



Research

# Study on deformation and instability characteristics and control technology of deep soft rock roadway with strong mining

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## Abstract

With the development of deep mining of coal resources, the problem of deep roadway management is becoming increasingly prominent. This study aims to analyze the deformation and failure mechanism of deep-buried soft rock roadway under the influence of mining. Taking the material roadway of 3205 working face in Xin'an Coal Mine as the engineering background, this study analyzed the characteristics of roadway deformation and failure and its influencing factors through field investigation. This study firstly expounded the principles of graded continuous anchorage for roadway roof and then proposed a new support scheme whose feasibility was verified by numerical simulation. Field verification shows that the optimized support scheme can effectively control the deformation of roadway surrounding rock. Specifically, the deformation of roof and two sides is reduced by 91% and 50% respectively; the maximum crack depth decreases from 9.56 to 3.26 m, indicating that the roof crack development has been significantly inhibited. The new support scheme builds a solid anchored rock beam structure with thick roof layer, which can maintain the long-term stable bearing capacity of deep-buried composite roof roadway. This study provides a solution for the surrounding rock control of deep soft rock roadway.

## Article Highlights

A typical deep-buried strong mining-induced soft rock roadway survey and monitoring have been conducted, and the deformation and failure patterns of the roadway surrounding rock have been analyzed.

The principle of graded continuous anchoring and its complete set of technologies have been developed, achieving structured support for fractured surrounding rock.

A variety of monitoring methods were employed to conduct a systematic evaluation and analysis of the new support technology.

**Keywords** Roadway · Deep soft rock · Graded continuous anchorage · Grouting

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## 1 Introduction

Coal resources with a burial depth of more than 1000 m in China account for more than 50% of the total resources [1]. With the depletion of shallow coal resource, deep mining has become normal in coal resource development [2–5]. Currently, the mining depth is increasing rapidly at a rate of 10–25 m per year [6]. Deep roadway is featured with high stress, high seepage pressure, high temperature, and strong disturbance. Under this circumstance, roadway excavation would inevitably face a highly complex stress environment [7, 8]. With a high degree of deterioration, the surrounding rock is prone to have large and irreversible deformation under mining disturbance. The safety and stability control of deep soft rock roadways is one of the main technical problems restricting deep coal mining. Such roadways display strong rheological characteristics after excavation [9]. Without effective support, the roadway expansion and deformation will continue to develop, which pushes up the maintenance cost of roadways and makes it hard to release the excavation capacity.

At present, significant research achievements have been made on the stability control of deep roadway surrounding rock as a result of long-time commitment from scholars both at home and abroad. Kang et al. proposed the high prestressed strong support system and maintained that the deformation and failure of surrounding rock in fault area can be effectively controlled by improving the support strength and stiffness at the initial stage of support [10]. Zhang et al. put forward the continuous beam roof control theory and efficient long anchoring support technology and applied it widely to deep coal roadway with large section composite roof, which greatly reduced the development of surrounding rock cracks [11]. He et al. put forward the coupling support strategy of key positions and developed the constant resistance large deformation anchor cable which played a good control effect for deep composite roof roadway [12]. Through comprehensive field investigation and numerical model analysis, Li et al. evaluated the influencing factors, law and processes of deformation, and failure modes of deep soft rock roadway and proposed the improved combined support scheme of double-layer long anchor net spraying and concrete-filled steel tube to effectively control roadway deformation [13]. By means of field investigation and numerical simulation, Liu et al. revealed the characteristics and mechanism of mining abnormal deformation and failure of deep high stress roadway and pointed out the importance of supporting structure parameters for roadway maintenance and control [14]. Sun et al. established a three-dimensional time-varying numerical model of the

CVISC constitutive model and discovered the rheological characteristics of the surrounding rock of deep coal roadway [15]. On this basis, they proposed the grouting technology such as high-pressure jet grouting and delayed grouting to restrain the coal rheological. Although some achievements have been made in terms of the treatment of deep soft rock roadway, the deformation and failure mechanism of this kind of roadway is still unclear. Moreover, the geological occurrence of different mining areas means significantly different mining environments. Accordingly, much still needs to be done on the surrounding rock control of deep soft rock roadway.

Therefore, with the material roadway of 3205 working face in Xin'an Coal Mine as research setting, this study analyzed the deformation and failure mechanism of surrounding rock of deep soft rock roadway, and expounded the principle and support technology of graded continuous anchoring of roadway roof. On this basis, it proposed a new support scheme to provide reference for the engineering treatment of similar deep soft rock roadway.

## 2 Engineering geology overview

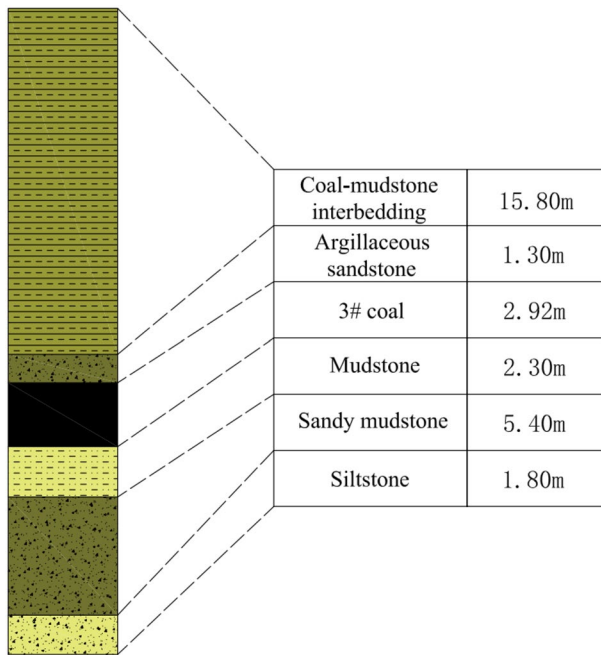
### 2.1 The overview of roadway

Xin'an Coal Mine is located in Pingliang City, Gansu Province, China, with an annual output of 1.2 million tons of coal. At present, the maximum mining depth of coal in this mine has reached 800 m, and the high ground stress and strong mining disturbance induced by deep mining have caused the continuous large deformation of the roadway. Accordingly, repeated repair and restoration is highly common, which seriously restricts the safe and efficient mining of coal mine. The material roadway of 3205 working face is a typical large-deformation roadway affected by deep mining. The coal seam of this roadway is 3# coal seam, with an average burial depth of 800 m, average thickness of 2.92 m, and dip angle of 3°–16°. Three working faces, namely 1203, 1205, and 1207, are located 50 m vertically above the 3205 working face, and 40 m coal pillars are left between the working faces, all of which have been mined. According to the geological data of the mine, No. 3 coal seam is a single synclinal structure, with the west wing dip angle of 6°–8° and the east wing dip angle of 8°–35°.

The section design of the material roadway of 3205 working face is a three-center arch section. The specific parameters are as follows: roadway width: 5.0 m, roadway height: 3.2 m; arch height: 1.25 m; straight wall height: 1.95 m; and roadway section area: 14.7 m<sup>2</sup>. The roof of coal seam is mainly composed of multiple layers of mudstone

**Table 1** Characteristics of geological occurrence

Lithology	Thickness/m	Petrographic description
Coal-mudstone interbedding	15.80	Low strength, gray-black
Argillaceous sandstone	1.30	Gray-white sandstone, argillaceous cement, wavy bedding development, the bottom 0.3 m being gray-black mudstone
3# coal	2.92	Black lump and powder
Mudstone	2.30	Deep gray- gray black mudstone, lump
Sandy mudstone	5.40	Gray sandy stone, lump, dense and hard rock mass
Siltstone	1.80	Gray siltstone whose composition is mainly quartz, and the argillaceous cement is dense

**Fig. 1** Characteristics of geological occurrence

and coal seams, which is featured with low strength, weak cementation, and easy weathering. The floor mainly comprises sandy mudstone and mudstone, which is a typical soft rock roadway. The specific geological occurrence is shown in Table 1 and Fig. 1.

