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Stability analysis of large-scale stope using stage subsequent filling mining method in Sijiaying iron mine

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ABSTRACT

To improve mining production capacity, a stage subsequent filling mining (SSFM) method is employed for Sijiaying iron mine. The height of the stage stope is approximately 100 m. As there are farmlands and villages on the surface of the mine, the surface deformation should be controlled when the ore is mined out for the purpose of stope stability and minimizing surface subsidence. In this paper, according to the site-specific geological conditions, the self-stability of the stage-filling body was analyzed, and the failure mechanism of backfilling body was defined. Thus the relationship between the exposed height of filling body and the required strength was obtained. Next, the stability of backfilling body and the characteristics of surface subsidence due to mining of −450 m level were analyzed using physical modeling. Finally, a three-dimensional numerical model was established using FLAC^{3D}, with which the surface subsidence and the stability of stope were achieved. The results show that the stope basically remains stable during the two-step recovery process. The maximum magnitude of the incline is 10.99 mm/m, a little larger than the permissible value of 10 mm/m, and the horizontal deformation is 5.9 mm/m, approaching the critical value of 6.0 mm/m, suggesting that the mine design is feasible for safety mining.

1. Introduction

Due to shortage of mineral resources, exploitation of mines under rivers, buildings, and railways (commonly called under-three-objects) is significantly important for China's economy. At present, much effort has been made on the environmental protection in China, thus hazards associated with exploitation of underground mines, including large surface subsidence and groundwater pollution, should be avoided. Therefore, the stage subsequent filling mining (SSFM) method, which can effectively reduce the dilution rate of ore and protect groundwater, is commonly used across the world. In this regard, many scholars made contributions to investigating the optimization of cement-tailing ratios and stope structure, as well as the stability of backfill. For example, Liu and Zheng (2007) studied the application of sublevel open stope with delayed backfill mining method at the Tongshan copper mine using field tests, which offer the basis for the improvement of stope structure and mining technology. Xue (2007) introduced the application of high-level and large-diameter delayed backfill method at the Anqing copper mine. Xiao and Yao (2006) applied the deep-drill and sublevel stope backfill mining method to the Ashele copper mine. Gao (2002) experimentally investigated the application of the sublevel stope delayed backfilling method at the Tonglvshan copper mine. Xiao (2005) conducted a physical modeling of high-level large-diameter long-hole mining at the Anqing copper mine to optimize the mining sequence. Huan (1994), Zhou (2000), and Su (2002) adopted the nonlinear programming, the genetic algorithm, and the neural network to optimize the structure of stope, respectively. Liu et al. (2006) studied the reliability of high-level backfill based on chaotic optimization. Zou and Wei (2005) investigated the factors affecting the stability of stope and proposed technical measures for mining activities. Chen et al. (1984) analyzed the stability of stope at depth of 1200 m using nonlinear finite element analysis. Tesarik et al. (2009) analyzed the long-term stability of a backfilled room-and-pillar panel at the Buick mine, Missouri, USA. Besides, Zhou and Gu (2001) optimized the cement-tailing ratios of Anqing copper mine with genetic algorithm. Li and Liu (2004) investigated the mechanical behaviors of highly consolidated tailings backfill and optimized the cement-tailing ratios with the game tree method. Nanthananthan (2006) proposed an optimization design method of cement-tailing ratios by experiments in combination with

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numerical modeling. Especially, lots of work has been done related to stability of structures such as slopes/stope/backfill in mines through numerical modeling, e.g. FLAC (Vishal et al., 2010; Gupte et al., 2013; Singh et al., 2013; Pradhan et al., 2014).

However, the design of SSFM significantly differs in different conditions of overburden strata. In the region of Sijiaying iron mine, a large number of villages and farmlands can be observed on the ground surface. It belongs to "under-three-objects" ore body mining. In order to meet the demand for steel production, the SSFM method is used for this mine according to site-specific mining conditions. The sizes of the stope are 50 m $\times$ 25 m $\times$ 100 m, and two-step mining method for the room and pillar is used. Therefore, the stability of stope and effective control of the surface subsidence are the key issues related to the safe extraction of the mine. In this regard, the stability analysis of super-stope and the surface subsidence of Sijiaying iron mine are necessary to verify the reliability of mining scheme, ensure the safe exploitation of mine, and avoid farmland destruction and villages relocation.

