

Study on Reinforcing Anchor Cable Support in Overreach Section of Deep Roadway

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Abstract: With the increasing depth of coal mining, the conditions of deep working face are becoming more and more complex, the stability of surrounding rock of roadway is getting worse and worse, and the problem of deep roadway support is becoming more and more prominent. In this paper, the 24100 working face of Pingdingshan No.11 Coal Mine is taken as the engineering background. In view of the problems of roof crushing and deformation and more net pockets in the return airway affected by the mining of the working face, the grouting anchor cable reinforcement and support technology in the advance section of the deep mining roadway is proposed, and the feasibility of the support effect is verified by numerical simulation. The results show that the influence range of the advance abutment pressure of the air roadway is about 60 m, and the reinforcement support strength required for the roof of the roadway is 50.6 kN/m². The grouting anchor cable reinforcement support technology system of the return air roadway can control the deformation of the surrounding rock of the roadway and achieve safe and efficient mining.

Keywords: Advanced Support Technology; Grouting Anchor Cable; Active Support; Deep Roadway; Numerical Simulation.

1. Introduction

At present, the reserves of coal seams with 'superior' occurrence conditions are becoming less and less. In order to meet the needs of energy and achieve the goal of sustainable development, it is necessary to mine deep coal seams with complex burial conditions. The proportion of deep mines is increasing year by year^[1]. Due to the special environmental conditions of deep mine, such as high ground stress, high temperature, rich groundwater, etc, the soft rock roof expands and cracks develop after roadway excavation, and the stability of roadway surrounding rock itself is poor^[2]. For the deep mining roadway, on the one hand, under the complex stress environment, the roadway is greatly affected by the mining disturbance of the working face, the mine pressure appears frequently, the cracks in the surrounding rock of the roadway are further expanded and developed, and the plastic zone of the surrounding rock continues to increase. On the other hand, the occurrence of coal seams is mostly soft rock inclined coal seams. At the same time, affected by the inclination of inclined coal seams and the structure of coal and rock mass, the surrounding rock of roadway is prone to stress concentration, which leads to frequent asymmetric deformation and failure of roadway roof, floor and two sides. Many field practices show that ^{[3]-[4]}, Fully grasp the distribution of advanced abutment pressure and the deformation law of roadway surrounding rock under the influence of mining, and put forward a targeted technical scheme for strengthening support and control of surrounding rock, so as to avoid the situation of "excavation before repair" in the roadway, improve the stress state of surrounding rock, and realize the safe and efficient mining of the mine.

According to the existing research and field practice, it shows that^{[5]-[6]}, The bolt-grouting support technology formed by grouting anchor cable and bolt can play a good supporting effect on the advance support of mining roadway. At present, this method has made some technical progress, but it still needs to be further studied and solved in terms of selecting

suitable slurry materials, determining the best grouting parameters, layout and design of grouting anchor cables.

2. Description of the Mine Study Site

The No.11 mine is located at the foot of Xiangshan in Pingdingshan City, Henan Province. The east-west direction of the mine is 6.6 km long, the north-south inclination width is 2.54 km, and the mine field area is 13.6252 km². The VII of the 16-17-24100 working face is -784.2 m ~ -872.1 m, the corresponding ground elevation is +130.00 m ~ +147.00 m, and the average dip angle of the coal seam is about 13.0°. The 16-17 coal is black massive and granular, mainly composed of bright coal, with a thickness of 2.0 m ~ 7.0 m, an average thickness of 5.3 m, and a thin layer of gangue with a gap of 0.06 m. The immediate roof is about 1.0 m ~ 3.5 m mudstone and sandy mudstone, the basic roof is about 7.0 m ~ 13.0 m sandstone and sandy mudstone, the direct bottom is 2 m ~ 5 m thick gray black massive mudstone, and the basic bottom is 8 m ~ 15 m thick sandstone and limestone.

