Research Article

Study on Anchor Cable instead of Single Hydraulic Prop Support in Advance Support of Deep Roadway

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In order to solve the problems of complicated advanced support process, high labor intensity, affecting the rapid advance of working face and the destruction of roof bolt (cable) by advance single hydraulic prop in ultra kilometer deep mine roadway, the deformation characteristics of roadway surrounding rock is analyzed. Taking the 27304 working face of Wanglou coal mine as the engineering background, numerical simulation, field monitoring, and theoretical calculation were used to analyze the deformation characteristics of roadway surrounding rock within the advanced influence range of 27304 working face. This paper puts forward the active advance support technology scheme, in which grouting anchor cable replaces the existing single hydraulic prop in the advance influence range of the working face in the ultra-kilometer deep mine, and observes and analyzes the deformation and failure characteristics of the surrounding rock of the working face advance roadway. The numerical simulation results show that in the advanced influence range of deep roadway, grouting anchor cable was used to replace the previous single hydraulic prop, and the vertical stress at both ends of the working face decreased by 15 MPa, with a decrease rate of 33.3%; the displacement of roadway roof, floor, and two sides decreased by 10 mm, 55 mm, and 20 mm, with a decrease rate of 40%, 68.75%, and 47.6%, respectively. The field monitoring results show that the roof separation is obviously improved after using grouting anchor cable as the active advance support scheme. It solves the problem of safe and efficient production faced by the ultra-kilometer deep shaft in Wanglou coal mine and provides theoretical and technical support for unmanned double roadway advance support under the condition of safe and efficient mining.

1. Introduction

With the depletion of China's shallow coal resources, China's coal mines are extending to the depth at the mining speed of 8–12 m per year [1, 2]. The mining of coal resources is gradually turning to the deep [3]. The engineering geology and hydrogeology conditions of roadway surrounding rock mass are becoming more and more complex, the physical and mechanical properties of deep rock mass change [4–6], the initial ground stress of the original rock is destroyed by engineering excavation disturbance, and the redistribution of ground stress leads to secondary stress field, which is the main reason for the occurrence of geological disasters such as roof caving and floor heave of surrounding rocks, and seriously threatens the safe and efficient production of mines

[7, 8]. The high ground stress along with the deep buried face of deep roadway increases the requirement of support. The supporting area of single support roof is small, and the problems of roadway roof sinking and coal side moving increase easily occur, which is difficult to meet the safety production requirements of ultra-deep working face. According to statistics, the total length of roadway of new coal mines in China is up to 12000 km every year, among which, the bolt support rate has reached over 80%, and that of state-owned coal mines is higher [9]. With the development of roadway support technology in coal mine, the proportion of bolt support will continue to increase [10, 11].

In view of the roadway active support, domestic and foreign scholars have carried out a lot of research. Du [12] proposed a multidimensional active control technology to

control the stability of surrounding rock of roadway by using a new type of high-prestressed truss cable supporting roof and cable-channel steel truss supporting roadway side. Practical engineering shows that this technology can effectively improve the self-settling ability and anti-deformation ability of surrounding rock. Wang et al. [13] proposed the secondary support theory of high-stress composite active support system. Through numerical simulation and field observation, it was shown that the support system could effectively control the deformation and failure of surrounding rock. Zhao and Ma [14] adopted the methods of theoretical analysis, numerical calculation, and engineering practice, and by analyzing the reasons for failure of bolt-anchor cable support, they proposed a technical method to increase the extension of anchor cable. Du et al. [15] established the mechanical model of the composite active supporting system and obtained the critical condition of the pretension Qy value when the truss anchor cable structure in the composite active supporting system reaches the balance of the system. Yao et al. [16] proposed the technical idea of adopting proactive support to replace proactive passive support in the area affected by the advance abutment pressure of the stoping face and realized the one-time active support in the full cycle of the service of the stoping roadway. Zheng and Wang [17] studied the mechanical adaptability of the bolt system accessories, and the results show that the tray arch structure has a significant impact on its bearing capacity. Cao et al. [18] established the two-phase failure mode of the bolt support system. The failure modes of cable bolting are discussed using a bond strength model as well as an iterative method. The interfacial shear stress model for ribbed bar is introduced, and a closed form solution is obtained using a triline stress strain relationship. Wu et al. [19] used numerical modeling to investigate the stress redistribution around roadway under different horizontal stress orientations. The influence of horizontal stress direction, pillar size between the roadways, and the excavation sequence of the roadways was examined. A reinforcing system which consists of high-strength rock bolts and cables was applied at the five roadways. Both the bolts and cables were installed with high pretension forces along the whole length of them. Field observation indicated that this reinforcing system was able to effectively restrain the deformation and failure of rock mass surrounding roadways. He et al. [20], through 2D finite element method (FEM), conducted numerical studies to investigate the arching effect and the importance of rock bolting. Results show that σ m could be used as a reliable indicator to assess the arching effect.

However, most of the above research studies focus on shallow buried roadways, while there are few related research studies on super-kilometer deep buried roadways. Based on the research background of 27304 working face roadway in Wanglou coal mine, aiming at the serious damage of roadway roof by high ground stress and the large investment of advanced support personnel, this paper adopts theoretical analysis, field measurement, and other means to study the application of grouting anchor cable instead of single hydraulic prop in the advanced roadway of coal mining face. The original advanced support system is optimized to ensure the support strength and further reduce the labor intensity and advance support cost.

2. The Engineering Situation

Wanglou coal mine 27304 working face is located in the north wing of seven mining area. Ground elevation is +33.3 m ~ +34.3 m. Working face elevation is −1141 m ~ −1029 m. The depth of coal seam is 1105 m, the length of working face advancing direction is 1500 m, the width of working face is 120 m, the width of roadway is 4.9 m, the mining height is 4.5 m, and the cutting depth is 0.8 m. According to the actual situation exposed by the two lanes on the working face, there is no false roof on the working face roof, but the direct roof is siltstone. The roadway layout scheme of 27304 working face is shown in Figure 1, and the top and bottom plates of the coal seam are shown in Figure 2.

