ABSTRACT
To obtain a better economic benefit, Xiadian Gold Mine intended to increase the stope structure parameter from 2.5 m (height) to 8 m. A test stope using the new structure parameter was mined to verify the surrounding rock's stability. Based on the mechanical indices relating to the ore and rock, the “static” stability of the test stope under the new stope structure was analyzed by the Mathews stability diagram method. Then, a microseismic (MS) monitoring system was installed around the test stope to determine the “dynamic” stability of the test stope during mining. The spatio-temporal evolution of MS events was analyzed to understand the failure behaviors of the surrounding rock. Based on the MS monitoring results, a damage model was established and used to quantify the damage degree and dimension within the surrounding rock during mining the test stope. The study results show that the damage degree increased from less than 0.05 to 0.6, the damage dimension increased from less than 1500 m² to more than 14,000 m², and the shear plastic zone appeared around the two sidewalls and the roof of the stope. The results show that the test stope is at risk of instability under the new stope structure.

1. Introduction
In underground mining engineering, with the excavation-induced disturbance of an orebody, the equilibrium of the in-situ stress state in the rock mass is disturbed; this changes the mechanical environment in the rock mass around the excavation (Xia et al. 2019). As a result, stress is released in some areas, while it is concentrated in others. When the stress reaches the limiting strength of a rock mass, instability will occur in the surrounding rock. In deep mining, a reasonable size of ore pillars plays an important role in controlling ground pressures. In response to ground pressure on ore pillars in the deeper parts of a mine, stress concentration, changes in...
displacement, and damage occur. These responses can be fed back to determine optimal spatial geometric dimensions of ore pillars. In mines, a reasonable stope structure not only affects the efficient recovery of ore, but also influences the safety of property and personnel: the stability evaluation of stope structures is particularly important in engineering practice.

Many scholars have conducted in-depth studies on stope structures and obtained a series of analytical methods, which can be classified into three categories: (1) widely used empirical and semi-empirical analogy methods (Barton 2002; Hoek and Brown 1980; Mawdesley et al. 2001), such as the widely used rock mass rating (RMR) method, $Q$ system, and Mathews stability diagram; (2) theoretical calculation methods (e.g., thickness-to-span ratio and the K.B. Lu Peinie estimation method) established on the basis of classical mechanics (Brown 2003); (3) numerical simulation methods as developed in recent years (Lisjak and Grasselli 2014; Liu et al. 2018; Tang et al. 2015), such as the finite element method and discrete element analysis method. These studies play an important role in the stability analysis of stope structures.

Despite meticulous calculation and reasonability, theoretical calculation methods can only be used to calculate some regular stope structures. The empirical analogy method is simple and widely used. Although this kind of method has some limitations in evaluating the stability of stope structure due to complex rock mass structure and changeable geological conditions, it can give preliminary evaluation results. For instance, since 1980, many researchers have verified the Mathews stability diagram method by collecting more field data of deep mining (mostly less than 1,000 m) and constantly modified it. After 2000, according to many newly increased example data, this method has been constantly improved and has achieved good application in many mines (Jia et al. 2020; Trueman et al. 2000). However, the above method can only be used for static evaluation and cannot consider the time effect of rock deformation and failure (Xia et al. 2020). Therefore, it will be a feasible method to combine with in-situ monitoring. Although the numerical simulation analysis methods can solve the problems of changeable geological conditions and complex stope structures, the difficulty in selecting parameters remains. Incorrect parameter selection affects the stability analysis results of stope structures, making it difficult to interpret the mechanism of stope instability and prevent and control the resulting disasters. Therefore, in-situ monitoring is an indispensable means to study rock mass stability (Sun et al. 2020; Xia et al. 2018).