2. Geological and mining conditions

As shown in Fig. 1, the Sijiaying iron mine is located at the southern Luanxian County, 55 km away from Tangshan, Hebei Province, China. The Sijiaying iron mine is owned and operated by the Tangshan Iron and Steel Group. The strike of the ore body is in the direction of NS, the dip direction is in EW, and the average dip angle is 46°. The ore body is composed of biotite granulite and some mica quartz schist above the −540 m level. The fresh rock is distributed on the levels from −540 m to −220 m with good mechanical properties; while the slightly weathered rock is distributed on the levels from −220 m to −150 m with slightly developed jointed fractures. In addition, the intensely weathered rock mass can be observed on the levels from −150 m to −90 m. Also the overburden of Quaternary soil is distributed on the levels from −150 m to −90 m.

Two-step sublevel delayed backfill mining method was used for this mine. The panel was arranged along the strike of the ore body with a length of 120 m. The length and width of the room and pillar are 50 m and 25 m, respectively; and the width of the interval pillar between panels is 20 m. The primary room was first extracted and backfilled by cemented tailings. The cement-tailing ratio was 1:8. Then, the secondary pillar was mined out and backfilled with non-cemented tailings. To avoid collapse of the farmlands and buildings on the surface due to mining activity, the level of −450 m should be first mined out.

3. Self-stability of the cemented backfilling body

In general, when the exposed area of backfilled stope is continuously increased, a typical failure mode of shear sliding can be observed, whereby a plane failure surface occurs that dips outwards in excavation (open stope), causing part of the exposed filling body to move to the new stope. When the height of backfilling body is sufficiently large, a filling block with sufficient weight will be formulated to combat any resistance to sliding along a critical failure plane orientated. The resistance to shear sliding results from a combination of cohesion and particle fraction both along the failure surface and at the side walls, and the driving force at failure is the net weight of the block.

It can be seen in Fig. 2 that the front side of the backfilling body is fully exposed, the back side is non-cemented, and the left and right sides are surrounded by wall rocks. Assuming that L_c is the strike length of the backfilled stope, B_c is the width, and H_c is the total height of the exposed backfilling body, the horizontal stress F_1 induced by the non-cemented tailings can be obtained:

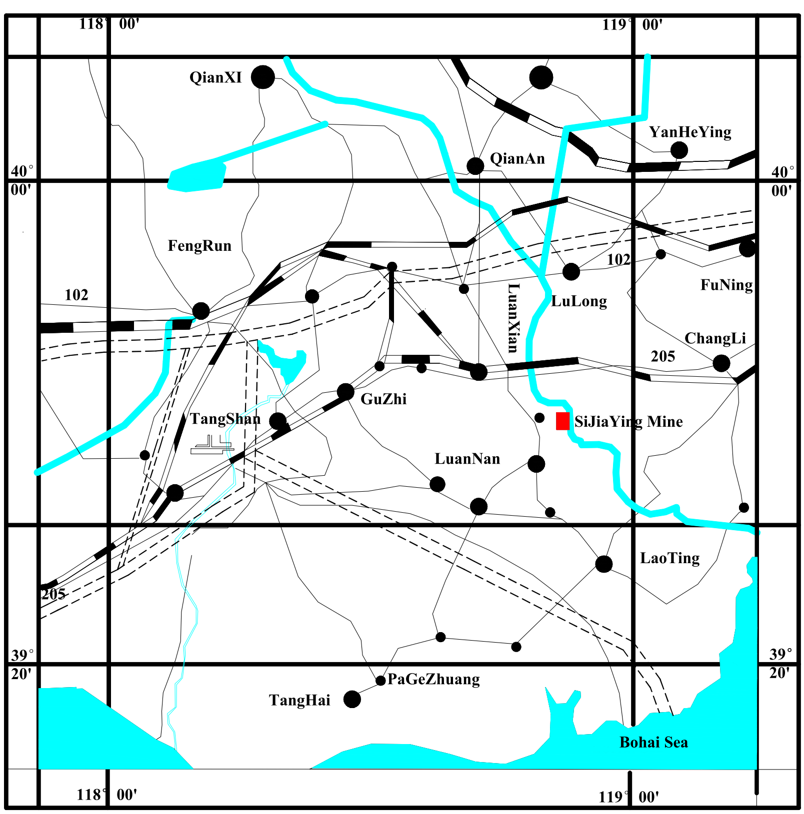

Fig. 1. Location of Sijiaying iron mine.