Based on the geological conditions of the coal seam where the 24100 working face is located, the basic quality index of the rock mass and the mechanical parameters of the coal and rock, the geomechanical evaluation of the surrounding rock of the 24100 roadway is carried out. The factors that have a great influence on the deformation of the surrounding rock of the roof in the advanced section of the 24100 working face mainly include the soft rock of the roof and floor, the development of the roof fissure, the water filling of the roof sandstone fissure, the influence of the mining and the stress of the deep high surrounding rock. The mine pressure of the roadway in the advanced influence area of the working face is severe, the roof crushing phenomenon is serious, the displacement of the mining side invading the roadway is large, and there is a great potential safety hazard. It is necessary to further strengthen the control of the surrounding rock of the roadway in the advanced section.

3. Grouting Anchor Cable Reinforcement Support Technology

3.1. Mechanical analysis of surrounding rock of roadway

The application of grouting anchor cable support technology in roadway can significantly improve the overall mechanical properties of surrounding rock, realize the consolidation effect by filling the fracture of surrounding rock with slurry, effectively control the development of surrounding rock loose circle, prevent the weathering of rock stratum and transform passive support into active support. The radius of the plastic zone of the roof of the roadway can determine the failure depth of the surrounding rock of the roof and the support range of the anchor cable. The anchor cable connects the soft and unstable rock mass of the lower layer with the complete rock mass of the upper layer to form a unified anchor solid, which controls the deformation of the surrounding rock of the roadway and prevents the plastic zone of the surrounding rock from continuing to expand. The surrounding rock structure of the roof of the inclined roadway is shown in Figure 1.

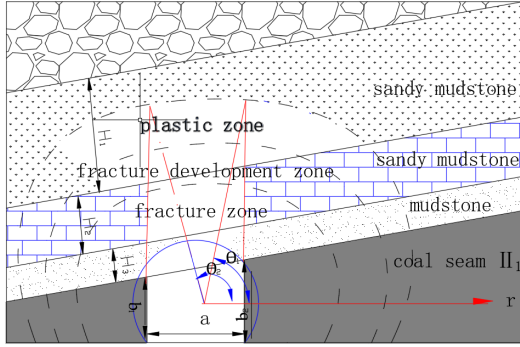


Figure 1. Mechanical structure of roadway roof surrounding rock.

Based on the suspension theory, the supporting strength that the anchor cable needs to provide is :

$$P_T = \frac{\gamma}{a} \left[\int_{\theta_1}^{\theta_2} \frac{R_1^2}{2} d\theta + \frac{aR_1 \sin\theta_1}{4} \left(\frac{R_1 \sin\theta_1 - b_2}{R_1 \sin\theta_1} \right)^2 + \frac{aR_2 \sin\theta_2}{4} \left(\frac{R_2 \sin\theta_2 - b_2}{R_2 \sin\theta_2} \right)^2 - \int_{\frac{a}{2}}^{\frac{a}{2}} \left(\frac{b_1 - b_2}{a} x + \frac{b_2}{2} \right) dx \right] \quad (1)$$

$$\theta_1 = \arccos \left(\frac{a}{R_1} \right) \quad (2)$$

$$\theta_2 = \arccos \left(\frac{a}{R_2} \right) \quad (3)$$

In the above formula: R_1 and R_2 are the radius of plastic zone corresponding to θ_1 and θ_2 . The calculation results are 6.05 m and 6.15 m; The height of low side and high side of roadway b_1 and b_2 is 2.9 m and 4.1 m respectively. a is 5.2 m. It is calculated that the support strength required for the static load state of the roof per unit area is 125.67 kN/m².

The supporting strength required under the influence of

mining is:

$$p_c = K p_T k_1 \quad (4)$$

In the above formula, K is the stress concentration factor, taking 1.92 ; k_1 is the safety factor, take 1.2. It is calculated that the support strength required for the roof per unit area under the influence of dynamic pressure is 289.5 kN / m².

