0.25	18.6	70.0

2.2.3 Hydrogeology

Annual rainfall in the mine area is 1,335.08-1,884.1 mm. It is the major source of underground mine water, which is stored in karst depressions and funnels into the overlying strata and rock fractures etc. Therefore, the hydrogeology condition is low to moderate and does not have a significant effect on underground mining.

2.3 Rock mass quality and maximum exposed area of roof

2.3.1 Rock mass quality evaluation

According to detailed field investigations and rock mechanical tests, the rock mass rating (RMR) (Bieniawski 1974) was used for assessing the quality of the rock mass. In the RMR classification system, parameters including uniaxial compressive strength (A_1), rock quality designation (*RQD*) (A_2), spacing of discontinuities (A_3), condition of discontinuities (A_4), groundwater (A_5) and rating adjustment for discontinuity orientations (B) were calculated; they are presented in Table 3. The total rating is $\sum$ (A_1,A_2,A_3,A_4,A_5,B) = 85. As a result, the rock mass was classified as good.

Table 3 Value of each evaluation index of rock mass in RMR

	A_1	**A_2**	**A_3**	**A_4**	**A_5**	**B**
Value	132-159 MPa	92%	1,200 mm	Note 1	Wet	Note 2
Rating	12	20	15	30	9	-2

Note 1: For very rough surfaces, not continuous, no separation and unweathered wall rock, the rating is 30 given by the RMR system.
Note 2: The RMR system gives a description of 'favourable' for the conditions assumed where the tunnel is to be driven with the dip of a set of joints dipping at 30°. Using this description for 'tunnels and mines' in the RMR system gives an adjustment rating of -2.

2.3.2 Maximum exposed area of roof

It was necessary to determine the stable excavation spans for underground mining. For this end, three methods, i.e. engineering analogy, Mathews stability chart (Mathews et al. 1980) and numerical simulation, were intensively utilised to determine the specifications for stoping.

The Mathews design method is based on the calculation and mapping of two factors. The first factor is the stability number, *N*, representing the stability level of the rock mass under given conditions of stress, rock structure and orientation of rock surfaces. The second factor is the shape factor or hydraulic radius, *S*, which takes the geometry of the stope excavation surface into consideration. The Mathews stability graph consists of three zones: a stable zone, a potentially unstable zone and a potential caving zone. There is a transitional zone between the stable and the unstable zones. The original graph was modified by Potvin et al. (1989), where, as shown in Figure 4, the unstable zone is excluded.

2.3.2.1 The stability number, N

The stability number, *N*, is defined as

$$N = Q' \times A \times B \times C \quad (1)$$

where:

Q' = modified rock quality factor of *NGI* (Barton et al. 1974).

A = rock stress factor.

B = joint orientation adjustment factor.

C = gravity adjustment factor.

According to empirical formula RMR = 9ln$Q$ + 44, then Q can be expressed as $Q = e^{\frac{RMR-44}{9}}$. Substituting RMR = 85 into this formula, we obtain Q = 95.16. In view of variation of practical engineering geology, using Q approximating for Q', then Q' = 95. Factor A is determined from the ratio of the uniaxial compressive strength, σ_c, to the induced compressive stress, σ_i, which is calculated by σ_c/σ_i. Based on graph to determine factor *A* from rock strength and rock stress (after Potvin et al. 1989), the value of $A = 1$ in this paper. $B = 0.5$ to 0.8, which is determined based on engineering geology survey and determination chart (Stewart & Forsyth 1995). Due to angle of dip in hanging wall being 18°, then $C = 8\text{-}7\cos 18° = 1.343$ (Stewart & Forsyth 1995). Given the calculated results, the value of *N* is in the range of 84.8-102.