3. Design of Active Support Scheme

According to the actual situation of Wanglou coal mine, the concrete construction scheme of anchor cable replacing single support in advance supporting section of roadway in working face is designed by theoretical calculation and field experience.

3.1. Original Support Scheme. Two groups of ZQL2 × 5000/21/40 advanced supports and one group of ZT14000/19/40 end frame combined with single pillar shed support were used for the advance support of the working face. The shed was supported by 3.5 m–4.0 m "π" beam combined with single hydraulic pillar, one beam with three pillars, shed spacing of 1.0 m, middle pillar which is 2.5 m from the upper pillar, and middle pillar which is 1.0 m from the lower pillar.

Three groups of ZQL2 × 5000/21/40 advance supports, single hydraulic prop, and hinged top beam support are used for the advance support of the lower channel of the working face, and single row prop support is used with the column spacing of 1.0 m.

3.2. Roof Bolt Support Strength. The bolts on the roof were made of Φ 22 × 2500 mm resin bolts with 7 bolts in each row, and the row spacing between bolts was 800 × 800 mm. Anchor is fastened with rubber washer and standard nut.

The breaking load of roof bolt is

$$F_1 = 95 \text{ kN}. \tag{1}$$

The support density of roof bolt is

$$p_1 = \frac{7}{a_1 \times a}, \tag{2}$$

where p_1 is roof bolt support density; a_1 is the row spacing of roof bolts, $a_1 = 800$ mm; and a is the width of working face roadway, $a = 4900$ mm.
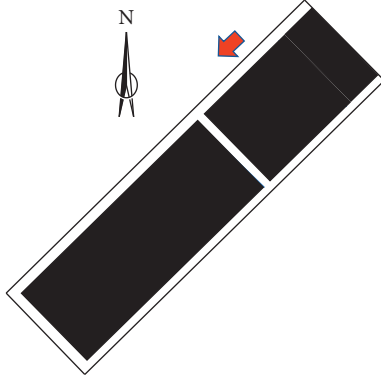
FIGURE 1: Roadway layout of 27304 working face.

Number	Stylolitic	Thickness/m	Rock category
1		27	Siltstone
2		17	Mudstone
3		4.5	Coal seam
4		7	Mudstone
5		22	Fine sandstone

FIGURE 2: Roof and floor of coal sea.

$$p_1 = \frac{7}{a_1 \times a} = \frac{1.79}{\text{m}^2}. \tag{3}$$

The support strength of roof bolt is

$$W_1 = F_1 \times p_1. \tag{4}$$

Get $W_1 = 0.170$ MPa.
Then, the supporting force of roof bolt is

$$Q_g = P_1 \times a \times a_1. \tag{5}$$

3.3. The Supporting Strength of Roof Anchor Cable.
Three anchor cables were arranged to cooperate with anchor cable

TABLE 1: Relationship between number of anchor cables and row spacing.

a_s (mm)	800	900	1000	1200	1400	1600
n	2.56	2.88	3.2	3.84	4.48	5.12
N/root (recommended)	3	3	4	4	5	6

tray to strengthen the support of roadway roof. The anchor cables were made of low relaxation steel strands of Φ 22×8000 mm, and the row spacing was 1600×1600 mm.

The breaking load of roof anchor cable is

$$F_2 = 583 \text{ kN}. \tag{6}$$

Roof anchor cable support density is

$$p_2 = \frac{3}{a_2 \times a}, \tag{7}$$

where p_2 is roof anchor cable support density; a_2 is top plate cable row distance, $a_2 = 1600$ mm; and a is the width of working face roadway.

We can solve

$$p_2 = \frac{3}{a_2 \times a} = \frac{0.38}{\text{m}^2}. \tag{8}$$

Roof bolting strength is

$$W_2 = F_2 \times p_2. \tag{9}$$

Get $W_2 = 0.223$ MPa.

3.4. Checking the Original Support Strength during Tunneling.
Through the above calculation, it can be known that the roof bolt and anchor cable arranged during tunneling can provide support strength as follows:

$$\begin{aligned} W_1 &= 0.170 \text{ MPa}, \\ W_2 &= 0.233 \text{ MPa}. \end{aligned} \tag{10}$$

Therefore, it can be concluded that the support strength provided by the support during the excavation of 27304 roadway is 0.393 MPa, which cannot reach the reasonable support strength of advanced support.

3.5. Reinforcement Parameters of Anchor Cable instead of Single Pillar in Advance Support during Stoping.
According to the width of the roadway which is 4900 mm, it is concluded that the supporting strength to be reinforced is 0.382 MPa.

The specification of anchor cable was Φ 22.0 $\times$ 8000 mm. The leading anchor cable tension load is $F_s = 583$ kN. The relationship between the number of anchor cables and row spacing is as follows:

$$\begin{aligned} n &= p_s \times a_s \times a, \\ p_S &= \frac{P_S}{F_S}, \end{aligned} \tag{11}$$

where n is the number of anchor cable roots; P_s is the support strength to be provided; P_s is the support density of

advance anchor cable; a_s is the advance anchor cable row distance; and a is the width of working face roadway, $a = 4900$ mm.

The relationship between the number of anchor cables and the row spacing can be obtained as shown in Table 1.

The relationship between the number of anchor cables to be increased and the row spacing of anchor cables was obtained. According to the specific geological conditions of Wanglou coal mine and the support layout during excavation, the row spacing can be 900 mm, each row can be added with three Φ 22.0 × 8000 mm anchor cables, and the roof can be supported with W-type steel belt, with the row spacing of 2400 × 900 mm. Combined with the actual site support, the row spacing between reinforcement cables can be set at 2400 × 800 mm. According to the calculation, the specific layout is as follows.

