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Research Article

Roadway Protection by Roof-Cutting in the Support Removing Channel of the Long-Wall Mining Face

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Aiming at strong mining pressure and large deformation of surrounding rock in the main roadway, a mechanical model of the advanced abutment pressure distribution in the long-wall mining face was established. The main reason for the large deformation of the surrounding rock in the main roadway was found to be unreasonable position of the stop line, which brings the main roadway under the influence of mining operations. Therefore, on the premise of an unaltered stop line, the technique of cutting the roof and use of pressure relief to protect the main roadway were proposed. The technique will form a weak surface along the stop line, blocking the rock layer from transmitting the abutment pressure. Engineering practice shows that by using the advanced roof cutting and pressure-relief technique, the abutment pressure on the long-wall mining face is reduced and the main roadway is protected.

1. Introduction

The main roadways lie through the entire mine area and thus require a long service life. Once the surrounding rock deforms and is destabilised, the productivity of the mine will be affected, so the main roadways are required to be well supported and the surrounding rock remains stable [1]. The main roadways are usually located along the centre of the strike in mining areas and are affected by the mining of nearby working faces. The maintenance condition is affected not only by natural factors such as strata properties and the original stress state in the rock (the geostatic pressure), but also by advanced mining pressure on working faces (the dynamic pressure). The advanced mining pressure arising on a long-wall mining face consists of horizontal and vertical pressures, and the vertical pressure exerts the most influence on the surrounding rock. Wang et al. [2] investigated the mechanical response and failure mechanism of pillars under static pressure and dynamic pressure and found that vertical dynamic pressure is crucial in terms of its influence on the surrounding rock. Singh et al. [3] used a stress model to find that the vertical stress poses a threat to support structures. Kang et al. [4] assessed the influence on the surrounding rock of a bolt-mesh-anchor-supported main roadway under advanced abutment pressure and found that vertical loading causes stretch-draw and shear sliding failure in roof strata of main roadways. Therefore, in the present research, we focus on vertical stress.

It is difficult to change once the mining system begins to take shape on-site, and with the purpose of safety, there are relatively wide pillars left on each of the two sides to confirm the position of the stop line of the mining face. Based on Critical Layer Theory and a Winkler Elastic Foundation, Han et al. [5] analysed the distribution of advanced abutment pressure in the working face and confirmed the width of the protection coal pillar of the main roadway according to the influence range of advanced abutment pressure. Gu et al. [6] gave the calculation formula of coal pillar loading and confirmed the method of pillar width based on analysis of stress and loading of working faces in the final mining stage. The purpose of wide pillars is to increase the distance between main roadways and working faces, so that the influence on main roadways caused by dynamic pressure is attenuated, but this method not only leads to a significant waste of coal resources and a low mining-recovery rate but only affords a poor level of protection to the main roadway.

To diminish the influence of mining stress, besides widening pillars, research into roof-strata weakening has attracted attention among scholars. Currently, the common method is a precracking technique, including blasting, hydraulic or gas cracking, and static cracking. Hao et al. [7, 8] undertook a static cracking experiment on cuboidal limestone specimens under uniaxial stress, revealing the evolution of acoustic emissions and cracking during static fracturing, and applied it to forced roof caving in gassy mines and roof-cutting pressure-relief techniques. The boring of holes required to implement this method is labour-intensive, and the reaction to events is too slow. Chen et al. [9] tested the influences of dielectric viscosity of high hydraulic pressure, critical CO_2, etc., on the eliciting emergence and development of cracking: cracks elicited by high dielectric viscosity are smoother, and cracks elicited by hydraulic pressure develop in shear failure mode. Kang et al. [10, 11] applied hydraulic pressure to achieve advanced roof cutting in a main roadway, which reduces the advanced abutment pressure in the long-wall mining face. Explosion is an advanced method of rock-breaking. Gao et al. [12] developed a method that utilises directional stretch-draw explosion roof-cutting pressure relief to achieve control of the roadway surrounding rock and also used numerical modelling and industrial practice to examine its effects. Yang et al. [13] achieved explosive precracking of an overhanging roof in goaf based on overhanging theory, cutting side transmission of stress in the roof and thus decreasing the pressure thereon. Yardimci and Konicek [14, 15] used deep-hole cumulative explosions to achieve pressure relief in stiff roof strata above the coal seam: the method could eliminate stress concentration. Existing research mainly concentrates on a solution to the problem that the stiff roof strata in side roadways are overhanging and rock bursts are induced by mining; research into advanced roof-cutting pressure relief in the long-wall mining face by deep-hole explosion remains sparse.

Therefore, this work established a technique involving deep-hole precracking explosion in the key strata layer at the position of stop line in the long-wall mining face based on analysis of the mechanical model, so transmission of mining advanced abutment pressure along the main roadways is cut, effect of dynamic pressure inside working faces is weakened, and the purpose of protecting the main roadway is achieved.

2. Engineering Context

Shanxi Coking Coal Group Co., Ltd. (Nanguan Coal Company), whose production capacity is 14 Mt per annum, runs its 2# coal seam under the classification of a low-gas mine. There are three main roadways: a return air roadway, a conveyor roadway, and a rail roadway. The return air roadway and rail roadway lie along the floor of 2# coal seam, and the conveyor roadway is 12 m below the floor of 2# coal seam. The level interval among the three roadways is 30 m. The working face is located to the north of the air return roadway, the mining method is mechanised inclined long-wall mining with a mining height of 2.45 m, under U-type ventilation, and caving is applied to the roof of the goaf. The layout of main roadways and working faces is shown in Figure 1.