## 2.2 The original support scheme

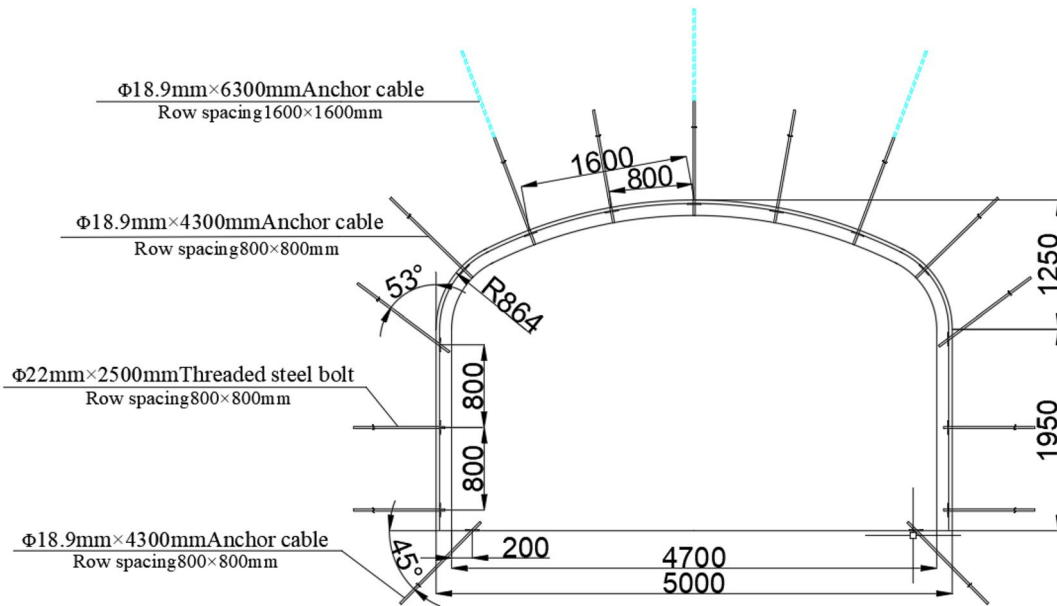
The original support scheme of the material roadway of the 3205 working face adopts the combined support of anchor cable-net-beam. The roof and the shoulder sockets of two sides are supported by  $\Phi 18.9 \times 4300$  mm anchor cables, which are arranged perpendicular to the roof outline, and the row spacing is  $800 \times 800$  mm, with 9 cables in each row. The roadway walls are supported by  $\Phi 22 \times 2500$  mm

rebar bolts, which are arranged perpendicular to the lane wall, with 4 bolts in each row. The bottom corners of the two sides are supported by  $\Phi 18.9 \times 4300$  mm anchor cables, which are arranged downwards at an inclined angle of  $45^\circ$ , with two cables in each row. The roof support is reinforced: the  $\Phi 18.9 \times 6300$  mm anchor cables are used for support, with a row spacing of  $1600 \times 1600$  mm, and 3 cables in each row. Specific supporting parameters are shown in Fig. 2.

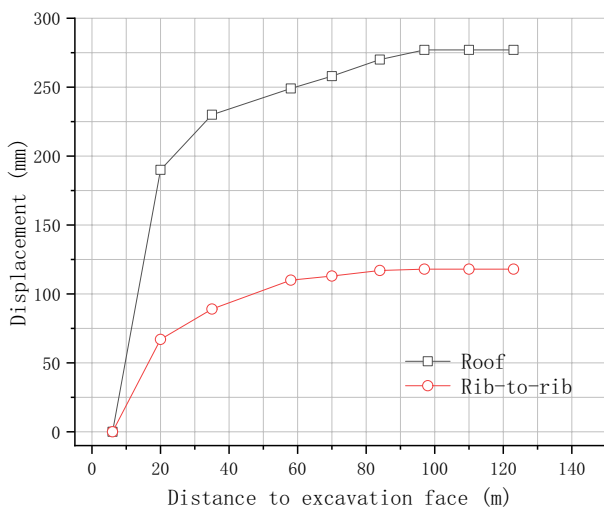
## 3 Analysis of roadway deformation and failure characteristics and influencing factors

### 3.1 The roadway deformation and failure characteristics

- (1) The surrounding rock of the material roadway of 3205 working face is seriously deteriorated. To be specific, large deformation occurred in the roadway, which is manifested by severe roof subsidence, side convergence and floor heave. The monitoring results of roadway surface displacement show that the roof subsidence and the convergence of the two sides display the same variation trend. The roadway deformation rate is rather high in the early stage of roadway excavation. The roof subsidence reaches 68% of the final deformation when the excavation advances by only 20 m. Meanwhile, the side convergence reaches 55% of the final deformation, indicating that the existing support scheme cannot realize the safety and stability control of roadway in the early stage of excavation. With the advance of the driving face, the roadway deformation rate tends to decrease. The roadway deformation gradually becomes stable until the monitoring point is 100 m away from the driving face. The final deformation of the roof and the sides



**Fig. 2** Original support scheme of 3205 working face



**Fig. 3** Deformation measurement under original support

are 277 mm and 118 mm, respectively, as is shown in Fig. 3.

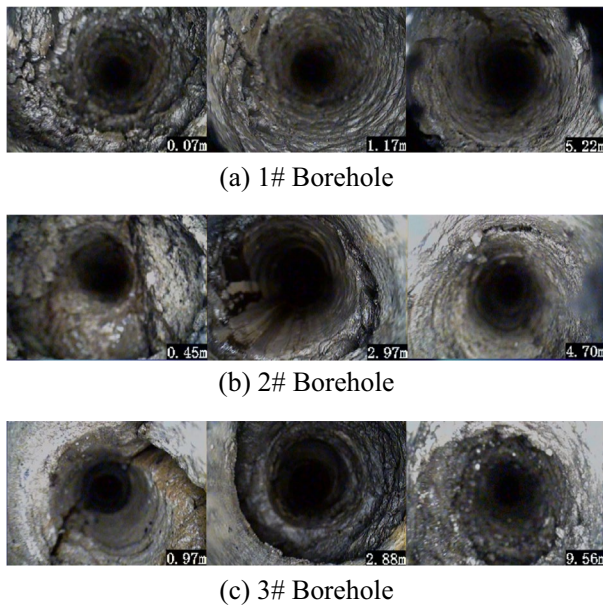
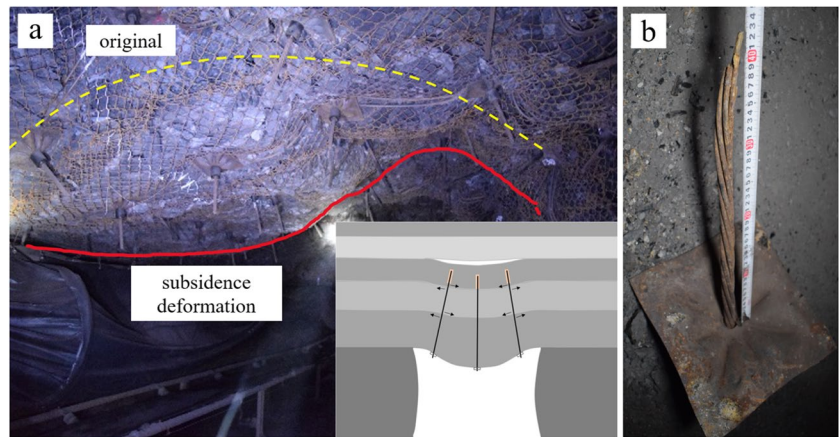
Field investigation shows that the material roadway roof of 3205 working face is abnormally broken and significantly affected by weathering. The original semi-circular arch section has become concave under the action of stress. In order to ensure the safe production of coal mining, the roadway needs to be repaired from time to time, as shown in Fig. 4a. The roadway deformation failure is mainly manifested by the roof subsidence deformation, which is consistent with the monitoring