In recent years, the high-accuracy microseismic (MS) monitoring technique developed using geophysical theories has provided a scientific means of solving these problems. The MS technique uses elastic waves generated by the rock to assess fracturing therein. Now, this technique has been applied in various geotechnical engineering projects for daily safety monitoring (Zhang et al. 2015), such as rock burst forecasting in the deep-buried tunnels of Jinping II Hydropower Station (Ma et al. 2015), stability evaluation of deep-buried TBM construction tunnel (Tang et al. 2018), roof collapse forecasting in the coal mine (Cheng et al. 2019; Li et al. 2017) and seepage channel identification in an iron mine (Zhang et al. 2016; Zhao et al. 2020); however, there is still no accepted quantitative model with which to determine whether some MS phenomena will induce an geological disaster, or not.
To explain the MS characteristics and forecast the MS development trend, the combination of MS monitoring technique and numerical method was used by many scholars. For example, Hazzard and Young (2004) used Itasca Consulting Group’s particle flow code to model the seismicity in rock, including seismic event location, magnitude, and mechanism (i.e., moment tensors). Snelling et al. (2013) investigated the relationships between geologic structure, stress, seismicity and explained the anomalous cluster of MS events resulting from induced stresses from mine development at deeper levels. Xu et al. (2014) used the MS data as input to degrade the mechanical parameters of rock elements in the numerical model and then revealed the potential sliding surface of the rock slope. Zhang et al. (2016) modeled damage zones in a grout curtain using COMSOL Multiphysics software to explain the spatial distribution of MS events. The above research shows that the effective combination of in-situ monitoring and numerical simulation is a feasible method to evaluate the dynamic stability of the rock mass.

With these considerations, this study was conducted based on Xiadian Gold Mine in Zhaoyuan City, Shandong Province, China. A test stope stability under a new designed stope structure was preliminarily evaluated utilizing the Mathews stability diagram method. After that, an MS monitoring system was arranged around the test stope. By analyzing MS source parameters, the test stope’s dynamic stability during excavation was obtained. Furthermore, to evaluate the dynamic damage of the rock mass under the new stope structure, a damage model was established based on the energy dissipation theory and the MS data. A program was developed with the FISH language embedded in FLAC3D to search rock units within the damage dimension from MS events and weaken the rock mass. The damage and plastic zones evolution process were visualized, analyzed, and assessed. The research was expected to help mining and support designs of deep mines and provide a reference for work in the same type of mine elsewhere.

2. Engineering background

The Xiadian Gold Mine, located 30 km south of Zhaoyuan City, Shandong Province, China (Figure 1a), is one of the most productive gold mines in China since 1984. The major orebody is hosted in the beresitization zone and granite, at burial depths ranging from +161 m to −1412 m. The ore body’s overall trend is approximately NE 45°, with an average dip angle of 47°. The average thickness of the ore body is 12.61 m, and its average grade is 3.65 g/t. Tonalite and fault gouge form the immediate roof of the ore body, beresitization granite, and biotite granite form its floor.

From 1984 to 2000, non-pillar sub-level caving was adopted as the underground mining method. To control surface subsidence, upward horizontal cut-and-fill stoping was used since 2001. The stope structure measured $38 \text{ m} \times 7 \text{ m} \times 2.5 \text{ m}$ (length $\times$ width $\times$ height): as the excavation reached greater depths, the economic benefits decreased so, to guarantee the economic viability of the operation, the stope structure is planned to expand to $38 \text{ m} \times 7 \text{ m} \times 8 \text{ m}$. In addition, a test stope with improved structure parameters was excavated to test the surrounding rock’s stability.
Figure 1. Engineering geological conditions: (a) Location of the Xiadian Gold Mine; (b) Three-dimensional geological model of the Xiadian Gold Mine.
3. Stability analysis of the test stope based on the Mathews stability diagram

3.1. Brief introduction the Mathews stability diagram

The Mathews stability diagram is designed based on two factors, i.e., a stability number $N$ and a shape factor $S$. The stability number $N$ represents the ability of the rock mass to maintain stability under the given stress conditions, and the shape factor $S$ reflects the size and shape of a stope.

The stability number $N$ is calculated as follows (Mawdesley et al. 2001).

$$N = Q' \times A \times B \times C$$

where, definition of each parameter is listed in Table 1.

