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Original scientific paper

Research on the Support Parameters Optimization of Large Section Roadway in Shallow Coal Seam

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Abstract: To explore the optimization design method of reasonable support parameters for large section roadway of shallow coal seam in Yulin area, this paper takes the support parameters of 3^{-1} coal return air roadway in North Second Panel of Hongliulin Coal Mine as the research object. The length, diameter and row spacing of anchor rods needed for support of return air roadway are calculated by using suspension theory. Combined with the calculation results, the use of the roadway and the specific engineering geological conditions in the excavation construction, the roadway size and support parameters are optimized. The numerical simulation of FLAC^{3D} computer shows that the plastic failure range and deformation displacement of surrounding rock of optimized support scheme meet the requirements of roadway use. The monitoring results of loose zone, and anchorage force in the engineering site show that the loose circle of roadway roof is 1.23 m, and the loose circle range of roadway side is 0.72 m. The anchoring force of roof bolt reaches 50 KN, that of front and auxiliary walls is 26 KN and 21.2 KN respectively, and that of anchor cable is 189 KN. The loose zone depth and stress intensity of anchor cable of the optimized support scheme all meet the use requirements of return air roadway. It improves the tunneling efficiency, shortens the construction period, reduces the support cost, and improves the economic benefit. It provides reference value for the optimization design of roadway support parameters in similar production coal mines.

Keywords: FLAC^{3D}; large section roadway; shallow coal seam; support parameters

1 INTRODUCTION

The Jurassic coalfield in northern Shaanxi is rich in coal resources, characterized by shallow burial, thin bedrock and thick loose overlying layer. The section of the preparation and mining gateway in the disc area of each production coal mine is large, and the support parameters determined by the coal design institute are generally adopted in the support scheme. In engineering practice, it is found that there are problems such as high support strength, high cost and low tunneling efficiency. Scientific research workers and field management personnel have actively explored and practiced the optimization of support parameters. Zhou Yongxing used the field measurement, numerical simulation and theoretical analysis to optimize the method [1]. Shang Zhiping took the mining gateway of 51103 working face of Guojiawan Coal Mine in Shenfu mining area as the research object, and optimized the roadway support scheme through field investigation, laboratory test, theoretical analysis, field monitoring and numerical simulation [2]. Zhang Mengliang conducted a research on the optimization of rapid tunneling of large-section coal roadway and anchor network cable support parameters, and optimized the design of W3303 working face of Gaohe coal mine [3]. Wang Shuaiming used the suspension theory to design the optimization scheme of the row spacing between anchor bolts and cables in the 9304 - machine lane of Hengsheng Coal Industry. The engineering site monitoring showed that the surrounding rock control effect is good, which realized the cost reduction and efficiency increase of the mine [4]. Du Shuai studied the influence of anchorage parameters and roof rock layer parameters on anchorage stress diffusion, and the research results provided a theoretical basis for the design of composite roof anchorage parameters [5]. The problem of roadway caused by the secondary mining, the roadway optimization support scheme of "anchor-anchor cable + network joint support" is put forward to ensure the safety of mining operation for Liuta Mine 3220401 [6]. Ma Longtao analyzed the causes of roadway surrounding rock damage in the mining face of Huangling No.2 coal mine, optimized the combined support of bolt anchor cable steel belt in the roadway, and achieved good support effect [7]. In view of the problems of large deformation and difficult control of surrounding rock in the thick coal seam, He used theoretical analysis, numerical simulation, normal matrix analysis and other methods to study the roof deformation and support parameter optimization of the roof coal roadway in the thick coal seam, and obtained the optimal support scheme [8]. In order to determine the main factors affecting the dynamic pressure deformation roadway, Ronghai designed a set of targeted support technology [9]. Reasonable supporting parameters such as bolt diameter, length, spacing between rows, number of anchor wire and spacing between the two are determined through numerical calculation [10]. Changing the length of the base-angle bolt, even if the length of the base-angle bolt is increased to a certain extent, will decrease the overall supporting effect of the supporting structure [11]. Based on the engineering geological conditions and support scheme of 3^{-1} coal return air main roadway in Beierpan District of Hongliulin Coal Mine, this paper puts forward the optimization scheme of support parameters of main roadway by means of theoretical analysis and computer numerical simulation. By monitoring the deformation of surrounding rock on the project site, the feasibility of the scheme is verified, the purpose of saving supporting materials and improving the driving speed is achieved, and the guarantee is provided for the safe and efficient mining of coal seam in the disk area.

