Experimental and Numerical Study of Rock Stratum Movement Characteristics in Longwall Mining

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The roof fracture is the main cause of coal mine roof accidents. To analyze the law of movement and caving of the roof rock stratum, the roof subsidence displacement, rock stratum stress, and the rock stratum movement law were analyzed by using the methods of the particle discrete element and similar material simulation test. The results show that (1) as the working face advances, regular movement and subsidence appears in the roof rock strata, and the roof subsidence curve forms a typical "U" shape. As the coal seam continues to advance, the maximum subsidence displacement remains basically constant, and the subsidence displacement curves present an asymmetric flat-bottomed distribution. (2) After the coal seam is mined, the overburden forms an arched shape forcechain, and the arched strong chain is the path of the overburden transmission force. The farther away from the coal seam, the smaller the stress concentration coefficient is, but it is still in a high stress area, and the stress concentration position moves toward the middle area of the goaf. The stress concentration in front of the coal wall is the source of force that forms the abutment pressure. (3) Above the coal wall towards the goaf, a stepped fracture was formed in the roof rock stratum. The periodic fracture of the rock stratum is the main cause of the periodic weighting of the working face. Understanding the laws of rock movement and stress distribution is of great significance for guiding engineering practice and preventing the roof accidents.

1. Introduction

China is a country with large coal production, and its coal production accounts for about half of the world's total production. There are six major disasters occurring in coal mines, namely, gas explosion, water inrush, roof caving, coal-gas outburst, transport, and poisoning. Among them, the roof accident accounted for 13.79% of the total accidents, and the death rate accounted for 14.98%, as shown in Figure 1. The roof fracture is the main cause of the coal mine roof accidents. Therefore, it is of great theoretical and practical significance to study the laws of the movement of the roof above the coal seam for preventing and controlling the occurrence of roof disasters.

For a long time, scholars at home and abroad mainly used theoretical calculation and physical simulation for analysis of roof fracture. Zhao and Liu [2] based on the engineering geology conditions of remaining face 3101 in Shenghua Mine in China analyzed the roof fracture and instability features of the roof caving zones through physical simulation, theoretical analysis, and field measurements. He et al. [3] built the structural and mechanical models to explore the mechanism of the new approach. The results indicate that effective bulking of the gob roof and reasonable support of the entry roof were key governing factors in improving entry stabilities and reducing roof deformations. Kong et al. [4] explored the stability of the roof at the end of the face by using theoretical analysis, numerical simulation, and field measurement. Huang et al. [5] analyzed the height of water-permeable fracture zone in overlying strata by the situ test. Wang et al. [6] analyzed the mechanical and acoustic emission characteristics of rock-like materials
under nonuniform loading by simulating the stress distribution of the coal body in front of the working face. Kang et al. [7] studied sudden massive roof collapse during longwall coal retreat mining through physical experiments. Wang et al. [8] studied dynamic structural evolution of overlying strata during shallow coal seam longwall mining. Zhu et al. [9] analyzed overburden movement characteristics of top-coal caving mining in multiseam areas. Zhang et al. [10] investigated the coal drawing from thick steep seam with longwall top-coal caving mining by experimental methods. These studies play an important role in understanding the roof rock fracture.

In recent years, with the developments of computer technology, the numerical method has become an important research method in the field of rock mass engineering. Numerical modeling is a promising and effective tool for the simulation of progressive caving of strata resulting from the longwall mining. Gao et al. [11] studied the coal longwall caving characteristics using an innovative UDEC Trigon approach. Hosseini et al. [12] analyzed the mechanism of roof caving in longwall mining by using Phase2 software of FEM. Yasitli and Unver [13] studied the top-coal caving mechanism by using the finite difference code FLAC3D. Islam et al. [14] analyzed the stress distributions for multislice longwall mining in Barapukuria coal mine of Bangladesh by using finite element modeling. Wang et al. [15] investigated the dynamic mechanical state of a coal pillar during longwall mining panel extraction by FLAC software. The research on roof movement and fracture law of longwall mining mainly focuses on using FLAC, UDEC, etc., while the research using PFC are relatively few. Particle flow code in 2 dimensions (PFC2D) is a discrete element method proposed by PA Cundall and ODL Strack to simulate the motion and interaction of circular particles [16, 17], which can solve the problems of solid mechanics and large deformation. At the same time, the multiple interacting deformable continuous, discontinuous, or fracturing bodies undergoing large displacements and rotations can be simulated and analyzed. It is widely used in geotechnical engineering, geological engineering, and mining engineering. Lu et al. [18] used the 3D discrete element method to study the unstable slope in the Lushan hot spring district, central Taiwan. Wang et al. [19] studied the height of the mining-induced fractured zone above a coal face by PFC and similar material experiment; however, there was little analysis of the rock stratum motion law of the overburden after coal seam mining, especially the stress and displacement of the rock stratum. Wang et al. [20] applied the discrete element method to study the characteristics of jointed rock masses. Liu et al. [21] simulated the damage evolution law of coal mine roadway by the particle flow code model. Cai et al. [22] studied AE characteristic in large-scale underground excavations by the FLAC/PFC coupled numerical simulation method. Wang et al. [23] used PFC2D to analysis the stability of the heavily jointed rock slope. An et al. [24] investigated the dynamic characteristics of inelastic rock impact using the discrete element method contact model. Numerical modeling has been used to investigate a variety of problems in underground mining and tunneling, subsidence induced by longwall coal mining, stresses generated when an open stope is filled cemented backfill, and the stability of exposures created during subsequent mining of adjacent stopes [25].

