processes

Article

Surrounding Rock Control Technology of Thick Hard Roof and Hard Coal Seam Roadway under Tectonic Stress

Zhongzong Cao [1,2], Honglin Liu [1,*], Chengfang Shan [3], Hongzhi Wang [1] and Haitong Kang [3]

1 College of Geology and Mining Engineering, Xinjiang University, Urumqi 830046, China; caozhongzong@stu.xju.edu.cn (Z.C.); wanghongzhi@xju.edu.cn (H.W.)
2 Department of Mechanical and Electrical Engineering, Aksu Vocational and Technical College, Aksu 843000, China
3 Kuqa Yushuling Coal Mine Co., Ltd., Aksu 843000, China; wenxiangcao@stu.xju.edu.cn (C.S.); czz880928@163.com (H.K.)
* Correspondence: liuhonglin@xju.edu.cn

Abstract: In the process of roadway excavation in thick and hard coal seams with a hard roof, the instantaneous release of a large amount of elastic energy accumulated in coal and rock mass causes disasters. Especially under the action of tectonic stress, dynamic disasters of roadway-surrounding rock are extremely strong. Therefore, this paper takes the 110,505 roadway of the Yushuling Coal Mine as the engineering background. Aiming at the serious deformation of roadway-surrounding rock and the problem of strong mine pressure, the deformation mechanism of roadway-surrounding rock is studied by means of theoretical analysis, indoor experimentation, numerical simulation and field testing, and the surrounding rock control technology is proposed. Firstly, the results show that the stress field type of the Yushuling Coal Mine is a σ_{Hv} type, the azimuth angle of the maximum horizontal principal stress is concentrated in 110.30°~114.12°, the dip angle is −33.04°~−3.43°, and the maximum horizontal principal stress is 1.94~2.76 times of the minimum horizontal principal stress. Secondly, the brittleness index of No. 5 is 0.62; the failure energy release of the surrounding rock compressive energy floor rock sample is up to 150,000 mv * ms. The more the cumulative number of rock samples, the greater the strength, and the more severe the damage. Thirdly, with the increase in tectonic stress, the stress of roadway-surrounding rock is asymmetrically distributed, and the plastic zone develops along the tendency. The maximum range of the plastic zone expands from 4.18 m to 10.19 m. Lastly, according to the deformation characteristics of roadway-surrounding rock, left side > roof > right side > floor, the surrounding rock control technology of 'asymmetric anchor net cable support + borehole pressure relief' is proposed, which realizes the effective control of roadway-surrounding rock deformation.

Keywords: stress relief method; thick and hard coal seam; acoustic emission technology; asymmetric anchor net cable support; drilling pressure relief

Citation: Cao, Z.; Liu, H.; Shan, C.; Wang, H.; Kang, H. Surrounding Rock Control Technology of Thick Hard Roof and Hard Coal Seam Roadway under Tectonic Stress. *Processes* **2024**, *12*, 1973. https://doi.org/10.3390/pr12091973

Academic Editors: Qingbang Meng, Xinglong Zhao, Yuanyuan Pu and Sitao Zhu

Received: 13 July 2024
Revised: 23 August 2024
Accepted: 10 September 2024
Published: 13 September 2024

1. Introduction

Coal will still be the most important energy resource in China for a long time in the future, and the development of coal is directly related to the development of the national economy [1]. Xinjiang is an important base for national energy security [2]. However, the tectonic stress of strata in the Xinjiang mining area is large, resulting in serious deformation of roadway-surrounding rock in shallow crust and strong mine pressure [3,4], which seriously threaten the safe and efficient mining of mines.

The hollow inclusion stress relief method is an important means to test the distribution characteristics of tectonic stress [5–7]. Aiming at the problem that it is easy to induce strong mine pressure in the hard roof, Yang Peiju et al. [8] found that the thick hard roof located in the fracture zone will cause strong mine pressure. Goszcz and Feng Longfei et al. [9,10] think that the breaking and dislocation instability of the hard roof is the main cause of strong mine pressure. A large number of scholars [11,12] believe that strong mine pressure is related to the thickness of the hard roof and mining speed.

Domestic and foreign scholars have put forward control technology for different types of roadway-surrounding rock [13,14], including: support, pillar, concrete injection, masonry, and bolt support. Especially since the modern bolt support design concept of "three high and one low" has been put forward, roadway excavation technology and work efficiency have been significantly improved [15]; with the development of a 1 × 19 structure with a large diameter and high elongation steel strand anchor cable, combined with a bolt, anchor net and steel strip, a variety of combined support methods have been formed, which have effectively helped the control of roadway-surrounding rock [16]. However, the surrounding rock of the roadway is prone to strong mine pressure in the thick and hard coal seam of the hard roof, and the modification technology of the surrounding rock of the roadway cannot be promoted efficiently [17,18]. The pressure relief technology of the surrounding rock of the roadway can effectively make up for the shortcomings of the above control technology. The pressure relief technology of roadway is mainly through the use of hydraulic fracturing [19–21], blasting [22–24], drilling pressure relief [25,26], and other technical means to realize the effective pressure relief technology of roadway-surrounding rock. Academician Kang Hongpu's team [27,28] studied the static and dynamic load mechanical properties of high impact toughness bolts and metal mesh and determined the bolt-mesh support method for impact roadway. Tahmasebinia [29] carried out the study of cable bolt impact load. At present, the technical parameters of surrounding rock control in the thick hard roof and hard coal seam roadway under tectonic stress are mostly based on experience, and the level of surrounding rock control technology needs to be improved. Therefore, it is necessary to further optimize the support parameters and pressure relief hole parameters.

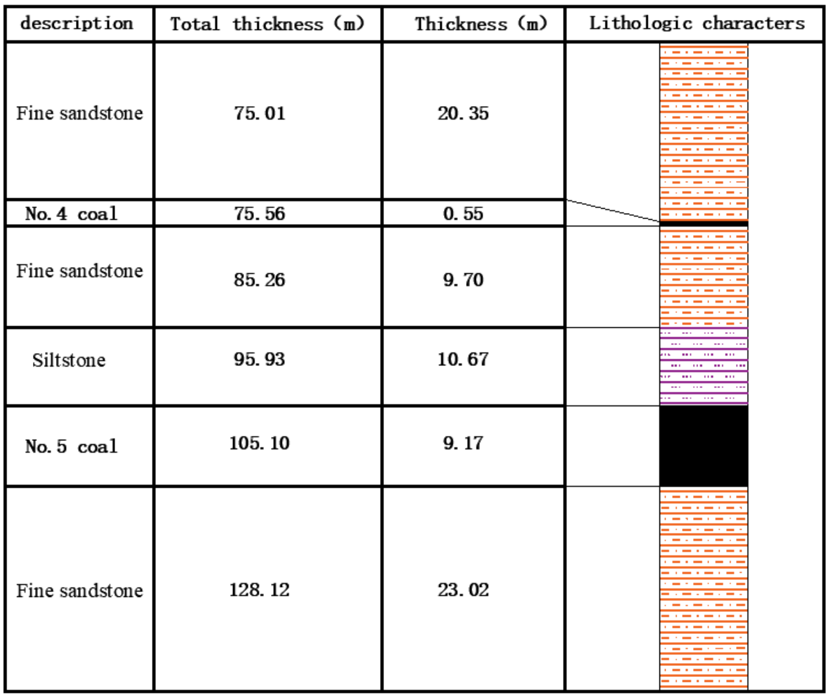
In summary, the Yushuling Coal Mine, which has the characteristics of a typical 'thick hard roof and hard coal seam' in Kubai Coalfield, is selected as the engineering background. On the basis of mastering the mechanical properties of stratum rock and soil and the size and direction of original rock stress, the FLAC^{3D} numerical model is established to study the stress distribution, plastic zone change, deformation, and failure characteristics of roadway-surrounding rock in the thick hard roof and hard coal seam under tectonic stress. Combined with the actual situation of the site, the control technology of roadway-surrounding rock is put forward. It provides a reference for the surrounding rock control technology of mine roadways with similar geological conditions in Kubai Coalfield.