The roof and floor strata of the 2# coal seam are shown in Figure 2; the thickness of the coal seam is 2.45 m, at a dip angle of 7°, with a simple structure and high stability; the thickness of the immediate roof is 6.65 m, which is interbedding composed of sandy mudstone, siltstone, and mudstone; the basic roof is composed of 6.0 m fine-grained sandstone.

As shown in Figure 1, according to design, a 60 m coal pillar was left between the stop mining line and return air roadway after mining of the 3207 working face was completed: as affected by dynamic pressure, there was high pressure around the return air roadway and the deformation was serious, leading to increase in ventilation resistance and the need for repair, making production difficult. When the surrounding rock of the return air roadway is relatively broken, the U-shaped steel shed is used for support, and in other cases, the anchor net and shotcrete support are used. The specific deformation and failure condition are shown in Figure 3. In the U-shaped steel supported area, the roadway was narrowed from a semicircular arch into a triangle, concrete and boarding were damaged, and bending and cracking occurred in the middle of the U-shaped steel support; in bolted net and shotcrete supported areas, there was outbursting on both sides of the roadway, bulging on the upper corner, and spalling of concrete from the exposed surface.

3. The Layout of Advanced Abutment Pressure in the Long-Wall Working Face

The mining area roadway belongs to the main roadway of the mine, which is an important channel for safe production. Studies have shown that the overlying basic roof layer of the long-wall mining face will form a cantilever beam with the coal wall as the fulcrum and the coal wall will bear a large vertical load and transfer this in the direction of the main roadway, which will not only affect the bearing capacity of the coal pillar but also decrease the stability of the main roadway (often to a dangerous extent). Zhu et al. [16, 17] found (by calculation) the influence range of the advanced support pressure on the long-wall working face to be 244 m, based on the key-layer transfer mechanism, so the remaining 250 m coal pillar was intended to protect the main roadway. Huang [18, 19] established an arch model based on Terzaghi's theory and studied the transmission of dynamic load in the roof rock.

To quantify the influence of advance-supporting pressure of long-wall mining face on the stability of the surrounding rock of the main roadway, the mechanical model of advance-supporting pressure distribution in the long-wall mining face shown in Figure 4 was established by combining the research results of Zhu et al. [16, 19].

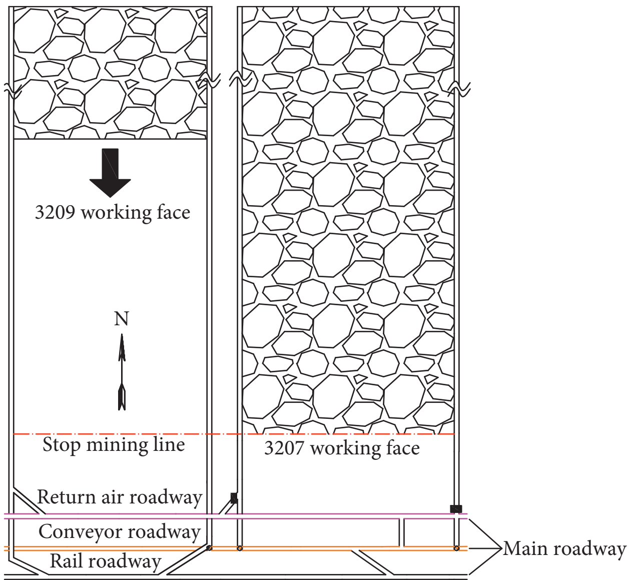


Figure 1: Layout of main roadways and mining faces.

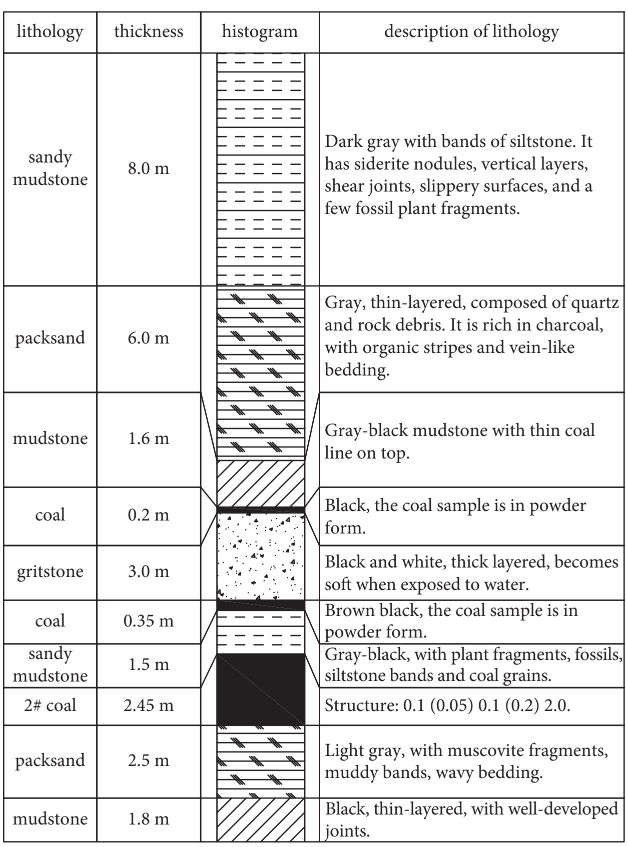


Figure 2: Roof and floor strata of 2# coal seam.