According to the suspension theory, the supporting strength of the original supporting bolt roadway is:

$$P_t = \frac{F_1 n_1 \omega_1}{a L_1} + \frac{F_2 n_2 \omega_2}{a L_2} \quad (5)$$

In the above formula: P_t is the original support anchor cable roadway support strength, kN/m²; F is the pre-tightening force of anchor rod and anchor cable, which is 150 kN and 200 kN respectively. n is the number of anchor bolts and anchor cables, taking 7 and 3; L_1 and L_2 are the spacing of anchor bolts and the row spacing of anchor cables, taking 0.78 m and 0.75 m respectively, and the minimum values of anchor bolts and anchor cables are taken from the original support design. ω_1 and ω_2 are the support efficiency of the anchor rod, respectively, 0.7 and 0.75 according to the actual situation and related literature.

It is calculated that the supporting strength of the roadway roof per unit area of the original supporting bolt is 238.9kN/m². The required strengthening support strength of the roof advance section of the return air roadway in the 24100 working face is not less than 50.6kN/m².

In order to further study the advanced stress distribution and surrounding rock deformation law of the return air roadway under the original support, taking the return air roadway of 24100 working face as the research object, the FLAC 3D model software is used to simulate the original support method, which provides a reliable basis for the reinforcement and support technology of grouting anchor cable in the return air roadway.

3.2. The distribution law of advance abutment pressure

3.2.1. Modeling

Taking the return air roadway of 24100 working face in No.11 Coal Mine as the engineering background, FLAC3 D software was used for research. The dip angle of the coal seam is 13 °, the average thickness of the coal seam is 5.3m, and the length of the 24100 working face is 297.8m (the model is 300m). X×Y×Z=400 × 400 × 130. The FLAC3D numerical model is established, as shown in Figure 2.

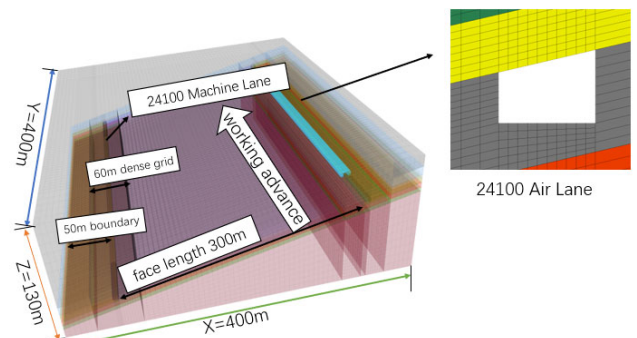


Figure 2. Numerical calculation model diagram.

The Mohr-Coulomb constitutive model is selected for the simulation study. The buried depth is 1000 m. In order to simulate the self-weight of the overlying strata, a vertical stress of 25 MPa is applied at the top of the model. The pressure coefficient is 1.1. The z direction of the top of the

model is set as the free surface, and the other surfaces are fixed. The physical constraints of the model are shown in Figure 3. The mechanical parameters of coal rock are obtained according to the field drilling core experiment, as shown in Table 1.

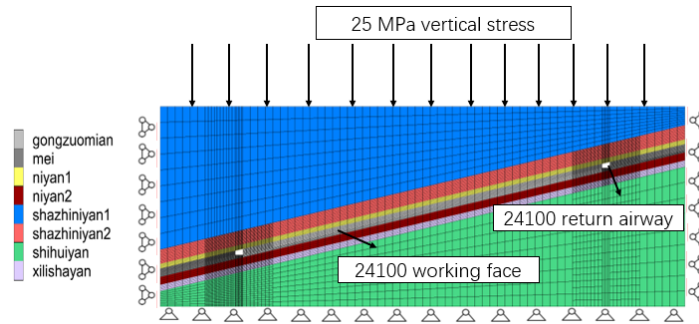


Figure 3. Physical constraint diagram of the model.

