Research Article

An Influence Study of Face Length Effect on Floor Stability under Water-Rock Coupling Action

Baojie Fu ^{1,2} and **Bo Wang** ^{3,4}

¹*School of Mining and Safety Engineering, Anhui University of Science and Technology, Road Taifeng No. 168, Huainan, Anhui 232001, China*

²*Key Laboratory of Coal Mine Safety and Efficiently Caving of Ministry of Education, Anhui University of Science and Technology, Huainan, Anhui 232001, China*

³*CCTEG Chongqing Research Institute, Chongqing 400037, China*

⁴*School of Mechanics and Civil Engineering, China University of Mining and Technology, Xuzhou, Jiangsu 221116, China*

Correspondence should be addressed to Baojie Fu; bifu@aust.edu.cn

Received 25 November 2020; Revised 21 January 2021; Accepted 22 February 2021; Published 10 March 2021

Academic Editor: Chuangzhou Wu

Copyright © 2021 Baojie Fu and Bo Wang. This is an open access article distributed under the Creative Commons Attribution License, which permits unrestricted use, distribution, and reproduction in any medium, provided the original work is properly cited.

The Taiyuan Formation limestone aquifer and Ordovician limestone aquifer are widely distributed in the coal seam floor of coal measures in North China; the water hazard safety problem of the stope floor under the influence of mining is very prominent. The risk of the water inrush from the coal seam floor is closely related to the degree of full exploitation, so it is necessary to study the stability of the stope floor under aquifer conditions, especially the influence of the working face length effect on floor stability. Through numerical simulation of water-rock coupling action, the mine pressure behaviors of the water-resisting floor under different face lengths were analyzed based on the measured formation permeability coefficient. The Fish program was used to adjust rocks entering the plastic failure state into a strain softening model to investigate the influence of the face length effect, the damage degree of the water-resisting floor, and the morphology and deformation bearing capacity of the failure zone. The results show the following: (1) the face length effect is one of the main influence factors of the failure mode and failure degree of surrounding rocks in the stope; (2) as the face length increases, the obvious pressure relief zone of surrounding rocks presents a staged change, and the obvious pressure relief zone at the seam roof and floor is in an obvious “reverse saddle shape”; (3) the closer to the seam floor, the more remarkable the rock softening characteristic because of the compaction action of gangues caving from the roof; and (4) the rock mass close to the seam floor undergoes local tensile failure, and the water-resisting floor near the coal wall at two sides mainly bears compaction-shear action, leading to compression-shear failure of the rock mass at the floor and formation of water-conducting fractures. The study results can provide a reference for taking precautionary measures of safety mining above a confined aquifer.

1. Introduction

The Taiyuan Formation limestone aquifer and Ordovician limestone aquifer are widely distributed in the coal seam floor of coal measures in North China. However, due to the relatively poor mechanical strength of the aquifer medium, and the large scale of the underground mining stope space, the water hazard safety problem of the stope floor under the influence of mining is very prominent. The risk of the

water inrush from the coal seam floor is closely related to the degree of full exploitation. Mining activities destroy the balance of the original ground stress. Then, the surrounding rock stress in the stope will then undergo redistribution, leading to interactions between permeability characteristics of the rock mass medium and stress state, thereby forming water-rock coupling action [1]. The surrounding rocks in the stope will experience deformation and failure under the action of a new stress field. The rock masses will suffer from fracturing

expansion, shear deformation, and displacement because of fracturing, extruding, and softening and corroding actions of confined water. Consequently, the porosity and connectivity of structural planes in the rock mass will be enhanced to strengthen the permeability of rock masses and groundwater inrush pathways can be easily formed [2].

Various factors influence the floor stability in a stope. Many Chinese and foreign scholars have conducted investigation work on influence factors such as floor lithology, water-resisting floor thickness, water pressure, face length, structure, and coal seam occurrence state, analyzing the load-carrying characteristics and deformation failure laws of the stope floor with one factor of combining multiple factors and obtaining important research results [3–8]. Wang et al. [9] used the discrete element fluid-structure interaction (FSI) simulation method to perform simulation analysis of the evolution laws of floor stress and seepage during the coal mining process above a confined aquifer. Zheng et al. [10] established a jointed rock mass seepage-damage coupling model and applied it to the prediction of water inrush disasters in mines. The studies carried out by other scholars [11–16] have deepened the understanding of the breeding and evolution process of the water inrush from the seam floor and provided a certain scientific basis for the prevention and treatment engineering practice of the water inrush from the seam floor. Most existing research is conducted on the theoretical basis of the traditional elastic-plastic mechanics or phenomenological damage mechanics, resulting in imperfect simulation of rock mesostructural damage and permeability evolution arising out of this. Thus, in-depth studies need to be implemented on damage, rupture, permeability change, and formation of the water inrush pathway of rock masses at the seam floor under different mining conditions.

