

Article

Study on the Stability of Mining Walls During the Recovery of Flank Pillars Adjacent to Massive Backfill

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Abstract

To address the difficulty of determining the safe reserved thickness of the mining wall in the test block of the panel pillar at Tongkeng Mine, The stress of mining wall is comprehensively analyzed. Combined with numerical simulation method and field monitoring, the optimal wall thickness is determined. By differentiating each stress component, the mathematical equations governing the locations where extreme values of the stress components occur are derived, and the mathematical expressions for the extreme value positions of each stress component are further determined accordingly. Considering the geological characteristics and mining conditions of the experimental stope with panel pillars, the eastern mining wall of the test block is selected as the research object. A mining wall thickness range of 3 m to 8 m is designed, and the optimal safe reserved thickness of the mining wall is determined through numerical simulation. Based on the optimal mining wall retention thickness, stopping operations are carried out on the orebody of the experimental stope. Meanwhile, monitoring points are reasonably arranged from the upper-middle section to the middle of the mining wall, and real-time monitoring is performed on the stress variation data at each monitoring point during the entire stopping process of the test block. Theoretical analysis results show that the exact locations of the extreme values of each stress component can be accurately determined within the two-dimensional plane of the mining wall, among which the extreme value of the horizontal stress component appears at the midpoint of the mining wall thickness. Numerical simulation results indicate that both the stress and displacement of the mining wall exhibit a gradual decreasing trend with an increase in mining wall thickness. However, when the mining wall thickness exceeds 5 m, the reduction rate of stress and displacement slows down significantly, and the mining wall tends to become stable. Maintaining a mining wall thickness of 5 m in the experimental stope can generally ensure the safe recovery of the orebody. However, pronounced stress concentrations occur at the geometric corners of the mining wall, which result from stress retention caused by changes in the mining wall geometry. Meanwhile, the stress concentration in the mining wall is synchronized with that in the drilling galleries of the experimental stope, and varying degrees of failure occur in the drilling galleries at locations where stress concentration appears in the mining wall. Monitoring results show that the maximum stress borne by the drilling gallery is approximately 26 MPa, beyond which rock mass collapse and fragmentation are prone to occur.



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Keywords: plate area pillar; stability analysis; test ore block; monitoring of mining wall stress; stress concentration

1. Introduction

In recent years, the rapid development of the country has resulted in a significant increase in the demand for mineral resources [1]. Long-term, large-scale mining has considerably depleted shallow-level resources, making deep-level resource extraction an essential strategy for ensuring China's energy security [2,3]. The ore pillar, which serves as a critical structure for supporting the goaf and maintaining the stability of surrounding rock in mines [4], possesses not only inherent stability but also substantial resource volume and an extremely low recovery rate [5–7]. As production shifts from shallow to deep mining, ore pillar resources will encounter various risks, including increased deformation, a highly complex stress environment, and greater fragmentation of the rock mass [8–11]. These factors will inevitably complicate the recovery of ore pillar resources [12–14]. Therefore, while ensuring production safety, effectively recovering a portion of the ore pillar resources and maximizing the utilization of mineral resources remains one of the primary challenges currently faced by mines [15–18].

Many researchers worldwide have investigated stable recovery of ore pillars through theoretical, empirical, and simulation studies [19,20]. Poulsen et al. [21] proposed a method to calculate stress on ore pillars during mining that accounts for the interaction between pillar geometry and spatial position; this method applies to planar solids. Cao et al. [22] developed a mechanical model of the surrounding rock based on the movement law of the overlying strata, used it to design pillar widths, and derived formulas for the elastic and plastic zone widths within a pillar. Zhou et al. [23] established an evaluation and mechanical model for the dynamic removal of adjacent ore pillars and used structural dynamics to analyze pillar recovery time and disturbance. Chen et al. [24] derived the structural expression governing ore pillar stability by examining pillar instability mechanisms. To improve resource utilization, practitioners commonly retain a thin mining wall during pillar recovery to support backfill and surrounding rock masses [25–27] and thus prevent backfill collapse during recovery [28]. Huang [29] determined the critical yield stress of the mining wall by combining a mechanical model of ore-wall failure with an energy-variation method, and established the relationship between ore-wall stress and spatial dimensions. Wang [30] derived the roof deflection equation from a beam-column composite model and proposed formulas for the safety factor and pillar stiffness ratio. Wang et al. [31] used catastrophe theory to build a mechanical model of the mining wall, formulated the cusp catastrophe equation, and derived an expression for the ore-wall ultimate thickness. Xie Xuebin et al. [32] developed an ore-wall mechanical model based on elastic thin-plate theory and examined how geometric parameters vary according to the mining wall's load-bearing mechanism. Shi [33] used FLAC numerical simulation to analyze the optimal strip-pillar dimensions. Vyazmensky [34] investigated how different structural planes affect pillar stability by coupling finite-element and discrete-element methods.

In summary, to ensure safe and efficient extraction of pillar resources and to maximize resource recovery, researchers worldwide have investigated multiple stability-related aspects. Most studies have concentrated on stress calculations, mechanical modeling, and optimal dimensional analysis of ore pillars and walls. They have not examined the synchronous relationship between mine-wall stress and roadway failure. Few have considered the dynamic evolution of stress over the entire stope-mining process, nor have they established a direct link between mine-wall stress distribution and the safe thickness of the

mining wall. This paper examines the occurrence of pillar resources in Orebody No. 92 of Tongkeng Mine and evaluates the ore-wall retention scheme for the first-mined area of the panel pillars. We analyze the stress distribution and mechanical behavior of the mining wall and identify the locations of extreme values. Based on numerical simulation results, we examine stress and displacement trends and characteristic values at key points under different retention schemes to identify the optimal ore-wall scheme. Finally, we combine monitoring methods used during ore-wall extraction with in situ stress measurements from the test ore block, providing data to support large-scale recovery of panel pillar resources in the mining area and offering a reference for analogous mines.

2. Project Overview

The large-scale mining of the No. 92 ore body at Tongkeng Mine has continued for nearly 20 years, producing over 23 million tons of ore. In the early phase, caving-induced stress concentrations were severe, which caused most panel pillars to become fragmented and difficult to extract. The No. 92 ore body is divided into seven panels, as shown in Figure 1. Panel widths range from 90 to 100 m, and panel pillars follow north–south exploration lines with a width of 20 m. Complex geological engineering conditions and overlying rock caving produce pronounced ground pressure, which reduces recovery rates for intermediate pillars, increases mining costs, and raises safety risks. These factors complicate the safe recovery of the mine’s pillar-group resources, and identifying safe, efficient recovery methods remains a major challenge for the enterprise. To address this issue, the east side of stope T204 and T205, located within the 204-panel pillar, was selected as the experimental stope for research. Studying this experimental stope enabled investigation of the panel pillars’ stress-field distribution and determination of a reasonable ore-wall thickness to be retained, which provide foundational data for large-scale panel-pillar recovery.

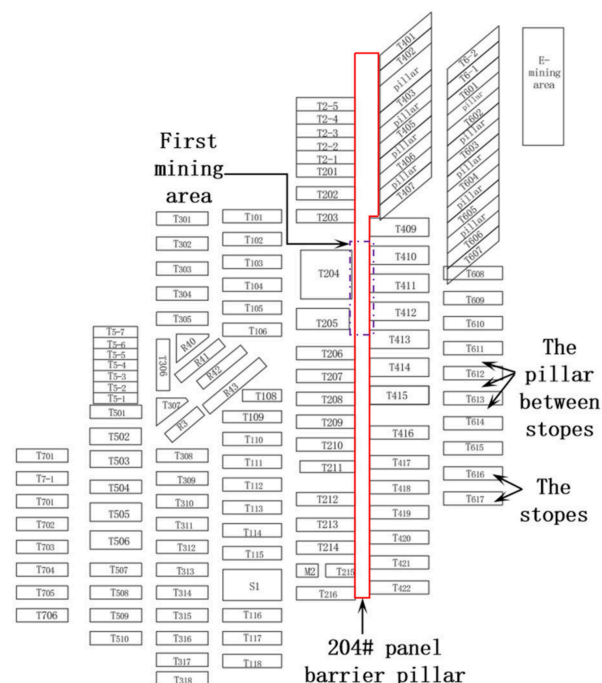


Figure 1. Schematic diagram of ore block layout.

