Development characteristics and field detection of overburden fracture zone in multiseam mining: A case study

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Abstract
During the multiseam mining process, the overlying strata were damaged due to different mining disturbances, where a large number of fractures were produced which were the main seepage paths of gas and water, easy to cause gas or water inrush accidents. Thus, it is necessary to explore the development characteristics of the fractured zone in the overlying strata from multiseam mining. In this paper, laboratory triaxial compression tests were firstly conducted on the rock specimens subjected to different mining disturbances. And the corresponding values of $m$ and $s$ could be obtained through a calculation based on the test results. Subsequently, the numerical simulation was conducted to simulate the field situation, in which the $m$ and $s$ values were used in the numerical model combined with the Hoek-Brown–modified criterion, and the results show that the overburden failure height was increased with the mining disturbances from 48 to 120 m, and then, the effects of mining height and interval thickness on the development height of fracture zone were studied by numerical simulation. Finally, the leakage of roof drilling at different stages was tested and calculated in the field. The results showed that the loss of leakage drilling can directly reflect the development degree of roof cracks, and the detection results were close to the development height of the numerical simulation fracture zone. It is verified that the simulation method used in this paper is feasible and can be used as a reference for similar research.

KEYWORDS
fractured zone, mining disturbances, multiseam mining, numerical simulation, overburden failure height
1 | INTRODUCTION

Coal mining is often accompanied by some accidents, so safe mining is becoming more important than ever. With the advancement of working face, due to the function of in situ stress and self-weight, a caving zone, a fracture zone and a curved subsidence zone are generated from the bottom upwards in the overlying strata. Since the fractures of coal and rock mass are the main channels for fluid migration and infiltration, it is of great significance to accurately determine the water-permeable fractured zone in the overlying strata caused by mining not only to ensure the safe production of coal mining under water, but also to improve the recovery rate of coal pillars and reduce the waste of coal resources. Numerous studies have been conducted with different methods to investigate the fracture evolution law of the overlying rock strata under different mining conditions. Majdi et al proposed the theoretical steady-state mathematical model of overburden strata movement and fracture extension and the calculation method for corresponding height of water-permeable fractured zone. Many domestic researchers represented by Qian and Miao established the theoretical system of “horizontal three zones” and “vertical three zones” on the basis of theoretical analysis, field measurement and experimental research on the distribution characteristics of water-permeable fractures zone in the overlying strata. Meanwhile, the factors which affect the development height of water-permeable fractures zone have been studied for a long time. Combined with the theoretical research on the water-permeable fracture zone, the prediction formulas under different surrounding rock properties and occurrence states have been basically formed. Miao et al have conducted comprehensive research on the effect of the location and composition of key strata on the height of water-permeable fractured zone and pointed out that the rupture of key strata controlled the failure height of the overlying strata. Qu et al proposed a three-zone conceptual model in the overlying strata of a longwall panel that accounts for the coupled behavior of strata deformation and gas flow. The model comprises a fractured gas-interflow zone, a destressed gas-desorption zone, and a confined gas-adsorption zone. With the development of the technology, various research and monitoring methods have been used to predict the height of water-permeable fractured zone in the overlying strata under different mining conditions. For instance, the microseismic monitoring technology has been used to study the evolution laws of the overburden mining-induced fissure. Based on the factors of mining thickness, hard rock proportional coefficient, length of working face and mining depth, Chai et al have accurately predicted the height of fractured zone of overburden strata in Qianjiaying coal mine using the GA-SVR methods. Based on the system theory, Zhang et al proposed a mining-bursting physics model to predict the conducting-water fractured zone in the overlying strata in Xinglongzhuang mining area, and the error was about 8.8%. In recent years, the spatial distribution characteristics and evolution laws of overburden fractures have been further studied and major achievements have been obtained. Based on previous researches, Yuan proposed the theories of methane control in depressurized mining, including methane extraction, simultaneous mining technique of coal and methane without coal pillar, and circular overlying zone for high-efficiency methane extraction in coal seams with low permeability, which provides important guidance for the safe coextraction of coal and gas.

