Research Article

Comprehensive Monitoring Research on the Effect of Roof Cutting and Pressure Release: A Case Study of the Jiulishan Coal Mine in China

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The longwall panel of the Jiulishan coal mine is affected by floor-confined water, while the roof rock weighting makes the production management of some longwall panels difficult. This has caused engineering methods of roof cutting and pressure release to be considered. To explore the dynamic effects of roof cutting and pressure release method, the 14141 longwall panel was taken as the test point, and a variety of monitoring methods were used to comprehensively analyze the characteristics of roof rock weighting, front abutment pressure, and floor mining failure. The main results are as follows: first, the weighting interval, the weighting intensity distribution, and the gateway deformation are asymmetrical. Second, the peak point of the front abutment pressure is approximately 6 m away from the coal wall, the peak stress is 7.02 MPa, the concentration coefficient is 1.54, the range of the pressure increasing area is approximately 16.4 m, and the farthest end of the front abutment pressure effect area is 22.4 m away from the coal wall. Furthermore, the maximum depth of the floor failure is at the No.17 measuring point (A117) position under the condition of one electrode distance. The maximum damage depth is 9.8 m, and the horizontal distance between the location of the maximum failure depth and the working face is 4.1~10.3 m. Finally, compared with the formula calculation value without considering the effect of roof cutting and pressure release or the adjacent 14121 longwall panel, the weighting interval and the weighting intensity, the degree of coal wall spalling, the influence range of the front abutment pressure, and the degree of floor mining failure are significantly reduced. The research results provide scientific and technological support for roof cutting and pressure release as a technical method to prevent floor water inrush and reduce the roof rock weighting intensity on the longwall panel.

1. Introduction

Coal is the main basic energy source in China and plays a leading role in the primary energy structure [1]. However, with the large-scale mining of coal resources, the current mining conditions have changed from easy to difficult [2, 3]. Coal mining under thick and hard roof is an important part of complex mining conditions. Roof controlling technology, especially roof cutting and pressure release, has become a priority in coal mining research. Roof cutting and pressure release have been used successfully to weaken thick and hard roof in several mines, such as the Tashan coal mine (Datong, China), Jiulishan coal mine (Jiaozuo, China), Fucun coal mine (Zaozhuang, China), and Halagu coal mine (Erdos, China).

Many scholars have studied the mechanism of overlying strata pressure under roof cutting conditions [4]. The “cutting cantilever beam theory” was first proposed to explain that advanced roof caving can be achieved by precutting to form a cantilever beam above a gob-side gateway [5, 6]. Subsequently, several methods have been used to analyze the impact of the roof cutting technology. The method of roof cutting and pressure release can effectively reduce the dynamic caused by the release of elastic energy when the
hard roof has a suspended roof structure [7]. Based on the theory of short cantilever beams, the evolution law of roof crack propagation under different blast hole spacing conditions is obtained, and the reasonable blast hole spacing is determined [8]. The effect of deep-hole blasting for gateway pressure releasing was studied through numerical simulation, which is believed that deep-hole blasting can adjust the gateway stress environment to achieve stress release and reduce the deformation of surrounding rock [9]. Taking the Natun coal mine in China as the engineering background, the presplitting cutting height of roof, the presplitting cutting angle, and the distance between presplitting blasting holes were determined as the key parameters based on the analysis of the stress state of roof in the process of mining [10]. According to the method of numerical simulation, zonal characteristics and its influence coefficients of working face pressure using roof cutting and pressure releasing method with no pillar and roadway formed automatically are obtained [11]. A micromechanical damage model considering the heterogeneity of the roof rock was developed using the finite element method (FEM), and the blasting-induced damage evolution in the roof rock was numerically explored using the FEM model, to analyze the effects of the directional roof split blasting technique (DRSB) on the stability of the entry surroundings. As the roof splitting effects were enhanced, more vertical stresses were transferred to the gob area, causing the stress concentrations in the entry surroundings to be mitigated [12]. Based on the most important data concerning the geological and mining conditions in Polish hard coal collieries with particular emphasis on tremors, rock bursts, and fatalities, different types of protective means are applied to avoid damage of powered supports in geominning conditions where dynamic phenomena occur [13]. However, at present, most research has mainly relied on numerical simulation and theoretical analysis, and field measurement research is still relatively lacking.

In addition, the Carboniferous-Permian coal-bearing strata in northern China coalfield directly overlie the Ordovician and Upper Carboniferous. Upper Carboniferous is close to the coal seam, thick, and strongly rich in water, coupled with the effect of fault structure, resulting in the problem of floor water inrush being very prominent [14, 15]. It is estimated that more than 55% of coal mines are limited by the threat of floor-confined water [16]. In recent years, scholars have studied the mechanism and evaluation method of floor water inrush, including “in situ fissures” and the “original destruction” theory [17], the five figures plus double coefficient method [18], the key layer “KS” theory of floor water inrush [19], the mechanism of delayed water inrush from the floor under combined action of confined water [20], the modelling mechanical layering effects on stability of underground openings [21], and Fisher’s discriminant model [22]. In addition, many statistical formulas were obtained to predict the depth of floor mining failure and conduct a regression analysis [23, 24]. However, few studies have linked roof cutting and pressure release to the depth of the floor mining failure and the risk of floor water inrush. Moreover, there is also little field measurement research on the characteristics of floor mining failure in the longwall panel with roof cutting and pressure release.