Table 1. Coal rock mechanics parameter table.

lithologic characters	bulk modulus /GPa	shear modulus /GPa	force of cohesion /MPa	angle of internal friction /(°)	tensile strength /MPa	density /(kg/m ³)
Sandy mudstone 1	3.2	2.4	2.9	37	2.3	2610
Sandy mudstone 2	3.6	2.6	2.6	39	2.2	2620
Mudstone 1	2.5	1.3	2.1	34	1.6	2640
coal	1.6	1.2	0.9	28	0.9	1638
Mudstone 2	3.2	1.5	1.9	31	1.4	2634
post office box stone	4.9	2.4	2.2	36	2.2	2662
limestone	4.5	4.2	3.1	38	2.4	2860

3.2.2. The influence range of advance abutment pressure of working face

Based on the geological conditions of 24100 working face, FLAC 3D software is used to simulate the mine pressure behavior of working face in the process of coal seam mining. The distribution characteristics of advance abutment pressure in roadway are analyzed when the first weighting, periodic weighting and working face of 24100 working face are squared for one time. According to the actual record: the first weighting step of the working face is 28 ~ 30m (30m in the

simulation calculation), and the periodic weighting step of the working face is 12 ~ 18m (15m in the simulation calculation). The inclined length of the working face is 300 m, and the working face is 300 m at a time. The change of advance abutment pressure of roadway is analyzed and recorded when the working face advances at different distances. The cloud diagram of vertical stress change at different distances of working face mining is shown in Figure. 4, and the change curve of advance abutment pressure at different distances of working face mining is shown in Figure. 5.

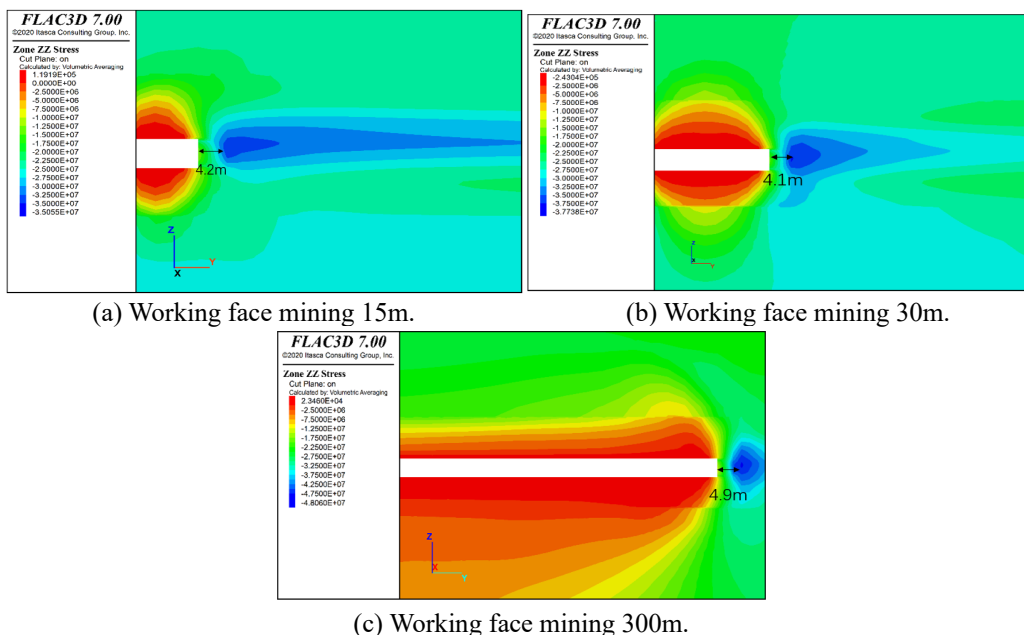


Figure 4. The vertical stress change cloud map of different distances in working face mining.

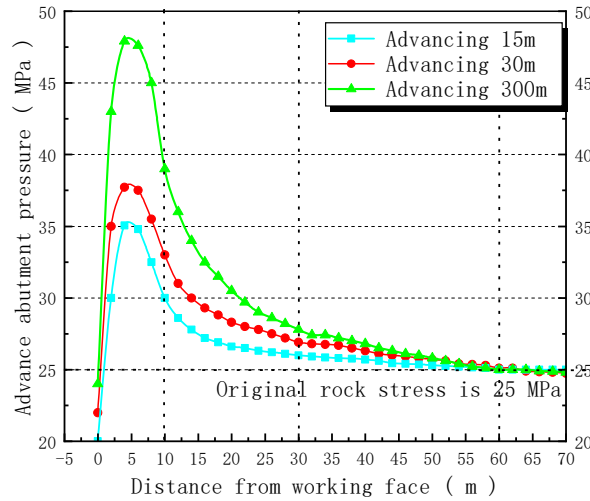


Figure 5. Working face mining advance abutment pressure curve at different distances.