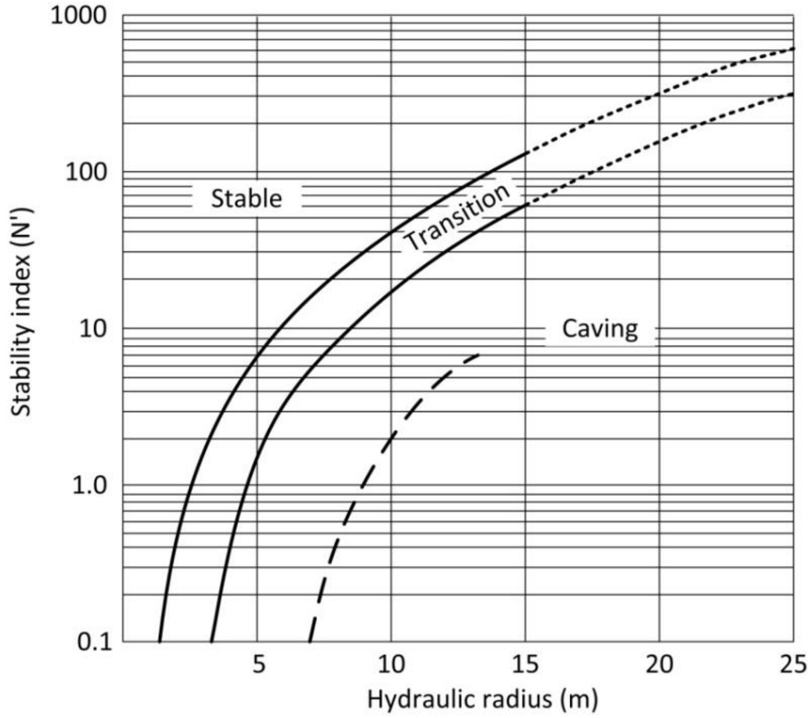

Figure 4 Updated Mathews stability graph (after Potvin et al. 1989)

2.3.2.2 Stope shape factor S

According to the value of stability number, *N*, the value of the stope shape factor, *S,* can be determined by the Mathews stability chart in the range of 11.8-13.5. Therefore, the maximum stable shape factor (hydraulic radius) of the exposed stope area is 13.5.

2.3.2.3 Stability exposed area and maximum exposed area of roof

Considering the length of stope is 80 m, six schemes of stope dimension are given as illustrated in Table 4. The appropriate span of stope is between 40 and 45 m and the maximum exposed area of roof is 3,200-3,600 m^2, thus the rock mass can sustain itself well.

Table 4 Evaluation results of roof stability by the Mathews method

Scheme	Length of stope (m)	Width of stope (m)	Exposed area of roof (m^2)	Shape factor *S*	Roof stability
1	80	20	1,600	8	Stable
2	80	25	2,400	10.9	Stable
3	80	30	2,800	12.2	Stable
4	80	35	3,200	13.2	Transition
5	80	40	3,600	14.4	Transition
6	80	45	4,000	15.4	Caving

3 Design of mining method of ore cluster I-9

This study considers the parameters of mine design on orebodies with a thickness of less than 5 m, which accounts for 77.4% in all orebodies. The disadvantages of the conventional breast stoping used in the ore cluster I-9 are as follows:

- Labour-intensive due to low level of mechanisation.
- Less production capacity of the stope. Only 40-60 tonnes per day of a single stope.
- High loss ratio of 14.6%.
- High cost of production with CNY 158.88 per tonne (in 2013).

Generally, there are three technological issues that need to be solved when exploiting the gently-dipping narrow vein. Firstly, a safe and reliable technology for roof support with low cost is required as the exposed area of roof increases. Secondly, the handling of ore by gravity in the stope is impossible because the orebody has a low dip; therefore, ore haulage efficiency must be improved. Thirdly, the interaction between the multilayer orebodies, especially for the barren rock interlayer in small thickness, must be considered.

At present, the technology development trend for exploiting the gently-dipping narrow vein includes studies of efficient mining methods with high recovery, the development and usage of trackless equipment, hydraulic and a high degree of automation facilities for improving mining efficiency. More importantly, scientific management and optimisation of mining system are significant for increasing the production capacity.

For mining of multilayer orebodies, on must consider whether the interstratified barren rock is to be mined with up-down layer veins together. One feasible method to determine workability of thickness of barren rock is to build a fuzzy break-even analysis model based on current minerals prices. Furthermore, numerical simulation could be used for the analysis of mining interstratified barren rock with an appropriate thickness to ensure that stope roof is safe. Admittedly, this is a more complicated design method of underground mining.

