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Optimization of Roadway Support in Deep Coal Seams Based on Loose Zone Testing

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Abstract

To address the problem of fractured surrounding rock and support failure in deep coal seam roadways under high-stress conditions, this study takes the auxiliary transportation roadway (west) in the 2103 panel of a mine in Shanxi Province as the research background. The ZKXG30 type mining borehole imaging device was used to conduct precise testing of the loose zone. Five observation boreholes were arranged at key cross-sections of the roadway to systematically analyze the fracture development characteristics and distribution patterns of the roof, shoulder, sidewalls, and floor corners. The results indicate that the loose zone thickness of the surrounding rock ranges from 2.36 m to 4.38 m, classifying it as a large loose zone. The shoulder area exhibits the most severe damage, showing an overall butterfly-shaped failure pattern. Based on the test results, an optimized combined support system of "ordinary cable bolts + grouting cable bolts + mesh + W steel belt + shotcrete" was designed. The sidewalls were reinforced using Φ22 mm × 5300 mm hollow grouting cable bolts. Field monitoring results show that after reinforcement, the maximum convergence of the two sidewalls is 199 mm, and the maximum roof–floor convergence is 211 mm, both within the safety limits. The stability of the surrounding rock was significantly improved.

Keywords

Loose Zone; Borehole Imaging; Grouting Cable Bolts; Support Optimization; Surrounding Rock Control.

1. Introduction

Under the conditions of underground coal mining, the stress state of the surrounding rock transitions from a three-dimensional stress condition to a biaxial stress state, leading to the release of accumulated energy in the rock mass. When the concentrated stress exceeds the strength of the surrounding rock, failure occurs and continues until the post-peak residual strength of the rock exceeds the applied stress, at which point the failure ceases. As a result, an irregular ring-shaped zone forms around the excavated opening, known as the loose zone of the surrounding rock[1]-[4]. The boundary of the loose zone is defined by the interface between newly developed fractures and the intact rock. To investigate the formation mechanism of the loose zone around roadways, researchers have employed various approaches, including theoretical analysis and field tests. These studies have led to the development of classification-based support technologies for the loose zone, forming a systematic control theory for managing surrounding rock stability[4]-[6].

Various methods are available for detecting the loose zone in roadway surrounding rock, such as the ultrasonic wave method, electrical resistivity method, permeability method, and multi-point extensometer method[7][9]. However, these techniques do not provide a direct visual assessment of the loose zone thickness[10]-[12]. In this study, borehole imaging technology was applied to visually inspect boreholes in the 2103 Panel Auxiliary

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Transportation Roadway (West) of a coal mine in Shanxi Province. The development and distribution of fractures in the roof, shoulder, sidewall, and floor corner were examined, enabling an analysis of the variation characteristics of the surrounding rock loose zone in this roadway. Based on the findings, the support design was optimized to maintain the integrity of the support structure. The effectiveness of the support was evaluated by monitoring roadway deformation, providing a valuable reference for safe and efficient mining under similar geological conditions.

2. Engineering Overview

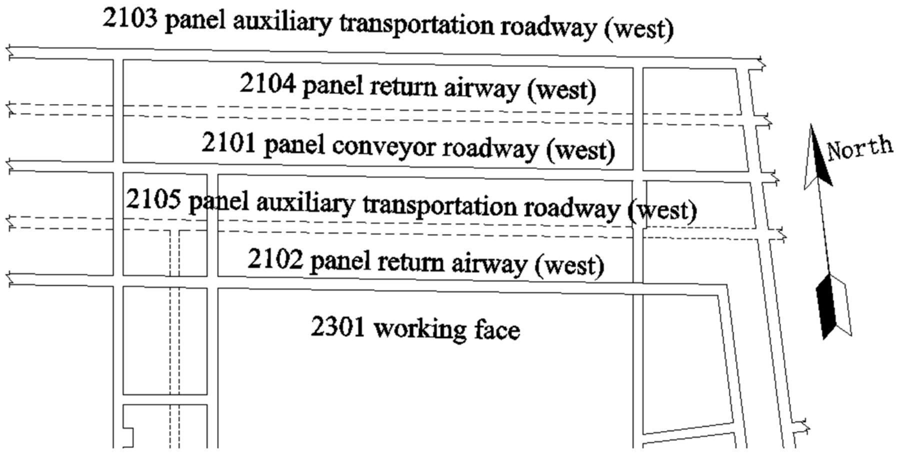

Figure 1. Plan layout of the 2103 Panel Auxiliary Transportation Roadway (West)

content	lithology	thickness(m)	burial depth(m)	rock histogram
main roof	sandy mudstone	2.53	766.98	
immediate roof	mudstone	2.30	769.28	
coal	3# coal	4.65	773.93	
immediate floor	mudstone	2.74	776.67	
main floor	siltstone	1.00	777.67	

