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CONTROL TECHNOLOGY FOR SOFT ROCK ROADWAY IN INCLINED COAL SEAM: A CASE STUDY IN NUI BEO MINE, QUANG NINH, VIETNAM

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ABSTRACT: The large deformation of soft rock roadway is still a main concern for a large number of coal mines, particularly in terms of these located at the inclined seam with the large section. This paper presented a successful case study conducted at Nui Beo coal mine in Vietnam. Borehole peep detector was first put into the application to investigate the distribution of fractures to clarify the influenced range of surrounding rock. Then, the finite element analysis software (i.e. RS2) program is adapted to establish the numerical model aiming at exploring critical parameters for bolt support. Finally, a combined support technology including primary support by using a high prestressed bolt and reinforced technology by cable have been proposed and put into application. The results revealed that 1) the rheological strain is closely related to time, it is about 105 days from the firstly caving to be stable, 2) the maximum displacement of the ribs and roof-to-floor are 28mm and 47mmrespectively. The deformation of surrounding rock obtained from case study was used to evaluate the effectiveness of proposed support technology. The use of high prestressed bolt can maintain the stability of surrounding rock incline coal seam. Moreover, the experience can be used as a reference for other mines in Vietnam.

Keywords: Inclined seam; Soft rock; Asymmetric deformation; Prestressed bolt; RS2

1. INTRODUCTION

According to the coal industry development plan (2015-2030) published by Vietnam National Coal-Mineral Industrial Group, the production of black coal in Vietnam will be 70-75 million tons in 2020 and 120 million tons in 2030 to meet the basic requirement of the rapid development of the economic indicating that the black coal is still the main energy in Vietnam [1, 2]. Different from the open-cut mining, the effective and economic of which mainly depend on the depth of overburden, underground mining is becoming more and more popular and desirable for Quang Ninh Vietnam's coal industry due to sharply decline of shallow sources and restrict limit of the environment. However, during the past decades, a few studies focusing on the deformation control of surrounding rock in roadway have been conducted resulting in a large number of roof collapses and fatal accidents. Different from other countries where the bolt support technology for roadway has been widely used in practical application, there is a little successfully case study in terms of the application of bolt in roadway owing to the historical problems. N. Q. Phich.et al proposed the use of the bolt together with U-shape support to control the large deformation in soft rock roadway affected by the adjunct caving pressure [3]. The main limits for the widespread use of bolt in underground coal can be summarized as follows: 1) It is still a new technology for roadway support since that it was first introduced to Vietnam at the beginning of 1999 [4]. Meanwhile, traditional support technologies including U-shape support and I section support can meet the basic requirement for the use of roadway in shallow depth [5, 6]. 2) With the increase of mining depth, the geological conditions have been becoming more complex. Moreover, most of the coal resource is with deep inclined and soft rock accounting for approximate 50% in total [1-5].

Mentioned difficulties directly limit the application of bolt support technology in Vietnam. However, it is significantly necessary that not only the industries and the research institution should take their efforts to push forward the application of bolt support aiming at the construction of modern coal mine. Against the current situation that the large deformation of surrounding rock is still the main concern not only for Vietnam lacking sufficient experience but for other countries [10-14], this paper presented a case study of the bolt support which was successfully conducted in Nui Beo coal mine, Vietnam to provide a guideline for other coal mines in Vietnam. The research starts with the analysis of the geological conditions of the coal mine and some drawbacks of traditional support technology to find out the main reason for the large deformation, followed by the theoretical analysis according to static stress model and the burgers model to evaluate the deformation mechanics of surrounding rock. Then, a finite element (FE) analysis method, the namely RS2 software program was adopted to establish the inclined soft rock roadway model to investigate the effect of some critical parameters. Based on both the theoretical analysis and FE simulation, the authors of this paper proposed the combination use of bolt as primary support and cable as secondary support in practice. The case study revealed that the combinational support technology presented in this paper can sufficiently control the large deformation of soft rock roadway. It is believed that the successful case study can provide a guideline for other coal mines in Vietnam with similar geological conditions.