2. Experimental Methods and Materials

2.1. Geological Conditions and Ground Stress Measurement

2.1.1. Geological Condition

The Yushuling Coal Mine has a large number of coal seams, large thickness, and slow dip angle. The minefield is located in the south wing of the Tesdrik anticline, which is a south-inclined monoclinic structure. The stratum strikes nearly east–west, with a tendency of 175° to 220°, and the dip angle is mostly between 9° and 13°, with an average of 10°. The Yushuling Coal Mine has the typical characteristics of geological conditions in the Xinjiang mining area. The strata are squeezed and mostly formed during the Yanshan orogenic movement. The coal seam roof and floor histograms are shown in Figure 1.



description	Total thickness (m)	Thickness (m)	Lithologic characters
Fine sandstone	75. 01	20. 35	
No. 4 coal	75. 56	0. 55	
Fine sandstone	85. 26	9. 70	
Siltstone	95. 93	10. 67	
No. 5 coal	105. 10	9. 17	
Fine sandstone	128. 12	23. 02	

Figure 1. No.5 Coal seam and roof and floor histogram.

2.1.2. In Situ Stress Measuring Point Arrangement

The measurement method adopts the hollow inclusion stress relief measurement method. According to the basic principle of the arrangement of the original rock stress measuring points, combined with the arrangement of the underground roadway in the Yushuling Coal Mine, three measuring points are selected to measure the original rock stress within the existing roadway exposure range. The first measuring point (YSL-1) is located at the + 1550 m transport stone gate, the second measuring point (YSL-2) is located at the lower monkey car in the main inclined shaft, and the third measuring point (YSL-3) is located at the No. 2 directional drilling field of mining area 12. In Figure 1, the main shaft is mainly used for transporting coal and pedestrians, the auxiliary shaft is mainly used for transporting materials, the transport stone gate is mainly used for pedestrians and transporting materials, and the water sump is used for storing water, as follows (Figure 2).

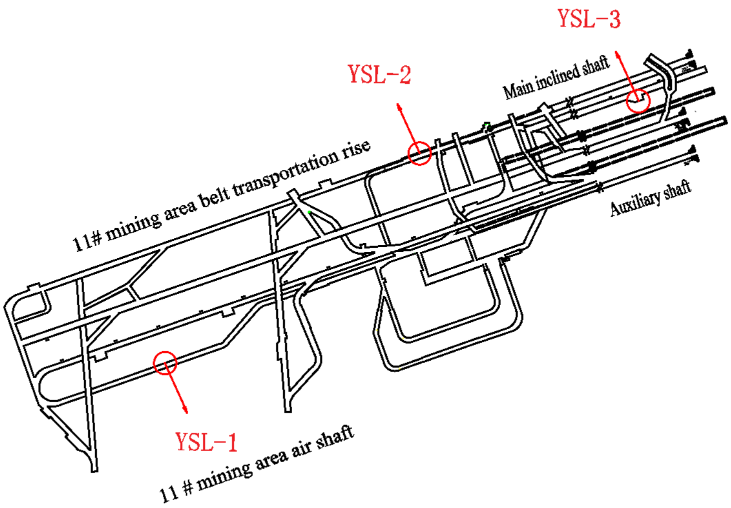


Figure 2. In-situ stress measuring point layout diagram of hollow core inclusion.

2.2. Mechanics and Acoustic Emission Experiments

The intrinsic mechanical characteristics of coal and rock formations, coupled with the failure mechanisms observed in rock samples, serve as fundamental parameters for the comprehensive analysis of mine pressure dynamics and the deformation patterns within mining roadways. These parameters are especially crucial when dealing with thick and hard coal seams that are capped with a hard roof. They are also vital for understanding the instability mechanisms of the surrounding rock structures and for developing effective control technologies for the stability of roadways. Furthermore, the role of acoustic emission (*AE*) experimental data in assessing the stability of the roadway's surrounding rock is complex and multi-dimensional. By employing *AE* monitoring technology, it is possible to conduct a thorough analysis and make accurate predictions regarding the stability and early warning signs of potential failures in the rock mass surrounding the roadway.

The coal and rock samples of roof, coal, and floor were prepared according to the International Society of Rock Mechanics (ISRM) test standard. The samples used in this test were three groups of roof, coal, and floor, and each group had three samples to ensure the stability of the test data. Figure 3 is the E45.605 mine rock mechanical property test system selected for this test equipment.

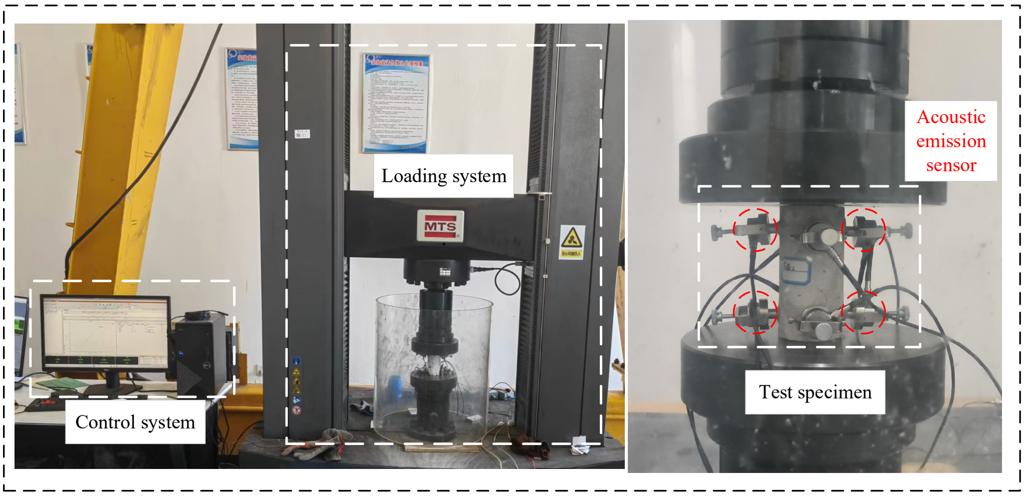


Figure 3. E45.605 mine rock mechanical properties test system.

3. Results and Discussion

3.1. In Situ Stress Distribution Characteristics

In Figure 4, the development roadway and preparation roadway are mainly arranged. The 110,505 belt conveyor roadway is mainly used for coal transportation, and the 110,505 rail conveyor roadway is mainly used for material transportation. The 110,505 working face mainly has a hydraulic support, shearer, scraper conveyor, and other equipment. According to the analytical data of the three measuring points, it can be seen that the azimuth angle of the maximum principal stress is concentrated in 110.30°~114.12°, the stress value is 6.34~8.78 MPa, and the inclination angle is −33.04°~−3.43°, indicating that the inclination angle of the maximum principal stress of the three measuring points is less than 35°, and the maximum principal stress can be regarded as horizontal stress. The azimuth angles of the intermediate principal stress are 14.66°, −37.99° and −70.84°, respectively, the stress values are 4.61~5.75 MPa, and the inclination angles are −58.58°, −78.94° and −56.91°, respectively, indicating that the inclination angles of the intermediate principal stress at the three measuring points are concentrated at about 50°~80°, and the intermediate principal stress can be regarded as vertical stress. The azimuth angle of the minimum principal stress is concentrated in 201.58°~205.00°, the stress value is 3.11~3.26 MPa, and the inclination angle is −31.19°~1.58°, indicating that the inclination angle of the minimum principal stress of the three measuring points is basically within 30°, and the minimum principal stress can be regarded as horizontal stress.

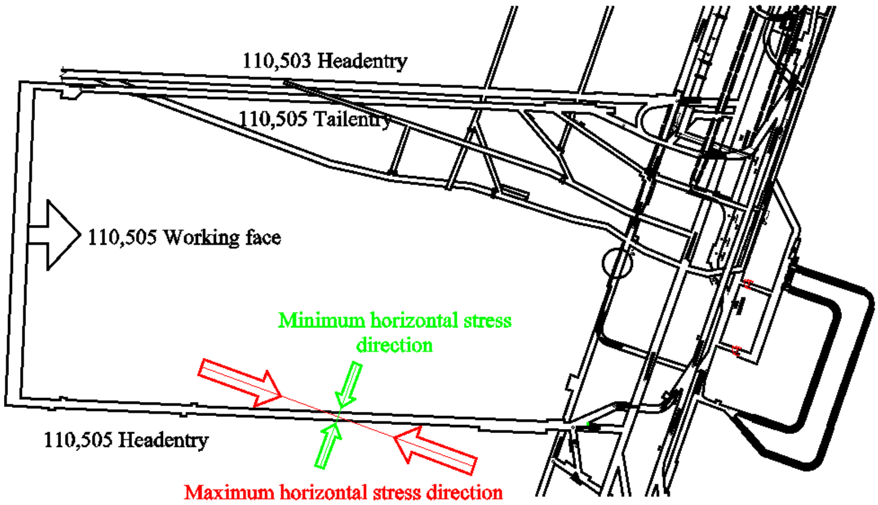


Figure 4. The orientation relationship between the direction of in situ stress and the 110,505 headentry.