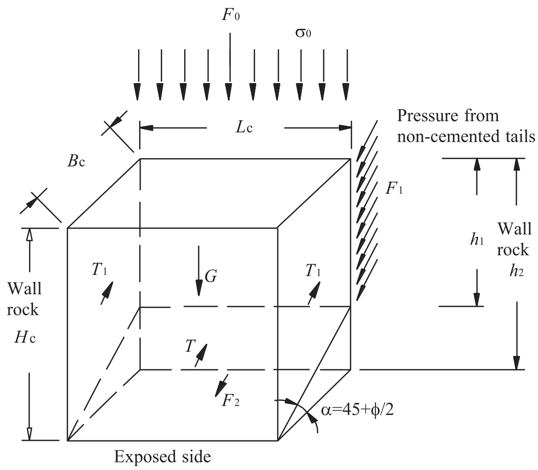

Fig. 2. Mechanical model of cemented backfill.

$$F_1 = \frac{1}{2}\lambda_c\gamma_2 h_1^2 L_c \tag{1}$$

$$h_1 = H_c - B_c \tan\alpha \tag{2}$$

where λ_c is the coefficient of horizontal pressure; γ_2 is the unit weight of the non-cemented backfill; h_1 is the upper height of sliding block, which can be calculated by Eq. (2); and α is the angle of the critical failure plane.

The weight of the backfilling block is

$$G = \gamma_c g h_2 B_c L_c \tag{3}$$

where γ_c is the unit weight of the cemented backfilling body (when more than one cement-tailing ratios are concerned, γ_c is the weighted mean value), h_2 is the average height of the sliding block, which can be calculated by the following equation:

$$h_2 = H_c - \frac{B_c \tan\alpha}{2} \tag{4}$$

The overlying load F_0 is

$$F_0 = B_c L_c \sigma_0 \tag{5}$$

The total driving force on the failure plane is

$$F_2 = (G + F_0)\sin\alpha \tag{6}$$

Assuming that the vertical average stress on the failure plane is σ, the resisting force T_1 obtained from the surrounding wall rock is

$$T_1 = B_c h_2\left[c' + \frac{1}{2}\lambda_c(\sigma+\sigma_0)\tan\phi'\right] \tag{7}$$

where c', ϕ' are the cohesion and internal friction angle of the hanging wall and/or footwall, respectively. If the backfilling material has merely one ratio, then we can consider $c' = c_c$ and $\phi' = \phi_c$; if more ratios are concerned, they can be considered to be the weighted values.

The resistant force induced by the cemented backfill, T, is

$$T = \left[c'' + \lambda_c(\sigma\cos\alpha\tan\phi'')\right]\frac{B_c L_c}{\cos\alpha} \tag{8}$$

where c'' and ϕ'' are the cohesion and internal friction angle of cemented backfilling body, respectively. If the backfilling body has merely one ratio, then we have $c'' = c_c$ and $\phi'' = \phi_c$; if more ratios are concerned, the weighted mean values are chosen.

If the backfilling body does not fail, the following condition must be satisfied:

$$F_2 + F_1\cos\alpha < T + 2T_1 \tag{9}$$

By solving Eq. (9), the following expression can be derived:

$$\begin{aligned}&\left[2B_cL_c(\gamma_c g h_2+\sigma_0)\sin\alpha + \lambda_c L_c\gamma_2 h_1^2\cos\alpha\right.\\ &\left.-2B_c h_2(2c'+\lambda_c\sigma_0\tan\phi') - \frac{2B_cL_cc''}{\cos\alpha}\right]\Big/\\ &(2B_ch_2\lambda_c\tan\phi'' + 2B_cL_c\lambda_c\tan\phi'') < \sigma\end{aligned} \tag{10}$$

After substituting the related mechanical parameters listed in Table 1 in Eq. (10), the relation curve between the required strength of backfilling body and the exposed height is obtained and shown in Fig. 3, where the length of the ore room is 50 m, and the width is 25 m. It can be seen from Fig. 3 that the required strength of backfilling body increases with the increase of the exposed height. In the earlier stage of the curve, the increasing trend is evident, and the required strength of backfilling body does not increase linearly with the increase of height because the arch function of surrounding rock is increased, thus reducing the vertical stress at the bottom of backfill. When the exposed height reaches 100 m, the required strength is 0.9 MPa. Therefore, the mixture ratio of the cemented backfilling body can meet the strength requirement of the stope.