3.1 Apparent dip layout of stope

Ore break down by medium depth bore blasting and the use of LHD units for loading ore play an important role in improving efficiency in the exploitation of the gently-dipping narrow vein. Limited by its low moving flexibility and the long load distance, the LHD is an applicable unit with high efficiency where the maximum grade is less than 12° and the load distance less than 150 m. The orebodies are mainly gently-dipping, but some are inclined from 15-25°. The high dip angle is beyond the reach of trackless equipment. In order to solve this problem, a proposal was initiated in the form of a large panel, small subsection, apparent dip layout and mining synergy.

It is unreasonable to arrange a haulageway for a LHD directly along the ore dip when the dip angle is larger than 15°. Based on space geometry and a trigonometric function relationship, the maximum grade can be reduced when there is an included angle between the haulageway of a LHD and the ore dip. This type of layout is termed apparent dip layout of stope (Zhou et al. 2012b). Figure 5 illustrates the spatial relationships about the layout.

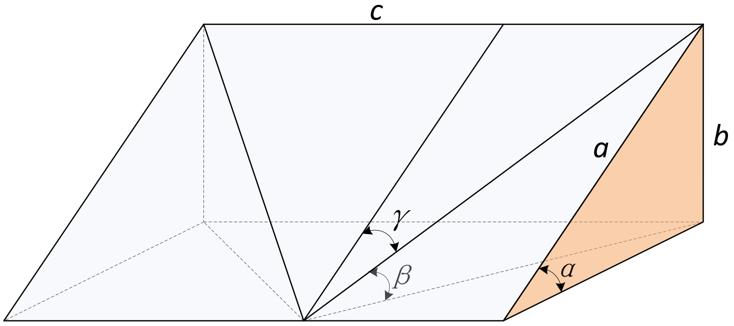

Figure 5 Spatial relationships between ore dip and the apparent dip: (*a*) length of stope along ore dip; (*b*) height of subsection; (*c*) length of stope along ore strike; (*α*) ore dip angle; (*β*) apparent dip angle; (*γ*) angle between apparent dip and ore dip

According to Figure 5, the relationship of α, β and γ is

$$\cos\gamma = \sin\beta / \sin\alpha \tag{2}$$

Therefore, the included angle needed for the layout of a haulage tunnel for a LHD in practice can be determined by Equation 2. Considering a valid load distance of LHD and quantity of stope, the length of stope along ore dip a is given as

$$a \leq (L - d \times n) \times \cos\gamma \tag{3}$$

where:

d = width of stope (m).

n = quantity of stope shared with one orepass (n is a natural number, and there is one stope with one orepass when n = 0).

L = effective load distance of LHD (m).

Consequently, stope parameters can be optimised according to Equation 3 for design of mining method.

3.2 Design of mining method

3.2.1 Parameters of stope

A schematic diagram of the mining method is shown in Figure 6. The apparent dip of the stope along dip direction decreased from 17° (section D-D in Figure 6) to 12° (section C-C in Figure 6), while the direction of stope and dip direction formed an angle of 45°. It creates a preferable condition of ore removal for LHD. However, this angle should be decreased to 40° where the orebody is steeper, in exploratory line from no. 9 to 8-1 in the mine.

Generally, the whole room is 80 m in length and 40 m in width, arranged along strike. The height of the room is the thickness of the orebody. There are two 6 m wide barrier pillars between two rooms at a spacing of 3 m. The pillar recovery will be undertaken after the stoping and gob handling of the upper stope room is completed. Under the apparent dip layout of the stope, each room can be divided into six stopes. The longest length of the side of stope is 56.5 m and the average width is about 20 m. Advancing mining in stoping can be used in mining under conditions of good rock quality, as well as large exposed areas of roof. Conversely retreating mining in stoping is used for bad rock quality. However, there are two sharp corners in each part when the stope layout is apparent dip. It would lead to some unexpected ore loss in practice.