3.5.1. Upper Roadway of Working Face. The design net width and height of 27304 upper roadway are 4.9 m and 3.7 m, respectively, which was constructed along the roof of coal seam. During the excavation, bolt mesh or anchor cable were used for roadway support. The bolts on the roadway roof were made of Φ 22 × 2500 mm resin bolts with row spacing of 800 × 800 mm. The side part adopts Φ 22 × 2500 mm right-handed full thread anchor rod, and the row spacing between anchor rods was 900 × 800 mm. Three anchor cables were arranged in each row of the roadway roof to cooperate with the anchor cable tray to strengthen the support. The anchor cables were made of low relaxation steel strands with a spacing of 1.6 × 1.6 m. The specific layout is shown in Figure 3.

3.5.2. Lower Roadway of Working Face. The width and height of the Lower roadway of 27304 are 4.3 m and 3.7 m, respectively, and the construction is carried out along the roof of coal seam. The roof and side of roadway are provided with Φ 20 × 2600 mm high-strength pretension bolt, the row spacing of top bolt is 800 × 800 mm, and the row spacing of side bolt is 1000 × 800 mm. The low relaxation steel strand of Φ 22 × 8000 mm is used as the anchor cable. Three anchor cables were arranged in each row, and the row spacing between anchor cables was 1600 × 1600 mm, which was arranged in the middle of two adjacent rows of bolts.

4. Numerical Simulation Analysis

Based on the geological conditions of 27304 working face of Wanglou coal mine, combined with the anchor cable support scheme of advance support section and the single support scheme, FLAC^{3D} finite element numerical simulation software was used to carry out a comparative analysis on the modeling of the two support types. The influence range of stress after roadway excavation is about 3 ~ 5 times the width of roadway. Based on the actual production of coal mine, the model size is finally selected: $X \times Y \times Z = 180$ m × 120 m × 100 m. In order to reduce the error caused by the grid division, the length-width ratio of the grid should not be greater than 5. For the key research area, the grid can be densified. The unit size far from the roadway area is about 1 m, and the size close to the roadway
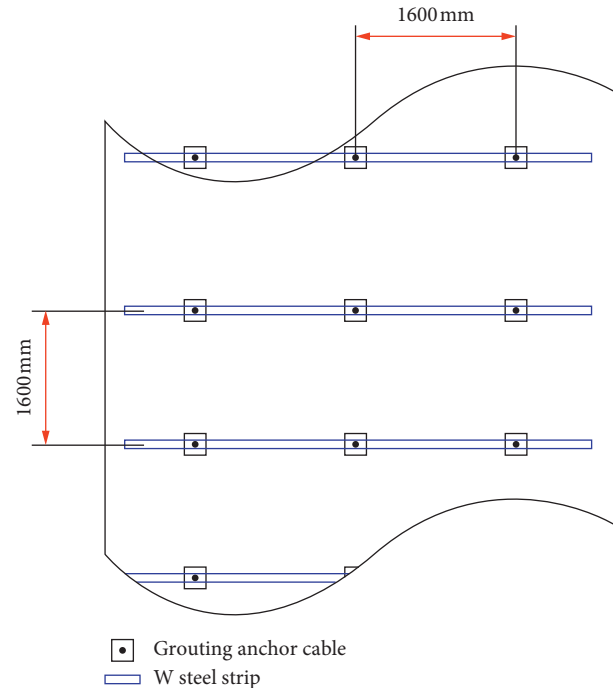

FIGURE 3: Schematic diagram of advanced support scheme design.

area and the roadway excavation area is 0.5 m. For the key research area, the grid can be densified, and the whole model is a cube. A uniform load of 26.5 MPa was applied to the roof of the model to simulate the pressure of rock strata along the negative side of X-axis; horizontal stress with a lateral pressure coefficient of 0.95 was applied around the model to simulate the horizontal stress. The bottom of the model was restrained to move in vertical direction, and the profile was restrained to move in horizontal direction. The initial displacement in each direction is 0. The establishment of 3D model is shown in Figure 4. See Table 2 for the lithological physical and mechanical parameters of each rock mass in the model.

The Mohr-Coulomb model was used in the calculation, "Cable" structural element was used to simulate bolt support, and "Beam" structural element was used to simulate single hydraulic prop support. The cloud diagram of the maximum principal stress section of the model is shown in Figure 5. The cloud diagram of vertical stress profile section is shown in Figure 6. By analyzing the cloud map of the simulation results, it can be found that the maximum principal stress on the end face of the anchor cable support is 30 MPa and the stress is concentrated at the front of about 5 m. The maximum principal stress on the end face of the monomer support reaches 45 MPa, and the stress concentration effect is increased. The normal stress of roof and floor is 97.5 MPa, and anchor cable support has a wider range of higher vertical stress distribution than monomer support.

The vertical displacement section of roadway surrounding rock is shown in Figure 7, and the section of horizontal displacement section is shown in Figure 8. It can be seen from the figure that the maximum displacement and settlement of the roof, the maximum displacement of floor heave, and the maximum displacement of side slope of single
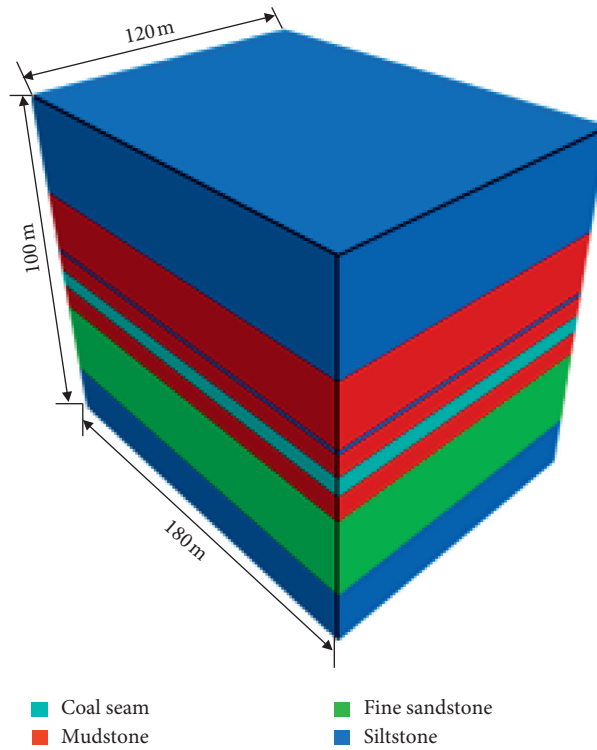
FIGURE 4: Numerical simulation model of mining roadway.

