Research on Floor Failure Features of Coal Seam above the Confined Aquifer in The Case of Backfill Mining

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Abstract. Water inrush accident is one of the most common disasters in the production process of coal mines, while coal mining in deep strata has always suffered from the threat of the confined aquifer. It is of great significance for mining safety assessment and water inrush prevention to grasp the failure features of coal seam floor during the mining. The article adopted the methods of numerical simulation and field test to make a comprehensive study on floor failure characteristics of 7# coal seam above confined aquifer in M coal mine in the case of backfill mining. Numerical simulation results of FLAC3D software showed that the backfill mining had reduced the pressure relief area, the working face floor-heave and the stress concentration of the surrounding rocks, so that the failure depth of the floor did not develop along with the advancing of working face. Meanwhile, water injection tests on the total 9 testing sections of two drills of the coal floor rocks indicated that during the working face advancing, the amount of water leakage of testing sections above 16.8m has significantly increased, and when the working face was far away from the testing point, the leakage would still be higher than that before mining and in a certain extent the rock stratum had been damaged. But water leakage of testing sections of 16.8-25.5m appeared no obvious change, which showed that the rocks kept the original state without being destroyed during the mining. Finally, it is seen that the results of field test and numerical simulation were basically the same that is in the case of backfill mining, the depth of floor rock failure of 7# coal seam was about 16m and there was no danger of water bursting.

1. Introduction
As the basic energy, coal will still be in the dominant position in China's energy consumption. After 20 years’ continuously intensive mining, easy mining resources in the shallow strata have been nearly exhausted, and coal mining has entered into the deep area. Compared with the shallow mining, the deep mining always suffers from the problem of water inrush for the coupling influence of high ground stress and confined aquifer. In order to realize safety recovery of the coal resources above confined aquifer, the domestic and foreign scholars have done a lot of research on mining above confined aquifer and backfill mining. Z.Y. Wang put forward the main research contents, methods and
frame structure of mining above confined aquifer [1]. Y.X. Zhao et al. used similar simulation experiments to reproduce the pressure relief process of confined aquifer during the mining, and obtained characteristics of stress and displacement fields at different depths of floor [2]. W.M. Li et al. analyzed the influence of different excavation steps on the stress and displacement fields of floor by numerical simulation [3]. W.S. Du et al. made research on the control method of floor in confined water by means of laboratory test, and put forward the control measures such as filling mining and floor reinforcement [4]. A.T. Jakubick et al. studied the effect of rock permeability on water inrush of floor combined with the action of confined water [5, 6]. Y.D. Zheng et al. studied the failure law of coal seam floor with filling mining by means of numerical simulation and field measurement [7]. The above results have provided an important reference for the floor control of coal mine under the threat of underground water. But the failure features of coal seam floor above the confined water under the condition of backfill mining have not been systematically studied. Taking M coal mine as an example, the floor failure features of the working face with backfill mining technology were studied systematically by the methods of numerical simulation and field test.

2. Geology situation

The ground elevation of M coal mine in Shandong Province of China is about +75m and that of the underground working face is about -600m. The 7# coal seam is the mainly mining seam with an average inclination of 10 and a thickness of 2.5m, which belongs to the medium thick coal seam. The coal seam floor is composed of fine sandstone, mudstone, and Ordovician limestone and so on. The Ordovician limestone is an aquifer with great thickness and large amount of water, which is about 35m from the bottom of 7# coal seam. Upon the Ordovician limestone is mudstone and fine sandstone interbed, with a thickness of about 17m and good performance of water insulation. The length of the mining working face is 1000m and the width is 140m.

3. Numerical simulation

3.1. Numerical model

At present, there are many softwares used to simulate the stress and strain of rock mass [8]. FLAC3D numerical simulation software developed by the Itasca Consulting Group Inc. was taken in this simulation. During the calculation, the rock mass was set as an elastoplastic body, whose failure criterion conformed to the Mohr - Coulomb model.

Based on the geological histogram, the numerical model was established according to the actual thickness and lithology of the rock or coal layer. The geometric size of the model is 400m, 400m and 120m in the direction of X, Y and Z respectively, as shown in Figure 1. As this research focused on the failure of coal seam floor, the thickness of coal seam floor in the model was set to be much larger, with the thickness of 80m from the coal seam bottom to Ordovician limestone. And as to eliminate boundary effect the rock strata up from the coal seam roof are chosen in the model was set to be 40m. In order to simulate the initial stress conditions of the coal strata before mining, displacement constraints were set in the model boundary, an average distribution load of 8MPa was imposed on the top and a pure water pressure of 3.5MPa was imposed on the upper boundary of the Ordovician limestone. The physical and mechanical parameters of rock, coal and fillings are shown in Table 1.