3. Mechanical Model of Mining Wall

3.1. Establishment of Mechanical Model

As a representative thin ore pillar, the mining wall is modeled as an elastic body that is continuous, homogeneous, isotropic, and intact. Its surface is assumed flat and its thickness constant. Because the wall's deformations and displacements are much smaller than its cross-sectional dimensions, the small-deformation assumption of elastic mechanics holds [35]. Accordingly, the stress problem is reduced to a plane elastic problem, and the material is taken as continuous, homogeneous, and isotropic. The mechanical model of the mining wall is shown in Figure 2. The wall height is h , thickness is b , γ denotes the unit weight of the mining wall, γ_1 denotes the unit weight of the filling body, q denotes the initial lateral load, P_0 denotes the initial vertical load, and λ denotes the lateral pressure coefficient. The cross section is rectangular, and the semi-inverse method is used to solve the stress state of the mining wall.

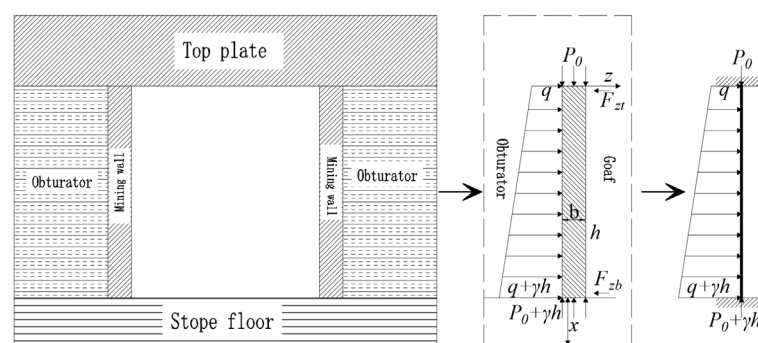


Figure 2. Stress analysis of mining wall.

3.2. Solution of Mechanical Model

$$\sigma_z = \frac{\sigma^2 \Phi}{\sigma^2 x} = x f(x) + a f(x) \quad (1)$$

$$\Phi = \frac{x^3}{6} f(z) + \frac{ax^2}{2} f(z) + x f_1(z) + f_2(z) \quad (2)$$

$$\frac{\partial^4 \Phi}{\partial x^4} + 2 \frac{\partial^4 \Phi}{\partial x^2 \partial z^2} + \frac{\partial^4 \Phi}{\partial z^4} = 0 \quad (3)$$

Substituting Formula (2) into Formula (3) and integrating them, the function expressions of $f(z)$, $f_1(z)$, and $f_2(z)$ can be obtained:

$$\begin{aligned} f(z) &= Az^3 + Bz^2 + Cz + D \\ f_1(z) &= -\frac{1}{10}Az^5 - \frac{1}{6}Bz^4 + Ez^3 + Fz^2 + Gz \\ f_2(z) &= -\frac{1}{10}Az^5 - \frac{1}{6}Bz^4 + Hz^3 + Iz^2 \end{aligned} \quad (4)$$

Further integration can yield the expression of the stress function:

$$\begin{aligned} \Phi &= \left(\frac{x^3}{6} + \frac{ax^2}{x} \right) (Az^3 + Bz^2 + Cz + D) \\ &+ x \left(-\frac{1}{10}Az^5 - \frac{1}{6}Bz^4 + Ez^3 + Fz^2 + Gz \right) + \left(-\frac{1}{10}Az^5 - \frac{1}{6}Bz^4 + Hz^3 + Iz^2 \right), \end{aligned} \quad (5)$$

Under the condition that only the self-weight stress of the mining wall is considered, the stress components in each direction can be obtained as follows [36]:

$$\begin{cases} \sigma_x = (\frac{x^3}{6} + \frac{ax^2}{2})(6Az + 2B) + x(-2Az^3 - 2Bz^2 + 6Ez + 2F) + (-2aAz^3 - 2aBz^2 + 6Hz + 2I) - \gamma x \\ \sigma_z = (x + a)Az^3 + Bz^2 + Cz + D \\ \tau_{xz} = \frac{1}{2}Az^4 + \frac{2}{3}Bz^3 - 3Ez^2 - 2Fz - G - (\frac{x^2}{2} + ax)(3Az^2 + 2Bz + C) \end{cases} \quad (6)$$

In the formula, σ_z represents the horizontal stress applied to the ore pillar, σ_x denotes the vertical stress applied to the ore pillar, and τ_{xz} indicates the shear stress on the mining wall. The variables $a, A, B, C, D, E, F, G, H$, and I are the undetermined coefficients of the equation.

$$\begin{cases} a = \frac{q}{2\lambda\gamma_1} + 3 \\ A = \frac{8\lambda\gamma_1}{b^3} \\ B = 0 \\ C = -\frac{6\lambda\gamma_1}{b} \\ D = -\frac{q}{2} \\ E = \frac{2\lambda\gamma_1}{5b} - \frac{3qh}{4b^2} - \frac{\lambda\gamma_1 h^2}{5b^2} \\ F = 0 \\ G = -\frac{\lambda\gamma_1 b}{20} + \frac{9qh}{16} + \frac{3\lambda\gamma_1 h^2}{20} \\ H = 0 \\ I = -\frac{P_0}{2} \end{cases} \quad (7)$$

During the mining of underground ore bodies, factors such as inadequate roof stability and local exposure development necessitate the simplification of the mine wall into a force model characterized by a fixed support at the base and a horizontal constraint at the top. Utilizing principles of structural mechanics, the supporting reaction forces at both the top and bottom of the mine wall are represented in the accompanying formula.

$$\begin{cases} F_{zt} = \frac{3}{8}qh + \frac{1}{10}\lambda\gamma h^2 \\ F_{zb} = \frac{5}{8}qh + \frac{1}{10}\lambda\gamma h^2 \end{cases} \quad (8)$$

On the side adjacent to the backfill body:

$$\sigma_{z|z=0} = -q - \lambda\gamma x \quad (9)$$

On the side adjacent to the goaf, both the horizontal stress and shear stress are zero.

$$\begin{aligned} \sigma_{z|z=b} &= 0 \\ \tau_{z|z=b} &= 0 \end{aligned} \quad (10)$$

At the top of the rib wall, the vertical stress and shear stress are given by:

$$\begin{aligned} \int_0^b \sigma_{x|x=0} dz &= -P_0 b \\ \int_0^b \tau_{xz|x=0} dz &= -\frac{3}{8}qh - \frac{1}{10}\lambda\gamma_1 h^2 \end{aligned} \quad (11)$$

At the bottom of the rib wall, the vertical stress and shear stress are given by:

$$\begin{aligned}\int_0^b \sigma_x|_{x=h} dz &= -P_0 b - \gamma h b \\ \int_0^b \tau_{xz}|_{x=h} dz &= -\frac{5}{8} q h - \frac{2}{5} \lambda \gamma_1 h^2\end{aligned}\quad (12)$$

Analysis of the mine-wall stress model shows that the horizontal load can be represented as a trapezoidal distribution, which is the sum of a rectangular and a triangular load, while the vertical load can be represented as a uniform distribution. Using the displacement method from structural mechanics together with the boundary stress conditions, the unknown coefficients are determined. The resulting expressions for the stresses are:

$$\begin{cases} \sigma_x = \left(-\frac{2}{b^2} x^3 z + \frac{4}{b^2} x z^3 - \frac{6h^2}{5b^2} x z - \frac{3}{5} x z \right) \frac{\lambda \gamma_1}{b} \\ \quad - \left(6x^2 z + \frac{9h}{2} x z - 4z^3 \right) \frac{q}{b^3} - \lambda x - p_0 \\ \sigma_y = \left(x + \frac{q}{\lambda \gamma_1} \right) \left(-\frac{2\lambda \gamma_1}{b} z^3 + \frac{3\lambda \gamma_1}{2b} z - \frac{\lambda \gamma_1}{2b} \right) \\ \tau_{xy} = \left(-\frac{z^4}{b^2} + \frac{h^2 z^2}{5b^2} + \frac{z^2}{10} + \frac{3x^2 z^2}{b^2} - \frac{3x^2}{4} \right) \frac{\lambda \gamma_1}{b} \\ \quad - \left(\frac{9hz^2}{4b^2} + \frac{6xz^2}{b^2} - \frac{3x}{2} \right) \frac{q}{b} + \left(\frac{3}{80} b - \frac{3h^2}{20b} \right) \lambda \gamma_1 - \frac{9qh}{16b} \end{cases} \quad (13)$$

