International Journal of Natural Resources and Environmental Studies
ISSN: 3006-2012 (Print), ISSN: 3006-0834 (Online) | Volume 8, Number 4, Year 2026
DOI: https://doi.org/10.62051/ijnres.v8n4.16
Journal homepage: https://wepub.org/index.php/IJNRES/index

ϕ28.6 mm High-Strength Anchor Cable Control Technology for Deep Soft Rock Roadways

Xianda Zang*

School of Energy Science and Engineering, Henan Polytechnic University, Jiaozuo 454003, China
*Corresponding Author: Xianda Zang (Email: 1801157937@qq.com)

ABSTRACT

Aiming at the engineering challenge of large surrounding rock deformation and conventional support failure in soft rock roadways under deep high stress and intense mining-induced conditions, this paper takes the 1101 roadway of Hudi Coal Mine as the engineering background. Adopting a combined method of theoretical analysis, numerical simulation, and field tests, this paper reveals the hierarchical coupling control mechanism of high-strength anchor cables and grouting reinforcement, optimizes and determines the support parameter system of "short anchor cables + long anchor cables + deep-shallow grouting" based on ϕ28.6 mm large-diameter high-strength anchor cables, and carries out industrial field tests. The results show that, compared with the original support scheme, the convergence of the two roadway sides, roof subsidence, and floor heave are reduced by 47.9%, 46.2%, and 40.1% respectively with the proposed technology. The surrounding rock deformation is effectively controlled, which provides a technical reference for surrounding rock stability control of similar deep soft rock roadways under strong dynamic pressure.

KEYWORDS

Deep Soft Rock Under Strong Dynamic Pressure, Hierarchical Coupling Support, Φ28.6 Mm Large-Diameter High-Strength Anchor Cables, Surrounding Rock Control.

1. INTRODUCTION

At present, the output of deep coal mines with a burial depth of over 800 m accounts for more than 40% of the total coal production in China[1], and deep underground mining has gradually become the norm in China's coal industry. However, the complex environment of high in-situ stress and intense mining-induced disturbance in deep mines[2-4] leads to prevalent problems including large surrounding rock deformation, frequent support failure, and repeated roadway rehabilitation in over 60% of soft rock roadways, which brings severe challenges to roadway support engineering[5]. Therefore, it is of great practical necessity to conduct research on reinforcement support technology for deep roadways that still suffer from large deformation after multiple rehabilitation works.

Zhu[6] concluded that the main inducement of large deformation in deep soft rock roadways is the shear failure of surrounding rock. Focusing on the creep characteristics of soft coal, Wu[7] proposed a combined support method using high-prestressed, high-strength bolts and cables, together with steel mesh and plastic woven mesh. Based on true triaxial simulation tests, Zhu[8] investigated the deformation law of surrounding rock after deep roadway excavation. Based on test results, Jia[9] concluded that the main causes of roadway deformation and floor heave are the creep deformation of surrounding rock under deep high-stress environments. Based on the dual action mechanisms of damage evolution and hardening strengthening coexisting throughout the whole creep deformation process of deep rock, Zhou[10] established a three-element nonlinear damage-hardening coupled creep model. Aiming at the difficulty of surrounding rock control in deep roadways subjected to lateral pressure and front abutment pressure, Kang[11] proposed a support scheme based on ground and roadway surrounding rock reinforcement. Yeng[12] proposed controlling surrounding rock deformation through surrounding rock modification, and improved the overall strength of fractured surrounding rock by studying grouting parameters, slurry seepage laws, and developing appropriate grouting materials. Based on the above existing studies, the main cause of surrounding rock instability in deep soft rock roadways is the creep deformation of surrounding rock under high in-situ stress, which induces large-area instability of the roadway section.