2.GEOLOGICAL AND MINING CONDITION

2.1 Introduction of Nui Beo Coal Mine

According to the strategic development of Vietnam's coal industry until 2015 with the review into 2025, which was approved by the Prime Minister in Decision No. 89/2008/QD-TTg dated 07/7/2008, Nui Beo Coal Mine topped as one of the new modern mines, which has an investment priorities in the future from The Vietnam National Coal-Minerals Industries Group.

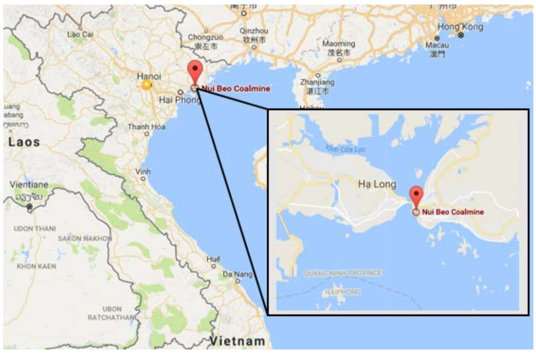


Fig. 1 Location of the research area

Nui Beo coal mine locates at the Ha Lam coal area with the designed production capacity of 2.0 Mt/a (Fig. 1). The age of the mining production area is 30 years starting from 2015. Nui Beo coal mine pit belongs to the Ha Lam coal area, 7 km north-east to Ha Long City-Quang Ninh. Mineral Coal Area is 5.6 Km2. The mine area is considered rich in coal resources, the density of geological exploration projects from the opening coal seam to -300 reaches to the detail prospecting, many areas of the open pit to reach the exploration point, part from -300 to -500 (the bottom of coal layer) reached a search in detail prospecting. The overburden strata range from 500 m to 700 m (average 540 m).

The thickness of the coal seam and the overlying strata changes significantly with the depth. In total, there are 14 layers of the coal seam which has been explored (Fig. 2). The seam 14 (10) is now caving by open-cut. According to the mining plan, coal 13 (9) and V11 (8) will be caving by open-cut as well. The seams 10 (7) V9 (6), V7 (4), V6 (3), and the rest of the V11 (8) is the main coal seams to mobilize the mining project. Existing research has revealed that the surrounding rock in Nui Beo coal mine is a typical engineering soft rock, the strength of which will be affected by moisture due to its physical properties [7-9]. Existing potential risks in terms of the stability of surrounding rock should be put more attention on.

Columnar	Thick-ness (m)	Depth (m)	Rock name	Description lithology
	12.7	60	Claystone	Dark gray, thin layered structure,15% mine stratigraphy, with dark gray. Often forming cliffs, head directly or mixed in the coal seam grip
	7.3	67.3	Packsand	Light gray, mainly composed of particles quartz, sustainable and very hard
	13.2	70.5	Siltstone	Ash gray, gray, mainly component clay minerals and fine-grained quartz.
	4.2	74.7	Claystone	Dark gray, thin layered structure,15% mine stratigraphy, with dark gray. Often forming cliffs, head directly or mixed in the coal seam grip
	15.8	100.5	Siltstone	Ash gray, gray, mainly component clay minerals and fine-grained quartz.
	14	114.4	Claystone	Dark gray, thin layered structure,15% mine stratigraphy, with dark gray. Often forming cliffs, head directly or mixed in the coal seam grip
	10.2	124.5	Siltstone	Ash gray, gray, mainly component clay minerals and fine-grained quartz.
	14	138.5	Claystone	Dark gray, thin layered structure,15% mine stratigraphy, with dark gray. Often forming cliffs, head directly or mixed in the coal seam grip
	3.5	142	Coal	Simple structure, black, shiny, small slope angle
	10	152	Claystone	Dark gray, thin layered structure, 15% mine stratigraphy, with dark gray
	10.7	162.7	Siltstone	Ash gray, gray, mainly component clay minerals and fine-grained quartz.
	7.3	170	Packsand	Light gray, mainly composed of particles quartz, sustainable and very hard

Fig. 2 Typical stratigraphic column

2.2 Traditional Support For Surrounding Rock

The experimental roadway is dragged under the floor of 10# coal with the distance of 8-15 m, the average elevation of the roadway is at -140 m level. To investigate the difference between traditional support technology (i.e. U-shape support) and bolt support technology, these two support technologies have been applied in two sections at the same roadway. The length of experimental roadway is 1500 m, with the width of 6100 mm and height of 4850 mm. The height of the arch and the straight wall are 3050 mm and 1800 mm.