results of roadway surface displacement mentioned above. As the broken roof extrudes into the roadway to form a large number of net bags and concaves, the anchoring structure sinks along with the roof as a whole in this process. When the subsidence reaches a certain extent, discontinuous surface is generated above the anchoring area. In this case, the stress cannot be continuously transferred, leading to the overall anchoring failure. Meanwhile, when the roof is sunk, the layered rock mass produces uncoordinated deformation and mutual dislocation, resulting in higher shear stress on the anchoring structure, even cutting the anchor cable, as shown in Fig. 4b.

- (2) In order to analyze the deformation and failure range of the roadway surrounding rock and the support effect of the original support scheme, a borehole scope was used to characterize the development of strata cracks in the roadway roof. Three boreholes, 1#, 2#, and 3#, were set in the middle of the roadway roof at a distance of 13 m, 221 m, and 435 m from the head, respectively. The borehole depth was all 10 m. Figure 5 shows the borehole camera images under original support.

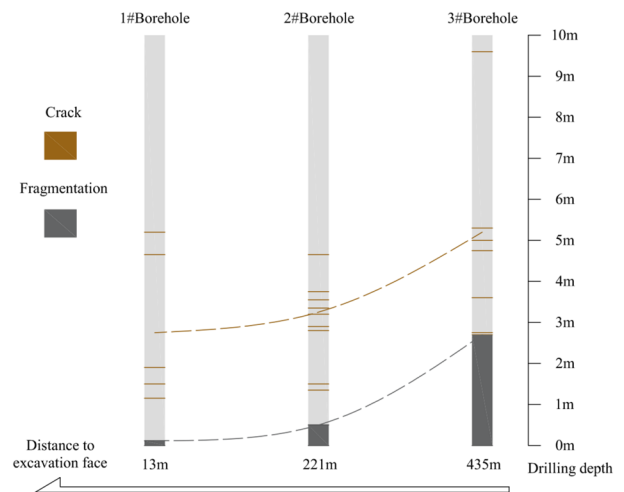
As is shown by the borehole images, the cracks in the surrounding rock of the roof expand greatly in the early stage of excavation, indicating that the original support scheme could not achieve timely control of the surrounding rock, resulting in continuous expansion and

**Fig. 4** Failure characteristics of surrounding rock of 3205 material roadway



**Fig. 5** Borehole camera images under original support

deformation of the surrounding rock. The cracks are generally distributed in the area shallower than 5.4 m, which indicates that the crack has crossed the primary anchor cable anchoring zone. Moreover, traces of crack evolution are also found outside the secondary anchor cable anchoring zone and the maximum development depth has reached 9.56 m. The reconstruction of borehole cracks reveals that the average development depth of cracks and the depth of fracture development display the same change characteristics with the advance of the working face: both show an increasing trend with time, as shown in Fig. 6. This indicates that the original support scheme could not effectively control the deformation of surrounding rock in the long run.



**Fig. 6** Analysis diagram of borehole under the original support scheme

### 3.2 Analysis of influencing factors of roadway surrounding rock deformation

- (1) Both the ground stress and concentrated stress are high in the test roadway. The average burial depth of 3#coal seam is 800 m. The material roadway of 3205 working face a typical roadway with large buried depth and high stress level and its original rock stress is high. At the same time, it is located in the area below the isolated coal pillar of the overlying coal seam, with a vertical distance of 50 m and a horizontal distance of 30 m. Therefore, the roadway is still within the stress concentration area of the isolated coal pillar, and the specific position relation of the high concentration stress is shown in Fig. 7.
- (2) The influence of multiple high-intensity mining. The material roadway of 3205 working face is a typical deep roadway featured with strong mining. Before excavation, it has been affected by repeated min-

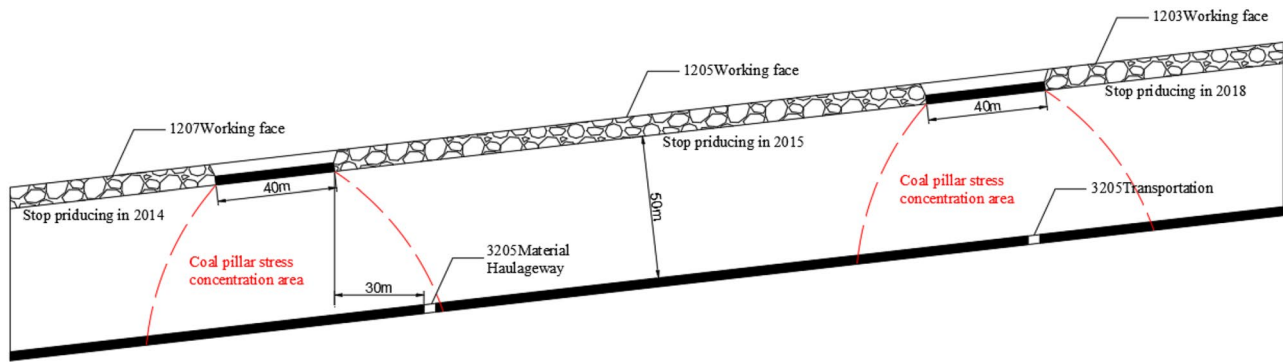


Fig. 7 Spatial relationship diagram of roadway

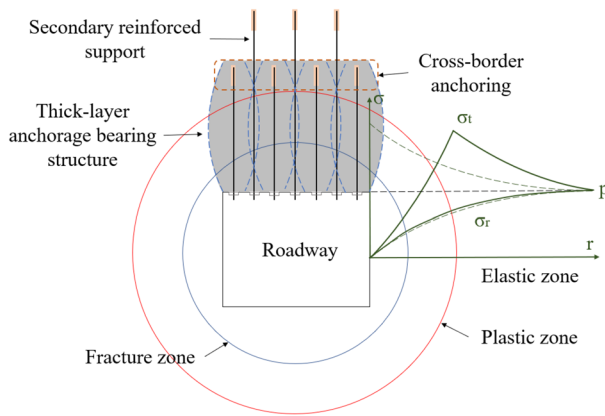
ing of three overlying working faces, namely, 1207 working face, 1205 working face, and 1203 working face. Having been damaged to some degree, its surrounding rock is prone to weathering damage, crack expansion and development.