Prestressed anchor cables are a critical method for surrounding rock support, and the prestress generated by anchor cables can effectively reinforce roadway surrounding rock[13-14]. For this reason, many scholars have carried out extensive research on prestressed anchor cable support. By analyzing field monitoring data, Li[15] investigated the load and failure mode of support structures under high in-situ stress conditions. Aiming at the difficulty of surrounding rock control in deep roadways with thick top coal, Xiao[16] proposed a support technology combining high-strength, high-pretightening force bolts with diagonal cable beams, which effectively controlled the large deformation of roadway surrounding rock. Aiming at the failure problem of roadway surrounding rock and support structures under strong impact loads, Wu[17] proposed a combined support and pressure relief control technology. Based on the basic principle of the convergence-confinement method, Meng[18] obtained through mechanical calculation that when the anchor cable diameter increases from ϕ18.00 mm to ϕ28.6 mm, its support strength increases by 126%, and the support performance of the anchor cable is significantly improved.

Taking the 1101 roadway of Hudi Coal Mine as the research object, this paper adopts the ϕ28.6 mm large-diameter high-strength anchor cable as the core load-bearing component for the first time, and constructs a hierarchical coupling control system of "shallow grouting consolidation - short anchor cable reinforcement - long anchor cable suspension". The support parameters are optimized through theoretical analysis and numerical simulation, and field industrial tests are carried out to verify the engineering applicability of the proposed technology.

2. ENGINEERING BACKGROUND AND DEFORMATION AND FAILURE CHARACTERISTICS OF SURROUNDING ROCK

2.1. Engineering Geological Conditions of the 1101 Roadway

The 1101 roadway of Hudi Coal Mine is the auxiliary transportation main roadway in Panel 1, with a burial depth of approximately 700 m, which is a typical deep roadway. A total of 5 main roadways are arranged in this panel, and the 1101 roadway is located in the middle. The center-to-center spacing between the 1101 roadway and the adjacent 1102 roadway and 1103 roadway on both sides is 30 m respectively, while the width of the reserved rock pillar between adjacent roadways is only 25 m, forming a dense roadway group. Fully mechanized working faces are arranged on both sides of the roadway: the 1301 (Upper) and 1303 (Upper) working faces on the south side have completed mining, the 1305 (Upper) working face is in the mining stage, and the safe distance between the stop line of the working face and the roadway is 100 m.

The surrounding rock of the roadway is dominated by carbonaceous mudstone, and the entire roadway is located in the carbonaceous mudstone stratum 15 m to 20 m above the roof of the No.3 Coal Seam. Figure 1 shows the diagram of underground borehole coring and sampling, and Figure 2 shows the process of the rock mechanics test. The test results show that the carbonaceous mudstone has an average uniaxial compressive strength of only 21.6 MPa, an elastic modulus of 8.03 GPa, and an average tensile strength of 0.70 MPa. Its tensile strength is only 3.01% of its compressive strength, which means it belongs to an extremely weak rock mass. It is prone to softening and argillization when encountering water, with extremely poor engineering properties.

Figure 1. Underground Borehole Coring, Sampling and Rock Sample Preparation

Figure 2. Rock Mechanics Test Process

2.2. Defects of the Original Support System and Deformation Characteristics of Surrounding Rock

The original support of the 1101 roadway adopted a combined support system of bolt-mesh-shotcrete, bolts and cables, as shown in Figure 3. After rehabilitation, the bolts were replaced with ϕ22 × 2400 mm high-strength bolts, with roof anchor cables of ϕ22 × 7300 mm and roadway side anchor cables of ϕ22 × 5300 mm, with a pretightening force of no less than 250 kN. However, this support system has three core defects:

First, conventional anchor cables of ϕ21.8 mm grade are adopted. Under a pretightening force of 300 kN, the load reaches 54.3% of its theoretical breaking force (645 kN), making it extremely prone to tensile fracture under additional mining-induced loads. Second, the maximum anchoring depth is only 7.3 m, which is smaller than the 8–9 m surrounding rock failure depth after mining, thus failing to form effective suspension. Third, no targeted grouting reinforcement measures are taken, so the fractured surrounding rock cannot form an integral load-bearing system with the support structure.