Fig. 3 The outline of the roadway with traditional support

The original support for surrounding roadway is typical U-shaped steel shrinkage bracket, the back reinforcement net of the bracket adopts Φ 6 mm round bar welding or reinforced concrete slab, the bracket is 0.7 m. As shown in Fig. 3, from which the deformation of surrounding rock and the corrosion of the steel support have been observed. Considering that the service time of the roadway is about 30 years, traditional support technology should be evaluated to make sure the safety mining. Moreover, compared with bolt support ease-install, the potential manual risk is another concern for U-shape support. Therefore, it is emergent to take some technical measures to control the deformation.

2.3 Rock Mass Properties

The rock-mass detection recorder (GD3Q-GA) was used to detect the lithology and the fissure of roof surrounding rock. The vertical borehole with the aperture of 65 mm and the depth of 10 m was drilled ahead with the distance of 10 m.

It is believed that with the increase of depth, the stability of the surrounding rock seems better than that in shallow depth. In detail, there exist lots of network fissures ranging from 0.5 to 1.5 m. With the increase of the depth, the surrounding rock is characteristic with cut- fracture located at approximate 2.0 m. Then, both the number and the size of cracks gradually reduced out of 2.0 m even though the damage of the borehole has been observed. It is the pretty integrity of the surrounding rock over the depth of 2.4 m. Moreover, no obvious cracks and fracture were observed on the surface of the borehole beyond 4 m, indicating that the influence zone ranges from the surface of the roadway to 2.4 m.

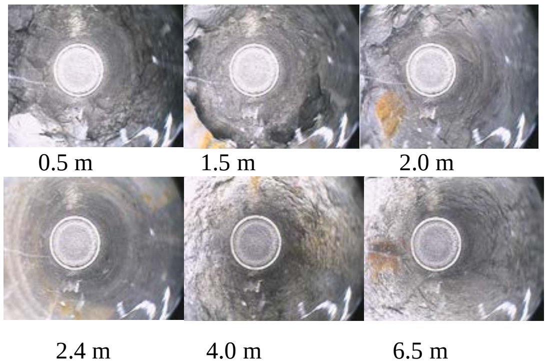


Fig. 4 Peeping at roof drilling

3. ANALYSIS MODEL FOR SOFT ROCK ROADWAY WITH ASYMMETRIC DEFORMATION

3.1 Analysis Model For Roadway In Inclined Coal Seam

According to the geological properties and the layout of the experimental roadway, the simplified stress analysis model was proposed as shown in Fig. 5. In Fig. 5, the σ1 was defined as the vertical stress and σ3 is the horizon stress which can be obtained from equations (1-2) where γ is the bulk weight, ν representing Poisson ratio, z is the height of the overburden strata.

$$\sigma 1 = \rho.g.z = \gamma.z = 2500\times 10\times 140 = 4.5\text{MPa} \quad (1)$$

$$\sigma_3 = \frac{\nu}{1-\nu}\sigma_1 \quad (2)$$

Herein, γ = 2500, z = 140 m and ν = 0.2.

The law describing the stress distribution is expressed mathematically as:

$$\sigma_1 = 2500\times 10\times 140 = 4.5\text{MPa}$$

$$\sigma_3 = \frac{0.2}{1-0.2}\times 3.5 = 1.125 \text{ MPa}$$

$$\sigma_1^{'} = \sigma_1 \cos 35^{\circ} + \sigma_3 \sin 35^{\circ} \quad (3)$$

$$\sigma_3^{'} = \sigma_1 \sin 35^{\circ} + \sigma_3 \cos 35^{\circ} \quad (4)$$

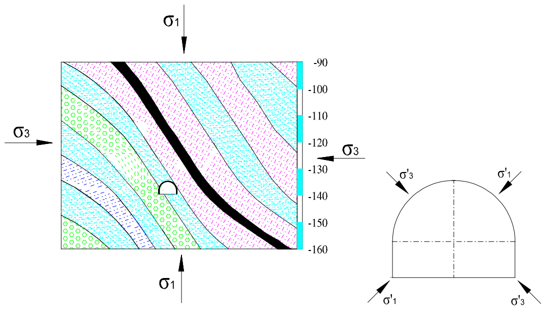


Fig. 5 The stress analysis model for the roadway

It is apparent that the vertical stress of the roadway is much larger than that on the horizon. The right shoulder and the left corner are affected by asymmetric force significantly followed by the left shoulder and the right corner which are relatively smaller.

