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Research on the Instability Mechanism and Key Control Technologies of Surrounding Rock in Ultra-large Section Dense Strip Roadways

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Abstract: To study the instability mechanism of the surrounding rock and the reasonable width of coal pillars in ultra-large cross-section dense strip roadways, the 7810 working face roadway in Wuyang Coal Mine was taken as the research object. Theoretical analysis, numerical simulation and on-site industrial tests were adopted to investigate the instability mechanism of the surrounding rock and the stress distribution law of the surrounding rock in ultra-large cross-section dense strip roadways. The research results show that when the width of the coal pillar between the strips is less than 5m, the vertical stress inside the coal pillar is generally small, and the stress peak is less than the initial stress; when the width of the supporting coal pillar is 5-8m, the vertical stress inside the coal pillar is greater than the initial stress, but the stress concentration is not severe, and the stress concentration coefficient is 1.05-1.49; when the width of the supporting coal pillar is between 8-12m, the peak supporting pressure appears in the center of the strip coal pillar, the overall supporting pressure inside the coal pillar is large, and the stress concentration coefficient is 1.73-2.68. The surrounding rock supporting pressure is high and it is easy to accumulate a large amount of elastic energy, which is not conducive to the long-term stability of the strip coal pillar and the safe use of the roadway. Therefore, under the existing conditions, the width of the coal pillar between the dense strip roadways is 5-8m. The research results provide a reference for the excavation and maintenance of dense strip roadways under similar geological conditions.

Keywords: Large section; Strip mining; Coal pillar protection; Surrounding rock control.

1. Introduction

China is rich in coal resources and has strong market demand. In the present and future period of time, coal is still the main energy in our country. Coal is a reliable, clean, convenient and practical energy, and a bridge to the future energy system [1-3]. At the same time, underground mining and charging is an important technical approach for green mining [4-7]. Generally, in order to better control the deformation of overlying strata in the mining area, strip mining is often adopted [8-9].

In the field of coal mining, strip mining has attracted much attention because of its unique mining mode. However, due to the different disposal methods adopted in the gob after strip mining, the final surface settlement has slightly different results. Among them, the use of filling method to control the goaf roof after strip mining is a more common and commonly used means. In theory, if this method can be implemented perfectly, it can indeed ensure the stability of the mining area to a large extent. But in the actual operation process, there are many thorny problems to be solved.

On the one hand, the choice of strip width is a difficult one. If the bandwidth is too narrow, the mining efficiency of coal resources will be greatly reduced, and the economic benefits cannot be maximized. However, if the bandwidth is too wide, the span of the goaf will increase, and the pressure on the roof will increase sharply, which will greatly increase the risk of mining. This requires that the mining team must rely on detailed and accurate geological survey data, rigorous mechanical calculations and rich practical experience to determine the most appropriate mining strip width.

On the other hand, the topping rate of the filling body often fails to meet the pre-set requirements, which has become a "stubborn disease" that cannot be ignored. If the backfill body is not closely and efficiently connected to the roof, it will form a void of different sizes between the roof and the backfill body. The existence of these voids not only weakens the supporting efficiency of the backfill to the roof, but also reserves "hidden danger space" for the subsidence and deformation of the roof. Once the roof begins to sink, the rate of subsidence will be accelerated due to the buffer effect of the void, and eventually the surface settlement will exceed the expected design value.

What is more serious is that there is often a lack of effective monitoring means throughout the mining and filling process. Without real-time and accurate monitoring data feedback, it is impossible to detect the subtle changes of goaf roof, backfill and surface in time. When it is found that the surface settlement has exceeded the design value, a series of irreparable losses such as the cracking of ground buildings and the fracture of underground pipelines will be caused, which will bring heavy blows to the surrounding environment, production and life, and even the economic development of the whole region.

After analysis, Zhang Yuanchao et al. [10] clearly put forward that once the small structure composed of supporting structure and coal pillar becomes unstable, it will directly lead to roadway instability. Zhang Hongwei et al. [11] believe that if overlying rock movement is in an insufficiently stable state, it will become a key factor leading to roadway deformation and instability. Yang Ke et al. [12] studied the evolution characteristics of surrounding rock under secondary mining operation of narrow coal pillar, and found that the stress distribution of surrounding rock was different and showed regional evolution, which was an important reason for the deformation of roadway. Wang Weijun [13] made a comparative analysis of the influence of uniform expansion of plastic zone and malignant expansion on roadway deformation, and concluded that effective control of malignant expansion of plastic zone is an effective way to achieve stability and maintenance of surrounding rock. Li Xuehua [14] et al., through the analysis of the classic case of narrow coal pillar, concluded that anchor bolt can strengthen the shallow surrounding rock and provide certain lateral binding force. Kang Hongpu et al. [15] proposed that high preload and high strength anchor cable support can significantly reduce the deformation of surrounding rock.

In this paper, the densely banded roadway at the 7810 working face of Wuyang Coal Mine is taken as the object. Based on theoretical analysis, numerical simulation and field industrial test, the plastic zone distribution and stress law of surrounding rock of the roadway are analyzed, the instability mechanism is explored, and the support scheme is proposed in view of the serious instability of the roadway at the working face. The industrial test was carried out in the field. It is expected that the research results of this paper can provide valuable reference for the roadway support work under similar conditions.