Besides, some researchers have carried out a series of studies on the damage law of overburden strata caused by mining under special geological conditions. For unconformable overlying strata, the transmission of stresses is blocked by the unconformity of strata during the process of stresses redistribution induced by coal mining, resulting in the formation of asymmetric saddle-type fracture zone, which is high at left and low at right, formed in the overlying strata over the goaf. Meanwhile, the size of reserved coal pillars also affects the development of an overburden fractured zone. Under the different coal pillar conditions, the fracture structure of the basic roof and the resulting fracture development zone demonstrate different evolution laws.

The damage degree of the overlying strata can directly affect the occurrence of coal mine disasters. The large intact hard overlying strata can easily cause violent mining pressure in the working face, which may result in the occurrence of rock and coal burst, while the fragmented overlying strata may easily cause the water inrush accident. For multiseam mining, the extraction of lower seam will activate the fracture zone over the upper seam, then enlarge the range of aquifer affected by roof fractures, and may further induce the water intrusion accident. However, as a major mining method, the multiseam mining method has been widely employed in most mining areas in China. Thus, it is of significance to explore the evolution of the water-permeable fractured zone in the overlying strata from multiseam mining. The objectives of this study are mainly as follows:

1. To calculate the Hoek-Brown parameters of $m$ and $s$ of the overlying strata under different disturbances.
2. To explore the development law of fracture zone in the overlying strata from multiseam mining through numerical simulation experiments.
3. To obtain the actual height of water-permeable fractured zone in the overlying strata.

2 | BACKGROUND OF MINING AREA

Jinhuaogong coal mine is located in the northeast of Datong city. The geological structure of the coalfield is simple, and the fault
is rare, as well as the fold. The current mining coal seams are 7#, 8#, 10#, 11-1# and 12-2#. The 7# coal seam is located in the middle of the coal layers (Figure 1). The immediate roof is mainly composed of sandy mudstone, and the main roof mainly includes silty and fine sandstone, and partially contains medium-coarse sandstone. The interval between 8# and 7# coal seams is about 15 m, and its main roof is mainly composed of siltstone and medium-coarse sandstone. The 11# coal seam is located in the lower position of the coal layers, and the distance from the 8# coal seam is about 35 m. The coal layer is stable and the structure is simple. Besides, the roof strata are mainly fine sandstone with compact structure, good cementation, and high compressive strength. In the working face, the fully mechanized long wall strike-retreat mining method has been employed.

3 | MECHANICAL RESPONSE PROPERTIES OF ROOF STRATA BY DIFFERENT MINING DISTURBANCE

3.1 | Rock sampling

In Datong mining area, there are many minable coal seams with small intervals (about 8-25 m). And the roof is mostly fine sandstone and siltstone with dense and hard structure. To investigate the mechanical properties of hard rock strata subject to different mining disturbances, the sampling positions were chosen at the rock strata over the working faces 8709 in 8# and 11# coal seams, respectively (Figure 1). The length of the working face 8709 in 8# coal seam is 165 m and the thickness of coal seam is about 1.6 m; while the length of the working face 8709 in 11# seam is 160 m and the thickness of coal seam is about 2.0 m. The sampling was conducted in the connection roadway between the working faces (8709) in 8# and 11# coal seams.