In this study, the $Q'$-value can be calculated indirectly by obtaining the quality $Q$ of the rock mass, as proposed by Barton (2002) based on 212 tunnel cases, and calculated by multiplication of six factors:

$$Q = \frac{RQD J_n}{J_a SRF} \frac{J_w}{J_a SRF}$$

where, $RQD$, $J_n$, and $J_r$ indicate the rock quality designation, number of joint sets, and joint roughness (the most unfavorable discontinuities or joint sets); $J_a$, $J_w$, and $SRF$ represent the alteration (variation) of joints (the most unfavourable discontinuities and joint sets), joint water reduction factor and stress reduction factor, respectively. When $J_w/SRF = 1$, the value of $Q$ is identical to that of $Q'$.

The shape factor $S$ is calculated according to the following equation:

$$S = \frac{a}{L}.$$
3.2. Analysis of the stope stability based on the Mathews stability diagram

Under the ShapeMetrix$^{3D}$ digital measuring system, a rock mass structural plane in the test stope was investigated. The detailed process of obtaining the parameters of the structural plane by this system can be found in Yang et al. (2015). By synthesizing the rock mass structural plane (Figure 2) with the obtained images based on pixel matching techniques at the same point and judgment of range of visibility, the geometrical information pertaining on rock joints and fractures was obtained (Table 2). Based on the engineering geological investigation results undertaken in-situ and rock mechanics test in the laboratory, the RQD was calculated, thence $Q'$ through Eq. (2) and Table 2.

$A$ represents the stress on the rock, as determined by the ratio of uniaxial compressive strength of intact rock to the compressive stress produced by mining in the mid-line of the stope: the maximum horizontal principal stress is 53.08 MPa at $-700$ m. $A$ is linearly correlated with $R_c/\sigma$, and its range of values is shown in Eq. (4). The results are listed in Table 3.

\[
\begin{align*}
R_c/\sigma < 2, A &= 0.1 \\
2 \leq R_c/\sigma \leq 10, A &= 0.1125(R_c/\sigma) - 0.125 \\
R_c/\sigma > 10, A &= 1
\end{align*}
\]  

(4)

Based on the statistical table containing data pertaining to major joints’ structural planes in Table 2, $B$ can be calculated by the angle between the exposed surface in
sereographic projection and major joints. \( C \) is mainly determined using \( C = \frac{8 - 6 \cos \alpha}{\cos \alpha} \) taken from the existing research and \( \alpha \) indicates the dip angle of the stope plane. Through the above calculation, the relevant parameters of the test stope are obtained, and the results are shown in Table 3.

The stability number \( N \) and hydraulic radius \( R \) of hanging wall rock and roof of the test stope calculated in Table 3 were plotted on the Mathews stability diagram (Figure 3). In the figure, the red, blue, and green dots are the mine cases that scholars used to plot the Mathews stability diagram (Mawdesley et al. 2001; Trueman et al. 2000). The black square and round dots in the figure represent the stability-evaluation results of the test stope’s sidewalls and roof, respectively.

It can be seen from Figure 3 that two sidewalls and roof of the test stope are in a transition zone from a stable state to failure. The reason is the increased exposed area of the stope after adjusting the stope structure and the poor lithologic conditions of the orebody. Therefore, it is challenging to ensure stope stability if using the new stope structure. In the following, the test stope’s stability during mining is further studied using the MS monitoring data.

4. Dynamic evaluation of the stability of the test stope based on MS monitoring

MS source parameters, such as the number, apparent stress, and apparent volume of MS events, change with the state of the rock mass from inception to final occurrence of dynamic disasters in the mine. By studying the spatio-temporal changes of such parameters, failure characteristics of the rock mass were found, thus allowing evaluation of the test stope’s stability. Based on this, the test stope’s damage process could be studied by analyzing the spatio-temporal evolution of each MS source parameter. Moreover, the test stope’s stability under the new stope structure was judged, guiding the subsequent design of the stope structure and backfill support operations in the mine.