As the longwall panel of the Jiulishan coal mine is affected by floor-confined water and roof rock weighting, making the production management of some longwall panel difficult, engineering measures of roof cutting and pressure release are being examined. To explore the dynamic effects of roof cutting and pressure release measures, the 14141 longwall panel was considered as the test point, and a variety of monitoring methods were used to comprehensively analyze the characteristics of roof rock weighting, front abutment pressure, and floor mining failure. Compared with the theoretical calculation and the observed value of the adjacent 14121 longwall panel, this study analyzed the various effects of roof cutting and pressure release in the process of advancing the working face. The study results further understand the effects of roof cutting and pressure release measures on the characteristics of roof rock weighting, front abutment pressure, and floor mining failure and provide new ideas for the prevention of floor water inrush.

2. The Study Area and Its Engineering Background

2.1. Geological Conditions of 14141 Longwall Panel. As shown in Figure 1, the Jiulishan coal mine is situated northeast of Jiaozuo City, Henan Province, China, which is approximately 22.0 km from the center of Jiaozuo City. The 14141 longwall panel with an average depth of 350 m is located in coal seam 2 of the first mining area in the Jiulishan coal mine, Henan Province, China. The structure of the coal seam is simple, with an average thickness of 6.9 m, a hardness of 0.6, and an average dip angle of 9.5°. An inclined comprehensive longwall stratified mechanized mining method was applied in the 14141 longwall panel; that is, the width of the longwall panel is 111 m, the length of the longwall panel is 748 m, the average mining height is 3.5 m, and the average daily mining speed is 2.9 m/d. The layout of the 14141 longwall panel is shown in Figure 2. In addition, the 14121 longwall panel is adjacent to the 14141 longwall panel with similar geological conditions, but roof cutting and pressure release were not taken.

The immediate roof is siltstone, with an average thickness of 2.90 m, whereas the main roof is sandstone, with an average thickness of 9.90 m. The seam’s immediate floor is lime mudstone, with an average thickness of 1.00 m, whereas the seam basic floor is siltstone, with an average thickness of 9.60 m. The direct water-filled aquifer of the floor is L9 limestone aquifer with an average thickness of 7.8 m and a hydraulic pressure of 1.5 MPa, which is approximately 21.5 m away from the coal seam. The water bursting coefficient of the L9 aquifer is 0.07 MPa/m, which is higher than the critical value of 0.06 MPa/m. Furthermore, the indirect water-filled aquifer of the floor is L2 limestone aquifer with an average thickness of 12.0 m and a hydraulic pressure of 2.5 MPa, which is approximately 75.0 m away from the coal seam. The water bursting coefficient of the L2 aquifer is 0.02 MPa/m, which is lower than the critical value of 0.03 MPa/m. The basic parameters of the 14141 longwall
panel are listed in Table 1, and the lithological changes of the coal seam roof and floor are listed in Table 2.

2.2. Overview of Roof Cutting and Pressure Release and Comprehensive Observation on the 14141 Longwall Panel. The 14141 longwall panel has a hard roof with a total thickness of 12.8 m, which is unlikely to cave in during mining. At the same time, the longwall panel is adjacent to the L8 and L2 limestone aquifers and Liangmacun fault zone, which is seriously threatened by water disasters related to composite water bodies. To solve the problem of roof pressure and support pressure during the mining process and reduce the floor-broken depth, roof cutting and pressure release were applied to the 14141 longwall panel.

The width of the cross-section of the gateway was 4 m and the height was 3 m. The cross-section was supported by anchor nets and W steel belts with prestressed cable bolts. The row spacing between anchors was 850 mm × 750 mm.
The roof used 17.8 mm × 6500 mm prestressed cable bolts and laying steel mesh to strengthen the support, and two sides laid diamond-shaped nets.

A row of blasting holes was arranged 10 m downward from the open-cut, placed at the interval of the hydraulic support. A total of 48 blasting holes were constructed at the open-cut. The blasting hole diameter was 50 mm, the hole spacing was 1.8 m, the hole depth was 12 m, the hole dip was +90°, and the distance from the coal wall was 1 m. In addition, no blasting hole was arranged within 5 m before and after the fault.

As shown in Figure 3, a row of roof blasting holes was arranged outward along the haulage gateway, and the row direction was parallel to the central line of the haulage gateway. A total of 416 blasting holes were constructed along the haulage gateway. The blasting hole diameter was 50 mm, the hole spacing was 1.5 m, the hole depth was 10 m, the hole dip was +90°, and the distance from the coal wall was 1 m. In addition, no blasting hole was arranged within 5 m before and after the fault.

In order to understand the prevention effect of roof cutting and pressure release, three aspects of dynamic monitoring were considered during the production of the longwall panel, including the characteristics of roof rock weighting, front abutment pressure and floor mining failure.