2 ENGINEERING GEOLOGY OVERVIEW

3^{-1} Coal of Hongliulin coal mine north two disk area is a medium thick coal seam, with a recoverable thickness of 1.80 ~ 3.42 m, an average recoverable thickness of 3.03 m, the RQD (Rock Quality Designation) value is 17.2 ~ 70%, and the average uniaxial saturation compressive strength is 14.98 MPa. The top and bottom of coal seam are mainly siltstone and fine sandstone, and mudstone sand and mudstone are sporadically distributed. RQD value of roof siltstone is 30.0% ~ 92.3%, average uniaxial saturated compressive strength is 21.00 MPa, as weak rock, medium complete rock mass. RQD value of fine-grained sandstone is 15.3 ~ 92.0%, average uniaxial saturated compressive strength is 32.8 MPa, which belongs to semi-hard rock, moderate rock integrity, mudstone, soft rock, poor rock integrity with coal seam mining. The bottom plate is mainly powder and fine sand, the RQD value of siltstone rock is 23.8 ~ 86.1%, the average uniaxial saturated compressive strength is 37.90 MPa, which is semi-hard rock with moderate rock mass integrity. The RQD value of fine-sandstone is 67.3 ~ 91.7%, the average uniaxial saturated compressive strength is 26.40 MPa, which is weak rock and complete. Mudstone and sandy mudstone are prone to bottom drum phenomenon. The 3^{-1}coal return air lane in Beierpan District is arranged along the coal seam from south to north, with a length of 2527 m and a thickness of 36 ~ 75 m in the overlying strata. To ensure safe construction and prevent the weathering and crushing roof from falling, top coal with a thickness of 0.3 m is set up during excavation for undercover construction. The return air main roadway is designed with rectangular section, net width of 5800 mm, gross width of 6000 mm, net height of 3200 mm, gross height of 3500 mm, net fault area of 18.56 m^2 and excavated fault area of 21.6 m^2. The bolt row spacing is 1.0 m, the top bolt spacing is 1.10 m and the side bolt spacing is 1.20 m. There are two anchor cables in each row, and the row distance is 2.0 m. The specific support parameters are shown in Tab. 1.

Table 1 The original design support parameters of the return air lane

Lane name	The nature of surrounding rock	support pattern	Width / mm	Height / mm	Spray concrete thickness / mm	Anchor bar parameters		Anchor parameters	
						Specification and size / mm	Interrow spacing / mm	Specification and size / mm	Interrow spacing / mm
Back to the wind lane	Rock / coal	Anchor network cable spray	6000	3500	100	ø 20 × 2600	1100 × 1000	ø 17.8 × 7300	200 × 2000

3 DESIGN OF BOLT SUPPORT PARAMETERS BASED ON SUSPENSION THEORY

According to the regional geological data and drilling data analysis results, the 3^{-1} coal plate lane is a typical shallow buried thin bedrock coal seam roadway, which can be used according to the suspension theory [12-17]. Design of North 2 - disk zone 3^{-1} Support scheme of the return air lane in the coal plate area.

(1) The rod length.

The bolt length is usually calculated according to Eq. (1):

$$L = L_1 + L_2 + L_3 \tag{1}$$

In formula: L_1 - bolt exposed length, generally L_1 = 0.10 ~ 0.40 m. L_2 - Effective length, m. L_3 - Anchor rod anchor section length, general end anchor time L_3 = 0.3 ~ 0.4 m.

The L_2 value is equal to the value of the Platts pressure arch height, which can be calculated according to the Platts theory. When the rock Platts ruggedness coefficient $f > 3$, L_2 can be calculated according to Eq. (2).

$$L_2 = \frac{B}{2f} \tag{2}$$

When the rock firmness coefficient $f \le 3$, L_2 can be calculated according to Eq. (3):

$$L_2 = \frac{1}{f}\left[\frac{B}{2} + H\,cot\left(45° + \frac{\varphi}{2}\right)\right] \tag{3}$$

where: B - for the roadway width, m. H - roadway height, m. φ - the internal friction angle of the roof strata, °.

According to the occurrence conditions of the roof, the average uniaxial saturation compression strength of the roof siltstone is 21.00 MPa, the internal friction angle is 37°. The average of the fine sandstone is 32.8 MPa, and the internal friction angle is 38°, L_2 calculated according to the stable roof fine grain sandstone and siltstone calculation. L_2 = 2.25 m. L = 0.1 + 2.25 + 0.3 = 2.65 m. When the roof of the return air lane in the disc area is siltstone, the rock stability coefficient f = 2.1 < 3, in the return air lane. When the roof of the return air lane in the disc area is fine-grained sandstone, the rock stability coefficient is f = 3.28 > 3, then. L_2 = 0.915 m. L = 0.1 + 0.915 + 0.3 = 1.315 m. As can be seen from the above calculation results, the lithology of the headair lane in the disc area has a great influence on the length of the bolt. When the platan coefficient is greater than 3, there is a large optimization space for the length of the bolt, and when the anchor coefficient is less than 3, the optimization space is not large. The specific construction should always pay attention to the roof lithology of the large lane, and the length of the support bolt should be adjusted according to the mechanical properties of the roof rock layer.