3.2 Burgers Model For Creep Behavior

For engineering soft rock characteristic with creep behavior, it is necessary to consider the relationship between the rheology and time to maintain the stability of surrounding rock during the designing period. Considering the large deformation of the roadway and the service period of the experimental roadway in Nui Beo coal mine, in present research, it was assumed that the law of the creep behavior in Nui Beo mine is under the control of Burgers model which is composed of Maxwell body and Kelvin body owing to that this model is suitable for fitting the rheological curve of soft rock under lower stress [15].

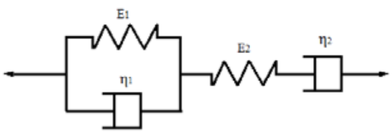


Fig. 6 Burgers model

The relationship between the strain and some critical parameters including time is expressed by following equation [15].

$$\varepsilon = \frac{\sigma_0}{\eta_1} + \frac{\sigma_0}{E_1} + \frac{\sigma_0}{E_2} - \frac{\sigma_0}{E_2} \exp\left(-\frac{E_2}{\eta_2} t\right) \quad (5)$$

Where, σ_0 In-situ stress, MPa; σ_0 = 4.5 MPa. E_1 Elastic modules of mudstone, MPa; E1 = 2875 MPa. E_2 Elastic modules of coal, MPa; E2 = 650 MPa. η_1 Viscoelastic Coefficient of mudstone, MPa.h; η_1 = 8380 MPa.h. η_2 Viscoelastic Coefficient of coal, MPa.h; η_2 = 35539 MPa.h.

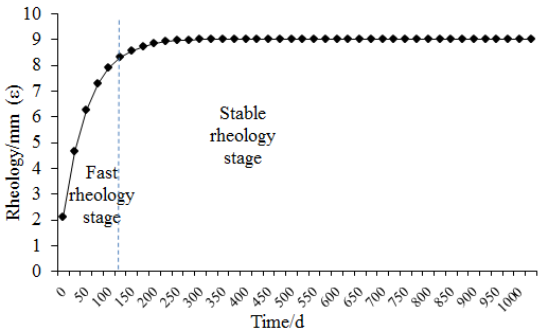


Fig. 7 Relationship between Surrounding rock displacement and time

It can be seen from Fig. 7. The rheological characteristics of the soft rock in the inclined coal seam are proportional to the time, and the rheological deformation begins in 10 days after the excavation of the roadway. With the increase of the time, the surrounding rock gradually becomes stable. It is about 105 days that the surrounding rock to be calmed. Correspondingly, the maximum rheological deformation is 9,02 mm. After 105 days of excavation, no obvious rheology can be observed. From the above observation, the misuse of rigid support rather than flexible support structure will not play its expected role in practice.

4. FINITE ELEMENT SIMULATION BY RS2

4.1 The Simulation Model

In this section, the simulation results obtained from RS2 (i.e. Rock and Soil 2-dimensional analysis program) are presented due to it is believed to be an effective method for support design which has been widely used in mining engineering, especially in terms of the deformation mechanics of surrounding rock [16-18].

The length and the height of the numeric model are 88 m and 70 mm respectively. The top boundary of the model is applied to the weight of overlying strata, the bottom boundary is fixed. The left and right boundary are fixed horizontally to simulate the real application. The preload vertical stress on the top of the model is applied according to the depth of 80 m. The roadway is symmetrical to the straight wall semicircle arch, the net width and height of the roadway are 6.1 m and 4.85 m, respectively.