Affected by the above factors, the 1101 roadway has suffered severe deformation and failure, with the failure mode shown in Figure 4. The cumulative roof-to-floor convergence reaches 2.5–3.5 m, the cumulative convergence of the two roadway sides reaches 2–3 m, and the roadway cross-section has undergone significant convergence. The support structure has experienced large-area failure modes including tensile fracture of bolts and cables, anchor disengagement, and cracking and spalling of the shotcrete layer. Despite multiple rounds of roadway re-driving, rehabilitation and reinforcement support, the surrounding rock deformation continues to develop, forming a vicious cycle of "instability - roadway rehabilitation - re-instability", which seriously threatens safe production in underground mines.

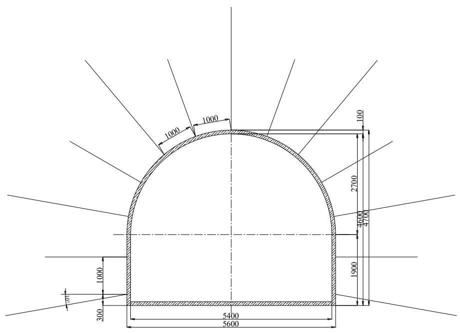


Figure 3 Original Support Layout of the 1101 Roadway

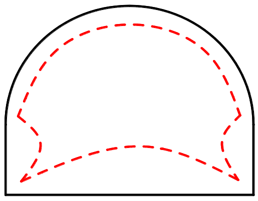


Figure 4 Deformation and Failure Modes of Surrounding Rock of the 1101 Roadway

3. HIERARCHICAL COUPLING CONTROL MECHANISM OF HIGH-STRENGTH ANCHOR CABLES AND GROUTING

3.1. General Idea of Hierarchical Coupling Control

Based on the deformation and failure mechanism of surrounding rock of the 1101 roadway, this paper proposes a general control idea of "high-strength active support + hierarchical anchoring + surrounding rock modification", and constructs a three-level composite load-bearing system of "shallow grouting consolidation - 4.3 m short anchor cable reinforcement - 8.3 m long anchor cable suspension". The fractured surrounding rock is consolidated through shallow grouting to provide a reliable anchoring foundation for the short anchor cables; the shallow surrounding rock is reinforced by short anchor cables to form an initial load-bearing arch; the ϕ28.6 mm large-diameter long anchor cables are adopted to suspend the shallow load-bearing structure to the deep stable rock stratum, blocking the outward expansion of the plastic zone; finally, an integral collaborative load-bearing structure is formed through the coupling effect of grouting and anchor cables.

Due to the severe failure of the surrounding rock of the 1101 roadway, the study determines to re-profile and enlarge the roadway into a rectangular roadway with dimensions of 5600 mm in width × 5000 mm in height. The two top corners are constructed into circular arc profiles with a radius of 588 mm, and the cross-section is shown in Figure 5.

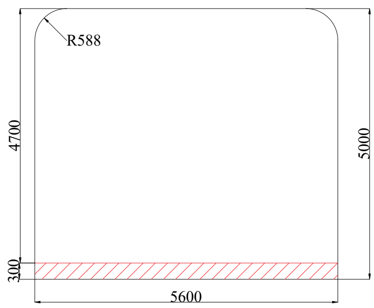
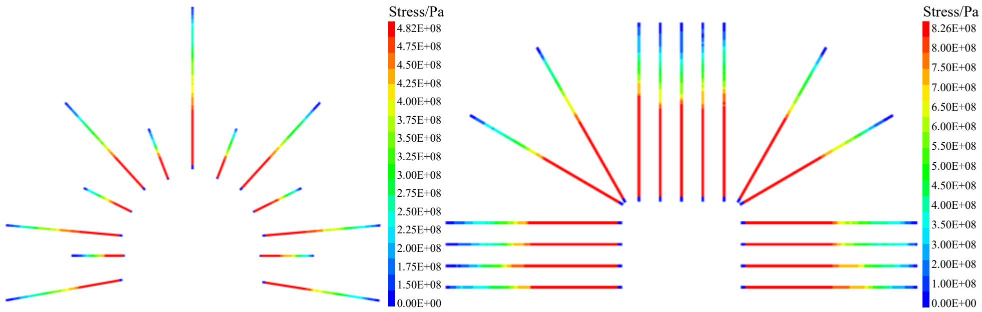