With an overview of the previous studies found that there are few studies which used particle flow code and similar material simulation test to analyze the laws of movement and fracture of the roof rock stratum in the longwall mining. Therefore, this paper uses PFC2D software and similar material simulation test to establish the analysis model of longwall mining. The roof subsidence displacement, rock stratum stress, and the rock stratum movement law will be analyzed. This research will provide more information about rock stratum movement laws for engineers.

Figure 1: Large accidents in China’s coal mines in 2016 [1]. (a) Percentage of different structure types of large accidents. (b) Percentage of deaths of different structure types of large accidents.
2. Similar Material Experiment

2.1. Similarity Principle and Similarity Coefficient. Similar material experiment is an experimental model that imitates the prototype and copied by following a certain proportional relation [26]. And model experiments and prototypes need to meet certain similarity proportional relations. The similar material experiments, taking the onsite geological conditions as the prototype on the basis of the similarity theory, design the corresponding experimental model to analyze the stress and displacement distribution characteristics of the overlying surrounding rock during the coal seam mining process and the movement law of the rock stratum, so as to provide a basis for the surrounding rock management and mining design of the working face.

It is well known that the similarity coefficient is very important for the simulation test. If the similarity coefficient is too large, the model will be larger and cost more. If the similarity coefficient is too small, some monitoring devices are difficult to install and physical parameters of similar materials are inaccurate [26, 27]. In similar material experiments, the geometry similarity coefficient of model to prototype is 1/100. Based on similar principles, the similarity coefficient that needs to be met are as follows.

(1) Geometric similarity coefficient \( (C_L) \)

Set the prototype size to \( L_p \) and the corresponding size of the experimental model to \( q_{m1} = C_L \times C_t \times y_p \times (H - H_m) \). Then, the geometric similarity coefficient is described by the following expressions:

\[
C_L = \frac{L_m}{L_p} = 1 : 100. \tag{1}
\]

(2) Time similarity coefficient \( (C_t) \)

To the prototype and the experimental model, the gravitational acceleration is equal, and thus the time similarity coefficient can be obtained through the following equation:

\[
C_t = \sqrt{C_L} = 1 : 10. \tag{2}
\]

(3) Bulk density similarity coefficient \( (C_\gamma) \)

Base on the rock stratum bulk density, the bulk density similarity coefficient is

\[
C_\gamma = \frac{\gamma_m}{\gamma_p} = \frac{1}{1.67}. \tag{3}
\]

(4) Stress similarity coefficient \( (C_\sigma) \)

Based on the similarity theory and \( \sigma = \gamma \cdot l \), the stress similarity constant can be calculated by substituting the bulk density similarity constant and geometric similarity coefficient into the following equation:

\[
C_\sigma = C_\gamma \times C_t = \frac{1}{1.67} \times 1 = \frac{1}{167}. \tag{4}
\]

(5) Poisson’s ratio similarity coefficient \( (C_\mu) \)

Set the rock stratum Poisson’s ratio of the prototype to \( \mu_p \) and the corresponding rock stratum Poisson’s ratio of the experimental model to \( \mu_m \). Then, Poisson’s ratio similarity coefficient can be described by the following equation:

\[
C_\mu = \frac{\mu_m}{\mu_p} = 1. \tag{5}
\]

(6) Elasticity modulus similarity coefficient \( (C_E) \)

Set the rock stratum elasticity modulus of the prototype to \( E_p \) and the corresponding rock stratum elasticity modulus of the experimental model to \( E_m \). Then, the elasticity modulus similarity coefficient is as follows:

\[
C_E = \frac{E_m}{E_p} = C_L \times C_t = \frac{1}{167}. \tag{6}
\]

2.2. Engineering Background and Similar Material Experiment Model. Take the geological conditions of the 1232(1) fully-mechanized mining face of Xieqiao Coal Mine of Huainan Mining Group in AnHui Province in China as the engineering background, as shown in Figure 1. The 1232(1) workface is from east to Group C uphill, west to F5 fault, north to −560 m track alley, and south to 1242(1) return air laneway. 1232(1) workface elevation is −540.6−604.3 m, and the ground level is +20.5−29.5 m. The average coal seam inclination angle is 13°, and the coal volume weight is 13.71 kN/m\(^3\), and the coal seam average thickness of 2.91 m. The main roof is fine sandstone, gray to light gray. The sandstone’s cementation matter is mainly mud, with an average thickness of 8.64 m. The main roof is fine sandstone, light gray, occasionally a small amount of fissions, calcite filling, with an average thickness of 4.57 m. The lithology comprehensive strata diagram based on the field drill hole are shown in Figure 2. The main mechanical parameters of different lithologies measured by onsite in situ tests and indoor mechanical tests are shown in Table 1.