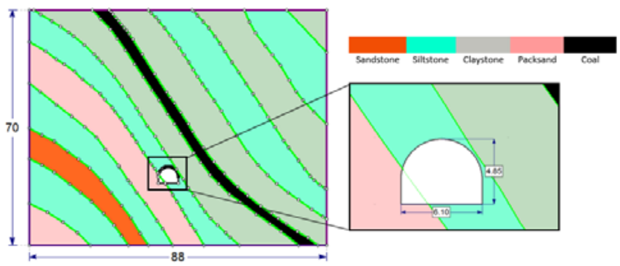


Fig. 8 Inclined rock strata roadway model

The mechanical parameters of rock strata are shown in Table 1.

Table 1 Mechanical parameters of rock mass

Stratum	U-W (kg·m-3)	Y-M E (GPa)	G S I	U-C-S (MPa)	Cohesion C (MPa)	F-A (°)
Sand-stone	2650	5.3	55	73.3	2.8	35
Siltst-one	2600	8.5	50	41.5	2.4	30
Coal	1400	1.4	20	20	0.7	22
Packs-and	2700	8.8	65	85.6	3.0	38
Clays-tone	2520	3.7	40	26.8	2.1	26

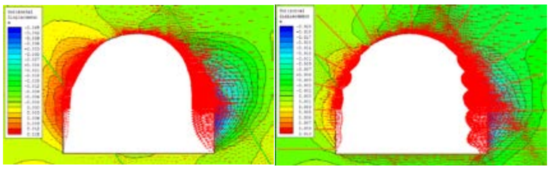
Considering the layout of the roadway located in inclined coal seam as shown in Fig. 8, three simulation schemes were proposed for simulation. 1) the first one is as reference that not any supports applied to the surrounding rock of the roadway; 2) the symmetrical bolt support was applied to the surrounding rock without the consideration of the influence caused by the inclined coal seam and 3) extra cables were anchored to one rib which nearby the coal seam as strengthening method to build up the asymmetrical support.

To clarify the difference between plastic zone distribution and deformation regularity of the experimental roadway, critical parameters, and different support schemes were selected to simulate in the present study. These critical parameters including the length, diameter and the space of the bolt and cable in finite element analysis. By comparing the primary analysis and considering the application conditions, finally, the proper parameters of the support were selected as follow: the bolt is Φ 22 x 2400 mm anchor rod with the space of 800 mm, the anchor cable is Φ 17.8 x 7300 mm spacing 1800 mm.

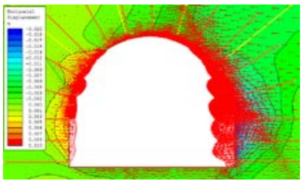
To investigate the deformation mechanics of surrounding rock and the effects of different support schemes, displacement vector, plastic zone and the deformation of surrounding rock was carefully studied. More detailed information can be found in following sections.

4.2 The deformation characteristics of different support methods

The simulation results showed that the asymmetry displacement of the surrounding rock is mainly caused by the influence of inclined coal seam. From the horizontal displacement distribution map, it is obvious that (Fig. 9) the right rib is significantly greater than that in the left for the roadway without support.



a) Without support, b) Support by bolts

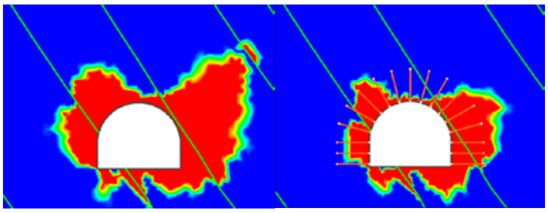


c) Support for bolts and cables

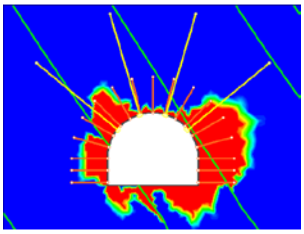
Fig. 9 Displacement vector diagram of the surrounding rocks

The maximum displacements in different ribs are 45 mm in the right and 15 mm in the left. While using symmetrical anchor bolt and cable support scheme in the roadway, the displacement of surrounding rock has been controlled to some extent, the maximum displacement of the right rib is only 20 mm, and the maximum displacement of the left rib is under 10 mm. The right displacement amount is equal to 62.5 % when the symmetrical anchor cable support scheme is adopted. Not supported, seriously affect the stability of the roadway. However, the surrounding rock still experiences obviously asymmetric deformation, indicating that traditional symmetrical support technology cannot sufficiently control the deformation in this situation.