- (3) Poor lithology of roof surrounding rock. As is shown in Table 1, the immediate roof of the material roadway is 1.3 m-thick argillaceous sandstone, while the main roof is composed of 6 groups of 15 m-thick coal-mudstone interlayers. The roadway roof is a typical ultrathick coal-mudstone composite roof whose cementation degree of coal-rock mass is rather low. Due to its weak lithology and low strength of surrounding rock, the self-bearing capacity of surrounding rock can hardly be given full play.
- (4) The existing support scheme is unreasonable. For the damaged rock mass of the material roadway of 3205 working face, the primary support fails to build a stable bearing structure successfully, which leads to the poor integrity and low strength of surrounding rock. In this case, the roof deforms at a large rate in the early stage of roadway excavation and the surrounding rock continues to deform and eventually reach the plastic limit and break. During this process, the surrounding rock volume increases and intrudes into the roadway, while the cracks continue to expand upward to the outside of the anchoring area, resulting in structural instability. In the case of the failure of the primary support, the secondary anchor cable can only play a passive suspension role as a reinforced support. Meanwhile, the crushing force generated by the expansion and deformation of the rock mass produces a high tensile stress at the anchorage end of the secondary anchor cable, and the separation cracks outside the anchoring area are easily generated under the weakly consolidated composite roof. The results of borehole observation also confirm that structural instability may occur in the secondary anchorage with the development of time.

In general, the existing support can hardly achieve the stability control of roadway damaged rock mass under the influence of stress-disturbance. On the one hand, the primary support is difficult to build high-strength bearing structure; on the other hand, the secondary reinforced support has poor coordination with the primary support, which needs to be changed urgently.

#### 4 The principle of roof graded continuous anchorage

After the excavation of roadway, the surrounding rock stress is released and redistributed around the roadway. In the shallow fracture zone, the damage level is significant, and it is under a lower stress level, penetrating deeper into the surrounding rock. Both the tangential stress ( $\sigma_t$ ) and radial stress ( $\sigma_r$ ) of the surrounding rock increase and surpass the original rock stress ( $p$ ). This region is referred to as the plastic zone, where the surrounding rock undergoes some degree of damage. Beyond the plastic zone, in the elastic zone, the surrounding rock stress gradually decreases until it returns to the original rock stress level. In this region, the surrounding rock is relatively stable and maintains a higher level of integrity. In order to achieve roadway stability control, it is necessary to construct thick and high quality anchored rock beams in time. On the one hand, the anchoring length needs to be increased. To be specific, the anchor cable could be anchored to stable rock formation in the elastic zone outside the plastic zone. If the anchoring zone is in the damaged rock mass, the concentrated tensile stress at the anchoring end can easily lead to the fracture penetrating, thus reducing the stability of surrounding rock [16]. On the other hand, high pretension and high strength anchor cables could be used to lock the multi-layer rock strata so as to make them bear the load cooperatively [17, 18]. The thick rock beam itself forms a self-bearing stable structure,



**Fig. 8** Diagram of cross-boundary graded continuous support technology

making its anchored rock strata in a state of three-way compression, thereby eliminating the roof tensile stress zone and realizing continuous transfer of roof stress. This can maintain the stability of the roadway roof to a certain extent, as is shown in Fig. 8. For the cracked surrounding rock of the material roadway of 3205 working face in Xin'an Coal Mine, the primary anchor cable cannot fundamentally inhibit the crack expansion, and the anchorage body should be modified by grouting to timely seal the crack in the primary support. This measure can prevent the further development of the crack, and improve the integrity of the surrounding rock, thus maintaining the stiffness and strength of the foundation anchoring bearing layer.

For the deep-buried roadway with a composite roof, the primary thick anchored rock beam is not sufficient to resist the stress disturbance during late mining. Consequently, it is necessary to re-reinforce the primary thick anchored layer, which is called secondary reinforced anchoring. From the perspective of continuous stress transfer, the thickness of secondary anchoring is about 1.5 times that of primary anchoring; and the stress transfer effect of the cable with high pre-tension is more efficient. The small displacement of secondary bearing layer is utilized to constrain the large deformation of shallow surrounding rock, so as to achieve linkage control of surrounding rock displacement. This concept of graded reinforcement anchoring serves as the principle of graded continuous anchorage. The multi-level large bearing ring can significantly improve the anti-disturbance performance of the surrounding rock mass, and then achieve the long-term stable bearing capacity of the strong mining roadway [19, 20].

## 5 New support scheme

### 5.1 Control principles

According to the engineering geological condition of the material roadway of 3205 working face and the principles of graded continuous anchorage, the stability control principles are put forward for the surrounding rock of deep-buried roadway with a composite roof.

#### (1) To optimize the roadway section

The roadway section should be changed from a three-center arch to a quasi-rectangle. The roadway should be excavated along the rock strata so as to avoid damaging the roof strata as much as possible. This can reduce the damage to the roadway roof. Meanwhile, in order to optimize the stress environment of the roadway shoulder corners, the roadway shoulder corners should be changed into an arc, which is conducive to realizing the long-term load bearing of the roadway.

#### (2) To improve the support material

The ordinary anchor cable for primary anchoring in the original support scheme should be replaced with high strength and high elongation grouting anchor cable. The existing anchor cable is featured with relatively small diameter and low strength and the excessive length reduces its sensitivity to working resistance, severely limiting its capacity of constraining surrounding rock expansion and deformation. The anchor cable is more often than not cut or broken in actual application. In this case, replacing high-strength supporting materials can enhance the rigid strength of the support system and guarantee the high load-increasing performance. Traditional hollow grouting anchor cables have a load-bearing capacity of less than 340 kN. If the load-bearing capacity of the hollow anchor cable is increased, its elongation will also decrease. The newly adopted high strength and high elongation grouting anchor cables have a load-bearing capacity of over 520 kN, and their elongation can reach 7%. This not only increases the load-bearing capacity of the anchor cable but also improves its elongation. The improved support material is shown in Fig. 9.

#### (3) To improve the preload

The deformation of soft rock roadway is time-sensitive, and the implementation of high preload is the key to reducing the deformation rate of surrounding

**Fig. 9** High strength and high elongation grouting anchor cable



rock in the early stage of roadway excavation. To a certain extent, it can seal the cracks, eliminate the discontinuous zone to achieve continuous stress transfer, and enable the anchor cable to achieve timely high-strength load.

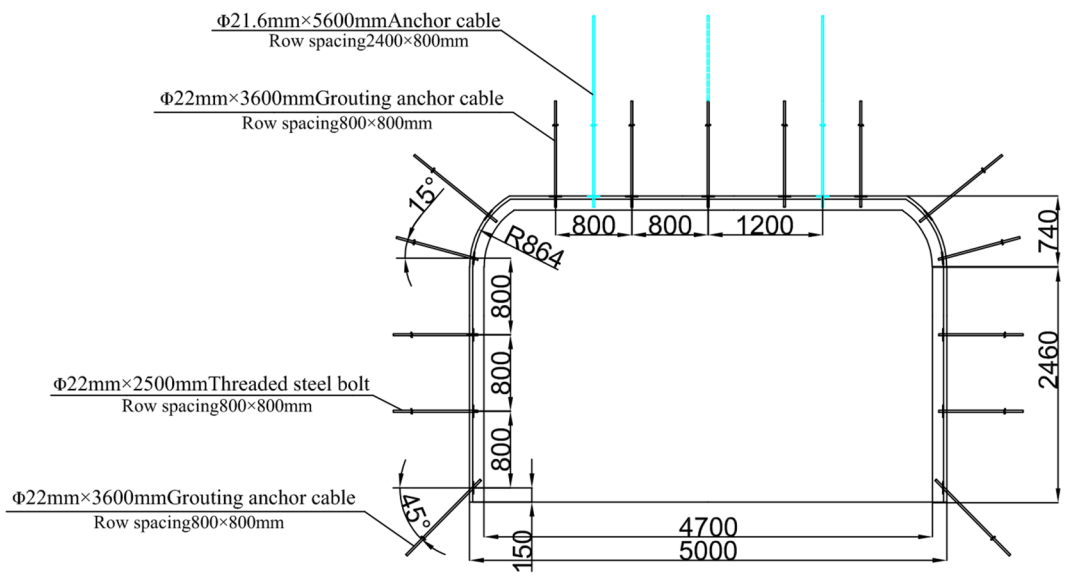
(4) Delayed grouting modified surrounding rock

In this soft rock roadway, the fractures of surrounding rock are highly developed under stress-disturbance. With roadway excavation, the fractures of surrounding rock develop rapidly at a large deformation rate. During this stage, the fractures can not be completely closed only by increasing the preload. The fracture sealing can be achieved through delayed grouting of the stable deformed surrounding rock. The loose surrounding rock can be cemented into a whole, which improves the cohesion and internal friction angle of the rock mass. In this way, the mechanical properties of the rock mass could be strengthened, constructing the high-rigid-strength bearing rock beam, and realizing the long-term stable control of the roadway.