In the above expressions for each stress component, only x and z are independent variables, and each stress component is a polynomial in x and z . By holding x or z fixed and taking the first-order derivative with respect to z or x , respectively, one obtains the locations of extreme values for each stress component at a given height or thickness on the mining wall section. The stress expressions after taking the first-order derivative of each component are as follows:

$$\begin{cases} \frac{d}{dx}(\sigma_x) = -\frac{6x^2 z \lambda \gamma_1}{b^3} - \frac{12xzq}{b^3} \\ \frac{d}{dz}(\sigma_x) = \frac{\left(-\frac{2x^3}{b^2} + \frac{12xz^2}{b^2} - \frac{6h^2 x}{5b^2} - \frac{3x}{5} \right) \lambda \gamma_1}{b} - \frac{(6x^2 + \frac{9}{2} hx - 12z^2)q}{b^3} \\ \frac{d}{dx}(\sigma_z) = 0 \\ \frac{d}{dz}(\sigma_z) = -\left(x + \frac{q}{\lambda \gamma_1} \right) \left(-\frac{6\lambda \gamma_1 z^2}{b^3} + \frac{3\lambda \gamma_1}{2b} \right) \\ \frac{d}{dx}(\tau_{xz}) = -\frac{3x\lambda \gamma_1}{2b} + \frac{3q}{2b} \\ \frac{d}{dz}(\tau_{xz}) = \frac{\left(-\frac{4z^3}{b^2} + \frac{2h^2 z}{5b^2} + \frac{z}{5} + \frac{6x^2 z}{b^2} \right) \lambda \gamma_1}{b} - \frac{\left(\frac{9hz}{2b^2} + \frac{12xz}{b^2} \right) q}{b} \end{cases} \quad (14)$$

When z is a fixed value, location of the extreme value of σ_x stress:

$$\begin{cases} x = 0 \\ x = -\frac{2q}{\lambda \gamma_1} \end{cases} \quad (15)$$

Position of the σ_z stress extreme value: No extreme value, Location of the extreme value of τ_{xy} stress:

$$x = \frac{q}{\lambda \gamma_1} \quad (16)$$

When x is a fixed value:

Location of the extreme value of σ_x stress:

$$z = \pm \frac{\sqrt{30} \sqrt{(x\lambda \gamma_1 + q)x(6b^2 \lambda \gamma_1 + 12h^2 \lambda \gamma_1 + 20x^2 \lambda \gamma_1 + 45hb + 60xq)}}{60(x\lambda \gamma_1 + q)} \quad (17)$$

Position of the σ_z stress extreme value:

$$\begin{cases} x = 0 \\ x = -\frac{2q}{lg_1} \end{cases} \quad (18)$$

Location of the extreme value of τ_{xy} stress:

$$\begin{cases} z = 0 \\ z = \pm \frac{\sqrt{10}\sqrt{\lambda\gamma_1(2b^2\lambda\gamma_1+4h^2\lambda\gamma_1+60x^2\lambda\gamma_1-45hb-120xq)}}{20\lambda\gamma_1} \end{cases} \quad (19)$$

The analysis of the differentiated formula reveals that, when the thickness or height of a specific mining wall is established, the locations of the extreme values of each stress component can be determined using the known parameters of the surrounding rock. In designing the thickness of the mining wall, it is sufficient to ensure that the stress design value at the positions of extreme values remains below the allowable maximum stress, thereby ensuring the safety and stability of the mining wall.

4. Analysis of the Mining Wall Retention Scheme and Numerical Simulation Results

To address the technical challenges associated with existing pillar mining in the disk area and to verify the location of the extreme stress value, the site depicted in Figure 3 was designated as the initial mining area for research on pillar recovery and the safe reserve thickness of the mining wall. This first mining area is located at the 374 m to 386 m level of the pillar in block 204, positioned between the T204 and T205 stopes in a north–south orientation. Both the eastern and western sides of the pillar are composed of filled bodies, and the existing infrastructure is relatively intact. In this area, permanent mining walls are retained on both sides to support the filling bodies. Specifically, an 8 m permanent ore pillar is maintained on the western side, while the eastern side serves as the primary focus for the mining wall setting scheme. Through simulation analysis conducted within a range of 3 to 8 m, the optimal thickness for the mining wall on the eastern side has been determined.

4.1. Model Calibration

A calculation and analysis model for the entire mining area was constructed, and excavation simulation analyses were performed on the mined ore bodies to obtain surface settlement data. By comparing the surface settlement data derived from the model with the observed surface collapse pits and surface monitoring data, it was determined that the locations of the surface collapse pits generally fell within the 42 mm settlement deformation range. The surface settlement monitoring data and the values extracted from the numerical simulation at corresponding positions were largely consistent, exhibiting an error margin of less than 5%, as illustrated in Figure 4. This simulation also revealed that the self-weight stress at the 430-level ranged from 18 MPa to 20 MPa, which was subsequently utilized as the boundary condition for the local model analysis.

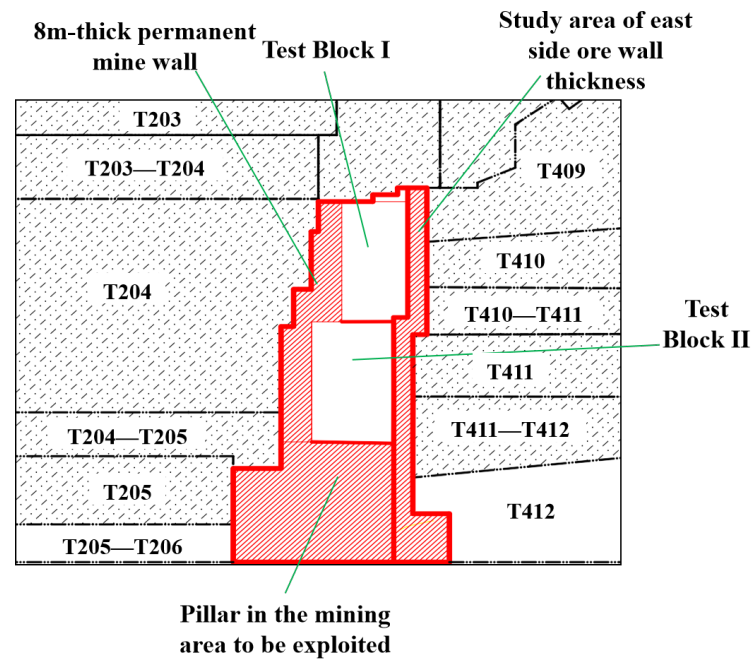


Figure 3. Design drawing of the first mining area.

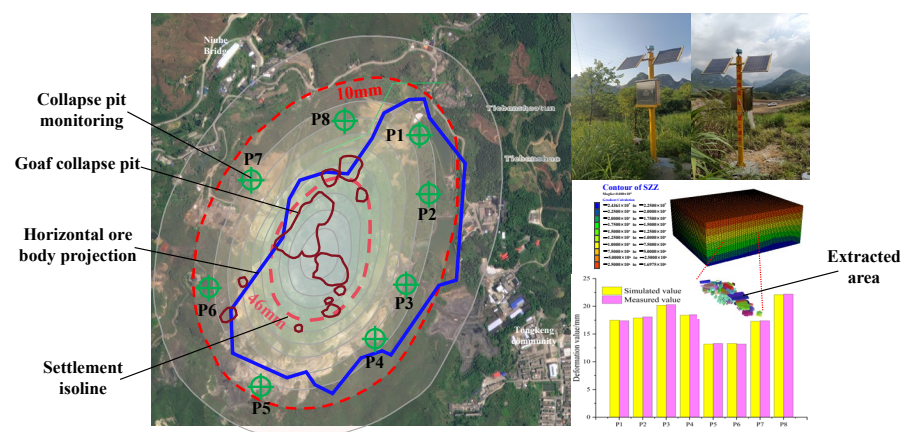


Figure 4. Surface subsidence monitoring and stress checking.