As shown in Fig. 10, the distribution of the plastic zone is seriously asymmetric due to the influence caused by the inclination of surrounding rock. With different support schemes, the distribution pattern of the plastic zone in surrounding rock roadway is very different. The plastic zone of the roof, floor and two ribs of the roadway reduced significantly owing to the reinforced support technology. Especially, the size of the plastic zone in two ribs have been reduced sharply. It indicated that the combination of the anchor bolt and cable supporting method can control the deformation of surrounding rock and to ensure the stability of the surrounding rock by using.



a) Without support, b) Support by bolts



b) Support for bolts and cables

Fig. 10 Plastic zone distribution of roadway surrounding rock with different support

4.3 Asymmetric deformation characteristics with different supporting schemes

To obtain the asymmetrical deformation characteristics of the deep surrounding rock in the inclined seam roadway, various monitoring points are attached to the deep surrounding rock of the roadway. In the two-shoulder of the arch and the roof, 3 positions with different depth points (0, 1.5 m, 3.5 m, 5.0 m, 7.5 m) were selected for monitoring. The displacement of the roof is monitored by point A#. B# and C# Point were used to investigate the displacement of the two ribs, the layout of the monitoring point is shown in Fig. 11.

Fig. 12 and Fig. 13 are the overall displacement cloud picture and overall displacement of surrounding rock at the different monitoring site. From Fig. 12-13, it can be found that the displacement of the surrounding rock is asymmetric due to the influence of strata formation, the total displacement of roadway roof is greater than that of two-help, and the displacement of C# monitoring helps is larger than that of B# point monitoring. The displacement of roadway surrounding rock is different when different support schemes are adopted, and the displacement of the measured point from 0 m to 7.5 m is gradually decreased.

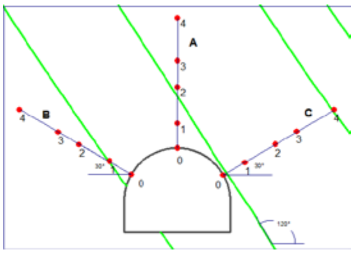
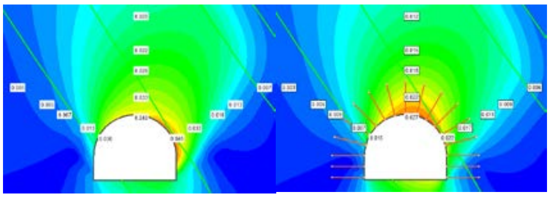
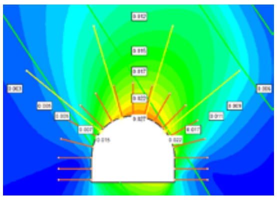


Fig. 11 Monitoring points arrangement



a) Without support, b) Support by bolts



c) Support for bolts and cables

Fig. 12 Displacement cloud of roadway deep part surrounding rock with different supports

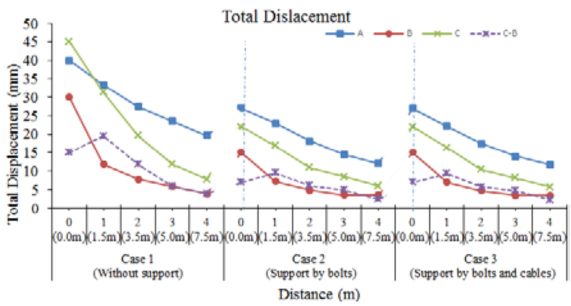


Fig. 13 Total displacements of surrounding rock with different supports

It is obvious that for surrounding rock without support, the overall displacement at the surface and the depth of 7.5 m are 40 mm and 20 mm observed from point A#, respectively. However, the same data obtained from point B# are 45 mm and 7.8 mm in total. For the other rib, corresponding to point B#, the displacement obtained from point C#, the overall displacement at the surface and the depth of 7.5m are 30 mm and 3.9 mm.