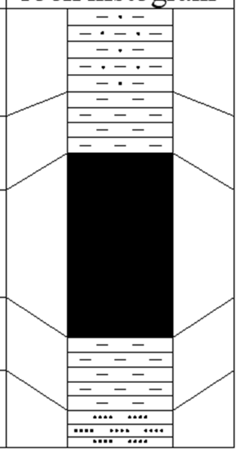

Figure 2. Comprehensive Geological Column

The ground elevation of the 2301 working face in the Shanxi mine ranges from 980 m to 1065 m, with the coal seam floor elevation ranging from 210 m to 312 m. The northern boundary of the 2301 working face is the west main roadway of the Nansu Second Panel currently under excavation, while the eastern boundary is an already excavated main roadway of the Second Panel. The seam being mined is the No. 3 coal seam of the Lower Permian Shanxi Formation, with a thickness ranging from 4.5 m to 4.8 m, averaging 4.65 m, categorized as a thick coal seam. The coal seam dip angle varies between 0° and 8°, with an average of 5°.

The auxiliary transportation roadway (west) of the 2103 panel serves as the return air roadway for the Second Panel and remains operational throughout its mining and excavation period. Located to the north of the 2301 working face, its coal seam floor elevation ranges from 201 m to 146 m, with an overburden thickness of 843.6–876.8 m. The roadway is oriented east–west,

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excavated along the roof of the coal seam while leaving a layer of bottom coal. The entrance is positioned at the west side of the intersection between the 2103 panel's auxiliary transportation roadway (west) and the northern section of crosscut No. 6, adjacent to the 2101 panel's belt roadway (west). The roadway adopts a rectangular cross-section with an excavation width of 5900 mm, height of 4500 mm, and an excavation area of 26.55 m^2. Within the No. 3 coal seam, the immediate roof consists of mudstone with an average thickness of 2.30 m, the main roof is sandy mudstone averaging 2.53 m, the immediate floor is mudstone averaging 2.74 m, and the main floor is siltstone averaging 1.00 m in thickness.

3. Loose Zone Testing

3.1. Monitoring Equipment and Principles for Loose Zone Testing

The testing employed a ZKXG30-type mining borehole imaging trajectory detection device, mainly consisting of a host imaging analyzer, probe, depth-measuring pulley, and auxiliary components such as cable rack, connecting cable, charger, and USB adapter. Additional accessories, including a probe centering protector and push rods, were used for horizontal and inclined borehole tests. The primary components of the equipment are shown in Figure 3.

The probe contains an LED white-light source (with adjustable brightness circuit) and a camera for capturing borehole wall images. It is also equipped with a high-performance 3D electronic compass for measuring the azimuth and inclination of the borehole. The video, control, and compass data are transmitted via cable to the host, which, combined with depth pulse signals, determines the probe's position and performs real-time video recording and image stitching. The system supports full-hole or partial recording and can synchronize or independently perform image stitching. As the probe moves deeper, the system automatically composes a continuous unfolded image of the borehole wall.