2. Engineering Background

Shanxi Lu 'an Group Wuyang Coal Mine 7810 working face is located in +600 level 78 mining area. 7810 return air lane strike length is 895m, working face elevation is 457-553m, 7810 cutting eye strike length is 135.5m, working face elevation is 542-553m, 7810 transport lane strike length is 863m, the working face elevation is in the range of 438m to 542 m, the surrounding situation is as follows: In the north of the working face is the gob 7808; The south is unmined solid coal; Preparation roadway in the west connecting the 78 mining area; The eastern part is also unmined solid coal.

The 7810 return air lane and 7810 transport lane are rectangular sections with a width of 5.0m and a height of 6m. 7810 return air lane opens from 78 belt lane with an Angle of 66° and is 34 meters away from the measuring point A06. After the opening, the waist line driving will pass under 78 track lane and 78 special return air lane respectively (mezzanine shall not be less than 800mm). After the opening is driven 68m, it will turn in at an Angle of 145.3°. Turn the middle and follow the waist line to continue driving through the bottom of 78.2# special return air lane (mezzanine shall not be less than 800mm), and then walk the waist line to find the bottom plate, and then drive along the bottom plate until the stop digging position. The total length of 7810 return air lane is 895m. The inclination of coal seam along the roadway is 5 ~ 8°, with an average of 6°.

3. Strip Mining Design Method

In strip mining, the recovery ratio is a key design element, which is like a tightly intertwined link between the stability of the coal pillar, the safety of the stope, and the effective control of surface deformation, and the recovery rate of the resource. Therefore, the core point of strip mining design, in the final analysis, is to take into account the control effect of rock and surface deformation, the stable state of the stope and the recovery rate of resources, based on which the appropriate drawdown ratio is drawn up.

1) Engineering analogy. Engineering analogy is a widely used method in the field of engineering. Based on the principle of similarity, it compares the engineering projects being studied or designed with the existing successful projects similar in geological conditions, engineering structure, construction technology and many other aspects, and analyzes the distribution of plastic zone of roadway surrounding rock, stress coping strategies, and implementation effects of support schemes in these similar projects. To infer the possible problems in the current project and the effective measures that can be taken [16]. The basic parameter selection is described below.

The extracted strip width D is calculated in the equation (1)

$$D = \left(\frac{1}{14} \sim \frac{1}{10}\right) \times H \tag{1}$$

The reserved strip width B is calculated in the formula (2).

$$\left(\frac{B}{M}\right) > 5 \tag{2}$$

In the above formula: H is mining depth, m; M is the mining height of coal seam, m.

2) Limit equilibrium theory. The theory of limit equilibrium is one of the most important basic theories in engineering mechanics and geotechnical engineering. It mainly studies the equilibrium conditions of objects or rock and soil bodies when the force reaches the limit state. In coal mining, for the surrounding rock of the roadway, when the combination of abutment pressure, ground stress and other external forces on the surrounding rock reaches the critical state of plastic deformation and imminent destruction of the surrounding rock, the limit equilibrium theory can be used to analyze the stress distribution of the surrounding rock and the development trend of the plastic zone, so as to provide a basis for reasonable design of supporting structure. To ensure that the project can maintain stability near the limit stress state, and ensure the construction safety and engineering quality. The theoretical calculation of limit equilibrium can be solved by the mechanical relation of Mohr-Coulomb stress circle [16-18]. The relation of the existence of three-way stress under the limit equilibrium condition is shown in equation (3).

$$\sigma_1 = \frac{1+\sin\varphi}{1-\sin\varphi}\sigma_3 + \frac{2c\cos\varphi}{1-\sin\varphi} \tag{3}$$

In the above formula:$\sigma 1$ is the axial stress, the unit is MPa; $\sigma 3$ is the lateral stress, unit: MPa; c is coal rock (volume) cohesion, MPa; is the internal friction Angle, unit(°).

At the edge of coal pillar, the lateral stress is 0. In the yield zone, when the lateral stress reaches the junction with the nuclear zone, the lateral stress is equal to the original rock stress before the strip mining, that is,$\sigma_3=\gamma H$. By substituting $\sigma_3=\gamma H$ into(3),the limiting equilibrium condition in the three-way stress state is obtained, as shown in(4).

$$\sigma_1 = \frac{1+\sin\varphi}{1-\sin\varphi}\gamma H + \frac{2\cos\varphi}{1-\sin\varphi} = \sigma_z \tag{4}$$

In the above formula, γ is the bulk density of overlying rock, kN/m^3; H is the depth of coal mining, m; In theory,$\sigma_1 = \sigma_z$.

In the engineering practice of coal mining, strip coal pillar plays an important role in carrying and supporting. A careful observation of the structure of the strip coal pillar shows that there are plastic zones with specific properties on both sides. This plastic zone is not random, it has a certain width range, and presents very special mechanical properties at the boundary, and its boundary supporting ability is almost zero, which means that at this boundary position, it is almost impossible to provide effective supporting force for the surrounding rock mass, and it is difficult to resist external pressure deformation. However, it is worth noting that although the plastic zone has been formed, the peak stress in it is strictly limited and does not exceed the ultimate strength of the coal pillar itself. The stress in the nuclear region is parabolic, and the details can be seen in Figure 1. In stable equilibrium, the axial stress of coal pillar presents a "saddle shape" distribution, as shown in FIG. 2.

