Cooperative Control Mechanism of Long Flexible Bolts and Blasting Pressure Relief in Hard Roof Roadways of Extra-Thick Coal Seams: A Case Study

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Abstract: The higher strength of a hard roof leads to higher coal pressure during coal mining, especially under extra-thick coal seam conditions. This study addresses the hard roof control problem for extra-thick coal seams using the air return roadway 4106 (AR 4106) of the Wenjiapo Coal Mine as a case study. A new surrounding rock control strategy is proposed, which mainly includes 44 m deep-hole pre-splitting blasting for stress releasing and flexible 4-m-long bolt for roof supporting. Based on the new support scheme, field tests were performed. The results show that roadway support failure in traditional scenarios is caused by insufficient bolt length and extensive rotary subsidence of the long cantilever beam of the hard roof. In the new proposed scheme, flexible 4-m-long bolts are shown to effectively restrain the initial expansion deformation of the top coal. The deflection of the rock beam anchored by the roof foundation are improved. Deep-hole pre-splitting blasting effectively reduces the cantilever distance of the “block B” of the voussoir beam structure. The stress environment of the roadway surrounding rock is optimized and anchorage structure damage is inhibited. The results provide insights regarding the safe control of roadway roofs under extra-thick coal seam conditions.

Keywords: extra-thick coal seam; hard roof; deep-hole pre-splitting blasting; flexible long bolt; in-situ observation

1. Introduction

The geological conditions of coal seams in China are complex and highly variable, among which coal seams with hard roofs account for approximately 1/3 [1]. Mining activity is typically insufficient to destroy hard roofs under extra-thick coal seam conditions. The suspension of long-distance roofs significantly influences the periodic pressure of the working face and affects the lateral support stress distribution characteristics [2,3]. Concentrated stress of the protective coal pillar deforms the roadway, deteriorates the surrounding rock stress field, and aggravates damage to the roof anchorage structure [4]. This problem is particularly pressing because roofing accidents alone have been reported to account for 70–80% of all working face accidents [5]. Extensive research attention has therefore been paid to understand the ore pressure behavior and damage mechanism of roadway rock mass mined under hard-roof conditions.

Wang et al. [6] conducted numerical simulations and field measurements and concluded that the emergence of a hard top tends to concentrate stress around the working face. Mandal et al. [7] observed the effects of the top coal caving method on the hard roof, caving pace, and bending deflection effects on strata movement. Ju et al. [8] analyzed the
structural characteristics and formation pressure of overlying hard strata after mining a
7-m-thick coal seam. Mondal et al. [9] monitored the strata behavior in the destressed
zone of a shallow Indian longwall panel with hard sandstone cover using mine-micro-
seismicity and borehole televiewer data. Tan [10] studied rock strata movement according
to hard-roof caving behavior, caving step distance, bending deflection, and energy change.
Zhang et al. [11] established mechanical and mathematical models of the overlying pillar
shell structure in a stope with a hard roof in an extra-thick coal seam using elastic plate
theory, and discussed the mechanism of stope size cyclic loading and dynamic loading
ore pressure. Zhao et al. [12] studied the fracture characteristics of extra-thick hard roofs
based on long-beam theory. Xia et al. [13] studied the ground pressure characteristics of a
mining panel area under the combined action of a hard roof and coal pillar in the goaf. Xie
and Xu [14] used a discrete element program to simulate and analyze the influence of hard
roofs of different thicknesses and levels on the peak abutment stress value and influence
range. Han et al. [15] studied the influence of hard roof transverse cantilevers on the gob
ore pressure. Li et al. [16] analyzed the fracture characteristics of hard roofs by combining
physical simulations and acoustic emissions. The above-mentioned studies showed that
stress concentrates more notably in hard roofs, owing to their high strength, as well as
in protective coal pillars, which significantly impacts the working face and roadway in
mining areas.