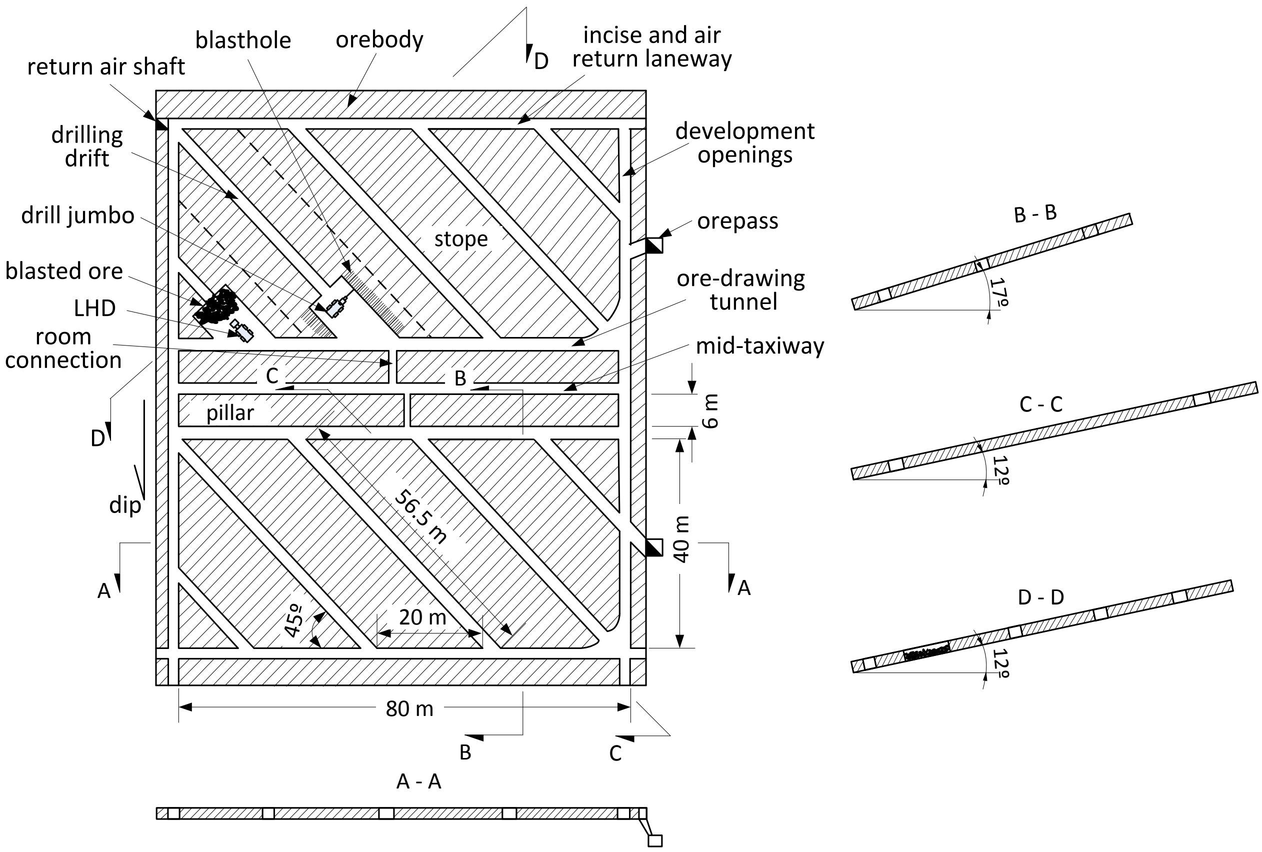

Figure 6 Schematic diagram of mining method

3.2.2 Stoping process

As can be seen from Figure 6, a return air shaft is located in the corner of the chamber, which connects with the upper level return air tunnel. One orepass is included in each chamber. Along the strike of the orebody, two tunnels are needed to be excavated in the two sides of the chamber. The upper one is used for ventilation and worker access, and the other is used for ore removal and worker access. The mine-out area is subsequently filled with mining waste rock.

The equipment used for mining comprises a YT-29 short-hole rock drill, a Sandvik DD210L jumbo, a TCY-2A diesel LHD and a JZC-12 underground ore truck from Tongguan Machinery Company in China. An Atlas Copco ST 1030 diesel LHD is used as a standby. The DD210L jumbo can be used for drilling blastholes 3.6 m in depth and 65 mm in diameter. The blasthole spacing is 1.0-1.2 m, and array pitch is 0.8-0.9 m. By using multi-millisecond blasting, about 20 rows holes are blasted at a time with about 300-600 tonnes of blasted ore.