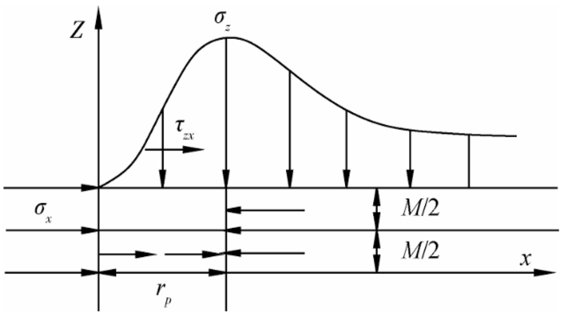

Figure 1. Mechanical calculation model of coal pillar yield zone width

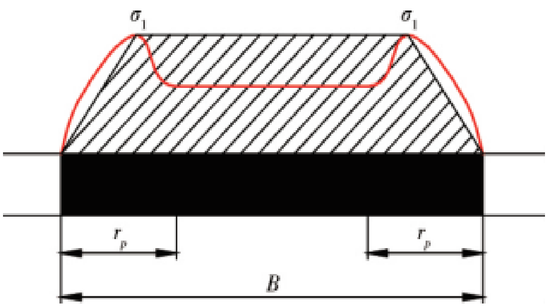

Figure 2. Mechanical model for calculating the width of coal pillar

The differential equation of interfacial stress balance in yield region is shown in the following equation (5).

$$\frac{\partial \sigma_x}{\partial x}+\frac{\partial \tau_z}{\partial_z}=0$$

$$\frac{\partial \tau_{zx}}{\partial x}+\frac{\partial \sigma_z}{\partial_z}=0 \tag{5}$$

$$\tau_{zx}=-(c+\sigma_z \tan\varphi)$$

The stress boundary conditions are shown in the equation (6).

$$[\sigma_z]=\sigma_z, [\sigma_x]=\beta[\sigma_z]=\beta\sigma_z \tag{6}$$

The calculation of the critical width of coal pillar in the yield area is shown in the formula (7).

$$r_p=\frac{M\beta}{2\tan\theta}\ln\left(\frac{\sigma_z+\frac{c}{\tan\theta}}{\frac{c}{\tan\theta}+\frac{\sigma_x}{\beta}}\right)+\left(\frac{M}{2}-z\right)\times \tan\theta \tag{7}$$

In the above formula, M is the mining height, the unit is m;β is the side pressure coefficient, μ is Poisson's ratio; σ_z is the ultimate strength of a coal pillar, expressed in MPa.

According to the above formula, the width of the yield zone of the coal pillar is different at different heights. For the convenience of engineering calculation, we can choose the width of yield zone at the position of k1=0.9=M/2 with the height of coal pillar to simplify the treatment. At the same time, the disturbance factor is introduced. The function of this factor is to fully consider the influence of different mining methods on the loosening and destruction of coal pillar side.

In view of the relatively hard characteristics of the coal seam roof, the occurrence of caving phenomenon in the roof is less frequent in the actual situation, and it can even be treated as if the caving does not occur to a certain extent. In this condition, if $\sigma_x=0$ is ordered, formula (7) can be simplified to formula (8).

$$r_p=\frac{Md}{2\tan\theta}\ln\left(1+\frac{\sigma_1\tan\theta}{c}\right)\beta \tag{8}$$

In the formula, d is the mining disturbance factor, whose value ranges from 1.1 to 3.3.

In order to make the theoretical calculation more convenient, the following assumptions are set: First, it is assumed that the width of the yield zone in all directions around the strip pillar is consistent, and there is no difference; Second, it is assumed that part or all of the weight of the overlying strata within the area delimited by the vertical line of the stope boundary needs to be carried by strip coal pillars. Based on these two assumptions, the corresponding calculation formula is shown in (9).

$$p=p_1+p_2 \tag{9}$$

Where: p is the total load per unit volume on the coal pillar, unit MN; p1 and p2 are respectively unit volume loads in the coal pillar yield zone and nuclear zone, unit MN.

4. Stability Analysis of Strip Coal Pillar

(1) In coal mining operation, the stability of strip coal pillar is directly related to the safety and sustainability of the whole mining area, so it is of great significance to analyze it deeply. From the perspective of structural composition, the plastic zone on both sides of the strip coal pillar is one of the key factors affecting the stability. As mentioned above, the plastic zone has a certain width and the boundary support ability is almost zero, which makes the coal pillar easily disturbed by external pressure at the boundary. When the surrounding rock mass changes slightly, the pressure is applied to the boundary of the plastic zone, where the first small deformation may occur due to the lack of effective support. Even if the plastic zone is subjected to greater pressure under complex stress environment, the main structure of the coal pillar can remain relatively intact and will not collapse rapidly due to local

excessive pressure, which provides a certain guarantee for the overall stability of the coal pillar.

Considering the influence of coal pillar size and shape on stability, the formula for calculating coal pillar strength [19-22] is as follows.

$$\sigma_p = \sigma_m (0.64 + 0.36B/M)^n \tag{10}$$

In the above formula: σ_p is the strength of coal pillar, MPa; σ_m is the strength parameter of coal and rock, MPa; n is a constant.

(2) According to the coal pillar area bearing theory, the load borne by the coal pillar is essentially equivalent to the weight of the overlying rock pillar within the stope range supported by it, and the average load of continuous coal pillar can be calculated according to formula (11).

$$\sigma_{pa} = \sigma_z (\frac{B+D}{B}) \tag{11}$$

In the above formula: $\sigma_z = \gamma H$, γ is the bulk density kN/m3 of the rock; H is the mining depth, m; σ_{pa} is the load with coal pillar, MPa; The remaining parameters are the same as above.