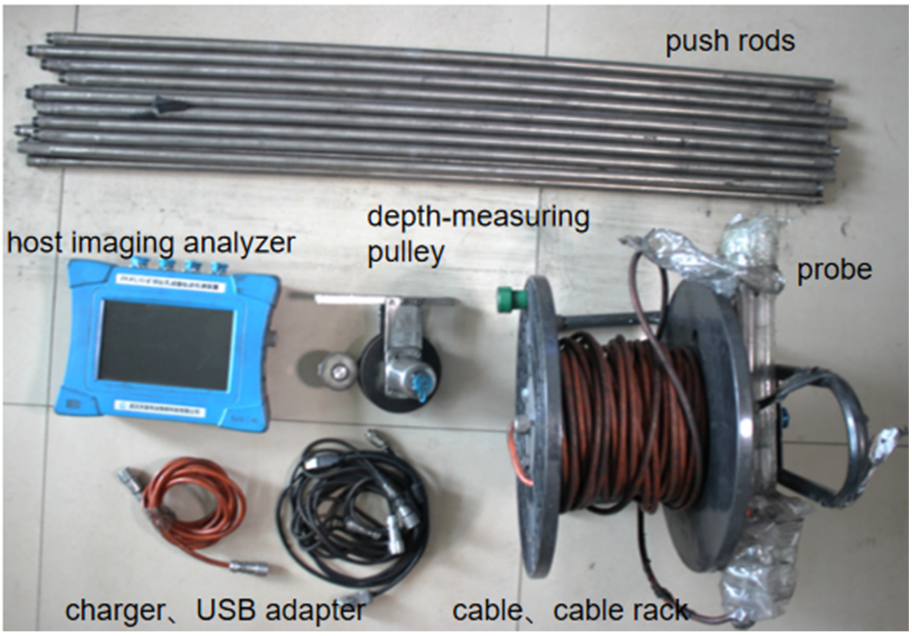


Figure 3. ZKXG30 Mining Borehole Imaging Trajectory Detection Device

3.2. Monitoring Section Layout

To assess the degree of rock loosening around the 2103 Panel Auxiliary Transportation Roadway (West), a monitoring section was established 1000 m from the entry point, as illustrated in Figure 4. Initially, seven boreholes were planned for borehole imaging; however, due to the presence of belts and ventilation equipment on the left side, the left upper and lower corner boreholes were not drilled, resulting in five completed observation boreholes in this section.

Details of each borehole are as follows: Borehole No.1 (floor corner) was designed to be 5 m deep with an inclination of about 45° to the horizontal but was not completed. Borehole No.2 (sidewall) was 9 m deep. Borehole No.3 (shoulder) was designed to be 9 m deep with an inclination of about 20° to the vertical but was not completed. Borehole No.4 (roof) was 9 m deep; Borehole No.5 (shoulder) was also 9 m deep with a 20° inclination to the vertical; Borehole No.6 (sidewall) was 9 m deep; and Borehole No.7 (floor corner) was 5 m deep with a 45° inclination.

