Mechanical Analysis of the Failure Characteristics of Stope Floor Induced by Mining and Confined Aquifer

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1. Introduction

Water inrush is a sudden catastrophic mine accident [1]. According to statistics from the Ministry of Emergency Management of the People’s Republic of China, more than 16 billion tons of coal resources are facing the risk of water inrush from the coal seam floor, which are mainly concentrated in industrially developed regions such as Huabei, Huadong, and Huanan. The coal reserves of these regions account for about 70% of the national coal resources, and nearly 60% of the coal mines are threatened by confined aquifer [2]. Water inrush has become the second most serious disaster after gas [3–5]. It not only causes heavy casualties and property losses but also severely damages local groundwater resources [6–8].

The fault is an important water channel, and the region near the fault is a high-prone area of water inrush. The prediction and prevention of water inrush from the working face across the fault has become a novel research hit [9–11]. Meng et al. [12] established a numerical model to analyze the impact of mining on the failure characteristics of the floor in the fault zone and pointed out that the risk of floor water inrush is greater when mining the footwall. Sun et al. [13] analyzed the sensitivity of floor water inrush to various parameters of the fault and used the limit hydraulic pressure that the floor can withstand as the criterion for floor water
inrush when the working face goes through the fault. Hu et al. [14] found that the phenomenon of abnormal water bearing occurs when the working face is close to the fault and believed that the fault activation caused by mining is an important reason of floor water inrush.

The research on the influencing factors of floor water inrush and the accurate prediction of dangerous locations are helpful to the precise implementation of measures. Yin et al. [15] analyzed the evolution law of underground water channels in the mining process by numerical simulation and found that the factors affecting the floor water inrush include the in situ stress, hydraulic pressure, working face width, mechanical properties of floor aquiclude, and breaking and swelling characteristics under hydraulic penetration. Liu et al. [16] used the orthogonal experiment method to analyze the sensitivity of floor failure depth to various influencing factors and evaluated the risk of water inrush from the floor. Cheng et al. [17] analyzed the influence of coal thickness, hydraulic pressure, thickness of the floor aquiclude, and other influencing factors on the risk of floor water inrush, comprehensively evaluated the influencing factors at different locations during the mining process, and believed that the open-off cut and the working face are the most likely locations for floor water inrush. Guo et al. [18] used physical simulation to study the evolution of floor stress during mining above confined aquifer and pointed that the floor water inrush occurs at the open-off cut before the first weighting of the roof.

With the rapid development of computer technology, various mathematical methods supplemented by computer have been widely used in mine floor water inrush. Wu et al. [19] proposed a mathematical evaluation method for evaluating the risk of water inrush from coal seam floor based on the variable weight model and unascertained measure. Zhao et al. [20] established a floor water inrush evaluation model based on a random forest (RF) intelligent machine learning algorithm and compared the risk assessment graph generated by this method with the probabilistic neural network (PNN) model. Wang et al. [21] used the two-level fuzzy comprehensive evaluation system to predict the risk of floor water inrush. Li and Chen [22] used grey relational analysis (GRA) and analytic hierarchy process (AHP) to establish an evaluation model, which effectively overcomes the uncertainty among evaluation indicators for floor water inrush and quantifies the significance of every indicator. Ma et al. [23] proposed a floor water inrush prediction model based on the combination of genetic algorithm and support vector machine. The model structure and related parameters were determined.