In the mechanical model shown in Figure 4, the abutment pressure is composed of three parts: the overlying rock weight I of the coal pillar (blue part), the rock weight II of the fissure zone controlled by key block ① (red part), and the rock weight III of the curve subsidence zone controlled by key block ① (green part). Among them, the weight II of the fissure zone controlled by block ① propagates toward the coal pillar in the form of an isosceles triangle through its articulation point A and the weight III of the curve subsidence zone propagates toward the coal pillar in the form of an isosceles triangle through the front arch foot G. Now, the mechanical expressions of the three parts of the advance-support pressure I, II, and III of the long-wall mining face were studied and calculated separately.

In Figure 4, the dividing point between the coal wall and the roof strata forms the coordinate origin O to establish an O-x-y right-angled coordinate system, the x-axis points to the direction of the coal pillar, and the y-axis points to the surface; then, the load distribution is as shown in Figure 5.

In △GPO (Figure 5(a)), $\overline{GP} = \overline{GO} = a$, $\overline{OP} = b$, and $\overline{OH} = b/2$. These similar triangles mean that $(\sigma_0/\sigma_x) = (b/2/x)$ (Figure 5(b)) and $\sigma_x = (\sigma_0 \cdot x/b/2)$, so the stress σ_x (kN/m) is as follows:

$$\sigma_x = \begin{cases} \dfrac{\sigma_0 \cdot x}{b/2}, & \left(0 \le x \le \dfrac{b}{2}\right), \\ \dfrac{\sigma_0 \cdot (b-x)}{b/2}, & \left(\dfrac{b}{2} < x < b\right), \\ 0, & (x > b). \end{cases} \tag{1}$$

By integration, $Q = \int_0^{b/2} (\sigma_0 \cdot x/b/2)\mathrm{d}x + \int_{b/2}^{b} (\sigma_0 (b-x)/b/2)\mathrm{d}x$ and $\sigma_0 = (2Q/b)$.

Then, the segmented calculation formula of σ_x is as follows:

$$\sigma_x = \begin{cases} \dfrac{4Qx}{b^2}, & \left(0 \le x \le \dfrac{b}{2}\right), \\ \dfrac{4Q(b-x)}{b^2}, & \left(\dfrac{b}{2} < x \le b\right), \\ 0, & (x > b), \end{cases} \tag{2}$$

where Q is the concentrated load on the isosceles triangle, kN; σ_0 represents the pressure (expressed as a line load) on the top of the system, kN/m; and σ_x is the pressure from any point on (x, 0), kN/m.

3.1. Calculation of the Height of the Three Zones.

3.1. Calculation of the Height of the Three Zones. The key strata in the overlying rock layer of the long-wall mining face appear to be mainly affected by the mining height and the location of the key layer. The height of space between the collapsed immediate roof and the upper key layer is as follows:

$$\Delta = M + (1 - K_P) \sum h_i, \tag{3}$$

where Δ is the height of space between the collapsed immediate roof and upper key layer, also known as rotation volume of the breaking block in the key layer, m; M denotes the thickness of the coal seam, 2.45 m; K_p is the bulking coefficient of broken rocks, 1.3; and $\sum h_i$ is the thickness of the immediate roof, 6.65 m.

Figure 3: Deformation of the main roadway.

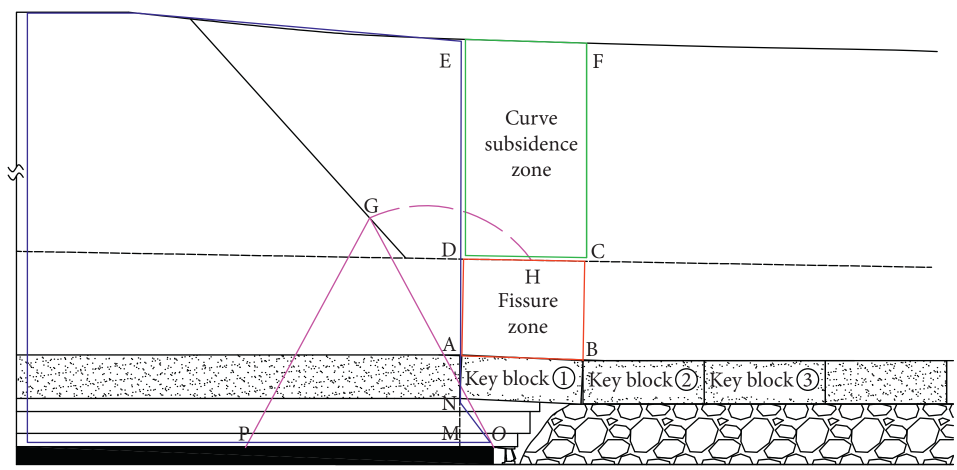


Figure 4: Mechanical model of the distribution of abutment pressure on the long-wall mining face.

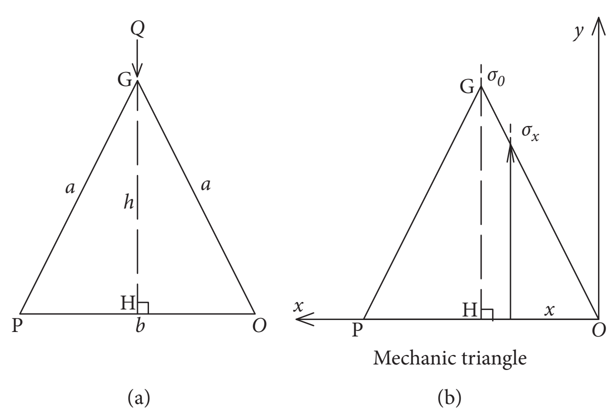


Figure 5: Distribution of force. (a) Geometric triangle. (b) Mechanic triangle.