4.2. Numerical Simulation Scheme

To accurately simulate the stress changes in the ore body before and after mining, the numerical simulation analysis model adhered strictly to the design requirements and involved detailed modeling of the ore body, the mining wall, and the filling material, as illustrated in Figure 5. Additionally, survey lines were positioned on the east and west sides of the ore body, as depicted in Figure 6. The derivative results of the aforementioned stress components indicate that the midpoint of the mining wall thickness is where the maximum horizontal stress occurs. Consequently, the measurement line is positioned in the center of the pillar. The positions of measurement points A1 to A5 along the A measurement line on the west side remain constant. In contrast, the B1 to B5 points on the east side represent the monitoring locations designated in each scheme for the 3 to 8 m mining wall. These points are utilized to monitor changes in stress and displacement of the mining wall before and after the extraction of the ore body, as modeled in the numerical simulation. Specifically, Scheme One corresponds to a 3 m mining wall, Scheme Two to a 4 m mining wall, and so forth, with Scheme Six involving an 8 m mining wall. The conditions for the simulation calculations are detailed in Table 1. The model employs the Mohr-Coulomb constitutive model, with the elevation in the Z direction ranging from 430 m to 350 m. According to the

mine production schedule, the extraction period for Block I Test Mining Block is 20 days, and the backfilling period for Block I Test Mining Block is 22 days. Extraction of Block II Test Mining Block commences 28 days after completion of backfilling of Block I Test Mining Block, by which time the backfill has reached the design strength. The extraction period for Block II Test Mining Block is 20 days, and the backfilling period for Block II Test Mining Block is 20 days. Therefore, in the numerical simulation the design strength parameters of the backfill are used as the basis for parameter assignment.

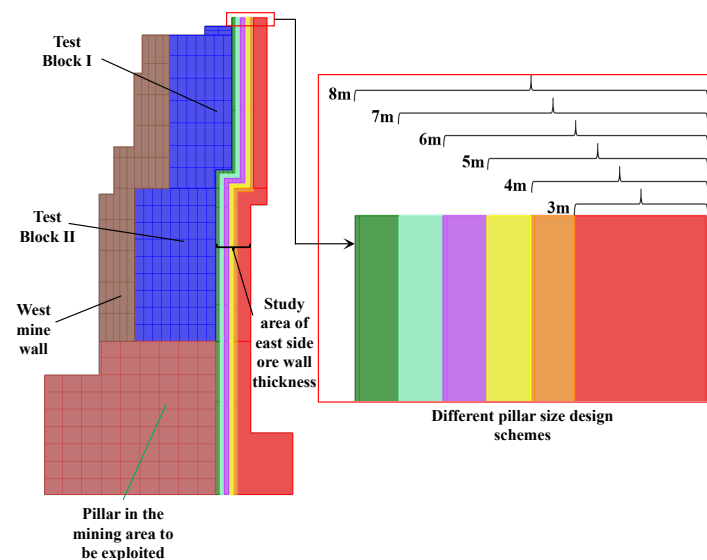


Figure 5. Layout plan of mineral blocks in the first mining area.

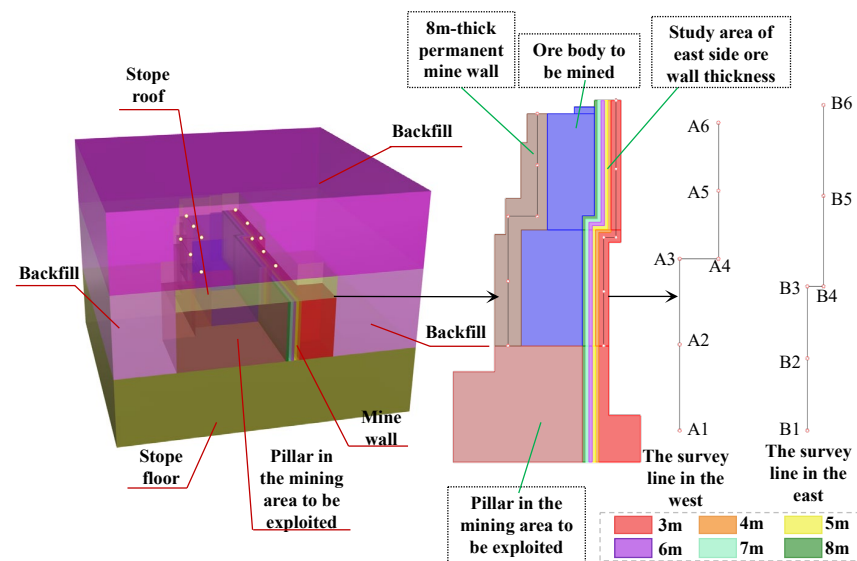


Figure 6. Layout plan of monitoring points.

Table 1. Simulated calculation conditions.

Simulation Steps	Detailed Description
Step 1	Calculate the initial stress field under the action of self-weight
Step 2	Comparison and Adjustment of Calculated Stress Values with In situ Stress Values
Step 3	Excavation of Block I Test Mining Block
Step 4	Backfilling of Block I Test Mining Block
Step 5	Excavation of Block II Test Mining Block
Step 6	Backfilling of Block II Test Mining Block

4.3. Mechanical Parameters and Boundary Conditions of the Model Rock Mass

To accurately understand the regional engineering geological conditions and provide reliable foundational data for subsequent mine engineering design and stability analysis, this paper systematically conducts a comprehensive global engineering geological survey of the area. By integrating the characteristics of the topography and geological structure, the survey employs surface mapping, drilling logging, and drilling rig coring to achieve thorough control over the distribution of rock strata, lithologic characteristics, and the development of structural planes within the study area. Notably, the drilling rig serves as the primary method for obtaining field rock samples, with strict control over drilling parameters and coring rates to ensure the integrity and representativeness of the core samples. Each obtained rock sample is labeled with its sampling depth and rock layer number, providing authentic and reliable physical samples for laboratory testing. Concurrently, in situ stress testing is performed, allowing for the precise determination of the magnitude, direction, and distribution of geostress in the study area. This is accomplished through the scientific arrangement of test points and the selection of testing methods suited to the regional geological conditions. The results of these tests are illustrated in Figure 7, which directly presents the fundamental characteristics of the regional geostress field.

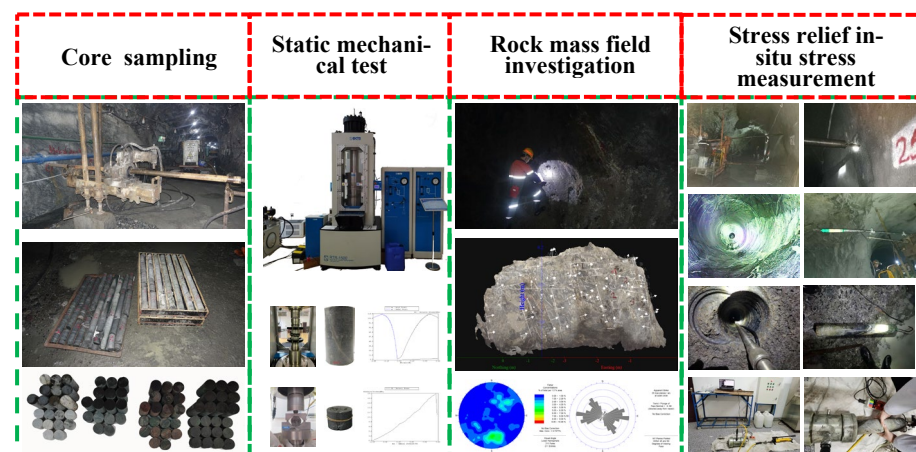


Figure 7. Mechanical parameter acquisition.