(2) The diameter of the bolt rod body.

The anchorage force of the bolt shall not be less than the weight of the suspended unstable rock layer, as calculated by Eq. (4).

$$Q = KL_2 a_1 a_2 \gamma \tag{4}$$

where: Q - anchor bolt anchor force, MN; K - safety factor, generally take 1.5 ~ 2; a_1a_2, - The row spacing between anchor bolts, m; γ - Average gravitational density of unstable rock strata, M N/m^3.

If the anchorage force of the bolt is equal to the breaking force of the rod, the diameter of the bolt can be calculated by Eq. (5).

$$d = \sqrt{\frac{4Q}{\pi \sigma_t}} \tag{5}$$

where: d - bolt rod body diameter, mm. Q - Anchor force can be determined by the pull test, the design is 100 kN. σ_t - Tensile strength of rod material. If the bolt material is selected as equal strength left rebar without longitudinal bars, the yield strength of rod material is 340 MPa and the tensile strength is 490 MPa.

According to the calculation of Eq. (5), the rebar bolt d = 16.05 mm, it can be seen that the bolt diameter has a certain optimization space.

(3) Anchor ages and row spacing.

According to the weight of the rock suspended by each bolt, the weight of the rock suspended by the bolt is equal to the anchorage force of the bolt. Usually, the anchors are arranged by isometric arrangement, that is, spacing and equal spacing, set to a, then there is Eq. (6): $a = a_1 = a_2$.

$$a = \sqrt{\frac{Q}{K\gamma L_2}} \tag{6}$$

where: K - bolt safety factor, here take 1.8. γ - The volume force of the rock, take 25 kN/m³.

L_2 = 2.25 m, a = 0.99 m. When the top of the return air lane is siltstone. Then the return air lane. When the roof is fine grain sandstone: L_2 = 0.915 m, then the return air lane a = 1.56 m. The above calculation analysis shows that the lithology of the roof has a great impact on the row spacing of the bolt. When the roof surrounding rock condition is not good, the row spacing should be properly reduced, and the row spacing should be appropriately increased when the roof condition is good.

4 OPTIMIZED DESIGN OF SUPPORTING SCHEME

The theoretical calculation result is Hongliulin 3^{-1} Coal Mine. The optimization scheme of bolt support design of return air lane in coal plate area provides the basis. According to the actual engineering in the process of coal mine excavation, the engineering analogy analysis and use of the roadway, the specific requirements of construction, the requirements of design specifications and regulations, and the ventilation system check, the excavation size and support parameters of the return air roadway are adjusted, and the north second plate area of Hongliulin 3^{-1} coal mine is finally determined. The optimized support design scheme of the return air roadway in the coal plate area is as follows. The design section size is adjusted from 6000 × 3500 mm to 5600 × 3100 mm, the diameter and length of the roof bolt are designed to be ø 20 × 2600 mm, the spacing between rows is 1000 × 1000 mm, the bolt preload is 100 KN, and the nut torque is not less than 150 N·m. The diameter and length of the side bolt are designed to be ø 18 × 1600 mm, the spacing between the rows is 1500 × 1000 mm, the bolt preload is 70 KN, and the nut torque is not less than 100 N·m. The support plate is made of Q235 steel plate, the thickness is 8 mm, and the specification is 150 × 150 mm. The roof and side support mesh of the roadway are made of φ = 6.0 mm reinforcement welding, the top mesh specifications are 3200 × 1200 mm, the side mesh specifications are 2000 × 1200 mm, the mesh specifications are 80 × 80 mm, and the mesh is overlapping 160 mm. Double rows of three flowers were tied manually with 14# wire, and the binding spacing was 240 mm. The cable length is 7.3 m, the cable is a steel strand of φ 17.8mm, the row distance between the cables is 2000 × 2000 mm, the anchoring force of the cable is not less than 200 KN, the rectangular layout, the pallet is made of Q235 steel plate, the thickness is 12 mm, the specification is 300 × 300 mm. The strength grade of bedding concrete is C30, and the strength grade of shotcrete is C20. The layout is shown in Fig. 1.