4. Stability analysis based on physical modeling

Based on the site-specific geological and mining conditions, the model size used in this physical modeling is 2.5 m × 0.3 m × 2.8 m. In this model, the geometric similarity ratio is 1:170, the bulk density similarity ratio is 0.6:1, and the stress similarity ratio is

Table 1
Physico-mechanical parameters used in numerical model.

Type	Density (g/cm^3)	Compression strength (MPa)	Tensile strength (MPa)	Cohesion (MPa)	Internal friction angle (°)	Elastic modulus (GPa)	Poisson's ratio
Ore body	3.30	35	1.6	4.50	70	55	0.20
Deep fresh rock (from −540 m to −220 m)	2.80	25	1.2	2.80	55	36	0.22
Weakly weathered rock (from −220 m to −150 m)	2.75	10	0.9	1.10	45	13	0.25
Intensely weathered rock group (from −150 m to −90 m)	2.60	5	0.5	0.45	33	10	0.30
Quaternary system (from −90 m to 20 m)	1.95	1	0	0.10	27	1.5	0.35
Cemented backfill	1.98	8.50	1.01	1.16	48	1.13	0.16
Non-cemented backfill	0.025	0.3	1.410	0.03	0.0	0.004	0.26

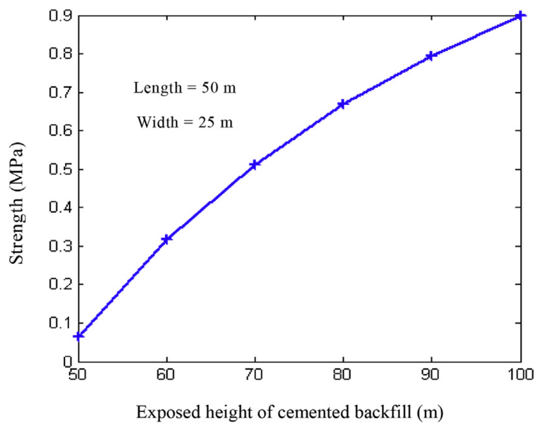

Fig. 3. Required strength with different exposed heights.

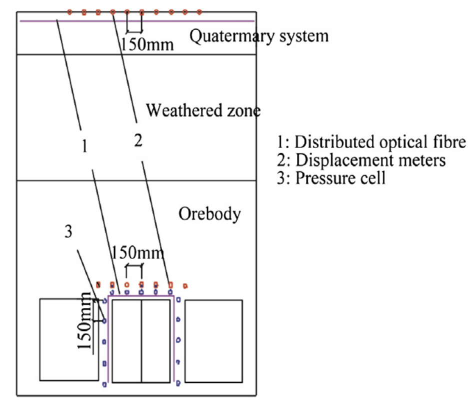

Fig. 5. Layout of monitoring equipments.

1:285. Sand is used as the main aggregate, and gypsum and calcium carbonate are considered as the cemented materials. The weight-type model test was used, where the hierarchical compaction method was adopted to construct the model. The thickness of each level is 2–3 cm. First the layers were compacted upward layer by layer. Then suitable amount of mica was sprinkled on the top of the compacted layer, and mortar was fed to the next layer. The compaction step was repeated until the completion of the whole model (see Fig. 4). Besides, the surface displacement of the model was monitored using the distributed optic fiber sensors and displacement meters. The displacements of the internal rock and pillar were monitored by optic fiber sensors, and the change of stress was recorded by pressure sensors. There were 17 displacement meters and 15 pressure boxes installed in the model in total, and their locations are shown in Fig. 5.

4.1. Simulation processes

The following cutting and filling processes were simulated in the test:

(1) Extract ore to form the primary room and measure the stress and displacement.
(2) Backfill the primary room with cemented tailings and record the stress and displacement.
(3) Mine out the pillar and record the magnitude of stress and displacement.
(4) Backfill the secondary void where the pillar is excavated with non-cemented tailings, then monitor the stress and displacement.

The mining processes mentioned above are shown in Figs. 6–9. It can be observed from these figures that after the extraction of the primary room, the model was in a good condition and no cracks or failures were developed. But when the secondary pillar was mined, it can be noted that some local failures were developed on the top of the exposed sides. Through backfilling by non-cemented materials, the loose tailings were caved immediately after the removal of baffle.