### 3. Roof Rock Weighting Characteristics on the Longwall Panel with Roof Cutting and Pressure Release in Medium Hard Roof Condition

#### 3.1. Monitoring Scheme for the Initial and Periodic Weighting Characteristics

As shown in Figure 4, three measuring points were set up on the roof cutting side, the middle area of the working face, and the uncut side during mining, with numbers 1, 2, and 3, respectively. Each measuring point corresponded to a hydraulic support. The spacing of each measuring point was 36 m. As shown in Figure 5, the HY60L intrinsically safe Bluetooth stress gauge was positioned at the measuring point to measure the working resistance of the hydraulic support. The observer collected data with a KT217-S3 wireless stress acquisition instrument every 3-7 days and analyzed the correlation between the change in working resistance and the advancing distance of the working face. The monitoring period of the working resistance of the hydraulic support was 43 days, during which the working face advanced by approximately 90 m.

#### 3.2. Analysis of Monitoring Results

Based on the monitoring data of the roof cutting side (No.1 measuring point), the middle area of the working face (No.2 measuring point), and the uncut side (No.3 measuring point), the roof rock weighting characteristic of the immediate roof and the main roof under the condition of roof cutting and pressure release was analyzed.

3.2.1. Analysis of the Initial Weighting Characteristics of the Immediate Roof

The working resistance monitoring results of the three hydraulic supports during the initial weighting characteristics of the immediate roof are shown in Figure 6. During the advancement of the working face, the immediate roof in the gob gradually caves with mining, and the working resistance of the support was stable. In addition, as shown in Figures 6(a) and 6(b), the working resistance of the support at the uncut side and the middle area of the working face was more than 30 MPa, which was close to the critical value of the safe-valve, whereas in Figure 6(c), the working resistance of the support at the cutting side was less than 25 MPa, which was obviously smaller than that of other measuring points. In general, the boundary conditions of the roof were changed by the emergence of the cutting section of the presplitting blasting, resulting in the reduction of the internal stress required for the fracture, the reduction of the energy required for the fracture, and the reduction of the limit span, which in turn reduced the initial weighting intensity. The initial weighting intensity of the immediate roof was larger at the uncut side and middle area of the working face, but smaller at the cutting side.

3.2.2. Analysis of the Initial Weighting Characteristics of the Main Roof

The working resistance monitoring results of the three hydraulic supports during the initial weighting of the main roof are shown in Figure 7. As shown in Figure 7(a), the working resistance of the No.3 support began to increase when the working face was advanced 24 m, up to the maximum at 30 m. However, it did not exceed the critical value of the support safe-valve. Similarly, in Figures 7(b) and 7(c), the working resistance of the No.2 and No.1 support began to rise.

### Table 2: Geological description of the coal seam roof and floor.

<table>
<thead>
<tr>
<th>Rock stratum</th>
<th>Lithology</th>
<th>Average thickness (m)</th>
<th>Lithology description</th>
</tr>
</thead>
<tbody>
<tr>
<td>Main roof</td>
<td>Sandstone</td>
<td>9.9</td>
<td>Dark gray, fine to medium grained, and rich in mica layer</td>
</tr>
<tr>
<td>Immediate roof</td>
<td>Siltstone</td>
<td>2.9</td>
<td>Grayish black with low sand content and plant fossils</td>
</tr>
<tr>
<td>False roof</td>
<td>Mudstone</td>
<td>0.2</td>
<td>Black, glossy, slippery, and fragile</td>
</tr>
<tr>
<td>Coal seam</td>
<td>No.2</td>
<td>6.9</td>
<td>Upper part is terminal, with dim luster; the lower part is flaky and massive, with metallic luster</td>
</tr>
<tr>
<td>Immediate floor</td>
<td>Mudstone</td>
<td>1.0</td>
<td>Black, crisp, and laminated</td>
</tr>
<tr>
<td>Main floor</td>
<td>Siltstone</td>
<td>9.6</td>
<td>Grayish black argillaceous with thin siliceous layer at the bottom</td>
</tr>
</tbody>
</table>

The roof used 17.8 mm × 6500 mm prestressed cable bolts and laying steel mesh to strengthen the support, and two sides laid diamond-shaped nets.

A row of blasting holes was arranged 10 m downward from the open-cut, placed at the interval of the hydraulic support. A total of 416 blasting holes were constructed along the haulage gateway. The blasting hole diameter was 50 mm, the hole spacing was 1.5 m, the hole depth was 10 m, the hole dip was +90°, and the distance from the coal wall was 1 m. In addition, no blasting hole was arranged within 5 m before and after the fault.

As shown in Figure 3, a row of roof blasting holes was arranged outward along the haulage gateway, and the row direction was parallel to the central line of the haulage gateway. A total of 416 blasting holes were constructed along the haulage gateway. The blasting hole diameter was 50 mm, the hole spacing was 1.5 m, the hole depth was 10 m, the hole dip was +90°, and the distance from the lower flank of haulage gateway was 1.1 m. In addition, one pilot hole was constructed in the middle of each of the two blasting holes, with a diameter of 50 mm and a spacing of 1.8 m. The blasting holes were constructed and blasted within 30 m from the lower safety opening before stopping, and the blasting holes were maintained at 25 m from the lower safety opening during mining.

In order to understand the prevention effect of roof cutting and pressure release, three aspects of dynamic monitoring were considered during the production of the longwall panel, including the characteristics of roof rock weighting, front abutment pressure and floor mining failure.