Only with a better understanding of the water inrush mechanism, it is possible to prevent and control mine water inrush and promote the integration of water inrush prevention and environmental protection [24]. Wu et al. [25] pointed out that mining activates the floor aquiclude and creates a water channel, which is the main cause of floor water inrush. Guo et al. [26] believed that the nature of floor water inrush from Ordovician limestone aquifer is the combined action of mining and confined aquifer, which caused the initiation, propagation, and penetration of microcracks in the floor aquiclude and finally led to the destruction of the floor. When the floor aquiclude cannot withstand the water pressure, the floor water inrush will occur. Lu and Wang [9] established a continuous coupled damage and flow modeling approach based on microscopic fracture mechanics and pointed out that floor water inrush is prone to occur when the floor rock strata are highly heterogeneous or the water pressure is high. Based on the theory of fluid-solid coupling, Ma et al. [27] analyzed the evolution of the maximum tensile stress as the working face advances, pointed out the similarity between the maximum tensile stress and the distribution of the plastic zone, and judged the maximum floor failure depth. Liu et al. [28] used similar simulation methods to study the evolution of pore water pressure during the whole process of the incubation, development, and occurrence of the floor water inrush and pointed out that the fluctuation of the pore water pressure after the sudden increase and the continuous increase of the vertical stress can be used as the precursor information of the floor water inrush.

In view of the early warning and prevention of floor water inrush from mining above confined aquifer, accurate calculation of floor failure depth is the key, and accurate description of the stress distribution caused by mining is a prerequisite [29, 30]. Based on the stress distribution of 41503 working face in Shandong Huatai Coal Mine, China, this study established a mechanical model in the semi-infinite body with the distributed load on the upper boundary (induced by mining) and the uniformly distributed load on the lower boundary (induced by confined aquifer) to calculate the stress in any point in the floor. The Mohr–Coulomb criterion was used as the floor failure criterion, and the analytical solution of the floor failure depth was obtained and compared with the in situ test data. The model was extended to different mining environments in different mining regions, achieving good results, which verified the reliability of the model.

2. Engineering Background

The No. 15 coal seam is being mined at Huatai coal mine in Shandong Province, China. The average coal thickness of the coal seam is 1.5 m, and the inclination angle is 2°–7°. It is a near horizontal coal seam with relatively positive geological conditions. According to the strata histograms of Huatai coal mine shown in Figure 1, the roof rock mass includes fine sandstone, sandy mudstone, and siltstone, and the floor rock mass is fine sandstone. The average depth of 41503 working face is 700 m, the length of roadway is 300 m, and the length of working face is 100 m. The longwall mining method is adopted, and all caving method is used to manage the roof. There is a confined aquifer at the depth of 50–60 m below the floor, and the maximum hydraulic pressure is 3.7 MPa, which poses a threat to the safety production.

3. Theoretical Analysis on Failure

Characteristics of Stope Floor

3.1. Basic Principles. Based on the half-plane theory [31], the stress redistribution caused by mining is simplified as a
model of the normal distribution load on the boundary of the semi-infinite body. As shown in Figure 2, the upper boundary of the semi-infinite body is subjected to a distributed stress with a concentration of $q$. In order to study the stress in any point inside it, a coordinate system is established. The microelement $d\xi$ is subjected to tiny concentrated force. The stress at point $M(x, y)$ caused by this microelement can be calculated. The stress vectors caused by all the microelements on the segment $AB$ are superimposed, and the stress caused by the distributed load of the entire segment at point $M(x, y)$ can be obtained. The calculation formulas for vertical stress $\sigma_x$, horizontal stress $\sigma_y$, and shear stress $\tau_{xy}$ are, respectively:

$$\sigma_x = -\frac{2}{\pi} \int_{b}^{a} \frac{qx^3 d\xi}{\left[x^2 + (y - \xi)^2\right]^2}$$

$$\sigma_y = -\frac{2}{\pi} \int_{b}^{a} \frac{qx(y - \xi)^2 d\xi}{\left[x^2 + (y - \xi)^2\right]^2} - \frac{2}{\pi} \int_{b}^{a} \frac{qx^2 (y - \xi) d\xi}{\left[x^2 + (y - \xi)^2\right]^2}$$

$$\tau_{xy} = -\frac{2}{\pi} \int_{b}^{a} \frac{qx^2 (y - \xi) d\xi}{\left[x^2 + (y - \xi)^2\right]^2}$$

(1)

3.2. Construction of Mechanical Model. According to the supporting pressure distribution of the working face [32], without considering the influence of tectonic stress, the stress in the stope floor along the strike is shown in Figure 3. Segments I and IV are the original stress zones, segment II is the stress increasing zone, and segment III is the stress decreasing zone. $S_1$ is the maximum principal stress, and $S_3$, $S_3 + \Delta\sigma$, and $S_3 - \Delta\sigma$ are the minimum principal stresses in the original stress zone, stress increasing zone, and stress decreasing zone, respectively [33, 34].