To convert on-site geological information into quantitative parameters suitable for numerical simulation, systematic indoor rock mechanics tests were conducted on the collected rock samples. A comprehensive technical process was established, encompassing “rock sample testing–quality classification–parameter assignment,” in conjunction with the rock mass quality evaluation system. Indoor tests adhered strictly to established rock mechanics testing standards. Key mechanical index tests, including uniaxial compressive strength, elastic modulus, and Poisson’s ratio, were performed on rock samples representing various lithologies. Throughout the testing process, variables such as loading rate and environmental temperature and humidity were meticulously controlled to ensure the accuracy and repeatability of the data. Building on this foundation, standardized evaluation methods were employed to classify the rock mass quality in the study area, utilizing indicators such as the development degree of the on-site rock mass structural surface and the core integrity index, while eliminating outliers from the test data. Utilizing rock mechanics parameters derived from laboratory tests and detailed findings from on-site geological investigations, the Roclab1.0 software facilitated a comprehensive analysis aimed at scientifically and rationally refining and calculating the pertinent parameters of rock mass quality. Through statistical analysis and correction, the mechanical parameters for different types of surrounding rocks in the study area were ultimately determined, encompassing core indicators

such as compressive strength, elastic modulus, cohesion, and internal friction angle. The specific parameters are detailed in Table 2, which provides quantitative support for the construction of the numerical model.

Table 2. Mechanical parameters of surrounding rock in the mine.

Lithology	Compressive Strength (MPa)	Tensile Strength (MPa)	Young's Modulus (GPa)	Poisson's Ratio	Cohesion (MPa)	Internal Friction Angle (°)	Density (KN/m ³)
Test Mining Block	22.31	1.45	19.76	0.14	2.45	43	28.3
Roof Pillar	22.31	1.45	19.76	0.14	2.45	43	27.5
Floor	22.65	1.50	11.30	0.25	1.68	35	28.3
Mining wall	22.31	1.45	19.76	0.14	2.45	43	28.3
Backfill	0.87	0.21	0.88	0.12	0.62	34	22.6

During the numerical model construction phase, it is essential to ensure that the simulation results align with actual engineering conditions. To achieve this, the mechanical properties of materials in each area of the model are precisely defined based on the engineering geological conditions of the mine in the study area and the predetermined numerical calculation scheme. In numerical analysis, Rhino 7.0 software is used for modeling and meshing, and FLAC3D 6.0 software is used for simulation calculation. By partitioning and classifying the grid cells within the model, the mechanical parameters of the surrounding rocks from different mines, as detailed in Table 2, are systematically assigned. This approach guarantees that the mechanical properties of each section of the model closely correspond to the actual geological formations, thereby minimizing the risk of simulation errors arising from parameter confusion or assignment discrepancies. To accurately replicate the true stress environment of the study area, the initial stress field of the model is established by integrating the model verification results with field stress test data, followed by a multi-round optimization adjustment. Ultimately, a self-weight stress of 19 MPa is applied to the model's top, calculated precisely based on the thickness of the overlying strata and the bulk density of the rock mass in the study area. A lateral stress coefficient, set at 1.5 times the dead weight stress, is applied around the model. This coefficient is calibrated by comparing field stress test results with the distribution patterns of the regional stress field, ensuring that the initial stress field of the numerical model closely approximates the actual stress state of the study area. This approach establishes a robust foundation for the subsequent numerical simulation analysis of the mine project's stability.

4.4. Numerical Simulation Results

Figure 8 shows the minimum principal stress contours for test block I and test block II in the Scheme Three test area after mining. The plots indicate relatively high compressive stress on the mining walls left on the east and west sides of both test blocks. From the minimum principal stress map, it is evident that the minimum principal stress on the mine walls increases substantially after stoping is completed. In Figure 8a, the stress on the rock walls bordering the goaf of test block I is markedly higher than in the unmined area. Prior to extraction, the mining wall was in a triaxial stress state; mining disrupts this equilibrium and converts it to a biaxial stress state. At the same time, the single-sided filling body adjacent to the mining wall imposes a persistent squeeze on the rock wall, which raises the loading on that wall. As a result, the stress on the goaf sidewall becomes significantly higher than in the surrounding unmined zones.

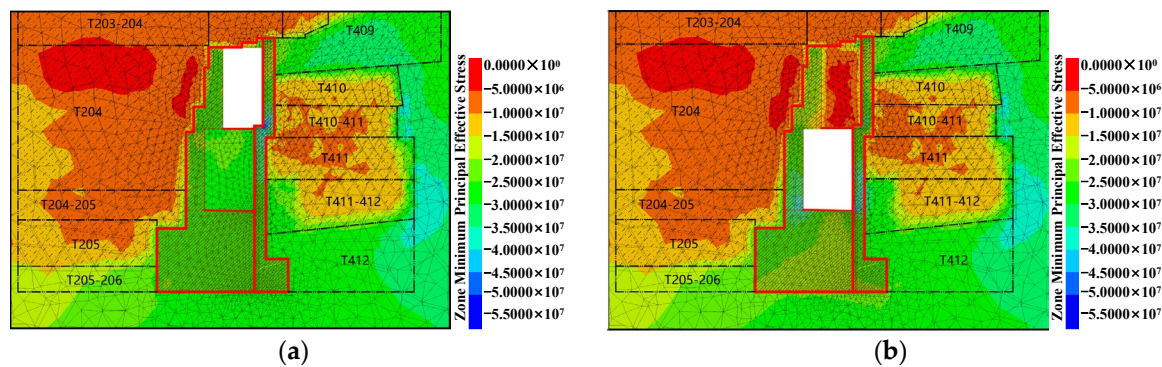


Figure 8. Minimum principal stress nephogram under the working condition of a 5 m mining wall. (a) Mining of test ore block I; (b) Mining of test ore block II.

To clarify the stress states of the east and west mining walls, we extracted the maximum and minimum principal stresses at monitoring points on both sides when test ore block I and test ore block II were mined in each scheme. The resulting comparisons for the test stope are shown in Figures 9–12. Figures 9 and 10 present the minimum principal stress comparisons for the east and west monitoring points, and Figures 11 and 12 present the corresponding maximum principal stress comparisons. Comparing the stress data under different remaining thicknesses shows a pronounced change in the east mining wall that is clearly negatively correlated with wall thickness. As the east wall thickens, both the minimum and maximum principal stresses at all monitoring points decline steadily, indicating that increased wall thickness reduces surrounding rock stress concentration and thereby improves the stress state. When the ore-wall thickness exceeds 5 m, the decline in stress slows markedly, the rate of decrease diminishes, and the stress curve flattens. This behavior indicates that for thicknesses below 5 m, the mining wall has not yet developed a fully self-bearing structure and thus shows higher sensitivity to stress adjustment. For thicknesses above 5 m, the mining wall attains adequate load-bearing and anti-deformation capacity, allowing the surrounding rock stress field to stabilize. Further increases in thickness produce only marginal improvements in stress. Accordingly, 5 m may serve as a practical reference threshold for selecting the reserved thickness of the eastern mining wall in engineering design. Unlike the eastern mine wall, the stress trends at the monitoring points on the western wall show little variation across different schemes, and the overall stress level remains stable with only minor fluctuations. This contrast stems from clear differences in design and initial stress conditions between the two walls. The eastern wall has a stepped thickness ranging from 3 m to 8 m, producing a large gradient in thickness and a correspondingly sensitive stress response. By contrast, the western wall is uniformly 8 m thick, giving it a relatively high initial thickness and strong rock mass integrity and bearing capacity. As a result, it sustains a stable stress-state under different mining schemes and is less affected by changes in external operating conditions.

