

Combined support mechanism of rock bolt and anchor cable for adjacent roadways in the external staggered split-level panel layout

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Case study

Keywords: Combined support, Split-level panel layout, External staggered arrangement, Adjacent roadways

Posted Date: August 13th, 2020

DOI: <https://doi.org/10.21203/rs.3.rs-22696/v2>

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Version of Record: A version of this preprint was published on January 25th, 2021. See the published version at <https://doi.org/10.1007/s40789-020-00399-w>.

Abstract

Utilizing the spatial structure characteristic of the external staggered split-level panel layout, the combined support technology of adjacent roadways was proposed, and combined support mechanism of rock bolt and anchor cable was analyzed. The influence of side rock bolt and anchor cable parameters respectively on mechanical properties of the anchorage body and support stress distribution of lateral coal body were revealed through FLAC 3D software. The optimal support parameters of side rock bolt and anchor cable were determined subsequently. And the support effect of the gob-side entry under mining influence was verified. Result shows that the support of side rock bolt and anchor cable can improve mechanical properties and the stress state of the anchorage body, which has a good protective effect on the coal body of the air-intake entry roof and side wall. It is beneficial to the stability of the side wall and the realization of the suspension effect for the roof rock bolt and anchor cable, which makes the surrounding rock maintenance of the gob-side entry of the thick coal seam more favorable.

1 Introduction

In recent years, with the increasing demand for coal and the increased efficiency of coal production, coal mining activities have gradually shifted from shallow mining to deep mining (Xie et al. 2015). Under the condition of deep mining, the roadway of fully mechanized top coal caving faces the control problems of surrounding rock such as rock burst, strong mining influence, soft thick coal roof, which is prone to large deformation and instability. In response to China's call to build resource-saving mines in recent years, gob-side entry driving technology is extensively used to avoid the waste of resources and secondary geological disasters caused by the large coal pillar size.

Scholars have done a lot of research on the control of surrounding rock in gob-side entry. Based on the characteristic of surrounding rock in gob-side entry, the stability principle of big and small structures was put forward, which provided a theory basis to the application of bolting (Hou et al. 2011). The width expression of internal stress field and three fracture forms of the overlying main roof were researched to determine the rational position of gob-side entry (Li et al. 2012; Wang et al. 2014).

Effectively controlling the deformation of the surrounding rock of the roadway and keeping it safe and unobstructed are the primary condition for safe production in the mine. In terms of roadway support technology, rock bolt and anchor cable have become the main form of roadway support in coal mine. According to the support requirement of roadway under different geological conditions, the support theory and technology have been further developed.

He et al (2014a, b, 2015) manufactured a type of anchor bolt with constant resistance and large deformation, which can provide the constant resistance and stable deformation. Kang et al (2007, 2010a, b) put forward the high pretensioned stress and intensive supporting theory, and developed the high pretensioned stress and intensive bolting system, which include intensive bolt, steel strip and cable bolt with a big increase in mechanical functions. Ma et al (2013, 2015) manufactured the long-extension bolt

support technology to ensure the support system has enough length and extension performance. He et al (2015a, b, 2016) put forward the multiple-cable-girder-truss system, key component of which is the composite structure of steel beam and channel steel and multiple cables anchored in the depth of the roof.

The common feature of existing research results is that adjacent roadways on both sides of coal pillar are driven along the floor of coal seam, and the support scheme basically involves only a single roadway. According to the three-dimensional spatial characteristic of the external staggered split-level panel layout (ESSPL), the design of support scheme and parameters of two adjacent roadways can be realized.

Therefore, based on the combined support technology in ESSPL, geological conditions of Xinjulong Mine were taken as research object to study on the combined support mechanism of rock bolt and anchor cable. This paper mainly studies influences of side rock bolt and anchor cable parameters on mechanical properties of the anchorage body and the distribution characteristic of the support stress field respectively, and verifies the support effect under the influence of mining.

2 Combined Support Technology For Adjacent Roadways In The External Staggered Split-level Panel Layout (Esspl)

2.1 External staggered split-level panel layout (ESSPL)

Since the split-level panel layout (SPL) or longwall mining with split-level gate roads (LMSG) was invented in 1998 (Zhao et al. 1998), it has been applied in many production mines to effectively control the mine pressure and improve economic benefits.

As shown in Fig. 1, there are three types of roadway layout in split-level panel: internal staggered layout, overlapping layout and external staggered layout. According to the actual engineering situation, the appropriate roadway layout can be selected. The original intention of the SPL is to solve the problem of recovery rate in the conventional longwall top coal caving. It has successively carried out the field application of the internal staggered split-level panel layout (ISSPL) in Shanxi, Hebei and Shandong Provinces (Zhao. 2014). When the length of end shields is not enough to arrange the internal staggered gob-side entry, the overlapping layout can be used for the panel.

However, in that case, skip-mining is required for the succession of panels, and the passive support of the gob-side entry greatly limits the advance rate. With the ESSPL, the air-intake entry can be driven in the area with lower stress, so that the support form of rock bolt and anchor cable can be adopted to ensure the mining continuation. At present, the theory system of ESSPL has been improved (Wang et al. 2016, 2020a, b). In the process of technology promotion, Xiegou Mine, Fenyuan Mine and other mines have used it to realize the active rock supporting.

Compared with the conventional longwall top coal caving and the ISSPL, the ESSPL presents the three-dimensional characteristic in space, which can be expressed as the horizontal staggered distance L_1 and longitudinal staggered distance L_2 between two entries, as shown in Fig. 2.

2.2 Combined support technology for adjacent roadways

According to the three-dimensional spatial characteristic of ESSPL, the design of support scheme and parameters of two adjacent roadways can be realized by the side support of air-return entry driven along the coal seam roof. After its excavation, rock bolts and anchor cables are installed into the side coal wall which can even be reinforced by grouting. When air-intake entry of the successive panel was driving along the coal seam floor, the rock bolt and anchor cable can be installed on the roof to realize the combined support technology for adjacent roadways.

1-the air-return entry of the active panel 2-the air-intake entry of the successive panel

The characteristics of combined support technology shown in Fig. 3 are as follows:

- (1) The air-return entry is driven along the coal seam roof. Roof rock bolts and anchor cables are directly installed into immediate roof and main roof, which can give full play to the suspension effect of rock bolts and anchor cables;
- (2) Combined anchorage area is formed between the adjacent roadways, in which the cohesion c , internal friction angle φ , compressive strength σ and shear strength τ of the anchorage body will be enhanced;
- (3) The combined anchorage area can provide strong anchoring force for the roof rock bolt and anchor cable of air-intake entry to realize stronger suspension effect.

On the whole, using ESSPL and combined support technology can give full play to the suspension effect of roadway support under various geological and mining conditions and expand the application scope of suspension theory with simple and economic characteristics.

3 Engineering Case

Xinjulong Mine is located in the Heze City of Shandong Province. In the second mining area, 3# coal seam is mined at depth of around 800 m. The thickness of the coal seam varies from 3.8 to 8.5 m, and the average thickness is 7.2 m. The inclination angle of the coal seam is 6.0-9.6° with average inclination of 7.8°. The coal protodyakonov coefficient is 1.59. The density is 1360 kg/m³. The coal seam has stable occurrence and complex structure, with 0.1-0.35m thick mudstone or carbonaceous mudstone in the middle. The coal quality is mainly fat coal and 1/3 coking coal.

The active 2301S panel, which adopts fully mechanized top-caving technology, was 2319 m long along the strike and 274 m wide along the dip. And 20 m coal pillar is planned to left between 2301S panel and

2302S panel. Rectangular roadways are driven along the coal seam floor, with the size of 4.5 m×3.5 m. The supporting parameters are as follows:

(1) Six rock bolts and two anchor cables are used to support the roof of the entry. The model of high-strength left-handed longitudinal reinforcement threaded steel bolt is $\Phi 22$ mm×2500 mm, the spacing and row spacing is 850 mm×800 mm. The model of anchor cable is $\Phi 18.9$ mm×6300 mm, the spacing and row spacing is 2400 mm×800 mm.

(2) Five rock bolts and two anchor cables are used to support the coal side of the entry. The model of high-strength left-handed longitudinal reinforcement threaded steel bolt is $\Phi 22$ mm×2500 mm, the spacing and row spacing is 750 mm×800 mm. The model of anchor cable is $\Phi 18.9$ mm×4300 mm, the spacing and row spacing is 2400 mm×800 mm.

4 Influence Of Rock Bolt Parameters On Mechanical Properties Of Anchorage Body

In this section, FLAC^{3D} was used to analyze the influence of rock bolt parameters on mechanical properties of anchorage body. Therefore, it is necessary to adopt the control variable method. The related parameters of the rock bolt are mainly the spacing, pre-tightening force, length and diameter, as shown in Table 1.

Table 1 Experimental parameters of rock bolt

Rock bolt parameter	Value				
Spacing (m)	0.6	0.8	1.0	1.2	1.4
Pre-tightening force (kN)	40	60	80	100	120
Length (m)	2.2	2.4	2.6	2.8	3.0
Diameter (mm)	16	18	20	22	24

This experiment was based on the original mechanical parameters of 3# coal seam in the second mining area of Xinjulong Mine, as shown in Table 2. FLAC^{3D} software was used for modeling, which is calculated by Mohr-Coulomb failure criterion. The size of the model was 3.5m(length) × 3.0m (width)× 3.5m (height), which were divided into 36,750 cells and 40,176 nodes, as shown in Fig. 4.

Table 2 Rock mechanics parameters of 3# coal seam in Xinjulong Mine

Uniaxial compressive strength (MPa)	Uniaxial tensile strength (MPa)	Elastic Modulus (GPa)	Poisson ratio	Cohesion (MPa)	Internal friction angle (°)
10.40	0.51	26.28	0.29	3.1	28.5

Because the compressing speed of the compression member during the compression test will have a certain effect on the final stress-strain result, in order to fit the original parameters, the numerical simulation of the specimen with different loading speed by FLAC^{3D} was carried out many times. In the process of numerical simulation, the test specimen was subjected to three mechanical tests of uniaxial compression, 1MPa confining pressure and 2MPa confining pressure, as shown in Fig. 5 (a), and the corresponding Mohr-Coulomb stress circle was obtained, as shown Fig. 5 (b). The original elastic modulus of the experimental body is 26.2 GPa, the uniaxial compressive strength is 10.35 MPa, the cohesion is 3.08 MPa and the internal friction angle is 28.46°.

4.1 Influence of rock bolt spacing on mechanical properties of anchorage body

According to the rock bolt spacing designed in Table 1, other variables were controlled as rock bolt pre-tightening force of 40 kN, rock bolt diameter of 16 mm and rock bolt length of 2.2 m.