4.1. Establishment of MS monitoring system

The test stope (the No. 550 stope) was buried between −692 m and −700 m, near the 550 exploration line. The test stope was mined from January 7 to January 23, 2016. The Institute of Mine seismology (IMS) MS monitoring system was installed around

| Table 2. Summary of parameters of the structural plane. |
|-----------------|----------------|----------------|----------------|----------------|----------------|----------------|
| RQD | \( J_o \) | \( J_r \) | \( J_a \) | \( J_w \) | SRF | Strike of major joints (°) | Dip angle of major joints (°) | Density of major joints (m⁻¹) | Trace length of major joints (m) | Compressive strength of rock mass /MPa |
| 64.92 | 12 | 1.5 | 1.5 | 1 | 1 | 202 | 74 | 3.45 | 0.61 | 70.62 |

| Table 3. Calculation of the stability number \( N \) in the test stope. |
|-----------------|----------------|----------------|----------------|----------------|----------------|----------------|
| Stope | \( Q' \) | \( A \) | \( B \) | \( C \) | \( N \) | \( R \) |
| Sidewall of the stope | 5.41 | 0.1 | 0.8 | 3.68 | 1.59 | 3.37 |
| Roof of the stope | 5.41 | 0.1 | 0.5 | 2 | 0.54 | 2.96 |
the test stope to monitor fracturing in the rock mass during mining. The sensor lay- 
out is shown in Figure 4: sensors 1 to 4 were installed at $-692$ m, and sensors 5 and 
6 at $-700$ m.

The hardware of the MS monitoring system can be mainly divided into three parts, 
namely data acquisition (NetADC), data communication (NetSP), and six geophones 
(including five uniaxial geophones and one triaxial geophone). The MS monitoring 
system can determine the MS source parameters, such as event location, energy, and 
magnitude, in either manual or automatic mode.

The MS event location algorithm used in the MS monitoring system of Xiadian 
Gold Mine is the simplex algorithm (Prugger and Gendzwill 1988), and the location 
error ($L_e$) then can be calculated as follows:

$$L_e = v_p \cdot t_e = v_p \cdot \| {t_0} - {t_i + R_i/v_p} \|_2 = v_p \cdot \sqrt{\sum_{i=1}^{N} (t_0 - t_i + R_i/v_p)^2}$$  \hspace{1cm} (5)

where $v_p$ is the velocity of the P-wave, $t_e$ is the time error, $N$ is the number of 
source-triggered sensors, $t_0$ is the origin time, $t_i$ is the P-wave arrival time recorded 
by the $i^{th}$ sensor, $R_i$ is the distance between MS event and the $i^{th}$ sensor.

The locational accuracy can be judged using Eq. (5). Figure 5 shows the location 
error of all the MS events in the study area. This figure shows that the location error 
is concentrated primarily on 4 to 12 m, demonstrating a high locational accuracy (He 
et al. 2013). Because of the high requirements of location accuracy for this study, MS 
events with location errors greater than 12 m were removed.
4.2. Dynamic evaluation of the stability of the test stope based on the MS source parameters

4.2.1. Analysis of the spatio-temporal evolution of MS events

Figure 6 shows the distribution of MS events before mining (Stage I, from December 11, 2015, to January 6, 2016), during mining (Stage II, from January 7 to January 23, 2016), and after mining (Stage III, from January 24 to February 29, 2016) test stope. Figure 6a–c correspond to Stages I, II and III, respectively. During the monitoring period, the seismic energy $E$ and the local magnitude of the MS events were $1.6E02$ J to $7.86E04$ J and $-2.8$ to $1.8$, respectively. In the figure, the colors of the spheres represent the magnitudes of MS events. The spatio-temporal evolutionary process of the damage and failure of surrounding rocks around the test stope was analyzed based on the spatio-temporal distributions of MS events and mining activities:

1. During Stage I (Figure 6a), drift excavation blasting was primarily conducted, in which the total charge used during a single blast was lower than that during stope blasting, and the excavation roadway presented a small area. Therefore, the
Figure 6. Temporal and spatial distribution of MS events: (a) Stage I, from December 11, 2015, to January 6, 2016 (before mining); (b) Stage II, from January 7 to January 23, 2016 (mining); (c) Stage III, from January 24 to February 29, 2016 (after mining).
disturbance of the drift excavation on the surrounding rocks of the test stope was relatively small and stable before mining the test stope. As a result, the distribution density of MS events was lower than in Stage II while higher than in Stage III. The MS events mainly at a magnitude of about -0.5, with a relatively uniform magnitude distribution.