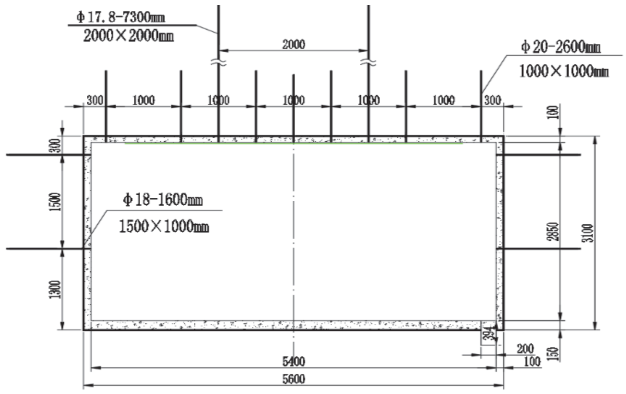


Figure 1 Supporting scheme of the return air lane

5 NUMERICAL SIMULATION AND ANALYSIS

To verify the rationality of the support optimization protocol, the computer numerical simulation software FLAC^{3D} was used to analyze the support effect, the plastic failure characteristics and the displacement deformation law. The FLAC^{3D} numerical calculation model is mainly constructed based on borehole 4-HB3. Combined with relevant geological data, the mechanical parameters [18] of surrounding rock are selected as shown in Tab. 2.

Table 2 Coal crag mechanics parameter

Lithology	Bulk modulus / GPa	Shear modulus / GPa	Cohesive force / MPa	Internal friction angle / °	tensile strength / MPa	Density / kg/m³
3^{-1}coal	1.3	1.1	1.4	30	1.9	1350
Siltstone	1.8	1.3	1.6	35	2.1	2360
Fine-grained sandstone	1.6	1.2	1.8	36	2.2	2530
Medium-grained sandstone	2.52	1.66	2.2	37	2.3	2220
Mudstone	1.33	1.13	1.35	34	1.8	2550

(1) Characteristics of plastic destruction.

After the excavation of the return air roadway, the support design scheme is adopted for support. After 50 steps of calculation, the plastic damage characteristics of the rock around the return air and the force state of the anchor rod and anchor cable are shown in Fig. 2.

As can be seen from Fig. 2, after the excavation of the return air roadway, the support design scheme is adopted, and the plastic yield damage occurs in the surrounding rocks around the roadway. The top of the roadway top model is 300 mm top coal, and the height of plastic damage is about 0.3 m. The top coal has basically suffered plastic damage. At the center of the roadway top, it is mainly pull failure (tension) and shear failure form (shear). The upper part of the two gangs of the return air lane is coal, the maximum depth of plastic damage is 1.6 m, the lower part of the two gangs of the return air lane is mudstone, and the maximum depth of plastic damage is 0.4 m.

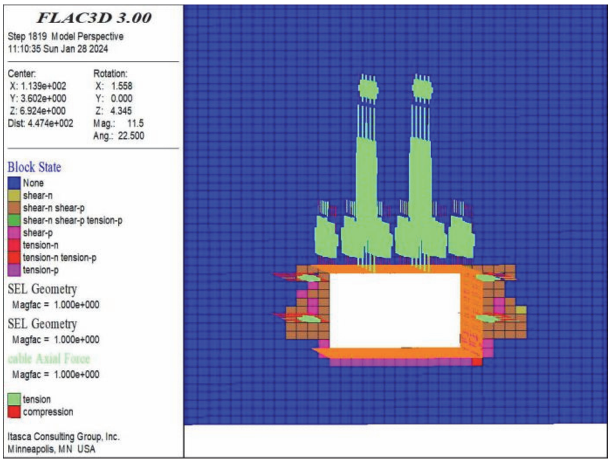


Figure 2 Plastic failure characteristics and stress status of the return air lane

The above plastic failure forms are shear failure and tensile failure. The bottom plate is medium grain sandstone, with the maximum plastic damage depth of 0.3 m, which is mainly manifested as pull damage. The design support scheme is adopted. In the support system, all the anchors are stretched and compressed in different degrees of tension and compression in the surrounding rock, indicating that they play a supporting role. The stress state of the anchor bolt and the anchor cable is different. The anchor rod and the anchor cable in the roadway roof play the suspension role, and the tensile stress of the suspension is mainly borne by the anchor cable. It can be read from the figure that the maximum tensile force on the free section of the bolt is 1005 MPa. It can be seen that the bolt has a strong obstruction effect in preventing further damage and deformation of the surrounding rock, and plays a supporting role. The bolt support limits the plastic failure of the roof and the two sides. The effect of the optimized support scheme is due to the stress state of surrounding rock changed by the bolt, cable, anchor net and concrete support. In this way, the development of plastic failure and deformation of surrounding rock is controlled, and the safety of working space of roadway is ensured.