In order to analyze the law of fracture movement of roof rock, similar material experiments were conducted based on the actual engineering geological conditions. The similar material experiments used a plane stress model with the size of length × height × width = 420 cm × 200 cm × 25 cm. In this experiment, the working face advancing distance is 200 cm, and 110 cm boundary coal pillars are set at both ends of the simulation experiments bench in order to eliminate the model boundary effect. For the unsimulated overburden, an equivalent stress was applied to the top of model. The experimental model and displacement monitoring points are shown in Figure 3. During the experiment, the movement of the rock stratum was photographed by the camera, and the subsidence displacement of the roof rock stratum was monitored by total station instrument.
Coal: black, block, semidark semibriar coal, locally developed 1 layer of mudstone, thickness 0.3–3.3m.

Mudstone: dark gray, massive, a small amount of plant fossil fragments, a thin layer of coal or carbonaceous mudstone on top.

Mudstone: dark gray, block, with carbonaceous components, a small amount of plant fossil fragments, thickness 0.4–4.24m.

Mudstone: gray to dark gray, massive, partially sandy, with a thickness 1.13 to 7.37m.

Mudstone: gray, blocky-thin layered, brittle, harder, thickness 4.5~13.50m.

Sandy mudstone: gray, blocky, muddy cemented, hard, with a small amount of siderite and charcoal, thickness 1.41~6.25m.

Fine sandstone: gray, mainly quartz, dense, and hard, with a thickness 3.15~12.66m.

Silty sandstone: gray, blocky, muddy cemented, hard, with a small amount of siderite and charcoal, thickness 1.41~6.25m.

Mudstone: dark gray, massive, containing a small amount of plant fossil fragments, thickness 0~4.7m.

Mudstone: dark gray, massive, a small amount of plant fossil fragments, thickness 0.4~4.24m.

Mudstone: gray, mainly quartz, dense, and hard, with a thickness 3.15~12.66m.

Fine sandstone: light gray, a few fissures, massive, dense, hard.

Silty sandstone: gray, blocky, muddy cemented, hard, with a small amount of siderite and charcoal, thickness 1.41~6.25m.

Fine sandstone: light gray, dominated by quartz, subfeldspar, and charcoal, massive and dense.

Table 1: Mechanical parameters of rock mass.

| Lithology          | Volume weight (kN/m³) | Young’s modulus (MPa) | Cohesion (MPa) | Internal friction angle (°) |
|--------------------|----------------------|----------------------|----------------|-----------------------------|
| Fine sandstone     | 28.72                | $1.41 \times 10^4$   | 4.28           | 40                          |
| Siltstone          | 26.98                | $1.08 \times 10^4$   | 3.95           | 40                          |
| Mudstone           | 25.12                | $0.45 \times 10^4$   | 2.02           | 31                          |
| Sandy mudstone     | 25.78                | $0.34 \times 10^4$   | 2.19           | 30                          |
| Coal               | 13.71                | $0.12 \times 10^4$   | 1.24           | 32                          |

Figure 2: The lithology comprehensive strata diagram.

Figure 3: Similar simulation experiment device and measuring point arrangement.
2.3. Experimental Results Analysis

2.3.1. Movement of Roof Rock Stratum. As coal seams were mined, the roof moves and sinks, and a camera was used to record the law of movement of the roof strata at the different advance distances during the test, as shown in Figure 4. It can be seen from Figure 4 that when the working face advanced 50 cm (actually advances 50 m), the immediate roof collapses and fills the gob area, and the caving rock mass presents an irregular state. As the face advances, the development height of the fracture zone is 35.28 m, which is about 14 times of the mining height, and the rock strata in the fracture zone are generally articulated. The fractures play a key role in the load control of the working face support. There are many cracks in the rock strata in the fracture zone. As coal seams are mined, cracks continue to expand and evolve. The roof rock strata separation is easy to be formed under the effect of transverse cracks, and the rock strata fracture is prone to appear under the action of longitudinal cracks. Above the fracture zone is a sagging zone, where the rock strata maintains integrity, and there is basically no developed fissure; only the rock layers show an overall tendency to bend and sink. The law of rock stratum fracture is similar to the literature results [19, 27].

2.3.2. Subsidence Displacement of Roof Rock Strata. The subsidence displacement of roof rock strata was monitored during the similar simulation test. Seven monitoring lines were arranged in the roof rock strata, and the evolution and distribution laws of the displacement field of the roof rock strata of the stope during the coal seam mining advancement were obtained, as shown in Figure 6.