Figure 5 Rehabilitated Cross-Section Design of the 1101 Roadway

3.2. Load-Bearing Mechanism of ϕ28.6 mm Large-Diameter High-Strength Anchor Cables

The bearing characteristics of ϕ28.6 mm and ϕ21.8 mm anchor cables are compared and analyzed through numerical simulation. The numerical simulation nephogram of the axial stress of the anchor cables is shown in Figure 6, and the results show that: under a pretightening force of 300 kN, the working resistance of the ϕ28.6 mm anchor cable is 71.4% higher than that of the ϕ21.8 mm anchor cable, with a significantly improved bearing capacity. From the perspective of safety redundancy, the theoretical breaking force of 1860-grade ϕ28.6 mm steel strand is about 990 kN, and the pretightening force of 300 kN only accounts for 30.3% of its breaking force, which reserves sufficient space for resisting mining-induced dynamic loads. In contrast, the 300 kN pretightening force of the ϕ21.8 mm anchor cable is close to its safety limit, making it extremely prone to fracture under mining-induced disturbance.



(a) ϕ21.8mm cable (b) ϕ28.6mm cable

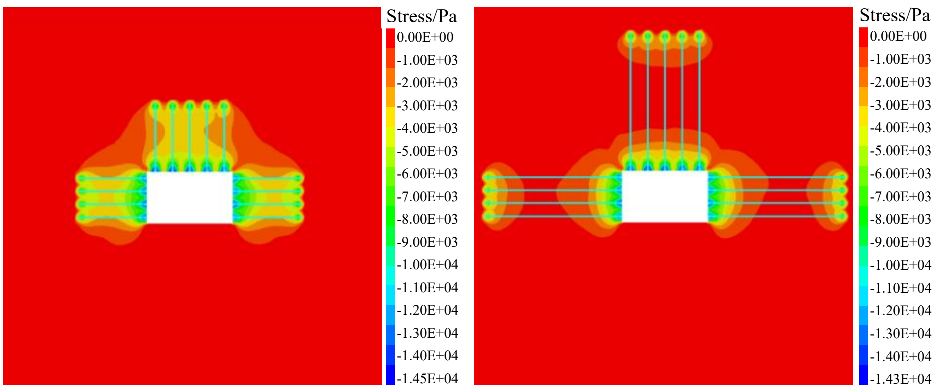
Figure 6 Axial Stress of Anchor Cables with Different Diameters Under the Same Pretightening Force

Meanwhile, the ϕ28.6 mm large-diameter anchor cables have superior stress diffusion characteristics. Under high pretightening force, they can form a continuous and complete compressive stress field in the surrounding rock, effectively offset the tensile stress generated by roadway excavation, and inhibit the development and propagation of joints and cracks. The simulation of axial stress distribution shows that the axial stress of the ϕ28.6 mm anchor cables is uniformly distributed along the length direction, with no occurrence of local stress concentration. The stress in the anchorage section attenuates smoothly, enabling the prestress to be effectively transmitted to the stable rock stratum at a depth beyond 8.3 m.

3.3. Hierarchical Anchoring Mechanism of Long and Short Anchor Cables

Figure 7 shows the nephogram of the minimum principal stress distribution of anchored rock mass with different anchor cable lengths. The simulation results show that the hierarchical support form with staggered arrangement of 4.3 m short anchor cables and 8.3 m long anchor cables realizes collaborative control of surrounding rock at different depths. The 4.3 m short anchor cables mainly act on the shallow fractured surrounding rock within 0–4 m depth. They compact the fractured rock blocks through high pretightening force, form a shallow load-bearing arch with certain load-bearing capacity, and inhibit dilatancy deformation and bed separation of the shallow surrounding rock.