4.2. Results of physical modeling

(1) When the primary room is formed, the displacements of the roof and ground surface are smaller due to the timely backfilling and the supporting effect of the pillar. The maximum displacement of the roof is 0.21 mm for the model and about

Fig. 4. The test model.

Fig. 6. Extraction of ore to form primary room.

Fig. 7. Backfilling the primary room with cemented material.

35.7 mm for the prototype (Fig. 10); the maximum surface settlements for the model and the prototype are approximately 0.012 mm and 2.3 mm, respectively (Fig. 11); and the maximum deformation of the interval pillar on the top (Fig. 12) are 0.09 mm and about 15.4 mm for the model and the prototype, respectively. As for the change of stress, the stress in the roof of the primary room increases rapidly (Fig. 13), suggesting that the stress is transferred to the roof after the excavation of the orebody room.

Fig. 8. Extracting ore to form secondary room.

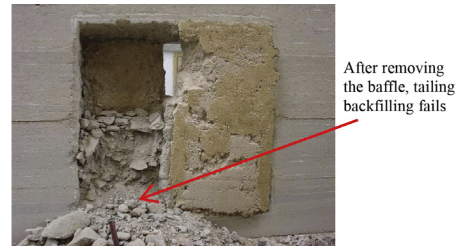

Fig. 9. Backfilling the secondary room with non-cemented materials.

(2) After the secondary pillar was mined, the cemented backfill was roughly stable. Due to the vibration induced by mining, a small amount of backfilling on the top fell down, but there was not a large area running off. After the secondary room was backfilled using the non-cemented tailings, the tailings backfilling failed when the baffle was removed. The main reason is that the tailings contain no cementation. In this regard, the roof displacement and the surface subsidence as well as the horizontal deformation of the interval pillar increase to some degree. The ultimate deformation of the roof was 0.42 mm, about 71.4 mm in the prototype; the maximum surface settlement was 0.03 mm, corresponding to a value of about 5.1 mm in the prototype; the lateral deformation of interval pillar reached 0.09 mm, which was about 15.4 mm for the prototype. Considering the changes in the stress field, the stresses in the roof and the interval pillar (the position of the second monitoring point is the origin of coordinate, so the height is negative at the bottom of pillar) increase (Fig. 14), but are less than the ultimate compressive strength.

(3) The designed cement-tailing ratio is feasible. When mining the secondary pillar, the cemented backfilling with the height of 100 m shows a good self-stability.

(4) Because the distributed fiber optic sensors are not subjected to electromagnetic interference and are almost insensitive to corrosion, more accurate and satisfactory monitoring data were obtained in comparison of those recorded by the displacement meters.

5. Three-dimensional numerical modeling and analysis

In order to investigate the influence of different mining stages on the surface subsidence, a three-dimensional numerical model was

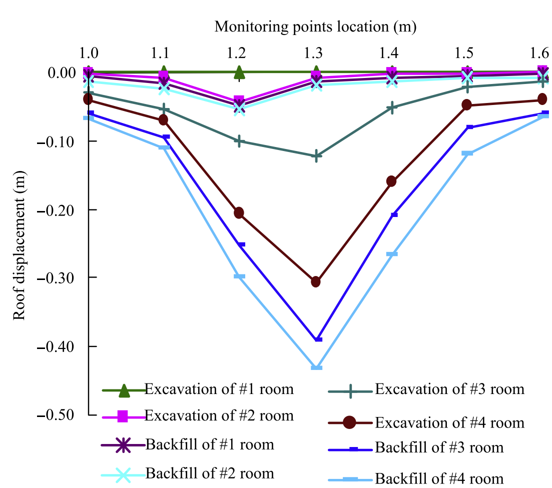

Fig. 10. Change of roof displacement with mining sequence.

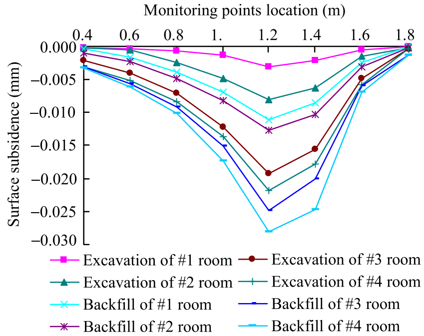

Fig. 11. Change of surface subsidence with mining sequence.