The maximum horizontal stress (σ_H), the minimum horizontal stress (σ_h), the vertical stress (σ_v). The relationship between the three is shown in Table 1.

Table 1. Test results of maximum principal stress and minimum principal stress.

Name	σ_H (MPa)	σ_h (MPa)	σ_v (MPa)	σ_H/σ_v	σ_H/σ_h
YSL-1	8.78	5.75	3.20	1.53	2.74
YSL-2	8.60	4.77	3.11	1.80	2.76
YSL-3	6.34	4.61	3.26	1.38	1.94

3.2. Analysis of Stress–Strain Relationship and Deformation Failure Characteristics of Coal Block

Figure 5 is the stress–strain curve of a typical coal block, which is divided into the compaction stage, linear elastic stage, pre-peak fracture stage, and post-peak fracture development stage. In the compaction stage, the microcracks inside the rock begin to crack and expand, and especially when the stress value is 3 MPa the stress–strain curve rises rapidly. During this period, the work done by the press on the rock is mainly converted into dissipated energy. The energy dissipation is unidirectional and irreversible, and the internal microstructure cohesion is lost. In the linear elastic stage, the rock strain begins to enter the elastic stage from 0.25%, and the rock mass begins to accumulate a large amount of elastic energy. The microcracks inside the rock mass further spread; in the pre-peak fracture stage, the stress enters the up-and-down slowly rising stage, and the macroscopic crack expands and penetrates. In the stage of post-peak fracture development, the rock specimen is suddenly destroyed, and the compressive capacity of the rock specimen decreases sharply, and the internal elastic energy is released rapidly.

According to the characteristics of energy accumulation and release, the stress–strain curve is divided into three regions, dissipation energy, elastic strain energy, and residual elastic energy. Assuming that the total input energy generated by the external force acting on the rock sample is W, there is the following relationship according to the law of conservation of energy [30]:

$$W = E_e + E_d \tag{1}$$

$$R = E/W \tag{2}$$

where W is the total energy, R is the brittleness index of rock, E_e is the elastic energy stored in the rock, and the energy dissipated by the rock during the E_d loading process is mainly due to the internal damage and plastic deformation of the rock, and E is the recoverable energy when the rock is destroyed. According to the calculation and analysis, the dissipation energy of rock in the region ① is 2.78 J, the elastic energy of rock in the region ② is 9.89 J, the residual elastic energy of rock in the region ③ is 3.18 J, and the total energy input by external force is 15.85 J. Considering that the recoverable energy *E* during rock failure is equivalent to the elastic energy Ee stored in the rock, *R* is approximately equal to *E*/*W*, that is, the rock brittleness index is 0.62.

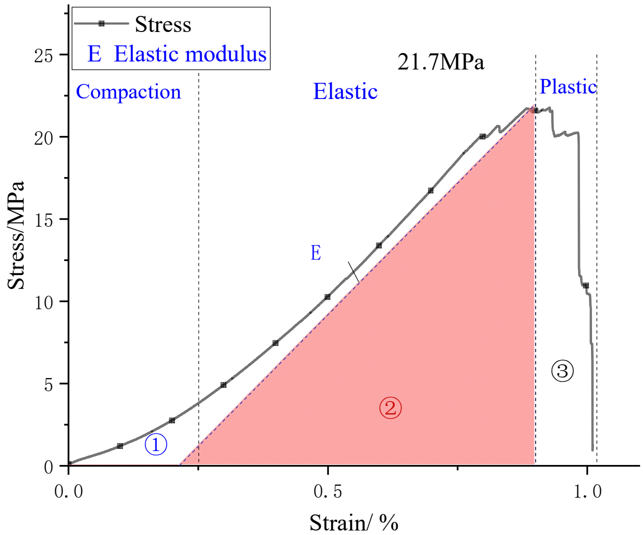


Figure 5. No. 5 Coal sample stress–strain curve.

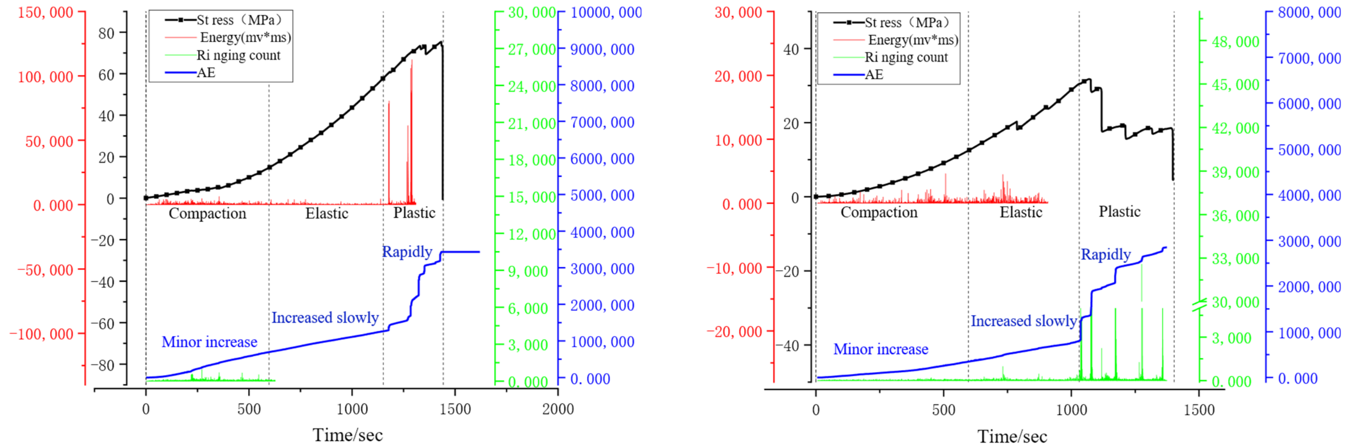
3.3. Analysis of Acoustic Emission Characteristics

Figure 6 is a typical acoustic emission characteristic curve of the bottom plate and the roof. The loading rate of the bottom plate rock sample is 0.02 mm/min. With the continuous pressure, the rock undergoes three stages of compaction-elasticity-yield, the cumulative acoustic emission count curve, and the stress–strain curve. There is a clear correspondence. The ringing count of rock specimens increases from a small increase to a slow increase, and finally increases suddenly in the yield stage, and the peak stress reaches 75 MPa. In the vicinity of the rock sample failure, the peak of the ringing count reached 12,000, and the cumulative *AE* events were about 3,430,000, but the cumulative ringing count stopped increasing after the stress peak. The acoustic emission energy is in a quiet period in the compaction stage and the elastic stage, but the energy increases sharply in the yield stage and finally reaches 150,000 mv * ms in the post-peak failure stage.

The axial stress of the roof increases gradually at the loading rate of 0.02 mm/min, and it also experiences three stages. However, the whole process time of the roof rock sample is slightly smaller, and the proportion of small acoustic emission events of the roof rock sample increases, and the rock is dominated by a small-scale fracture. The peak of ringing count reached 32,586, and the cumulative event of AE was about 2,830,000, and the maximum energy at the failure of rock sample reached 26,754 mv * ms.

Through the comparison of data from (a) and (b), it is found that the cumulative number of ringing in the floor is 600,000 more than that in the roof, and the greater the uniaxial compressive strength of the rock the more the cumulative ringing count of acoustic emission. The stress reduction rate of the bottom plate rock sample after the peak value is the fastest, and the excess energy of the specimen is dissipated in the form of impact, and the damage is the most severe.

Based on the experimental study of coal and its roof and floor rock mass in the Yushuling Coal Mine, the physical and mechanical properties of representative rocks in the mining area, such as density, uniaxial tensile strength, uniaxial compressive strength, internal friction angle, cohesion and elastic modulus, are obtained. The main physical and mechanical parameters are as follows (Table 2):



(**a**) Surrounding rock of roadway floor (**b**) Roadway roof surrounding rock

Figure 6. Acoustic emission ringing count and energy evolution law of No. 5 coal roof and floor rock samples.