3.3 Analysis on stability of stope and stoping sequence

3.3.1 Numerical model

Numerical simulation is an effective way to examine the stability of a stope and optimise the stoping sequence in rock engineering. In this paper, a coupling method of modelling was carried out as follows:

- Construction of a 3D digital model using Surpac software. A 3D digital model of the orebody surface and stratum for the whole mining area was built using Surpac, as illustrated in Figure 7(a).
- Meshing using MIDAS/GTS software. In this step, the meshing operation on the MIDAS/GTS model included seaming of the curved surface, trimming, partitioning and insetting of entity etc. The mode for numerical simulation with meshing is shown as Figure 7(b), which is a tetrahedron with

four nodes of grid unit. The meshed model of the stope with a 2 m grid unit is shown in Figure 8, in which the third row of stopes is apparent dip due to the dip angle.

- Calculation conducted by FLAC3D software. After the meshed model was imported into FLAC3D, the calculation was started once the pre-processing was completed.

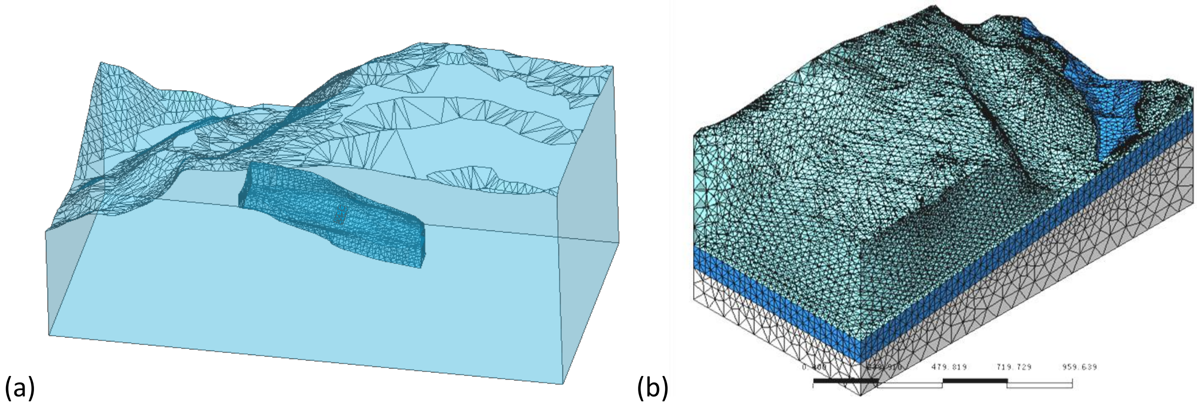

Figure 7 (a) 3D digital model; and (b) meshing model

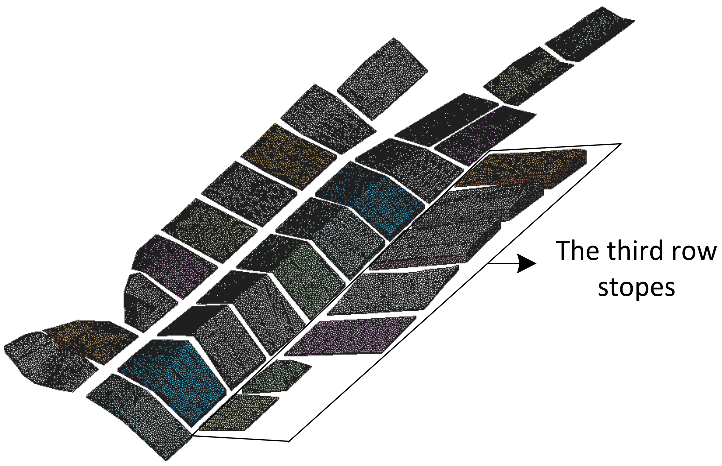

Figure 8 Meshing model of stopes

3.3.2 Parameters of numerical simulation

As rock parameters from laboratory testing are very different from a practical rock mass, they should be adjusted based on engineering geological survey before numerical simulation. The bulk modulus and shear modulus are parameters also needed in FLAC3D. The rock parameters used for numerical simulation are given in Table 5. The Mohr–Coulomb criterion was adopted during the simulation.