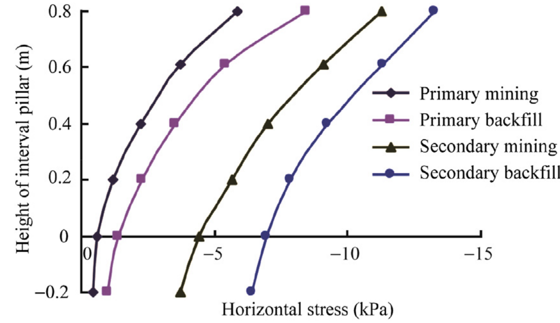

Fig. 14. Change of horizontal stress of interval pillar with mining sequence.

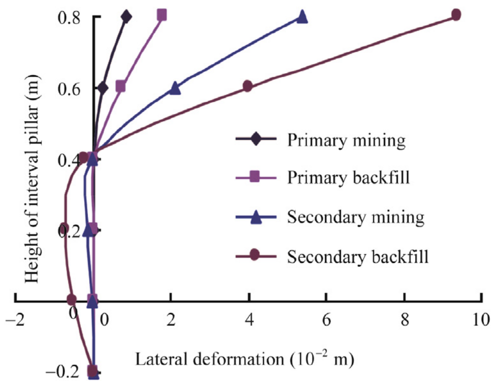

Fig. 12. Change of lateral deformation of interval pillar with mining sequence.

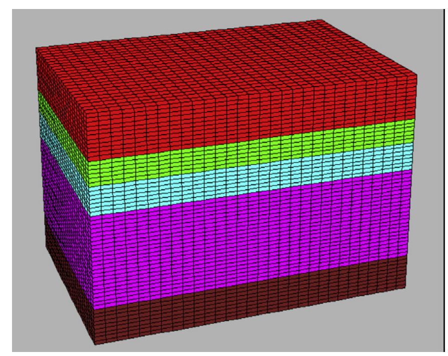

Fig. 15. Three-dimensional numerical model.

established by means of FLAC^{3D} according to the mining design. The size of the model is 800 m (x-axis: EW) × 600 m (y-axis: NS) × 540 m (z-axis: vertical direction) in terms of length × width × height. The calculation model is shown in Figs. 15 and 16, with 94,928 zones and 113,254 grids. The surrounding rocks and backfills are assumed to conform to the Mohr-Coulomb yield criterion, and the employed physico–mechanical parameters are listed in Table 1.

The variations of surface subsidence with mining steps are shown in Fig. 17. It can be seen from Fig. 17 that the maximum subsidence point moves to the right as the mining activity continues. The magnitudes of surface subsidence during mining in the first and second stages are 1.8 mm and 6.1 mm, respectively. However, in the third stage, the surface subsidence changes abruptly to a value of 1 m, especially when the secondary room is mined. After backfilled with non-cemented tailings, the surface subsidence reaches 1.2 m.

At the same time, according to the incline, curvature as well as horizontal deformation shown in Fig. 18, it can be noted that the curvature value is smaller than its allowable value, and the incline and horizontal deformation approach to their critical values. Therefore, it is concluded that the structure parameters of the stope in association with the mechanical parameters of the backfill can ensure the deformation of ground surface to a controllable range. Besides, the movement of ground surface due to mining activity will not cause damage to the villagers above the mine surface.

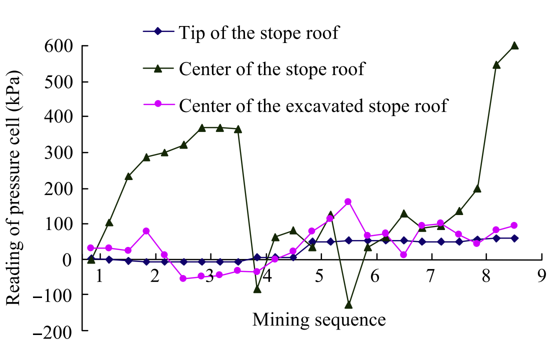

Fig. 13. Change of roof stress with mining sequence.