It can be seen from Fig.4 and Fig.5 that when the 300 m working face is squared once, the peak value of the advance abutment pressure is 48.06 MPa, the peak value of the concentrated stress is about 4.9 m away from the working face, and the stress concentration factor is 1.92. The advance abutment pressure increases first and then decreases with the increase of the distance from the head of the working face, and increases from the original rock stress to the peak value of the advance abutment pressure 4.1 ~ 4.9 m in front of the working face. When the distance of the advance working face is less than 10 m, the roadway is greatly affected by the advance abutment pressure, the surrounding rock is seriously broken, and the roof bearing capacity is poor. In the range of 10 ~ 30m, the slope of the abutment pressure curve and the stress concentration factor are obviously reduced, and the surrounding rock of the roadway is in a relatively stable state at this stage. When the advance working face is 30 m, it gradually tends to be stable. The advance abutment pressure changes little in the range of 30 ~ 60 m, and then decreases to 60 m in front of the working face, which is similar to the change trend of the original rock stress. Considering the site construction conditions and the maximization of economic benefits, grouting reinforcement should be carried out within the range of 10 ~ 30 m ahead of the original support to improve the bearing capacity of the surrounding rock of the roof.

3.3. Supporting parameters of grouting anchor cable

(1) Grouting time

The advance grouting anchor cable construction should be completed within 30 m of the advance working face, and the grouting reinforcement should be completed within 10-30 m of the advance working face.

(2) Density of grouting anchor cable

According to the previous analysis, the support strength (P_c) of the reinforced grouting anchor cable in the mining roadway of the advanced dynamic pressure zone of the 24100 working face is not less than 50.6 kN/m². Firstly, the support parameters of the reinforced grouting anchor cable are determined by theoretical calculation.

Combined with the actual situation of No.11 Coal Mine, the diameter of hollow grouting anchor cable is $\Phi 22$ mm, the anchoring force is 300 kN, and the breaking load is 420 kN. The number of grouting anchor cables at least constructed

within 30 m ahead is:

$$n = \frac{P_c S}{K_1 F} \quad (6)$$

In the formula: n is the number of grouting anchor cables, root; P_c reinforcement support strength, 50.6kN/m²; S is the area of roadway roof ahead of 30 m, 160 m²; K_1 is the effective coefficient, take 0.8; F is the anchoring force of grouting anchor cable, 300 kN.

After calculation, $n = 34$, that is, in the range of 30 m ahead, at least 34 grouting anchor cables are used in the roof, and the support density is 0.21/m².

(3) Anchor cable row spacing

In the range of 30 m ahead, at least 34 grouting anchor cables are constructed in the roof, and 2 grouting anchor cables are constructed in each row. There are at least 17 rows of anchor cables in the roof of the leading section, and the row spacing i is:

$$i = \frac{30}{17-1} = 1.875 \quad (7)$$

Therefore, the row spacing of the anchor cable is not more than 1.875 m.

(4) Length of grouting anchor cable

In order to meet the requirements of support strength, it should meet:

$$L = L_a + L_b + L_c + L_d \quad (8)$$

In the formula: L is the total length of the anchor cable, m; L_a is the anchorage length in stable rock strata.

$$L_a \geq K_1 \frac{d_1 f_a}{4 f_c} = 1.16 \quad (9)$$

In the formula: K_1 safety factor, take 1.2; the diameter of d_1 is 0.022 m; the tensile strength of f_a is 1760MPa; the bonding strength of f_c anchor cable and anchoring agent, 10MPa; L_b needs the thickness of the suspended unstable rock layer, which is 5.6 m. The thickness of L_c plate and anchorage is 0.15 m. The exposed tension length of L_d is 0.25 m.