The 8.3 m ϕ28.6 mm long anchor cables penetrate through the shallow load-bearing arch, are anchored into the intact stable rock stratum at a depth beyond 8 m, suspend the shallow load-bearing structure as a whole, and effectively control the expansion of the deep plastic zone. The staggered arrangement of long and short anchor cables enables their respective prestress fields to be superimposed with each other, forming a continuous stress reinforcement zone covering shallow to deep surrounding rock and avoiding the occurrence of support blind zones.



(a) 4m (b) 8m

Figure 7 Prestress Diffusion of Anchor Cables with Different Lengths

3.4. Modification and Coupling Action Mechanism of Deep and Shallow Grouting

Aiming at the failure characteristics of surrounding rock at different depths, a hierarchical grouting technology of "shallow borehole low-pressure grouting + deep borehole high-pressure grouting" is adopted. The shallow grouting borehole has a depth of 4 m and a grouting pressure of 3 MPa, which mainly consolidates the shallow fractured zone with large fractures, improves the integrity of surrounding rock, and provides a reliable anchoring foundation for short anchor cables. The deep grouting borehole has a depth of more than 8 m and a grouting pressure of 10–15 MPa, which is used to seal the deep plastic zone with micro-fractures and improve the mechanical properties of surrounding rock in the anchorage section of long anchor cables.

The coupling effect of grouting and anchor cables can significantly amplify the support efficiency. The slurry penetrates into the fractures of surrounding rock, cements the fractured rock blocks into a whole, and wraps the anchor cable body at the same time, forming a three-in-one integral load-bearing structure of "anchor cable - slurry - surrounding rock", which greatly improves the anchoring force and the integrity of the support system.

4. SUPPORT PARAMETER OPTIMIZATION AND NUMERICAL SIMULATION VERIFICATION

4.1. Establishment of FLAC3D Numerical Model

Based on the engineering geological conditions of the 1101 roadway, a FLAC3D numerical model with dimensions of 180 m × 20 m × 50 m is established. A uniform load is applied on the top of the model to simulate the self-weight of the overlying strata, and displacement constraint boundaries are adopted at the bottom and around the model. The surrounding rock adopts the Mohr-Coulomb constitutive model, and the mechanical parameters are taken from the laboratory rock mechanics test results, with the specific parameters shown in Table 1.

Table 1 Mechanical Parameters of Surrounding Rock for the Numerical Model

lithology	density/ kg·m-3	bulk modulus/ GPa	shear modulus /MPa	cohesion /MPa	internal friction angle/°	tensile strength /MPa
fine sandstone	2600	3.44	1.59	5.91	33	1.74
medium sandstone	2700	3.31	2.49	4.40	31	1.22
siltstone	2500	3.00	2.60	2.80	31	1.60
carbonaceous mudstone	2300	1.23	0.87	1.20	27	0.35
sandy mudstone	2350	2.24	1.20	2.26	28	0.54
mudstone	2500	1.95	0.74	2.11	25	0.28
No. 3 Coal	1400	1.31	0.46	1.11	20	0.42

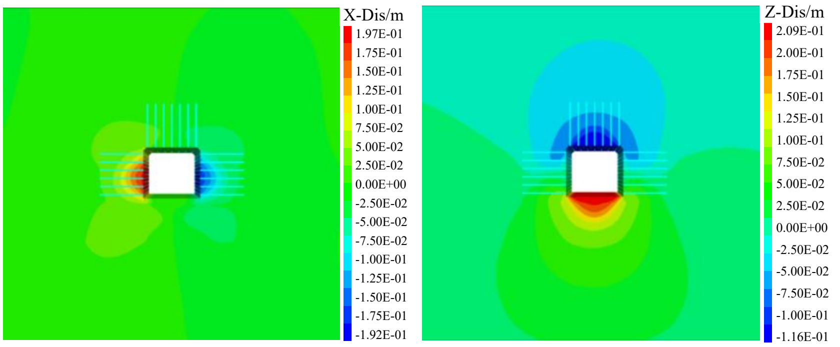
4.2. Optimization of Key Support Parameters