**5.2 New support scheme**

The new support scheme adopts two-stage continuous support technology. The primary support adopts high strength and high elongation grouting anchor cables, and the secondary support adopts high-strength regular anchor cables. According to practical engineering experience, the depth of fracture development in the material roadway of 3205 working face is generally about 2.8 m. Due to the influence of disturbance, the length of the primary anchor cable is set to be 3.6 m. To enhance the synergy between the primary and secondary support, the length of the secondary anchor cable should be 1.5–1.6 times that of the primary anchor cable, thus setting the length of the secondary anchor cable at 5.6 m. Specific support parameters are shown in Fig. 10.

As for the roof support, the specifications of the primary grouting anchor cables are  $\Phi 22 \times 3600$  mm, with a row spacing of  $800 \times 800$  mm and 7 cables in each row; and the pre-load is not less than 150kN. The middle five anchor cables should be equipped with the steel ladder beam. The specifications of ordinary anchor cables for roof secondary support are  $\Phi 21.6 \times 5600$  mm, with a row distance of  $800 \times 800$  mm. The distance between two anchor cables



**Fig. 10** New support scheme



in the same row is 2400 mm, and the preload is not less than 180 kN.

The two sides are supported by 6  $\Phi 22 \times 2500$  mm leftwing steel rebars without longitudinal reinforcement and 2  $\Phi 22 \times 3600$  mm grouting anchor cables, with a row spacing of  $800 \times 800$  mm. The anchor bolts are arranged perpendicular to the roadway wall. Near the shoulder corners, the elevation angle of the anchor rods is  $15^\circ$ ; near the bottom plate, the depression angle of the grouting anchor cables is  $45^\circ$ . The position of the grouting anchor cables is 150 mm away from the bottom plate. The torque of the anchor bolts shall not be less than 300 N m; the pre-tightening force of the grouting anchor cable shall not be less than 150 kN; and the steel ladder beams shall be arranged on the roadway sides.

According to roadway deformation observations and borehole images, it has been observed that during the early stages of excavation, roadway deformation develops rapidly. During this process, the expansion of fractures in the shallow surrounding rock is the direct factor causing deformation. The rate of roadway deformation gradually slows down at approximately 20 m of excavation progress. Therefore, the delayed grouting distance is determined to be 20 m. At the same time, this distance ensures that roadway excavation and grouting operations do not interfere with each other.

## 6 Simulation analysis

The Trigon module in UDEC software was used to design the model of the material roadway of 3205 working face in Xin'an Coal Mine. The model consists of 40 layers, including five lithologies of mudstone, coal, siltstone, fine sandstone, and coarse sandstone. The mechanical parameters are shown in Table 2.

The overall design of the model is near-horizontal stratum with a model size of  $160 \times 90$  m. In order to compare the roadway maintenance control effect of the new support scheme and that of the original support scheme, the rock joints in the surrounding area of the roadway

are encrypted with a size of  $20 \times 23$  m. The deformation and crack development of the roadway roof and sides are mainly observed. The calculation model is shown in Fig. 11.

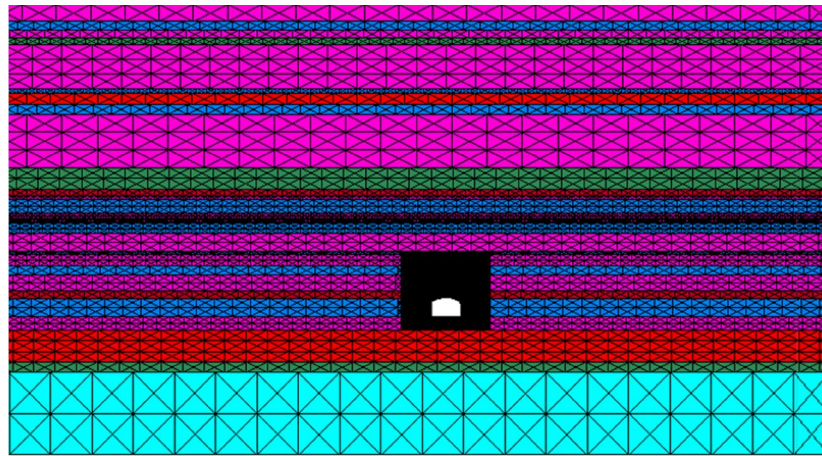
In order to better conform to the gradual process of stress release on the engineering site, the stress is gradually released in ten stages in the experiment, with 10% released in each stage until the stress drops to zero in the tenth stage. The stress release coefficient  $R$  of these ten stages is 0.1, 0.2, 0.3–1.0, respectively [21, 22].

As can be seen from Fig. 12, in the original supporting state, the plastic zone of the roadway is distributed symmetrically and reaches a deeper range. The tensile failure is concentrated around the roadway, and the failure is more serious in the roof and bottom plates than the two sides, showing a triangular distribution, and the maximum depth of roof tension failure is up to 2.8 m. Under the new support scheme, the number of plastic deformation zones decreases from 4389 to 3438, a reduction of 21.6%; and the number of tensile failures reduce from 865 to 677, a reduction of 21.7%. This indicates that the control effect of the original support on the surrounding rock of the roadway is not ideal. To be specific, the plastic zone of the surrounding rock has a large development range, and tensile failure concentrates mainly in the roadway roof and two sides. The improved support scheme effectively controls the plastic deformation range of the roof and improves the surrounding rock integrity of the roof and two sides. Accordingly, only a small amount of tensile failure occurs in the roof.