Table 5 Rock parameters for numerical simulation

Rock type	Tensile strength (MPa)	Poisson's ratio	Cohesion (MPa)	Angle of internal friction (deg)	Modulus of elasticity (GPa)	Bulk modulus (GPa)	Shear modulus (GPa)
Marble	3.14	0.25	10.72	58	30.12	20.08	12.05
Orebody	6.85	0.25	17.65	50.7	40.51	27.01	16.20
Basalt	4.46	0.25	13.96	56	36.07	24.04	14.43

3.3.3 Results of numerical simulation

The numerical simulation results show that the mining area is stable during the excavation. The maximum compressive stress in the ore-drawing tunnel is 26.3 MPa (section (a) along dip, as shown in Figure 9), compared with the maximum compressive stress of 50 MPa in the pillar (section (b) along strike, as shown in Figure 9). Furthermore, the maximum tension stress is about 4 MPa in the roof and the maximum vertical displacement of stope is 0.1 m (section (c) along strike as shown in Figure 9) when stoping is finished. The value for maximum stress is within the safety range; tensile stress and plastic zone do not appear in the model. It proves that the designed mining parameters are feasible.

According to simulation results, there are no significant differences of subsidence displacement and the maximum and minimum principal stress in rock induced by different sequence of stoping. Due to the gently-dipping orebody, the main factors that affect the secondary stress distribution between adjoining stopes should be considered when making decisions on the stoping sequence. The optimum stoping sequence is from bottom to top and from the centre to the sides of the panel mining.