3.2. Analysis of Monitoring Results

Based on the monitoring data of the roof cutting side (No.1 measuring point), the middle area of the working face (No.2 measuring point), and the uncut side (No.3 measuring point), the roof rock weighting characteristic of the immediate roof and the main roof under the condition of roof cutting and pressure release was analyzed.

3.2.1. Analysis of the Initial Weighting Characteristics of the Immediate Roof

The working resistance monitoring results of the three hydraulic supports during the initial weighting characteristics of the immediate roof are shown in Figure 6. During the advancement of the working face, the immediate roof in the gob gradually caves with mining, and the working resistance of the support was stable. In addition, as shown in Figures 6(a) and 6(b), the working resistance of the support at the uncut side and the middle area of the working face was more than 30 MPa, which was close to the critical value of the safe-valve, whereas in Figure 6(c), the working resistance of the support at the cutting side was less than 25 MPa, which was obviously smaller than that of other measuring points. In general, the boundary conditions of the roof were changed by the emergence of the cutting section of the presplitting blasting, resulting in the reduction of the internal stress required for the fracture, the reduction of the energy required for the fracture, and the reduction of the limit span, which in turn reduced the initial weighting intensity. The initial weighting intensity of the immediate roof was larger at the uncut side and middle area of the working face, but smaller at the cutting side.

3.2.2. Analysis of the Initial Weighting Characteristics of the Main Roof

The working resistance monitoring results of the three hydraulic supports during the initial weighting of the main roof are shown in Figure 7. As shown in Figure 7(a), the working resistance of the No.3 support began to increase when the working face was advanced 24 m, up to the maximum at 30 m. However, it did not exceed the critical value of the support safe-valve. Similarly, in Figures 7(b) and 7(c), the working resistance of the No.2 and No.1 support began to rise.
when the working face was advanced to 26.6 m, up to the maximum at 28 m. However, it did not exceed the critical value of the support safe-valve. Moreover, the working resistance of the No.1 support was smaller than that of No.2 and No.3. Therefore, the initial weighting step of the main roof was 24-26.6 m. Affected by roof cutting and pressure release and the dip of 10-13 degrees of the longwall panel, the initial weighting interval and the initial weighting intensity of the working face were asymmetrical. The initial weighting interval of the roof cutting side was larger than that of the uncut side, but the initial weighting intensity was reduced. In addition, as shown in Figure 8, the coal wall at the cutting side was stable during the initial weighting of the main roof, but the coal wall collapsed badly at the uncut side. However, the adjacent 14121 longwall panel had an inclination angle of 12 degrees, but the upper and middle parts of the working face had similar initial weighting.
interval, with an average value of 25.72 MPa, showing that the coal seam inclination had little effect on roof rock weighting.

3.2.3. Analysis of the Periodic Weighting Characteristics of the Main Roof. The working resistance monitoring results of the three hydraulic supports during the periodic weighting of the main roof are shown in Figure 9. The periodic weighting interval of the main roof was 9~15 m, with an average of 11 m. During periodic weighting of the main roof, the average working resistance of the support was 16.33 MPa, and the maximum was 33.95 MPa, approximately 50.5% and 97% of the rated working resistance of 35 MPa, respectively. The average working resistance of the support at the roof cutting side was more affected by the roof cutting and pressure release, whereas the peak working resistance of the support was less affected. Specifically, compared with the support in the middle area, the peak working resistance of the support at the cutting side was reduced by 10.47%, and the average working resistance was reduced by 8.7%. Compared with the support at the uncut side, the peak working resistance of the support at the cutting side was reduced by 7.4%, and the average working resistance was reduced by 30%. Therefore, owing to the small fault structure in the roof layer in the middle of the working face, the peak working resistance of the support was the largest in the middle area, followed by the uncut side, and the cutting side was the smallest. However, the average working resistance was the largest at the uncut side, followed by the middle area, and the cutting side was the smallest. In addition, there was little coal wall spalling with the range varied slightly during the periodic weighting of the main roof.

3.2.4. Analysis of the Gateway Deformation. The trend of the gateway deformation during the advancement of the working face is shown in Figure 10. Two measuring stations were established at 600 m and 710 m of the haulage gateway, and two measuring stations were established at 650 m and 760 m of the air-return gateway. In the process of advancing the working face, the leading influence range of the gateway was 119 m; that is, when the distance from the working face was 119 m, the gateway deformation change began to clearly increase. Furthermore, the gateway deformation change of the air-return gateway was clearly greater than that of the haulage gateway, and the roof-to-floor convergence in each gateway was greater than that of the horizontal convergence. Specifically, at 50 m from the working face to the gateway deformation measuring station, the horizontal convergence at 760 m of the air-return gateway was 202 mm, and the roof-to-floor convergence was 312 mm. At the same time, the horizontal convergence at 760 m of the air-return
The critical value of support safe-valve

The working resistance of the support (MPa)

Advanced distance of the working face (m)

The working resistance of the No.3 measuring point support

(a)

The working resistance of the No.2 measuring point support

(b)

Figure 6: Continued.
The gateway was 56 mm, and the roof to floor convergence was 106 mm. In addition, the horizontal convergence at 760 m of the haulage gateway was 78 mm, and the roof to floor convergence was 192 mm. The horizontal convergence at 600 m of the air-return gateway was 78 mm, and the roof to floor convergence was 98 mm. As the lower part of the haulage gateway was unmined solid coal, and presplitting blasting was carried out on the roof of the haulage gateway, cutting off the roof pressure transmission, the surrounding rock stress of the air-return gateway would change more significantly.