The increasing working face length will undoubtedly enlarge the scope of mining influence and intensity of mine pressure behaviors, deepen the damage scope of the floor, and effectively reduce the thickness of water-resisting strata. Therefore, the working face length is one of the key factors influencing the failure depth of the stope floor. However, when geological conditions are unchanged, the mine pressure behaviors under different face lengths and the influence mechanism on the floor stability remain to be further discussed. Thus, a strain softening model was introduced into the floor failure zone based on hydrogeological conditions in Group A coal mining of the Panxie Coal Mine of the Huainan Mining Group to investigate the influence mechanism of face length effect on floor stability.

2. Stress-Bearing Zoning and Failure Mechanism of the Floor

The coal seam and surrounding rock are in a stress balance state under the primary rock stress state, while the excavation of the coal seam or rock strata will damage the stress balance state and cause the redistribution of surrounding rock stress, finally leading to deformation and failure [17]. Affected by the forward supporting pressure, the coal seam and floor within a certain distance from the front of the working face are located in a pressurization zone, so the seam floor at this

location is compressed. This zone is called the compression zone (Zone B in Figure 1). As the working face continues to advance, the area from the working face to the rear goaf is in a pressure relief zone. The floor rocks of this part are converted from a compression state into an expansion state. The floor heave will be generated on the working face, and cracks of rock bedding stratification will appear. This zone is called the expansion and pressure relief zone (Zone D in Figure 1). Between Zones B and D, the rock strata within this scope are gradually transitioned from a compression state into a pressure relief state. The deformation state is also transitioned from compression into expansion. This area is called a transition zone (Zone C in Figure 1). After the working face is advanced again, the rock masses caving in the goaf are compacted by the overlying strata, and this part of the floor is located within the new pressurization zone again due to the overburden pressure. Then, it is converted from an expansion state into a compaction state again (Zone E in Figure 1).

Each point at the floor rock strata will undergo the process of “compaction-stress relief-recompression” with the advancement of the working face [18]. The crack ratio in the floor rock strata is changed due to the action of these stresses. Three crack types, namely, vertical tension, bedding, and shear, arise. Accordingly, this part of floor strata loses its water-resisting ability, and the floor failure depth refers to the depth of rock strata, causing the failure of the seam floor with a certain depth and water-conducting ability due to the combined action of factors such as mine pressure during the mining process of the working face.

3. Permeability Test of Floor Strata

3.1. Site Conditions. The A1 and A3 seams are mainly mined in Group A of the Panxie Mining Area of the Huainan Mining Group, and the average dip angle is 10° ; the coal thickness at the A1 seam is within 1.56–7.77 m (average value: 2.8 m); the coal thickness at the A3 seam is 2.09–9.17 m (average value: 5.8 m). The spacing between the two coal seams is 1–5 m, and they are combined into one seam at the local part. Strong Taiyuan Formation limestone and Ordovician limestone aquifers occur at the seam floor, where the Taiyuan Formation limestone aquifer C_3^3 at 28.1 m away from the A3 seam floor is a strong aquifer, resulting in potential water-induced damage to coal mining of Group A. The thickness and water pressure of the C_3^3 aquifer are 7.8 m and 4.5 MPa, respectively. The positional relationship of the working face and lithological histogram is shown in Figure 2.

3.2. Permeability Test. The test rock specimens, namely, sandstone, mudstone, sandy mudstone, and limestone at the floor of Coal Group A1, were collected from the field. Each rock was processed into five blocks, and standard specimen dimensions were obtained: height $H = 100$ mm and sectional diameter $d = 50$ mm. The instantaneous permeability test of the rock specimens was conducted on an MTS816 electrohydraulic servo rock mechanics experimental system. The test principles are shown in Figures 3 and 4.

The volumes of both pressure stabilizers in the pore pressure system are B ; their pressures are p_1 and p_2 , respectively;

FIGURE 1: Simulated diagram of the coal seam floor failure.

FIGURE 2: Working face arrangement and lithologic column.

and the height and cross-sectional area of each rock specimen are H and A , respectively. As the pressures at two ends of each rock specimen are different at the initial time ($p_{10} > p_{20}$), the pressure gradient $\xi_0 = (p_{20} - p_{10})/H$ exists, and the liquid in water tank 1 enters water tank 2 through the rock specimen. Then, the pressure in water tank 1 is continuously lowered while that in water tank 2 is continuously elevated until the pressures in the two water tanks become equal and reach an equilibrium state. The mass flow of liquid entering the rock specimen from water tank 1 is set as q . If the pore water of the rock specimen is saturated, then the mass flow of liquid flowing into water tank 2 from the rock specimen is also q , and the seepage velocity in the rock specimen is $V = q/\rho A$. According to the fluid compressibility, the following can be obtained:

$$\frac{1}{c_f} = \rho \frac{dp_1}{d\rho}. \quad (1)$$

The relational expressions $d\rho = -qdt/B$ and $q = \rho AV$ indicate that

$$\frac{dp_1}{dt} = -\frac{AV}{c_f B}. \quad (2)$$

Similarly,

$$\frac{dp_2}{dt} = -\frac{AV}{c_f B}. \quad (3)$$

The following can be obtained through (2) and (3):

$$\frac{d(p_2 - p_1)}{dt} = 2 \frac{AV}{c_f B}, \quad (4)$$

$$V = \frac{c_f B H}{2A} \frac{d\xi}{dt}, \quad (5)$$

FIGURE 3: MTS816 rock mechanics permeability experimental system.