(3) Based on the above discussion, the optimal design process of strip mining is obtained as follows: (a) The design parameters are selected by engineering analogy method; (b) The strip width was optimized according to the limit equilibrium theory; (c) Select the appropriate safety factor and carry out numerical simulation analysis on the selected parameters; (d) Based on the above analysis and calculation results, the optimal remaining width of coal pillar is determined.

5. Analysis of Reasonable Width of Remaining Coal Pillar

In order to study the yield failure and stress characteristics of surrounding rock under different coal pillar widths in 7801 working face, a numerical model based on FLAC3D was built. According to the geological data, the model size is selected, and the displacement constraint and uniform load are set to the surrounding boundary, so as to simulate the related characteristics of roadway surrounding rock when the coal pillar width is 4m, 5m, 6m, 8m, 10m and 12m.

5.1. Comparative analysis of surrounding rock stress with different pillar widths in strip roadway

According to the actual geological data, the buried depth of the 3# coal seam in the 7810 working face is 386m on average, and its direct overburden load q is about 9.6MPa, and the side pressure coefficient is λ=1.2. 7810 working face with rectangular section of roadway, 5m wide, 6m high. Width of coal pillar between strip lanes: 4m, 5m, 6m, 8m, 10m, 12m. The numerical simulation model is shown in Figure 3. The comparative analysis of surrounding rock stress with different coal pillar widths in strip roadway is shown in Figure 4-9. The maximum stress diagram of inter-strip coal pillar is shown in Figure 10.

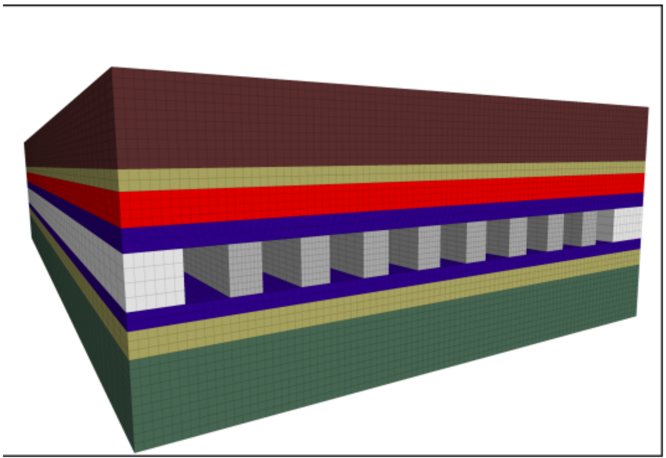

Figure 3. Numerical simulation model diagram

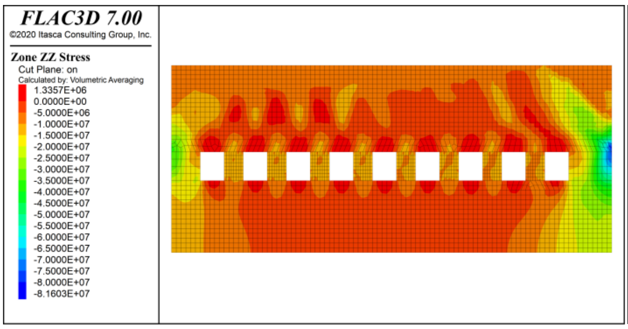

Figure 4. Stress nephogram when the width of coal pillar between strip lanes is 4m

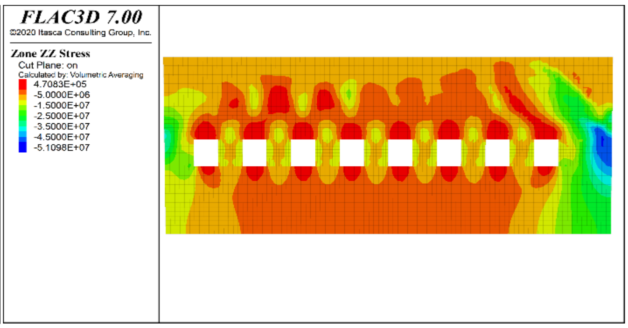

Figure 5. Stress nephogram when the width of coal pillar between strip lanes is 5m

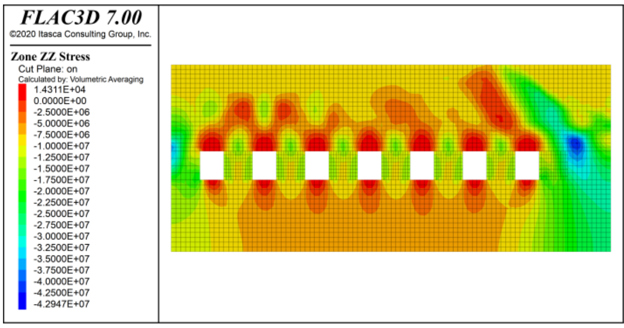

Figure 6. Stress cloud diagram when the width of coal pillar between strip lanes is 6m

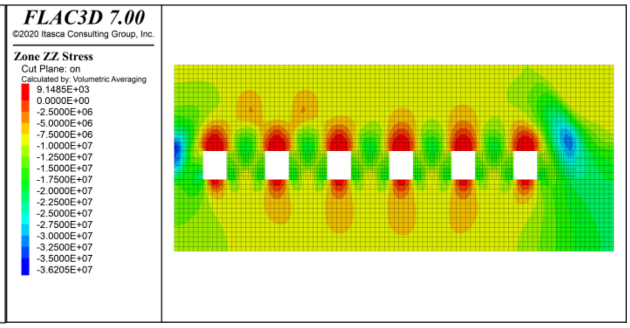

Figure 7. Stress nephogram when the width of coal pillar between strip lanes is 8m