Table 2. Physical and mechanical parameters of No.5 coal and roof and floor rock samples.

Name of Rock Sample	Volumetric Weight	Tensile Strength	Compressive Strength	Angle of Internal Friction	Cohesion	Elastic Modulus	Poisson's Ratio
	N/m^3	MPa	MPa	°	MPa	GPa	/
No. 5 Coal seam	12,107.68	0.780	21.7	43	1.7546	2.2545	0.21–0.25
Roof	23,587.52	2.709	47.67	36	20.2701	7.274	0.30–0.33
Floor	23,883.05	4.472	60.03	46	9.7177	7.862	0.31–0.34

3.4. Numerical Model Calculation and Analysis

In order to study the stress distribution, plastic change, deformation, and failure characteristics of the surrounding rock of roadway in thick and hard coal seams with a hard roof under the influence of tectonic stress, based on the engineering geological conditions of the Yushuling Coal Mine, a continuous medium with a stratigraphic dip angle of 10° was established, and the deformation mechanism of surrounding rock of roadway in thick and hard coal seams with a hard roof under the action of tectonic stress was simulated and further analyzed (Figure 7).

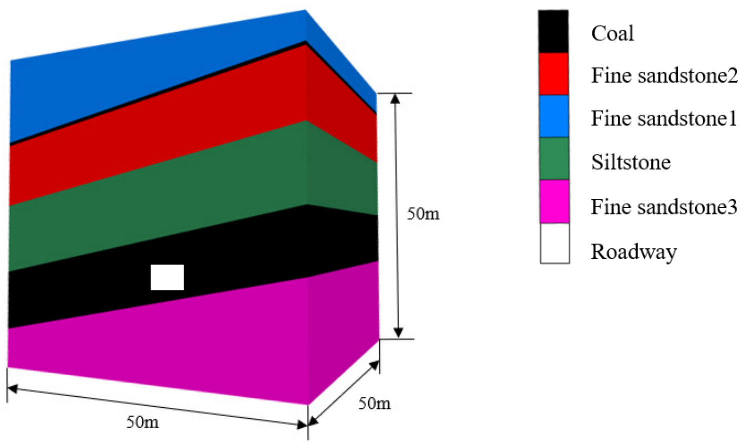


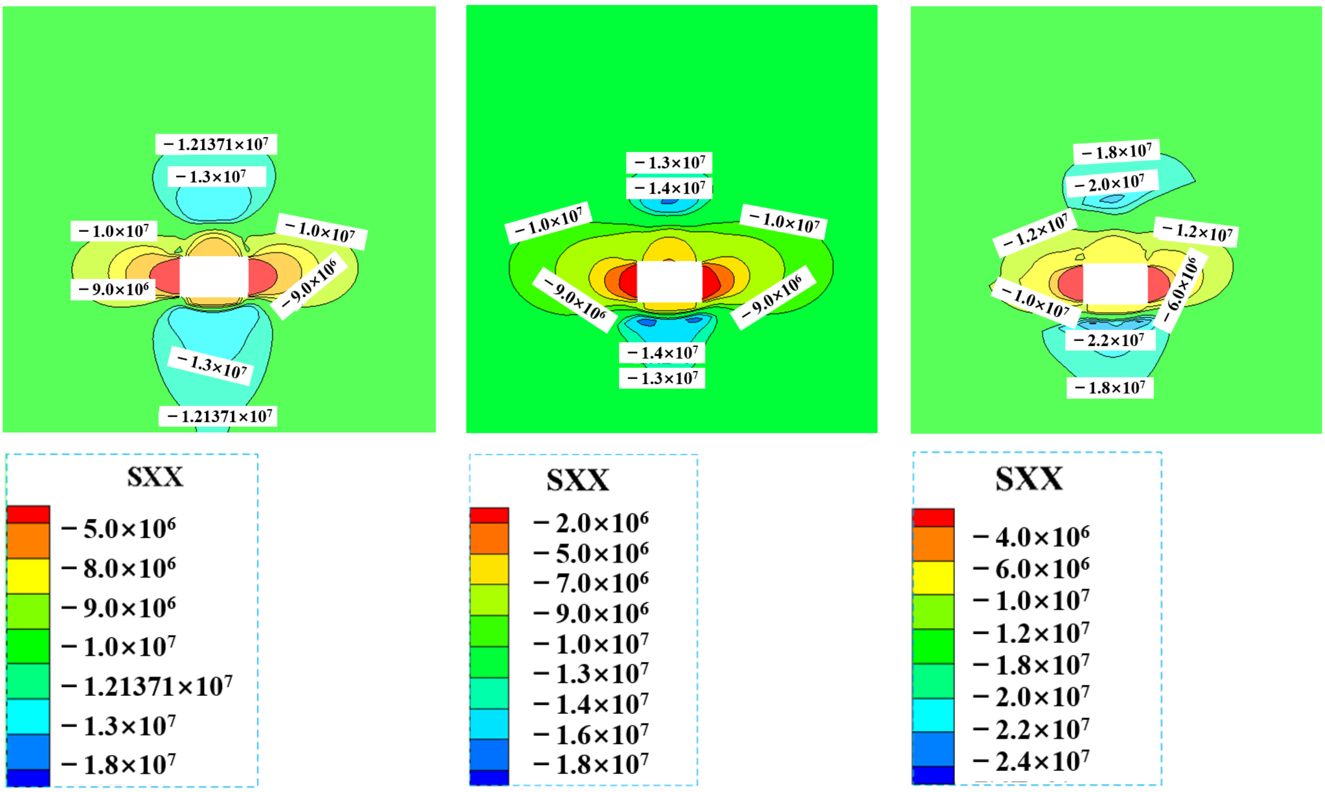
Figure 7. Construction of numerical calculation model of surrounding rock strata in 110,505 headentry roadway.

The model adopts the Mohr–Coulomb criterion, and the size adopts the length × width × height = 50 m × 50 m × 50 m. The model generates a total of 280,000 units. The roadway is arranged in the 5-coal seam under the model. The size of the roadway is based on the actual size of the site; the width is 5.8 m, the height is 3.5 m, the net section is 20.3 m^2, and the initial model diagram is 7 m^2. The model rock parameters are shown in Table 3.

Table 3. Lithology of surrounding rock and adjacent rock strata in 110,505 haulage roadway.

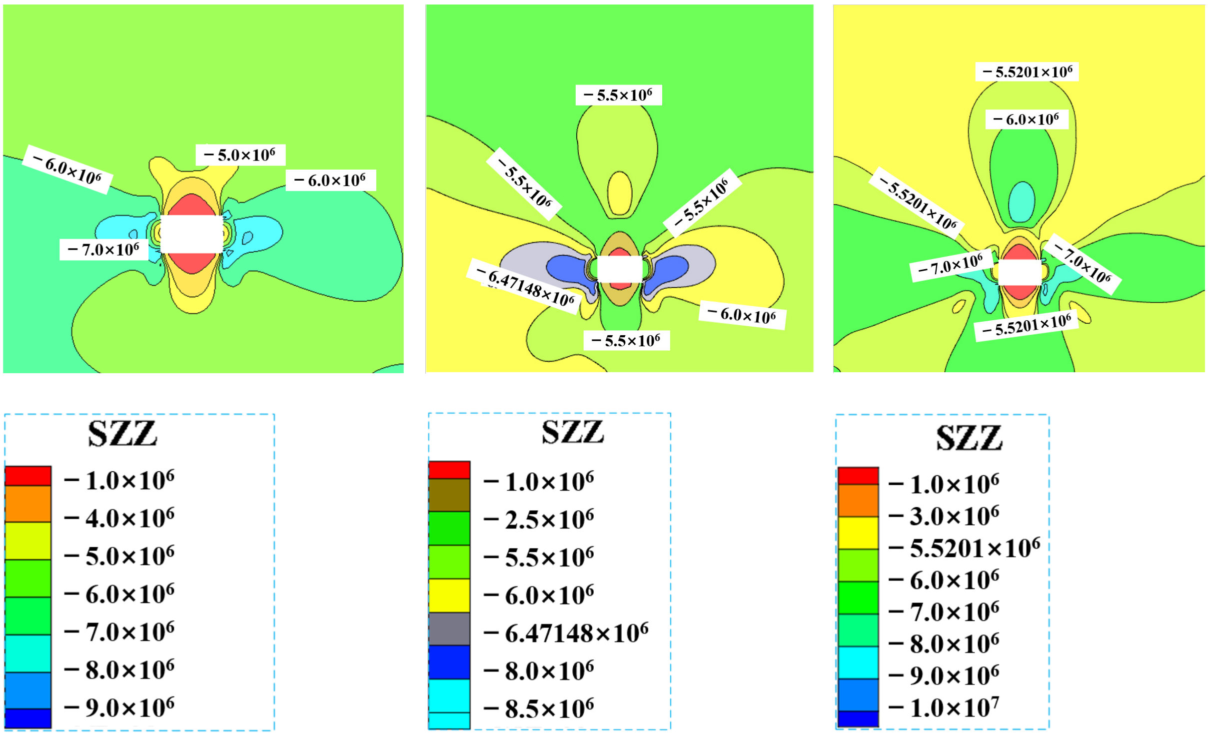
No.	Lithologic Characters	Thickness (m)	Density (kg/m^3)	Tensile Strength (MPa)	Bulk Modulus (MPa)	Shear Modulus (MPa)	Angle of Internal Friction (°)
1	Fine sandstone	20.35	2630	4.472	2643	1820	46
2	No. 4 coal	0.55	1350	0.780	2139	1204	43
3	Fine sandstone	9.70	2630	4.472	2643	1820	46
4	Siltstone	10.67	2660	2.709	3550	2345	36
5	No. 5 coal	9.17	1350	0.780	2139	1204	43
6	Fine sandstone	23.02	2630	4.472	2643	1820	46