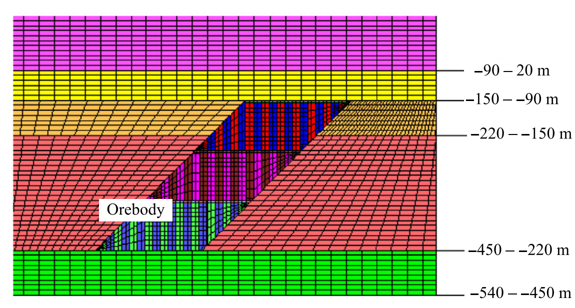

Fig. 16. Distribution of wall rock and ore body.

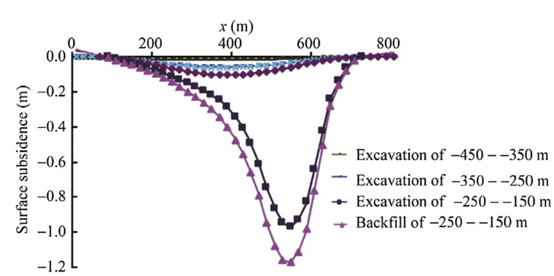

Fig. 17. Surface subsidence change with mining steps.

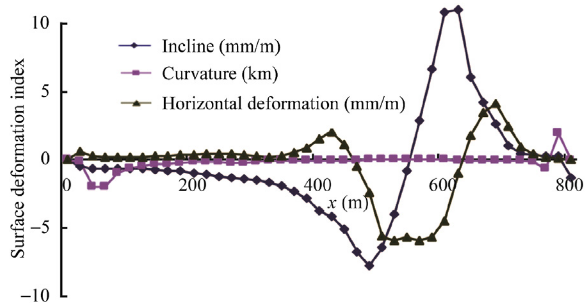

Fig. 18. Surface incline, curvature and horizontal deformation caused by mining.

6. Conclusions

Through theoretical analysis, physical modeling and numerical simulations, the stope stability and the surface subsidence of the Sijiaying iron mine were analyzed in detail, and the main conclusions are drawn as follows:

(1) According to the results of self-stability test on cemented backfill, the backfilling block can basically remain stable even when the exposed height reaches 100 m where the cement to sand ratio of 1:8 is adopted.
(2) According to the results of physical modeling, it is noted that during the mining of primary room at the −450 m level, the stope is stable. However, after the secondary pillar was mined out, pieces of backfilling body fell down. The displacements of the stope and ground surface were small, although lateral flexural deformation occurred in the interval pillar, and the maximum deformation of 15.4 mm can be recorded at the top of the interval pillar. The maximum vertical stress of the roof is 15 MPa and the horizontal stress is 3.5 MPa, which do not exceed the ultimate compressive strength of the pillar. Therefore, the stope is stable and the ground surface can be safely controlled.
(3) According to the results of the three-dimensional numerical modeling, the curvature value is smaller than its critical value. The maximum magnitude of the incline is 10.99 mm/m, a little lager than the permissible value of 10 mm/m; and the horizontal deformation is 5.9 mm/m, approaching to the critical value of 6.0 mm/m. The calculation shows that the stope can be controlled with the surface subsidence within its allowable threshold, and the surface subsidence caused by mining activity will not exert major influence on the safety of the villagers and local residence.
(4) Among the four steps of mining and backfilling process, the stability of stope during the third step is the worst, so it is necessary to pay more attention to the monitoring of stope stability to assure the stability of stope and safety of production.
(5) During the third-step mining of pillar, the self-stability of the second-step cemented backfills, as the exposed area increases, will reduce. As for the stope with height of 100 m, only when the design strength of cemented backfills is greater than 1.0 MPa, the stability of backfill could be maintained.

In general, theoretical analysis, physical model as well as numerical modeling are utilized in this paper to study the stability of stope and backfills. Advantages and disadvantages of each method are summarized. Theoretical analysis is based on the empirical and semi-empirical method to analyze and evaluate the stability of cementing backfills. The physical modeling is only used for the stress-strain simulation during the course of mining and backfilling at the level of −450 m, but the stress of multi-staged stope is not available. The numerical simulation can analyze the stability of stope and stress distribution due to mining, but the simulation results rely on the reliability and accuracy of rock mass parameters. Therefore, combination of the three methods used to perform comprehensive analysis can not only increase the reliability of stability analysis, but also offer important basis for design optimization.

Conflict of interest

The authors wish to confirm that there are no known conflicts of interest associated with this publication and there has been no significant financial support for this work that could have influenced its outcome.

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