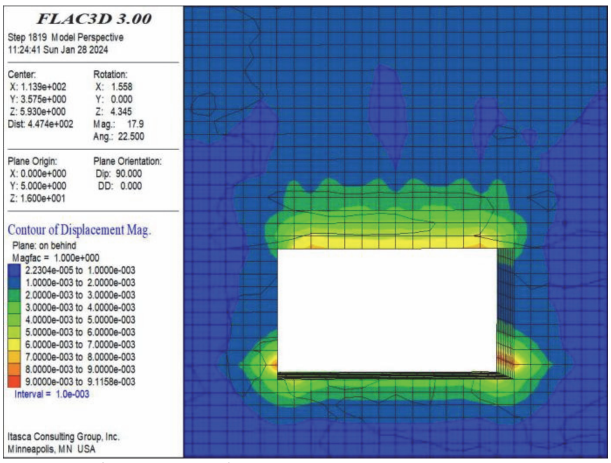


Figure 3 Cloud diagram of the displacement contour in the return air lane

(2) The displacement and deformation characteristics.

In this paper, support design scheme is adopted for support. After 50 - time step calculation, the displacement cloud diagram of roadway surrounding rock is shown in Fig. 3. It can be seen that the position of deformation and displacement of auxiliary transportation main roadway is the roof and two sides of the roadway. It can be seen from the figure that the maximum displacement of the roadway mainly occurs in the roof of the roadway, and the maximum value is 0.0091158 m. The roadway hardly deforms under the optimized support scheme.

6 INDUSTRIAL FIELD MONITORING

In order to test the implementation effect of the optimized support design scheme, this paper systematically monitors the deformation and failure state of the roadway surrounding rock, the stress distribution and size of the anchor rod (cable). By obtaining the stress information of the supporting body and deformation characteristics of the surrounding rock, the rationality and reliability of the support scheme design are further verified, and the stability and safety of the surrounding rock of the roadway are judged.

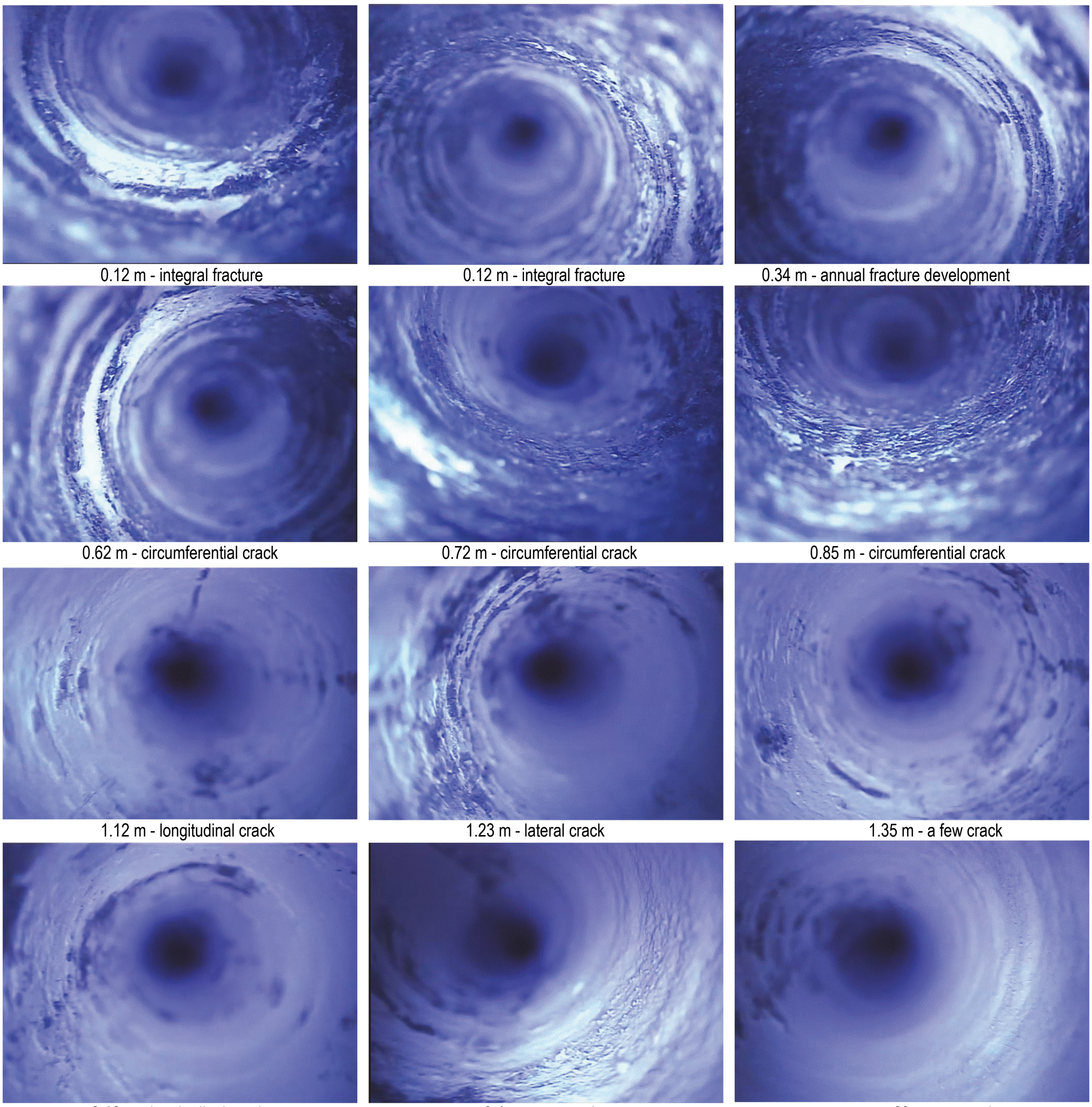
(1) OFF-layer monitoring.