In this paper, λ values are 1.0, 2.0, and 3.0, respectively. The working conditions of the construction roadway under different pressure measurement coefficients are simulated. The stress distribution characteristics and the plastic zone variation law of roadway-surrounding rock are studied and analyzed to study the influence mechanism of tectonic stress on roadway-surrounding rock deformation. The numerical simulation results are shown in Figures 8–10.



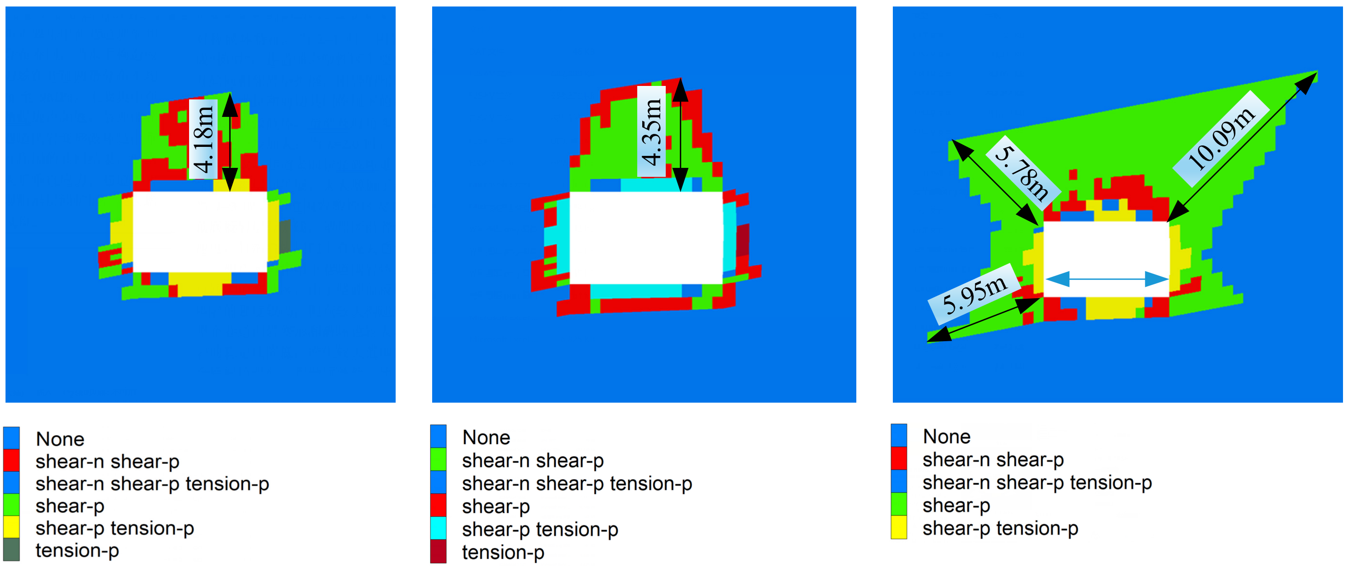
(**a**) λ = 1 (**b**) λ = 2 (**c**) λ = 3

Figure 8. Horizontal stress nephogram of surrounding rock of 110,505 headentry under different tectonic stress.



(**a**) $\lambda = 1$ (**b**) $\lambda = 2$ (**c**) $\lambda = 3$

Figure 9. Vertical stress nephogram of surrounding rock of 110,505 transportation roadway under different tectonic stress.



(**a**) $\lambda = 1$ (**b**) $\lambda = 2$ (**c**) $\lambda = 3$

Figure 10. Cloud diagrams of plastic zone of surrounding rock of the 110,505 transportation roadway under different tectonic stress.

From the analysis of Figures 8 and 9, it can be seen that with the excavation of a coal seam roadway, the stress of roadway-surrounding rock is redistributed and reaches a new dynamic equilibrium state. The overall distribution of horizontal stress is a unique variation 'butterfly' shape in inclined strata. The two sides are approximately symmetrically distributed. The depths of the roof and floor are quite different. The stress concentration ranges of the roof and floor are also different. The deep concentration stress of the bottom foot of the two sides of the roadway is the largest. As the λ value gradually increases, the maximum horizontal principal stress increases from 18 MPa to 24 MPa. The shallow parts of the two sides of the roadway are mainly subjected to tensile stress. After passing the tensile stress area, the compressive stress of the roadway increases first and then decreases with the direction away from the normal direction of the roadway until it stabilizes to the original rock stress of the roadway. The horizontal stress redistribution of the roadway is mainly concentrated on the top and bottom of the roadway, which is different from the distribution of the horizontal stress field. When the horizontal tectonic stress gradually increases, the vertical stress field is unevenly distributed on the two sides of the roadway, with a small increase from 9 MPa to 10 MPa, mainly concentrated on the left side of the roadway. It is one of the reasons for the frequent coal gun sound and serious joint breakage in the left side of the roadway. The deformation and failure of the surrounding rock of the roadway is the common result of the interaction between vertical stress and horizontal stress, but the horizontal stress failure effect is significantly greater than the vertical stress. The plastic zone of roadway-surrounding rock is large and shows expansion along the interlayer, which greatly increases the risk of roadway roof separation.

According to the analysis of Figure 10, it can be seen that with the increase of the lateral pressure coefficient λ, the plastic zone of the surrounding rock of the roadway increases greatly, indicating that the tectonic stress is very destructive to the surrounding rock of the roadway. According to the analysis of the plastic zone distribution map, it is found that the plastic zone develops to the top and bottom of the coal rock layer. The junction has certain limitations. The specific reason is that due to the difference in the physical and mechanical properties of the coal rock layer, it also restricts the development of the plastic zone of the surrounding rock of the roadway to a certain extent, forming an asymmetric failure feature in the plastic zone. When λ = 1, the plastic zone of surrounding rock is approximately 'ladder'. The plastic zone of roadway-surrounding rock is mainly concentrated in the roof, and the maximum value of the plastic zone is 4.18 m, which begins to expand to adjacent rock strata. The yield mode of surrounding rock is that the roof surface yields under the combined action of tension and shear. The tensile failure occurs on the surface of the roadway, and the shear yield occurs on the side and shoulder corner, and the failure is further increased. When λ = 2.0, the range of plastic zone is further increased, and the maximum area of roof separation is 4.35 m. The tensile failure of the roadway surface is further increased, and it shows the expansion along the interlayer, which greatly increases the risk of separation. When λ = 3, the development of the plastic zone of the surrounding rock of the roadway is limited to the boundary between the roof and floor of the coal and rock, and the plastic zone develops along the coal seam. The speed increases, and there is a large upward trend along the coal seam, with a maximum value of 10.09 m.