3.2 | Testing system and procedures

The raw coals were processed into standard specimens with 50 mm in diameter and 100mm in length approximately according to the ISRM. The test system mainly includes the MTS815 servo testing system (Figure 2(A)) and the AUTOLAB-1000 multifunctional acoustic parameter automatic measurement system (Figure 2(B)). To achieve the different damage effect on the rock strata caused by different mining disturbances, the samples were processed as follows: Firstly, the uniaxial compressive strength of rock samples should be obtained; to simulate the damage effect on the rock strata caused by first mining disturbance. The specimen was loaded
to about 70% of the uniaxial compressive strength and then unloaded to about 40%; to simulate the damage effect on the rock strata caused by the second mining disturbance, the specimen was then loaded to about 85% of the uniaxial compressive strength and unloaded to about 40%. In order to comply with the field situation, all the rock specimens from the 11# rock strata were only subjected to mining disturbance treatment once. And part of rock specimens from the 8# rock strata were subjected to mining disturbance treatment once, while the other part was subjected twice. The sensor’s layout was processed as shown in Figure 2(C), during the test, the center frequency of acoustic longitudinal wave probe is 700 kHz, and the measurement principle is shown in Figure 3(A). Figure 3(B) showed the paths of stress loading and unloading.

In addition, triaxial tests were carried out on the specimens affected by different mining disturbances. First, the confining pressure ranging from 5 to 40 MPa was applied to the rock specimens with a loading rate of 0.025 MPa/s. Then, the displacement controlled axial loading was conducted on the specimens with a displacement loading rate of 0.02 mm/min until the specimens were fully destroyed. In this process, the constant confining pressure was maintained and the whole load-displacement curves were recorded. Thus, the values of $m$ and $s$ in the Hoek-Brown criterion were calculated with the test results. Besides, the effect of confining pressure on the strength of rock strata affected by different mining disturbances also can be investigated.

### 3.3 Results and analysis

#### 3.3.1 The characteristics of wave velocity of rock specimens under different disturbances

The average UCS value of fine sandstone and siltstone from 8# rock strata is 110 MPa, while the UCS of the rock specimens from 11# rock strata is 77 MPa.
To investigate the damage effect of mining disturbance on the rock strata, the wave velocity was measured on the rock specimens subjected to different simulated disturbances. Figure 4 shows the results of the wave velocity tests. It can be seen that the wave velocity range of intact rock specimens from 11# Roof and 8# Roof is

**FIGURE 4** Wave velocity of rock specimens from 8# Roof (A) rock specimens from 11# Roof (B)

The strength of rock specimens subjected to different simulated disturbed under different confining pressure (A) 11# Fine sandstone (B) 11# Siltstone (C) 8# Fine sandstone (D) 8# Siltstone

**FIGURE 5** The strength of rock specimens subjected to different simulated disturbed under different confining pressure (A) 11# Fine sandstone (B) 11# Siltstone (C) 8# Fine sandstone (D) 8# Siltstone
about 2.2-2.5 km/s. In the initial stage, the wave velocity increases with strength. This is due to that the internal fractures of the rock specimens were compacted and the specimens became relatively more uniform. When the load increased to about 150 kN, the wave velocity began to decrease, indicating that the internal fractures of the rock samples began to initiate and propagate. With the load increased, the damage gradually developed from microscopic to macroscopic, and ultimately the rock specimen was fully destroyed, the wave velocity rapidly decreased, accompanied by the appearance of macrolongitudinal fractures on the surface of the rock samples. For the rock samples subjected simulated mining disturbance, the overall wave velocity showed a decreasing trend, indicating that fractures existed in the rock samples at the initial stage. With the increase in load, the fractures further propagate and the wave velocity gradually decreases.

3.3.2 | The effect of confining pressure on the strength of rock specimen subjected to different disturbance

Rock masses are generally in a triaxial stress state; thus, it is necessary to study the influence of confining pressure on the strength of rock mass. In this study, triaxial tests were carried out on specimens with different damage degree caused by different simulated mining disturbances. The confining pressure $\sigma_m$ was increased successively with a gradient of 5 MPa, and the corresponding strength of specimens can be obtained successively. Figure 5 illustrates the variation characteristics of $\sigma_1-\sigma_3$ curves of rock specimens with different damage degree. The strength of rock specimens increased with the confining pressure and decreased with the disturbances. Table 1 shows that rock strength and confining pressure of different disturbance degrees can be expressed by quadratic functions.