FIGURE 4: Mechanical model of the transient permeation experimental system.

where ξ is the pressure gradient of the rock specimen, namely, $\xi = (p_2 - p_1)/H$.

For Darcy's flow, the relationship between the seepage velocity and the pressure gradient follows Darcy's law, that is,

$$\xi = -\frac{\mu}{k_D} V, \quad (6)$$

where μ is the dynamic viscosity of seeping liquid and k_D is the permeability of Darcy's flow in the rock specimen.

Equation (5) is substituted into Equation (6) to obtain

$$\frac{d\xi}{dt} = -2 \frac{Ak_D}{c_f BH \mu} \xi. \quad (7)$$

The sampling time interval and the total number of sampling times were set as τ and n , respectively, in the test. The pore pressure gradient is ξ_f at the sampling ending time $t_f = n\tau$. We use the integral of Equation (7) to obtain

$$\ln \frac{\xi_0}{\xi_f} = 2 \frac{Ak_D t_f}{c_f BH \mu}. \quad (8)$$

In Equation (8), pressure gradients ξ and ξ_0 are negative, ξ_0/ξ is positive, so $\ln(\xi_0/\xi)$ is significant. The permeability

and permeability coefficient of the rock specimen can be obtained through Equation (8):

$$k_D = \frac{c_f BH \mu}{2t_f A} \ln \frac{\xi_0}{\xi_f} = \frac{c_f BH \mu}{2t_f A} \ln \frac{p_{10} - p_{20}}{p_{1f} - p_{2f}}, \quad (9)$$

$$K = \frac{k_D \gamma}{\mu}, \quad (10)$$

where γ is the specific gravity of the liquid.

Equations (9) and (10) are formulas used to calculate the permeability K_D and permeability coefficient K in the stress-strain whole-process instantaneous permeability test of the rock specimens, which was performed on an MTS816 rock mechanics experimental system.

To guarantee the sealing property of the rock specimen, the stains on cylindrical surfaces of the rock specimen, pressure head, and porous disc were wiped away first. Then, these cylindrical surfaces were intertwined with a layer of plastic insulating tapes in a spiral form from bottom to top. A segment of the thermal-shrinking plastic jacket was cut off to cover rock specimens, porous discs, and upper and lower pressure heads. The plastic jacket was uniformly baked using an electric blower so that the plastic jacket fitted in well with the insulating tape. Information like experimental parameters, components, and number of rock specimens was recorded on the insulating tape. Finally, insulating tapes were

FIGURE 5: Sealing of rock sample.

used to twine the upper and lower parts of the plastic jacket. Rock specimens, pressure heads, and porous discs were encapsulated together using the plastic insulating tape and thermal-shrinking plastic jacket. Sealing of the rock sample is shown in Figure 5.

3.3. Analysis of Test Results. In the test, the confining pressure p_c was set as 4 MPa. The seeping liquid was water, where its mass density ρ , dynamic viscosity μ , and compression coefficient c_f were $1,000 \text{ kg/m}^3$, $1.01 \times 10^{-3} \text{ Pa} \cdot \text{s}$, and $0.472 \times 10^{-9} \text{ Pa}^{-1}$, respectively. The volume of pressure stabilizer B was $0.32 \times 10^{-3} \text{ m}^3$. According to the time series of the pore pressure difference acquired in the test, the permeability coefficient K of Darcy's flow is calculated according to Equation (10).

3.3.1. Whole Stress-Strain Permeability Characteristics of Sandstone. Figure 6 provides the strain ε -dependent change laws of the sandstone permeability coefficient K . The data analysis indicates the following. (1) In the elastic deformation phase, a small number of microcracks in the sandstone specimen start presenting tensile deformation as the strain increases. Moreover, the porosity is elevated, the permeability coefficient K is gradually enlarged, but the increased rate of K is low in this phase. (2) After the strain satisfies $\varepsilon > 0.0106$, the permeability coefficient K abruptly increases and reaches the maximum value: $K_{\max} = 278.02 \times 10^{-11} \text{ m} \cdot \text{s}^{-1}$. The increased amplitude reaches 4~20 times that in the elastic deformation phase. In reality, the cracks in the specimen are expanded, fused together, and run through each other. An evident damaged macrostructural plane is already formed. When the strain satisfies $\varepsilon > 0.0118$, the permeability coefficient K declines rapidly, and the decreased amplitude is approximately 50%. (3) Under $\varepsilon > 0.0133$, the permeability coefficient K is decreased slowly and stabilized at $K_{\max} = 124.83 \times 10^{-11} \text{ m} \cdot \text{s}^{-1}$, approximately. In this phase, some large cracks and the space of the damaged structural plane are filled by small crushed particles as the failure degree is aggravated. Consequently, the porosity and the permeability

FIGURE 6: Variation curve of the permeability coefficient K with strain ε .

coefficient K decline. The confining pressure p_c also exerts a limiting effect on crack expansion.