The width of the working face in the numerical simulation model is 120m and the length 380m. It is assumed that filling and mining are synchronized during the backfill mining process, and the ratio of filling height to coal seam thickness is 0.8.
Table 1. Physical and Mechanical Parameters of Rock, Coal and Fillings.

| Parameters          | Lithology     | Elasticity Modulus/GPa | Poisson's Ratio | Cohesion/MPa | Friction Angle/° | Tensile Strength/MPa | Density/kg/m³ |
|---------------------|---------------|------------------------|-----------------|--------------|------------------|-----------------------|--------------|
|                     | Siltstone     | 1.05                   | 0.85            | 4.9          | 40               | 1.2                   | 2660         |
|                     | Coal          | 0.48                   | 0.15            | 1            | 25               | 0.03                  | 1500         |
|                     | Limestone     | 4.49                   | 2.82            | 4            | 39               | 0.95                  | 2650         |
|                     | Fine sandstone| 0.98                   | 0.92            | 5            | 42               | 1.1                   | 2760         |
|                     | Mudstone      | 1.78                   | 0.51            | 1.9          | 30               | 0.89                  | 2710         |
|                     | Filling       | 0.03                   | 0.02            | 2.55         | 23               | 1.4                   | 1870         |

3.2. Numerical simulation results and discussion

3.2.1. Distribution of effective stress. In the case of backfill mining, the effective stress distribution of coal seam roof and floor during the mining process is shown in Figure 2. Supported by the fillings, the concentration coefficient of abutment pressure in the front of the working face is very small. It is about
1.22 when the working face is propelled to 50m, and about 1.51 when advanced to 150m with small increase.

At the beginning of mining, the contact effect between the roof and fillings is not obvious due to the small roof subsidence. The supporting function of the fillings has not been exerted, as shown in Figure 2 (a). Similar to the roof caving mining method, a large area of pressure relief in roof and floor of the working face has emerged.

When the working face is advanced to 150m, as shown in Figure 2 (b). As the overlying rock has been fully subsided, the supporting capacity of the fillings is fully developed. At this time, the vertical stress in the middle of the goaf has been restored to the level of the original rock stress. In addition, due to the existence of confined aquifer, the effective stress of Ordovician limestone in the coal seam floor is very low. Especially after the excavation, there occurs to be a large range of low stress zone in limestone, and the change of stress state causes the Ordovician rock to be damaged easily, which is likely that the cracks in the rock are developed and interconnected, to form the rising area of confined aquifer.

3.2.2. Distribution of vertical displacement. The displacement distribution of the working face roof and floor during the mining process is shown in Figure 3. The maximum roof subsidence is 320mm and the maximum floor-heave is 180mm when the working face is advanced to 50m, as the roof was not fully touched with the fillings, the control effect of the floor by fillings is weaker, as shown in Figure 3 (a). When the working face is advanced to 150m, the maximum subsidence of the roof has reached 720mm, and as shown in Figure 3 (b). Having been solidly touched with the roof, the control effect of floor-heave by the fillings turns to be better with the maximum floor-heave of 120mm. All this indicates that the roof and floor displacement and the pressure relief range has been restrained by the fillings.

![Figure 3](image-url)

**Figure 3.** Vertical Displacement Distribution of Roof and Floor during the Mining Process.

3.2.3. Failure depth of floor. The distribution of plastic damage area of surrounding rock when the working face advancing to different distances is shown in Figure 4. Supported by the fillings, the
failure depth of the floor is approximately the same when the working face is advanced to 50m and 150m, which is about 16m and does not develop in the vertical direction as the working face moves forward.

![Image](image_url)

Figure 4. Plastic Damage Zones Distribution of Floor during the Mining Process.

4. Field test of floor failure depth

4.1. Project design

In order to analyze the floor failure features of the working face, the floor damage depth of 703 working face is measured on-site by adopting the drill leakage detector, which is a patent product of Shandong University of Science and Technology. The device is mainly composed of blocklet, pressure hose, drill pipe, water injecting and sealing platform and so on, as shown in Figure 5. The destroyed status of the rock could be confirmed by measuring and comparing the leakage flow of each testing section of the drill before and after mining.

![Image](image_url)

Figure 5. Schematic Diagram of Drill Leakage Detecting Device.
According to the regulations for the retention of coal pillars and mining under buildings, waters, railways and main roadway, the floor failure depth of working face could be calculated by formula (1):

$$h=0.0085H+0.1665\alpha+0.1079L-4.3579$$

(1)

Where, $h$ is the floor failure depth, $H$ is the mining depth, $\alpha$ is the average inclination angle of the coal seam, and $L$ is the width of working face [9]. According to the values of $H$, $\alpha$ and $L$ of M mine, it could be calculated that $h$ is 18.1m. Combining with the experience data, two dive drills named A and B were constructed from the auxiliary transportation roadway nearby, going through the protective coal pillar and reaching the floor of 703 working face. The parameters of drills are shown in Table 2, and the drills layout profile is shown in Figure 6.

| Table 2. Parameters of Drills. |
|--------------------------------|
| Drills | Inclination Angle/° | Length/m | Diameter/mm | Azimuth/° | Vertical Depth of Testing /m | Sections/m |
|--------|---------------------|----------|-------------|-----------|----------------------------|------------|
| A      | 10                  | 96.5     | 89          | 170       | 12-16.8                    | 12-16.8    |
| B      | 15                  | 98.4     | 89          | 180       | 16.8-25.5                  | 16.8-25.5  |

Figure 6. Drills Layout Profile.