Hard-roof failure and instability occur gradually over large areas under extra-thick
ccoal seam conditions. Rakesh [17] thinks that the roof stability of thick coal seam mining is
closely related to the thickness of the coal seam, the characteristics of rock mass, and the
stress environment of the surrounding rock. Li et al. [18] showed the large thickness and
amount of moveable space involved in high and hard roofs in extra-thick coal seams. Unver
and Yasitli [19,20] used FLAC3D to simulate the roof failure mechanism of thick seam caving
mining in the M3 longwall panel of Omerler Underground Mine located at Tuncbilek. Singh
et al. [21] studied the strata movement during underground mining of a thick coal seam.
The sliding instability of high and hard roofs produce high ground pressures [22]. For coal
seams thicker than 10 m, the widely used fully mechanized caving technique leads to a
wide range of strong mining effects. The hard roofs of thick coal seams tend to have large
rotation and sinking distances, thus the stope bearing pressure exerts a strong influence over
long distances and times [23,24]. Surrounding rock fracture damage is intensified by the
combined action of the goaf lateral support pressure and advanced abutment pressure. The
theory and technology of using a hard roof to control mining pressure is well established
in the research areas of support reinforcement [25], hydraulic fracturing [26,27], blasting
pressure relief [28,29], and water injection softening [30]. However, the application of deep-
hole pre-splitting blasting technology in large pillar roadways under hard-roof and extra-
thick coal seam conditions remains limited. Most previous studies focused on the stope
structure and mine pressure distribution, whereas the reliability and stability of roadway
support structures is poorly constrained under extra-thick coal seam conditions. This is
particularly important because the force and deformation characteristics of roadway roofs
can be directly determined by the integrity and stability of the roadway support structure.

This paper focuses on the problem of surrounding rock control under hard-roof and
extra-thick coal seam conditions in the Wenjapo Coal Mine (Shaanxi Province, China). The
damage and failure mechanism of the surrounding rock is analyzed using field tests. A
cooperative control technique is proposed that involves long flexible bolts and deep-hole
pre-splitting blasting. Field experiments were performed and the results are discussed. The
design methods and concepts provide a practical reference for deep coal mine roadways
with similar problems.

2. Engineering Background

2.1. Geologic Conditions

This study investigates the surrounding rock instability mechanism of a roadway
induced by strong mining in an extra-thick coal seam and the synergistic control mechanism
of blasting pressure relief and application of long bolts. The Wenjiapo Coal Mine (Binchang Mining Co., Ltd., Shaanxi Coal Chemical Group, Binzhou, China) with an annual output of 5 Mt is taken as a case study. The Wenjiapo Coal Mine is located in the eastern Binchang mining area in the administrative area of Binzhou City, Shaanxi Province, China (Figure 1a). The position of air return roadway 4106 (AR 4106) is shown in Figure 1b. There are five main lanes in the 41-panel area south of the working face. The western side is the design position of working face 4107 and the eastern side is working face 4105 (WF 4105). The width of the coal pillar between the working faces is 44.5 m. The Lujia-Xiaolingtai anticline is present in the southern part of the working face with a strike of ~46–51°, which causes the working face to be generally higher in the south and lower in the north with a downhill slope of 1–4°. The risk level of rockburst in the roadway is weak. The burial depth of the working face is 711–730 m with an average depth of 718 m. The coal seam is 9.5–11.32 m thick with an average thickness of 10.37 m, and is therefore classified as an extra-thick coal seam. The longwall top coal caving method is adopted for mining with a mining-to-caving ratio of 0.4. The roof in the working face is managed using the caving method.

Figure 2 shows the stratigraphy of AR 4106. The immediate roof of AR 4106 is sandstone with a thickness of 1.33 m and moderately developed cracks; the main roof is fine sandstone with a thickness of 10.52 m and dense and hard lithology; the immediate floor is aluminum mudstone with a thickness of 6.26 m; and the main floor is mudstone with a thickness of 12.20 m. X-ray diffraction experiments were carried out on the aluminum mudstone floor, as shown in Figure 3. The results show that the rock mass contains 59.9% clay minerals. A 500-mm bottom-coal layer is reserved on the floor to retain the mudstone strength due to the serious roof water seepage during roadway excavation. The water source was mainly confined fissure water with an inflow of 10–20 m³/h during excavation.

2.2. Original Supporting Design

The width and height of AR 4106 are 5500 and 3850 mm, respectively, and the roadway was originally supported by bolts and cables. The parameters are shown in Figure 4. The roof is supported by rows of eight rebar bolts of 22 mm in diameter and 2500 mm in length (parameter design is mainly based on the previous experience of field technicians). Each bolt has a preload force of 30 kN and array spacing of 800 mm. The bolts are used with W-steel strips and cables are applied to support the roof. Five cables with a diameter of 17.8 mm and length of 8800 mm were used in each row of the support during roadway excavation. Prior to mining of the adjacent working face, two cables were added to each row as reinforcement support (seven cables for each row of the roof). The measured preload force of each cable is 60 kN, the cable row spacing is 800 mm, and the cable is used with an I-shaped steel beam. The roof supports are reinforced with 100-mm steel mesh with a length and width of 5500 and 900 mm, respectively, and an overlap area of 100 mm. The
two sides of the roadway are supported by a row of rebar bolts of the same size as the roof, with five bolts in each row. The bolt row spacing is 800 mm and plastic mesh is used for watch protection. The length and width of the plastic mesh are 3400 and 1000 mm, respectively, with an overlap of 100 mm. Because the coal seam thickness is 10.37 m, the bolts and most of the cables are located within the coal seam itself.