3.3.2. Mudstone Whole Stress-Strain Permeability Characteristics. Figure 7 shows the change laws of the mudstone permeability coefficient K with strain ε . The data analysis indicates the following. (1) Relative to sandstone and limestone, the permeability coefficient K of mudstone is small, meaning that mudstone has better water-resisting performance than sandstone and limestone. (2) Similar to sandstone and limestone, the permeability coefficient K of the mudstone specimen increases slowly in the elastic deformation phase with the increase of strain. It increases rapidly when approaching the peak deformation point and tends to be a steady value in the residual deformation phase. (3) During the entire stress-strain permeability test process, the change amplitude of the permeability coefficient K of the mudstone specimen is not large, only being $K = 0.24 \times 10^{-13} \text{ m} \cdot \text{s}^{-1} \sim 68.79 \times 10^{-13} \text{ m} \cdot \text{s}^{-1}$. The permeability coefficient K in the residual deformation phase is stabilized at approximately $31.28 \times 10^{-13} \text{ m} \cdot \text{s}^{-1}$. (4) The strain of the mudstone specimen is larger than those of sandstone and limestone when the maximum permeability coefficient $K_{\max}$ is reached. $K_{\max}$ is reduced by three orders of magnitude than those of sandstone and limestone.

3.3.3. Whole Stress-Strain Permeability Characteristics of Sandy Mudstone. The change laws of the permeability coefficient K of sandy mudstone with strain ε are presented in Figure 8. The curve analysis shows the following. (1) For the sandy mudstone specimen, the change law of its permeability coefficient K is similar to that of mudstone on the whole as the strain ε increases. (2) The permeability coefficient K of the sandy mudstone specimen is slightly larger. During the entire stress-strain permeability test, the change amplitude of its permeability coefficient K is not large, only being $K = 2.84 \times 10^{-12} \text{ m} \cdot \text{s}^{-1} \sim 61.37 \times 10^{-12} \text{ m} \cdot \text{s}^{-1}$. (3) The strain ε of the sandy mudstone specimen is approximate to that of mudstone when the maximum permeability coefficient $K_{\max}$ is reached, indicating approximate plasticity of sandy mudstone and mudstone.

3.3.4. Limestone Whole Stress-Strain Permeability Characteristics. Figure 9 shows the change laws of the

FIGURE 7: Variation curve of the permeability coefficient K with strain ε . FIGURE 8: Variation curve of the permeability coefficient K with strain ε . FIGURE 9: Variation curve of the permeability coefficient K with strain ε .

permeability coefficient K of limestone with strain ε . The curve analysis indicates the following. (1) The permeability coefficient K of the limestone specimen is reduced by 50% in comparison with that of sandstone, manifesting better compactness of limestone than sandstone. (2) The permeability coefficient K of the limestone specimen is slightly increased in the elastic deformation phase as the strain ε increases, indicating few effective pores in the rock specimen. Only a small number of microfractures are under tensile deformation. (3) When the strain satisfies $\varepsilon > 0.0063$, the permeability coefficient K is increased rapidly and reaches the maximum value $K_{\max} = 134.82 \times 10^{-11} \text{ m} \cdot \text{s}^{-1}$. At this time, macrocracks exist in the rock specimen. (4) Similar to sandstone, the permeability coefficient K of the limestone specimen is in a rapid decline process under the condition of

$\varepsilon > 0.0084$, but it is finally stabilized at approximately $K_{\max} = 66.89 \times 10^{-11} \text{ m} \cdot \text{s}^{-1}$.

4. Numerical Analysis of Face Length Effect under Water-Rock Coupling Action

4.1. Mechanical Model of the Floor Failure Zone. During the FLAC3D numerical calculation process, the stress-strain curve is linear before the element reaches a yield point, and only elastic strain ε^e is generated in this phase, namely,

$$\varepsilon = \varepsilon^e, \quad (11)$$

where ε is the total element strain.

After the element yields, the total strain consists of elastic strain ε^e and plastic strain ε^p , namely,

$$\varepsilon = \varepsilon^e + \varepsilon^p. \quad (12)$$

The subsequent yield surface of the rock is related not only to the instantaneous stress state but also to the plastic deformation history. If the rock is regarded as the isotropic material and equivalent plastic shear strain ε^{ps} as the parameter recording the plastic loading history of rock material, then the postpeak subsequent yield surface [19] can be expressed as

$$\phi(\sigma_1, \sigma_2, \sigma_3, \varepsilon^{ps}) = 0, \quad (13)$$

TABLE 1: Physicomechanical parameters of rock layers.