At the end of stoping in test stope I, the east and west mining walls exhibit markedly different stress distributions. Monitoring point 4 shows a clear stress jump, with its value fluctuating sharply over a short interval. By contrast, stress at points 5 and 6 remains relatively stable and is generally slightly lower than at point 4. This pattern indicates that unloading from stoping test stope I produces spatially variable stress transfer within the mining wall. Because point 4 lies closer to the stoping zone, it experiences the surrounding-rock stress readjustment first. After test stope I is backfilled and stoping of test stope II is completed, the ore-wall stress field enters a second adjustment stage. At that stage, stress at monitoring points 1–3 on both the east and west mining walls trends upward, and the surrounding-rock bearing pressure increases gradually. The increase is most pro-

nounced at monitoring point 3, whose stress value becomes essentially equal to that at monitoring point 4. Comparing the spatial relationships between ore blocks I and II and each monitoring point shows that points 3 and 4 lie precisely in the turning region of the ore-wall geometry. This location explains the elevated stress readings and confirms a pronounced stress-concentration effect at the turning area. From the stope unloading perspective, the stress-release mode differs fundamentally between a straight-section mining wall and a broken-line turning point. During unloading of a straight-section mining wall, the surrounding boundary conditions change gradually, the stress-transmission path remains uniform, and stress is released steadily and orderly, so no localized stress increase occurs and the surrounding rock remains in a relatively stable stress state. In contrast, the unloading boundary at a broken-line turning point is abrupt and disrupts the continuity and uniformity of the original stress path. The resulting stress redistribution cannot occur rapidly, which permits stress to persist and accumulate at the turning area and ultimately produces a stress-concentration zone.

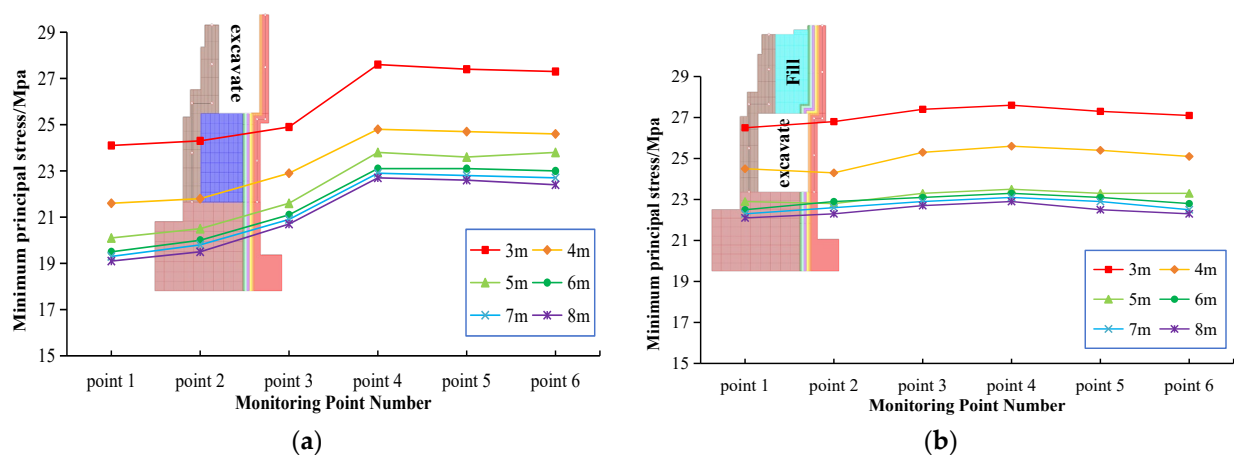


Figure 9. Comparative diagram of minimum principal stress data at the monitoring points on the east side of the mine wall. (a) Mining of test ore block I; (b) Mining of test ore block II.

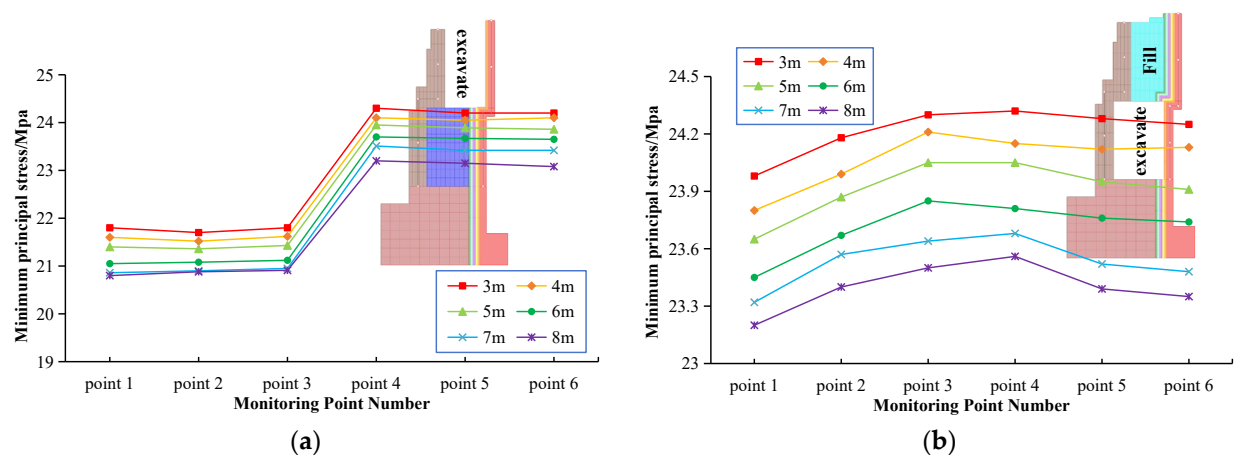


Figure 10. Comparative diagram of minimum principal stress data at the monitoring points on the west side of the mine wall. (a) Mining of test ore block I; (b) Mining of test ore block II.

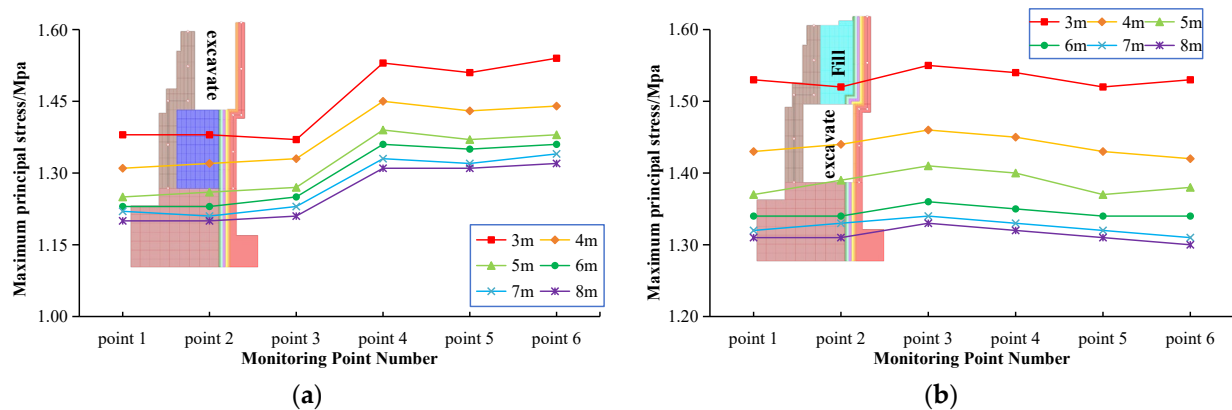


Figure 11. Comparative diagram of maximum principal stress data at the monitoring points on the east side of the mine wall. (a) Mining of test ore block I; (b) Mining of test ore block II.

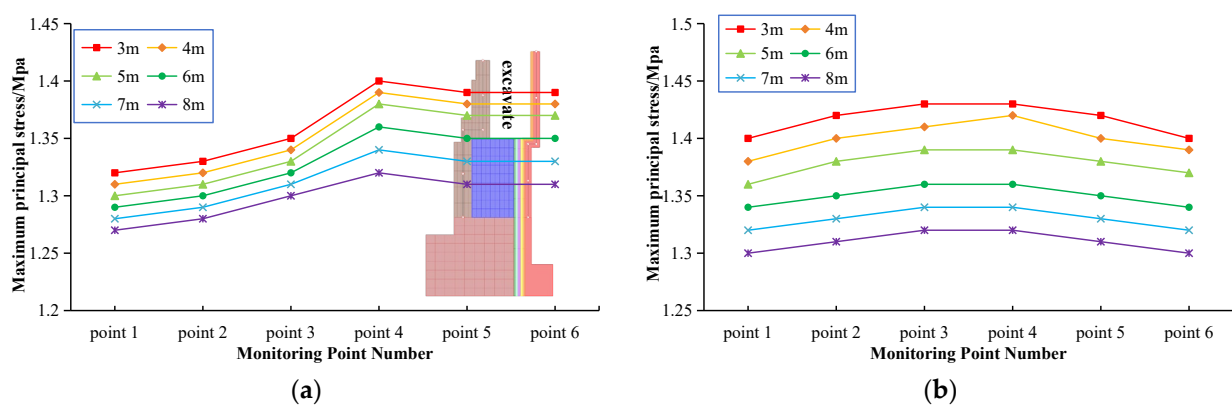


Figure 12. Comparative diagram of maximum principal stress data at the monitoring points on the west side of the mine wall. (a) Mining of test ore block I; (b) Mining of test ore block II.