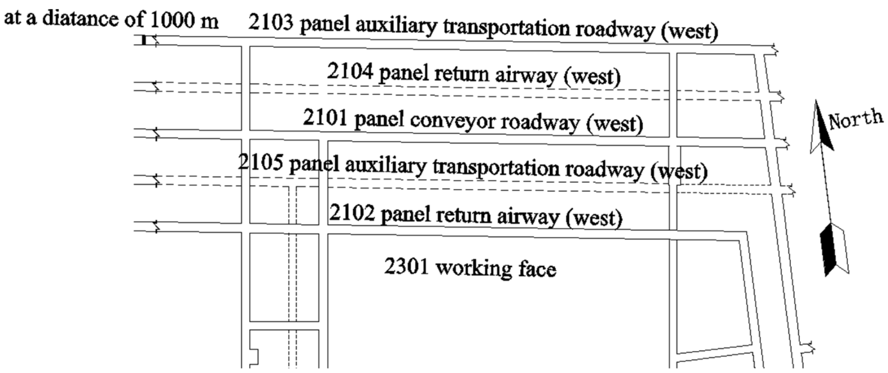


Figure 4. Layout of Borehole Observation Points

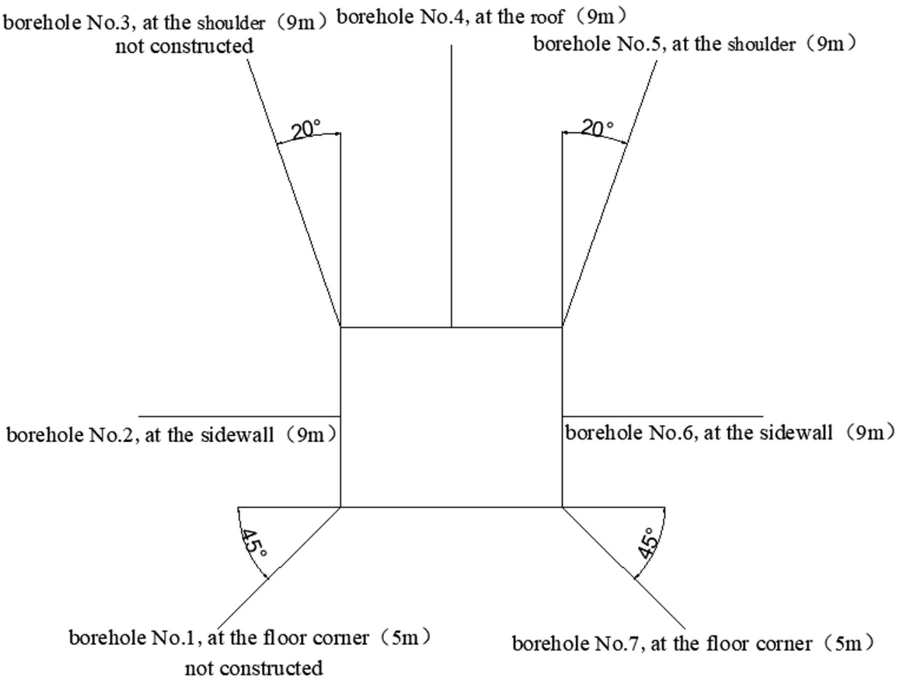


Figure 5. Arrangement of Observation Boreholes in Monitoring Section

3.3. Analysis of Loose Zone Monitoring Results

Given that the roadway was excavated within the coal seam, where the surrounding rock has low strength and poor self-stability, and the roof and floor consist of weak mudstone susceptible to collapse during drilling, the monitoring was conducted immediately after borehole completion to ensure data accuracy and to capture real-time fracture development within the surrounding rock for precise delineation of the loose zone.

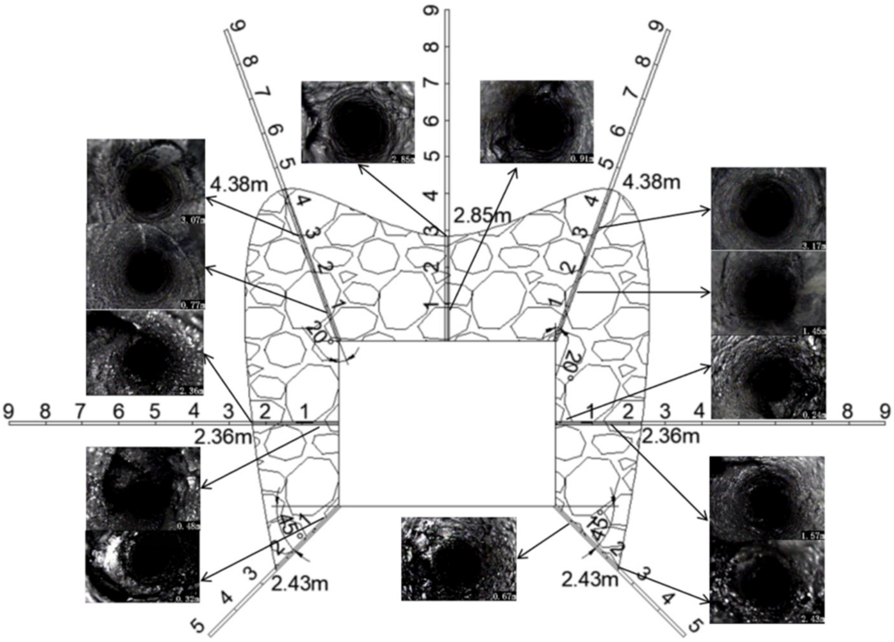


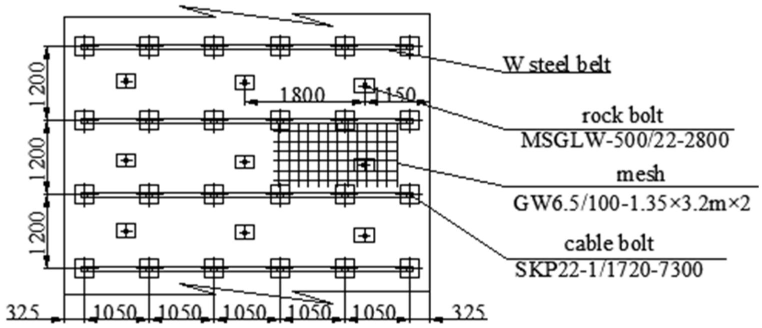
Figure 6. Distribution Map of Loose Zone Range

Due to site constraints, boreholes No.1 and No.3 could not be visually inspected. Considering the geological similarity and comparable deformation characteristics of both sidewalls, the loose zone range of borehole No.3 was assumed to be consistent with that of No.5, and that of No.1 with No.7. Based on this, the overall distribution of the loose zone in the monitoring cross-section was plotted as shown in Figure 6.