Repeat tests for each model: test uniaxial compressive strength first; Then, under the condition of 1, 2 MPa confining pressure, the compressive strength experiment was conducted to get the cohesion and internal friction angle. The shear strength of anchorage body was calculated from the above data. The experimental results were summarized to obtain influence curves of rock bolt spacing on the uniaxial compressive strength, shear strength, cohesion and internal friction angle of anchorage body, as shown in Fig. 6.

As shown in Fig. 6 (a), the uniaxial compressive strength and shear strength of anchorage body showed an increasing trend with the increase of rock bolt density. The uniaxial compressive strength of the 0.6 m spacing was 0.43 MPa higher than that of the 1.4 m spacing and 0.54 MPa higher than the original rock parameter. The shear strength of the 0.6 m spacing increased by 0.65 MPa compared with that of 1.4 m spacing, and 0.82 MPa compared with the original rock parameter.

As shown in Fig. 6 (b), with the decrease of rock bolt spacing, the cohesion and internal friction angle of anchorage body increased. The cohesion increase was not obvious, which was only 0.04 MPa when the spacing decrease from 1.4 m to 0.6 m. The internal friction angle of the 0.6 m spacing was 1.4° higher than that of the 1.4 m spacing and 1.83° higher than the original rock parameter.

In summary, effects of the rock bolt spacing on uniaxial compressive strength σ , internal friction angle φ , cohesion c and shear strength τ all increased with increasing support density, but the increase rate

decreased after 0.8 m. Each mechanical parameter had the largest increment between 0.8 m and 1.2 m, and subsequent studies control the rock bolt spacing to 0.8 m.

4.2 Influence of rock bolt pre-tightening force on mechanical properties of anchorage body

According to Table 1, the rock bolt pre-tightening force was designed as 40 kN, 60 kN, 80 kN, 100 kN and 120 kN. The rock bolt spacing was 0.8 m, and other variables were controlled as diameter of 16 mm and length of 2.2 m. The experimental results are shown in Fig. 7.

As shown in Fig. 7 (a), the uniaxial compressive strength and shear strength of anchorage body showed an increasing trend with the increase of rock bolt pre-tightening force. When the rock bolt pre-tightening force increased from 40 kN to 120 kN, the uniaxial compressive strength increased by 0.49 MPa, and finally increased by 0.98 MPa compared with the original rock parameter. The shear strength of 120 kN was 0.6 MPa higher than that of 40 kN, and 1.36 MPa higher than the original rock parameter.

As shown in Fig. 7 (b), the cohesion and internal friction angle of anchorage body also increased with the increase of rock bolt pre-tightening force. When the rock bolt pre-tightening force was 120 kN, the cohesion increment was the largest, which was 0.09 MPa higher than that of 40 kN, 0.131 MPa higher than that of the original. And the effect of rock bolt pre-tightening force on the internal friction angle was significant. The internal friction angle with pre-tightening force of 120 kN was 0.82° more than that of the 40 kN, and had an increase of 2.47° from the original rock parameter.

According to the comprehensive analysis of Fig. 7, with the increase of rock bolt pre-tightening force, the uniaxial compressive strength σ , internal friction angle φ , cohesion c and shear strength τ increased gradually. Considering that the increase rate decreased rapidly when pre-tightening force reached 80 kN, the rock bolt pre-tightening force is taken as 80 kN.

4.3 Influence of rock bolt diameter on mechanical properties of anchorage body

According to Table 1, the rock bolt diameter was selected as 16 mm, 18 mm, 20 mm, 22 mm and 24 mm. The rock bolt spacing was 0.8 m, the pre-tightening force was 80 kN and the length was 2.2 m. The experimental results are shown in Fig. 8.

As shown in Fig. 8 (a), the uniaxial compressive strength and shear strength of anchorage body increased with the increase of rock bolt diameter. When the anchor diameter reached 24 mm, the uniaxial compressive strength increased by 0.27 MPa compared with that of 16 mm, and increased 1.1 MPa compared with the original rock parameter. And the shear strength with rock bolt diameter of 24 mm increased by 0.4 MPa compared with that of 16 mm diameter, and increased by 1.54 MPa compared with that of the original.

As shown in Fig. 8 (b), the cohesion and internal friction angle of anchorage body also increased with the increase of rock bolt diameter. When the rock bolt diameter was 24 mm, the cohesion was 0.026 MPa larger than that of 16 mm, 0.13 MPa larger than the original rock parameter. The internal friction angle

with the diameter of 24 mm was 0.78° higher than that of 16 mm and 2.96° higher than that of the original.

Comprehensive analysis shows that when the rock bolt diameter reached 22 mm, the increment of mechanical properties was large, but the increase rate slowed down when rock bolt diameter exceeded 22 mm. Therefore, the rock bolt diameter is taken as 22 mm.

4.4 Influence of rock bolt length on mechanical properties of anchorage body

According to Table 1, the design rock bolt length was 2.2 m, 2.4 m, 2.6 m, 2.8 m and 3.0 m respectively. The rock bolt spacing was 0.8 m, the pre-tightening force was 80 kN and the diameter was 22 mm. The experimental results are shown in Fig. 9.

As shown in Fig. 9 (a), when the rock bolt length increased from 2.2 m to 3.0 m, the uniaxial compressive strength and shear strength of anchorage body showed an increasing trend. The uniaxial compression strength of 3.0 m rock bolt length increased by 0.34 MPa compared with that of 2.2 m, and 1.42 MPa compared with that of the original. And the shear strength of 3.0 m increased by 0.49 MPa compared with that of 2.2 m, and 1.98 MPa compared with the original rock parameter.

As shown in Fig. 9 (b), with the increase of rock bolt length, the cohesion and internal friction angle of anchorage body increased. When the rock bolt length was 3.0 m, the cohesion and internal friction angle was respectively 0.048 MPa and 0.86° larger than that of 2.2 m, which was 0.168 MPa and 3.77° larger than original rock parameters.

According to the comprehensive analysis of Fig. 9, the rock bolt length was positively related to all parameters, while the length of 2.4 m was the inflection point. When the rock bolt length reached 2.4 m, the growth rate of mechanical parameters started to increase rapidly. In order to get the most efficient increment, the rock bolt length is taken as 3.0 m.

4.5 Optimal support parameters of side rock bolts for air-return entry

In the above section, the influence of rock bolt spacing, pre-tightening force, diameter and length on mechanical parameters of anchorage body were studied respectively, and the reasonable values of rock bolt support parameters are determined as rock bolt spacing of 0.8 m, rock bolt pre-tightening force of 80 kN, rock bolt diameter of 22 mm and rock bolt length of 3.0 m. The mechanical properties of anchorage body in this support scheme are the uniaxial compressive strength of 11.78 MPa, shear strength of 10.67 MPa, cohesion of 3.24 MPa and internal friction angle of 32.2° . The above parameters are respectively increased by 1.42 MPa, 1.98 MPa, 0.168 MPa, 3.77° compared with original rock parameters.

By comparing the variation interval of mechanical properties caused by the variation of various parameters of the rock bolt, combining with the actual engineering situation, it is found that the rock bolt pre-tightening force and spacing have a great influence, the length and diameter of rock bolt have little

effect on mechanical properties of anchorage body. Therefore, in the actual production, the design of rock bolt support scheme should focus on the rock bolt spacing and pre-tightening force.

5 Influence Of Anchor Cable Parameters On The Support Stress Distribution Of Lateral Coal Body

The action mechanism of the rock bolt and anchor cable can be divided into two aspects: one is the influence on mechanical parameters of the anchorage body, the other is to change the stress state of the surrounding rock, which makes the stress state develop in a direction conducive to stability. During underground mining, original rock stress field, mining stress field and support stress field interact with each other. The original rock stress field and mining stress field are much larger than the support stress field formed by the support structure, which is not conducive to observing the support stress field (Wang et al. 2008). Under the condition of not considering the original rock stress field, based on the determined rock bolt scheme, FLAC^{3D} was used to analyze changes in the distribution of support stress field due to different anchor cable parameters to provide a basis for the reasonable design of support parameters. Similar to the study of rock bolt parameters, the parameters related to the anchor cable mainly include length, spacing and pre-tightening force, as shown in Table 3.

Table 3 Experimental parameters of anchor cable

Anchor cable parameter	Value			
Length (m)	6.0	8.0	10.0	12.0
Spacing (m)	1.4	1.2	1.0	0.8
Pre-tightening force (kN)	150	200	250	300

The model size was set as 50m(length) × 20m(width) × 50m(height), and air-return entry size was 4.5m(width) × 3.5m(height), resulting in a total of 295000 elements and 314874 nodes, as shown in Fig. 10. The rock bolt and anchor cable were staggered installed along the strike, as shown in Fig. 11.

The model was calculated by Mohr-Coulomb failure criterion. The horizontal and vertical velocity of upper and bottom boundaries were constrained, and the horizontal velocity of side boundaries is constrained. Five rock strata with different properties were established in the model, among which rock parameters were the same as those of Xinjulong Mine, as shown in Table 4.

Table 4 Properties of rock strata for numerical modeling

Lithology	Bulk modulus (GPa)	Shear modulus (GPa)	Density (kg/m ³)	Internal friction angle (°)	Cohesion (MPa)	Tensile strength (GPa)
Fine Sandstone	15.80	11.30	2 700	42.0	5.50	2.40
Siltstone	8.40	5.90	2 350	39.0	4.20	1.20
3# Coal Seam	11.90	7.10	1 400	28.5	3.10	0.51
Siltstone	8.40	5.90	2 350	39.0	4.20	1.20
Fine Sandstone	15.80	11.30	2 700	42.0	5.50	2.40

5.1 Influence of anchor cable length on the support stress distribution of lateral coal body

According to Table 3, the anchor cable length was set as 6.0 m, 8.0 m, 10.0 m and 12.0 m. The remaining variables were controlled as the pre-tightening force of 150 kN and the spacing of 1.4 m. The experimental results are shown in Fig. 12.

Observing Fig. 12 (a), the right side of the air-return entry was in turn the rock bolt-anchor cable support compressive stress zone, the anchor cable support compressive stress zone and the anchor cable support tension stress zone. In the rock bolt-anchor cable support compressive stress zone, due to the combined support effect of side rock bolts and anchor cables, a compressive stress zone of about 0.25 MPa was formed. In the anchor cable compressive stress zone, only a compressive stress zone of about 0.15 MPa was formed due to the support effect of the anchor cable only. A stress concentration zone appeared at the initial stage of anchor cable anchoring, and the tensile stress was about 0.07 MPa.

Through a comparative analysis of Fig. 12, it is found that as the length of the side anchor cable increases, the influence range of the side anchor cable gradually increased. However, with the increase of the side anchor cable length, the stress value in the compressive stress zone of the anchor cable support gradually decreased. When the length of the anchor cable reached 10 m, the stress value in the compressive stress zone of the anchor cable was significantly reduced, only about 0.08 MPa.