Lithology	Density (kg/m ³)	Elastic modulus (GPa)	Poisson's ratio	Cohesion (MPa)	Friction angle (°)	Tensile strength (MPa)
Sandy mudstone	2,467	1.6	0.36	1.8	30	0.8
Siltstone	2,531	3.2	0.32	2.3	31	1.0
Grit	2,602	8.3	0.21	5.3	35	2.1
Coal	1,320	1.0	0.39	1.2	26	0.4
Mudstone	2,368	1.3	0.35	1.6	31	0.7
Fine sandstone	2,545	7.2	0.24	4.2	32	2.2
Limestone	2,680	6.4	0.26	4.4	33	2.1

where σ_1 , σ_2 , and σ_3 are the principal stresses in three directions.

When the rock is under a peak state, it satisfies the Mohr–Coulomb criterion. When it is under the postpeak state, the characteristic parameters (namely, cohesion and internal friction angle) used to describe this state will be changed. Therefore, two parameters, which can reflect the postpeak strain softening characteristics, that is, generalized cohesion $\bar{c}$ and generalized internal friction angle $\bar{\varphi}$, are introduced to describe the stress level on the Mohr–Coulomb yield surface. The subsequent yield criterion can be expressed by Equation (14), as follows:

$$f = \sigma_1 - \sigma_3 \frac{1 + \sin \bar{\varphi}(\Delta \epsilon^{ps})}{1 - \sin \bar{\varphi}(\Delta \epsilon^{ps})} + 2\bar{c}(\Delta \epsilon^{ps}) \sqrt{\frac{1 + \sin \bar{\varphi}(\Delta \epsilon^{ps})}{1 - \sin \bar{\varphi}(\Delta \epsilon^{ps})}}, \quad (14)$$

$$\Delta \epsilon^{ps} = \left\{ \frac{1}{2} (\Delta \epsilon_1^{ps} - \Delta \epsilon_m^{ps})^2 + \frac{1}{2} (\Delta \epsilon_m^{ps})^2 + \frac{1}{2} (\Delta \epsilon_3^{ps} - \Delta \epsilon_m^{ps})^2 \right\}^{1/2}, \quad (15)$$

where $\Delta \epsilon_m^{ps}$ is the main increment of plastic shear strain, $\Delta \epsilon_m^{ps} = (1/3)(\Delta \epsilon_1^{ps} + \Delta \epsilon_3^{ps})$, $m = 1, 2, 3$.

4.2. Water-Rock Coupling Model. When the FSI mechanism of the rock mass is simulated using FLAC3D, the rock mass is regarded as a porous medium, and the fluid flow in the porous medium conforms to Darcy's law and satisfies the Biot FSI equation as follows:

$$\begin{cases} G \cdot \cdot u_j - (\lambda + G) \frac{\partial \epsilon_v}{\partial x_j} - \frac{\partial p}{\partial x_j} + f_{xj} = 0, \\ K \cdot \cdot p = \frac{1}{S} \frac{\partial p}{\partial t} - \frac{\partial \epsilon_v}{\partial t}, \end{cases} \quad (16)$$

where λ and G are the lame constants; p is the pore water pressure; ϵ_v is the volume strain; x_j , u_j , and f_{xj} , $-j$ are the directional coordinates, displacement, and body force, respectively; $\cdot \cdot$ is the Laplace operator; and S is the elastic storativity.

Equation (16) is an expression based on the Biot classical percolation theory, where $\partial p / \partial x_j$ reflects the influence of the seepage field on a solid skeleton, and its essence is that the pore pressure generated in the fluid flow influences the effective stress of a solid skeleton, influencing its deformation. ∂

$\epsilon_v / \partial t$ expresses the influence of volume deformation of a solid skeleton on the seepage field. The classical Biot equation can characterize the interaction between pore pressure dissipation and deformation of the medium skeleton very well.

4.3. Establishment and Analysis of a Numerical Model. The A1 and A3 seams are mainly mined in Group A of the Panxie Mining Area of the Huainan Mining Group, and the average dip angle is 10°; the coal thickness at the A1 seam is 2.8 m; the coal thickness at the A3 seam is 5.8 m. The spacing between the two coal seams is 1~5 m; Strong Taiyuan Formation limestone and Ordovician limestone aquifers occur at the seam floor, where the Taiyuan Formation limestone aquifer C_3^3 at 28.1 m away from the A3 seam floor is a strong aquifer, resulting in potential water-induced damage to coal mining of Group A. The thickness and water pressure of the C_3^3 aquifer are 7.8 m and 4.5 MPa, respectively. The dimensions of the established model were 300 m (length) $\times$ 300 m (width) $\times$ 212 m (height). The model was divided into 210,112 elements. The Mohr–Coulomb yield criterion was used, and the attribute of strain softening was judged and assigned via Fish language after the floor rock reached yield failure. The model boundary conditions were as follows. Front, rear, left, and right boundaries were fixed in directions x and y , and the bottom was a full-constrained boundary, and the top boundary was set as stress constraint (the top boundary stress is 12 MPa, the buried depth is 520 m, and the average rock weight is 2500 kg/m³). The boundary condition of seepage was as follows. A boundary with fixed water pressure was used at the bottom to simulate confined water of the limestone aquifer, and the others were water-resisting boundaries. The goaf was a water drainage boundary after mining of the working face. No water was present in the goaf. The fixed water pressure was 0 at the boundary, and physical and mechanical parameters of rock strata are seen in Table 1. The stress, displacement, and plastic zone distribution laws of the water-resisting floor during the A3 coal mining under face lengths of 120, 160, and 200 m were analyzed. The interface is set in the roof and floor of the coal seam to avoid model zone embedding; the mining simulation of the working face adopts the method of gradual excavation to simulate the mining process of the working face, and the excavation step is taken as the periodic mining press step.