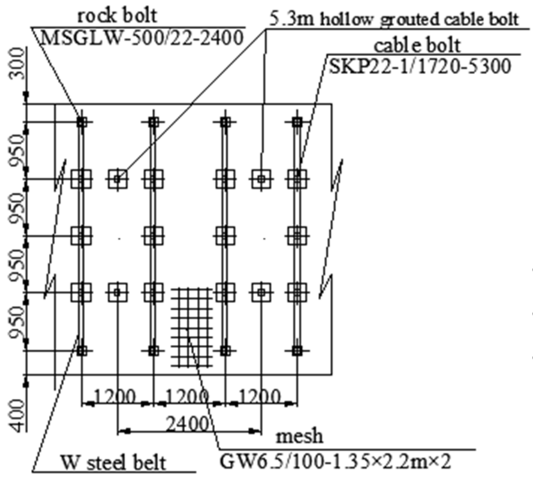
Comprehensive analysis of the borehole imaging data from the five observation points indicates that the shoulder region experienced the most severe damage, with a maximum depth of 4.38 m. The damage range along the sidewalls was comparatively smaller; however, since the roadway lies entirely within the coal seam, the coal mass in the sidewalls is weak and fragmented, with limited bearing capacity, prone to continuous deformation under mining-induced stress. Hence, sidewall reinforcement is particularly crucial in the support design.Overall, the roof, shoulder, and sidewall regions are significantly affected by mining stress, resulting in poor stability. The thickness of the loose zone varies from 2.36 m to 4.38 m, categorizing it as a highly unstable surrounding rock type that requires a systematic combined support approach.

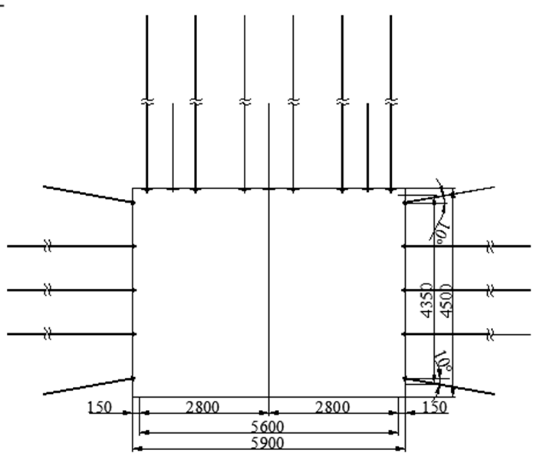
4. Optimization of Roadway Support Scheme

In the original support design, the roof was reinforced using 7.3 m-long cable bolts, which were effective in controlling roof deformation and maintaining arch foot stability, meeting the basic requirements for roof support. However, since the entire roadway was laid within the coal seam, the sidewalls composed of fractured coal with low strength and poor self-supporting capacity were prone to continuous deformation under mining-induced stress. The original sidewall support structure was insufficient to restrain this progressive deformation.To address this, the optimized design focuses on reinforcing the sidewalls through the use of grouting cable bolts, which effectively improve the integrity and mechanical strength of the fractured coal mass, transforming the anchorage mode from end-anchoring to full-length anchoring, and forming a framework-type load-bearing structure.Based on the above analysis, a combined support system of “ordinary cable bolts + grouting cable bolts + mesh + W steel belt + shotcrete” was developed, achieving systematic control over both roof and sidewall stability.