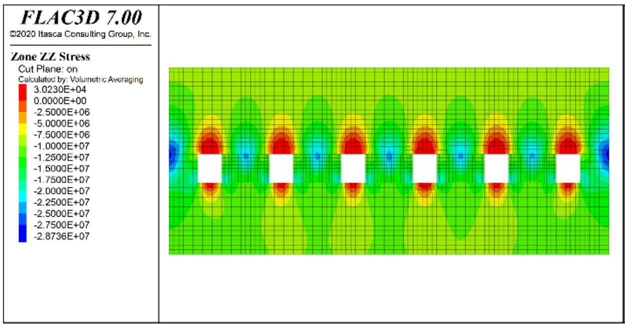

Figure 8. Stress nephogram when the width of coal pillar between strip lanes is 10m

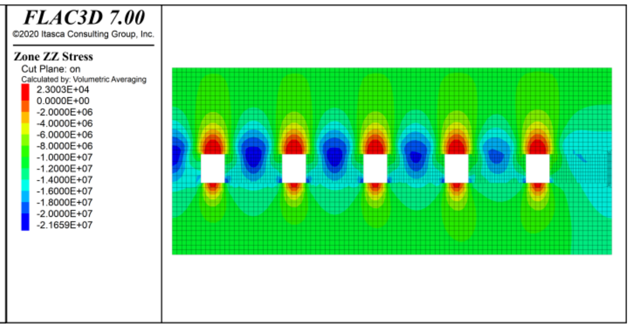

Figure 9. Stress cloud diagram when the width of coal pillar between strip lanes is 12m

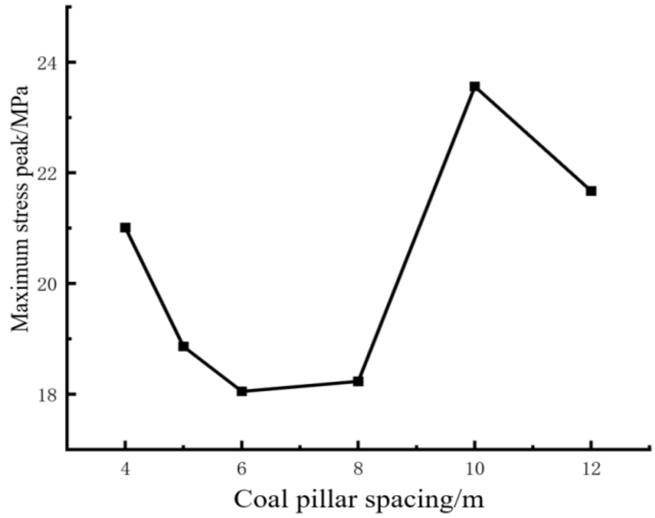

Figure 10. Maximum stress diagram of coal pillar between strip lanes

According to the vertical stress distribution cloud diagram of coal pillar with different widths from FIG. 3 to FIG. 9, it can be seen that when the width of coal pillar is less than 5 meters, the internal vertical stress is low, and the peak value is lower than the initial stress, because the coal pillar is seriously damaged and the supporting capacity is lost. At 5-8 meters, the vertical stress is greater than the initial stress, the concentration is not serious, and the coefficient is 1.05-1.49. When 8-12 meters, the peak abutment pressure is in the coal pillar of the section, the whole is large, the coefficient is 1.73-2.68, the surrounding rock pressure is large, and the elastic energy is easy to gather. With the increase of the width of the coal pillar, the stress concentration coefficient increases, the peak adverse impact prevention and roadway support, from the vertical stress distribution law, the width of the coal pillar should be arranged at 5-8 meters.

5.2. Comparative analysis of displacement of surrounding rock with different pillar widths in strip roadway

In the research and practice of coal mining engineering, the setting of pillar width between strip lanes has a significant effect on surrounding rock displacement, so it is significant to carry out in-depth comparative analysis of surrounding rock displacement under different pillar widths. Figure 11-16 shows the vertical displacement diagram of different pillar widths between strip lanes.

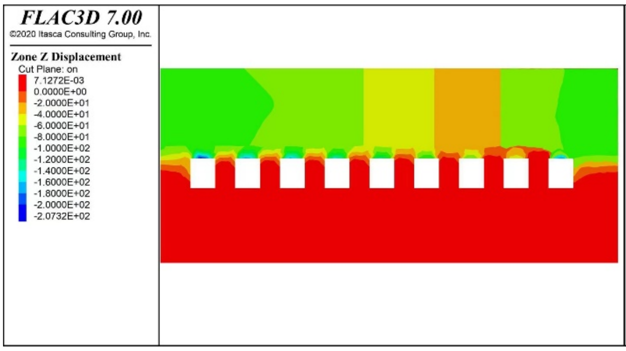

Figure 11. Distribution Cloud map of vertical displacement when the width of coal pillar is 4m

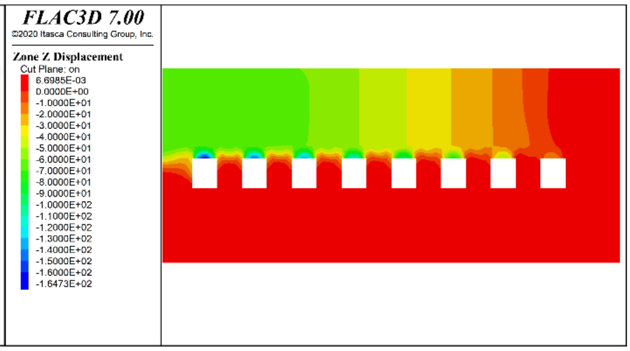

Figure 12. Distribution cloud map of vertical displacement line line when the width of coal pillar is 5m