In order to simplify the calculation, the supporting pressure and the floor hydraulic pressure are simplified as linear loads. Taking the working face as the origin $O$, a rectangular coordinate system is established with the positive direction of the $x$-axis being perpendicular to the floor downward and the positive direction of the $y$-axis being
parallel to the roadway to the region to be mined, as shown in Figure 4.

Assuming that the floor strata are continuous, complete, uniform, and isotropic, this conforms to the basic assumptions of elasticity. The segment OQ is in the area to be mined. The segment QB has not been affected by mining and belongs to the original stress zone, the segment BA is the elastic zone, and the segment AO is the plastic zone. The segment OP is a region that has been mined. The segment OC is the gradually compacted zone where the gangue falling in the goaf is gradually compacted, and the load gradually increases toward the depth of the goaf until it returns to the original stress (i.e., segment CP). Assuming the original stress is γH, the peak (point A) of the stress concentration region is nyH. The length of segment OA is a, the length of segment AB is b − a, and the length of segment OC is c. In order to make the calculation more accurate, part of the original stress in segments QB and CP is also taken into consideration. The length of segment QB is q − b, the length of segment CP is p − c, and the length of the floor hydraulic pressure (segment MN) is p + q.

Based on the model shown in Figure 4, the calculation formula of the distributed stress concentration q of every segment along the strike is

\[
q_1 = \frac{yH}{c} \xi, \quad \xi \in (-c, 0)
\]

\[
q_2 = \frac{nyH}{a} \xi, \quad \xi \in (0, a)
\]

\[
q_3 = \frac{yH}{b - a} [(n - 1)\xi - (bn - a)], \quad \xi \in (a, b)
\]

\[
q_4 = yH, \quad \xi \in (-p, -c)
\]

\[
q_5 = yH, \quad \xi \in (b, q)
\]

\[
q_6 = P, \quad \xi \in (-p, q)
\]

where, \(q_1, q_2, q_3, q_4, q_5, q_6\) are the normal distribution stress concentrations of the strike, kN·m⁻¹, \(H\) is the depth of the coal seam, \(m\), \(γ\) is the coal seam floor rock mass bulk density, kN·m⁻³, and \(n\) is the stress concentration coefficient.

Substituting \(q\) in equation (2) into equation (1), the vertical stress, horizontal stress, and shear stress caused by distributed stress concentration \(q_1, q_2, q_3, q_4, q_5, q_6\) at any point in the stope floor are obtained.

The stress components caused by \(q_1\) (segment OC) are
The stress components caused by \( q_2 \) (segment OA) are

\[
\begin{align*}
\sigma_{x2} &= -\frac{nyH}{na} \left[ \frac{x^3 + xy(y - a)}{x^2 + (y - a)^2} - \frac{x^3 + xy^2}{x^2 + y^2} + y \left( \arctan \frac{y - a}{x} - \arctan \frac{y}{x} \right) \right] \\
\sigma_{y2} &= -\frac{nyH}{na} \left[ \frac{x^3 + xy^2}{x^2 + y^2} - \frac{x^3 + xy(y - a)}{x^2 + (y - a)^2} - y \left( \arctan \frac{y - a}{x} - \arctan \frac{y}{x} \right) + x \ln \left( \frac{x^2 + y^2}{x^2 + (y - a)^2} \right) \right] \\
\tau_{xy} &= -\frac{nyH}{na} \left[ x \left( \arctan \frac{y}{x} - \arctan \frac{y - a}{x} \right) - \frac{ax^2}{x^2 + (y - a)^2} \right]
\end{align*}
\]