(a) Plan View





(b) Side View (c) Cross-section View

Figure 7. Reinforced Support Design of the 2103 Panel Auxiliary Transportation Roadway (West) Using Grouting cable bolts

In the 2103 Panel Auxiliary Transportation Roadway (West), a rectangular cross-section support system was adopted, with an excavation width of 5.9 m and a height of 4.5 m.To address the issues of fractured surrounding rock and support failure, a reinforcement scheme incorporating grouted cable bolts was implemented based on the original support design. The optimized support layout is shown in Figure 7.For the roof support, ordinary cable bolts of type SKP22-1/1720-7300 were used. The spacing between rows and columns was 1050 mm × 1200 mm, with six cable bolts per row, and the cable bolt nearest to the roadway shoulder was placed 325 mm from the sidewall. In addition, rock bolts of type MSGLW-500/22-2800 were installed with a spacing of 1800 mm × 1200 mm, three per row, with the side bolt located 1150 mm from the roadway shoulder.For the sidewalls, a combined support system of rock bolts and cable bolts was employed. Ordinary cable bolts of type SKP22-1/1720-5300 were installed at positions No. 2, No. 3, and No. 4 along each sidewall, with a spacing of 950 mm × 1200 mm, and three cable bolts per row. Rock bolts of type MSGLW-500/22-2400, each 2400 mm in length, were installed at the top and bottom corners of each sidewall. The top corner bolt was positioned 300 mm from the roof, while the bottom corner bolt was 400 mm from the floor.Furthermore, hollow grouted cable bolts with a diameter of 22 mm and a length of 5.3 m were used, spaced at 1900 mm × 2400 mm, with two bolts per row on each sidewall. Each cable bolt was anchored using two cartridges of resin type MSK2335 and two cartridges of type MSZ2360, while each rock bolt was anchored using one cartridge of MSK2335 and two cartridges of MSZ2360.

5. Engineering Test

5.1. Test Plan

A 1200 m section of the 2103 Panel Auxiliary Transportation Roadway (West) was selected as the test area for implementing the optimized combined support system. To verify the support performance, the convergence of both sidewalls and the approach of the roof and floor were continuously monitored using the convergence rod method, with measurements taken once daily over an 80-day monitoring period.

5.2. Analysis of Monitoring Results

The monitoring results (Figure 8) reveal that roadway deformation exhibited a trend of “rapid initial increase followed by stabilization.” During the first 25 days, deformation developed rapidly, with high convergence rates of both sidewalls and roof–floor displacement. After 25 days, the deformation rate gradually decreased, reaching stability after approximately 45 days.Final data indicate that the maximum sidewall convergence reached 199 mm, and the maximum roof–floor convergence was 211 mm—both within the permissible safety limits. Compared with the original support scheme, sidewall convergence control improved by more than 30%, and roof–floor displacement decreased by 25%.

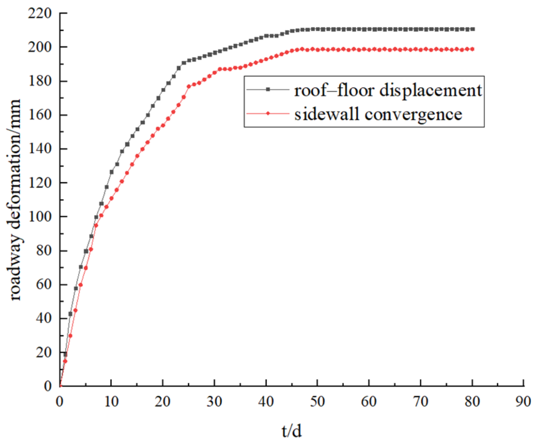


Figure 8. Convergence Deformation of Roadway Surrounding Rock

6. Conclusion

(1)Borehole imaging technology revealed the fracture development and distribution characteristics of the roadway surrounding rock. The maximum loose zone depth was 2.36 m at the sidewalls, 2.85 m at the roof center, and 4.38 m at the shoulder, presenting a typical butterfly-shaped failure pattern. Therefore, support in the shoulder and sidewall areas should be particularly strengthened.

(2)Based on the loose zone testing results, a combined support system of “ordinary cable bolts + grouting cable bolts + mesh + W steel belt + shotcrete” was proposed. The use of grouting cable bolts in the sidewalls effectively enhanced the integrity and mechanical performance of the fractured coal, realizing the transition from end anchoring to full-length anchoring and forming a framework-type bearing structure.Field test results demonstrate that the maximum sidewall convergence (199 mm) and roof–floor displacement (211 mm) remained within safe limits after support optimization, effectively controlling surrounding rock deformation.

(3)Borehole imaging technology provides accurate identification of the loose zone range and failure mode, offering crucial data for the scientific design of support parameters. The combined control approach integrating grouting cable bolts conventional support elements can effectively improve the stability of deep fractured roadway rock, providing both theoretical and practical reference for support design under similar geological conditions.

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