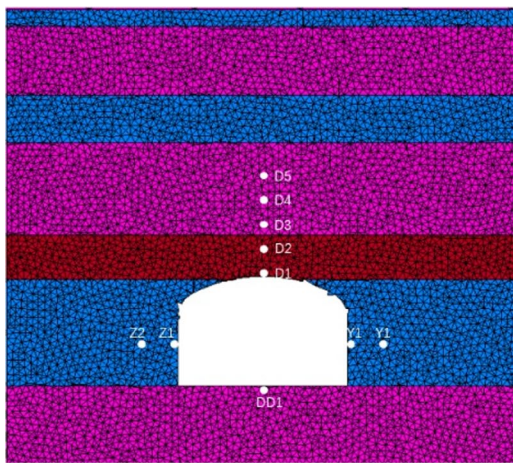
As can be seen from Fig. 13, the roadway deformation presents a “three-stage” characteristic under the original support scheme. In the early stage, the deformation of surrounding rock develops rapidly after excavation; in the middle stage, the deformation rate decreases to a slow increase as the stress of surrounding rock is gradually balanced; in the later stage, the deformation rate increases again with the long-term strength reduction of rock mass. The deformation trend of the simulated deformation curve in the early and middle stages is the same as that of the in-situ deformation monitoring, and the final simulated subsidence is 276 mm, indicating that the original scheme

**Table 2** Stratum Distribution and Mechanical Parameters

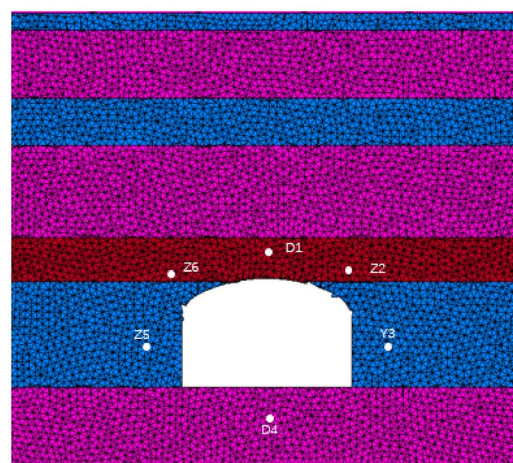
Lithology	Density/kg m <sup>-3</sup>	Elasticity modulus/GPa	Normal stiffness/GPa m <sup>-1</sup>	Shear stiffness/GPa m <sup>-1</sup>	Internal friction angle/ $^\circ$	Cohesion/MPa	Tensile strength/MPa
Mudstone	2150	2.6	424.9	127.5	2.9	31	0.7
coal	1367	0.9	153.2	46.0	1.38	27	0.3
Siltstone	2575	3.3	517.6	155.3	11.2	34	1.8
Fine sandstone	2633	6.3	912.2	273.7	11.7	35	2.7
Coarse sandstone	2075	3.9	550.6	165.2	6	29	0.6



(a) Numerical calculation model



(b) Displacement monitoring



(c) Strain monitoring

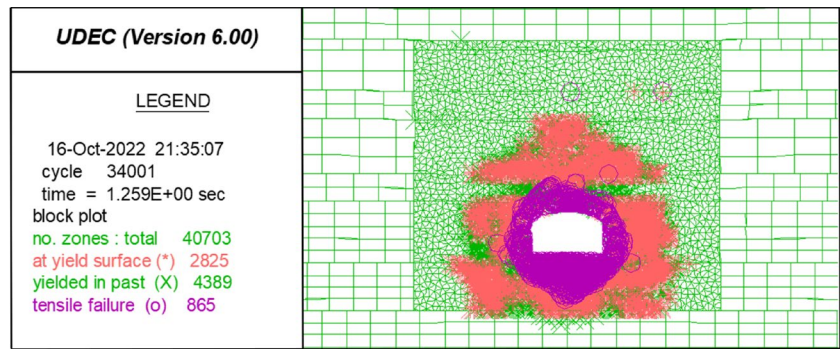
Fig. 11 Numerical calculation model

fails to play a good role in roadway maintenance and control. Under the new support scheme, the roof deformation of the roadway presents a steady and slow growth trend. This indicates that higher support strength can be achieved in the early stage after roadway excavation, thus inhibiting the deformation development of surrounding rock. The final roof subsidence is 82 mm, which is 70.28% lower than that of the original scheme. The final deformation of the lower side under the condition of no support and the original support scheme is 430 mm and 354 mm, respectively. In contrast, the final deformation of the new support scheme is 78 mm, which is 77.96% lower than that of the original support scheme.

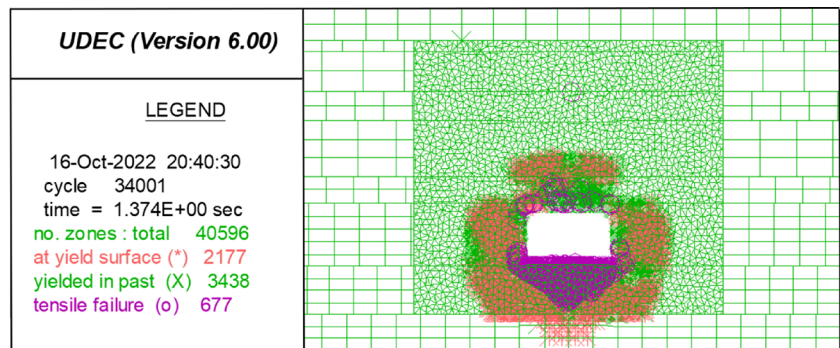
As shown in Fig. 14, after the roadway is excavated, the stress state of surrounding rock changes from three-way loading to two-way loading, and the vertical stress concentration coefficient of roof and horizontal stress concentration coefficient of the two sides gradually

decrease. The monitoring results are shown in Fig. 17. In this process, it is worth noting that under the new support scheme, the vertical stress of the roof decreases with the stress release coefficient in stages; whereas under the original support scheme, the stress of the roadway decreases rapidly after excavation, from the original rock stress (21.7 MPa) to (5.1 MPa). This indicates that the original support cannot achieve timely bearing and only plays a passive suspension role when the roadway deformation is large in the middle and late stages. In addition, under the new support scheme, the horizontal stress of the surrounding rock can be controlled at a higher level than that in the original support scheme. This is of great help to realize continuous stress transfer and build high-strength bearing rock beams to resist stress disturbance, thus realizing stable control of the roadway roof.

**Fig. 12** Nephogram of plastic zone distribution of surrounding rock under different support schemes



(a) Original support scheme

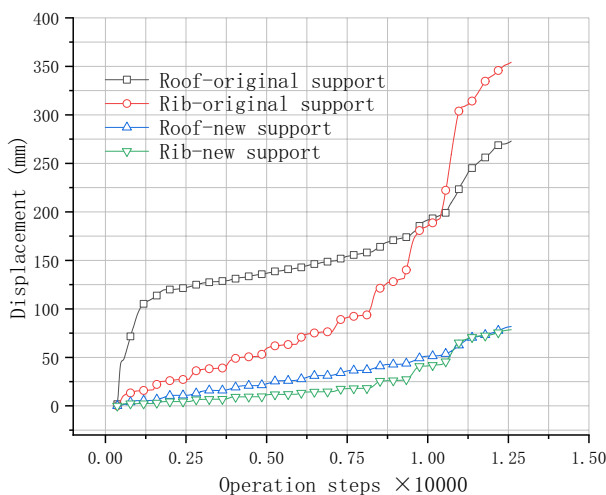


(b) New support scheme

### 7 Mine pressure monitoring and analysis

An experimental section of new support scheme with a length of 50 m was set up at 1800 m of the material roadway of 3205 working face. In order to comprehensively evaluate and analyze the surrounding rock control effect

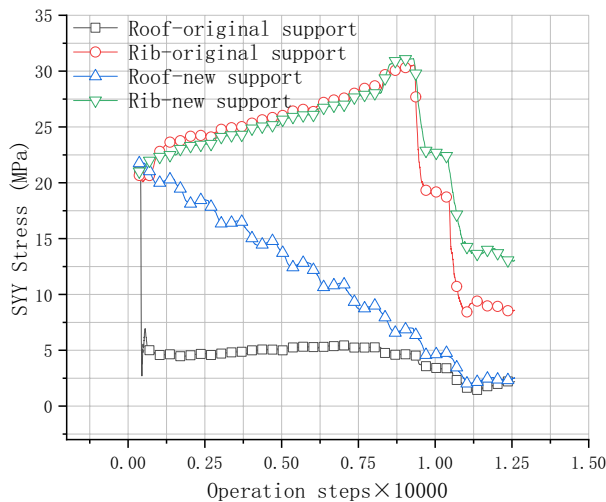
of the new support scheme, surface displacement monitoring, anchor cable axial force monitoring, and roof borehole monitoring were conducted. Displacement measurement stations and anchor force measurement stations are arranged together using an equal distribution principle. In order to achieve better borehole inspection results, borehole is positioned in the middle of the experimental section. It is crucial to ensure that all measurement stations do not interfere with normal production activities. The specific monitoring stations arrangement is shown in Fig. 15.