The stress components caused by \( q_3 \) (segment AB) are

\[
\begin{align*}
\sigma_{x3} &= -\frac{yH}{\pi(b - a)} \left[ \frac{(a - b)x^3 + x(y - a)[(n - 1)y - (bn - a)]}{x^2 + (y - b)^2} - \frac{(a - b)x^3 + x(y - a)[(n - 1)y - (bn - a)]}{x^2 + (y - a)^2} \right] \\
\sigma_{y3} &= -\frac{yH}{\pi(b - a)} \left[ \frac{(a - b)x^3 + x(y - a)[(n - 1)y - (bn - a)]}{x^2 + (y - b)^2} - \frac{(a - b)x^3 + x(y - a)[(n - 1)y - (bn - a)]}{x^2 + (y - a)^2} \right] \\
\tau_{xy} &= -\frac{yH}{\pi(b - a)} \left[ \frac{(a - b)x^3 + x(y - a)[(n - 1)y - (bn - a)]}{x^2 + (y - b)^2} - \frac{(a - b)x^3 + x(y - a)[(n - 1)y - (bn - a)]}{x^2 + (y - a)^2} \right]
\end{align*}
\]
The stress components caused by \( q_4 \) (segment PC) are
\[
\sigma_{x4} = \frac{yH}{\pi} \left[ \frac{x}{\sqrt{x^2 + (y + p)^2}} - \frac{x}{\sqrt{x^2 + (y + c)^2}} \right] \\
\sigma_{y4} = \frac{yH}{\pi} \left[ \frac{x}{\sqrt{x^2 + (y + p)^2}} - \frac{x}{\sqrt{x^2 + (y + c)^2}} \right] \\
\tau_{xy4} = -\frac{yH}{\pi} \left[ \frac{x^2}{x^2 + (y + p)^2} - \frac{x^2}{x^2 + (y + c)^2} \right]
\]

The stress components caused by \( q_5 \) (segment BQ) are
\[
\sigma_{x5} = \frac{yH}{\pi} \left[ \frac{x}{\sqrt{x^2 + (y - b)^2}} - \frac{x}{\sqrt{x^2 + (y - q)^2}} \right] \\
\sigma_{y5} = \frac{yH}{\pi} \left[ \frac{x}{\sqrt{x^2 + (y - b)^2}} - \frac{x}{\sqrt{x^2 + (y - q)^2}} \right] \\
\tau_{xy5} = \frac{yH}{\pi} \left[ \frac{x^2}{x^2 + (y - b)^2} - \frac{x^2}{x^2 + (y - q)^2} \right]
\]

The stress components caused by \( q_6 \) (segment MN) are
\[
\sigma_{x6} = \frac{p}{\pi} \left[ \frac{x}{\sqrt{x^2 + (y + p)^2}} - \frac{x}{\sqrt{x^2 + (y - q)^2}} \right] \\
\sigma_{y6} = \frac{p}{\pi} \left[ \frac{x}{\sqrt{x^2 + (y + p)^2}} - \frac{x}{\sqrt{x^2 + (y - q)^2}} \right] \\
\tau_{xy6} = -\frac{p}{\pi} \left[ \frac{x^2}{x^2 + (y + p)^2} - \frac{x^2}{x^2 + (y - q)^2} \right]
\]

According to the principle of vector superposition, the stress components under the action of the stress in every segment are correspondingly added, and the vertical stress \( \sigma_z \), horizontal stress \( \sigma_x \) and \( \sigma_y \), and shear stress \( \tau_{xy} \) at any point in the floor along can be obtained. The calculation formulas of \( \sigma_z \), \( \sigma_x \), \( \sigma_y \), and \( \tau_{xy} \) are
\[
\sigma_z = \sum_{i=1}^{6} \sigma_{zi} + \sum_{i=1}^{6} \sigma_{xi} + \sum_{i=1}^{6} \tau_{xyi} = \sum_{i=1}^{6} \tau_{xyi}
\]