3.3.3 | The characterization of damage degree of roof strata based on $m$ and $s$

The Hoek-Brow criterion was first proposed by E. Hoek and E. Brown in 1980 based on a large number of mechanical tests and laboratory tests, which can effectively reflect the nonlinear empirical relation between the maximum and minimum principal stresses during the process of rock failure. It can be expressed as:

$$\sigma_1 = \sigma_3 + \sqrt{m\sigma_3\sigma_{ci} + s\sigma_{ci}^2}$$

where $\sigma_1$ and $\sigma_3$ are the maximum and minimum principal stresses, respectively. $\sigma_{ci}$ is the UCS of the intact rock mass. $m$ and $s$ are constants related to the rock mass properties. And $m$ reflects the hardness of rock mass with value range of 0.001 (highly fractured rock mass) to 25 (hard intact rock), while $s$ reflects the fragmentation degree of rock mass with a value range of 0 (shattered rock) to 1 (intact rock).

To calculate the values of the parameters $m$ and $s$, the Equation (1) can be transformed into:

$$(\sigma_1 - \sigma_3)^2 = m\sigma_3\sigma_{ci} + s\sigma_{ci}^2$$

For the rock specimens without disturbance, let $s = 1$. Then, the values of $m$ and $s$ of the intact rock mass can be obtained by taking the average value of 3-5 groups of triaxial test data. For the rock specimens subjected to different disturbances, the values of $m$ and $s$ of the damaged rock mass can be achieved by fitting 3-5 groups of triaxial test data (Figure 6 and Table 2).

To further elaborate the relationship between the values of $m$ and $s$ and the mining disturbance, the reduction rate of $m$, $s$ affected by different mining disturbance is proposed to characterize the damage degree of rock mass (Table 3). It is shown that the second disturbed aggravates the damage of the specimens.

$$\eta_m = (m_0 - m_{i+1})/m_0$$

### Table 1 Fitting formulas of different confining pressure strength for roof typical rock

| Lithology          | Times of disturbance | Fitting formulas       | $R^2$ |
|--------------------|----------------------|------------------------|-------|
| 11# Fine sandstone | 0                    | $y=87.94+6.58x-0.72x^2$ | 0.945 |
|                    | 1                    | $y=79.64+6.35x-0.09x^2$ | 0.900 |
| 11# Siltstone      | 0                    | $y=108.03+4.31x+0.01x^2$ | 0.963 |
|                    | 1                    | $y=73.58+4.89x-0.43x^2$ | 0.981 |
| 8# Fine sandstone  | 0                    | $y=81.91+5.35x-0.05x^2$ | 0.992 |
|                    | 1                    | $y=79.02+2.69x+0.02x^2$ | 0.942 |
| 8# Siltstone       | 0                    | $y=106.8+2.06x+0.04x^2$ | 0.953 |
|                    | 1                    | $y=82.64+3.95x-0.01x^2$ | 0.975 |
|                    | 2                    | $y=77.29+2.45x-0.03x^2$ | 0.892 |

### Figure 6 The stress-displacement curves of rock specimens under different confining pressure (A) 11# Fine sandstone, undisturbed (B) 11# Siltstone, undisturbed (C) 8# Siltstone, undisturbed (D) 8# Siltstone, undisturbed (E) 11# Fine sandstone, once disturbed (F) 11# Siltstone, once disturbed (G) 8# Fine sandstone, once disturbed (H) 8# Siltstone, once disturbed (I) 8# Siltstone, twice disturbed (J) 8# Siltstone, twice disturbed
NUMERICAL SIMULATION OF FRACTURE ZONE IN THE OVERLYING STRATA

The UDEC is a numerical simulation software which can effectively simulate the mechanical behaviors of a discrete medium in two-dimensional space. It is widely used to simulate the response mechanical characteristics of rock mass under static or dynamic loading. In the mining field, it has been successfully employed to simulate the process of roof separating and caving with the advancement of working face, and the results can accurately reflect the effect of mining disturbances on the roof strata. In this study, based on the geological conditions of Jinhuagong coal mine, the UDEC software is used to simulate the failure process of roof strata from multiseam mining, which may provide scientific guidance for the safe production of close-distance multiseam mining.