The numerical simulation tests show that increasing the length of the anchor cable can significantly increase the support range. But under the same pre-tightening force, as the length of the anchor cable increases, the prestressing effect of the side anchor cable tends to weaken. Therefore, the pre-tightening force of the side anchor cable should be increased as the length of the anchor cable increases. According to the analysis, the length of the side anchor cable should not be too short or too long. It should be related to the air-intake entry position of the successive panel, so as to fully control the roof coal of the air-return

entry. In this study, the horizontal staggered distance is set to 3.0 m, adding roadway width 4.5 m, so the anchor cable length of 8.0 m is selected as the following research basis.

5.2 Influence of anchor cable spacing on the support stress distribution of lateral coal body

According to Table 3, the anchor cable spacing was designed to be 1.4 m, 1.2 m, 1.0 m and 0.8 m. In order to ensure the staggered installation of anchor cable and rock bolt, the row spacing of anchor cable was set as 1.6 m. The length of the anchor cable was 8.0 m, and the pre-tightening force was 150 kN. The horizontal support stress fields of different spacing are shown in Fig. 13.

With the increase of the anchor cable spacing, the stress value in the support stress field gradually increased. When the spacing between anchor cables was 1.4 m, the stress value in the rock bolt-anchor cable support compressive stress zone was 0.175 MPa, and the stress value in the anchor cable support compressive stress zone was 0.075 MPa. When the spacing between anchor cables was 0.8 m, the stress values were 0.28 MPa and 0.175 MPa respectively, which proves that reducing the spacing between anchor cables is beneficial to improve the control effect of anchor cables on lateral coal body.

When the spacing distance was reduced to 1.0 m, high stress ranges interconnected in the anchor cable compressive stress zone, and the support stress field formed by the rock bolt and the anchor cable were superimposed on each other. The control effect on the lateral coal body was more obvious. Therefore, the follow-up study controls the anchor cable spacing to 1.0 m.

5.3 Influence of anchor cable pre-tightening force on the support stress distribution of lateral coal body

According to Table 3, the pre-tightening force of the anchor cable was designed as 150 kN, 200 kN, 250 kN and 300 kN. The length of the anchor cable was 8.0 m, and the spacing between anchor cables was 1.0 m. The test results are shown in Fig. 14.

Observing Fig. 14 (a), due to the small pre-tightening force, a support compressive stress field of only about 0.15 MPa was formed at the tail and beginning of the anchor cable. Through a comparative analysis of Fig. 14 (a) ~ (d), it is found that under the control of other variables, with the increase of the pre-tensioning force of the side anchor cable, the range of the support pressure stress area formed by the side anchor cable gradually increased. When the 250 kN pre-tightening force was applied, high stress ranges interconnected in the anchor cable compressive stress zone, in which the stress value can reach about 0.245 MPa. The stress value and range of the rock bolt-anchor cable support compressive stress zone were large, which can give full play to the combined support effect.

5.4 Optimal support parameters of side anchor cable for air-return entry

The reasonable value of the anchor cable support parameters has been determined as the length of 8.0 m, the spacing of 1.0 m and the pre-tightening force of 250 kN. Under this support scheme, the superposition effect between rock bolt and anchor cable was obvious. A support compressive stress field

of about 0.3 MPa can be formed in the rock bolt-anchor cable support compressive stress zone, and that of 0.245 MPa can be formed in the anchor cable support compressive stress zone.

After the excavation of the air-return entry, the support effect of the rock bolt and anchor cable can timely apply a certain compressive stress to the lateral coal body to reduce the tensile shear stress caused by the excavation and unloading, which makes the lateral coal body's stress state develop in a direction conducive to stability.

Different anchor cable support parameters lead to different stress fields, which in turn affect the mechanical parameters of the coal body. Increasing the control range and capacity of the anchor cable is conducive to the formation of a prestressed load-bearing structure with greater stiffness in the anchorage area, which can become the main bearing structure, thereby preventing the rock layer outside the anchorage area from separating and facilitating the air-intake entry roof maintenance.

6 Analysis Of Support Effect Under Mining Influence

In the case of the ESSPL, the air-return entry is driven along the roof of the coal seam, which is equivalent to reducing the mining height of the thick coal seam. In addition, the trapezoidal bottom coal body which left between panels will play a certain role in protecting the air-intake entry of the successive panel. So its external staggered distance should be smaller than that of the traditional gob-side entry. When the entry driving along goaf, normally no coal pillar or only 3-5m wide coal pillar are left to block gangue, water or harmful gas in goaf (Du et al. 2014). In this paper, the external staggered distance between panels is selected as 3.0 m.

In the mining process of 2302S panel, the superposition of the front abutment pressure and side abutment pressure will change the stress state of the gob-side entry, which will lead to the increase of the plastic area and roadway deformation. Considering the influence of mining process in successive panel, the combined supporting effect of the external staggered gob-side entry is verified by numerical simulation.

The dimension of the model was 392 m (length) × 300 m (width) × 120 m (height), as shown in Fig. 15. The panel length along the strike is set to 110 m in consideration of the calculation amount of the model. The side boundaries were roller constrained and the bottom boundary was fixed both horizontally and vertically. A uniform stress of $800 \text{ m} \times 0.025 \text{ MN/m}^3 = 20 \text{ MPa}$ was applied to the top of the model corresponding to 800 m of overburden strata by assuming the overlying unit weight was 0.025 MN/m^3 . The rock mass engineering properties used for numerical modeling are given in Table 4. The optimal side rock bolt and anchor cable support parameters of the air-return entry in the ESSPL have been determined in Section 3.5 and Section 4.5.

According to the monitoring result of the abutment pressure distribution, the distance from the stress peak position to the coal wall of working face is about 20m. So after the mining of 2301S panel was

completed, the 2302S panel was excavated 150m. The plastic zone distribution which was 20 m in front of the working face around the external staggered gob-side entry was extracted, as shown in Fig. 16.

The blue color in the figure indicates that no damage has occurred, that is an elastic zone. The red color in the figure indicate that failure is currently occurring, indicating that the state of the surrounding rock in the area is unstable in the current state. The green color in the figure indicates that damage has occurred in the past but the damage is not continuing. It shows that in the current state, the surrounding rock state of this area is relatively stable.

Due to the support effect of side bolts and anchor cables along the air-return entry, the roof of the air-intake entry in the combined support area has not been completely damaged. The red areas on the roof and sides of the air-intake were small, most of the remaining area were green, which indicates that under the current stress state, the plastic failure trend of the surrounding rock is little, and red areas were all within the control range of the rock bolt. The plastic depth in the middle of the air-intake entry roof is only 2.3 m, and the maximum depth is 3.5 m. The upper rock layer of the combined support area had no plastic failure, which can provide an effective anchor point for the roof rock bolts and anchor cables of air-intake entry to prevent the plastic zone from further development and ensure the stability of the roadway roof.

The analysis shows that under the condition of the ESSPL, mechanical properties and the stress state of the surrounding rock are improved due to the anchoring effect of the side rock bolt and anchor cable of air-return entry, which plays a good role in protecting the coal body of the air-intake entry roof and side wall. It is conducive to the stability of the side wall and the realization of the suspension effect for roof rock bolts and anchor cables, which makes the surrounding rock maintenance of the gob-side entry of the thick coal seam more favorable.

7 Conclusion

(1)Utilizing the spatial structure characteristic of the external staggered split-level panel layout (ESSPL), the combined support technology of adjacent roadways was proposed, and its characteristics were summarized and analyzed.

(2)FLAC^{3D} was used to analyze the influence of rock bolt parameters of on mechanical properties of the anchorage body. The optimal values of rock bolt support parameters are determined as the rock bolt spacing of 0.8m, the rock bolt pre-tightening force of 80 kN, the rock bolt diameter of 22 mm and the rock bolt length of 3.0 m. Under the support scheme, the uniaxial compressive strength of anchorage body is increased by 1.42 MPa, the shear strength is increased by 1.98 MPa, the cohesion is increased by 0.168 MPa and the internal friction angle is increased by 3.77°. It is shown that the method of installing rock bolts to the coal body along the air-return entry can improve mechanical properties of the anchorage body, which provides mechanical basis for the roof support of the external staggered gob-side entry.

(3)FLAC^{3D} was used to analyze the influence of anchor cable parameters on the support stress distribution of lateral coal body. It is determined that the anchor cable length of 8.0 m, the anchor cable spacing of 1.0 m and the anchor cable pre-tightening force of 250 kN. Under this support scheme, the superposition effect between rock bolt and anchor cable support parameters is obvious. A support compressive stress field of about 0.3 MPa can be formed in the rock bolt-anchor cable support compressive stress zone, and that of 0.245 MPa can be formed in the anchor cable support compressive stress zone.

(4)Through the verification of the support effect under the influence of mining, it was found that in the case of the ESSPL, due to the supporting of air-return entry side rock bolts and anchor cables, mechanical properties and the stress state of the surrounding rock are improved, which plays a good role in protecting the coal body of the air-intake entry roof and side wall. It is conducive to the stability of the side wall and the realization of the suspension effect for roof bolts and anchor cables, which makes the surrounding rock maintenance of the gob-side entry of the thick coal seam more favorable.

Declarations

Acknowledgement

This study was funded by the National Natural Science Foundation of Surface Project of China (No. 51774289), the National Natural Science Foundation of the Youth Science Foundation of China (No. 51404270), the Fundamental Research Funds for the Central Universities (2011QZ06) and the Open Fund of State Key Laboratory of Coal Resources in Western China (SKLCRKF1903)

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Conflicts of interest/Competing interests

The authors declare that they have no conflict of interest.

Availability of data and material

All data generated or used during the study appear in the submitted article.

Code availability

Not applicable

Authors' contributions

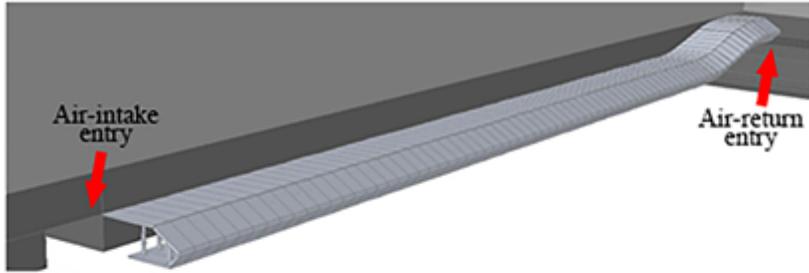
Adrian Batugin and Wang Zhiqiang contributed to the conceptualization of the paper. Wang Zhiqiang conceived the technology. All authors were involved in designing and carrying out the numerical modeling studies. Wang Zhiqiang and Su Zehua wrote the paper. Adrian Batugin, Lv Wenyu and Shermatova Sayyora Sidikovna performed significant review and editing of the paper.