3.3. Effect of Roof Cutting and Pressure Release. The stress generated by the periodic instability of the rock layer in the fracture zone is referred to as the periodic weighting of the main roof. The periodic weighting interval is the advancing distance of the working face during the roof rock weighting period [25].

The periodic weighting interval \( L \) and the periodic weighting intensity \( F \) of the main roof can be obtained using the following formulas.

\[
L = h \sqrt{\frac{R_t}{3Q}} \quad (1)
\]

\[
F = \left[ 2 - \frac{L \tan (\varphi - \theta)}{2(h_1 - s)} \right] Qb \quad (2)
\]

where \( R_t \) is the tensile strength of the main roof (MPa), \( h \) is the thickness of main roof (m), \( Q \) is the load per unit area of main roof, \( \varphi \) is the internal friction angle (°), \( h_1 \) is the thickness of key stratum (m), \( s \) is the sinkage of key stratum (m), and \( b \) is width of support.

The relevant parameters of the roof, considering \( R_t \) as 3.15 MPa, \( h \) as 9.9 m, \( Q \) as 16.82 MPa, \( \varphi \) as 38.7 °, \( \theta \) as 0, \( h_1 \) as 9.9 m, \( s \) as 0.7 m, and \( b \) as 1.9 m, are substituted into formulas (1) and (2) for calculation. The calculation results show that the periodic weighting step of the main roof is 19.62 m and the stress is 36.6 MPa.

3.3.1. Comparative Analysis of Roof Rock Weighting. From the monitoring results, it can be seen that the weighting interval and the weighting intensity were asymmetrical. The weighting interval of the roof cutting side was larger than that of the uncut side, but the weighting interval intensity is reduced. The coal wall at the cutting side was stable during the initial weighting of the main roof, but the coal wall collapsed badly at the uncut side.

The gateway deformation change of the air-return gateway was greater than that of the haulage gateway in the process of mining. Besides, the gateway deformation changes of the air-return gateway and haulage gateway were not much different on the adjacent 14121 longwall panel. Its horizontal convergence was approximately 220 mm-240 mm, and the roof-to-floor convergence was approximately 240 mm-260 mm.

The above results showed that roof cutting and pressure release had a significant impact on the characteristics of the roof rock weighting on the longwall panel, rather than the impact of the coal seam inclination.

The actual monitoring results were compared with the calculated values of the formula. The roof structure and pressure characteristics change after roof cutting and
The critical value of support safe-valve

The working resistance of the support (MPa)

Advanced distance of the working face (m)

Figure 7: Continued.
pressure release. Compared with the theoretical calculation, the average periodic weighting interval was reduced by 43.9%, and the average weighting intensity was reduced by 55.4%. Compared with the adjacent 14121 longwall panel without roof cutting and pressure release, the average periodic weighting interval was reduced by 43.75%, the average weighting intensity was reduced by 36.5%, and the peak weighting intensity was reduced by 25.6%. Therefore, roof cutting and pressure release of the 14141 longwall panel could significantly reduce the weighting interval and the weighting intensity. A comparison of the parameters between the two longwall panels is presented in Table 3.

4. Characteristics of Front Abutment Pressure on the Longwall Panel with Roof Cutting and Pressure Release

The traditional method of obtaining the characteristics of the front abutment pressure is via mechanical calculation or numerical simulation, which lacks the field monitoring data [26]. The borehole stress meter and ultrasonic logging with "one transmitter and two receivers" are used to monitor the effect section of the front abutment pressure. In addition, the compressive analysis of the characteristics of the front abutment pressure lays a foundation for the study of the effects of roof cutting and pressure release on floor mining failure.

4.1. Monitoring Scheme for the Characteristics of Front Abutment Pressure on the 14141 Longwall Panel

4.1.1. Monitoring Scheme of Borehole Stress Meter. Three groups of horizontal boreholes were drilled in sequence at a distance of 70 m from the haulage gateway to the coal wall. The spacing between boreholes was 20 m and the depth of each borehole was 15 m. The borehole layout is shown in Figure 11. Then, the borehole stress meter was fed horizontally into the borehole with 15 attachable mounting rods. Monitoring points were established, monitoring data were collected after all three boreholes were pushed, and the distribution curve of the front abutment pressure was
Figure 9: Continued.
combined with the daily footage of the longwall panel. The monitoring equipment is illustrated in Figure 12. Affected by the advancement of the working face, the gateway loose circle was not observed, and only observation holes were designed.

4.1.2. Monitoring Scheme of Ultrasonic Logging with “One Transmitter and Two Receivers.” To verify the action range of the front abutment pressure measured by the borehole stress meter, the ultrasonic test was arranged in the normal area of the haulage gateway, which was not affected by mining. Rock ultrasonic test is to study the sound velocity of longitudinal waves inside the rock, from which the relevant physical and mechanical state of the rock is inferred. It provides a reference for evaluating the quality of engineering rock. The measuring borehole was arranged 10 m outward from the open-cut, and a group of boreholes was arranged every 10 m inside the haulage gateway. There were two holes in each group, the dip angle of each hole was 5°, the spacing between holes in the same group was 0.5 m, and the depth was 20 m. Observations were carried out once a day for five days. The borehole layout is shown in Figure 11, and the ultrasonic measurement equipment is shown in Figure 13.