The theoretical length of grouting anchor cable is calculated to be $L = 7.16$ m. Therefore, the length of grouting

anchor cable should not be less than 7.16 m. Combined with the actual situation on site and considering a certain safety factor, the length of grouting anchor cable is determined to be 7.3 m. Strengthen the support of the mining roadway in the advanced dynamic pressure area of the 24100 air roadway. Combined with the existing roadway support, the specifications of the grouting anchor cable are $\Phi 22 \text{ mm} \times 7300 \text{ mm}$, and the support density is 0.21 roots/m^2 .

(5) Injecting paste material

Through field experiments, cement-based grouting materials were selected for anchorage support. P.O 42.5 ordinary Portland cement and water were selected to prepare cement slurry according to water-cement ratio of 0.8, and 4 % ACZ-I additive was added to improve the rheological

properties of slurry and enhance the stability of surrounding rock.

4. Simulation Verification

4.1. Grouting anchor cable reinforcement support scheme

Based on the above analysis, the reinforcing anchor cable support design is carried out for the advance section of 24100 return air roadway. The specification of grouting anchor cable is $\Phi 22 \text{ mm} \times 7300 \text{ mm}$. The supporting section is shown in Figure 6.

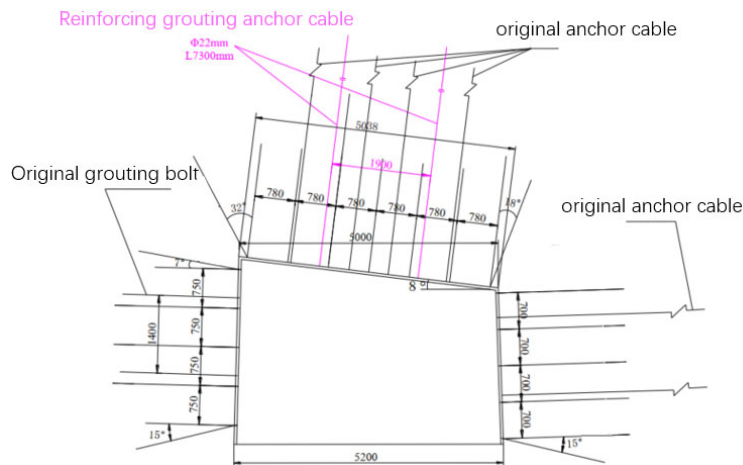
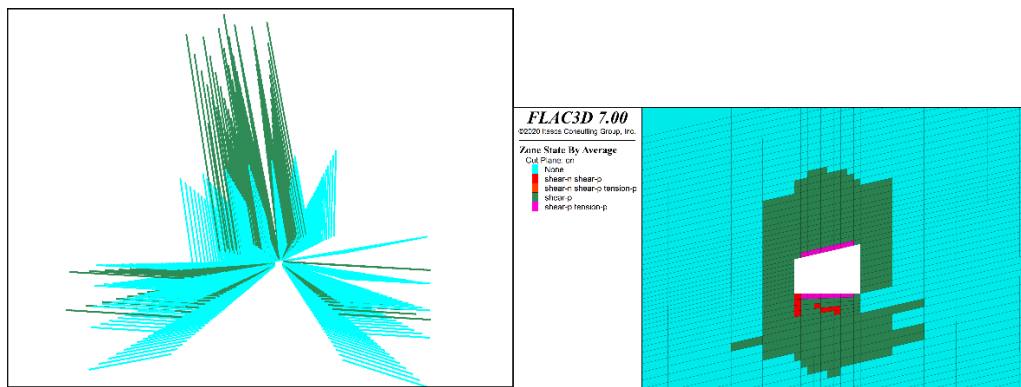


Figure 6. Pre-support section diagram of reinforcing grouting anchor cable in 24100 return air roadway.