From Figure 6, it can be seen that as the working face advances, the bending and subsidence appears in the roof rock strata, and the different degrees of subsidence appear in the measuring points above the gob area, and the roof subsidence curves form a typical “U” shape. When the working face advanced 50 cm, the subsidence displacement is the largest at the position 11.54 m away from the coal seam. The farther away from the coal seam, the smaller the subsidence displacement of the roof rock stratum is, and the subsidence displacement is little when the distance exceeds 23.87 m, and the influence of the mining is small. When the working face is advanced by 100 cm, the subsidence displacement of the roof rock stratum reaches a maximum of about 20 mm. The subsidence displacement is little when the distance away from the coal seam exceeds 49.28 cm, and it is basically not affected by mining. The “U” curve formed by the subsidence displacement of the roof rock stratum are more obvious. When the working face is advanced by 150 cm, the subsidence displacement of the roof rock stratum reaches a maximum of about 24 mm.

The farther away from the coal seam, the smaller the subsidence displacement is, but the patterns of the subsidence displacement curves are similar. At the same time, due to the collapse of the rock mass, there is a broken expand characteristics that makes the amount of subsidence displacement of each rock stratum smaller than the thickness of the coal seam. Due to the differences in the mechanical characteristics of each rock stratum (such as the degree of joint and rock stratum strength and thickness), the movement and collapse of each rock stratum are different, representing the coordinated movement characteristics of rock stratum. The law of rock stratum subsidence displacement is similar to the literature results [26, 27].

3. Numerical Simulation by Particle Flow Code

3.1. Particle Flow Code Theory and Parameter Calibration. The PFC2D model is a collection of discrete circular particles. Particle flow program based on the discrete element method uses the explicit time-step circulation rule to calculate the model particles cyclically [17]. The contact force between particles obeys the law of force-displacement, and particle’s motion is based on Newton’s second law. The force-displacement law which is applied to interparticle contact was used to update interparticle contact force, and the Newtonian law of motion which is applied to particles was used to find the position between particles and boundaries [16], as shown in Figure 7. The bond particle model (BPM) of PFC2D contains two main types of models, namely, “contact bond model (CBM)” and “parallel bond model (PBM)” [26]. Particles in the CBM are joined by a point of glue, and contacts cannot transfer moment. While in the PBM, when the local stresses exceed the bond strength between particles, a bond breakage, and a microcrack form. With the occurrence, propagation, and penetration of microcracks, macroscopic failure is finally formed.

The mechanical properties of PBM are similar to that of cementing materials in rocks, and the force between particles is reflected through the contact force chain. Getting microscopic mechanical parameters is a difficult problem in PBM. There is no clear correspondence between the micromechanical parameters of particles and macroscopic mechanical parameters of the physical test. The microscopic parameters are obtained by matching the stress-strain curves of numerical simulation of uniaxial compression with the physical test curves. The basic process is to adjust the microscopic parameters such that the numerical simulation results approximate the physical test results. During this
process, according to the laboratory physical tests, a large number of numerical simulation tests were performed under similar conditions. The numerical simulation results were compared with the laboratory tests results, and the microscopic parameters were repeatedly adjusted via trial and error [17]. The trial and error method process of PFC is shown in Figure 9 [29]. The microparameters obtained are shown in Table 2.

3.2. Numerical Model Establishment. A numerical model was to establish based on the geological conditions in Section 2.2. A rectangle particle assembly surrounded by four walls of top, bottom, left, and right is generated, and the size of the rectangle is 300 m × 200 m, as shown in Figure 10. The boundary conditions of the model are set to free for the top surfaces. The bottom, right, and left surfaces are fixed. The force of the top boundary wall is controlled by servo. The stress value applied to the top wall in this model is 10.53 MPa. The similar process of particle generated is used, which is described in Potyondy and Cundall [28]. The procedure can be divided into three steps, namely, an initial compact assembly, installation of the isotropic stress (1 MPa in the present study), and elimination of the floating particles. Then, according to the integrated lithology map of Figure 2, the model was grouped; at the same time, rock stratum with similar mechanical properties was grouped together, the thinner rock stratum was treated as adjacent thick rock stratum, and the corresponding mesomechanical parameters of Table 2 were assigned to each group. Finally, the model was used for mining simulation with the “delete” command step-by-step. The mechanical properties of the bedding and joint between the rock strata were set by the “joint” command. In order to eliminate the boundary effect on the mining simulation, 50 m coal pillars were left on the left and right sides of the model.

3.3. Numerical Results Analysis. In this part, the rock strata movement, the subsidence displacement, and the rock strata stress distribution are analyzed by using PFC.

3.3.1. Movement of Roof Rock Stratum. At the same time, the law of movement of the roof strata at different advance distances during the test using numerical simulation is shown in Figure 11. It can be seen from Figure 11 that when the working face is advanced by 30 m, the immediate roof is directly filled with the gob area due to its low strength. As the working surface continues to advance, a fixed beam structure is formed in the roof strata. When the main roof (fixed beam structure) reaches the ultimate strength, the main roof breaks; that is, the fixed beam breaks, and the working face forms first roof weighting (as shown in Figure 11(b)). A fixed cantilever beam structure is formed in the fractured rock beam (as shown in Figure 11(c)). As the extension length of the cantilever beam increases, the cantilever beam breaks easily and forms a roof weighting. At the same time, the
repeated breaking of the cantilever beam is the periodic weighting of the working face. The fracture location of the rock strata at the coal wall is shown in Figure 12. Above the coal wall towards the goaf, a stepped fracture was formed in the roof rock strata. The periodic fracture of the rock stratum is the main reason for the periodic weighting of the working face. It reveals the laws of movement and collapse of roof rock stratum from a new perspective.