In order to accurately demonstrate the fracture evolution process in overburden strata over the upper seam after the extraction of the lower seams, Hoek-Brown criterion has been implemented and combined with the UDEC software. This may provide guidance and a basis for multiseam mining.

4.1 Hoek-Brown–modified criterion

The Hoek-Brown–modified criterion can be expressed in the form of normal stress and shear stress and used for the estimation of shear strength parameters in the Mohr-Coulomb criterion. The strength parameters of the two criteria follow the relations:

\[ c = \sigma_{ci} \left( 4s + m \sigma_{3n} \right) \sqrt{s + m \sigma_{3n}} / 15 \]  
\[ \phi = \sin^{-1} \left( \frac{6m \sqrt{s + m \sigma_{3n}}}{15 + 6m \sqrt{s + m \sigma_{3n}}} \right) \]  
\[ \sigma_c = \sigma_{ci} \sqrt{s} \]  
\[ \sigma_i = -\sigma_{ci} s / m \]  
\[ \sigma_{3n} = \sigma_{3 \text{max}} / \sigma_{ci} \]  

where \( c \) is the cohesion, MPa; \( \phi \) is the internal frictional angle. \( \sigma_{ci} \) is the UCS of the intact rock mass. \( m \) and \( s \) are parameters related to the rock mass properties; And \( m \) reflects the hardness of rock mass with value range of 0.001 (highly fractured rock mass) to 25 (hard intact rock), while \( s \) reflects the fragmentation degree of rock mass with value range of 0 (shattered rock) to 1 (intact rock).

To realize the use of the Hoek-Brown–modified criterion in the numerical simulation using the UDEC software, the values of \( m \) and \( s \) can be obtained through laboratory experiments. Then, the strength parameters of Mohr-Coulomb, such as the cohesion and internal frictional angle, can be obtained using Equations (5), (6), and (7) based on the Hoek-Brown–modified criterion. Finally, the strength parameters would be introduced into the numerical model. Table 4 shows the converted mechanical parameters of coal and rock.

4.2 Establishment of the numerical model

According to the geologic and current mining conditions of Jinhuagong coal mine, the two-dimensional numerical calculation model with a length of 600 m and a height of 200 m...
as shown in Figure 7 was established. And the corresponding boundary conditions can be set as follows. The upper boundary condition is related to the gravity of the overburden strata. For convenience, the upper load distribution can be simplified to a uniform load, and the upper boundary can be set as a stress boundary condition. The bottom boundary was set as fixed hinge support, that is, the displacement boundary condition. Similarly, the lateral boundaries were also set as fixed hinge support.

4.3 | Results and analysis

4.3.1 | The fracture evolution process of the overlying strata during mining process

Figure 8 illustrates the fracture evolution process of overburden strata with the advancement of working faces. As shown in Figure 8(A), with the advancement of 7# 8705 working face, a caving zone, a fault zone and a bending zone were formed in the overlying strata, and the failure height was about 32 m, while the horizontal damage range is about 160 m. Then, influenced by the mining disturbance caused by the extraction of 7# 8707 working face, the failure height of overburden strata over 7# 8705 working face increased to 41 m, as shown in Figure 8(B). Since the 7# 8707 working face is longer than 7# 8705 working face and affected by the extraction of adjacent working faces, the failure height of the overburden strata over the 7# 8707 working face reached 48 m. Finally, due to the mutual influence of mining disturbances caused by the extraction of two adjacent working faces, a curved subsidence arch was formed in the overlying strata over the 7# coal seam.