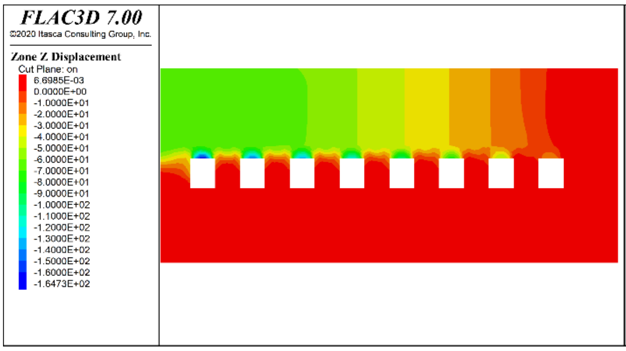

Figure 13. Distribution Cloud map of vertical displacement when the width of coal pillar is 6m

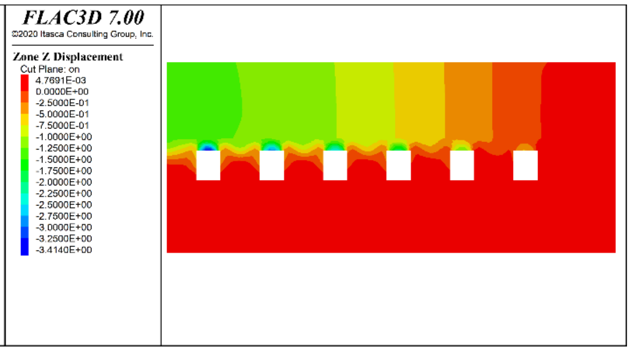

Figure 14. Distribution cloud map of vertical displacement line line when the width of coal pillar is 8m

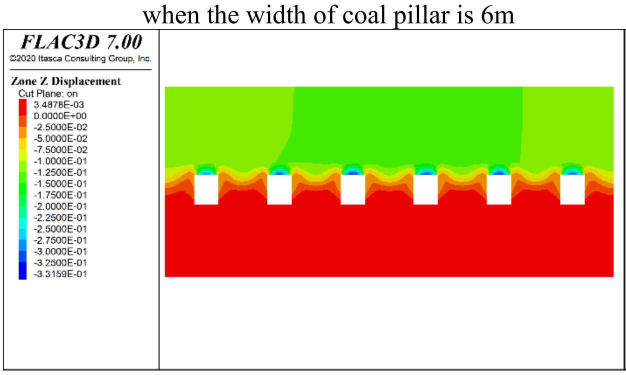

Figure 15. Distribution Cloud map of vertical displacement line when the width of coal pillar is 10m

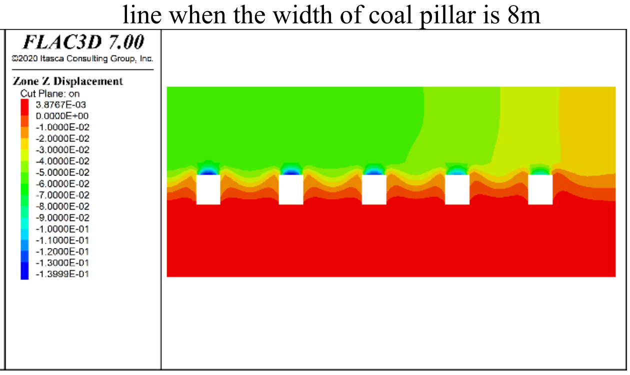

Figure 16. Distribution cloud map of vertical displacement line when the width of coal pillar is 12m

Since the contour cloud map of surrounding rock can only see the deformation characteristics and rules of surrounding rock near the roadway macroscopically, and cannot reflect the change characteristics of surrounding rock in different directions quantitatively, it can only understand the deformation characteristics and rules of surrounding rock of the roadway as a whole, and cannot reflect the deformation characteristics of surrounding rock of the roadway at a certain point. In order to further analyze the distribution of vertical displacement of surrounding rock of the roadway, Therefore, it is necessary to study the characteristics of displacement change of roadway surrounding rock. FIG. 17 shows the deformation curves of strip roadway surrounding rock with different pillar widths.

Through the above data and analysis, it can be seen that after the optimal plan is adopted to support the roadway at the 7810 working face of Wuyang Coal Mine, the stability of the roadway surrounding rock is improved, especially the deformation of the roadway roof and floor surrounding rock is controlled, which can basically ensure the normal and safe production of the roadway during the service period.

As can be seen from Figure 11-16, the roof of a strip roadway bends and dips after mining. The maximum roof dips of each strip roadway are in the middle of the goaf, and the maximum subsidence values of each strip goaf are symmetrically distributed. When the strip roadway spacing coal pillar is 4m, the deformation of the top and floor of the strip roadway is 448mm, which is relatively large, and the coal pillar is seriously broken inside, which is not conducive to the effective anchoring function of the anchor rod, and the supporting ability of the overburden rock is greatly reduced, resulting in serious roadway deformation, which is not conducive to the maintenance of the roadway. When the width of the coal pillar is between 4-12m, the deformation of the top and bottom shows a trend of decreasing first and then increasing. The deformation is the smallest at 5m, and the deformation of the top and bottom is 252mm. The change trend of the deformation is relatively gentle between 5-12m, and the deformation has been kept in a small range.