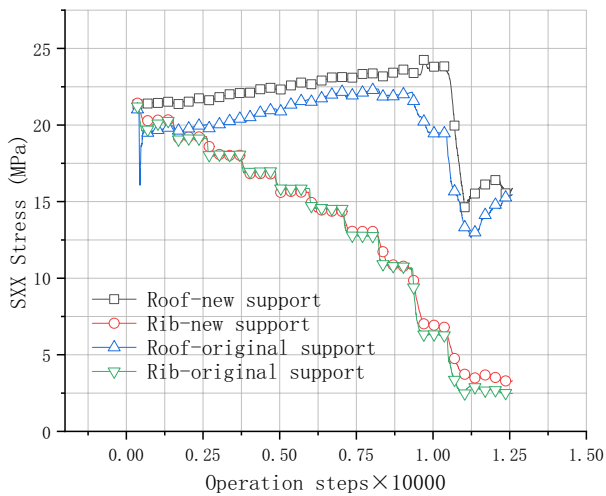
**Fig. 13** Deformation of surrounding rock under different support schemes

#### 7.1 Evolution law of roadway surface displacement

Surface displacement station A and B are placed at a distance of 20 m and 38 m to the starting position of the new support scheme. The monitoring results are shown in Fig. 16. As can be seen from the figure, the roof subsidence and the convergence of the two sides present a trend of “sharp increase–slow increase–eventual stability” as the excavation advances farther and farther away from the working face. The final roof subsidence and side convergence displacement observed at station A are 33 mm and 78 mm, respectively; and at station B, 32 mm and 97 mm, respectively. Within 40 m from the driving face, the roof subsidence is relatively severe, accounting for 73% and 78% of the final deformation amount, respectively.



(a) SYX Stress



(b) SXX Stress

Fig. 14 Strain of surrounding rock under different support schemes

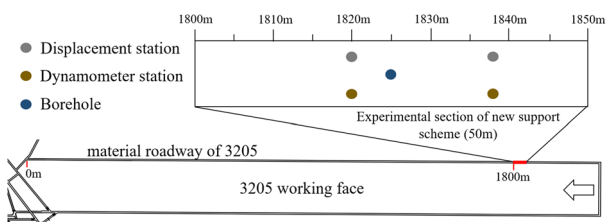


Fig. 15 Position of monitoring stations

Similarly, the severe deformation range of the two sides is also within 40 m from the driving face. The convergence amount accounts for 51% and 80% of the final deformation amount, respectively. Compared with the original support scheme, the new scheme reduces the roof deformation by 91% and the deformation of the two sides by 50%.

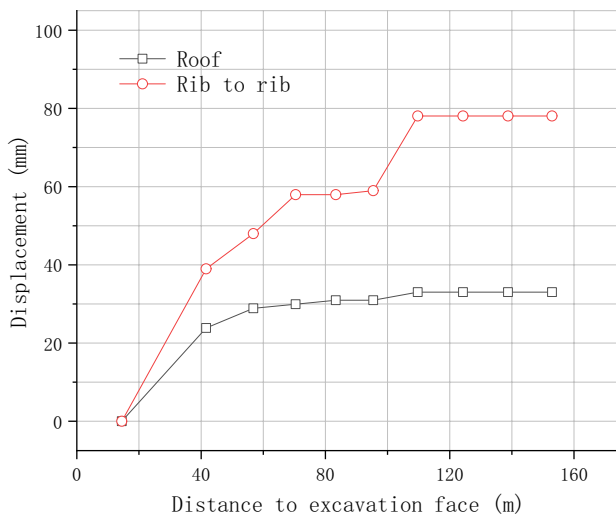
In summary, under the new support scheme, the deformation of the roof and the two sides is significantly reduced, which can better control the deformation of the roadway.

### 7.2 Evolution law of anchor cable load

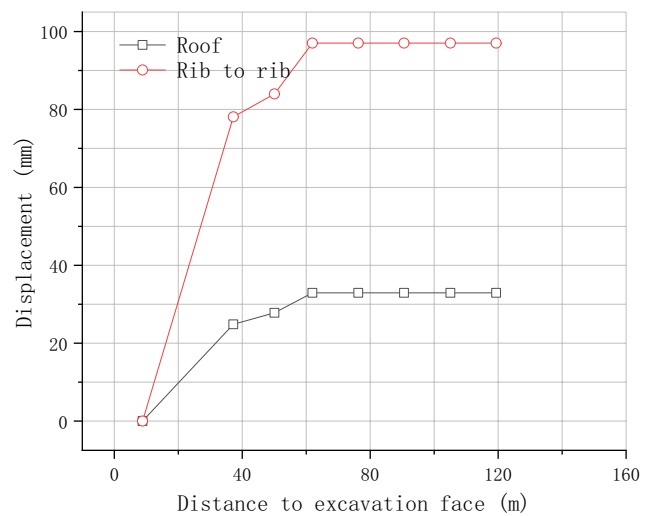
Stations 1# and 2# for anchor cable load monitoring were installed at a distance of 20 m and 38 m to the starting position of the new support scheme, respectively. The monitoring results are shown in Fig. 17. As is shown in the figure, the anchor cable axial force increases significantly in the early stage of roadway excavation. To be specific, when the excavation advances by 15 m, the load increment monitored by 1# dynamometer is 43 kN, an increase of 25%; that of 2# dynamometer reaches up to 17 kN, an increase of 11%. In the early stage, the rapid increase of anchor cable axial force indicates that the anchor cable is highly sensitive to resistance and can bear the load in time. As the roadway excavation further advances, the anchor cable axial force tends to increase slowly and becomes stable gradually, indicating that the current support can achieve the stability control of the surrounding rock of the roadway. One sudden drop appears in the axial force curve of 1# dynamometer. After careful observation, it is found that the shallow surrounding rock comes to break correspondingly. According to the analysis, this is because the rapid increase of load generates relatively high shear stress on the surface of surrounding rock. Once the stress exceeds the ultimate strength, the rock mass would fail and break after the peak, resulting in the sudden load drop.