After the excavation of the main roadway, the surrounding rock deformation occurs and the roof subsidence occurs. The displacement of the roof at different depths is not the same. Generally, the displacement of the shallow surrounding rock is large, and the displacement of the deep rock is small, resulting in the displacement difference between the shallow rock and the deep rock, which is called the roof separation. The LBY-4 multi-point separation indicator of soil type roof produced by Shandong Weishan Longong Machinery Co., Ltd. was used for monitoring. In order to accurately determine the deformation and failure range of the main roof and two key areas, a reasonable improvement has been made to the length of the measuring line of the earth-shaped roof separation instrument. Roof measuring line length selection: 1.8 m, 2.4 m, 3.0 m, 7.0 m. Two sides measuring line length selection: 1.6 m, 2.2 m, 2.6 m, 5.0 m. When the excavation of the working face reached about 930 m, the measurement lines of the roof and the two sides remained unchanged, and the values were displayed at the "0" scale position. The roof and the two sides did not appear to be separated during the excavation of the roadway. In addition, the second team of integrated excavation in Hongliulin Coal Mine installed a roof electronic separation instrument at the roof integrity of the opening position of the return air main roadway in the 3^{-1} coal pan area. Under normal circumstances, one roof electronic separation meter is installed for every 100 m tunneling, and one roof electronic separation meter is installed at the complete roof of the opening position of the turning tunnel (the center of the intersection of the two tunnels). The deep base point of the roof separation instrument is 7.3 m, the shallow base point is 2.6 m, and the initial reading is 0 when installed. The newly installed roof separation meter is measured and recorded once a day within 50 m of the working face, and the one 50 m away is observed once a week. The inspection personnel and technicians are responsible for daily reading work, and the separation recording ledger is established. During the execution of the project, the mine roof separation monitoring results are also 0 mm.

(2) Surrounding rock loose circle (surrounding rock damage range) drilling peep.

The main return air roadway in the 3^{-1} coal pan area is driven to 1050 m. In order to monitor the development of surrounding rock loose circle after the surrounding rock looseness and deformation after the opening of the main roadway, the drilling and observation position were originally designed to be carried out at 982 m, and the roof drilling was arranged in the center of the roadway with a depth consistent with the anchoring depth of the anchor cable - 7300 mm and the diameter of the drilling hole 42 mm. Due to the limitation of the construction technical conditions of the pneumatic drilling rig, the borehole perpendicular to the roadway side cannot be constructed. During the construction process, the Angle between the borehole and the horizontal plane is 61° (as shown by the drilling scope), the drilling depth is 5 m, and the drilling diameter is 42 mm. The peep results of the roof loose ring are shown in Fig. 4, and the peep results of the side loose ring are shown in Fig. 5.

Through the observation of the roof loosening ring telescope, it can be seen directly that the lateral cracks of surrounding rock are relatively developed and the degree of breakage is relatively high in the range of 0.12 m - 0.62 m. There are a few transverse and longitudinal cracks in 0.72 m - 1.23 m, and the degree of development is low. Within 1.23 m - 7 m, the rock remains intact without obvious cracks, and the loosening circle of the roadway roof is 1.23 m. Through the observation of the side loosening ring telescope, it can be intuitively seen that the lateral crack of surrounding rock is relatively developed in the range of 0 - 0.48 m. There are a few transverse and longitudinal cracks in 0.48 m - 0.72 m, and the degree of development is low. Within 0.72 m - 2.32 m, the rock remains intact without obvious cracks, and the loosening circle of the roadway side is 0.72 m.