References

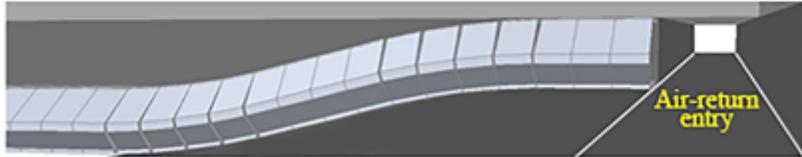
1. Du JP, Meng XR (2014) Mining science. China University of Mining and Technology Press, Xuzhou.
2. He FL, Bo YS, Xu ZH, Li EP, Xu L, Li TD (2015) Research on stability of open-off cut surrounding rock with thick coal roof and its reasonable anchor cable configuration. *J Min Saf Eng* 32(2):233-239.
3. He FL, Gao F, Sun YJ, Li SJ, Song BH, Yang YF (2015) Multiple-cable-girder-truss asymmetric support mechanism and its application in the roadway of fully mechanized top coal caving face with narrow coal pillar. *J China Coal Soc* 40(10):2296-2302.
4. He FL, Zhang GC (2016) Deformation and failure mechanism and control technology of large section coal roadway subjected to severe mining dynamic load. *J Min Saf Eng* 33(3): 423-430.
5. He MC, Guo ZB (2014) Mechanical property and engineering application of anchor bolt with constant resistance and large deformation. *Chin J Rock Mech Eng* 33(7): 1297-1308.
6. He MC, Li C, Gong WL (2015) Elongation and impacting experimental system for bolts with constant resistance and large deformation and finite element analysis. *Chin J Rock Mech Eng* 34(11):2161-2178.
7. He MC, Wang J, Sun XM, Yang XJ (2014) Mechanics characteristics and applications of prevention and control rock bursts of the negative poisson's ratio effect anchor. *J China Coal Soc* 39(2): 214-221.
8. Hou CJ, Li XH (2011) Stability principle of big and small structures of rock surrounding roadway driven along goaf in fully mechanized top coal caving face. *J China Coal Soc* 26(1): 1-7.
9. Kang HP, Wang JH, Lin J (2007) High pretensioned stress and intensive bolting system and its application in deep roadways. *J China Coal Soc* 32(12): 1233-1238.
10. Kang HP, Wang JH, Lin J (2010) Case studies of rock bolting in coal mine roadways. *J China Coal Soc* 29(4): 649-664.
11. Kang HP, Wang JH, Lin J (2010) Study and applications of roadway support techniques for coal mines. *J China Coal Soc* 35(11): 1809-1814.
12. Li L, Bai JB, Wang XY (2012) Rational position and control technique of roadway driving along next goaf in fully mechanized top coal caving face. *J China Coal Soc* 37(9):1564-1569.
13. Ma NJ, Zhao XD, Zhao ZZ, Li J, Guo XF (2015) Stability analysis and control technology of mine roadway roof in deep mining. *J China Coal Soc* 40(10): 2287-2295.

15. Ma NJ, Zhao ZZ, Feng JC (2013) Technology of butt long bolt on roadway supporting in difficult conditions. *Coal Sci Technol* 41(9):117-121.
16. Wang HS, Zhang DS, Li SG, Wang L, Wu LZ (2014) Rational width of narrow coal pillar based on the fracture line location of key rock B in main roof. *J Min Saf Eng* 31(1):10-16.
17. Wang JH, Kang HP, Gao F (2008) Numerical simulation on load-transfer-mechanisms and stress distribution characteristics of cable bolts. *J china coal soc*, 33(1):1-6.
18. Wang ZQ (2016) Combined support technology of adjacent roadways for gob-side entry driving in Thick Coal Seams. China Patent 201610547374.7, 12 July.
19. Wang ZQ, Guo L, Su ZH, Wang SS, Shen C (2020) Layout and combined support technology of alternate exterior stagger arrangement roadway and adjacent roadways in inclined and medium-thick coal seam. *J China Coal Soc* 45(2): 542-555.
20. Wang ZQ, Shi L, Wang P, Wu C (2020) A Novel Gob-Side Entry Method in China: A Case Study. *Geotech Geol Eng* 38:215-226.
21. Xie HP, Gao F, Ju Y (2015) Research and development of rock mechanics in deep ground engineering. *Chin J Rock Mech Eng* 34(11):2161-2178.
22. Zhao JL, Li B (1998) A study on raising the recovery ratio of fully-mechanized coal-talking about stagger-position-roadway-arrangement system. *J. Liaoning Tech. Univ. (Nat. Sci.)* 17, 237-239.
23. Zhao JL (2014) Development and Outlook of 3-D Stagger Arrangement Roadway Layout Technology. *Coal Eng* 46(1):1-3.

Figures



(a) 3-D view of the split-level panel



(b) Enlarged view of the split-level panel



(c) Internal staggered layout



(d) Overlapping layout



(e) External staggered layout

Figure 1

Split-level panel layout (SPL)

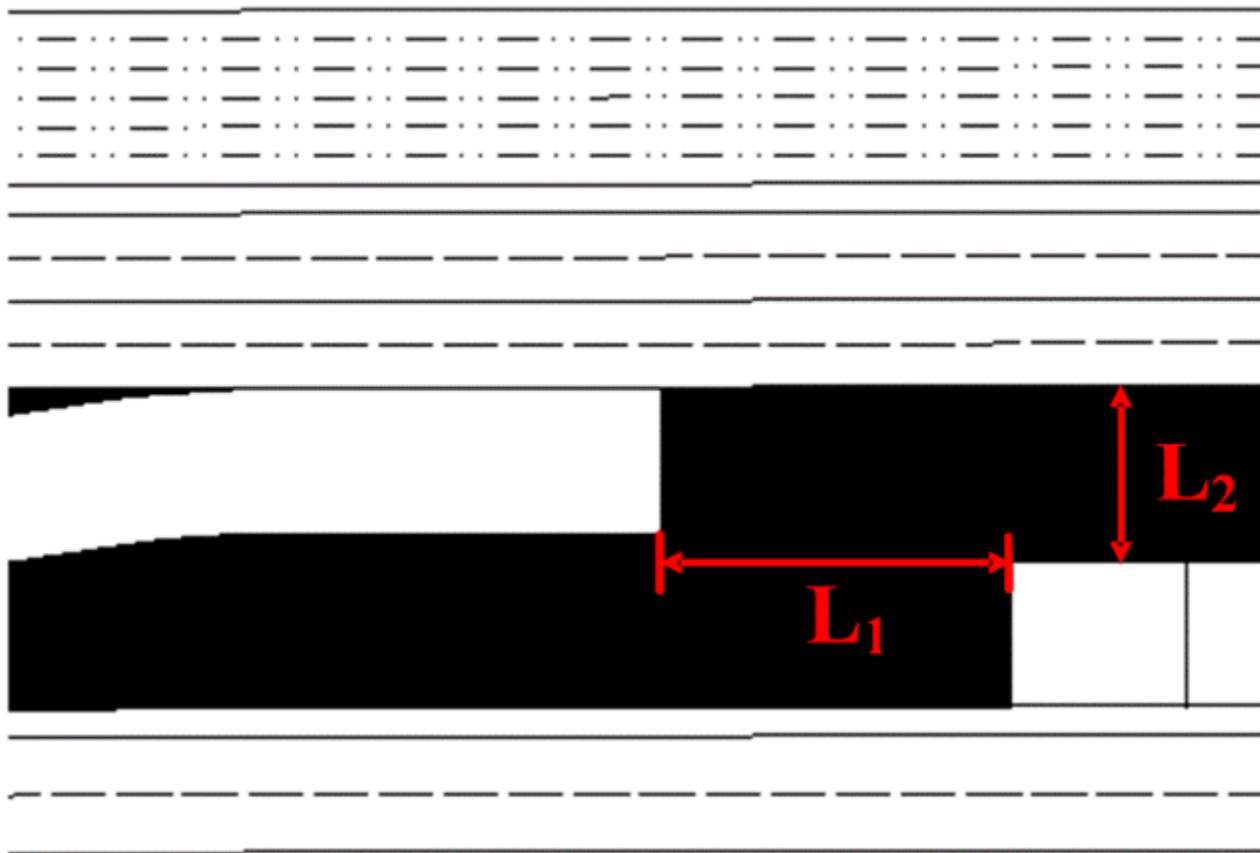


Figure 2

External staggered split-level panel layout (ESSPL)