Assuming the ultimate rotary subsidence volume required to form a stable "masonry beam" structure is Δ_{max}, when $\Delta < \Delta_{max}$, the key layer will be in the fissure zone and present a "masonry beam" structural form. According to the deformation instability model of such a "masonry beam" as found by Qian et al. [20],

$$\Delta_{max} = h - \sqrt{\frac{2ql^2}{\sigma_c}}, \quad (4)$$

where h is the height of key strata, 6.0 m; l is the pace of key-layer failure, and we take a periodic pressure spacing of 15 m; q denotes the stress on the key-layer imposed by the upper strata, 2.1 MPa; and σ_c is the uniaxial compressive strength of key-layer rock, 32.3 MPa. Applying the parameters from Nanguan Coal Company, we can obtain

$$\begin{aligned}\Delta &= M + (1 - K_P)\sum h_i \\ &= 0.46\,\text{m}, \\ \Delta_{\max} &= h - \sqrt{\frac{2ql^2}{\sigma_c}} \\ &= 0.6, \\ \Delta &< \Delta_{\max}.\end{aligned} \tag{5}$$

Therefore, the key layer can form a stable "masonry beam" structural form located in the fissure zone and control the movement of the upper strata. Therefore, the caving zone is the total thickness of the immediate roof (6.65 m). The height of the fissure zone is calculated according to the empirical formula as follows:

$$\begin{aligned}H_1 &= 30\sqrt{M} + 10 \\ &= 56.96\,\text{m}.\end{aligned} \tag{6}$$

The burial depth of this coal seam is 553.65 m, and the thickness of the curve subsidence zone is as follows:

$$\begin{aligned}H_2 &= H - H_1 \\ &= 553.65 - 56.96 \\ &= 496.69\,\text{m},\end{aligned} \tag{7}$$

where H_1 is the height of the fissure zone, m, and H_2 is the thickness of curve subsidence zone, m.

3.2. Advanced Abutment Pressure Composition and Quantitative Analysis. The advanced abutment pressure of the long-wall mining face is composed of three parts: the weight of overlying rock layers I of the coal pillar, the weight of strata II of the fissure zone controlled by key block ①, and the weight of strata III of the curve subsidence zone controlled by key block ① (Figure 4). The mechanical expressions of each component of the advanced abutment pressure were studied and calculated separately by drawing on the dynamic pressure and key-layer loading transmission mechanism of the long-wall working face and using the calculation methods of Zhu et al. [16, 19]. Due to limitations of space, only the relevant calculation results are given and the detailed calculation methods and values are shown in the literature [16, 19].

Part one: the weight of overlying rock layers I.

Calculate the original rock stress according to burial depth:

$$\begin{aligned}\sigma_0 &= \gamma \cdot H \\ &= \left(25\,\text{kN}\cdot\text{m}^{-1}\right) \times 553.65\,\text{m} \\ &= 13.84\,\text{MPa}, \\ \sigma_I &= \begin{cases} 0.062x, & (0 \le x \le 4), \\ 13.84, & (x > 4). \end{cases}\end{aligned} \tag{8}$$

Part two: the weight of strata II of the fissure zone controlled by key block ①.

$$\sigma_{\text{II}} = \begin{cases} 1.56x, & (0 \le x \le 11.5), \\ 35.72 - 1.56x, & (11.5 < x \le 23), \\ 0, & (x > 23). \end{cases} \tag{9}$$

Part three: the weight of strata III of the curve subsidence zone controlled by key block ①.

$$\sigma_{\text{III}} = \begin{cases} 0.12x, & (0 \le x \le 39), \\ 9.56 - 0.12x, & (39 < x > 78), \\ 0, & (x > 23). \end{cases} \tag{10}$$

3.3. Distribution of Advanced Abutment Pressure. According to the above analysis and calculation, the stress on the advanced abutment is as follows:

$$\sigma = \sigma_I + \sigma_{II} + \sigma_{III}. \tag{11}$$

By substituting the actual parameters of Nanguan Coal Company into the formula giving the advanced abutment pressure, the advanced abutment pressure distribution characteristics of the 3209 working face can be obtained (Figure 6). The advanced abutment pressure curve distribution characteristics are consistent with the calculated results of Zhu et al. [16, 19] and other researchers, which verifies the reliability of the mechanical model established in this study.

Analysis of the stress distribution (Figure 6) shows that the peak advanced abutment pressure of the 3209 working face occurs some 12 m in front of the working face with $\sigma_{\text{peak}} = 34.0$ MPa. This is 2.46 times the original stress. The range of vertical pressure in front of the working face that is greater than the original rock stress is defined as the range of advanced abutment pressure affected by mining; then, the range of mining influence range of the long-wall working face is 78 m. The distance between the return air roadway and the stop line is 60 m. The vertical pressure at this location is 16.2 MPa, which is 1.2 times the original stress and is affected by the advanced abutment pressure of the long-wall mining face. Therefore, the main reason for the severe deformation of the main roadway surrounding the rock is the unreasonable position of the stop line of the mining face, which brings the return air roadway within the influence of advanced mining of the working face.

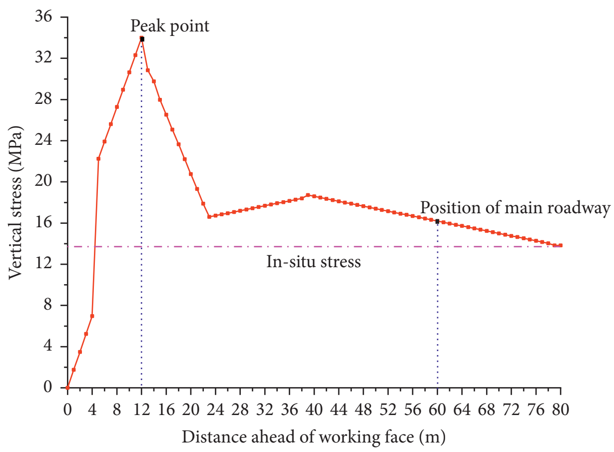


FIGURE 6: Abutment pressure distribution on the 3209 mining face.