When the coal seam advances 100 m, the collapse and fracture distribution of the roof rock strata is shown in Figure 13. The A and C are fissure-intensive zones. The rock blocks in the two areas are regularly stacked, and the blocks are inclined. The fractured rock block is located on one side of the coal wall, and on the other side is the falling goaf area. In addition, there are more cracks in the areas, and the two zones are also a gathering place for gas. In zone B, the collapse of the roof rock fell directly into the goaf and slowly compacted. The zone B is a compacted zone where the fallen rock masses bear the weight of some of the upper rock mass.

Compared with the results of rock stratum obtained from the experimental, the numerical simulation of rock strata movement law is similar to the similar material experiment results. However, the numerical simulation of rock strata obviously shows the initial fracture of the roof and the cantilever beam periodic fracture phenomenon, which explains the initial pressure and periodic pressure of the working face. In the similar experimental test, the goaf is
Table 2: Mechanical parameters of rock mass.

| Lithology          | ρ (kg/m³) | E_C (GPa) | k_n/k_s | Ebar_C (GPa) | k_n/k_s | μ   | σ_c (MPa) | τ_c (MPa) |
|--------------------|-----------|-----------|---------|--------------|---------|-----|-----------|-----------|
| Fine sandstone     | 2872      | 8         | 1.6     | 8            | 1.2     | 0.63| 40        | 40        |
| Siltstone          | 2698      | 7         | 1.6     | 7            | 1.2     | 0.60| 35        | 35        |
| Mudstone           | 2512      | 2         | 1.6     | 2            | 1.2     | 0.50| 10        | 10        |
| Sandy mudstone     | 2578      | 3         | 1.6     | 3            | 1.2     | 0.40| 15        | 15        |
| Coal               | 1371      | 1.2       | 1.6     | 2            | 1.2     | 0.40| 8         | 8         |

Note: ρ is the density of the particles; E_C is the elastic modulus of the particles; k_n/k_s is the ratio of the normal and tangential stiffness of the particles; Ebar_C is the elastic modulus of the parallel bond; k_n/k_s is the ratio of the normal and tangential stiffness of parallel bonds; μ is the coefficient of friction between the particles; σ_c is the normal bond strength of the parallel bond; τ_c is the tangential bond strength of the parallel bond.
3.3.2. Subsidence Displacement of Roof Rock Strata. The roof rock strata are deformed and fractured, and the roof rock strata move and sink under the influence of the overlying load and its own gravity. During the advancement process of coal seam mining (the direction to the right is the direction of advancement), the subsidence displacement of roof rock strata is monitored (Figure 10), and the subsidence displacement curves of the roof rock strata under different working distances (50 m, 80 m, and 100 m) were obtained, as shown in Figure 14.

From Figure 14, it can be seen that as the working face advances, the regular movement and subsidence of the roof rock stratum appears, and the measuring points above the gob area sink at different degrees; in addition, the roof subsidence curves form a typical “U” shape. When the working face is advanced by 50 m, the subsidence displacement is the largest appearing in the middle of the goaf at the position 10 m away from the coal seam. The farther away from the coal seam, the smaller the subsidence displacement of the roof rock stratum is; the subsidence displacement is little when the distance exceeds 30 m; at the same time, the influence of the mining is small.

When the working face is advanced by 80 m, the subsidence displacement of the roof rock strata reaches a maximum value of about 2.5 m, and the maximum sinking...
Figure 13: The fracture distribution of roof rock stratum.

Figure 14: The subsidence displacement curves of different workface advance distances: (a) 50 m; (b) 80 m; (c) 100 m.
position appears in the middle area of the goaf. The subsidence displacement is little when the distance away from the coal seam exceeds 50 m, and it is basically not affected by mining. The “U” curves formed by the subsidence displacement of the roof rock strata are more obvious. When the working face is advanced by 100 m, the subsidence displacement of the roof rock strata reaches a maximum of about 2.6 m. As the coal seam continues to advance, the maximum subsidence displacement remains basically constant, and the subsidence curves present an asymmetric flat-bottomed distribution. The simulations of the fracture and collapse process of the overlying rock mass in the stope were analyzed by the particle discrete element, and the movement law of the roof rock strata in the stope was further understood. Compared with the results of rock stratum obtained from the experimental, the numerical simulation of rock strata movement law is basically consistent with the similar material experiment results. The subsidence displacement curves all show “U” shape.