4.2.1. Influence of Different Spacing and Row Spacing of Anchor Cables on Roadway Stability

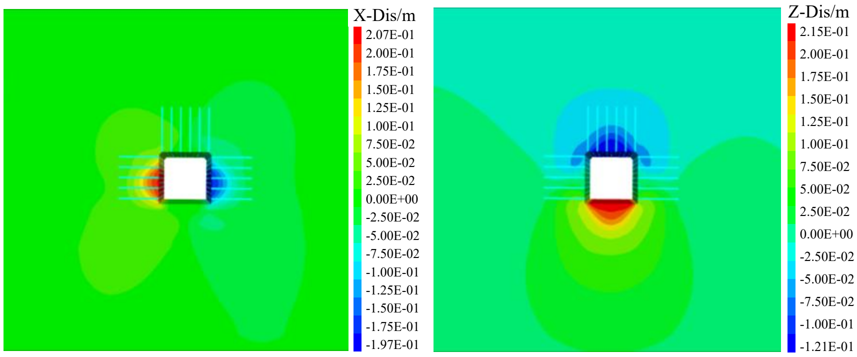
Figure 8 shows the nephogram of roadway surrounding rock displacement under different spacing and row spacing conditions. From the analysis of the relationship trend between surrounding rock displacement and anchor cable row spacing, the spacing and row spacing of 1000 mm shows significant advantages. When the spacing and row spacing is reduced from 1200 mm to 1000 mm, the attenuation amplitude of surrounding rock displacement is the most prominent: the roof subsidence is reduced by 14 mm with a reduction rate of 10.61%, the convergence of the two roadway sides is reduced by 16 mm with a reduction rate of 3.79%, and the floor heave is reduced by 9 mm with a reduction rate of 4.17%. The inhibition effect is far better than that of other adjacent row spacing intervals.

From the analysis of the relationship trend between support cost and support effect, the spacing and row spacing of 1000 mm has more advantages than that of 800 mm or lower. The spacing and row spacing of 800 mm can only achieve an additional displacement reduction of 6 mm for the roof, 4 mm for the floor heave, and 9 mm for the two roadway sides compared with the 1000 mm scheme, with corresponding reduction rates of 5.36%, 2.79%, and 3.71% respectively, showing a negligible control effect. However, the support density of the 800 mm spacing and row spacing scheme increases by 20%, leading to a rise in support cost. In contrast, the 1000 mm spacing and row spacing scheme effectively avoids unnecessary cost waste by reasonably reducing the support density on the premise of ensuring displacement control effect.

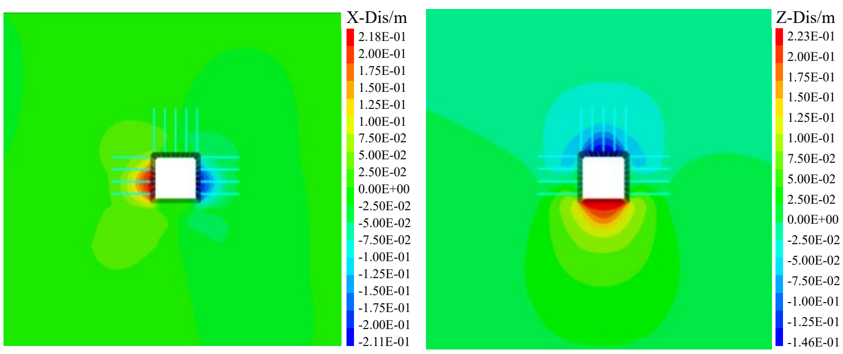
It is determined to adopt the combination of 4.3 m short anchor cables and 8.3 m long anchor cables, with the optimal spacing and row spacing of 1000 mm × 1000 mm. The long and short anchor cables are arranged in a staggered and parallel manner with an interval of 500 mm, which can realize effective superposition of the stress fields.