3.3. Stress Redistribution of Stope Floor. The hydrogeological report and monitoring data of Huatai coal mine show that the parameters are as follows [35–37]: the average bulk density of the overlying strata is \( \gamma = 25 \text{kN/m}^3 \), the depth of the coal seam is \( H = 700 \text{m} \), the stress concentration coefficient is \( k = 2.1 \), the length of the plastic zone is \( a = 8 \text{m} \), the length of the elastic zone is \( b - a = 18 \text{m} \), and the length of the gradually compacted zone is \( c = 60 \text{m} \). In order to analyze the stress distribution and floor failure range more accurately,
the original stress zone length of 150 m outside the mining-affected zone has been taken into consideration [37, 38], so \( p = 210 \text{ m} \) and \( q = 176 \text{ m} \). For safety reasons, the floor hydraulic pressure is \( p = 4 \text{ MPa} \).

Substituting the above parameters into equation (9), the stress redistribution of the coal seam floor under the combined action of mining and confined aquifer can be calculated. Mathematica, a visualization software application, is used to visualize the stress distribution of the vertical stress, horizontal stress, and shear stress in the floor along the strike within a certain range, as shown in Figures 5–7.

As shown in Figure 5, the vertical stress distribution of the floor shows obvious stress concentration and stress release phenomenon, and it takes the junction of stress increasing area and stress decreasing area as the dividing line, forming two groups of “convex arches” at the solid coal side and the goaf side, respectively. In the stress concentration area, the shallow vertical stress is greater than the deep in the vertical direction, and the peak stress in the horizontal direction is at the peak of the advanced abutment pressure, which is about 30.63 MPa, and the stress concentration coefficient is about 1.75. The stress gradient on the solid coal side is higher. The reason is that under the action of the advanced supporting pressure, the floor stress changes rapidly with the increase of depth. At the solid coal side, the floor strata within 18 m are in varying degrees of stress concentration, while within the range of 18–50 m below the stope floor at the solid coal side, the rock mass is in varying degrees of stress relief. Whether it is on the solid coal side or the goaf side, the shallow stress gradient is higher than that of the deep section, indicating that as the depth increases, the influence of supporting pressure on the stress distribution of the floor rock mass becomes lower.

It can be seen from Figure 6 that the horizontal stress in the stope floor presents a wide range of stress release and local stress concentration. The horizontal stress in the stope floor takes the junction of stress increasing area and stress decreasing area as the dividing line, forming two groups of “convex arches” at the solid coal side and goaf side, respectively. The stress gradient on the solid coal side is greater than that on the goaf side, indicating that under the action of advanced supporting pressure, the changing rate of horizontal stress with increasing depth is similar. Due to the Poisson effect of the rock mass, the horizontal stress concentration range below the solid coal is relatively small, and it has been restored to the original stress at 4 m below the floor. The maximum horizontal stress appears in the shallow part of the floor at the junction of the elastic and plastic zones, which is about 29.05 MPa, and the stress concentration coefficient is about 1.66.

As shown in Figure 7, the shear stress in the stope floor is divided into three zones by two dashed lines. The maximum shear stress in the middle zone appears in an ellipse of 4–14 m below the floor at the plastic zone, which is 11.22 MPa. The shear stress in the left zone is mainly affected by the elastic zone and the original stress zone, and its range is relatively small, and the maximum impact depth is about 12 m. The shear stress in the right zone is mainly affected by the stress decreasing zone, and its range is relatively large, and the maximum impact depth is up to 42 m. There are two sets of shear couples in the floor, and a boundary line of positive and negative shear stress is formed under the solid coal and the goaf, causing the floor rock mass at the boundary line to show compressive shear or tensile shear failure.