TABLE 2: The physical and mechanical parameters of surrounding rock.

Rock category	Thickness (m)	Shear modulus (GPa)	Bulk modulus (GPa)	Cohesion (MPa)	Internal friction angle (°)	Tensile strength (MPa)
Siltite	27	2.8	7.21	3.1	33	0.8
Mudstone	17	4.13	2.41	1.78	25	0.7
Siltite	1.3	2.5	6.8	2.8	31	0.8
Mudstone	6.2	4.2	2.45	1.8	26	0.9
Coal seam	4.5	1.35	2.13	1.21	30	0.12
Mudstone	7	4.19	2.48	1.75	34	0.8
Fine sandstone	22	2.8	7.21	3.1	33	0.8
Siltite	15	3.05	7.25	3.5	32	0.8

(a)

(b)
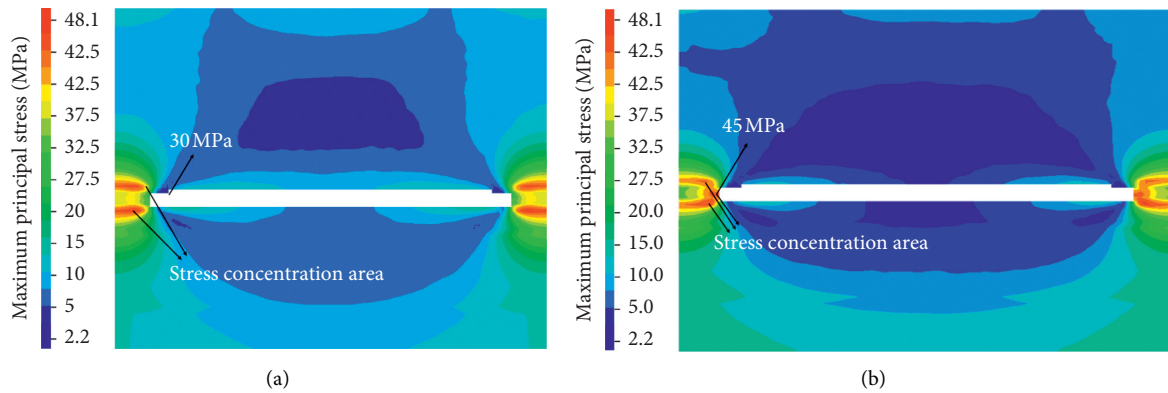
FIGURE 5: Section nephogram of maximum principal stress profile of different support types. (a) Single hydraulic prop support. (b) Anchor cable support.
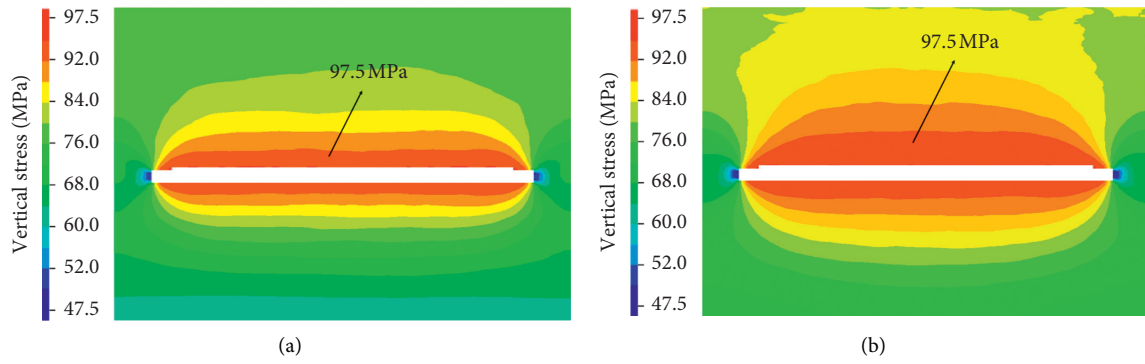
FIGURE 6: Vertical stress section nephogram of different support types. (a) Single hydraulic prop support. (b) Anchor cable support.
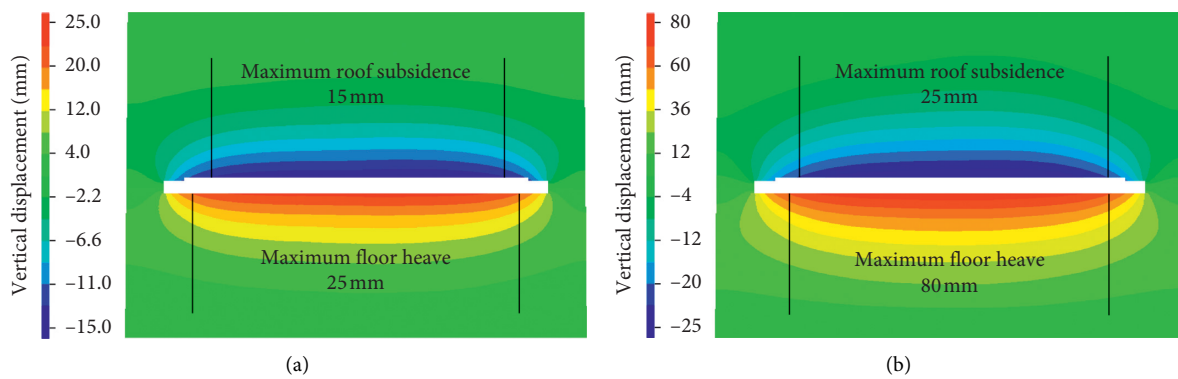