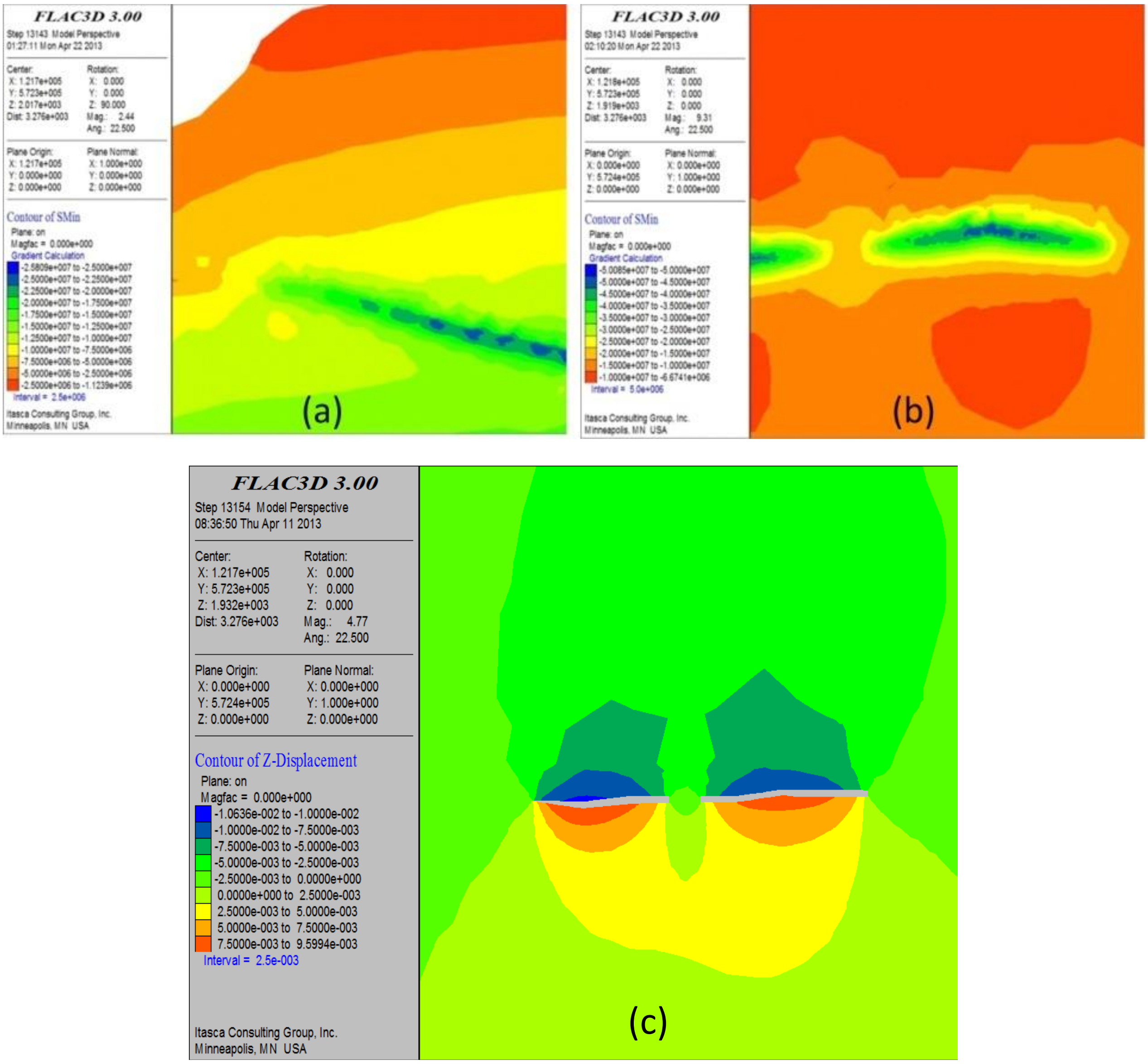

Figure 9 Sections of numerical simulation results: (a) section across ore-drawing tunnel along dip; (b) section across pillar along strike; (c) section across stopes along dip

3.4 Technical & economical comparison

By improving the mining method of ore cluster I-9 of the Kafang Mine, the daily production capacity of single copper ore has been raised from 1,396 to 2,400 t. The mining efficiency per man-shift have been nearly tripled from 7.5 to 20.6 t. Since the new stoping technology was applied, especially because scraper hoists were replaced by LHD in ore loading-hauling, the number of front-line miners had been reduced from 324 to 163. More importantly, the manual tasks of miners decreased with the increasing level of mechanisation. Therefore, the direct mining cost dropped from CNY 47.32 to CNY 31 per tonne, the total mining cost decreased from CNY 158.88 to CNY 104.32 per tonne. Due to the cost reduction, the ore cut-off grade was reduced from 0.6 to 0.4%. Hence, a great number of low grade resources became economically minable, including 242 million tonnes of ore containing 9,240 tonnes of copper. Table 6 summarises the technical and economic comparisons between the current and previous mining methods.

Table 6 Comparison between the technical and economical indexes

Number	Item	Unit	Value		
			Present	Past	Range of change
1	Ore removal cutoff grade	%	0.592	0.603	-1.8%
2	Dilution rate	%	11.2	9.6	16.7%
3	Loss rate	%	8.2	14.6	-43.8%
4	Mining-cut ratio	m/kt	9.2	20.6	-55.3%
5	Mining efficiency per man-shift	t/gang	20.6	7.5	174.7%
6	Stope production cycle	day	120	390	-69.2%
7	Stope production capacity	t/d	400	100	300%

4 Conclusion

In order to address the issues related to exploiting the gently-dipping narrow vein of the ore cluster I-9 in the Kafang Mine, a modified mining method of an apparent dip layout of stoping was proposed based on the traditional breast stoping at this mine. The stope with an apparent dip layout facilitated the employment of trackless equipment such as LHD and jumbo. The new layout possessed the following advantages over traditional methods: it prevented under-cutting in the process of stope preparation; it brought a higher level of mechanisation and production efficiency; it realised continuous mining over the full thickness of the orebody; it ensured safe and low-intensity working conditions; and it reduced the overall cost. A series of problems in practice have been solved in the initial stage using new mining technology, by optimising the development and transport system and providing skill training for operators using the trackless equipment. Practical applications suggest that the goal, as expected, has been reached in terms of the economic and social benefits. It can be concluded that the mining method not only presents a new way of extraction of the gently-dipping narrow vein in Kafang Mine, but also highlights its potential for the mining of similar orebodies worldwide.

Acknowledgement

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