4.4. Simulation Result

4.4.1. Maximum Principal Stress. After the coal mining, the rocks within a certain scope of the roof and floor will undergo

FIGURE 10: Stress distribution of rock.

pressure relief to different degrees. As the mining size is enlarged, the scope of influence of coal mining on surrounding rocks in the stope will also be enlarged, and the pressure relief degree and morphology of the pressure relief zone will present staged features.

As shown in Figure 10(a), the stope roof presents a “semi-circular arch” pressure relief zone under a face length of 120 m, and the maximum principal stress in the zone is $-2.0 \sim -0.12$ MPa, where the roof strata near two ends of the stope are tensioned while other scopes are compressed. The pressure relief zone at the stope floor is divided into upper

and lower parts by using the aquifer as the boundary, where the upper pressure relief zone is in a “basin shape.” In particular, an obvious pressure relief zone appears within the 6–7 m scope near the stope floor. The maximum principal stress value is $-2.0 \sim 0$ MPa. This obvious pressure relief zone is under an obvious “reverse saddle shape.”

Figure 10(b) shows that the pressure relief zone at the stope roof is “saddle-shaped” under a face length of 160 m and an advancement distance of 160 m. Under compaction of gangues caving from the coal seam roof, the pressure in a large scope at the middle of the stope is increased to

FIGURE 11: Floor movement and deformation curve.

−3.0~1.0 MPa, and obvious pressure relief takes place at the position 11.0~14.0 m away from the roof at two ends of the stope. The pressure value is within −1.0~0.04 MPa.

Corresponding to the stope roof, the pressure relief zone at the seam floor is in a “reverse saddle shape,” where the stress close to the middle position of the stope is higher than those at two sides.

The situation under a face length of 200 m is shown in Figure 10(c). Under the actions of water pressure and mining stress, the relative movement deformation of rock strata at the roof and floor is enlarged, and a compaction zone is formed within a certain scope in the middle of the working face. Two stress arches are formed in the large pressure relief zone of the roof and floor of the stope, and the arch angles are located in the compaction zone between the floor and the goaf, presenting an even more obvious “saddle shape.” The stress inside the obvious pressure relief zone at the floor ranges from −1.0 MPa to 0.07 MPa, being within the 14.0~18.0 m scope of the floor.

4.4.2. Floor Deformation. The influence scope of coal mining on the floor is enlarged with the increase of the face length, so

the movement deformations at different rock strata in the floor are enlarged. Due to the water-rock coupling action, the displacement laws of upper and lower rock strata in the aquifer of the seam floor will be analyzed in the simulation, as shown in Figure 11.

After coal mining, the entity coal side is compressed, the deformation is turned into a negative value, the floor rock strata in the stope undergo pressure relief, and the floor heaves are formed, so the deformation value becomes positive. As the face length increases, the heave deformation is gradually enlarged. For example, at 5.0 m from the floor, the maximum heave deformations under face lengths of 120, 160, and 200 m are 83, 148, and 786 mm, respectively.

Based on the rock strain softening characteristics in the floor failure zone, the closer to the seam floor, the more obvious the rock softening characteristics due to the compaction of gangues caving from the roof. Under a face length of 120 m, as shown in Figure 11(a), the deformation curve at 5.0 m from the floor is slowed down somehow within a certain scope in the middle of the stope. The heave deformation is higher than the 15.0 m displacement curve of the floor by 13 mm, indicating that the failure depth is over 5.0 m. The failure depth at the

FIGURE 12: Plastic zone distribution.

floor is approximately 15.0 m under a face length of 160 m, as shown in Figure 11(b). At the time, the deformation curves at 5.0 and 15.0 m from the floor are slowed down within a certain scope at the middle of the stope, which are lower than the corresponding positions of the 25.0 m displacement curve by 21 and 12 mm, respectively. Therefore, the obvious failure depth of the floor is 15.0~25.0 m under a face length of 200 m, as shown in Figure 11(c).