FIGURE 7: Cross section cloud map of vertical displacement profile. (a) Single hydraulic prop support. (b) Anchor cable support.
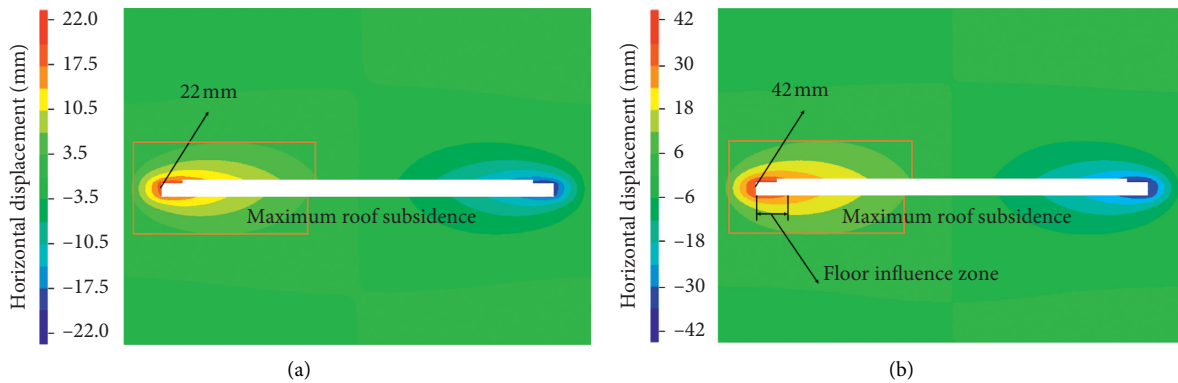

FIGURE 8: Horizontal displacement section nephogram. (a) Single hydraulic prop support. (b) Anchor cable support.

support roadway are 25 mm, 80 mm, and 42 mm, respectively; after using anchor cable support, the maximum displacements of roof, floor, and side slope are 15 mm, 25 mm, and 22 mm, respectively, which are 40%, 68.75%, and 47.6% lower than those of single support; especially, the displacement of floor heave decreased obviously. The displacement area of roof and floor with anchor cable support is smaller than that of single support. The higher horizontal displacement appears in the floor under the condition of single support, but not under the condition of anchor cable.

The schematic diagram of roadway deformation in surrounding rock is shown in Figure 9.

5. Field Monitoring Plan and Data Analysis

5.1. On-Site Monitoring Scheme. On-site monitoring is mainly divided into two parts: the roof separator is used to monitor the roof separation of material roadway and transportation roadway. A borehole peephole was arranged to monitor the development of loose circle. The specific monitoring scheme is arranged as follows:
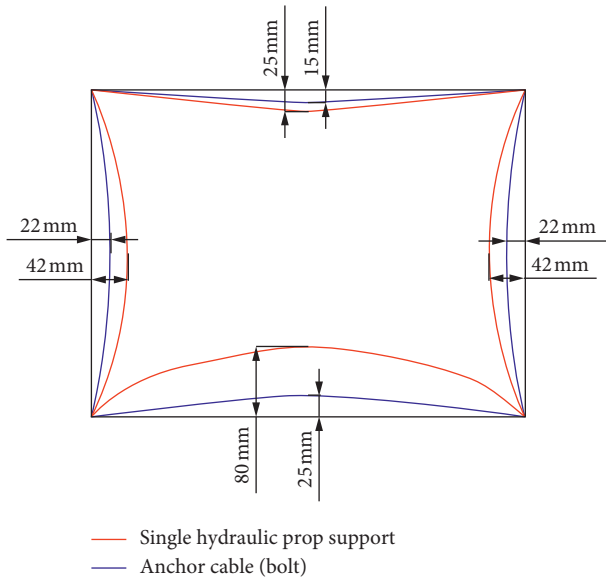
FIGURE 9: Displacement of model roadway.

(1) Layout scheme of roof separation instrument: in the material lane and transportation lane of 27304 working face, the first roof separation instrument is arranged at the top of each roadway 20 m away from the working face, and then one is arranged every 20 m, with 5 in each roadway.

(2) Layout scheme of borehole peeping instrument: due to the same geological conditions of material lane and transportation roadway in 27304 working face, combined with the site construction situation and equipment placement, it was decided to arrange the peeping hole on the roof of material lane for construction and observation. In the material lane of "3 lower" 2326 working face, the first group of peeping holes is arranged at 10 m away from the working face in the middle of the material lane after the stoppage of mining, and then a group of 5 is arranged every 10 m. Each group of peeping boreholes (on the same section) includes two boreholes, which are, respectively, arranged at the roof and the roadway side waist line on one side of the coal body, and the drilling depth is 5.5 m. The peeping boreholes are arranged at the track side of the haulage roadway of 27304 working face, 1 m away from the coal wall, and the working face is 10 m, 15 m, 20 m, 25 m, 30 m, 40 m, 50 m, 60 m, 70 m, and 80 m in advance, and the drilling depth is 9 m, so as to carry out construction and observation. The location diagram of the measuring point is shown in Figure 10.