With the extraction of 8# coal seam, the rock strata between the 7# and 8# coal seams were broken and

| Lithology          | Coal and rock mass | Coal and rock joint surface |
|--------------------|--------------------|----------------------------|
|                    | Uniaxial compressive strength (MPa) | Tensile strength (MPa) | Cohesion (MPa) | Internal friction angle (°) | Elastic modulus (GPa) | Normal stiffness (GPa) | Shear stiffness (GPa) | Cohesion (MPa) | Internal friction angle (°) |
| Coarse sandstone   | 65                  | 6.7                        | 1.7            | 34              | 5.7                        | 2.35                | 1.2                | 0.19              | 12              |
| Arenaceous shale   | 43                  | 3.1                        | 1.5            | 32              | 4.5                        | 2.1                 | 1.1                | 0.16              | 13              |
| 7# Coal            | 21                  | 1.7                        | 0.7            | 23              | 2.3                        | 1.85                | 0.7                | 0.1               | 8               |
| 8# Fine sandstone  | 111                 | 7.6                        | 2.4            | 35              | 18.28                      | 3.2                 | 1.7                | 0.25              | 19              |
| 8# Siltstone       | 109                 | 6.8                        | 3              | 38              | 16.81                      | 2.9                 | 1.5                | 0.3               | 17              |
| 8# Coal            | 23                  | 1.9                        | 0.9            | 23              | 2.6                        | 1.9                 | 0.75               | 0.1               | 9               |
| 11# Fine sandstone | 79                  | 5.4                        | 2.2            | 34              | 13.75                      | 2.4                 | 1.3                | 0.23              | 14              |
| 11# Siltstone      | 75                  | 5.2                        | 2.6            | 36              | 12.32                      | 2.2                 | 1.2                | 0.27              | 15              |
| 11# Coal           | 24                  | 1.9                        | 0.9            | 24              | 2.6                        | 1.9                 | 0.75               | 0.1               | 9               |

| Lithology          | Elastic modulus (GPa) | Normal stiffness (GPa) | Shear stiffness (GPa) | Cohesion (MPa) | Internal friction angle (°) |
|--------------------|-----------------------|------------------------|-----------------------|----------------|---------------------------|
| 7# Coal            | 21                    | 1.7                    | 0.7                    | 23             | 2.3                       |
| 8# Fine sandstone  | 111                   | 7.6                    | 2.4                    | 35             | 18.28                     |
| 8# Siltstone       | 109                   | 6.8                    | 3                      | 38             | 16.81                     |
| 8# Coal            | 23                    | 1.9                    | 0.9                    | 23             | 2.6                       |
| 11# Fine sandstone | 79                    | 5.4                    | 2.2                    | 34             | 13.75                     |
| 11# Siltstone      | 75                    | 5.2                    | 2.6                    | 36             | 12.32                     |
| 11# Coal           | 24                    | 1.9                    | 0.9                    | 24             | 2.6                       |

**FIGURE 7** The simulation model of the working face's layout
connected with the caved and broken strata structure above 7# coal seam. Meanwhile, the separation layer was generated in the bending zone of rock strata above the 8# coal seam. With the continuous extraction of 8# coal seam, the bending zone of rock strata further subsided and fractured, and transformed into the fracture zone, resulting in the increase of the failure height of the overlying strata, reaching about 85 m, as shown in Figure 8(C).

From Figure 8(D), it can be seen that the extraction of 8# 8709 working face has little influence on the development of the fracturing zone.

Similarly, with the extraction of the 11# coal seam, the rock strata between the 8# and 11# coal seams were broken. Meanwhile, the fracture zone in the rock strata above 7# coal seam further developed, accompanied by the expansion of the original fractures. And the final failure height of the overlying strata reached 120 m (Figure 8(E,F)).

### 4.3.2 The effect of mining height and interval thickness on the overburden failure height

In order to study the influence of mining height on the failure height of the overlying strata, the fracture evolution process of the overlying strata under the conditions of mining heights of 2, 4, 6, and 8 m has been simulated. From Figure 9, it can be seen that, when the mining height of 8#
When the mining height exceeds 6 m, the height of fracture zone increases with a greater extent. In a word, the mining height has a significant influence on the height of the fracture zone. Figure 10 shows the effect of intervals of lower coal seams on the overburden failure height. When the thickness of the interval layer is within the range of 45 m, the smaller the thickness of the interval layer is, the greater the influence on the overlying rock fracture zone will be. Obviously, the thicker of the interval strata, the less influence on the height of the fracture zone.