### 7.3 Distribution characteristics of cracks in roof surrounding rock

A 10 m-deep borehole was drilled at a distance of 25 m to the starting position of the new support scheme to observe the structure and fracture evolution of the roof surrounding rock with a borescope. According to the borehole observation results in Fig. 18, under the new support scheme, the crack expansion of the roof surrounding rock is significantly reduced. Similarly, the shallow separation and cracks are significantly reduced, too. The crack depth is generally within 2.5 m. Even though the maximum development depth of local cracks reaches up to 3.26 m, it does not exceed the primary anchorage zone. In contrast, under the original support scheme, the maximum crack depth is 9.56 m. As can be seen, under new support scheme, the maximum crack depth is reduced by 6.3 m, greatly improving the control effect of surrounding rock and realizing the stability control of the roadway. Grouting can seal the cracks in the anchorage zone in time and enhance the



(a) station A



(b) station B

Fig. 16 Deformation measurement under new support scheme

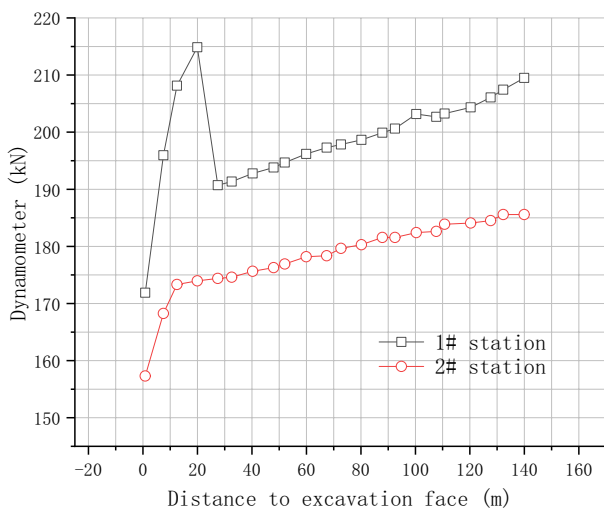


Fig. 17 Dynamometer indication of 1# station

bearing performance of anchorage zone, which is conducive to the long-term stable bearing capacity.

## 8 Discussion

### 8.1 Graded continuous support

Graded continuous support relies on the combined action of long and short anchor cables, with the primary support structure being critical, and the secondary

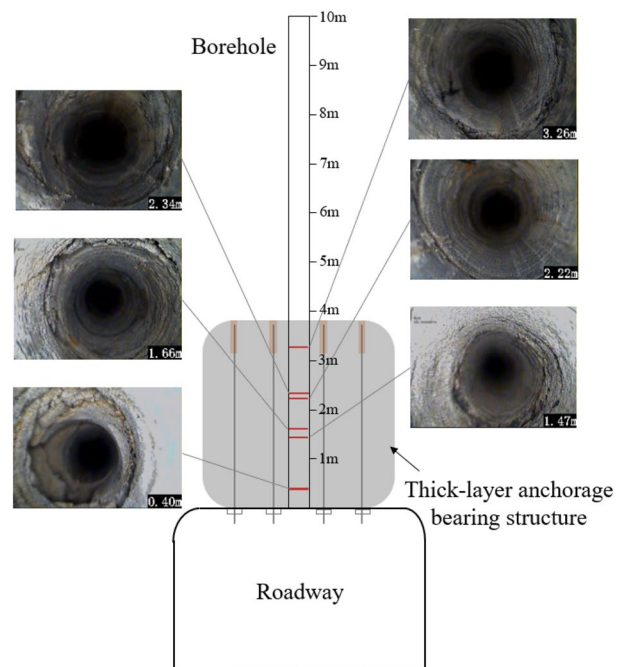


Fig. 18 Drilling characteristics of the material roadway roof of 3205 working face

support structure reinforcing the load-bearing capacity [23]. It is worth noting that both the original support scheme and the new support scheme for the subject of this study are two-stage support systems. However, they differ significantly in the effectiveness of roadway maintenance, primarily due to the poor quality of the primary support structure in the former. Under

the constraint of weaker anchor cables, continuous deformation of the surrounding rock eventually leads to support failure. To address this issue, in the early stages of roadway excavation, high pre-tension forces are applied to the anchor cables to increase their sensitivity to working resistance, restrain further development of fissures in the thick anchoring zone, limit surrounding rock deformation, and achieve active load-bearing of the surrounding rock under the action of the anchor cables. Additionally, as a strengthening support measure, when the length of the secondary anchor cable is too short, its reinforcing effect is limited, and excessive length can reduce the coordination between the two stages of support. Therefore, the length of the secondary anchor cable is controlled at 1.5 to 1.6 times that of the primary anchor cable, thus achieving continuous support.

## 8.2 Delayed grouting reinforcement of fractured surrounding rock

The roadway under study in this research falls into the category of typical deep-buried soft rock roadways, characterized by significant fissures after excavation. Applying higher pre-tension forces can partially close these fissures [24]. To further enhance the integrity of the surrounding rock, grouting is utilized to seal and reinforce the anchoring zone, thereby improving the strength and stiffness of the primary support structure [25]. Through the study, it has been observed that there are distinct stages of deformation after roadway excavation: In the first stage, the roadway deformation rate is relatively high as the surrounding rock stress is released, accompanied by the expansion and development of fissures. In the second stage, the roadway deformation rate slows down and gradually stabilizes. At this point, the surrounding rock stress reaches a new equilibrium and remains stable for some time. In the third stage, the roadway deformation rate increases again, primarily because the roadway deformation exceeds its load-bearing limit, ultimately leading to support failure. Grouting during the first stage would lead to the degradation of the reinforcement material during the stress release process, and grouting during the third stage would result in significant grout loss. Therefore, grouting is chosen to be performed during the second stage. According to on-site monitoring results, after excavating 20 m of roadway, the deformation rate gradually slows down. Performing delayed grouting at this stage achieves better reinforcement of the surrounding rock. Subsequent ground pressure monitoring results have also confirmed this observation.

## 8.3 The engineering significance of this support technology

Roadways are often surrounded by abundant weak rock layers such as mudstone, which have low strength and present significant challenges for reinforcement. As mining depth increases, the issues related to the management of soft rock roadways become more pronounced, especially in the high-stress environments of deep mining areas. Therefore, a support technology is proposed that begins by improving the support materials, using high strength and high elongation grouting anchor cables for robust reinforcement, followed by grouting for consolidation, aiming to control roadway surrounding rock deformation. This support technology demonstrates strong adaptability to soft rock and fractured rock roadway conditions. It provides valuable insights for coal mine roadway grouting support projects and the management of soft rock roadways, holding significant importance for roadway safety maintenance and control.

## 9 Conclusions

In response to the significant deformation issues in deep-buried soft rock roadways, a new support technology was developed based on on-site studies that summarized the deformation patterns of the roadways. It was observed that roadway deformation occurs in stages. Following the principles of graded continuous anchoring, the new support scheme utilizes high strength and high elongation grouting anchor cables to construct a two-stage continuous support structure. Additionally, delayed grouting is employed to further reinforce the fractured surrounding rock. Simulation results and mine pressure monitoring data both demonstrate effective control of surrounding rock deformation. Field mine pressure monitoring indicates a significant reduction in surrounding rock deformation, with a 91% reduction in roof deformation and a 50% reduction in rib deformation. Borehole inspection results show that the maximum depth of roof fissures has decreased to 3.56 m, indicating a substantial improvement in roadway stability. It is important to note that grouting reinforcement is essential for deep-buried soft rock roadways, and the timing of grouting is crucial. This study determined the grouting timing based on the staged deformation of the roadway. However, roadway deformation is not solely correlated with the development of internal rock fissures. Therefore, future efforts should involve complementary methods to assess the extent of fissure development within the surrounding rock, allowing for precise determination of the grouting timing.

**Author contributions** JL is responsible for manuscript writing and related illustrations, YL is responsible for manuscript writing and on-site tracking, as well as illustrations. ZX is responsible for manuscript writing and on-site tracking. NZ is responsible for revising and polishing the paper and creating illustrations. FG is responsible for on-site tracking and numerical simulations. QC is responsible for on-site tracking. SL is responsible for on-site tracking. Furthermore, all authors have reviewed the manuscript.

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**Data availability** The datasets generated during and/or analysed during the current study are available from the corresponding author on reasonable request.

## Declarations

**Conflict of interest** The authors have no competing interests to declare that are relevant to the content of this article.

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