5.2. Data Analysis of Roof Separation Instrument. The same monitoring scheme is used to monitor the adjacent roadways which are similar to the roadways in 27304 working face under the same geological conditions. Through the analysis of monitoring data: when the shallow base point is 3.0 m and 1# measuring point adopts single support, there is a total
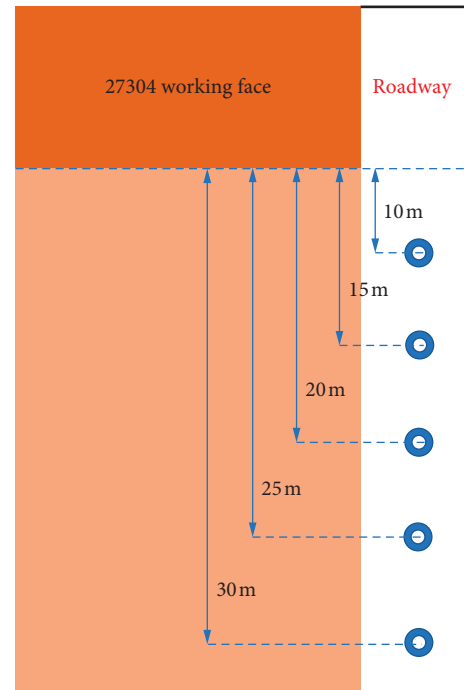

FIGURE 10: Schematic diagram of measuring point position.

displacement of 0.5 mm in 6 days before monitoring, compared with 0.5 mm displacement in 8 days before monitoring of bolt and cable support; within 20 days of monitoring, the displacement of single support roadway and anchor cable support roadway is 6.5 mm, but it takes longer time for anchor cable support to reach the same displacement. During the monitoring period of 2# measuring points, the cumulative displacement of single hydraulic prop support and anchor cable support was 1.8 mm and 0.5 mm, respectively; during the monitoring period of 3# measuring points, their displacement was basically the same; during the 28 days of monitoring, the cumulative displacement of anchor cable support roadway of 4# measuring points was only 30.7% of that of single hydraulic prop support roadway. No displacement was detected at 5# measuring point during the monitoring period. The monitoring data of shallow base points are shown in Figure 11.

In the middle base point of 5.0 m, the monomer support roadway and the anchor cable support roadway at 5# measuring point both eventually generated a 6.5 mm displacement, but the cumulative displacement curve "step" of the active support was relatively short, indicating that the anchor cable support was more sensitive to the change of surrounding rock stress and could give full play to its support ability rapidly. The cumulative displacement of 3# and 4# measuring points' anchor cable support is 1.5 mm and 1 mm less than that of single support, respectively. The decline was 22.2% and 30%, respectively. During the monitoring period, no data were detected at 2# and 5# measurement points. The monitoring data of midbase point are shown in Figure 12.

The deep base point is 9.0 m, and the change trend and value of cumulative displacement of each measuring point are basically the same during the monitoring period, and the measuring points of 3#, 4#, and 5# of monomer support and
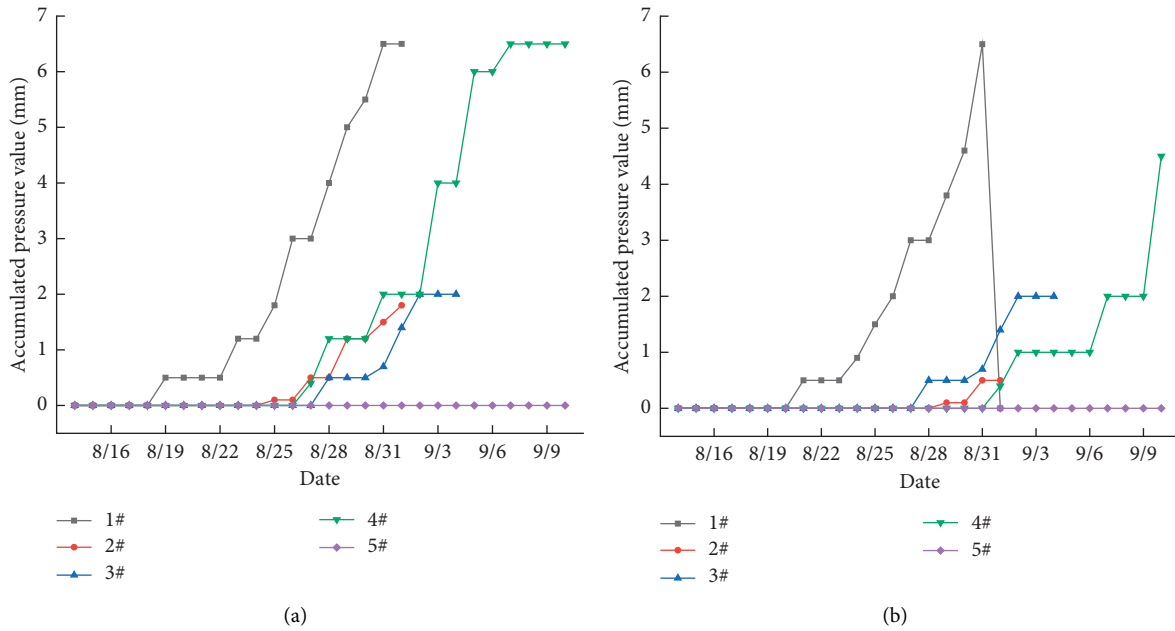
(a)

(b)