4. Advanced Roof-Cutting Pressure-Relief Roadway-Protection Technique

In response to the problem that the main roadway is affected by the advanced abutment pressure of the lateral mining face, the deformation of the surrounding rock is significant and the roadway needs repairing: we propose carrying out advanced roof-cutting in the support removing channel of the working face to cut off or effectively weaken the transmission of the advanced abutment pressure toward the main roadway, so as to control the deformation of the main roadway to the maximum extent.

4.1. Mechanism of Roof-Cutting to Relieve Pressure and Protect the Main Roadway. The previous analysis shows that the main reason for the deformation of the main roadway in the mining area is the influence of the advanced dynamic pressure of the lateral mining face. Therefore, cutting off or preventing the dynamic pressure from spreading to the main roadway can protect the main alley in the mining area. According to the strata control theory, the key layer controls the overburden structure and load transmission; that is, the advanced supporting pressure mainly propagates forward through the key strata [21]. Therefore, the innovation of this research lies in the novel cutting of the key layer at the stop mining line in advance through the method of deep-hole presplit blasting, blocking the propagation path of the advanced supporting pressure, shrinking the range of influence of the advanced supporting pressure, and ensuring that the main roadway lies beyond the range of influence of such pressures [22].

Deep-hole presplitting blasting, as a rock weakening control technique, weakens the integrity of the roof strata through blasting cracks and can realise active control of the collapse location and scope of the surrounding rock. Through deep-hole presplitting blasting, the key layer is cut off along the stop mining line, and the fracture position of key block ① is artificially controlled to be located on the side of the coal wall near the goaf, reducing the range and extent of the influence of advanced abutment pressure. Figure 7 shows the comparison of the leading support pressure distribution on the long-wall mining face before and after roof-cutting.

Analysing the distribution of advanced abutment pressure before and after roof-cutting shown in Figure 7, we can see that, after the implementation of deep-hole precracking roof-cutting explosions in the support removing channel, a large number of fissures will be generated around the holes in the strata, which destroys the integrity of key strata, and the fissures between adjacent blasting holes overlap and penetrate each other, forming a weak surface along the stop line. When the working face advances to the position of the stop line, the roof rock layer caves in time under the influence of its own gravity and mining, which reduces the length of the overhanging roof in the direction of the mining area, decreases the dynamic pressure on the coal wall and the coal pillar in front of it, cuts off the transmission path of the advanced abutment pressure, and realizes the role of protecting the main roadway and reducing the size of the coal pillar.

4.2. Timing of Roof-Cutting and Layout of Boreholes

4.2.1. The Range and Time of Roof-Cutting. The key to advanced pressure-relief roadway-protection technique is to implement precracking explosion only in the key-layer range, and the immediate roof range is not damaged by controlling the sealing hole length and sealing hole quality assurance, so as not to affect the safety of the top plate of the withdrawal channel. Therefore, before precracking explosion, the location and thickness of the key layer should be accurately judged according to the key-layer theory and the top slab rock layer histogram. The target of roof-cutting is the key layer of the top slab, and only the basic roof is cut without destroying the integrity of the immediate roof strata [23].

The time to cut the roof is before the advanced abutment pressure is transmitted to the stop line position, so that the transmission path of the advanced abutment pressure to the direction of the main roadway is cut. In other words, the support removing tunnel is tunnelled to some designed stop line position whereupon a precracking roof-cutting explosion is detonated.

4.2.2. The Layout of Deep-Hole Precracking Boreholes. The directional requirement for roof-cutting pressure-relief roadway-protection technique is onerous, requiring the cracks after an explosion to be opened in parallel with the direction of the stop line. When the explosives are detonated, they will produce a cavity area, crushing area, and fracturing area around the boreholes in turn [24, 25]. Therefore, this study achieves directional rock breaking through the pilot hole setup [26]. By the interhole charging explosive method, an empty hole is designed between two adjacent explosion holes, which provides a free surface and guides the explosion fracture along the preset direction, forming penetrating fractures and weak surfaces in the direction of the explosion hole connection line (Figure 8).

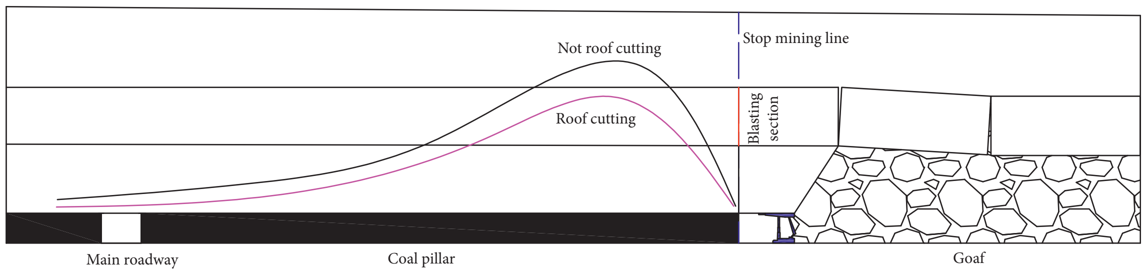


FIGURE 7: Abutment pressure distribution before and after roof-cutting.