By contract, the displacement of surrounding rock seems to be changed a lot. The overall displacement of surrounding rock at the surface and the depth of 7.5 m at different monitoring points, i.e. point A# (27 mm/11.7 mm), point B# (15 mm/3.5 mm) and point C# (22 mm/5.8 mm), are significantly different from that without supports. However, the difference of displacement observed in the deep site, namely at the position of 7.5 m, is not obvious due to the relative stability of surrounding rock in depth.

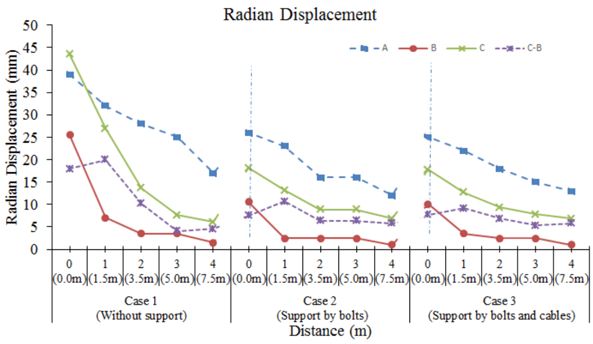


Fig.14 Radian displacement of surrounding rock with different supports

Fig. 14 is the overall radian displacement of surrounding rock, from which it can be seen that the radial displacement of surrounding rock is smaller than that of overall displacement shown in Fig. 13. Moreover, the displacement observed from the surface and at the depth of 7.5 m experience the same trend.

5. CASE STUDY

According to mentioned theoretical analysis, FE simulation, the combined support approach was proposed to put into the application to evaluate the supporting technology presented earlier.

5.1 Key parameters for support design

5.1.1 Roof support

The rebar bolt with Φ 22 x 2400 mm was installed with the space of 800 mm in a row. All bolts were fixed with the surrounding rock by CK2335-CK2350 resin roll. Steel pallet with the size of 150 x 150 x 10 mm plus steel tray with Φ 100 mm was used together with the bolts. The preload of the bolt is not less than 60 KN.

The round steel bar with Φ 12 mm as beam used together with the steel mesh manufactured with Φ 6, 2400 x 900 mm steel bar. The cable with Φ 17.8 x 7000 mm made of high strength prestressed anchor rope was used as secondary standing support. The row spacing and column spacing were 1800 mm and 2400 mm, respectively. Four resin rolls were used for each cable. Steel pallet with the size of 300 x 300 x 50 mm was taken as well. The preload of the cable is not less than 150 KN.

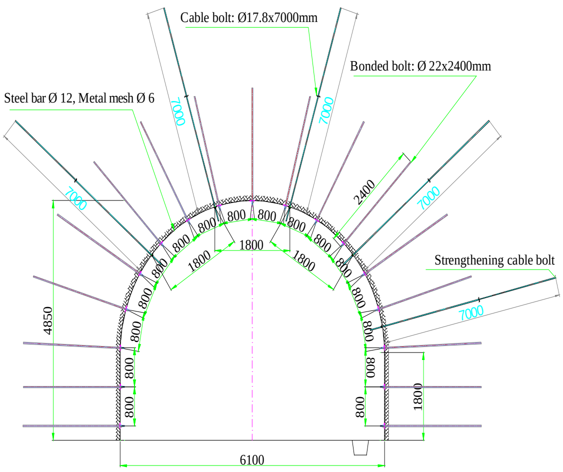
5.1.2 Rib support

As discussed earlier, different support parameters for two ribs were selected to control the asymmetric deformation of surrounding rock, correspondingly, more detail information can be found as follows: (1) Strengthening rib