3.3.3. Contact Force Chain and Stress of Roof Rock Strata. In the discrete particle flow program, the force is reflected by the contact force chain between the particles [16, 17, 30]. Figure 15 shows the distribution of the contact force chain network of the model at the stage of unmined and mined 100 m. There are two types of force chain structures, namely, a criss-crossing shape force chain (Figure 15(a)) and an arc shape force chain with obvious directionality (Figure 15(b)). The width of the force chain line is related to the magnitude of the force; the wider the force chain, the greater the force is.

It can be seen from Figure 15 that the contact force distribution is mainly closed force chain in the unmined state, and the contact force chain is evenly distribution. The force chain of model begins to adjust when it mines; that is, the force chain is from the criss-crossing shape turns to the arch shape force chain. Therefore, the arc shape force chain gradually increases, and the criss-crossing force chain decreases, forming a strong force chain zone with the arc-shaped force chain. And in the left and right bottom corners of the model, force chain concentration is obviously present. The load of the overburden strata is mainly concentrated on the particles of the arc force chain. The weight of the overburden is transmitted to the front of the coal wall and behind the goaf area through the force chain to form a force chain concentration zone. The strong force chain is mainly concentrated on the arched force chain, the weak chain assists the strong chain to maintain the stability of the structure, and the arched strong chain is the path of the overburden transmission force.

The stress is the macroscopic expression of contact force chain, and there is a positive correlation between force chain and stress. The magnitude of the contact force chain directly affects the macroscopic value of stress. In order to get the stress distribution of rock strata, the measurement circles are used to analyze the stress. As many previous studies have introduced the principle of the measurement circle [16, 17], this article will not introduce it. The measurement circles are arranged according to the stress line of Figure 10.

The vertical stress of coal seams at different advance distances of working face (20 m, 40 m, 60 m, 80 m, and 100 m) was monitored, and the results are shown in Figure 16. At the same time, the stress concentration coefficients in front of the coal wall and behind the goaf were analyzed, as shown in Figure 17.

From Figures 16 and 17, it can be seen that the stress of the surrounding rock is adjusted after the coal seam is mined. Stress concentration occurs in front of the coal wall and behind the goaf area. As the working face continues to advance, the stress concentration factor increases and eventually the increasing trend slows to stabilize. The location of stress concentration moves deep into the coal and rock mass. When the working face is advanced by 20 m, the maximum stress behind the goaf is 18.57 MPa (the original rock stress is 12.5 MPa), the maximum stress in front of the coal wall is 19.51 MPa, and the stress concentration coefficients are 1.48 and 1.56, respectively. When the working face is advanced by 60 m, the maximum stress value behind the goaf is 22.74 MPa, the maximum stress in front of the coal wall is 24.85 MPa, and the stress concentration coefficients are 1.82 and 1.95, respectively. When the working face is advanced 100 m, the maximum stress behind the goaf is 23.65 MPa, the maximum stress in front of the coal wall is 25.17 MPa, and the stress concentration coefficients are 1.89 and 2.01, respectively. And the area in the goaf is a stress reduction zone. The stress concentration in front of the coal wall is the source of force that forms the abutment pressure.

Due to the mining of coal seams, the stresses in the overlying roof rock layers are adjusted, the stress in the same layer is different, and the stress in the different rock layers is not the same at different distances from the roof to the coal seam. Therefore, three stress measuring lines were arranged at different levels above the working face to monitor the stress of different layers when the working face is advanced 100 m, and then the stress distributions of the rock layers at different distances from the roof to coal seam (0 m, 15 m, and 30 m, respectively) were obtained, as shown in Figure 18.

From Figure 18, it can be seen that stress concentration occurs at different layers above the coal seam, and the stress curves present a “double hump” shape. In the coal seam position of the working face (Figure 18(a)), the maximum stress in front of the goaf is 25.17 MPa, the stress concentration coefficient is 2.01, the maximum stress behind the goaf is 23.65 MPa, and the stress concentration coefficient is 1.89. When the distance from roof to coal seam is 15 m (Figure 18(b)), the peak stress intensity in front of the coal wall and back of goaf decreases, the stress concentration coefficient decreases (1.61 and 1.62, respectively), and the location of stress concentration moves toward the goaf. When the distance from roof to coal seam increases to 30 m (Figure 18(c)), the stress concentration coefficient continues to decrease (1.41 and 1.45, respectively), and the stress concentration continues to move toward the goaf. The vertical stress distribution law of the overlying rock stratum above the stope is that the stress concentration is largest in front of the coal wall and behind the goaf. The farther away from the coal seam, the smaller the stress concentration coefficient is, but it is still in a high stress area, and the stress
concentration position moves toward the middle area of the goaf.

4. Discussion

The results of the similar material simulation test and particle discrete element simulation are basically consistent, and the feasibility and rationality of numerical simulation are proved. Rock stratum fractures and collapse patterns are similar, and the laws of subsidence displacement curves are consistent. It is mainly because the mechanical behavior of materials is similar. Similar materials are rock-like materials formed by bonding fine sand through cement and gypsum. Particle discrete elements are bonded together to form rock-like materials through the parallel bond. And the destruction of the material is the breaking of the bonds between the particles. However, due to the limitations of computational capability of the computer, the particle size used in the

\textbf{Figure 15:} The distribution morphology of contact force chain: (a) unmined (criss-crossing shape force chain); (b) mined 100 m (arch shape force chain).