Under the water-rock coupling action, the influence degree borne by the rock strata below the aquifer is relatively low, and the displacement quantity is within 20~20 mm. Under a face length of 120 m, the deformation in the corresponding floor zone in the stope is turned positive. However, when the face length is 160~200 m, the deformation in the middle of this zone

becomes negative while those at the two sides are positive. The deformation curve is under the wavy line shape.

The displacement and deformation curves of different strata of the roof under different face lengths indicate that the 30.0~45.0 m scope at two sides of the stope is a roof and floor unobvious contact zone. Other scopes constitute the zone of significant roof/floor interaction.

4.4.3. Distribution of the Plastic Zone. Following the coal mining, the load-bearing state of the water-resisting floor is turned from compression into tension. As the advancement distance of the working face is continuously increased, the tensile action borne by the rock mass close to the seam floor within a certain scope at the rear of the stope becomes

FIGURE 13: Depth of floor failure by the electrical method test.

increasingly obvious. Local tensile failure will occur when the tensile stress exceeds the tensile strength of the rock mass. The water-resisting layer of the seam floor near the coal wall at two sides of the stope mainly bears compression-shear action. The compressive stress is also elevated as the advancement distance increases. As a result, the compression-shear action strength is continuously reinforced, and the rock mass at the floor will undergo compression-shear failure when the strength of the rock mass is exceeded.

FLAC3D judges whether the element enters plastic deformation via the Mohr–Coulomb criterion, but the plastic failure of rock strata does not indicate the formation of water-conducting fractures. The strain softening characteristic was assigned to the plastic deformation element. Whether the element had water-conducting ability after secondary failure was judged through Equation (4). The simulation analysis manifests that under a face length of 120 m, water-conducting zones are formed at the 5.6 m tensile failure depth at the middle of the floor and 8.8 m compression-shear failure depth at two ends, as shown in Figure 12(a). When the face length is 160 m, the tensile failure depth at the middle of the stope does not obviously change. The compression-shear failure depth at two ends is elevated to 16.2 m, and the communication trend with the floor aquifer becomes evident, as shown in Figure 12(b). With the increase in the face length, the influence scope borne by the floor from coal mining is enlarged. Under the face length of 200 m, the tensile failure zone at the middle of the stope is extended downward,

reaching 13.2 m, and the compression-shear depth at two ends becomes 22.3 m, as shown in Figure 12(c).

5. Engineering Verification

The 11123 working face of Panji Coal Mine 2 is the working face of the Group A coal mine in the Panxie Mining Area. Here, the ground elevation is +19.5~+22.5 m and the elevation of the working face is -429.9~-497.5 m. The average thickness of the A3 coal seam is 5.5 m with a dip angle of 10°. It is classified as a stable coal seam. Influenced by the sliding and fault between coal seams within the scope of the working face, the face length 500 m forward from the cutting hole is 120 m, and the rear face length is increased to 200 m. To analyze the scope of the floor failure depth under different face lengths, the optical fiber and electrical integrated test system was laid by drilling holes on the floor to conduct continuous monitoring. The physical properties and strain change characteristics of the floor during the rock stratum mining process were acquired, followed by the analysis of failure depth. As shown in Figure 13, the floor failure depth is approximately 8.2 m under the face length of 120 m and approximately 21.6 m under the face length of 200 m. The measurement result accords with the numerical analysis result.

6. Conclusion

- (1) By analyzing the permeability characteristics of limestone, mudstone, sandy mudstone, and sandstone in

a coal mine, the numerical calculation model of floor stability under a confined aquifer is established. The strain softening characteristics are judged by using the Fish function, and then the mechanical behavior, bearing capacity, and deformation of floor rock are described, and the floor failure degree under different working face lengths is obtained; compared with the measured data, the feasibility of this method is verified

- (2) The face length effect is one of the main influence factors of the failure mode and failure degree of surrounding rocks in the stope. As the face length increases, the obvious pressure relief zone of surrounding rocks presents a staged change. When the size of the working face is short, the stope roof presents a “semicircular arch” pressure relief zone, and the obvious pressure relief zone within a certain scope at the stope floor is in an unobvious “reverse saddle shape.” When the size of the working face is long, the obvious pressure relief zone at the seam roof and floor is in an obvious “reverse saddle shape” due to the compaction action of gangues caving from the seam roof
- (3) Based on the strain softening characteristic of rocks in the roof failure zone, the closer to the seam floor, the more remarkable the rock softening characteristic because of the compaction action of gangues caving from the roof. The floor deformation characteristic corresponds to the stress distribution in the floor pressure relief zone. The 30–45 m scope at two sides of the stope is a roof/floor unobvious contact zone, and other scopes constitute a zone with significant roof/floor interaction
- (4) After the coal is mined, the rock mass close to the seam floor undergoes local tensile failure, and the water-resisting floor near the coal wall at two sides mainly bears compaction-shear action. The influence scope of mining is enlarged with the increase of the face length, leading to compression-shear failure of the rock mass at the floor and formation of water-conducting fractures

Data Availability

The others can access the data supporting the conclusions of the study from this research article. The nature of the data is the laboratory experimental data, the field observation data, and the theoretical calculation data. The laboratory experimental data used to support the findings of this study are included within the article; mainly, the mechanical parameters used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

Acknowledgments

The work presented in this paper is financially supported by the National Natural Science Foundation of China (No. 51674007) and the Natural Science Foundation of Chongqing (cstc2020jcyj-msxmX1052). The authors gratefully acknowledge the assistance of the Huainan Mining Group at the Anhui University of Science and Technology for providing access to the mine, data, and physical resources necessary to complete this work.