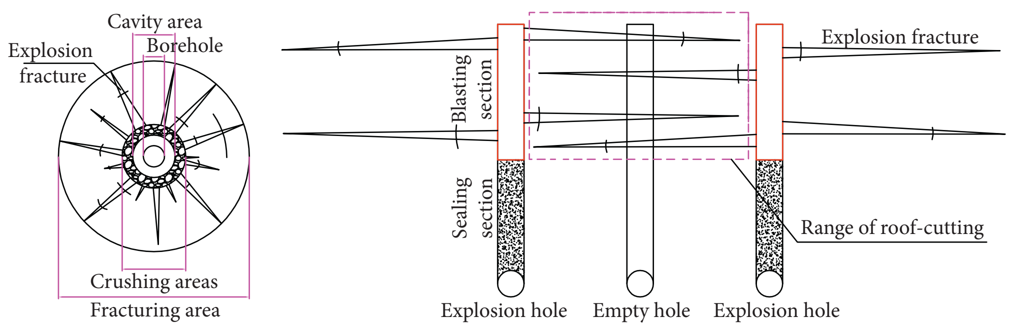


FIGURE 8: Diagram of bursting cracks and roof cutting.

4.3. Numerical Simulation Verification. We selected 2-dimensional distinct element software UDEC (Universal Distinct Element Code), based on the actual engineering conditions of roadways in Nanguan Coal Company's 3209 working face, established a 2-d numerical calculation model (Figure 9), and analysed the rationality of roof-cutting pressure-relief roadway-protection technique proposed in the present work [27]. The rock formation parameters measured on-site are shown in Table 1.

The numerical model uses the Cullen–Moore intrinsic model over an area measuring $x \times y = 200\,\text{m} \times 67.4\,\text{m}$ [28]. The boundary effect is eliminated according to the St. Venant principle; that is, 20 m coal pillars are left on the left and right sides of the model, the left and right sides and bottom boundary are fixed, the top is free, a vertical stress of 13.5 MPa is applied at the top to simulate the overlying strata [29], and no pressure is applied to the left and right sides of the model [30]. The section size of the main roadway in the model is width × height = 4 m × 3 m, and the model is advanced from 173 m to the left side until 118 m at the stop mining line and excavated in steps of 5 m. In the key-layer range at the stop line, the roof cutting was instigated, the precracking was set in advance at the roof-cutting position, and the precracking medium was deleted before the excavation of the working face to simulate the stress changes in the surrounding rock under the two conditions of cutting and not cutting, respectively. During the numerical modelling, a stress-monitoring line was arranged in the middle of rock layer 6 along the length of the model to monitor the roof stress when affected by mining and the stress change curves of surrounding rock under the two conditions of roof-cutting and not roof-cutting the top: the stress change cloud and diagrams shown in Figures 10 and 11 were obtained.

Analysing Figures 10 and 11, the peak stress on the advanced abutment when the numerical model is not cut is 35.7 MPa, arising at a point some 10.2 m in front of the stop line (2.64 times the original rock stress); the advanced pressure at a location some 60 m along the main roadway in front of the stop line is 8.4 MPa. The peak overhead stress after roof-cutting is 32.6 MPa, found at 11.3 m in front of the stop line, which is 2.41 times the original rock stress; the overhead stress at the location of the main roadway is 8.0 MPa. The peak advanced stress was reduced by 3.1 MPa, or 8.7%, by cutting the roof, and the advanced stress at the location of the main roadway was reduced by 0.4 MPa, or 4.8%. Therefore, cutting the key rock layer of the roof at the location of the stop line can play a role in blocking the forward transmission of the advanced abutment pressure and weakening the range of influence and degree of the advanced abutment pressure, which is consistent with the results of the study done by Yue-jin Zhou [31], which verifies the rationality of the technical solution proposed in this study.

5. Industrial Practice and Engineering Effect

5.1. Precracking Explosion in the Support Removing Channel. When the 3209 working face was advanced to 300 m from the stop line, the support removing channel was tunnelled in advance at the designed stop line position. The support removing channel was tunnelled along the roof of 2# coal seam to a rectangular section (width × height = 3.0 m × 2.5 m), with bolt-mesh-anchor support installed. According to the rock column diagram of 2# coal seam shown in Figure 2 and the key-layer theory, it is known that the basic roof of the support removing channel is a 6.0 m-thick fine-grained sandstone. The main target of deep-hole roof-

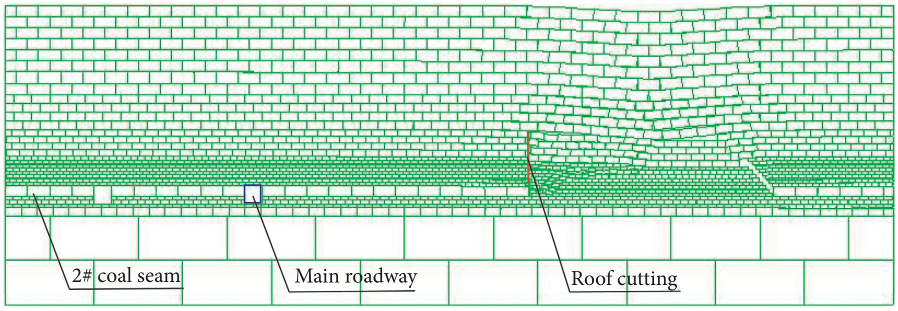


Figure 9: Numerical calculation model.

Table 1: Mechanical property parameters of the strata in the model.