2. During Stage II (Figure 6b), MS events primarily occurred around the test stope. They were most densely distributed in the stress concentration region in the intersection between the stope and the roadway. In this stage, the magnitude, number, and distribution density of MS events all increased, and multiple MS events with a relatively large magnitude (> 1.5) and energy (> 10,000 J) occurred. This result was primarily due to the substantial stress concentration at the ends of the stope because of the influence of in-situ stress. Mining was conducted using sublevel open stoping and subsequent filling methods. The above analysis shows that under the new stope structure, the surrounding rock will be greatly disturbed by this mining method.

3. During Stage III (Figure 6c), the mining activities were less intense. As a result, MS events were more widely disperse distributed, and they showed decreasing magnitude; in this case, the stress on the rock mass was adjusted. After completing the mining of the test stope, rock failure-induced MS events in the vicinity of the stope decreased owing to the stope being filled timeously; however, several MS events with a medium-level magnitude occurred within the drift excavation area. Therefore, it is necessary to pay attention to the surrounding rocks and deal with the unstable surrounding rock mass blocks in the roadway.

4.2.2. Analysis of apparent stress and cumulative apparent volume

Due to the complex underground mining environment, the failure of the rock mass cannot be measured by a single MS event. Therefore, according to seismology theory, statistical analysis based on the temporal evolution of MS events is generally undertaken to judge changes in the stability of a rock mass with time. As two important parameters describing seismogenesis, the apparent stress and volume are used to describe changes to the stress and deformation of the rock mass before and after an MS event occurred. The apparent stress $\sigma_A$ is defined as the ratio of radiation energy $E$ of MS events to deformation potential $P$ of MS events and represents the radiated MS energy of rock mass per unit area of an inelastic strain zone around a source (Mendecki 1997).

$$\sigma_A = \frac{E}{P}$$  \hspace{1cm} (6)

As a robust parameter measuring source volume, the apparent volume $V_A$ has scalar properties and represents the volume of a rock mass in the inelastic deformation zone around a source. Therefore, when analyzing seismic activity, the line’s slope representing changes in cumulative volume $\Sigma V_A$ with time is generally considered as a vital index representing the strain rate experienced by a rock mass.
where, $\mu$ denotes the shear modulus of the rock mass. Like the apparent stress, the apparent volume also depends on the deformation potential and radiation energy of MS events: because it is a scalar, it can be easily expressed in the form of cumulants or an isoline map.

Although the absolute stress and strain cannot be obtained directly using MS events, the reliable estimates of the apparent stress and cumulative apparent volume can be used as indices to measure the relative stress and strain. Many scholars (Dai et al. 2016; Li et al. 2016; Ma et al. 2018; Zhao et al. 2018) have performed extensive research on apparent stress and apparent volume, and the main conclusions are as follows: The higher the apparent stress of an MS event is, the greater the driving force of the rock failure will be, and the more likely rock failure is to occur. The surge in cumulative apparent volume is a significant feature before the occurrence of a rock burst or large-scale rock failure. In this paper, based on a field investigation and research on the changes in source parameters such as apparent volume and apparent stress, the information, and rules of the failure in stopes are obtained.

As shown in Figure 7, in the early stage of mining the test stope (from 7 January to 11 January, 2016), the apparent stress fluctuated (albeit within a small range), indicating that the accumulation and release of stress in the surrounding rock was not severe, and the mining disturbance was minor in this period. Soon afterward, from

$$V_A = \frac{\mu P^2}{E}$$

(7)
12 January to 14 January, 2016, the apparent stress dropped sharply, suggesting that the surrounding rock was damaged in this stage, leading to stress release. Since 15 January, the apparent stress increased continuously and exceeded that in the early stage of mining. The corresponding apparent volume also increased significantly, indicating that the stress and deformation in the rock surrounding the stope gradually accumulated to a high level. Until the end of mining work in the test stope, there was no typical rock mass failure seen such that the apparent stress decreased abruptly and the apparent volume increased substantially. This analysis shows that the surrounding rock of the test stope will be unstable in the mining process; that is, the new mining method cannot guarantee the safety of rock masses surrounding the test stope.