\textbf{Figure 16:} The stress distribution of coal seam.

\textbf{Figure 17:} The stress concentration coefficients under different workface advance sizes.
discrete element calculation is magnified, so the stress curve is not smooth and has a certain degree of volatility during the solution process. The effects of reducing the particle size and taking into account the multiple mining factors on the fracture and collapse of the roof remains to be further studied in the future. Compared with the continuous medium simulations [12–14], the particle discrete element could represent the fracture and collapse processes of the rock stratum. At the same time, the article expounds from a new perspective that the strong arched force chain is the path of the transmission force of the overlying strata.

5. Conclusion

In this paper, the methods of combining the particle discrete unit and similar material simulation test are used to analyze the stress distribution, sinking displacement curve, and movement law of rock stratum in stope, and the following conclusions are obtained:

(1) As the working face advances, regular movement and subsidence appearing in the roof rock stratum and the roof sinking curves form a typical “U” shape. As the coal seam continues to advance, the maximum subsidence displacement remains basically constant, and the subsidence curves present an asymmetric flat-bottomed distribution. The farther away from the coal seam, the smaller the subsidence displacement is. Due to the differences in the mechanical characteristics of each rock stratum (such as the degree of joint, rock stratum strength and thickness), the movement and collapse of each rock stratum are different, representing the coordinated movement characteristics of rock stratum.

(2) The vertical stress distribution law of the overlying rock stratum above the stope is that the stress concentration is largest in front of the coal wall and behind the goaf. The farther away from the coal seam, the smaller the stress concentration coefficient is, but it is still in a high stress area, and the stress concentration position moves toward the middle of the goaf. The stress concentration in front of the coal wall is the source of force that forms the abutment pressure.

(3) As the working face advances, a fixed cantilever beam structure is formed in the fractured rock beam. With the extension length of the cantilever beam...
increases, the cantilever beam breaks easily and forms a roof weighting. At the same time, the repeated breaking of the cantilever beam is the periodic weighting of the working face. Above the coal wall towards the goaf, a stepped fracture is formed in the roof rock stratum. The periodic fracture of the rock stratum is the main reason for the periodic weighting of the working face. The roof fracture is the main cause of the coal mine roof accidents. Therefore, it is of great theoretical and practical significance to study the laws of the movement of the roof above the coal seam for preventing and controlling the occurrence of roof disasters.

Data Availability

The data used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

The author declares that there are no conflicts of interest.