No.	Lithology	Strata thickness (m)	Bulk density (kg/m^{-3})	Poisson's ratio	Bulk modulus (GPa)	Shear modulus (GPa)	Tensile strength (MPa)	Cohesion (MPa)	Friction angle (°)
1	Overlying strata	20	2550	0.21	11.49	8.26	3	3	34
2	Sandy mudstone	8	2320	0.26	9.72	5.56	2.36	3.86	32
3	Packed sand	6	2550	0.18	13.54	11.02	2.18	4.32	35
4	Mudstone	1.6	2200	0.2	5.56	4.17	1.56	1.52	30
5	Coal	0.2	1400	0.22	1.61	1.11	1.2	0.8	26
6	Gritstone	3	2400	0.29	11.11	5.43	2.36	3.86	32
7	Coal	0.35	1400	0.22	1.61	1.11	1.2	0.8	26
8	Sandy mudstone	1.5	2320	0.26	9.72	5.56	2.36	3.86	32
9	2 # coal	2.45	1400	0.22	1.61	1.11	1.2	0.8	26
10	Packed sand	2.5	2550	0.18	13.54	11.02	2.18	4.32	35
11	Mudstone	1.8	2200	0.2	5.56	4.17	1.56	1.52	30
12	Underlying strata	20	2500	0.19	10.75	8.40	3.5	4.3	32

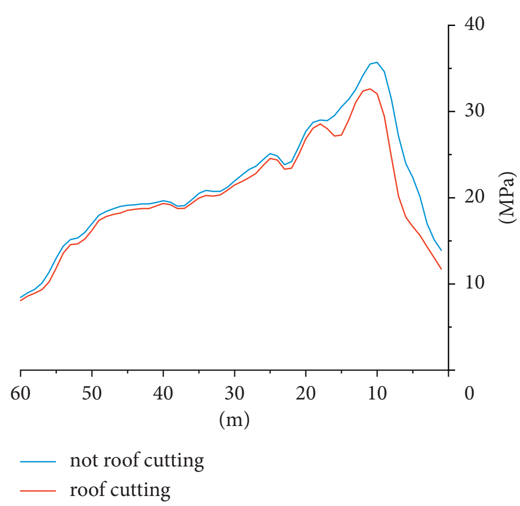


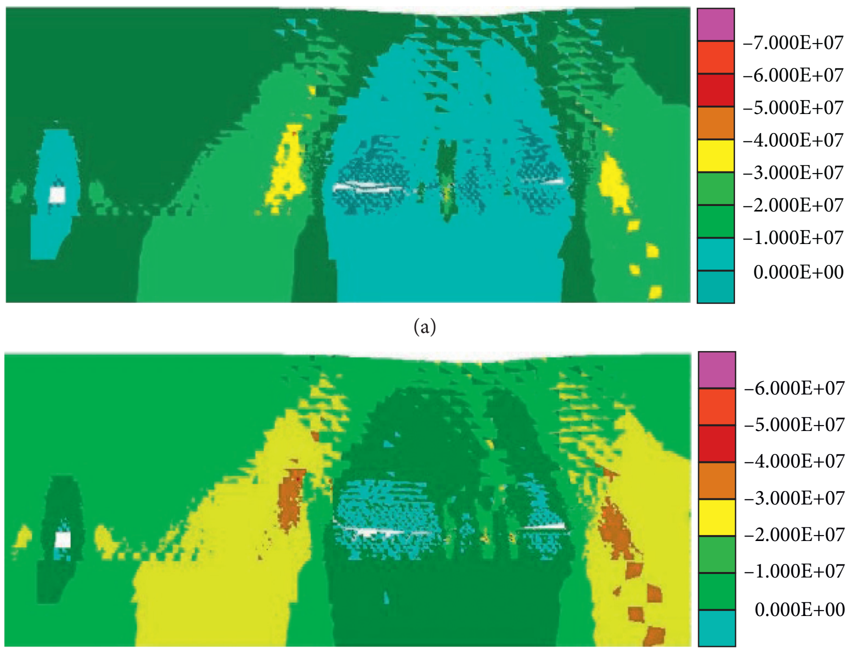
Figure 10: Stress curve of the surrounding rock.

cutting pressure-relief explosion and the weakening treatment is the basic roof, so the vertical depth of roof-cutting explosion is 12.65 m. Under the premise of protecting the safe use of the 3209 working face support removing channel, only explosion for the basic roof range is conducted.

The diameter of the explosion hole is 94 mm, the blasting length of each hole is 8.0 m, and the amount of explosive used is 20 kg/hole.

Based on the hole diameter, the physical characteristics of the basic roof, the performance of the explosives in the cartridge, the structure of the charge, and the radius of the fissure zone produced by a single-hole explosion under this condition were theoretically calculated to be 5.7 m. The spacing of the explosion holes was determined to be 5.0 m based on the ability of the fissure zone produced by explosion to pass through each other and considering a certain safety factor. In total, there were 65 boreholes along the axial direction of the support removing channel (33 charged explosion holes and 32 uncharged auxiliary holes). The layout of the roof-cutting holes is shown in Figure 12.

5.2. The Effect of Roof-Cutting. After the roof-cutting explosion in the support removing channel of the 3209 working face, the immediate roof of the support removing channel was intact and did not affect the removal of the support after mining, with no increase in safety risk. The smoke in the auxiliary hole diffused, which means that the fissures produced by the explosion holes on both sides of the face are interconnected.



(a)

(b)

Figure 11: Vertical stress cloud on the surrounding rock. (a) No roof-cutting. (b) Roof-cutting.