4.2. Testing procedure
After the completion of drilling, water injection tests were carried out respectively for each testing section (with a vertical depth of 1.5-2m) before and after mining activities. In order to prevent water leakage at both ends of the blocklet, the pressure was kept between 2 to 3MPa during the test process. And the pressure of water injection would be kept between 0.3-0.5MPa, and the leakage flow was observed once every 5 mins. When the flow rate didn’t show the continuously increase trend, and the difference between the maximum and the minimum of the 3 flow records was less than the 10% of the final value, it was considered that the leakage of water had been stable, then taking the final record as the observation value.

4.3. Results analysis
Water injection tests were carried out in the 9 testing sections of drill A and B before mining, and the leakage flow records of each section are shown in Table 3 and Table 4, which are in the range of 1.7 /min to 2.3L/min with an average of 1.9L/min.

| Table 3. Leakage Flow of Testing Sections in Drill A before Mining. |
|---------------------------------------------------------------|
| Testing Sections /m | 12-13.5 | 13.5-15 | 15-16.8 |
| Leakage Flow /L/min | 2.3     | 2.0    | 1.8     |
Table 4. Leakage Flow of Testing Sections in Drill B before Mining.

| Testing Sections /m | 16.8-18.3 | 18.3-19.8 | 19.8-21.3 | 21.3-22.8 | 22.8-24.3 | 24.3-25.5 |
|---------------------|-----------|-----------|-----------|-----------|-----------|-----------|
| Leakage Flow /L·min⁻¹ | 2         | 1.8       | 1.9       | 2.2       | 2.1       | 1.7       |

With advancing of the working face, the leakage flow of testing sections of drill A and B varies with different advancing distance of the working face, which is shown in Figure 7 and Figure 8 respectively. The leakage flow of each section was measured respectively while the distance between working face and testing point was reduced to 40m. Water injection test was carried out for every 10m advancing of the working face and finished until the working face was about 40m away from the test point.

It could be seen from Figure 7 that in the early stage of water injection test, the leakage flow of the three sections of drill A is about 1.8-4.0L/min when the distance between working face and testing point is more than 10m. At this time, the rock layers with different depths of the floor are not affected by the mining activities, and the water injection flow reflects the permeability of the primary fissure in the floor rock. But as the working face was advanced forward, the leakage flow of the three sections has been changing significantly. Taking the 12-13.5m section curve as an example, when the working face is advanced near it, the leakage flow could increase to 15L/min from 4L/min, and reaches the maximum value of 25L/min when the working face is moved 10m away from the testing point. Then as the working face continues to advance, the leakage begins to decline, but obviously is higher than the level before mining. All this indicates that during the mining process, the floor rock 12-16.8m below the working face are affected by the mining activities in varying degrees, the closer to the floor, the greater the influence. The significant increase of leakage flow indicates that the coal seam mining has caused the floor rock stratum to be destroyed, with the primary fissures expanding or new fissures emerging.

As shown in Figure 8, there is no significant change of the leakage flow of each section in drill B during the mining process, except the section 16.8-18.3m whose leakage has reached to 2.9L/min. In general, the 16.8-25.5m range of the floor rock has not been affected by the mining activities.

The water leakage curves of the nine sections of drill A and B are drawn together, as shown in Figure 9. It could be clearly seen that during the advancing of working face, there appears to be no obvious change in the leakage flow of the floor rock deeper than 16.8m, which indicates that the floor failure depth of the working face has developed to 15-16.8m.

Figure 7. Leakage Flow Curves of Testing Sections in Drill A.
5. Conclusion

Based on the research above, the following conclusions can be drawn:

(1) Numerical simulation results through the FLAC3D software showed that the backfill mining reduced the pressure relief area, the working face floor-heave and the stress concentration of the surrounding rocks. So the failure depth of the floor was about 16m and did not develop downwards with the working face advancing.

(2) Field water injection tests on the total 9 testing sections of drill A and B of the coal floor rocks indicated that during the working face advancing, the amount of water leakage of testing sections above 16.8m significantly increased, and the leakage was still higher than that before mining when the working face had been far away from the testing point. It demonstrated that the rock stratum had been damaged in a certain extent. Water leakage of testing sections of 16.8-25.5m appeared no obvious change, which showed that the rocks could keep the original state and be undestroyed during the mining.

(3) The results of field test and numerical simulation were basically the same. The depth of floor rock failure of 7# coal seam working face was about 16m in the case of backfill mining and there was
no danger of water bursting. However, in the process of production, the geological exploration work should be strengthened to avoid water inrush accidents caused by faults or collapse columns in the floor.

Acknowledgments
The authors would like to appreciate Professor Zhang Wenquan, Zhang Peisen, Li Ruifeng and other experts for their kind and patient guidance.

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