| Column       | Lithology       | Thickness (m) | RQD (%) | Remark                        |
|--------------|-----------------|---------------|---------|-------------------------------|
|              | Mudstone        | 4.58          | 30      |                               |
|              | Fine sandstone  | 7.30          | 72      |                               |
|              | Mudstone        | 2.65          | 30      |                               |
|              | Fine sandstone  | 10.52         | 66      | Main roof                     |
|              | Sandstone       | 1.33          | 55      | Immediate roof                |
|              | 4#coal          | 10.37         | 20      | Coal seam with thinner dirt   |
|              | Aluminum mudstone | 6.26      | 26      | Immediate floor               |
|              | Mudstone        | 12.20         | 32      | Main floor                    |

![Figure 2. The stratigraphy of AR 4106.](image)

![Figure 3. X-ray diffraction pattern of aluminous mudstone.](image)
2.3. Roadway Failure Characteristics

The deformation and failure characteristics of the roadway were investigated on site using a ZHS2400 digital camera (Shanghai Beiter Safety Equipment Co., Ltd., Shanghai, China), and roadway deformation data were recorded during roadway excavation and mining of the adjacent working face. Figures 5 and 6 show that the roof support density is high with an average of seven cables and eight bolts in each row and a row spacing of 800 mm.

Figure 4. Sections of main support of AR 4106.

3. Field Tests and Results

3.1. Principles for a Support System

The roof support was completed over two construction periods in the original scheme and included an average of eight bolts per row (bolt diameter = 22 mm, length = 2500 mm, preloading force = 30 kN) with seven cables per row (cable diameter = 17.8 mm, length = 8.8 m, preloading force = 60 kN) and a row spacing of 800 mm. Despite the very

Figure 5. Field photos of roadway deformation and failure: (a) roof subsidence; (b) failure of supporting material; (c) floor heave; and (d) large bulging of the sides.
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Figure 6. Monitoring data of roadway deformation under original support: (a) roadway deformation during tunneling and (b) deformation of roadway during mining of adjacent working face.

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the side rock mass, which aggravated the convergent deformation of the two roadway sides and further enhanced the shrinkage of the effective roadway space.

According to the cross-border anchoring theory of thick roofs [31], with the increase of roof bolt length, the damage zone of the bolt end will move up gradually. The key difficulty of roof support in an extra-thick coal seam is that bolts should be anchored in the deep and stable rock mass. According to previous research [32,33], the length of the bolts should be more than 3.5 m under the condition of an extra-thick coal seam. Combined with the actual conditions and the previous experience of the research group, the flexible 4-m-long bolts are determined to replace rebar bolts in the roof. At the same time, to ensure the continuous transmission of stress between anchorage structures, the length of the cable is reduced to 6.1 m.

In the aspect of roadway stress optimization, the long-distance cantilever structure of the main roof must be destroyed. Considering the convenience of construction, deep-hole blasting technology is selected. The depth design of blasting drilling should consider the hard fine sandstone stratum of the roof. In this case, the vertical distance between the stratum and the coal seam is 28.28 m, which is the vertical length of the blasting borehole. Combined with the previous research [34] and the actual conditions, the blasting drilling angle is determined to be 40°, and the length of drilling and blasting is 44 m.

Based on the above analysis, a new surrounding rock control strategy considering deep-hole blasting pressure relief and thick anchoring zone were proposed. The new support system accounts for the following principles by integrating both field experience and the results of field investigations. (1) The bearing structure of the roof foundation is constructed using long flexible bolts to suppress the initial rock mass expansion and deformation. (2) Short cables are used to further strengthen the bearing structure of the roof foundation and enhance the anti-disturbance ability of the anchor structure. (3) Deep-hole blasting is adopted to pre-split the roof, which effectively reduces the coal pillar stress concentration, optimizes the roadway surrounding rock stress field, and minimizes damage to the anchor structure.