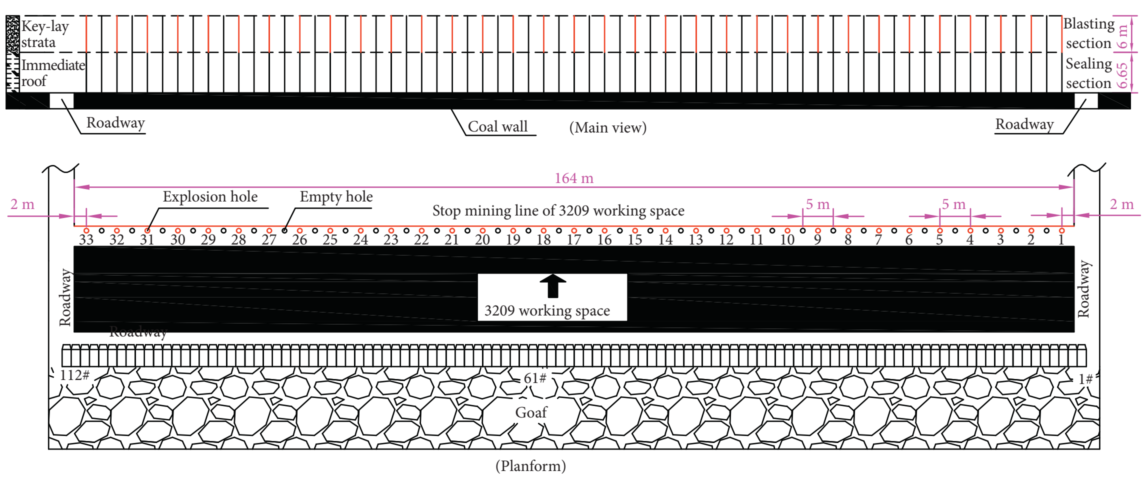


Figure 12: Layout of roof-cutting holes.

Five mining pressure observation stations were established in the return air roadway to observe the convergence and deformation of the surrounding rock during the mining of 3209 working face towards the stop line and to compare and analyse the pressure manifest in the return air roadway with that of the adjacent uncut roof working face. The distance between adjacent observation stations is 30 m, and they are symmetrically arranged.

The average amount of surrounding rock deformation of the five measuring stations is used as an index to evaluate the effect of roadway protection, which is shown in Figure 13. Under the condition of not roof-cutting, the average displacement of the two sides is 180.0 mm, and the average displacement of the roof and floor is 348.4 mm when the return air roadway is affected by the mining of 3207 working face. The average displacement of the two sides is 52.4 mm and the roof and floor is 68.9 mm, which is 29.1% and 19.8% of the average displacement of the uncut roof. The results indicated that the roof-cutting pressure-relief technique proposed in this study can reduce the influence of the advanced abutment pressure of the long-wall mining face on the main roadway and can protect the main roadway.

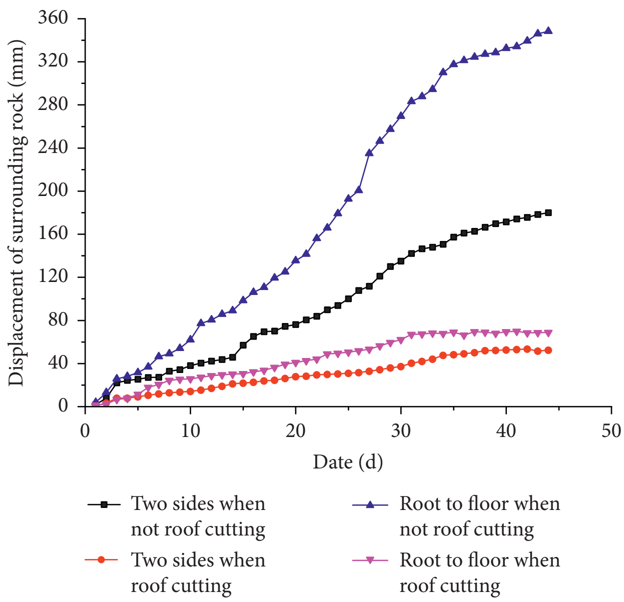


Figure 13: Deformation of the rock surrounding the main roadway.

6. Conclusion

This paper addresses the problem that the stop mining line of the long-wall mining face is unreasonable, resulting in the strong pressure in the main roadway being affected by the advanced abutment pressure. The following main conclusions are drawn:

(1) The main reason for the strong deformation of the surrounding rock of the main roadway is the unreasonable position of the stop line, which makes it within the influence range of the advanced mining of the lateral coal seam.

(2) The peak value and influence range of advanced abutment pressure of the long-wall mining face are mainly affected by the fracture characteristics of key-layer roof, and advancing deep-hole explosion in the support removing channel to precrack the key layer of the roof can cut off or weaken the transmission of advanced abutment pressure to the main roadway, thereby protecting the main roadway.

(3) Through explosion hole and auxiliary hole interval arrangement, it can realise directional presplit blasting. The auxiliary holes do not blast and play the role of providing a free surface and guiding the blasting crack to develop in the preset direction. The cracks generated by adjacent blasting holes penetrate each other, forming through cracks and weak surfaces in the direction of the hole connection.

(4) The key-layer roof is filled with explosives for blasting, and the immediate roof area is blocked by soil can achieve a segmental controlled explosion; it does not affect the integrity of the immediate roof of the support removing channel but cuts off the key-layer roof.

Data Availability

The figure data used to support the findings of this study are included within the article.

Conflicts of Interest

The authors declare that there are no conflicts of interest.

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