4.2. Analysis of Monitoring Results. Monitoring results of the No.1 borehole stress meter are shown in Figure 14. After the stress meter was placed at the bottom of the horizontal borehole for some time, the imbalance between the oil pressure of the instrument and the borehole wall caused the stress to increase abnormally, and then, the stress relaxation occurred on the borehole wall; that is, the total deformation remained unchanged, the plastic deformation continued to increase, and the elastic deformation decreased accordingly, resulting in stress decreases to a normal state. After a short period, the stress relaxation disappeared, while the borehole stress remained stable. This can be regarded as the borehole stress meter is in the stable stress area, that is, the initial rock stress area. The stress increased from 22.4 m from the coal wall and reached the peak value at 6 m in front of the coal wall. The peak point was approximately 6 m away from the coal wall, and the peak stress was approximately 7.02 MPa. After the peak point, the stress maintained a small high value and then decreased rapidly in a straight line, and the borehole stress meter was moved at 1 m away from the coal wall. At this time, the stress was reduced to below 4 MPa, which was lower than 4.5 MPa in the stable pressure zone. Therefore, the peak point of the front abutment pressure was approximately 6 m away from the coal wall, the peak stress was 7.02 MPa, and the concentration coefficient of the front abutment pressure was 1.54. The range of the stress increasing area was approximately 16.4 m, and the farthest end of the front abutment pressure effect area was 22.4 m away from the coal wall.

The monitoring results of ultrasonic logging with “one transmitter and two receivers” are shown in Table 4. The ultrasonic velocity of the coal body unaffected by mining is 1.6 km/s, which is smaller than the value measured in the laboratory (2.2 km/s), indicating that there are many primary fractures in the coal seam. With the advance of the working face, the ultrasonic velocity of the solid coal at 18 m in front of the coal wall began to decrease under the effect of the front abutment pressure. Therefore, the range of the front abutment pressure was approximately 18 m, which was consistent with the monitoring results of borehole stress meter.
Figure 10: The gateway deformation. (a) The trend of the roof-to-floor convergence. (b) The trend of the horizontal convergence.

Table 3: Comparison of parameters of 14141 and 14121 longwall panel.

<table>
<thead>
<tr>
<th></th>
<th>Roof cutting and pressure release</th>
<th>Initial weighting step of main roof (m)</th>
<th>Average intensity (MPa)</th>
<th>Peak intensity (MPa)</th>
<th>Periodic weighting step (m)</th>
<th>Condition of coal wall collapsing</th>
</tr>
</thead>
<tbody>
<tr>
<td>14141</td>
<td>Yes</td>
<td>24–26.6</td>
<td>16.33</td>
<td>25.24</td>
<td>9–15</td>
<td>Not obvious</td>
</tr>
<tr>
<td>14121</td>
<td>No</td>
<td>48</td>
<td>25.72</td>
<td>33.95</td>
<td>18–21</td>
<td>Obvious</td>
</tr>
</tbody>
</table>
30° 0.3 m

14141

Longwall panel

170 m

Ultrasonic test borehole
Loose circle borehole
Stress-meter borehole

15 m

70 m 10 m 10 m 10 m

2 m 18 m

Figure 11: Borehole layout of front abutment pressure.

Figure 12: Borehole stress meter monitoring station.

Figure 13: Ultrasonic logging with "one transmitter and two receivers" monitoring station.

Figure 14: Front abutment pressure of No.1 borehole stress meter.
4.3. Effect of Roof Cutting and Pressure Release. The front abutment pressure can be obtained by formulas (3) and (5)

$$\sigma_f = \tau_0 \cot \phi \frac{1 + \sin \phi}{1 - \sin \phi} e^{2f x_0/M (1 + \sin \phi (1 - \sin \phi))},$$  \hspace{1cm} (3)

$$K = \frac{\sigma_f}{\gamma H},$$  \hspace{1cm} (4)

$$x_0 = \frac{M (1 + \sin \phi)}{2f (1 - \sin \phi)} \ln \left[ \frac{K \gamma H (1 + \sin \phi)}{\tau_0 \cot \phi (1 - \sin \phi)} \right].$$  \hspace{1cm} (5)

where \(\sigma_f\) is the peak front abutment pressure (MPa), \(K\) is the concentration coefficient of the front abutment pressure, \(x_0\) is the distance between the peak point and the coal wall (m), \(\phi\) is the internal friction angle (°), \(\tau_0\) cot \(\phi\) is the self-supporting pressure of the coal body (MPa), \(H\) is the buried depth of the coal seam (m), \(M\) is the coal seam thickness (m), \(f\) is the friction coefficient between coal seams, and \(\gamma\) is the bulk density of the overlying strata (kN/m⁴).

The relevant parameters, considering \(H\) as 350 m, \(\tau_0\) as 1.1 MPa, \(\phi\) as 38°, \(f\) as 0.4, \(\gamma\) as 27 kN/m⁴, and \(M = 3.5\) m, are substituted into formulas (3), (4), and (5) for calculation. We can obtain the concentration coefficient of front abutment pressure (\(K = 2.04\)), the peak front abutment pressure (\(\sigma_f = 8.26\) MPa), and the distance between the peak point and coal wall (\(x_0 = 7.1\) m).