3.2. Roadway Support Scheme Based on Long Flexible Bolts

3.2.1. Main Support Scheme for Roadway

The roadway roof is mainly supported by long flexible bolts, as shown in Figure 7, with a diameter and length of 17.8 and 4000 mm, respectively. Each row is arranged with eight roots with a row spacing of 800 mm. Each flexible bolt is equipped with a 300 × 300 × 16 mm arched steel tray. A torque amplifier is used twice to increase the torsion of the flexible bolt to ensure that the pre-tightening torque of the nut is no less than 400 N·m. The roof network uses Φ6.0-mm steel fabric with a mesh size of 50 mm, length of 5600 mm, width of 1000 mm, and overlap area of the reinforcement fabric of 100 mm. The sides of the roadway are supported by five screw steel bolts with a diameter of 22 mm, length of 2500 mm, and row spacing of 800 mm. Each bolt is matched with a 150 × 150 × 10 mm steel support plate. The pre-tightening torque of the bolts is not less than 200 N·m. The mesh is #8 iron wire diamond mesh with a length of 4200 mm, width of 1100 mm, and overlap of 100 mm.

3.2.2. Short Cable Reinforcement Support

The roof should be reinforced twice prior to mining the adjacent working face and constructing the blasting borehole. A row of high pre-stressed cables with a diameter of 21.8 mm and length of 6100 mm are arranged in the middle of the flexible bolts in the roof. Five cables are arranged in each row with a row spacing of 1600 mm. Each cable has a large tray (300 × 300 × 16 mm) and the preloading force is not less than 120 kN.
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3.3. Deep-Hole Blasting for Hard-Roof Pre-Splitting

Deep-hole pre-splitting blasting was applied to destroy the hard roof structure in advance of AR 4106 and prior to mining WF 4105 to optimize the surrounding rock stress environment. The deep-hole blasting scheme is as follows:

1. The holes were cut in the roof of the coal pillar side of AR 4106. The opening position was 1.5 m from the coal pillar side and the final hole was in the coal pillar. The drilling arrangement is shown in Figure 8.

2. The construction parameters of deep-hole blasting:
   (a) Borehole azimuth: 90°.
   (b) Borehole inclination: 40°.
   (c) Aperture: 75 mm.
   (d) Length: 44 m.
   (e) Hole spacing: 10 m.
   (f) Charging length: 24 m/60 section.
   (g) Charge quantity: 66 kg.
   (h) Hole sealing length: 20 m.

3. Charge and detonation:
   (a) Construction machinery: ZDY-4000LR B-type full hydraulic tunnel drill for coal mining.
   (b) Explosives: Class-III permissible emulsion tube explosive for coal mining. The cartridge was 60 mm in diameter, 400 mm in length, and...
weighed 1.1 kg/roll. (c) The forward charge used a series connection of detonating cord + detonator + detonator to connect the initiation.

3.4. On-Site Monitoring and Result

Field measurements of the roof and two sides of AR 4106 were made during tunneling and mining of the adjacent working faces with the new support to determine the synergistic roof control mechanism of the long flexible bolt support and deep-hole blasting pressure relief scheme, as shown in Figure 9 and Table 1.

Figure 9a shows that surface displacement initially increased rapidly after roadway tunneling and then gradually slowed and stabilized. The main convergent deformation occurred 20 m from the excavation face. The maximum roadway roof subsidence under the new scheme was 35 mm and the maximum convergence of the two sides was 58 mm. Deformation of the roof and two sides was reduced by 69.57% and 81.93%, respectively, compared with the original scheme (Table 1). Upon mining of WF 4105, the roof strata of the mined-out side of the roadway sank to form a voussoir beam structure. Roadway deformation gradually increased owing to the stress disturbance until stabilizing. As shown in Figure 9b, the maximum subsidence of the roadway roof was 57 mm and the maximum convergence of the two sides was 114 mm; a reduction of 87.19% and 97.09%,
respectively, compared with the original support scheme. The deformation and stability distance of the roadway was approximately 150–160 m, which is essentially the same as the original scheme. The stability of the roadway was fundamentally altered during mining of the adjacent working face, as shown in Figure 10.

![Figure 10](image_url)

**Figure 10.** Field construction and tunnel maintenance effects: (a) torque amplifier preloading flexible bolt; (b) hydraulic pit drill construction blasting drilling; (c) maintenance effect of sides; and (d) maintenance effect of roof.