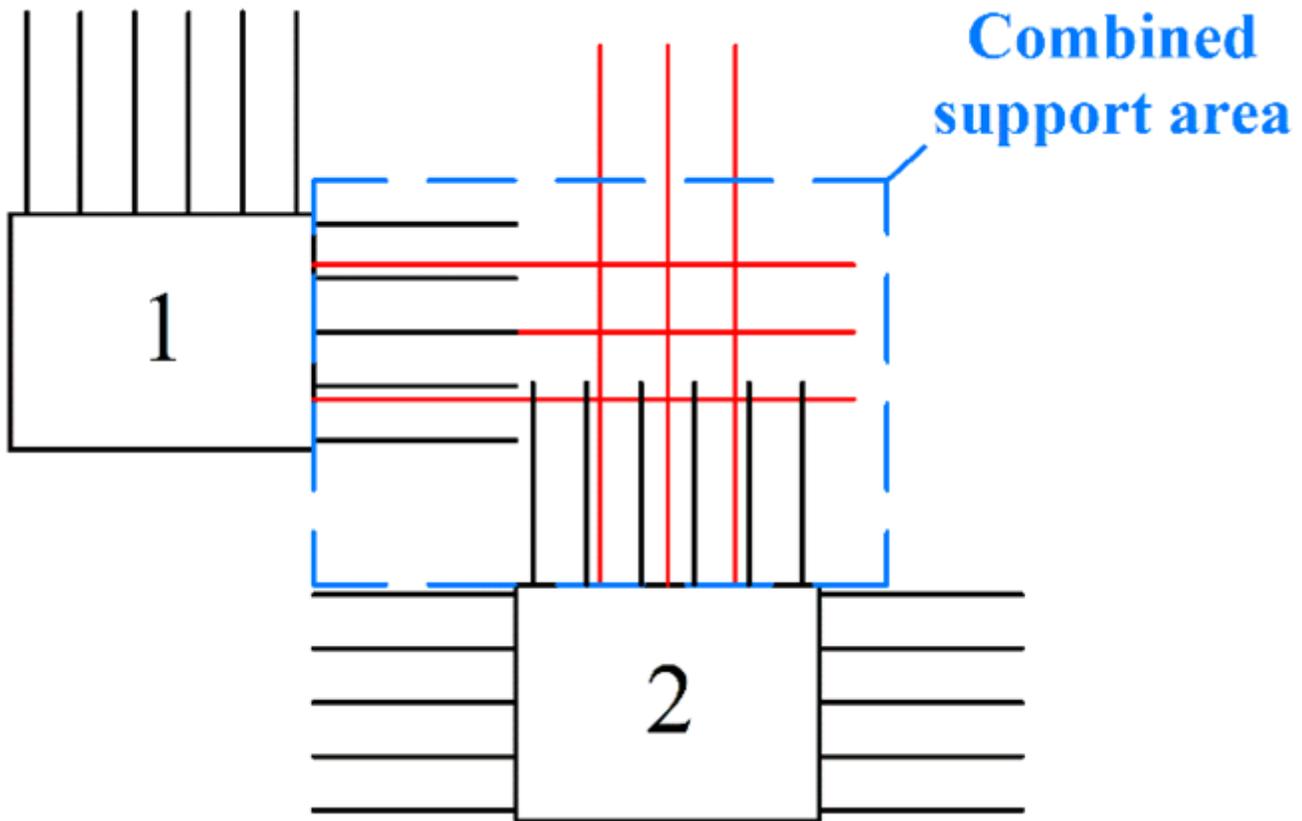


Figure 3

Combined support technology for adjacent roadways

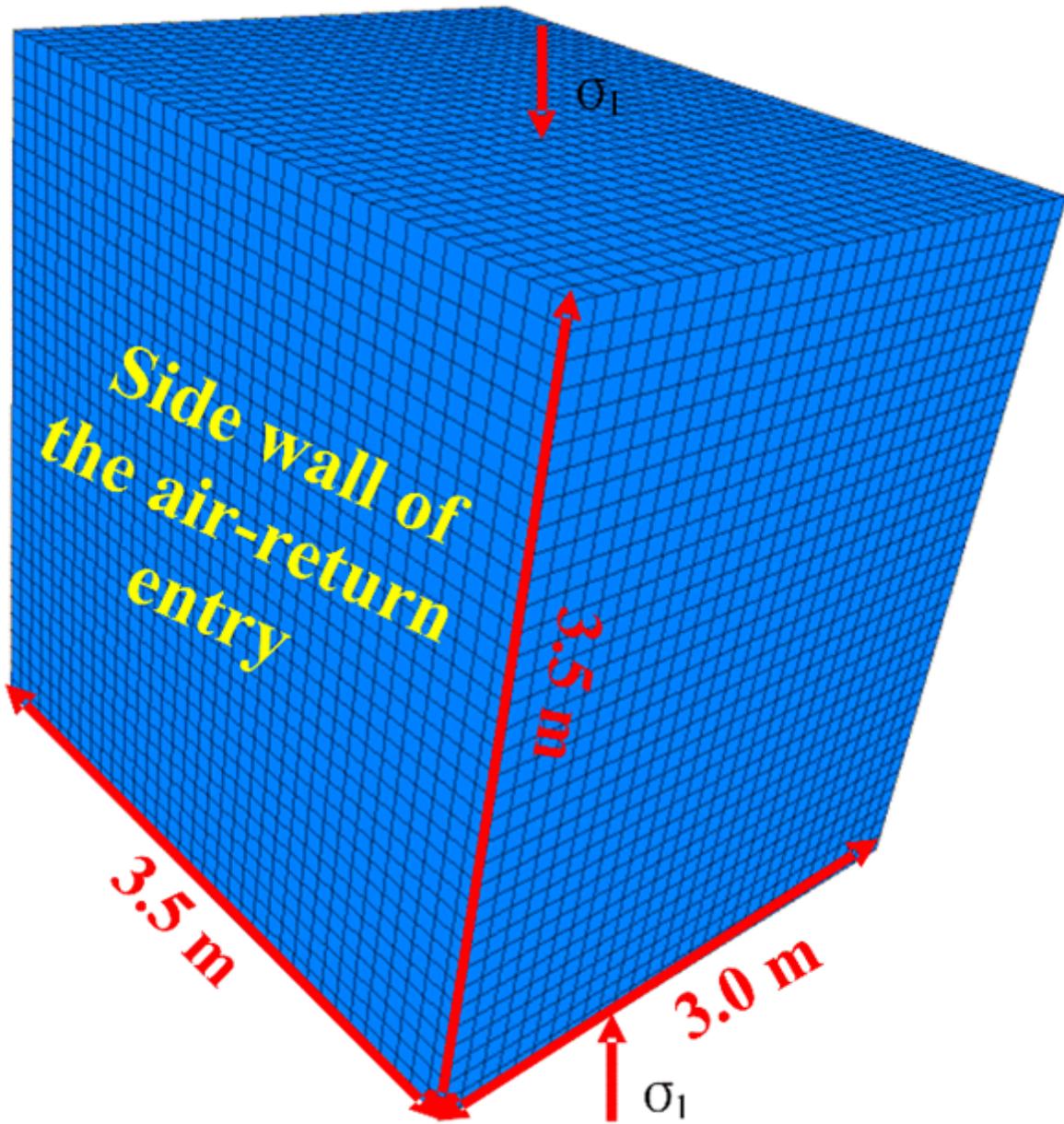
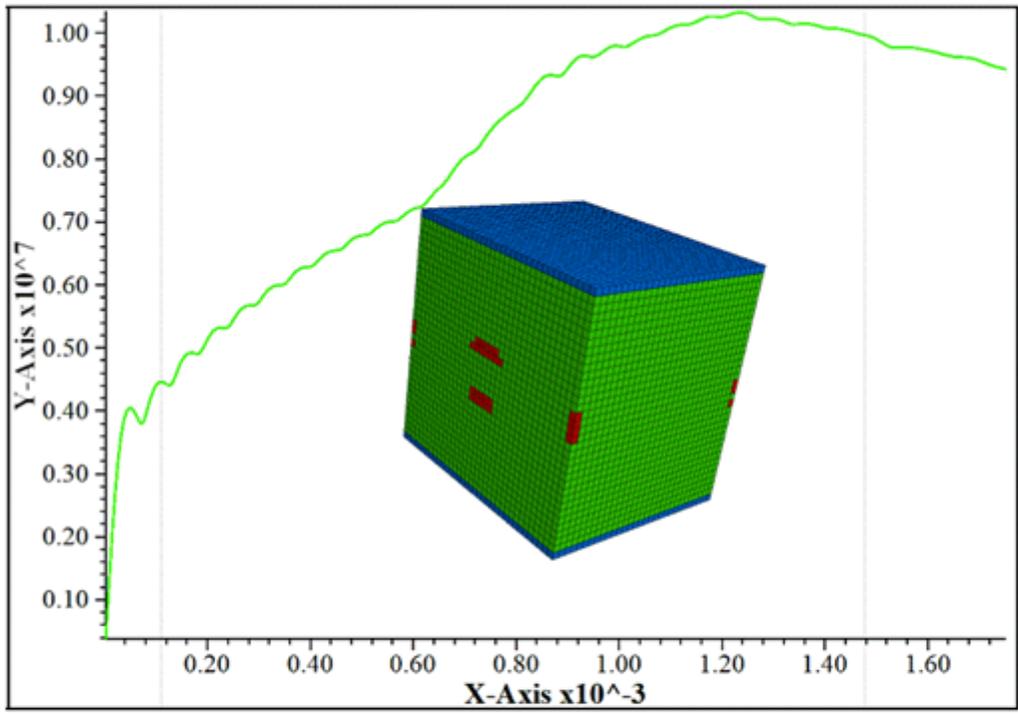
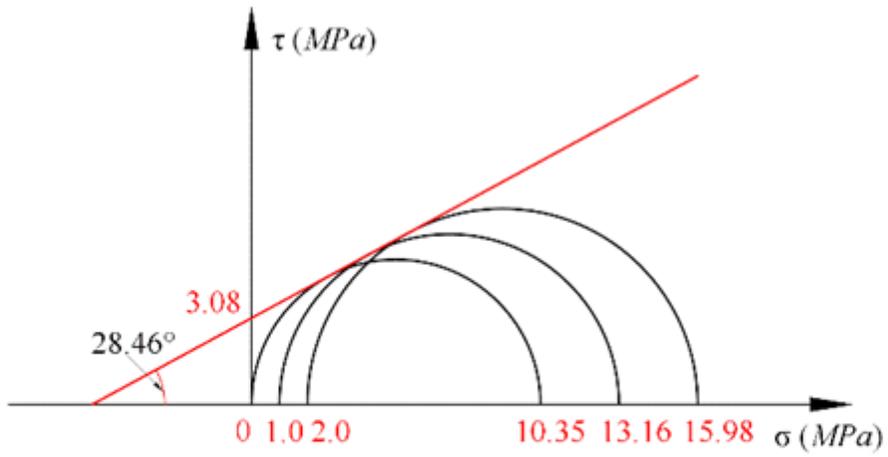