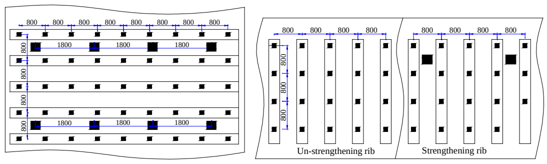
The rebar bolt with Φ 22 x 2400 mm was installed with the space of 800 mm in a row. All bolts were fixed with the surrounding rock by CK2335-CK2350 resin roll. Steel pallet with the size of 150 x 150 x 10 mm plus steel tray with Φ 100 mm was used together with the bolts. The round steel bar with Φ 12 mm as beam used together with the steel mesh manufactured with Φ 6, 2400 x 900 mm steel bar. The cable with Φ 17.8 x 7000 mm made of high strength prestressed anchor rope was used as secondary standing support. The row spacing and column spacing were 1800 mm and 2400 mm, respectively. Four resin rolls were used for each cable. Steel pallet with the size of 300 x 300 x 50 mm was taken as well. The preload of bolt and cable are not less than 60 KN and 110 KN, respectively.

(2) Un-strengthening rib

Different from another rib, all parameters kept at the same as the strengthening rib except for the cable. In addition, after the complete installment of bolts and cables, the sprayed concrete surface was construed to maintain the closure of the surrounding rock to prevent the creep of the surrounding rock.



(a) Cross section of the roadway



(b) Roof support (c) Rib support

Fig. 15 Roadway supported with bolt and reinforced with cable

5.2 Site monitoring

To verify the control effect of the main roadway, the displacement of the roadway surface is observed during tunneling. From continuous monitoring, the deformation of surrounding rock can be divided into three stages (Fig. 16).

Stage I (0-25 days): It is apparent that the speed of deformation developed sharply during the first 25 days. The total deformation of the roof is 41 mm, and the total deformation of the two ribs is 24 mm. The total deformation of the two is equal to the total deformation of 41.4 % roof.

Stage II (26-45 days): The trend did not start to change until 45 days, during which the deformation of the surrounding rock kept a stable slow increasing. The total deformation of the roof is 47mm, and the total deformation of the two is 28 mm.

Stage II (46-ongoing): The deformation of surrounding rock can be regarded as stale after 46 days even though the deformation speed of the top floor is still slightly larger than that of the two ribs

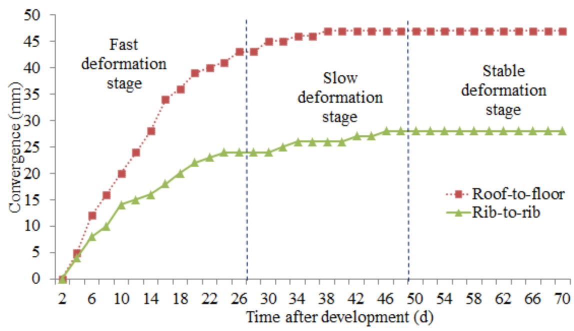


Fig. 16 Displacement-time curves of gateway surrounding rock

6. CONCLUSIONS

A case study on the stability of soft rock roadway in Nui Beo coal mine, Vietnam was carried out. The mine is one of the new modern coal mines operated by Vietnam National Coal-Minerals Industries Group. Due to the lack of industrial experience in bolt support technology during past decades, the roadway typical with its soft rock and inclined coal seam have experienced large deformation and severe creep behavior of surrounding rock.

The rock proprieties obtained by borehole video as part of this research show that there are lots of fractures existing in surrounding rock and the depth of the loose area is about 2 meters. In practices, it is significant to take some measures to prevent the development of the loose area.

Theoretical analysis results show that the maximum principal stress of the in-situ force is 4.33 MPa and the minimum principal stress is 3.50 MPa, the creep starts from 10 days after excavation and this period will last about 105 days.

The numerical simulation results obtained from FE software (i.e. RS2) show that the deformation of roadway in the inclined seam is not homogeneous. Particularly, the plastic zone and deformation of roadway surrounding rock mainly occur in roadway surrounding rock tendency. Moreover, asymmetry of roadway deformation in the inclined seam is mainly concentrated in the shoulder corner of the ribs. Correspondingly, the asymmetric support method should be taken to meet the basic requirement of the deformation mechanics of surrounding rock.

Engineering application verifies proposed support technology by using a bolt as primary support and cable as secondary support is a sufficient technology. Correspondingly, these critical parameters obtained from theoretical analysis and FE simulation are reasonable. The successful case study can push forward the use of bolt support for other coal mines in Vietnam with similar geological conditions.

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