3.4. Theoretical Failure Range of the Floor. The exploration of the stress redistribution in the stope floor rock mass is a prerequisite for analyzing the failure range of the floor. The weak surface of floor strata is often opened by shear force during the mining, which leads to the destruction of the floor and the formation of potential water channels. Therefore, the shear strength of the rock mass is a key factor affecting the failure of the floor. The failure occurs when the shear stress in the floor strata is higher than the shear strength \( \tau \) [39, 40]. This study adopts the Mohr–Coulomb criterion, and the failure criterion at any point in the floor is

\[
\sigma_1 < \frac{1 + \sin \phi}{1 - \sin \phi} \sigma_3 + \frac{2C \cos \phi}{1 - \sin \phi}
\]

where \( C \) is the cohesion, MPa, and \( \phi \) is the internal friction angle, °. The calculation formula for the maximum and minimum principal stresses \( \sigma_1 \) and \( \sigma_3 \) is

\[
\sigma_{1,3} = \frac{\sigma_x + \sigma_y}{2} \pm \sqrt{\left(\frac{\sigma_x - \sigma_y}{2}\right)^2 + \tau_{xy}^2}
\]

According to equation (10), define the parameter \( F(y, x) \):

\[
F(y, x) = \sigma_1 - \frac{1 + \sin \phi}{1 - \sin \phi} \sigma_3 - \frac{2C \cos \phi}{1 - \sin \phi}
\]

When \( F(y, x) < 0 \), the floor strata are destroyed. The laboratory test on the coal samples of Huatai coal mine shows that \( C = 6 \text{ MPa} \) and \( \phi = 30° \), which are substituted into equation (12), and the failure range of the floor is obtained, as shown in Figure 8.

It can be seen from Figure 8 that the floor strata have been destroyed in the shallow area below the dashed line. The failure range of the floor in the horizontal direction is 50 m behind the working face, and the maximum failure depth is located in the middle of the failure range, and the maximum failure depth is 9.68 m. The main roof in this area has not touched the gangue, and the shallow floor is highly unloaded. Coupled with the action of confined aquifer, the floor strata in this area are easy to be damaged by tensile force. It is worth noting that there is also a failure region located 40–50 m behind the working face and 34–44 m below the floor. This region is only about 5 m away from the confined aquifer, and the water can easily enter this region along the cracks in the rock mass. Therefore, this region should be taken into consideration when calculating the thickness of the floor aquiclude. Through calculation, the thickness of the floor aquiclude is about 24 m. In the goaf far away from the working face, as the main roof gradually falls and compacts, the floor rock mass in this region is in a compressed state and is relatively stable. The rock mass

\[
\begin{align*}
\tau & = \frac{1}{2} \left( \frac{\sigma_1 + \sigma_3}{2} - \frac{\sigma_1 - \sigma_3}{2} \right) \\
\tau & = \frac{1}{2} \left( \frac{\sigma_1 + \sigma_3}{2} + \frac{\sigma_1 - \sigma_3}{2} \right)
\end{align*}
\]
Figure 5: The vertical stress distribution.

Figure 6: The horizontal stress distribution.
below the solid coal is also in a compressed state and is relatively stable.

4. In Situ Test

4.1. Device and Scheme. In order to understand the damage of stope floor in Huatai coal mine, the double-end plugging leakage detection device was used to measure the floor failure depth, as shown in Figure 9. Drilling holes with different dip angles were constructed at the selected location, and then plugging water injection test was carried out.

According to the water leakage of sealing section at different depths, the floor failure after mining was determined.

The detection site was selected in the return air roadway of 41503 working face, and three detection boreholes were constructed in the stope floor, including one premining borehole (No. 1) and two postmining boreholes (No. 2 and No. 3), as shown in Figure 10.