References

- [1] L. Xiaoli, “Study on the theory and application of multi-scale behavior of water-rock coupling,” Tsinghua University, Beijing, 2008.
- [2] F. Meimei and M. Xianbiao, “Experimental research on fracture evolution law of water-resisting strata in coal seam floor above aquifer,” *Chinese Journal of Rock Mechanics and Engineering*, vol. 28, no. 2, pp. 336–341, 2009.
- [3] T. Min, “Study on the growth height of separation fracture of mining rock in Panxie area,” *Journal of China Coal Society*, vol. 29, no. 6, pp. 641–645, 2004.
- [4] L. Dejun, “The stress distribution of the goaf and its relationship of water inrush,” *Safety in Coal Mines*, vol. 7, pp. 35–39, 1988.
- [5] G. Yingbin, L. Haimei, and L. Junchen, “Research of No.9 coal seam floor’s fracture regularity in Xiandewang Coal Mine,” *Journal of China Coal Society*, vol. 28, no. 2, pp. 121–125, 2003.
- [6] S. Longqing and H. Jin, “Floor water inrush mechanism and prediction,” China University of Mining and Technology Press, Beijing, 2004.
- [7] H. Weiyue and Y. Shangxian, “Dynamic mechanism of water inrush from floor of mining face,” *Chinese Journal of Rock Mechanics and Engineering*, vol. 29, no. 1, pp. 3344–3348, 2010.
- [8] Z. Yujun, L. Shucui, Q. Sifeng, and L. Liping, “A catastrophe model for floor water-resisting key stratum instability induced by dynamic disturbance,” *Rock and Soil Mechanics*, vol. 31, no. 8, pp. 2361–2366, 2010.
- [9] W. Jinan, W. Xianhao, and C. Shaojie, “Fracture and seepage characteristics in the floor strata when mining above a confined aquifer,” *Journal of China University of Mining & Technology*, vol. 41, no. 4, pp. 536–542, 2012.
- [10] Z. Shaohe, Z. Weishen, and W. Shufa, “Study on the coupling problem between flow and solid of mine in confined aquifer,” *Chinese Journal of Rock Mechanics and Engineering*, vol. 19, no. 4, pp. 421–424, 2000.
- [11] C. Jisheng, “Numerical simulation study on water inrush mechanism of working face under high confined water,” Shandong University of Science and Technology, Qingdao, 2006.
- [12] H. Yaoqing, Y. Guochao, and S. Xiuwe, “Study on physical and numerical simulation of water inrush prediction theory for coal mining above confined aquifer,” *Chinese Journal of Rock Mechanics and Engineering*, vol. 27, no. 1, pp. 9–15, 2008.
- [13] Y. Xiaoge, “Study on broken depth of damage floor,” Shandong University of Science and Technology, Qingdao, 2011.
- [14] Y. Qingdao, “The possibility of detecting water in the Ordovician carbonate with objective seismic exploration,” *Journal of China Coal Society*, vol. 27, no. 1, pp. 36–40, 2002.

- [15] J. Fuxing, L. Weijian, Y. Genxi, and L. Wei, "Coupling study of microseismic monitoring and numerical simulation for tectonic activation," *Chinese Journal of Rock Mechanics and Engineering*, vol. 29, no. A2, pp. 3590–3597, 2010.
- [16] J. Fuxing, Y. Genxi, W. Cunwen, Z. Dangyu, and G. Yongqiang, "Application of high-precision microseismic monitoring technique to water inrush monitoring in coal mine," *Chinese Journal of Rock Mechanics and Engineering*, vol. 27, no. 9, pp. 1932–1938, 2008.
- [17] L. Shucui, "Mechanism of water inrush from coal seam floor and continuous survey of fractured zones in coal seam floor," China University of Mining Technology, Xuzhou, 2008.
- [18] P. L. Gong, Y. Q. Hu, Y. S. Zhao, and D. Yang, "Three-dimension simulation study on law of deformation and breakage of coal floor on mining above aquifer," *Chinese Journal of Rock Mechanics and Engineering*, vol. 24, no. 23, pp. 4396–4402, 2005.
- [19] J. W. Zhou, W. Y. Xu, M. W. Li, X. Q. Zhou, and C. Shi, "Application of rock strain softening model to numerical analysis of deep tunnel," *Chinese Journal of Rock Mechanics and Engineering*, vol. 28, no. 6, pp. 1116–1127, 2009.