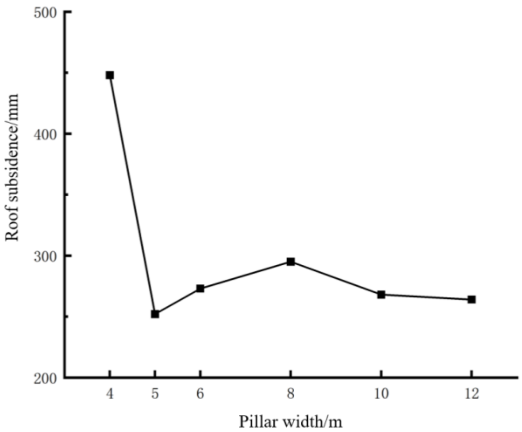

Figure 17. Surface subsidence curve of strip roadway after excavation

6. Industrial Experiment

6.1. Support scheme design

After the excavation work is completed, the stress state of surrounding rock immediately changes and begins to be re-distributed due to the disturbance caused by excavation construction. At this time, due to the pressure exerted by the overlying strata, the deformation amplitude of both sides of the coal pillar increases significantly, and its load-bearing capacity also decreases. As a result, the roof and top of the coal pillar side have more significant deformation than that of the solid coal side, showing typical asymmetric deformation characteristics. If measures are taken to strengthen the support strength of coal pillar side, the overlying rock pressure can be transferred to the solid coal side, and then the degree of asymmetric deformation can be effectively reduced.

After the end of the mining process, the coal pillar behind the working face will be subjected to the advance abutment pressure, and the stress inside the coal pillar will inevitably increase significantly. In this case, strengthening the side support work of coal pillar can effectively deal with a series of problems caused by the increase of stress inside coal pillar.

In this paper, based on the above analysis content, combined with the engineering analogy method, the layout scheme of the anchor cable is initially drawn up. In the future, it is necessary to further consider the optimization adjustment and improvement of the bolt design scheme according to the data obtained from the actual field observation. Specific support schemes are as follows:

(1) The top plate is made of 5 left-handed non-longitudinal rebar steel bolts with specifications of 22mm×2400mm. This specific specification of the bolt has significant advantages, 22mm rod diameter gives the bolt enough strength to withstand the huge pressure from the top of the roof; With the use of 5000mm×220mm×23mm W-shaped steel belt, its ultra-long 5000mm size can cover a large area of the roof area, making the support range more extensive, to avoid weak points in support; Two in a row, staggered with the bolt.

(2) The supporting design of the two walls is as follows: Each wall adopts 5 bolts, and the specifications of these bolts are completely consistent with the specifications of the bolts used in the roof, so as to ensure the integrity and stability of the supporting system. In terms of arrangement spacing, the arrangement is strictly in accordance with the parameters of 700mm × 1000mm, that is, the transverse adjacent bolt spacing is 700mm, and the spacing between the longitudinal rows and rows is set to 1000mm. Such a density is accurately calculated, which can provide just the right reinforcement effect for the two rows. In particular, it should be noted that the top and bottom bolts should be tilted outward for anchoring by 15° respectively during installation. The setting of this tilt Angle has a clever mechanical principle. On the one hand, it can better fit the irregular shape of the surface of the two sides and enhance the friction between the bolts and the coal and rock mass. On the other hand, the outward tilt can form an effective pulling effect on the weak areas of the upper and lower parts of the two sides, preventing the occurrence of local caving phenomenon.

(3) Reinforcement support operation is carried out for coal pillar wall. In view of the serious deformation degree of the roadway at the side of the coal pillar, it is necessary to take corresponding reinforcement measures: add an additional anchor cable at the position of the roof at the side of the coal pillar 300mm from the side of the roadway, so that the cable layout in this area will form a 2-1-2 arrangement pattern with the original anchor cable; At the same time, two anchor cables are added to the side, and channel steel or waste I-steel is used together to enhance the supporting ability of the side and ensure the stable state of the roadway. The anchor cable specification is 22mm×6200mm, and the row distance is 1400mm×2000mm.

6.2. Roadway surface displacement monitoring

The displacement monitoring of the roadway surface for a period of 60d was carried out by using the "cross" layout method on the roadway surface, and a total of 3 measuring stations were arranged. The location of the measuring points of the roadway section at the measuring stations was shown in Figure 13. The three stations are located at 20m away from the intersection of 7810 return air Lane, 15m away from the intersection of 7810 Transport lane return air Lane and 20m away from the intersection of 7810 Transport lane return air lane.

The results obtained from the observation data of station 1#, station 2#, and station 3# are summarized in Figure 18-20.

As shown in Figure 18-20, according to the monitoring data of roadway surface displacement, it can be seen that the deformation of surrounding rock is mainly reflected in the two walls and the floor of roadway 7810. The maximum displacement of the two walls is about 503mm, while the maximum displacement of the top and floor is relatively small (389mm). It is particularly noteworthy that on the 18d after excavation, the deformation rate of the roof and floor of the roadway and the surrounding rock of the two sides reach the peak value, and the average deformation rate is 11.8mm/d and 13.6mm/d, respectively. However, after this initial severe deformation stage, although the deformation of the roadway continues to increase, the overall deformation rate slows down significantly. After 35d, the deformation speed of the roof and floor of the roadway and the two sides of the roadway gradually become stable, which indicates that the deformation of the surrounding rock of the roadway has basically reached saturation, and the subsequent changes are no longer significant.