Figure 4

FLAC3D model of anchorage body



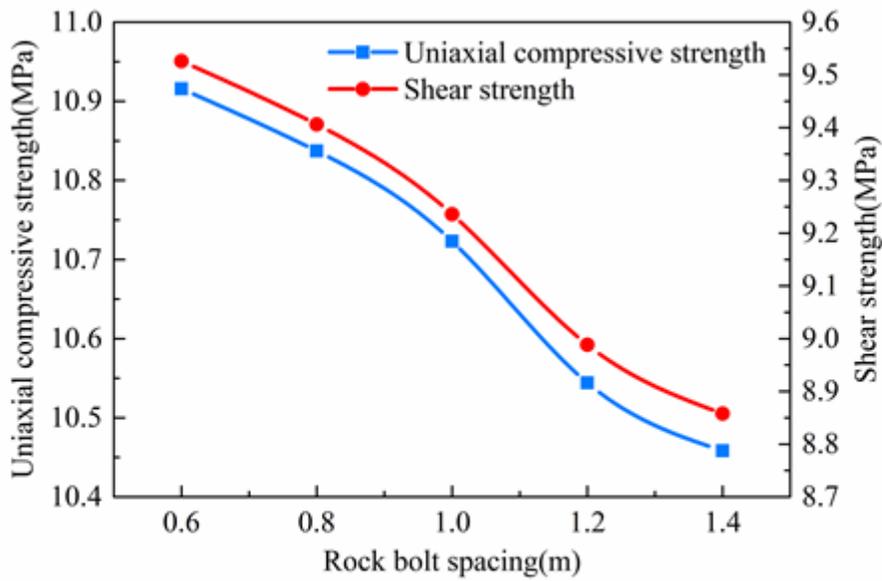
(a) Numerical compression test



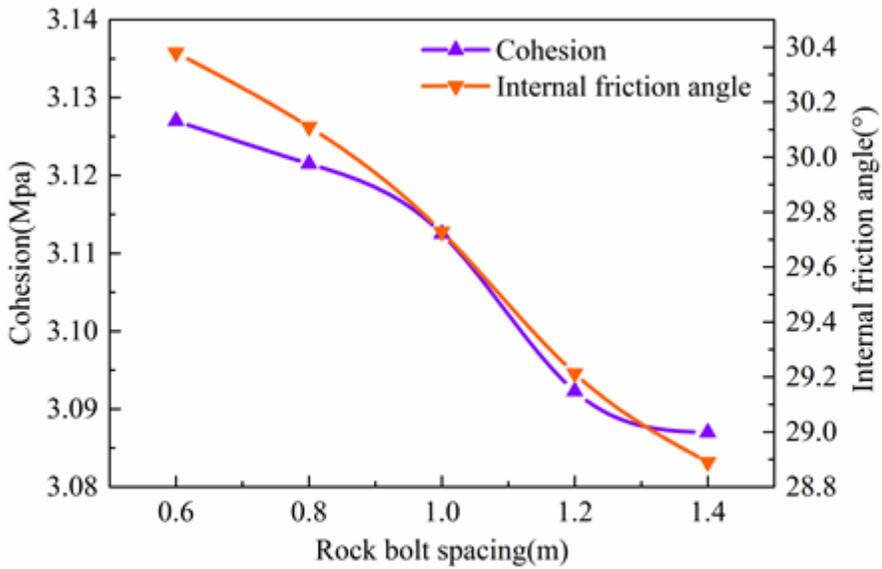
(b) Mohr-Coulomb stress circle

Figure 5

Compression test to fit original parameters



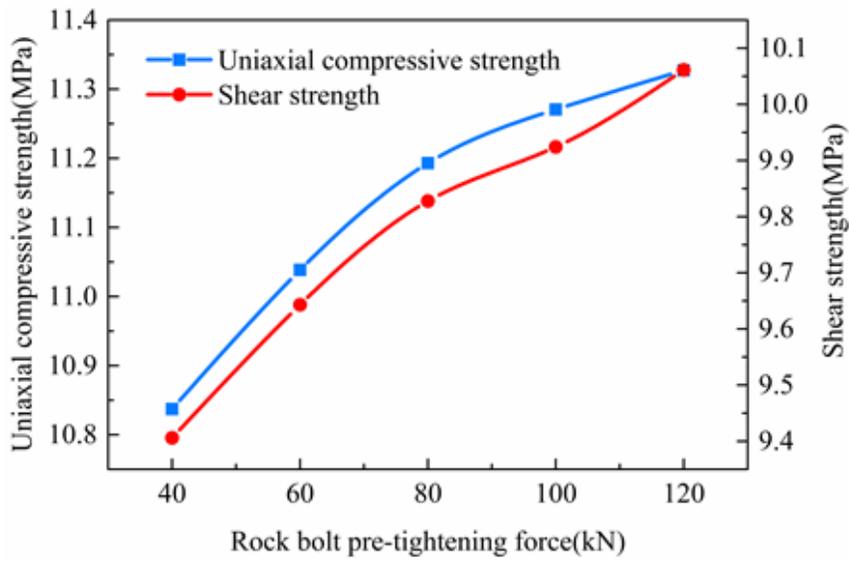
(a) Influence of rock bolt spacing on uniaxial compressive strength and shear strength



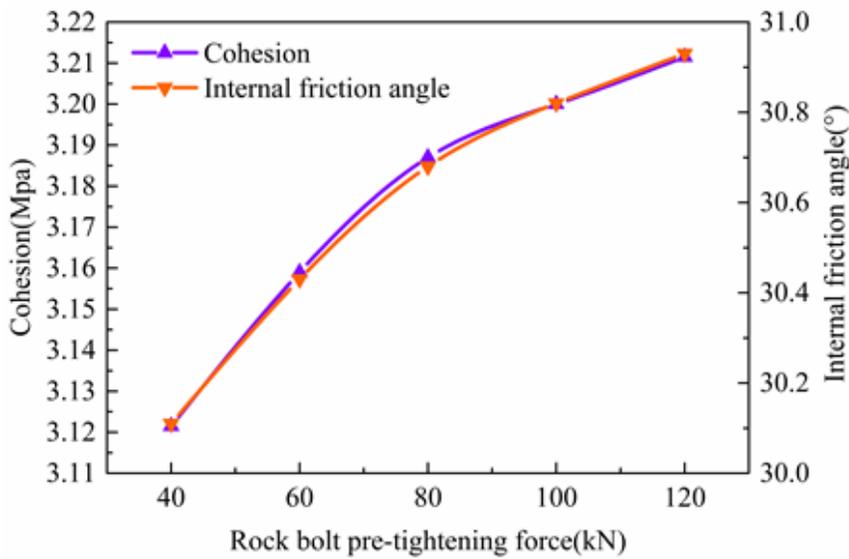
(b) Influence of rock bolt spacing on cohesion and internal friction angle

Figure 6

Influence of rock bolt spacing on mechanical properties of anchorage body



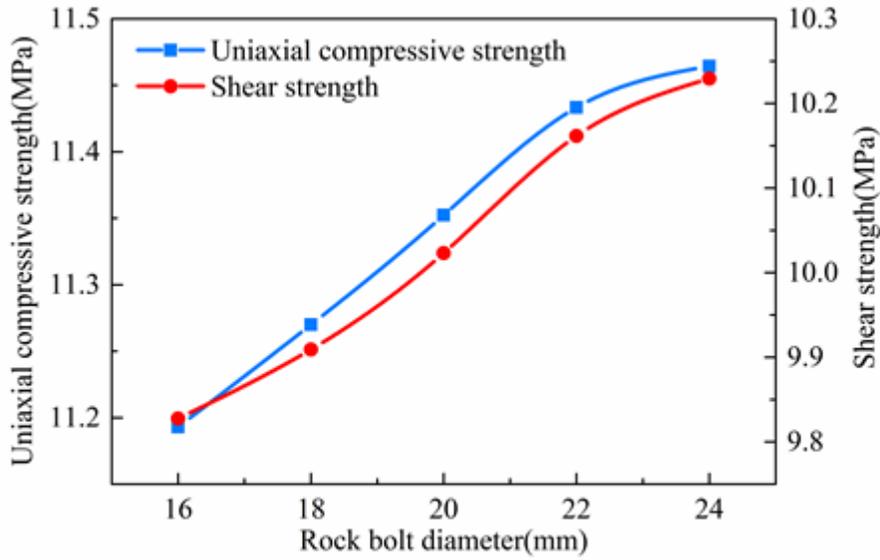
(a) Influence of rock bolt pre-tightening force on uniaxial compressive strength and shear strength



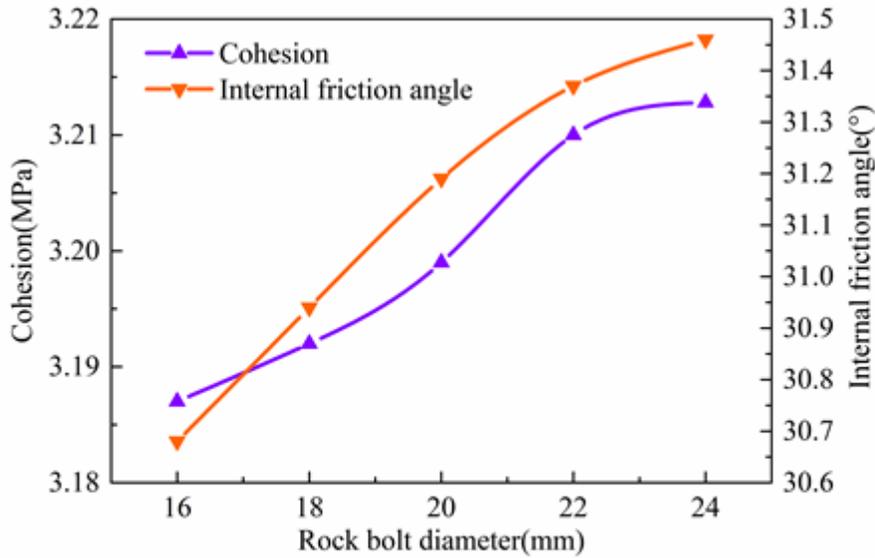
(b) Influence of rock bolt pre-tightening force on cohesion and internal friction angle

Figure 7

Influence of rock bolt pre-tightening force on mechanical properties of anchorage body



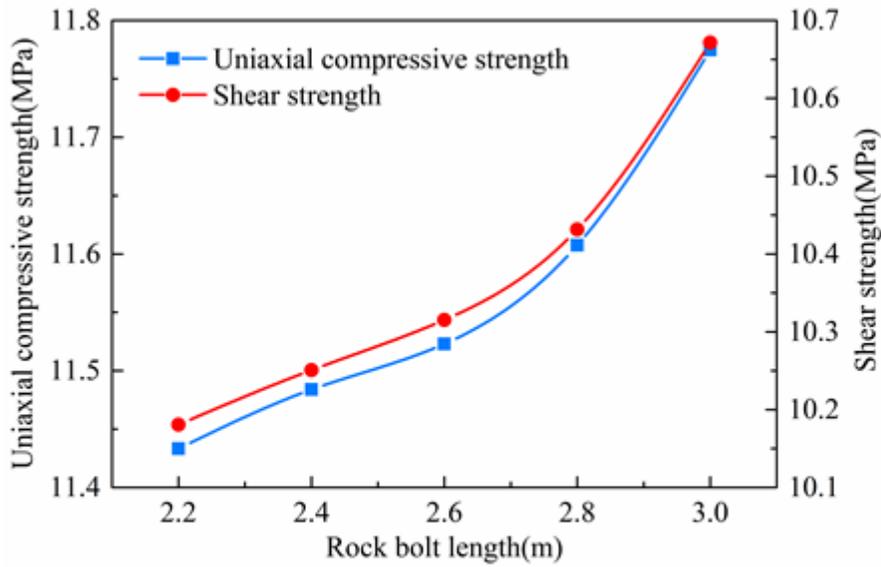
(a) Influence of rock bolt diameter on uniaxial compressive strength and shear strength



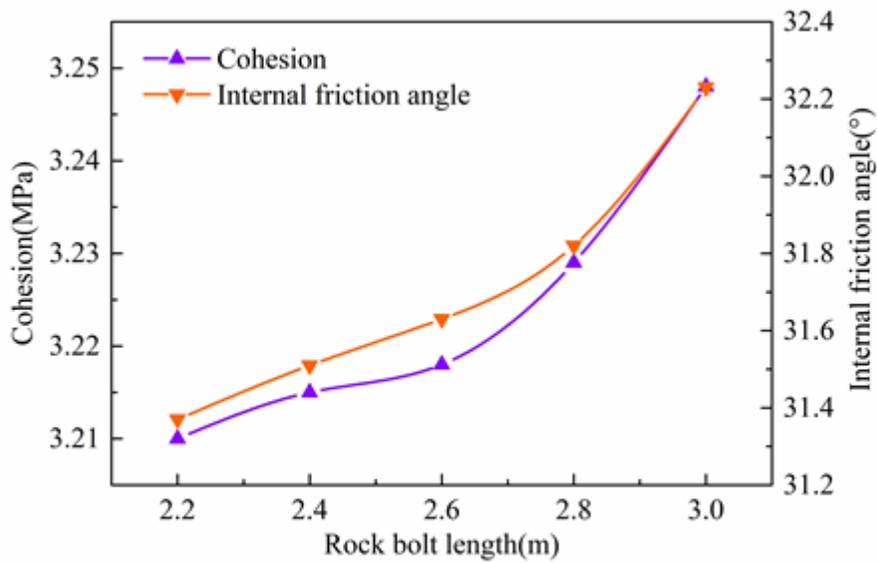
(b) Influence of rock bolt diameter on cohesion and internal friction angle