FIGURE 11: Monitoring data of shallow basis points. (a) Single hydraulic prop support. (b) Anchor cable support.
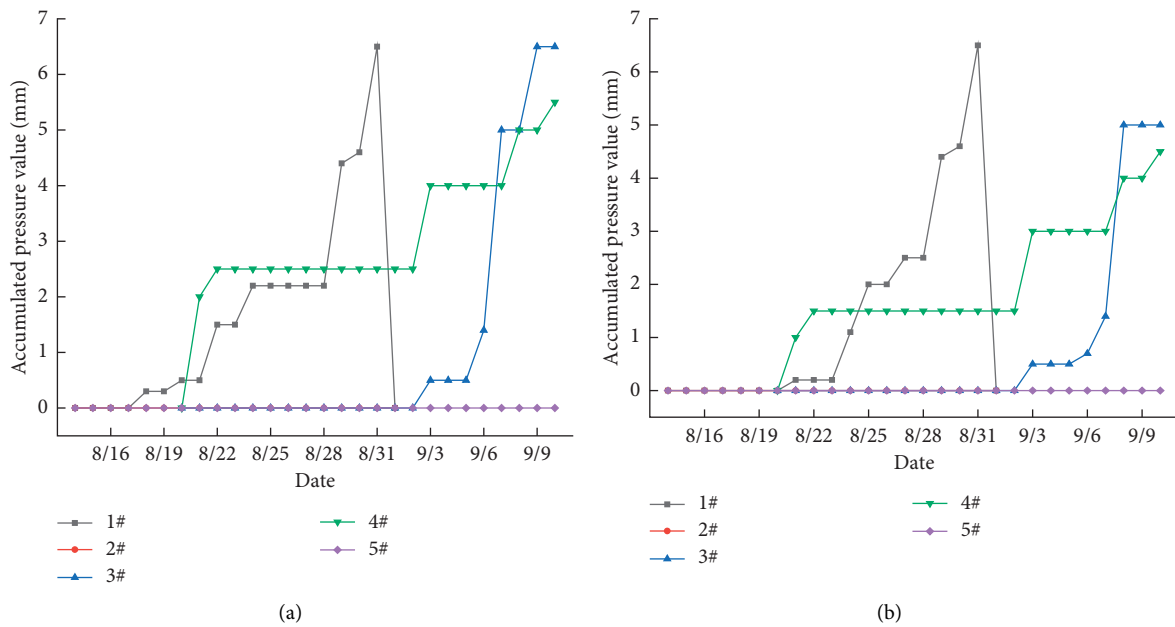

(a)

(b)

FIGURE 12: Medium base point monitoring data. (a) Single hydraulic prop support. (b) Anchor cable support.

anchor cable support are 1 mm, 1 mm, and 0.5 mm, respectively. No displacement was detected at 2# measuring point. It shows that the bolt (anchor cable) support is more effective than the single hydraulic prop support in controlling the displacement in the deep part of the roadway roof. The monitoring data of deep base point are shown in Figure 13.

Based on the comprehensive analysis of the roof separation data of each measuring point under the bolt (anchor cable) support scheme and single hydraulic prop support scheme, the shallow base point depth is 3 m, the middle base point depth is 5 m, and the deep base point depth is 9 m. The time required for anchor cable support under the same displacement is shorter,

and the cumulative displacement is also less than that of single support. Anchor cable support has the advantages of less roof separation, good integrity, and good roof support effect.

5.3. Data Analysis of Borehole Peepers. The station is 10 m away from the mining face, and the boreholes of 27304 haulage roadway are detected by RBIT-30 borehole imager. In order to fully understand the information inside the borehole, through intercepting the development area of typical fractures in the borehole rock layer, the development and development characteristics of rock fractures in
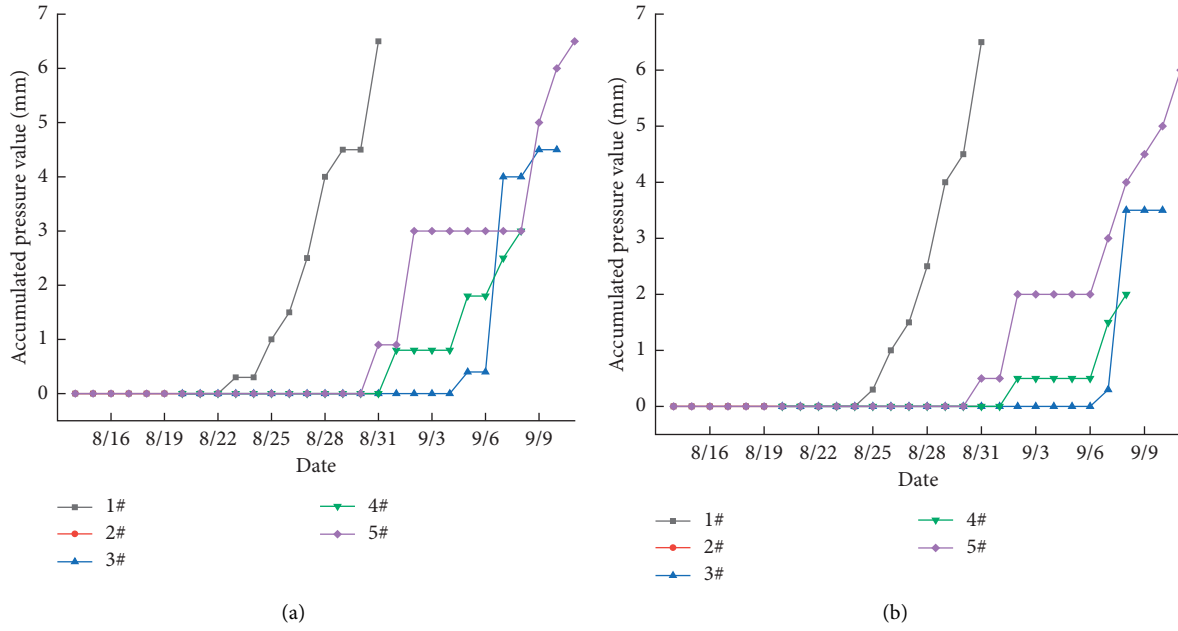
(a)

(b)

Figure 13: Monitoring data of deep base point. (a) Single hydraulic prop support. (b) Anchor cable support.

Table 3: Data of borehole peeping instrument.