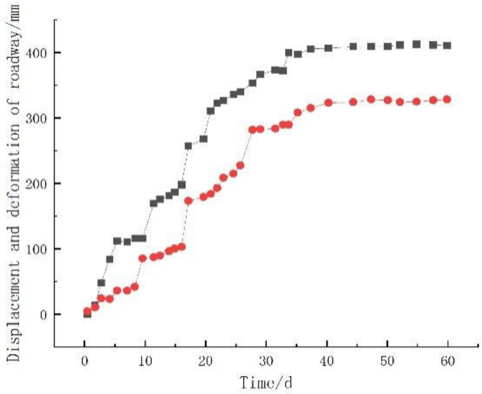

Figure 18. Displacement of roadway surface at station 1#

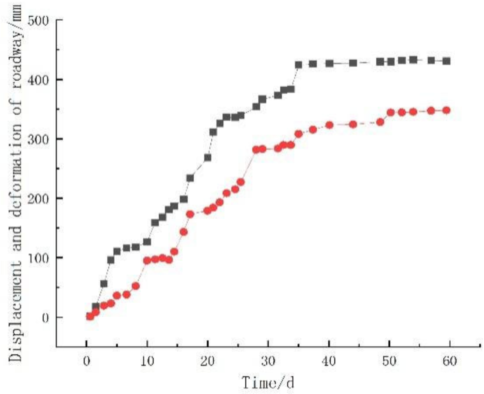

Figure 19. Displacement of roadway surface at station 2#

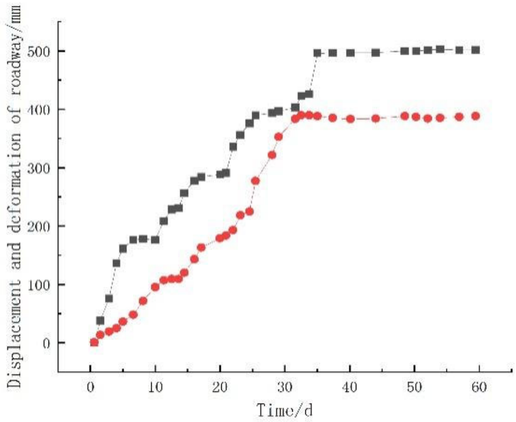

Figure 20. Displacement of roadway surface at station 3#

7. Conclusion

Numerical simulation software is used to carry out numerical analysis models for different coal pillar widths between strip lanes, and study the schemes for maintaining coal pillar widths in various sections, providing basis for the final determination of reasonable coal pillar widths and drawing conclusions:

(1) The comparative analysis of surrounding rock stress of coal pillars with different widths between strip lanes shows that: after the excavation of strip lanes, once the width of coal pillars is less than 5 meters, the internal vertical stress of coal pillars is generally at a low level, and the peak stress is lower than the initial stress. This is because when the width of the coal pillar is less than 5 meters, the damage of the coal pillar is extremely serious, and the ability to bear the weight of the overlying rock and provide support is almost lost. When the width of the coal pillar is 5-8 meters, the vertical stress inside the coal pillar is higher than the initial stress, and the stress concentration degree is not obvious, and the stress concentration coefficient is between 1.05 and 1.49. When the width of the coal pillar is 8-12 meters, the peak abutment pressure appears in the coal pillar area of the section, the abutment pressure inside the coal pillar is large, and the stress concentration coefficient reaches 1.73-2.68. Correspondingly, the surrounding rock is also under large pressure, and it is easy to accumulate a large amount of elastic energy, which brings hidden trouble to the stability of the roadway. When the stress peak value is too high, it will cause many adverse effects on the prevention and control of impact disaster and roadway support work. Considering the above factors, from the perspective of vertical stress distribution in the coal pillar, the width of the coal pillar is suitable to be arranged in the range of 5-8 meters.

(2) According to the comparative analysis of surrounding rock displacement with different coal pillar widths between strip lanes, when the coal pillar spacing between strip lanes is 4m, the deformation of the roof and floor of strip lanes is 448mm, which is relatively large, which is not conducive to the effective anchoring effect of the anchor rod, resulting in serious deformation of the roadway, which is not conducive to the maintenance of the roadway. When the width of the coal pillar is between 4-12m, the deformation of the top and bottom shows a trend of decreasing first and then increasing. The deformation is the smallest at 5m, and the deformation of the top and bottom is 252mm. The change trend of the deformation is relatively gentle between 5-12m, and the deformation has been kept in a small range.

(3) The coal pillar between strip lanes should ensure the stability of the roadway and reduce the waste of resources, so as to achieve the purpose of safe and efficient mining. Therefore, the reasonable width of coal pillar in the working face under strip lanes is determined to be 5m in consideration of the impact disaster prevention and control, roadway support and the maximum subsidence of the surface after the strip lanes are driven.

(4) Through the industrial test in the 7810 transport lane and return air lane of Wuyang Coal Mine, it can be found that

when the optimal support scheme is adopted, the deformation of the surrounding rock of the roadway is significantly reduced, and the stability of the surrounding rock is significantly improved, and remarkable results are achieved.

(5) According to the monitoring data of roadway surface displacement, the maximum displacement of the two sides of roadway 7810 is about 503mm and the maximum is 389mm. On the 18d after roadway excavation, the deformation rate of the surrounding rock on the roof and floor and the surrounding rock on the two sides of roadway is the maximum, and the average deformation rate is 11.8mm/d and 13.6mm/d, respectively. After 35d, the deformation speed of the roof and floor of the roadway and the two sides gradually tended to be stable, and the deformation of the surrounding rock of the roadway basically did not change significantly.

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