Figure 8

Influence of rock bolt diameter on mechanical properties of anchorage body



(a) Influence of rock bolt length on uniaxial compressive strength and shear strength



(b) Influence of rock bolt length on cohesion and internal friction angle

Figure 9

Influence of rock bolt length on mechanical properties of anchorage body

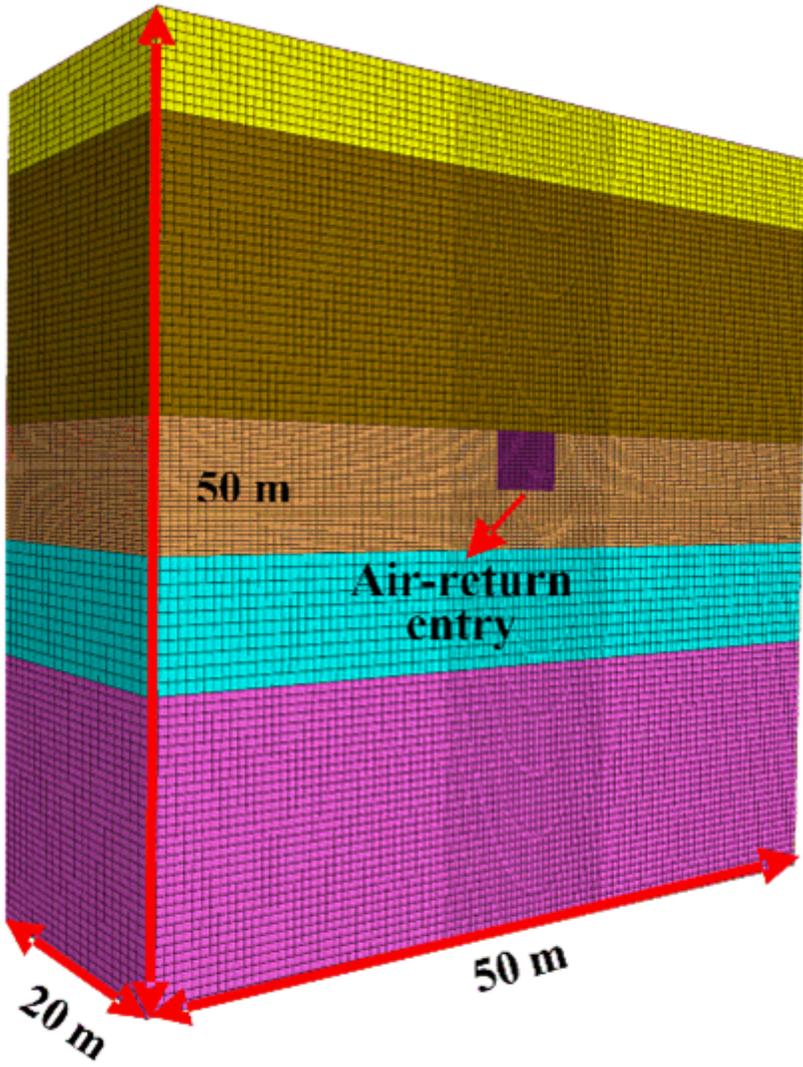


Figure 10

FLAC3D model used for research of anchor cable parameters

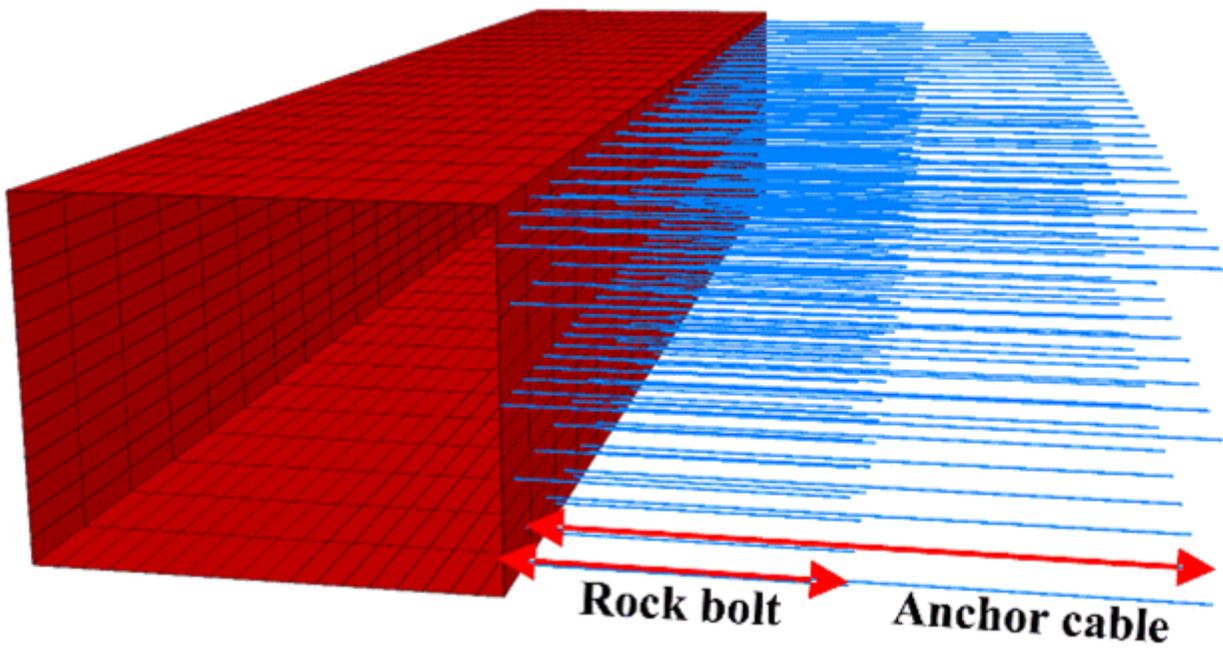
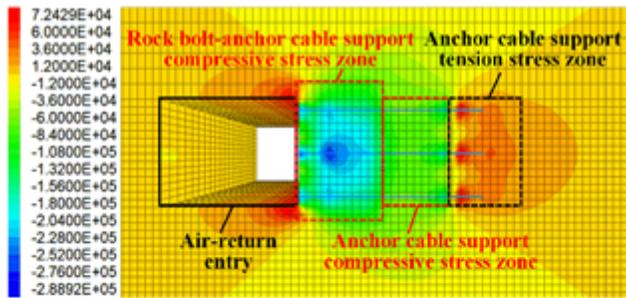
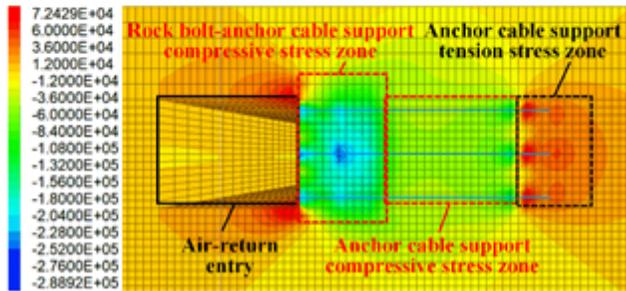


Figure 11

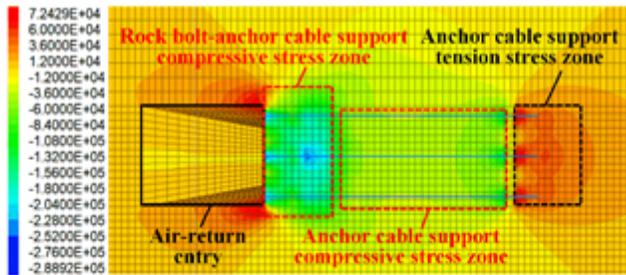
Side support structure of air-return entry



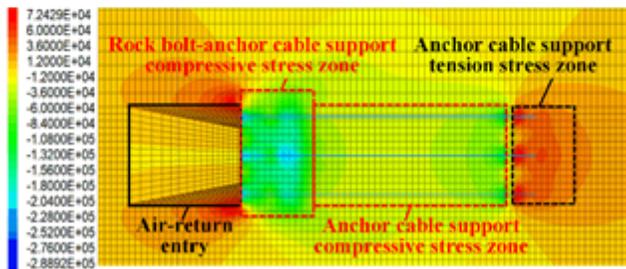
(a) Anchor cable length of 6.0 m



(b) Anchor cable length of 8.0 m



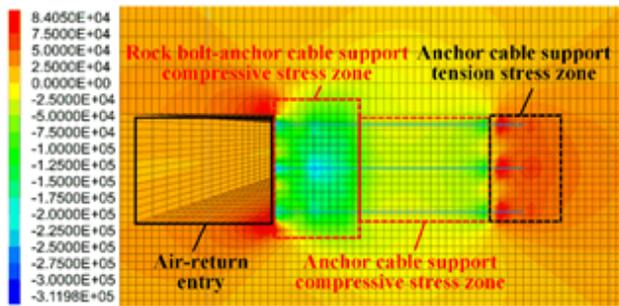
(c) Anchor cable length of 10.0 m



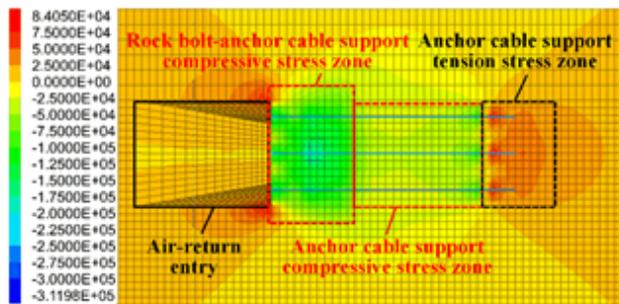
(d) Anchor cable length of 12.0 m

Figure 12

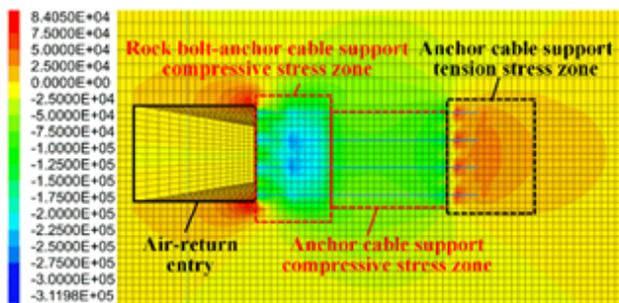
Horizontal support stress fields of different anchor cable lengths