	Fracture development area (m)	Separation zone (m)	Crushing zone (m)
1#	0.1–0.4 1.9–2.2	0.8–1.0	0.5–0.8 1.9–2.2
2#	0.8–1.3	0.4–0.8	0.8–1.8
3#	0.1–1.0	1.5–1.6	1.0–1.4 2.7–3.0
4#	0.1–0.5	0.7–1.0	1.5–2.2
5#	0.1–1.0	1.6–1.8	0.9–1.1 1.4–2.7
6#	0.1–0.3	0.7–1.2	0.9–1.1 1.4–2.7
7#	0.1–0.7 0.9–1.4	1.7–1.9	0.3–1.4
8#	0.3–0.7	2.7–3.0	1.6–3.0
9#	0.1–0.3	\	1.7–1.8 2.6–3.0

boreholes are summarized and analyzed. This paper summarized and analyzed the development characteristics of rock fractures in each borehole and selected some representative screenshots (as shown in Figure 12) for

explanation. According to the characteristics of roof structure and fracture development, the roof strata are divided into (1)fracture development area; (2) separation zone; and (3) crushing zone. The borehole data are shown
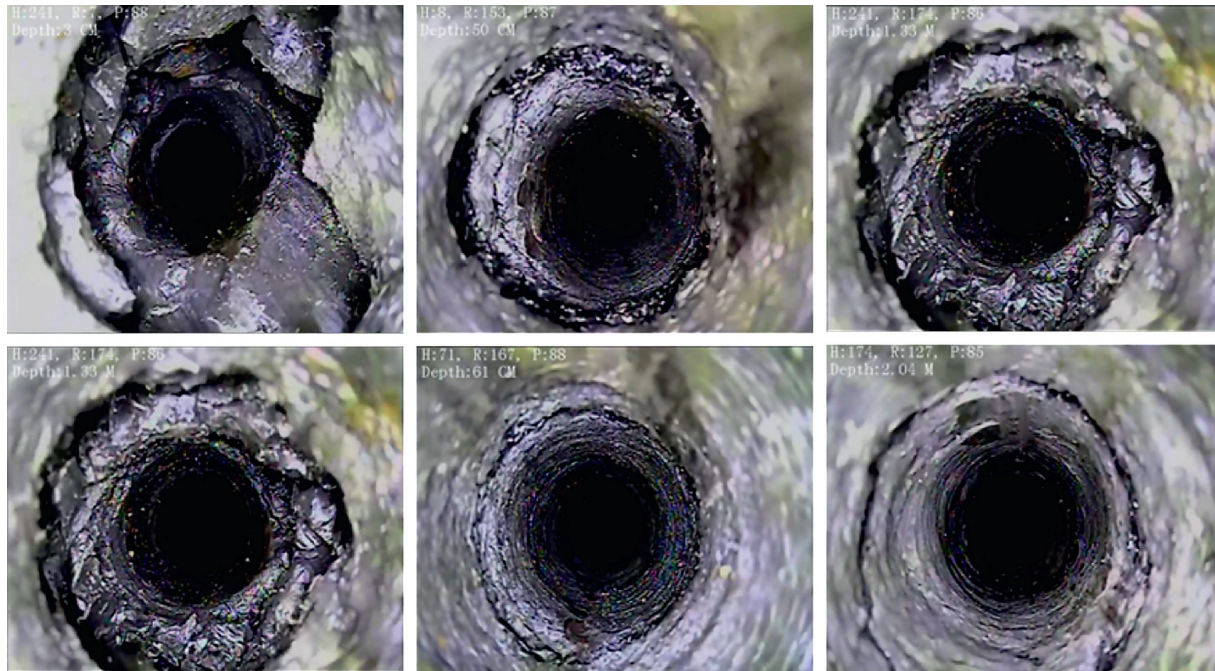
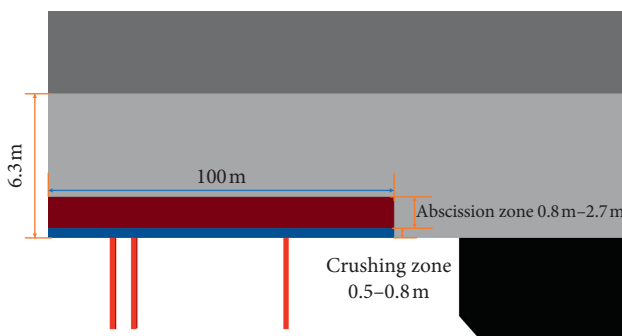FIGURE 14: Distribution law of cracks in roof strata (3# drill hole).

FIGURE 15: Distribution of roof separation zone.

in Table 3, and the distribution law of cracks in roof strata（3# drill hole）is shown in Figure 14.

Combined with the analysis of roof separation instrument and borehole imaging results, the fracture area is roughly distributed in the range of 0.5 m–0.8 m; the roof separation layer is generally distributed in the range of 0.8 m–2.7 m. Through the data analysis of roof separation instrument, the maximum separation amount is 4.5 mm, and the highest position is 2.7 m–3.0 m. The distribution of roof separation zone is shown in Figure 15.

6. Conclusion

(1) By using FLAC^{3D} numerical simulation software, the stress distribution of the roadway supported by single hydraulic prop and the roadway supported by grouting anchor cable is compared and analyzed. The results show that the maximum principal stress of single hydraulic prop support is 33.3% higher than that of anchor cable support, and the stress concentration effect of single hydraulic prop support is more obvious, and its vertical stress distribution is poor. The stability of roadway is further improved by using anchor support instead of single hydraulic prop support. The numerical simulation results of roadway displacement show that the displacement of roof, floor, and two sides of roadway is reduced by more than half. The displacement of floor heave is reduced by 68.75%, and the supporting effect is remarkable.

(2) Combined with the results of numerical simulation and the actual situation of 27304 working face in Wanglou coal mine, the specific scheme of grouting anchor cable instead of single hydraulic prop for advance support is designed, and the roof separation instrument and borehole peeper are used to test the support effect. The monitoring results show that compared with the single hydraulic prop support method, the bolt pressure is moderate in grouting anchor support, which indicates that the bolt plays a good supporting role; the displacement of each part of the roadway is small, the roof separation is small, and the integrity is good, and the fracture zone has no effect on the stability of the roadway. The numerical simulation results are in good agreement with the engineering practice. After the anchor cable is used in the advance support section of 27304 working face in Wanglou coal mine, the roadway can meet the requirements of safe and efficient mining.

Data Availability

The data used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

Acknowledgments

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