4.3. Damage evolution model of rock masses driven by MS data

To analyze the damage field evolution process under the mining influence, a damage evolution model of a rock mass driven by MS data was used. The degree and dimension of the damage can be quantified using this damage evolution model.

4.3.1. Quantification of damage degree

Rock failure can be described as a state instability phenomenon driven by energy. Each deformation mode corresponds to one or more forms of energy. Such as (Zhao et al. 2017): elastic energy $W_E$ produced by recoverable deformation after unloading, plastic energy $W_p$ corresponding to the difference value between total strain energy and recoverable strain energy, surface energy $W_X$ corresponding to the initiation and propagation of crack, radiation energy $W_m$ and kinetic energy $W_v$ during the failure of the rock and the energy dissipated in other forms $W_x$. Although the total energy during rock mass deformation process is conserved, it is not the superposition of all the energy listed above. Instead, there is a functional relationship between them and the total energy $W_{total}$, which can be expressed as Eq. (8). In this paper, the energy except elastic energy is unified as dispersion energy.

$$W_{total} = f(W_E, W_p, W_\Omega, W_m, W_v, W_x) = W_E + W_D$$ (8)

The damage process, weakening, and rock failure correspond to the accumulation, dissipation, and energy release. If a rock mass is assumed to be a closed system, according to the first law of thermodynamics:

$$U = U_d + U_e$$ (9)

where, $U$ is the total work done by external forces on rock mass, $U_d$ represents the internal energy dissipated, and $U_e$ is the stored strain energy. The relationship between $U_d$ and $U_e$ in a rock mass is as shown in Figure 8.

This study assumes that the fracture process of rock mass per unit is elastic-brittle deformation. The dissipation energy ($U_d$) of a rock mass comes from the unit volume’s stored strain energy as it undergoes brittle failure. The relationship between
the releasable strain energy ($\Delta U$) and the residual elastic strain energy ($U_h$) per unit volume is depicted in Figure 9. Before the failure, the unit volume’s deformation modulus remains the same; after failure, the deformation modulus changes from $E_0$ to $E_i$.

The total energy of rock mass units in the principal stress space can be expressed as follows:

$$U_U = \frac{1}{2E_0} \left[ \sigma_1^2 + \sigma_2^2 + \sigma_3^2 - 2\nu(\sigma_1\sigma_2 + \sigma_2\sigma_3 + \sigma_1\sigma_3) \right]$$

(10)

During failure, the releasable strain energy ($\Delta U$) stored in a rock unit is released and transformed into kinetic energy, MS energy, etc. The MS energy monitored by MS equipment is $\Delta U'$, defined as follows:

![Figure 8. Quantitative relationship between energy release and energy dissipation.](image1)

![Figure 9. Quantitative relationship between energy release and energy dissipation during elastic deformation.](image2)
\[ \Delta U' = \Delta U\eta \]  \hspace{1cm} (11)

where \( \eta \) is the MS efficiency. Only part of the energy was released in the form of the elastic wave during the damage process of the rock mass, most of which were consumed in other ways, such as the surface energy of newborn fracture. Therefore, the so-called MS efficiency \( \eta \) is the ratio of the MS energy \( E \) monitored and the total energy released when the rock mass breaks.

The proportional reduction of the deformation modulus is defined as the damage degree \( D \), and the \( D \) of a single rock mass unit within the damage dimension can be defined as:

\[ D = \frac{\Delta U}{U_U} = \frac{\Delta U'}{\eta U_U} \]  \hspace{1cm} (12)

The deformation modulus after the failure of the unit volume can be expressed as:

\[ E_1 = E_0(1-D) \]  \hspace{1cm} (13)

By substituting Eqs. (11) and (12) into Eq. (13), the deformation modulus after the failure of the unit volume is obtained as follows:

\[ E_1 = \left[ 1 - \frac{2E_0\Delta U'}{\eta[\sigma_1^2 + \sigma_2^2 + \sigma_3^2 - 2\nu(\sigma_1\sigma_2 + \sigma_2\sigma_3 + \sigma_1\sigma_3)]} \right] E_0 \]  \hspace{1cm} (14)

where, \( \sigma_1, \sigma_2, \) and \( \sigma_3 \) can be obtained by numerical simulation, \( \Delta U' \) can be monitored by MS monitoring equipment, and \( \eta \) takes 1.5% in this research (McGarr 1976; Zhao et al. 2017).