4. Discussion

4.1. Destruction Mechanism of Hard-Roof Roadway Surrounding Rock in Extra-Thick Coal Seams

The zonal fracture characteristics of roadway surrounding rock significantly affect the stability of roadway support structure [35]. The thickness of extra-thick coal seams above the roof is typically more than 4 m, which makes the roof support structure mostly arranged within the coal seam. Coal has a weak shear strength and primary fractures are often developed. Top coal affected by mining often undergoes expansion and dislocation deformation. A large number of joints and cracks develop on the microscopic level and failure and extensional deformation of the inner separation layer occur on the macro level [36]. The size of the plastic failure circle of roadway roof surrounding rock is typically larger than that of conventional roadways. The anchoring depth is therefore greater for the bearing structure of a shallow foundation supported by bolts. In this case, the supporting structure of the original scheme is limited by the rigid material and limited bolt length (2.5 m). The supporting range of the bolt remains in the fracture zone of the surrounding rock, as shown in Figure 11. However, the lack of bolt preloading force worsens the result. This causes a large number of stratum separation cracks to form within and outside of the bolt anchoring area, seriously damages the integrity of the roof rock mass, and strongly reduces the strength and stability of the shallow bearing structure constructed by the bolts. In this case, even the preload provided by the cable cannot enhance the strength of the shallow bearing structure and integrity of the rock mass. The anchor cable plays a more important role in suspending shallow fractured rock mass, but active reinforcement is not achieved. The roadway surrounding rock is thus affected by mining of the adjacent working face and large deformation and instability occur.
Failure characteristics of hard roof roadway in extra-thick coal seam.

The long distance of suspended roofs is another important cause of the severe roadway deformation and failure. According to the voussoir beam theory proposed by Qian [37], a voussoir beam structure will form on both sides of the stope after coal mining, as shown in Figure 12. In this case, the key strata of the roof are hard fine sandstone with high strength and strong self-stabilizing ability and the rock structure does not easily cave. This extends the cantilever length of the key block B that bears the high rock pressure above it. Caving rock blocks cannot effectively fill the goaf owing to the large movement space of the overburden rock in the goaf and small thickness of the direct roof in the fully mechanized caving face. The key block B of the voussoir beam structure cannot be supported by gangue in the goaf over long time periods. The key block B will continue to rotate and sink until its weight is balanced by transferring the pressure from the overburden to the coal body and pillar.

FLAC\textsuperscript{3D} software was used to simulate the surrounding rock stress field changes of AR 4106 under the influence of the mining of WF 4105. The model with dimensions is 268 m × 50 m × 100 m. The Mohr-Coulomb model is used to simulate the rock strata and coal seam. The horizontal and bottom sides are roller constrained. The vertical load is applied to the top boundary of the model which equals to the self-weight of the overlying strata. The simulation results are shown in Figure 13. As shown in Figure 13a,b, high stress concentrations formed on the side of the protective coal pillar near the roadway after mining WF 4105 and the stress environment of the surrounding rock changed. The maximum stress was 28.3 MPa and the stress concentration coefficient was 1.58. The concentrated stress in the coal pillar, which is far greater than the original rock stress, intensified the expansion of cracks in the roadway surrounding rock, which destabilized

Figure 11. Crack expansion of short bolt support.

Figure 12. Failure characteristics of hard roof roadway in extra-thick coal seam.

Figure 13. Stress field changes of AR 4106 under the influence of the mining of WF 4105.
and destroyed the support structure supported by short bolts and long cables, resulting in the full destruction of the roadway.

![Figure 13](image)

**Figure 13.** Numerical simulation results: (a) stress nephogram of mining numerical simulation at adjacent working face and (b) stress nephogram section.

### 4.2. Mechanism of Roof Strengthening Using Long Flexible Bolts

The strength and deformation resistance of the foundation bearing structure significantly improve upon increasing the bolt length and applying a large preload force over time. The operating principle is shown in Figure 14a. The flexible rod body overcomes the restriction of the roadway height on the length of the rigid bolt and effectively increases the bolt length to 4 m. The thickness of the rock mass supported by the longer bolt therefore increases, which can control the rock deformation mass over a wider range. This allows the bolt to pass through the fracture zone of the coal body and effectively control rock mass separation failure deformation within the anchorage zone. The anchoring end is located in the intact rock mass, which notably improves the bonding force between the anchoring agent and rock mass. The high applied preload force restrains the initial rock expansion deformation, reduces the damage to surrounding rock, increases the deflection of rock beam anchored by the thick-roof foundation, and significantly improves the anti-deformation ability. The deformation of the surrounding rock roof is reduced by 69.57% during roadway excavation. The roof stability can also relieve the extrusion deformation of the side, the latter of which is reduced by 81.93%.