Figure 13 shows the horizontal displacement nephogram in the X-direction after stoping of Test Ore Block I and Test Ore Block II. The nephogram reveals a pronounced spatially heterogeneous displacement pattern. Displacements of the mining wall adjacent to the stope are markedly larger than those in more distant areas, and all displacement vectors point inward toward the stope, consistent with the expected stress-release behavior and rock-mass deformation during stoping. To quantify ore-wall displacement and clarify deformation patterns under different conditions, we extracted the X-direction horizontal-displacement values from the monitoring points for each test scheme on the eastern mining wall. After sorting and visualizing these data, we produced the comparative plot of ore-wall monitoring-point horizontal displacements shown in Figure 14. Detailed analysis shows that ore-wall thickness is a primary control on horizontal displacement, following a clear pattern: displacement decreases as thickness increases. When the mining wall is thin, the rock mass has low intrinsic bearing capacity and, under stope mining disturbance, undergoes large-scale deformation that produces substantial horizontal displacement. As the mining wall thickens, rock-mass integrity and load-bearing capacity improve, increasing resistance to stope disturbance and reducing horizontal displacement. Once the ore-wall thickness exceeds 5 m, displacements at all monitoring points tend to stabilize and the differences among points diminish, indicating that the thicker mining wall can effectively resist mining-induced disturbance and that rock-mass deformation has reached a relatively stable state.

After stoping of Test Ore Block I was completed, the displacement distribution at the monitoring points exhibited clear segmentation. Monitoring Points 4–6 showed sub-

stantially larger displacements than Monitoring Points 1–3. A closer comparison of the schemes revealed that Monitoring Point 4 experienced markedly greater displacement than Monitoring Points 5 and 6. This difference stems primarily from the proximity of Monitoring Point 4 to the geometric turning area, where stress concentration produces a more pronounced displacement change. After filling test ore block I and completing the stoping of test ore block II, the stress state of the mining wall changes markedly. The backfill provides partial support to the mining wall. Concurrently, stoping in test ore block II induces secondary disturbance of the eastern mining wall, which increases displacements at monitoring points 1–3 and brings them closer to the displacements at points 4–6. At this stage, the displacement distribution across the eastern mining wall becomes more uniform. Displacements at monitoring points 4 and 5 remain slightly larger than at the other points, a difference that reflects the local positions of those sensors where the surrounding rock mass experiences more complex stress and greater deformation.

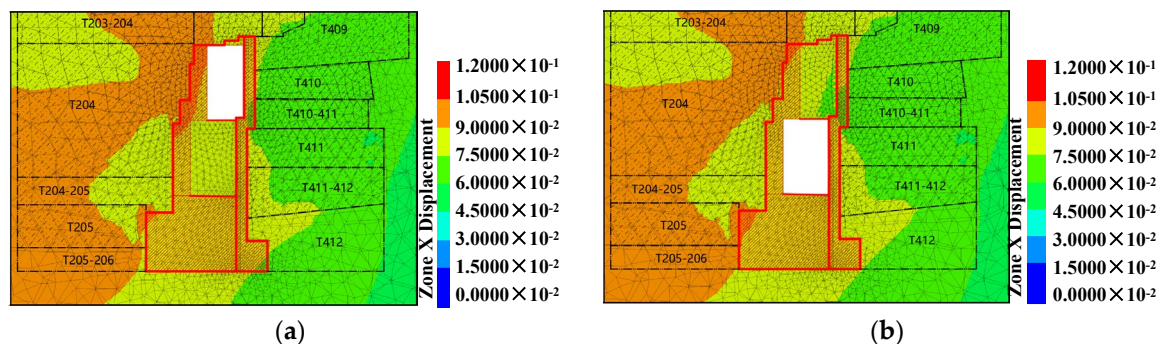


Figure 13. Displacement nephogram of scheme 1. (a) Mining of test ore block I; (b) Mining of test ore block II.

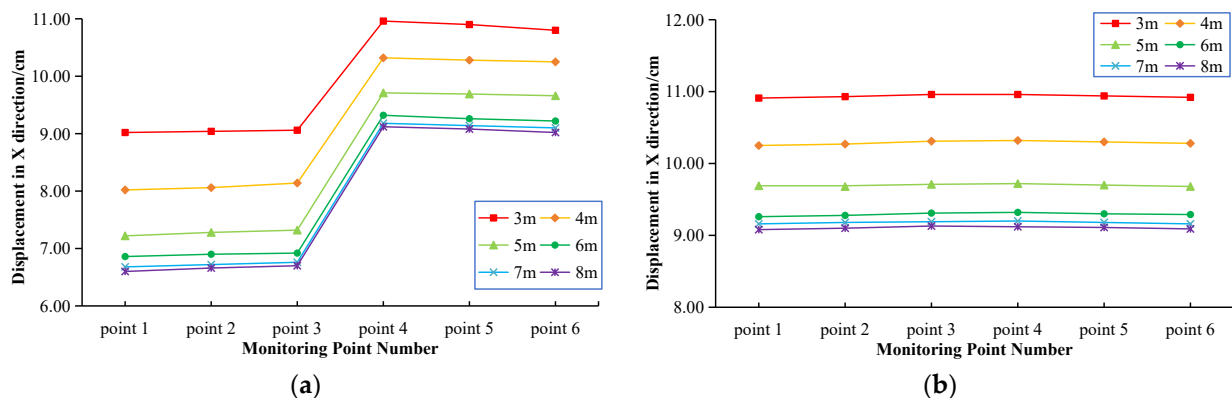


Figure 14. Comparison diagram of vertical displacement of the monitoring point of the mine wall on the east side. (a) Mining of test ore block I; (b) Mining of test ore block II.

5. Analysis of On-Site Monitoring Effect and Mine Wall Retention Effect

Due to long-term geostress and early mining disturbances, the rock mass integrity of the 204 panel pillars in the Tongkeng Mine is poor and the fracture phenomenon is obvious. This geological condition poses a potential threat to the mining safety of the panel. The spatial stress environment of the ore pillars in this panel area is complex. There are artificial filling bodies on the east and west sides. The mechanical properties of the filling bodies and the original ore pillars are quite different, which can easily cause uneven stress transmission. The other side is the ore pillars to be recycled, forming a composite stress system of “filling bodies and broken ore pillars”. The stability of the mine wall serves as a critical control point within the system, directly influencing the safety of personnel and equipment during

pillar mining. Additionally, it holds significant engineering implications for enhancing the recovery rate of mineral resources and minimizing resource waste. Consequently, by establishing stress monitoring points around the mining area, one can capture in real time the stress evolution, distribution characteristics, and transfer pathways of the orebody throughout the mining process. This approach provides essential data support for the safe extraction of the orebody. Furthermore, it establishes a theoretical and practical foundation for optimizing schemes and predicting risks associated with overall pillar recovery in subsequent stages, thereby contributing valuable insights for both scientific research and engineering applications.

Stress monitoring is conducted using the ZLGH-40 (Hubei Seaquake, Wuhan, China) borehole dynamometer in conjunction with the GSJ-2A (Hubei Seaquake, Wuhan, China) vibrating wire data technology memory, thereby establishing a comprehensive online stress monitoring system. This system links each stress meter probe to the main unit and sub-stations, enabling real-time measurement of the frequency from each probe and subsequent calculation of the stress value at each location. By connecting to a computer, users can view and record stress variations at each point in real time.