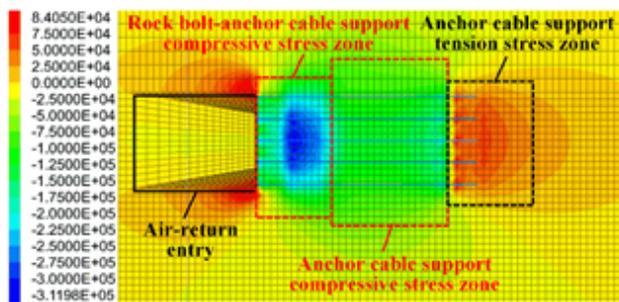
(a) Anchor cable spacing of 1.4 m



(b) Anchor cable spacing of 1.2 m



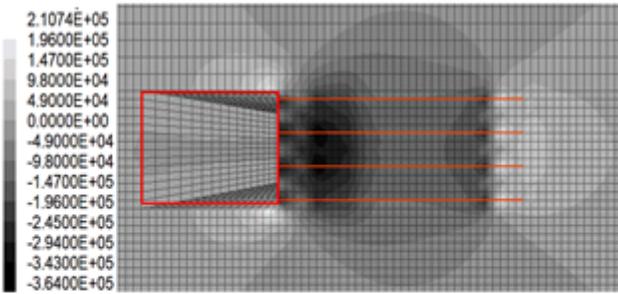
(c) Anchor cable spacing of 1.0 m



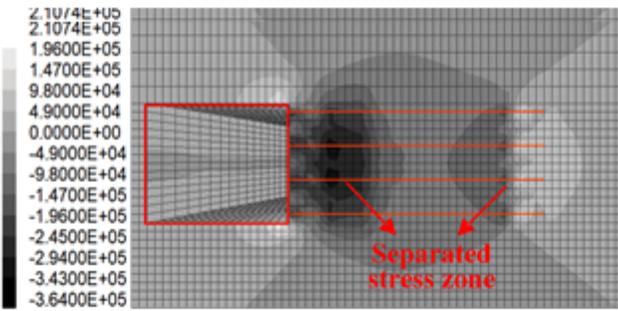
(d) Anchor cable spacing of 0.8 m

Figure 13

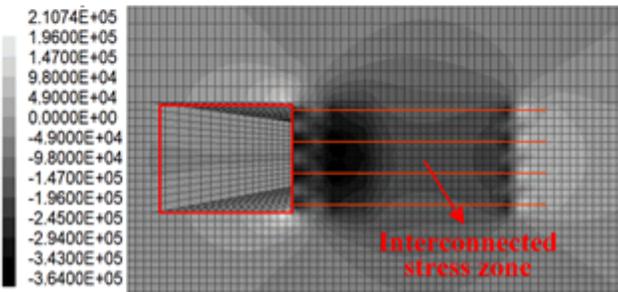
Horizontal support stress fields of different anchor cable spacing



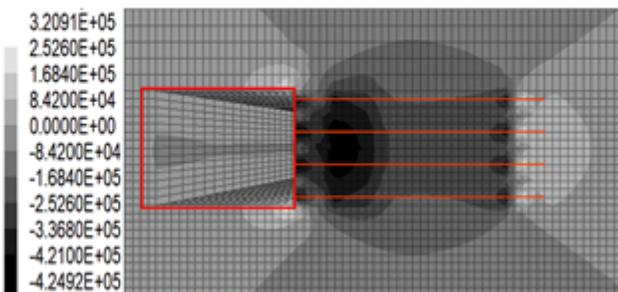
(a) Anchor cable pre-tightening of 150 kN



(b) Anchor cable pre-tightening of 200 kN



(c) Anchor cable pre-tightening of 250 kN



(d) Anchor cable pre-tightening of 300 kN

Figure 14

Horizontal support stress fields of different anchor cable pre-tightening force

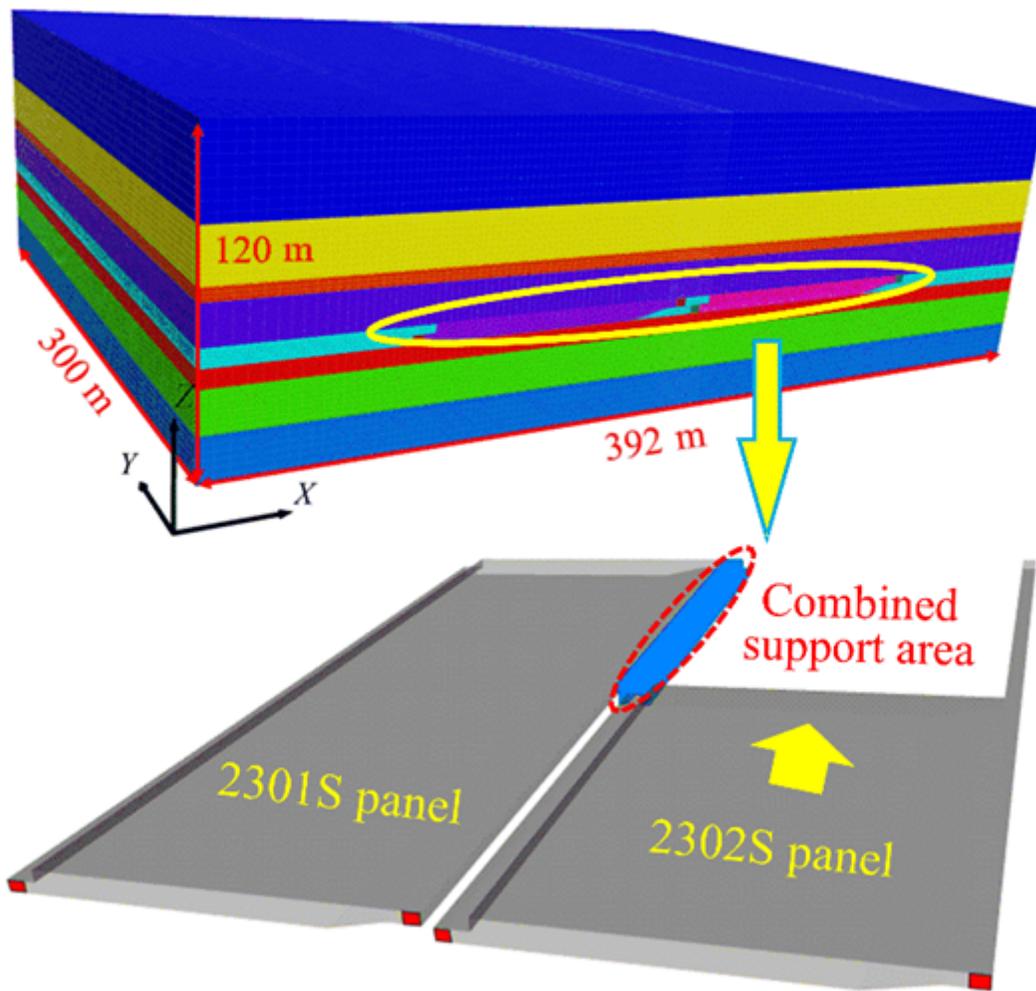


Figure 15

Numerical modelling of ESSPL

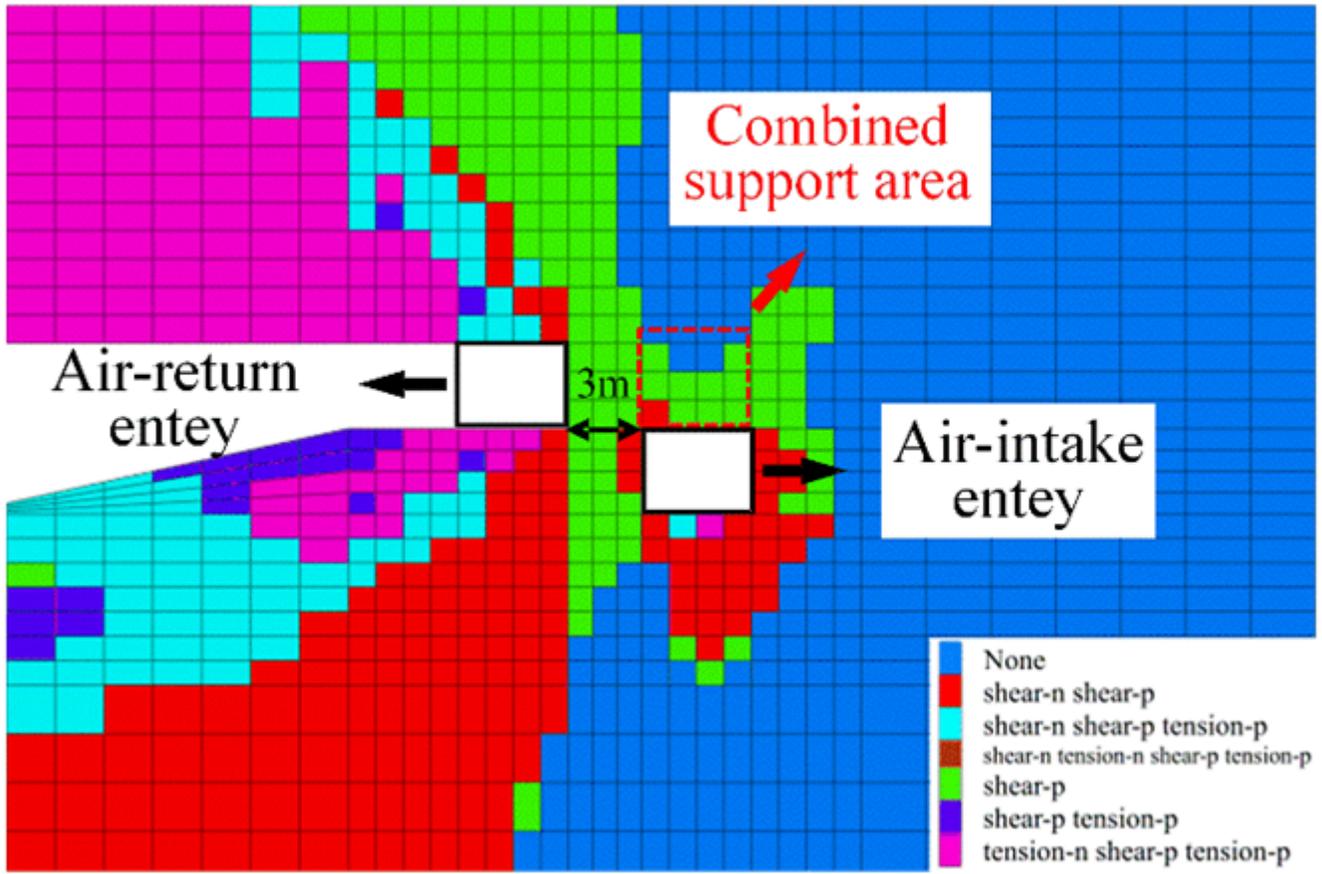


Figure 16

Plastic zone distribution of gob-side entry