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References

[1] X. Wang and F.-b. Meng, “Statistical analysis of large accidents in China’s coal mines in 2016,” Natural Hazards, vol. 92, no. 1, pp. 311–325, 2018.
[2] T. Zhao and C. Liu, “Roof instability characteristics and pre-grouting of the roof caving in residual coal mining,” Journal of Geophysics & Engineering, vol. 14, no. 6, pp. 1463–1474, 2017.
[3] M. He, Y. Gao, J. Yang, and W. Gong, “An innovative approach for gob-side entry retaining in thick coal seam longwall mining.” Energies, vol. 10, no. 11, p. 1785, 2017.
[4] D. Z. Kong, W. Jiang, Y. Chen, Z. Y. Song, and Z. Ma, “Study of roof stability of the end of working face in upward longwall top coal,” Arabian Journal of Geosciences, vol. 10, no. 8, p. 185, 2017.
[5] W. P. Huang, C. Li, L. W. Zhang, Q. Yuan, Y. S. Zheng, and Y. Liu, “In situ identification of water-permeable fractured zone in overlying composite strata,” International Journal of Rock Mechanics and Mining Sciences, vol. 105, pp. 85–97, 2018.
[6] X. Wang, Z. Wen, Y. Jiang, and H. Huang, “Experimental study on mechanical and acoustic emission characteristics of rock-like material under non-uniformly distributed loads,” Rock Mechanics and Rock Engineering, vol. 51, no. 3, pp. 729–745, 2018.
[7] H. Kang, J. Lou, F. Gao, J. Yang, and J. Li, “A physical and numerical investigation of sudden massive roof collapse during longwall coal retreat mining,” International Journal of Coal Geology, vol. 188, pp. 25–36, 2018.
[8] C. Wang, C. Zhang, X. Zhao, L. Liao, and S. Zhang, “Dynamic structural evolution of overlying strata during shallow coal seam longwall mining,” International Journal of Rock Mechanics and Mining Sciences, vol. 103, pp. 20–32, 2018.
[9] Z. Zhu, H. Zhang, J. Nemcik, T. Lan, J. Han, and Y. Chen, “Overburden movement characteristics of top-coal caving mining in multi-seam areas,” Quarterly Journal of Engineering Geology and Hydrogeology, vol. 51, no. 2, pp. 276–286, 2018.
[10] J. W. Zhang, J. C. Wang, W. J. Wei, Y. Chen, and Z. Y. Song, “Experimental and numerical investigation on coal drawing from thick steep seam with longwall top coal caving mining,” Arabian Journal of Geosciences, vol. 11, no. 5, p. 96, 2018.
[11] F. Gao, D. Stead, and J. Coggan, “Evaluation of coal longwall caving characteristics using an innovative UDEC Trigon approach,” Computers and Geotechnics, vol. 55, pp. 448–460, 2014a.
[12] N. Hosseini, K. Goshtashi, B. Oraee-Mirzamani, and M. Gholinejad, “Calculation of periodic roof weighting interval in longwall mining using finite element method,” Arabian Journal of Geosciences, vol. 7, no. 5, pp. 1951–1956, 2014.
[13] N. E. Yasitli and B. Unver, “3D numerical modeling of longwall mining with top-coal caving,” International Journal of Rock Mechanics and Mining Sciences, vol. 42, no. 2, pp. 219–235, 2005.
[14] M. R. Islam, D. Hayashi, and A. B. M. Kamruzzaman, “Finite element modeling of stress distributions and problems for multi-slice longwall mining in Bangladesh, with special reference to the Barapukuria coal mine,” International Journal of Coal Geology, vol. 78, no. 2, pp. 91–109, 2009.
[15] H. Wang, Y. Jiang, Y. Zhao, J. Zhu, and S. Liu, “Numerical investigation of the dynamic mechanical state of a coal pillar during longwall mining panel extraction,” Rock Mechanics and Rock Engineering, vol. 46, no. 5, pp. 1211–1221, 2013.
[16] P. A. Cundall and O. D. L. Strack, “A discrete numerical model for granular assemblies,” Géotechnique, vol. 29, no. 1, pp. 47–65, 1979.
[17] Itasca Consulting Group, PFC2D (Particle Flow Code in 2Dimensions) Users Guide, Itasca, Minneapolis, MN, USA, 2008.
[18] C.-Y. Lu, C.-L. Tang, Y.-C. Chan, J.-C. Hu, and C.-C. Chi, “Forecasting landslide hazard by the 3D discrete element method: a case study of the unstable slope in the Lushan hot spring district, central Taiwan,” Engineering Geology, vol. 183, no. 31, pp. 14–30, 2014.
[19] G. Wang, M. Wu, R. Wang, H. Xu, and X. Song, “Height of the mining-induced fractured zone above a coal face,” Engineering Geology, vol. 216, pp. 140–152, 2017.
[20] P. Wang, T. Yang, H. Liu, Q. Yu, and F. Zhang, “Characterization on jointed rock masses based on PFC2D,” Frontiers of Structural and Civil Engineering, vol. 7, no. 1, pp. 32–38, 2013.
[21] W.-r. Liu, X. Wang, and C.-m. Li, “Numerical study of damage evolution law of coal mine roadway by particle flow code (PFC) model,” Geotechnical and Geological Engineering, vol. 37, no. 4, pp. 1–9, 2019.
[22] M. Cai, K. Kaiser, H. Morioka et al., “FLAC/PFC coupled numerical simulation of AE in large-scale underground excavations,” International Journal of Rock Mechanics and Mining Sciences, vol. 44, no. 4, pp. 550–564, 2007.
[23] C. Wang, D. D. Tannant, and P. A. Lilly, “Numerical analysis of the stability of heavily jointed rock slopes using PFC2D,” International Journal of Rock Mechanics and Mining Sciences, vol. 40, no. 3, pp. 415–424, 2003.
[24] B. An and D. D. Tannant, “Discrete element method contact model for dynamic simulation of inelastic rock impact,” *Computers & Geosciences*, vol. 33, no. 4, pp. 513–521, 2007.

[25] M. A. Coulthard, “Applications of numerical modelling in underground mining and construction,” *Geotechnical & Geological Engineering*, vol. 17, no. 3-4, pp. 373–385, 1999.

[26] X. Zhang, H. Yu, J. Dong et al., “A physical and numerical model-based research on the subsidence features of overlying strata caused by coal mining in Henan, China,” *Environmental Earth Sciences*, vol. 76, no. 20, p. 705, 2017.

[27] M. Ju, X. Li, Q. Yao, S. Liu, S. Liang, and X. Wang, “Effect of sand grain size on simulated mining-induced overburden failure in physical model tests,” *Engineering Geology*, vol. 226, pp. 93–106, 2017.

[28] D. O. Potyondy and P. A. Cundall, “A bonded-particle model for rock,” *International Journal of Rock Mechanics and Mining Sciences*, vol. 41, no. 8, pp. 1329–1364, 2004.

[29] X. Wang and L. G. Tian, “Mechanical and crack evolution characteristics of coal–rock under different fracture-hole conditions: a numerical study based on particle flow code,” *Environmental Earth Sciences*, vol. 77, no. 8, p. 297, 2018.

[30] S. F. Edwards and R. B. S. Oakeshott, “The transmission of stress in an aggregate,” *Physica D: Nonlinear Phenomena*, vol. 38, no. 1–3, pp. 88–92, 1989.