The stress calculations and analyses indicate that the stress load on the eastern side of the ore body designated for recovery is significantly greater than that on the western side. This disparity arises from differences in compaction degree and load transfer efficiency between the eastern and western sides of the filling body, rendering the eastern wall a critical area for stress monitoring. The theoretical analysis of stress generation positions reveals that, when the horizontal coordinate x is held constant, the maximum vertical stress σ_z occurs at half the thickness of the mining wall. Thus, the midpoint of the mining wall represents the core area of horizontal stress concentration. Considering the feasibility of engineering layouts in the adjacent middle section of the shaft and roadway, along with the convenience of monitoring equipment installation and data representativeness, the upper 405 m middle section was ultimately selected as the site for the stress monitoring point. The specific installation location of the monitoring point is illustrated in Figure 15. To ensure the accuracy and reliability of monitoring data, as well as the safety of equipment in the complex mining environment, the drilling hole is oriented obliquely downward toward the mine wall. The drill hole is initiated from the roadway wall at a midpoint of 405 m, extending to a depth of 2 m below the top elevation of the mining block. This configuration effectively mitigates the impact of surface rock mass disturbances on monitoring results, thereby ensuring that the data accurately reflects the stress state in the deeper sections of the mine wall.

As illustrated in Figure 16, the stress changes at the monitoring point of the mine wall exhibit a pronounced characteristic of stress concentration throughout the mining process. The degree of stress concentration and the range of its influence demonstrate a pattern of differential evolution across various monitoring sites as mining progresses. This observation provides direct evidence of a significant stress transfer phenomenon occurring during the mining of the test ore block. Notably, the area of stress concentration has extended to the surrounding rock mass of the drilling roadway, aligning with the results of prior stress field simulation analyses. Further examination reveals that, while stress concentration is evident at the mine wall monitoring point, the rock mass of the drilling roadway simultaneously experiences a substantial ground stress response, indicating a clear mechanical correlation between the two regions. The coupling analysis of monitoring data and the on-site assessment of rock mass conditions reveals that when the stress value at the monitoring point is below 26 MPa, the integrity of the side rock mass in the rock drilling roadway remains intact. In this state, no fractures, rock spalling, or other failure phenomena are observed, indicating stability. Conversely, when the stress

value exceeds 26 MPa, localized rock mass failures occur on the sides of the rock drilling roadway, including instances of localized collapse and rock fragmentation. However, these failures are confined to specific areas and do not propagate throughout the entire roadway structure. The orebody on both sides experiences minimal impact from stress concentration, allowing the overall stability to satisfy the engineering requirements for safe mining. This finding provides a critical basis for establishing the stress control threshold for future mining operations.

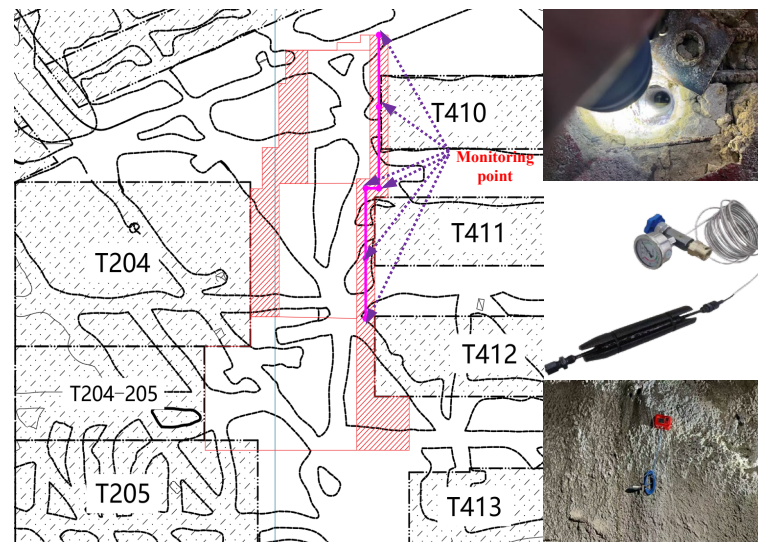


Figure 15. Installation position of monitoring points.

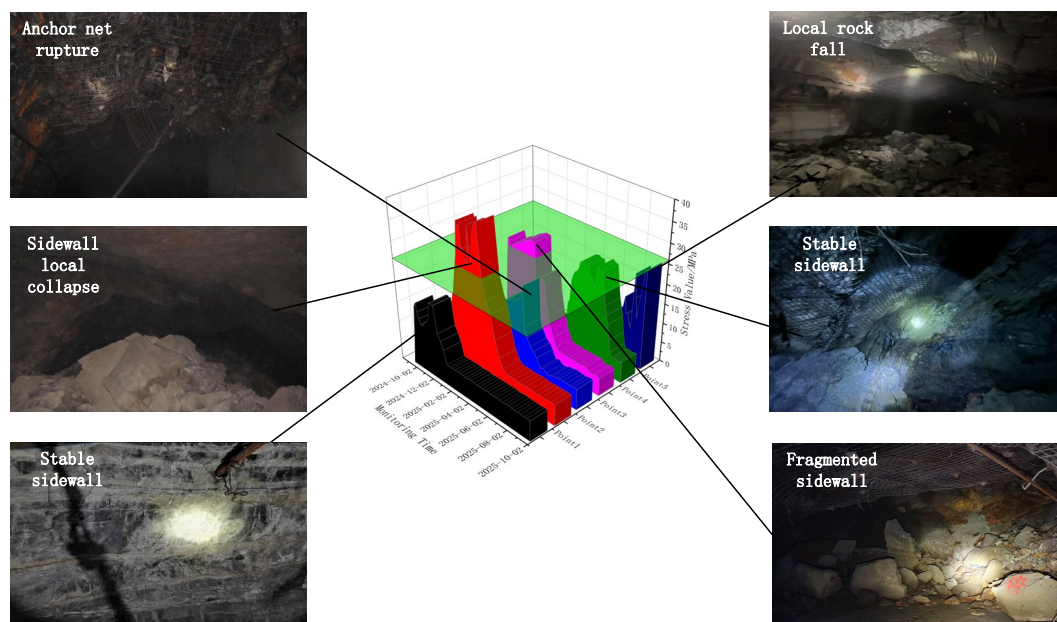


Figure 16. Comparison between stress monitoring results and roof damage.

6. Conclusions

This study presents a comprehensive analysis conducted from three perspectives: theoretical analysis, numerical simulation, and on-site monitoring. The optimal safety retention thickness for the Tongkeng mine was determined, with the main conclusions outlined as follows:

- (1) Differentiating the stress components allows for the identification of the locations of extreme values for each stress component on the two-dimensional plane of the mine wall. The extreme values of the horizontal stress components are found at half the thickness of the mine wall. To ensure the stability of the mine wall, it is sufficient to verify that the stress design value at the location of the extreme value is less than the allowable maximum stress when determining the thickness of the mine wall.
- (2) As the thickness of the mine wall increases, both the stress value and displacement at the monitoring point exhibit a decreasing trend, suggesting that a thicker mine wall enhances its stability. When the thickness of the mine wall exceeds 5 m, the rate of reduction in stress value and displacement at the monitoring point diminishes. A comprehensive analysis indicates that a 5-m-thick mine wall is sufficient to maintain structural stability, while further increasing the thickness yields negligible benefits and results in resource wastage.
- (3) Throughout the mining process of each test ore block, significant stress concentration occurs at the turning points of the mine wall, accompanied by an increase in horizontal displacement in this region. This phenomenon primarily arises from the stress release process at the turning point of the mine wall, which is impeded by the abrupt structural change, resulting in relatively concentrated stress generation.
- (4) The ore body in the test stope is extracted while maintaining a 5 m mine wall. The stress concentration in the mine wall corresponds with that in the rock excavation roadway of the test stope. Although the stress concentration in the mine wall is associated with varying degrees of stress manifestation in the rock excavation roadway, it ultimately ensures the safe recovery of the ore body.
- (5) The maximum stress that a rock excavation roadway can endure is approximately 26 MPa. When the stress at the monitoring point surpasses this threshold, significant stress manifestations will occur in the rock excavation roadway, leading to local rock mass collapse and fragmentation. During production, it is essential to maintain the stress level strictly within this critical value to ensure the overall stability of the roadway and the safety of operations.

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