### 4.3.2. Quantification of the damage dimension

Source radius has a larger correlation with the selection of source model and often results in unrealistic results (Cai et al. 1998, McGarr 1994). However, the apparent volume is a relatively robust source parameter, so it is used to determine the damage dimension in this research. It is assumed that the apparent volume of MS events is an isotropic sphere volume. Accordingly, the damage dimension is defined as:

\[ R = \sqrt[3]{\frac{3M^2}{8\pi G\varepsilon^3}} \]  \hspace{1cm} (15)

Therefore, \( R \) is used to quantify the damage dimension of rock masses corresponding to the recorded MS events. The damage model weakens the deformation modulus of the rock mass units within distance \( R \) to the MS event location.

### 4.3.3. Analysis of damage evolution processes

1. Establishment of the numerical model

Based on geological data, a numerical model that consists of stopes, drift, a backfill body, and rock masses was established (Figure 10). The size of the established
calculation model is 1400 m long, 900 m wide, and 1000 m high, where the calculation model length direction is along the strike direction of the ore body. The width direction is perpendicular to the strike direction of the orebody, and the vertical direction is from +170 m to −830 m level. The Mohr-Coulomb model was used as the constitutive model. The calculation model boundary applies the in-situ stress according to the following equations.

\[ \sigma_v = \gamma H \]  \hspace{1cm} (16)

\[ \sigma_{\text{max}} = 0.0611H + 9.928 \]  \hspace{1cm} (17)

\[ \sigma_{\text{min}} = 0.0242H + 7.764 \]  \hspace{1cm} (18)

where, \( \sigma_v \), \( \sigma_{\text{max}} \) and \( \sigma_{\text{min}} \) denote the vertical stress, maximum horizontal principal stresses, and minimum horizontal principal stresses, respectively. The in-situ stress distribution (Eqs. (17) and (18)) is measured using the acoustic emission method. Both the maximum and minimum horizontal principal stresses increase linearly with depth. The average angle between the maximum horizontal principal stress and the orebody’s plumb line is 8.28°, which is perpendicular to the strike of the orebody.
The sequence of stope excavation during the numerical simulation was carried out as follows. Firstly, the stope with the caving mining method and the stope with the filling mining method above \( C_0 \) 652 m (Figure 10) was excavated. Secondly, the stope with the filling mining method was filled, and the drift between \( C_0 \) 692 m with \( C_0 \) 700 m and corresponding stopes were excavated to simulate the actual stress state before 7 January, 2016. Finally, the test stope and its surrounding 548, 549, 551, and 552 stopes at \( C_0 \) 700 m level were excavated in three steps. The plastic zone and damage zone of profiles 1-1 and 2-2 were extracted according to the profile orientation shown in Figure 10. Profile 1-1 is a profile vertical the orebody strike direction, and profile 2-2 is a profile along the ore body strike direction. The physico-mechanical parameters are listed in Table 4.

### Table 4. Physico–mechanical parameters of the rock mass, ore body, and filling body.

|                         | Tensile strength (MPa) | Bulk modulus (GPa) | Poisson’s ratio | Cohesion (MPa) | Frictional angle (°) | Bulk density (kg/m³) |
|-------------------------|------------------------|--------------------|-----------------|----------------|----------------------|---------------------|
| Rock mass               | 8.1                    | 14.67              | 0.25            | 5              | 29                   | 2.7                 |
| Ore body                | 8.6                    | 18.28              | 0.23            | 6.5            | 33                   | 2.75                |
| Filling body            | 0.5                    | 7.32               | 0.36            | 0.84           | 38                   | 2.2                 |

The surrounding rock has suffered severe damage during mining. The maximum degree of damage to the surrounding rock mass reached 0.6, which may not ensure the stability of the stope. The abovementioned analytical result was confirmed by in-situ observations of local instability and the analysis result using the Mathews stability diagram method in Section 3.2.