![Figure 14](image)

**Figure 14.** Roof support structure: (a) the bearing structure of surrounding rock foundation is strengthened by long bolt and (b) the bearing structure of foundation is continuously strengthened by short cable.

Prior to mining WF 4105, a 6.1-m short cable was used as a secondary reinforcement of the anchor structure of the roadway roof foundation. The acting principle is shown in Figure 14b. Rock strata tend to be more disturbed when mining extra-thick coal seams. A roof fracture zone develops upward to prevent the foundation bearing structure of the
long-bolt support from being affected by the lateral support pressure. It is necessary to once more adopt a short cable to strengthen the rock beam thickness and roof stability. It is sufficient to handle the influence of the mining stress disturbance of the adjacent working face and improve the overall maintenance effect of the roadway during single-seam mining.

4.3. Deep-Hole Pre-Splitting Blasting Weakens the Hard Roof

The large suspended roof area worsens the stress environment of roadway surrounding rock, with high stress concentrations in the coal pillars. The horizontal stress of the roof and floor and highly concentrated vertical stress in the two sides lead to plastic failure and shattering-swelling deformation of the surrounding rock and notably damages the support structure. The entire hard roof structure must be destroyed to minimize the damage caused by the high stress field of the anchor structure. The blasting pressure relief principle for this case is shown in Figure 15.

![Figure 15](image)

**Figure 15.** Schematic diagram of pressure relief by deep-hole blasting: (a) blasting weakens the strength of rock mass around the borehole and (b) lateral masonry beam structure of stope after mining of adjacent working face.

Deep-hole blasting of the construction roof was carried out prior to mining WF 4105. The rock mass around the borehole was damaged under the combined action of the shock wave and high-pressure gas generated by the blasting, which promoted the generation of new fractures from the original fractures, and the hard roof integrity was destroyed, as shown in Figure 15a. The damaged hard roof was directly broken along the blasting crack after mining in the working face, as shown in Figure 15b, and the roof voussoir beam structure was modified. The cantilever distance and overhead area of block B of the long cantilever in the roadway surrounding rock was alleviated, the lateral support pressure of coal pillar was reduced, the stress environment of surrounding rock of roadway was optimized, the damage of surrounding rock anchoring structure was inhibited, and the supporting effect of roadway was significantly improved. The stabilization time of the block B contact gangue did not significantly change owing to the large mining space of the extra-thick coal seam. Therefore, although the stress level of surrounding rock effectively
decreased, the stabilization period of the rock mass structure cannot be reduced during the mining of adjacent working faces.

5. Conclusions

(1) The plastic failure circle is substantially larger in roadway roofs in extra-thick coal seams than in those of conventional roadways, and the traditional support range of screw bolts remains in the fracture zone. Main roofs that are dense and hard tend to increase the cantilever length of the key block B in the lateral voussoir beam structure of the stope. Extensive movement space of the overburden in fully mechanized caving mining increases the rotational subsidence distance and time of key block B. The pressure of the overlying strata is transferred to the coal pillar, which worsens the stress environment of the roadway surrounding rock. The unsteady support structure under short-bolt support is therefore further damaged.

(2) The adoption of a flexible rod body overcomes the tunnel height limitation on the traditional screw bolt length, which is increased to 4 m. The increased bolt length can effectively pass through the coal fracture zone and control the rock separation failure within the anchorage zone. The high applied preload restrains the initial expansion deformation of the rock mass, reduces the damage of the surrounding rock, and improves the deflection of the anchor rock beam with a thick-roof foundation. The deformation of the roof and side during roadway excavation is reduced by 69.57% and 81.93%, respectively, compared with the original scheme.

(3) Deep-hole pre-splitting blasting was used to promote the expansion and convergence of primary rock fractures. The damaged roof collapsed directly along the blasting crack after mining of the working face, which reduced the cantilever distance of the key block B in the voussoir beam structure. The high lateral support pressure in the coal pillar caused by the rotary subsidence of the long cantilever block B was effectively alleviated. This approach can optimize the stress environment of the surrounding rock and inhibit anchor structure damage. Under the proposed support scheme, the deformation of the roadway roof and side decreased by 89.19% and 87.09%, respectively, after the first mining period and the supporting effect was significantly improved.

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