0.12 m - integral fracture | 0.12 m - integral fracture | 0.34 m - annual fracture development

0.62 m - circumferential crack | 0.72 m - circumferential crack | 0.85 m - circumferential crack

1.12 m - longitudinal crack | 1.23 m - lateral crack | 1.35 m - a few crack

2.16 m - longitudinal crack | 3.45 m - no crack | 5.63 m - no crack

Figure 4 Roof loose ring peep

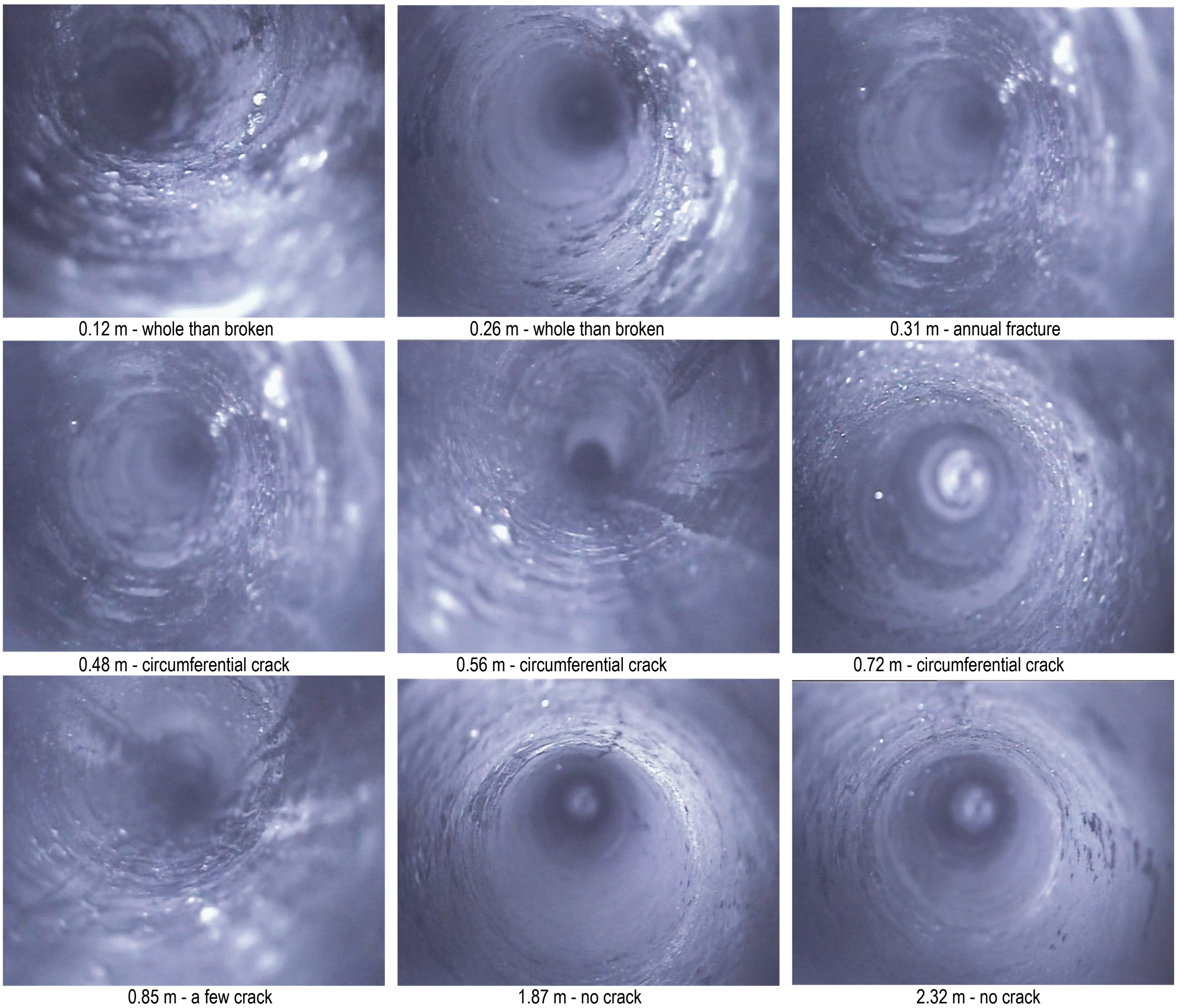


Figure 5 Left side loose ring peep view

(3) Bolt and cable tension stress monitoring.

The stress monitoring of bolt and cable is an important part of mine pressure monitoring of roadway. By monitoring the force size and distribution of the supporting body, the working condition of the anchor rod and cable can be comprehensively understood, and whether the anchor rod has yielded or broken can be judged, and the stability and safety of the surrounding rock of the roadway can be evaluated. Whether the bolting design is reasonable or not, reasonable suggestions for modifying the bolting design scheme are put forward according to the monitoring data. The hydraulic pillow is used to monitor the axial force of the end anchor rod. In the application, the pressure box is first set between the bolt tray and the nut of the outer anchor head, and then the nut is tightened to apply prestress to the bolt, record the initial value indicated by the pressure gauge, and then measure the change of the bolt pressure and time regularly. The anchor rod adopts MC200 dynamometer with a maximum measuring range of 200 KN and the anchor cable adopts MC300 dynamometer with a maximum measuring range of 300 KN. The position of the hydraulic pillow and the separation indicator are the same, and the anchor rod (cable) stress measuring station is arranged closely with the driving head after 820 m is driven into the return air main roadway in the disk area. One roof anchor cable is arranged in each station, one roof anchor is arranged, and one dynamometer (hydraulic pillow) is arranged on each side of the roadway. The monitoring results of bolt stress are shown in Fig. 6, and the monitoring results of cable stress are shown in Fig. 7.