Under the action of tectonic stress, the surrounding rock mass of roadway usually develops a large plastic zone after excavation. Even under the geological condition of a thick and hard coal seam with a hard roof, the deformation and failure of the surrounding rock of the whole roadway are also large, which lead to the decrease of the stability of the surrounding rock of the roadway, producing a large range of plastic zone. Combined with the engineering geological conditions, in situ stress test, roadway-surrounding rock characteristics, and numerical simulation analysis of the Yushuling Coal Mine, it is concluded that the asymmetric deformation mechanism of roadway-surrounding rock in thick and hard coal seams with a hard roof in the Yushuling Coal Mine under the action of tectonic stress is an asymmetric mechanism of stress distribution as well as an interlayer expansion deformation mechanism and tectonic stress shear deformation mechanism.

3.5. Control Technology of Roadway-Surrounding Rock

Based on the test results of the engineering background of the 110,505 haulage gateway in the Yushuling Coal Mine, this paper puts forward the control technology of 'asymmetric anchor net cable support + borehole pressure relief of roadway-surrounding rock according to the deformation characteristics of roadway-surrounding rock, and analyzes the support effect of the field surrounding rock deformation monitoring data, verifies the rationality of support, and provides reference for mine support under similar geological conditions.

3.5.1. Project Profile

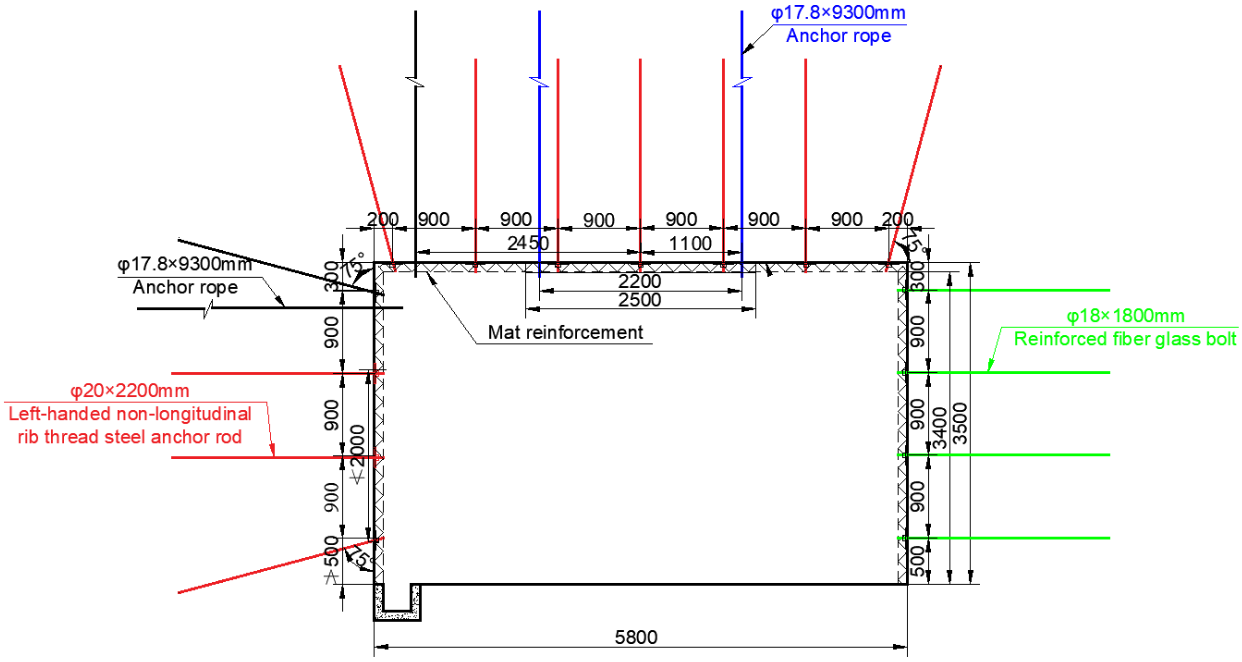
The 110,505 haulage gateway is arranged in the No. 5 coal seam. The slope of the roadway is arranged along the coal seam to the top coal, and the surrounding rock of the roadway often appears coal-blasting and rib-spalling. Symmetrical support is adopted in the construction process of the roadway, but the support strength of the left side of the roadway is still insufficient, which leads to the damage of the left side, and then this affects the stability of the whole roadway. Through the analysis of underground field observation, in situ stress test, theoretical analysis and numerical simulation results aiming at the special physical and mechanical properties, deformation characteristics, and insufficient support strength of the roadway-surrounding rock under the action of tectonic stress in the Yushuling Coal Mine, the single symmetrical support structure cannot attain the ideal support effect.

3.5.2. Control Technique

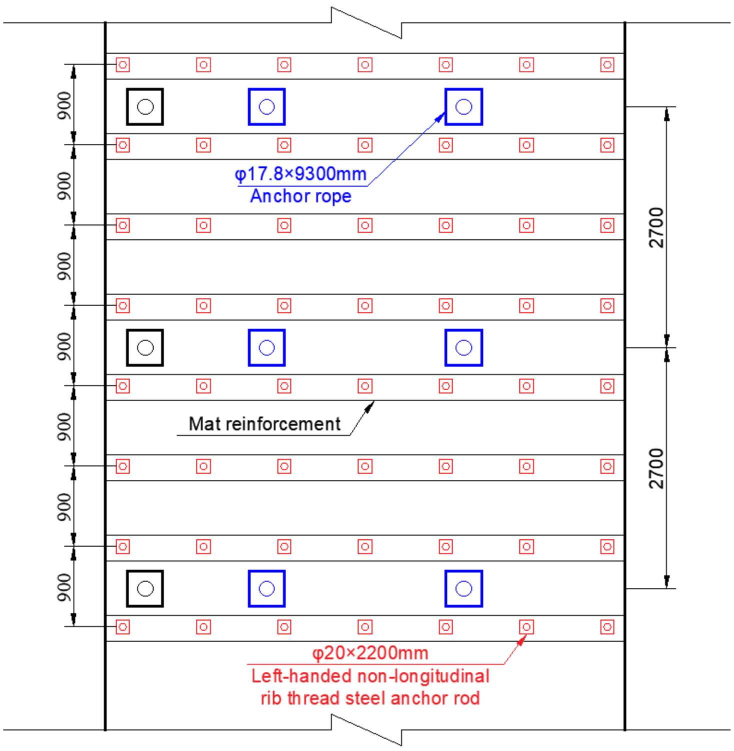
The asymmetric support technology of roadway-surrounding rock, such as 'deep anchor-shallow injection' and 'high supplement and low unloading', has a significant effect on the application of roadway-surrounding rock under the condition of non-uniform load [31–35]. An appropriate bolt-mesh-cable combined support technology can inhibit the damage of the surrounding rock and make the surrounding rock stable [36–39]. The drilling construction in the advance area of working face has achieved good results in the control of roadway-surrounding rock [40]. The grouting weakening filling in the roof separation area of the roadway further weakens the impact of a hard overburdened rockburst disaster [41]. The increase of horizontal stress will cause damage to the two sides of the coal body, and the larger the angle between the horizontal stress and the axial direction of the roadway, the greater the displacement deformation of the surrounding rock of the roadway [42,43]. For the roadway support and pressure relief mode under the action of tectonic stress, referring to the existing successful cases of adjacent mines, the roadway-surrounding rock control technology of 'asymmetric anchor net cable support + borehole pressure relief' is adopted to solve the problems of asymmetric deformation and strong mine pressure appearance of the roadway. However, the problem of a small amount of floor heave in the roadway is solved by undercover pressure relief. In summary, on the basis of the original support scheme of the 110,505 belt conveyor roadway in the Yushuling Coal Mine, the anchor cable and anchor rod (marked as black) are added as shown in Figure 11 below. The support and borehole pressure relief parameters are shown in Table 4.

Table 4. The technical parameters of surrounding rock control of the 110,505 roadway combined with an 'asymmetric anchor cable + pressure relief hole' are as follows.