In order to macroscopically characterize the damage caused by mining disturbances on the roof strata, the downhole segmented water injection method was utilized to detect the overburden failure height. As presented in Figure 11, this equipment consisted of four parts: the water injection station, test bit, hose connection, and plug operation station. The test bit consisted of two swelling capsules. The principle of this detection method is that the upward-inclined boreholes were drilled toward the overlying fractured strata above the mining goaf, and several holes should be created along the tubes to allow high-pressure water injection into the borehole. Then, the capsules were inflated by the gas injection pipeline to seal the detecting segment of the borehole. Besides, the water injection station and the water injection tube of the test bit were connected by the water injection hose, constituting a water injection detecting system. The test bit was connected to the drill pipe and moved by the rig. Thus, we can judge the damage of the overlying strata by measuring the water seepage amount. The detection was conducted from the orifice to the end of borehole. Each detecting segment was approximately 2-3 m.

According to the engineering geological conditions and layouts of working faces, three boreholes were separately drilled from the contact roadways in 7# 8707, 8# 8709, and 11# 8709 working faces, with dip angles of 30°, 50°, and 60°, with lengths of 100, 110, and 140 m, respectively. Figure 12 shows the water seepage histogram of boreholes through the roof strata subjected to different mining disturbances. As shown in Figure 12(A), near the end of the borehole, the water seepage was approximately 1.2 L·min⁻¹; thus, we considered that this region was not located within the fractured zone. Therefore, we can obtain the length of fractured zone of
the borehole was about 94 m, and the corresponding vertical height was 47 m. It was smaller than the numerical result of 48 m, which may be resulted from that the borehole did not pass through the highest point of the overburden fracture zone.

Similarly, from Figure 12(B), we can obtain the length of the fractured zone of the borehole was about 107 m, and the corresponding vertical height was about 82 m. Although it was also smaller than the numerical result of 85 m, the relative error was less than 5%, which met the engineering requirement. From Figure 12(C), we can obtain the length of the fractured zone of the borehole was 136 m. It can be seen that the overburden failure height was increased to 118 m. According to the above analysis, it can be known that the water seepage histogram of boreholes can effectively reflect the development of the failure height of the overlying strata, which was in coordination with the numerical results.

6 CONCLUSION

In this paper, we conducted a series of laboratory experiments, numerical simulation and field investigation, aiming at exploring the effect of the mining disturbance on the damage of the rock specimens, and further investigate the crack development of roof strata from multiseam mining. Major conclusions of this study are as follows:

1. Laboratory experiments show that, with the increase of the damage degree of rock specimens, the wave velocity decreases obviously, and the strength of rock specimens with different damage degree can be well expressed by a polynomial function of confining pressure. Meanwhile, a method based on triaxial data to calculate Hoek-Brown parameters $m$ and $s$ of rock samples with different initial damage states was proposed, providing a new approach for the next numerical simulation to study the nonlinear failure of the overlying strata from multiseam mining.

2. Based on the geological conditions of Jinhuagong coal mine and relative parameters $m$, $s$, the numerical model was established to simulate the fracture development in the overlying strata during the process of multiseam mining. The results show that overburden failure height was increased with the mining disturbances, from 48 to 120 m. The mining height has a significant influence on the height of the fracture zone, the thicker of the interval strata, and the less influence on the height of the fracture zone.

3. From fieldwork, it can be known that the variation characteristics of water seepage histogram of boreholes in different stages mining disturbances agree with the development characteristics of the overburden failure height, which indicates that the water seepage histogram of boreholes can
effectively reflect the development of the damage degree and failure height of the overlying strata under different mining disturbances.

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