4.3.1. Comparative Analysis of the Front Abutment Pressure. As shown in Table 5, compared with the theoretical calculation without roof cutting, the peak front abutment pressure was reduced by 15.01%, the concentration coefficient was reduced by 24.5%, and the interval between the peak point and the coal wall was reduced by 15.5%. In addition, compared with the adjacent 14121 longwall panel without roof cutting, the peak front abutment pressure was reduced by 11.8%, the concentration coefficient is reduced by 23%, and the distance between the peak point and the coal wall was reduced by 11.7%. Therefore, roof cutting and pressure release can reduce the effect range and concentration of the front abutment pressure.

5. Characteristics of Floor Mining Failure on the Longwall Panel with Roof Cutting and Pressure Release

5.1. Monitoring Scheme for the Characteristics of Floor Mining Failure. To obtain the maximum depth of the floor mining failure, a monitoring station is arranged on the upper side of the haulage gateway, which was 170 m away from the open-cut. The station location is shown in Figure 15. The relationship between the measurement of borehole apparent resistivity and the position of the longwall panel is shown in Figure 16. A borehole was constructed in the coal body from the station to the open-cut. The borehole diameter was 98 mm, the hole azimuth angle was 239°, the hole dip was -17°, and the hole depth was 90 m. The borehole parameters are shown in Figure 17. A special cable is buried in the borehole. There were 40 copper-ring electrodes at the front of the cable. The electrode spacing was 2 m. The cable was buried in the borehole (the total length of the cable was 78 m), and high-pressure grouting was carried out to seal the hole. During the advancing of the working face, the apparent resistivity of the floor strata was acquired repeatedly under the conditions of one electrode distance (\(AB/2 = 3\) m, \(MN/2 = 1\) m), double electrode distance (\(AB/2 = 6\) m, \(MN/2 = 2\) m), and triple electrode distance (\(AB/2 = 9\) m, \(MN/2 = 3\) m). Combined with the spatial location of the borehole and the spatial distribution of apparent resistivity anomalies, the parameters of floor failure were determined.

5.2. Analysis of Monitoring Results

5.2.1. Analysis of Monitoring Data. Monitoring was carried out six times after the installation of a special cable with electrodes. The horizontal distance between the working face and monitoring station was 150 m (background value),
48 m, 36 m, 32 m, 26 m, and 20 m, respectively. As shown in Figure 18, the characteristics of apparent resistivity along the cable are observed under the condition of one electrode distance, two times the electrode distance, and three times the electrode distance.

From Figure 18(a), we can see that under the condition of one electrode distance, the apparent resistivity before the No.20 measuring point is significantly smaller than the background value, which is caused by the compacting effort of shallow rock fracture water and front abutment pressure.
on the deeper rock mass. In addition, the apparent resistivity of the No.12-17 measuring points in the shallow part fluctuates significantly, and the maximum amplitude is 12 Ω·m. The results show that the floor rock mass in this area has experienced compaction and pressure release, resulting in different degrees of closure and opening of the internal fractures. When the fractures were filled with water, the apparent resistivity was further reduced. When the distance from the working face to the measuring station was reduced from 26 m to 20 m, the apparent resistivity of the No.16 measuring point was reduced. The results show that the floor rock mass is damaged and the fracture is filled with water because it enters the pressure release stage. Therefore, the No.16 measuring point was not the deepest point in the failure area and the apparent resistivity was further reduced. When the distance from the working face to the measuring station was reduced from 26 m to 20 m, the apparent resistivity of the No.16 measuring point was reduced. The results show that the floor rock mass is damaged and the fracture is filled with water because it enters the pressure release stage. Therefore, the No.16 measuring point was not the deepest point in the failure area and the apparent resistivity was further reduced. When the distance from the working face to the measuring station was reduced from 26 m to 20 m, the apparent resistivity of the No.16 measuring point was reduced. The results show that the floor rock mass is damaged and the fracture is filled with water because it enters the pressure release stage. Therefore, the No.16 measuring point was not the deepest point in the failure area.

5.2.2. Spatial Location of Geological Points in the Depth of Floor Failure. The apparent resistivity fluctuates abnormally when the floor rock mass is damaged. The approximate range of the "saddle-" shaped floor failure area can be determined according to the spatial distribution of these abnormal points under different polar distance conditions, as shown in Figure 18. There may be two situations in the vertical direction of the obtained apparent resistivity value; that is, the observed value reflects the electrical conductivity of the measuring point above the cable with electrodes (the first letter of the number is A) or the measuring point below the cable with electrodes (the first letter of the number is B). The first letter in the name of the measuring point indicates the polar distance, while the last two digits indicate the number of measuring points.