Control Technique	Parameter
Roof bolting	The top and left side bolts adopt Φ20 mm left-handed non-longitudinal rib screw thread steel bolt. The length of the rod body is 2200 mm, and its supporting components include a wear-reducing washer and a damping nut. The Φ18 × 1800 mm FRP anchor rod is used for the right-side anchor rod. The length of the anchor rod is 1800 mm and the row spacing is 900 mm.
Roof anchor rope	The anchor cable adopts a Φ17.8 mm prestressed steel strand with a length of 9300 mm. The prestress of the anchor cable is not less than 30 MPa, and the row spacing is 2700 mm.
Pressure tap	The boreholes are arranged in the front of the roadway, with a row spacing of 6m and a spacing of 800 mm. There are 6 boreholes in the front of the roadway, with an aperture of 75mm and a hole depth of 10m.



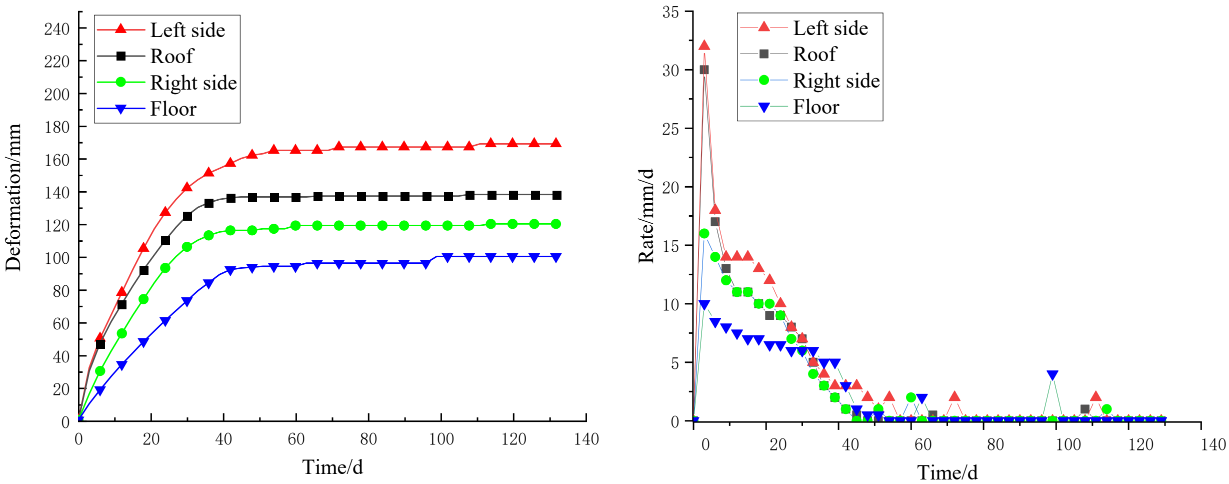
(**a**) Mine working vertical section



(**b**) Roadway top view

Figure 11. Surrounding rock control technology of the 110,505 roadway combined with an 'asymmetric anchor net cable + pressure relief hole'.

The evaluation methods of surface displacement and displacement deformation rate of surrounding rock are important indexes to evaluate the stability of surrounding rock of roadway. In order to deeply understand the supporting effect of the supporting scheme and the activity law of roadway-surrounding rock affected by excavation under tectonic stress, displacement stations are arranged in roadway, and reliable measuring points are selected for data research. The analysis of displacement data of measuring points is shown in Figure 12.



(a) Surface displacement (b) Displacement deformation rate

Figure 12. Deformation monitoring of surrounding rock of 110,505 roadway under tectonic stress.

After the excavation of the roadway, the displacement of the roadway surface first increases sharply, then increases slowly, and finally enters the stage of basically unchanged displacement about 40 days after the excavation of the roadway, indicating that the surrounding rock of the roadway has been stable. According to the observation results, the surface displacement within two weeks of roadway excavation accounts for nearly 75% of the total displacement. In the early stage of roadway excavation, the displacement of the left side of the roadway is the largest, and the roof subsidence is larger than the deformation of the left side, the floor heave, and the displacement of the right side of the roadway. The main reason is that the stress of the surrounding rock of the roadway will be redistributed after the excavation of the roadway. After excavation, the disturbance to the surrounding rock of the roadway is the largest, and the deformation is 10 mm/d~33 mm/d every day according to the monitoring results. When the stress is balanced, the surface displacement velocity of each part of the roadway is reduced. When observed on the 10th day, the displacement velocity is reduced to 7 mm/d–15 mm/d. When observed on the 30th day, the average displacement is 5 mm/d. It shows that the deformation of the surrounding rock of the roadway began to stabilize, and the surface displacement velocity has decreased greatly; especially on the 40th day, the surrounding rock of the roadway has been reduced. Such as on-site support effect comparison is shown in Figure 13.

(a) Surface displacement (b) Surface displacement

Figure 13. Comparison of supporting effect of 110,505 roadway before and after taking measures.

In summary, the roadway-surrounding rock control technology of an 'asymmetric anchor cable + pressure relief hole' has achieved a good control effect of roadway-surrounding rock.

4. Conclusions

(1) The stress field type of the Yushuling Coal Mine is the σ_{Hv} type, which involves mainly horizontal stress ($\sigma_H > \sigma_V > \sigma_h$). The maximum horizontal principal stress is 1.38~1.80 times that of the vertical principal stress, and the maximum horizontal principal stress is 1.94~2.76 times that of the minimum horizontal principal stress.
(2) The brittleness index of No. 5 is 0.62; the failure energy release of the surrounding rock compressive energy floor rock sample is up to 150,000 mv * ms.
(3) The greater the cumulative number of acoustic emission ringing, the greater the strength of the rock sample, the more severe the damage.
(4) Under the influence of tectonic stress, the deformation of the surrounding rock of the left side of the roadway is the largest, and the deformation of the left side is greater than that of the right side. The range of plastic zone near the roof and floor of roadway increases with the increase of tectonic stress, and the maximum range of the plastic zone is 10.09 m, which is mainly developed along the direction of coal seam.
(5) The surrounding rock control technology of an 'asymmetric anchor net cable + pressure relief hole' roadway can play an important control role in the deformation of surrounding rock of thick hard roof and hard coal seam roadways under the action of tectonic stress.

Author Contributions: Conceptualization, Z.C. and H.L.; methodology, C.S.; software, Z.C.; validation, Z.C., H.W. and H.K.; formal analysis, Z.C.; investigation, H.K.; resources, C.S.; data curation, Z.C.; writing—original draft preparation, Z.C.; writing—review and editing, H.L.; visualization, Z.C.; supervision, H.L.; project administration, H.W.; funding acquisition, H.L. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the Xinjiang Uygur Autonomous Region Special Program for Key R&D Tasks (2022B01034, 2022B01051, 2023B01010), Natural Science Foundation of Xinjiang Uyghur Autonomous Region (2022D01E31), Xinjiang Uygur Autonomous Region "Tianshan Talent Training" Program (2022TSYCCX0037, 2023TSYCJC0009), and Aksu Area Science and Technology Research Project (202305). The conception and launch of this work are also supported by the Rock Mechanics Laboratory (RML) of Xinjiang University.

Data Availability Statement: The research data used to support the findings of this study are currently under embargo while the research findings are commercialized. Requests for data, 12 months after publication of this article, will be considered by the corresponding author.

Conflicts of Interest: Authors Chengfang Shan and Haitong Kang were employed by the company Kuqa Yushuling Coal Mine Co., Ltd. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

References

1. Xie, H.P.; Wu, L.X.; Zheng, D.Z. Forecast of China's energy consumption and coal demand in 2025. *Coal J.* **2019**, *44*, 1949–1960.
2. Liu, H.L. Study on Water-Preserved Mining Mechanism and Classification of Extra-THICK coal Seam in Yili Weakly Cemented Strata. Ph.D. Thesis, China University of Mining and Technology, Xuzhou, China, 2020.
3. Wang, T.; Xia, Y.C.; Wei, B. Structural styles of Jurassic coalfields in Xinjiang and their coal-controlling effects. *Coal J.* **2017**, *42*, 436–443.
4. Kang, H.P. Analysis of stress field types and interaction in underground coal mine. *Coal J.* **2008**, *33*, 1329–1335.
5. Kang, H.P.; Zhang, X.; Si, L.P. In-situ stress measurements and stress distribution characteristics in underground coal mines in China. *Eng. Geol.* **2010**, *116*, 333–345. [CrossRef]
6. Kang, H.P.; Yi, B.D.; Gao, F.Q. Chinese Coal Mine In-situ Stress Database and In-situ Stress Distribution Law. *Coal J.* **2019**, *44*, 23–33.
7. Cai, M.F.; Guo, Q.F.; Li, Y. In-situ stress measurement and its application in Pingdingshan No.10 Coal Mine. *J. Beijing Univ. Sci. Technol.* **2013**, *35*, 1399–1406.