4.2. Measured Results. According to the results of the in situ test, the results of measured water leakage in the premining and postmining boreholes are shown in Figure 11.
(1) The No.1 borehole is the premining borehole, which is used to observe the fracture situation of stope floor without the impact of mining. It can be seen from Figure 11(a) that the water leakage of borehole 1 is relatively large at the depth of 8–11 m, with a peak value of 2.2 L/min, which may be caused by primary fractures in the stope floor. In addition, there is sporadic water leakage at the depth of 2–3, 35, 38, and 49 m, and the leakage of other drilling sections is very small, indicating that the development degree of cracks in stope floor is low, and the floor strata are relatively complete.

(2) The No. 2 borehole is the postmining borehole. As can be seen from Figure 11(b), there is a continuous leakage section within the range of 30 m of borehole depth, and the average leakage is about 1.35 L/min. Compared with the premining conditions, the water leakage in the section of 15–30 m section increases significantly, indicating that new fractures are produced in this section due to mining. Except for the leakage point at 41–44 m, the leakage of other sections is close to 0. Therefore, the critical failure depth is 30 m at the sudden drop point after the continuous leakage section. Due to the inclination angle of 20°, the failure depth of stope floor in the No. 2 borehole is 10.16 m.

(3) The No. 3 borehole is the postmining borehole. It can be seen from Figure 11(c) that there is a continuous water leakage section within 44 m of the borehole depth. Compared with No. 2 borehole, No. 3 borehole with a smaller dip angle is closer to the coal seam in the vertical direction and thus more severely damaged by mining. It is shown that the 31–40 m section is connected to the top and bottom leaking section, indicating that cracks in this section have been extended under the impact of mining, and the floor is damaged. According to the calculation of the sudden drop point of 44 m in the water leakage section and the borehole angle of 15°, the measured floor failure depth of the No. 3 borehole is 11.38 m. Therefore, the maximum failure depth of the stope floor of the 41503 working face is 11.38 m, and this estimation is of high reliability.

4.3. Popularization and Application. In order to further verify the accuracy and applicability of the model, five mines with different mining conditions, Xinzhi mine, Yangcun mine, Liujiazhuang mine, Suntuan mine, and Chenghe mine, are selected from different mining areas for verification. The production conditions, theoretical calculation, and in situ measurement results of floor failure depth of each mine are shown in Table 1. Through the comparison of the
measured values and the theoretical calculation values of the stope floor failure depth of five coal mines, it is found that the theoretical calculation results can better reflect the results of the in situ test. The maximum absolute error of theoretical calculation result is 1.1 m, the mean absolute error is 0.8 m, the maximum relative error is 8.2%, and the mean relative error is 6.5%. Therefore, the model has good applicability and popularization and provides a reference for the prediction of floor failure depth.

5. Conclusion

Based on the half-plane mechanical model, the distribution of vertical stress, horizontal stress, and shear stress was
obtained. The failure range of stope floor was calculated based on the Mohr–Coulomb criterion and verified by the in situ test. Finally, the model was popularized and applied, and the following conclusions were made:

(1) The vertical stress in stope floor forms two groups of “convex arches” at the solid coal side and the goaf side, respectively, and the gradient at the solid coal side is higher; the horizontal stress of the stope floor presents a wide range of stress release and local stress concentration. The shear stress in the stope floor is divided into three regions and two groups of shear couple, forming a boundary between positive and negative shear stress, which makes the stope floor in this area to show compression shear or tension shear failure.

(2) Based on the in situ test, the maximum failure depth of the stope floor is 11.38 m, which is roughly equivalent to the theoretical calculation result of 9.68 m. The mechanical model is used to analyze the floor failure depth in other five coal mines with different mining conditions and stress states. The maximum absolute error is 1.1 m, the average absolute error is 0.8 m, the maximum relative error is 8.2%, and the average relative error is 6.5%. The feasibility of using this mechanical model to study the floor failure depth is verified.

Data Availability

The data supporting the conclusion of the article are shown in the relevant figures and tables in the article.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

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