If the observed value reflects the apparent resistivity of the rock mass above the cable with electrodes, the boundary of the floor failure area is shown as a solid line in Figure 19. The rock above the curve is in the floor failure zone. The range of the failure zone is gradually reduced from shallow to deep, and the failure zone conforms to the inverted saddle-shaped feature. According to the space position of the measuring points, the maximum depth of the failure area is determined to be at the No.17 measuring point (A117) position under the condition of one electrode distance. If the observed value reflects the apparent resistivity under the cable with electrodes, the boundary of the floor failure area is indicated by the dotted line in Figure 19. The rock mass above the curve is in the floor failure zone. As the failure zone gradually expanded from shallow to deep, the observed shape of the failure area is inconsistent with the theoretical saddle-shaped floor failure area, and hence, this situation is excluded.

Therefore, as shown in Figure 20, the maximum depth of the floor failure was at the No.17 measuring point (A117) position under the condition of one electrode distance. The maximum damage depth was 9.8 m, and the working face was 20-24 m from the measuring station. The distance between the horizontal position of the maximum failure depth and the horizontal position of the coal wall of the working face was 4.1-10.4 m, and the distance from the horizontal position of the cable end was 31.9 m.
Figure 18: Continued.
5.3. Effect of Roof Cutting and Pressure Release. The depth of the floor mining failure can be determined using empirical formulas [27].

\[ h_1 = 0.0085H + 0.1665\alpha + 0.1079L - 4.3579, \]  
(6)  

\[ h_2 = 0.1079L + 0.7007, \]  
(7)  

\[ h_3 = 0.303L^{0.8}, \]  
(8)  

where \( h \) is the maximum depth of the floor failure (m), \( L \) is the width of the longwall panel (m), \( \alpha \) is the dip angle, and \( H \) is the mining depth (m).

The relevant parameters of the longwall panel, considering \( H \) as 350 m, \( \alpha \) as 9.5°, and \( L \) as 111 m, are substituted into equations (6), (7), and (8) for calculation. The calculation results show that the maximum depths of the floor failure are 12.2 m, 12.7 m, and 13.11 m, respectively.

The depth of floor mining failure can be obtained by plastic mechanics formula [27].

\[ W = 0.015H, \]  
(9)
where $W$ is the plastic zone width (m), $h$ is the maximum depth of the floor failure (m), $H$ is the buried depth of the coal seam, $\gamma$ is the rock density (kN/m$^3$), and $\phi_0$ is the average internal friction angle of floor rock mass.

Considering $\phi_0$ as 38° and $H$ as 350 m, the plastic zone width can be calculated using equation (9): $W = 5.25$. Therefore, using equation (10), we can obtain the maximum floor failure depth: $h = 11.31$ m.

### 5.3.1. Comparative Analysis of Floor Mining Failure

As shown in Table 6, the actual monitoring results were compared with the calculated values of the formula. It can be seen that the characteristics of floor mining failure on the longwall panel with roof cutting and pressure release show obvious changes. Compared with the theoretical calculation of the four formulas without roof cutting and pressure release, the maximum depth of the floor failure decreased by 19.7%, 22.8%, 25.2%, and 13.4%, respectively, with an average value of 20.3%.

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<td>13.11</td>
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<td>22.8%</td>
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<td>13.4%</td>
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### 6. Summary and Conclusions

Considering the 14141 longwall panel with roof cutting and pressure release of the Jiulishan mine as the test point, the monitoring was carried out for several aspects. The results show that roof cutting and pressure release have significant impact on the characteristics of the roof pressure, front abutment pressure, and floor mining failure.

The working resistance of the hydraulic supports on the 14141 longwall panel was monitored, which revealed that the weighting interval and the weighting intensity of the working face were asymmetrical; that is, the weighting interval of the roof cutting side was larger than that of the uncut side, but the weighting intensity decreased. During the initial weighting of the main roof, the coal wall at the cutting side remained stable, while the coal wall at the uncut side collapsed badly. The gateway deformation change of the airreturn gateway was greater than that of haulage gateway, and the roof-to-floor convergence in each gateway was greater than that of the horizontal convergence. Furthermore, roof cutting and pressure release of the 14141 longwall panel can significantly reduce the weighting interval, the weighting intensity, coal wall collapsing degree, and the gateway deformation when compared to the theoretical calculation and observed value of the adjacent 14121 longwall panel.

The characteristics of the front abutment pressure on the 14141 longwall panel were monitored, showing that the peak point of the front abutment pressure was approximately 6 m away from the coal wall, and the peak stress was 7.02 MPa, the concentration coefficient of the front abutment pressure 1.54, the range of the pressure increasing area 16.4 m, and the farthest end of the front abutment pressure effect area 22.4 m away from the coal wall. The ultrasonic velocity of the solid coal at 18 m in front of the coal wall began to decrease, and the range of the front abutment pressure was approximately 20 m, which was consistent with the borehole stress meter monitoring results. Furthermore, roof cutting...
and pressure release can minimize the effect range and concentration of the front abutment pressure when compared to the theoretical calculation. Roof cutting and pressure release can also minimize the depth of floor mining failure when compared to theoretical calculations.

Finally, the characteristics of floor mining failure on the 14141 longwall panel were studied, revealing that the maximum depth of the floor failure occurred at the No.17 measuring point (A117) position under the condition of one electrode distance was maintained. The maximum damage depth was 9.8 m, and the horizontal distance between the location of the maximum failure depth and the working face was 4.1–10.3 m.

Data Availability

The [DATA TYPE] data used to support the findings of this study are included within the article.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

Acknowledgments

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