8. Yang, P.J.; He, Y.; Guo, W.B. Disaster-causing mechanism and prevention and control measures of extremely thick and hard magmatic rock overlying the stope. *Coal J.* **2013**, *38*, 2106–2112.
9. Goszcz, A. Roof rock bursts and ways to combat them. *Arch. Gor.* **1991**, *36*, 239–261.
10. Cao, J.R.; Dou, L.M.; Zhu, G.G. Mechanisms of Rock Burst in Horizontal Section Mining of a Steeply Inclined Extra-Thick Coal Seam and Prevention Technology. *Energines* **2020**, *13*, 6043. [CrossRef]
11. Cao, S.G.; Jiang, H.J.; Wang, F.H. Numerical simulation study on the fracture of overlying hard rock strata in stope. *J. Min. Saf. Eng.* **2013**, *30*, 205–210.
12. Feng, L.F.; Dou, L.M.; Wang, X.D. Influence mechanism of mining speed on energy released by hard roof movement. *Coal J.* **2019**, *44*, 3329–3339.
13. Liu, H.L.; Chen, X.; Wang, H.Z. Investigation on Surrounding Rock Stability Control Technology of High Stress Roadway in Steeply Dipping Coal Seam. *Adv. Civ. Eng.* **2021**, *2021*, 5269716. [CrossRef]
14. Kang, H.P. 60 years of development and prospect of bolt support technology in coal mine roadway in China. *J. China Univ. Min. Technol.* **2016**, *45*, 1071–1081.
15. Kang, H.P.; Wang, J.H. *Bolt Support Theory and Complete Technology of Coal Roadway*; Coal Industry Press: Beijing, China, 2007; pp. 112–121.
16. Kang, H.P.; Lin, J.; Wu, Y.Z. Full-section high pre-stressed strong anchor cable support technology and its application in dynamic pressure roadway. *Coal J.* **2009**, *34*, 1153–1159.
17. Zhang, Y.C. *Grouting Technology*; Coal Industry Press: Beijing, China, 2012; pp. 1–10.
18. Zhang, N. *Theory and Practice of Lag Grouting Surrounding Rock Control in Roadway*; China University of Mining and Technology Press: Beijing, China, 2004; pp. 14–57.
19. Gao, H.; Zhao, W.S.; Chen, W.Z. Continuous three-dimensional stress monitoring in roof of coal mines for investigating the rockburst control effect with hydraulic fracturing. *Environ. Earth Sci.* **2022**, *81*, 433. [CrossRef]
20. Gu, S.T.; Chen, C.P.; Jiang, B.Y. Study on the Pressure Relief Mechanism and Engineering Application of Segmented Enlarged-Diameter Boreholes. *Sustainability* **2021**, *14*, 5234. [CrossRef]
21. Xu, D.; Deng, G.Z.; Liu, H. Energy dissipation mechanism and application of zonal failure of fractured coal rock. *J. Min. Saf. Eng.* **2021**, *38*, 410–418.
22. Lan, T.W.; Fan, C.J.; Han, J. Controlling Mechanism of Rock Burst by CO_2 Fracturing Blasting Based on Rock Burst System. *Shock Vib.* **2020**, *2020*, 8876905. [CrossRef]
23. Yan, P.; Zhao, Z.G.; Lu, W.B. Mitigation of rock burst events by blasting techniques during deep-tunnel excavation. *Eng. Geol.* **2015**, *188*, 126–136. [CrossRef]
24. Zhou, X.X.; Ouyang, Z.H.; Zhou, H.H. An Approach to Dynamic Disaster Prevention in Strong Rock Burst Coal Seam under Multi-Aquifers: A Case Study of Tingnan Coal Mine. *Energies* **2021**, *14*, 7287. [CrossRef]
25. Zhao, T.B.; Guo, W.Y.; Yu, F.H. Numerical Investigation of Influences of Drilling Arrangements on the Mechanical Behavior and Energy Evolution of Coal Models. *Adv. Civ. Eng.* **2018**, *2018*, 3817397. [CrossRef]
26. Gong, F.Q.; He, Z.C.; Jiang, Q. Internal Mechanism of Reducing Rockburst Proneness of Rock Under High Stress by Real-Time Drilling Pressure Relief. *Rock Mech. Rock Eng.* **2022**, *55*, 5063–5081. [CrossRef]
27. Kang, H.P.; Wu, Y.Z.; He, J. Research and practice of bolt support in deep rock burst roadway. *Coal J.* **2015**, *40*, 2225–2233.
28. Fu, Y.K.; Sun, Z.Y.; Ju, W.J. Experimental study on static and dynamic mechanical properties of bolt support metal mesh in rock burst roadway. *Coal J.* **2019**, *44*, 2020–2029.
29. Tahmasebinia, F.; Yang, A.; Feghali, P.; Skrzypkowski, K. A Numerical Investigation to Calculate Ultimate Limit State Capacity of Cable Bolts Subjected to Impact Loading. *Appl. Sci.* **2023**, *13*, 15. [CrossRef]
30. Zhou, H.; Meng, F.Q. Quantitative evaluation method of rock brittleness characteristics based on stress-strain curve. *J. Rock Mech. Eng.* **2014**, *33*, 1114–1122.
31. Wang, W.W. Research and Application of Deep Anchor-Shallow Injection Asymmetric Enhanced Control Technology. *Coal Technol.* **2024**, *43*, 86–88.
32. Zhong, H.S. Study on the influence of coal-rock dip angle on asymmetric deformation and failure of gob-side entry retaining. *Min. Res. Dev.* **2024**, *44*, 74–80.
33. Gao, T.Y. Asymmetric deformation characteristics and control of mining roadway. *Metall. Mater.* **2024**, *44*, 52–54.
34. Scholz, C.H. Microfracturing and the inelastic deformation of rock in compression. *J. Geophys. Res.* **1968**, *73*, 1417–1432. [CrossRef]
35. Eberhardt, E.; Stead, D.; Stimpson, B. ldentifying crack initiation and propagation thresholds in britle rock. *Can. Geotech. J.* **1998**, *35*, 222–233. [CrossRef]
36. Zhang, Y.; Zhuge, C.H. Application research on asymmetric support technology of narrow coal pillar gob-side entry driving in deep mine. *Min. Saf. Environ. Prot.* **2021**, *48*, 64–68+74.
37. Zhen, W.Y.; Liu, H.L.; Chi, M.B. Investigation into the Influence of Stress Conditions on the Permeability Characteristics of Weakly Cemented Sandstone. *Appl. Sci.* **2023**, *13*, 12105. [CrossRef]
38. Zhou, G.; Wang, P.J.; Zou, C.L. Research on asymmetric support of gob-side entry driving in complex tectonic stress mining area. *J. Min. Saf. Eng.* **2014**, *31*, 901–906.
39. Li, G.D.; Liu, H.L.; Wang, H.Z. Research on reasonable layout and surrounding rock control technology of mining roadway in extremely close lower coal seam. *Coal Eng.* **2021**, *53*, 42–47.

40. Gao, Y.B. Research on Key Problems of 110 Construction Method in Thick Coal Seam of Ningtiaota Coal Mine. Ph.D. Thesis, China University of Mining and Technology, Beijing, China, 2018.
41. Lai, W.A. Research on The Mechanism of Weakening Rock Burst by Gangue Fluidization Grouting Filling in the Abscission Zone of Hongqinghe Coal Mine. Master's Thesis, China University of Mining and Technology, Beijing, China, 2023.
42. Xu, Y.C. Deformation characteristics of roadway surrounding rock considering rotation angle of horizontal in-situ stress field. *Jinkong Sci. Technol.* **2023**, *6*, 36–38+43.
43. Guo, L.; Huang, C.L.; Zhang, C.L. Destruction mechanism and support scheme of surrounding rock of large section roadway under high level stress. *Coal Mine Saf.* **2018**, *49*, 266–270.