### 2. Damage evolution analysis

Figure 11 shows the evolution of the damage to the surrounding rocks based on the damage model. As shown in Figure 11a, the degree of damage of the rock mass around the test stope was less than 0.05 before 6 January, 2016. The damaged area was mainly distributed within the backfill body and the rock surrounding the drift (covering an area of less than 1500 m²).

During the mining of the test stope (from 7 January to 23 January, 2016, as shown in Figure 11b), both the degree of damage and damaged area increased within the rock surrounding the test stope. The maximum damage degree increased from 0.4 to 0.54, and the damaged dimension extended to approximately 11,000 m².

After the test stope was mined out, the damaged dimension continued to extend to more than 14,000 m², and the maximum damage degree reached 0.6 (Figure 11c). So far, the damaged dimension and damage degree therein began to stabilize.

### 3. Plastic zone analysis

Figure 12 shows the distribution of the plastic zone around the stope. Combined with Figures 11 and 12, it can be seen that before the mining of the test stope (Figure 12a), a small range of tensile and shear plastic zones appeared in several stopes around the test stope, which was caused by the low tensile strength of the filling body. The shear plastic zone was mainly distributed in the pillar between the drift...
and stopes, which indicated that the pillar was mainly sheared. After the test stope mining, the shear plastic zone appeared around the two sidewalls (2.3 m) and the roof (3.6 m) of the stope (Figure 12b and c). The research results in Figure 12 verify the analysis of the stability of the rock surrounding the test stope using the Mathews...
Figure 12. Plastic zone distribution of the surrounding rock around test stope; (a) Plastic zone of the surrounding rock of the test stope before mining (Profile 2-2); (b) Plastic zone of the surrounding rock of the test stope after mining (Profile 2-2); (c) Plastic zone of the surrounding rock of the test stope after mining (Profile 1-1).
stability diagram method. Therefore, it is challenging to ensure stope stability if using the new stope structure.

Based on the analysis results using the Mathews stability diagram, damage, and plastic zone distributions, it can be concluded that there are potential safety hazards in Xiadian Gold Mine under the new stope structure. The two sidewalls of the test stope have the risk of local rock wall spalling, and the roof of the test stope has the risk of local collapse. Considering the instability of fault gouge in hanging wall, if this new stope structure is selected, cemented filling should be used immediately after mining to isolate and support hanging wall and structure stable stope roof, to prevent further damage of two sidewalls of the stope and ensure safe mining of next layer.

5. Conclusion

In this study, the stability of the test stope was investigated using the semi-empirical and semi-theoretical Mathews stability diagram method. Then, an MS monitoring system and a damage model were used to evaluate the stability of the new stope structure in the Xiadian Gold Mine. Based on the above analysis, it can draw the following conclusions:

1. The test stope structure was analyzed using the Mathews stability diagram method. The research shows that the sidewalls and roof of the test stope under the new stope structure are in a stage of transition from a stable state to one of imminent failure, giving rise to the risk of instability in the new stope structure.
2. By analyzing the spatio-temporal distribution characteristics of MS events and the variation characteristics of apparent stress and accumulated apparent volume, it was concluded that the surrounding rock of the test stope was unstable during the mining process under the new stope structure, that is, the new mining method cannot guarantee the safety of the rock surrounding the test stope. As the representative of rock failure and damage, the results of MS monitoring were employed to establish a damage model. The damage model can quantify both the damage degree and damage dimension based on MS energy and apparent volume data.
3. The damage field evolution process and plastic zone distribution were analyzed based on the damage model and MS monitoring results. Due to the mining of the test stope, the damage degree increased from less than 0.05 to 0.6, the damage dimension increased from less than 1500 m² to more than 14,000 m², and the shear plastic zone appeared around the two sidewalls and the roof of the stope. According to the results of the Mathews stability diagram, damage, and plastic zone distributions, there are potential safety hazards in the new stope structure of the Xiadian Gold Mine.

Disclosure statement

No potential conflict of interest was reported by the author(s).
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**Data availability statement**

The data that support the finding of this study are available from the corresponding author upon reasonable request.

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