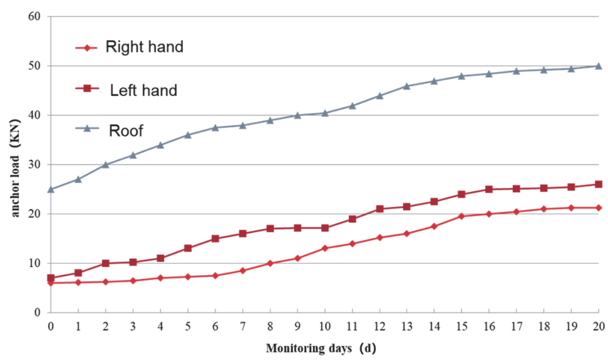


Figure 6 Bolt stress monitoring

It can be seen from Fig. 6 and 7 that the initial anchoring force of the two sides of bolt is 6 KN and 7 KN, the roof bolt is 25 KN, and the cable is 150 KN. With time, the anchorage force of bolt and cable gradually increased, and basically stabilized at 16 days. The final observed anchoring force of the roof bolt reaches 50 KN, the front side and the auxiliary side are 26 KN and 21.2 KN respectively, and the final observed anchoring force of the anchor cable is 189 KN. The anchorage stress received by the bolt and cable has achieved the expected supporting effect.

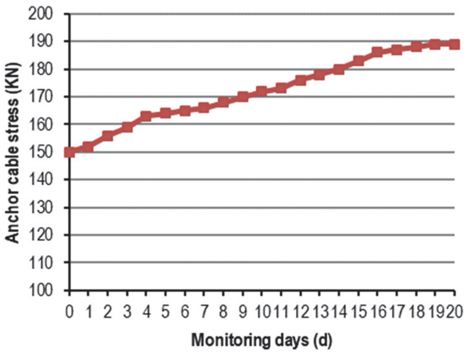


Figure 7 Anchor cable stress monitoring

7 DISCUSSION

(1) Based on engineering analogy and support experience, Coal mine roadway support design is monitored and revised in specific field application to determine the final support scheme.

(2) The preliminary design of the support scheme is carried out through the suspension theory. The numerical simulation software is further applied to analyze the effect of the support scheme. The rationality of the monitoring scheme in engineering can provide a formulation design idea of the coal mine support, which has certain reference value for optimizing support parameters of similar production coal mines.

8 CONCLUSION

(1) According to the engineering geological conditions of Beierpan Area of Hongliulin Coal Mine and the physical and mechanical properties of 3^{-1} coal and the surrounding rock of the roof and floor, this paper calculates the length, diameter and spacing of the bolt required by the large-section roadway support by using the suspension theory. Compared with the original design, the length, diameter and interrow distance of the anchor rod in the main return air roadway in the pan area have some optimization space, but should be adjusted reasonably according to the roof lithology.

(2) The size and support parameters of the roadway are optimized according to the theoretical calculation results, the use of the roadway and the specific engineering geological conditions in the excavation construction. The numerical simulation results of the FLAC3D computer show that under the conditions of the support scheme, the plastic failure range of the top and bottom and the two sides is small, and the anchor rod and cable are in a better stress state with little deformation and displacement and can ensure the safe use of roadway requirements.

(3) The values of soil type separation meter and electronic separation layer are 0, and no roof deformation occurs in the monitoring stage of the return air main roadway. The loose ring depth of the roof of the return air main roadway is 1.23 m, and the loose ring depth of the side is 0.72 m, which do not exceed the length of the anchor rod. The final observed anchoring force of the roof bolt is 50 KN, the front side and the side side are 26 KN and 21.2 KN respectively, and the final observed anchoring force of the anchor cable is 189 KN. The roof anchor cable and the roof and two side bolts can play the designed support strength in the support system, meeting the design and use requirements.

(4) By optimizing the support parameters on the length, diameter and interrow distance of the anchor rod, and the section size of the return air main roadway in the panel area, under the premise of ensuring safety, the tunneling efficiency of the roadway is improved, the supporting cost is reduced, the economic benefits are improved, and the supporting scheme is optimized and reasonable to meet the requirements of the roadway.

Acknowledgments

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