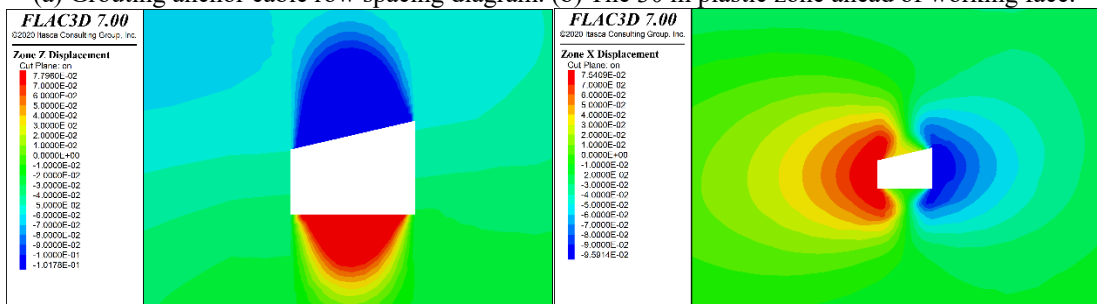
4.2. Feasibility analysis of advanced support scheme

Continue to use the above chapter model, after the model is assigned to the balance, the return air roadway is excavated. After the excavation is completed, the bolt anchor cable and

grouting anchor cable are set. The bolt and anchor cable support parameters continue to use the original support technology scheme, and the operation is balanced. The working face is excavated in a square (300m), and the deformation of the surrounding rock of the roadway is analyzed, as shown in Figure 7.



(a) Grouting anchor cable row spacing diagram. (b) The 30 m plastic zone ahead of working face.



(c) The 30 m vertical displacement ahead of working face. (d) The horizontal displacement.

Figure 7. Deformation law of roadway surrounding rock.

Through numerical simulation, it is found that when the air roadway is 30 m ahead of the working face, after the advanced strengthening support, only the shallow surrounding rock damage occurs in the roadway floor, which is less affected by the mining of the working face; after strengthening support, the roof subsidence is 101.8 mm, which is 31.6 % lower than that of the original support. The roof subsidence is 76 mm, which is reduced by 37.5 % compared with the original support. The strengthening support of the leading section of the air roadway can maintain the stability of the surrounding rock of the roadway.

5. Conclusions

In view of the problem that the 24100 return air roadway is affected by the mining of the working face, the roof cracks of the inclined roadway develop, break and sink, and form more net pockets. Through a series of studies, the following conclusions are obtained:

(1) Based on the suspension theory, the original support strength of the roadway roof is calculated to be 238.9 kN/m², and the support strength required for the roof under the influence of dynamic pressure is 289.5 kN/m². It is determined that the support strength required for the advanced section of the roof of the 24100 working face should not be less than 50.6 kN/m².

(2) The stress distribution and deformation law of surrounding rock of roadway in advance section are analyzed by FLAC 3D. The numerical simulation results show that the influence range of roadway by advance abutment pressure is about 60 m, the peak value of advance abutment pressure is 48.06 MPa, and the stress concentration coefficient is 1.92.

(3) The slope of the abutment pressure change curve and the stress concentration coefficient are obviously reduced in the range of 10 ~ 30m, and gradually tend to be stable when the working face is 30 m ahead. The abutment pressure changes little in the range of 30 ~ 60m. The support should be strengthened in the range of 30 m ahead of the working face, the deformation of the surrounding rock of the roadway should be controlled, and the grouting reinforcement should

be carried out in the range of 10 ~ 30m ahead of the working face.

(4) The hollow grouting anchor cable with a specification of $\Phi 22\text{mm} \times 7300\text{mm}$ and a row spacing of $1900\text{mm} \times 1500\text{mm}$ can achieve better reinforcement support effect and ensure safe and efficient mining of the working face. The FLAC 3D numerical simulation is used to verify the advanced reinforcement support scheme. After the grouting anchor cable is used to strengthen the support, the roof subsidence is reduced by 40.3 % compared with the original support, and the two-side convergence is reduced by 37.5 %. The anti-deformation ability of the surrounding rock of the roadway roof after the grouting anchor cable reinforcement support is improved.

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