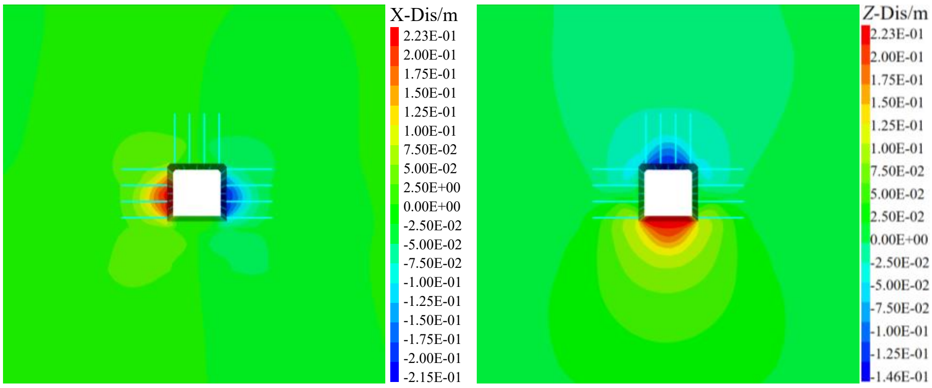
(a) spacing and row spacing 800mm



(b) spacing and row spacing 1000mm



(c) spacing and row spacing 1200mm

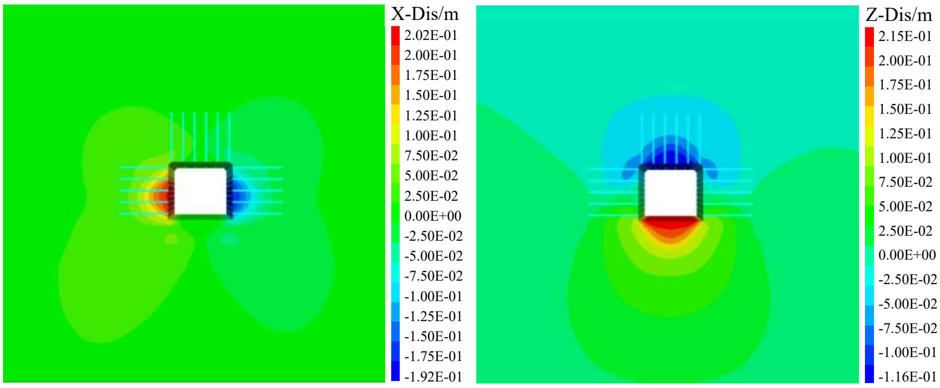


(d) spacing and row spacing 1400mm

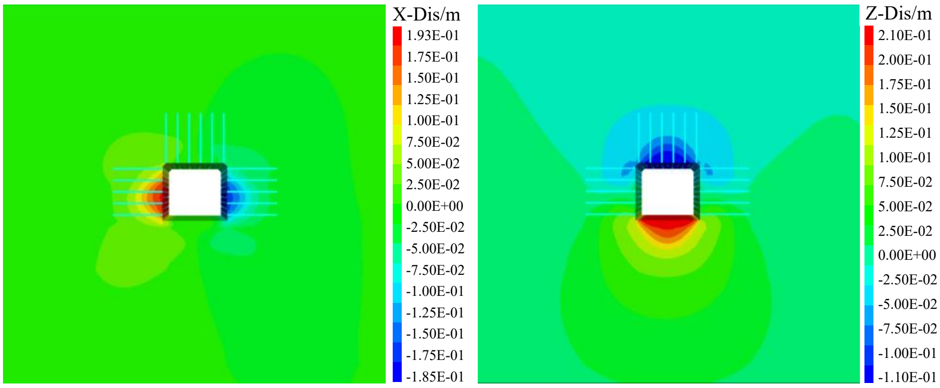
Figure 8 Nephogram of Roadway Displacement Distribution Under Different Spacing and Row Spacing Conditions

4.2.2. Influence of Different Pretightening Forces Applied to Anchor Cables on Roadway Stability

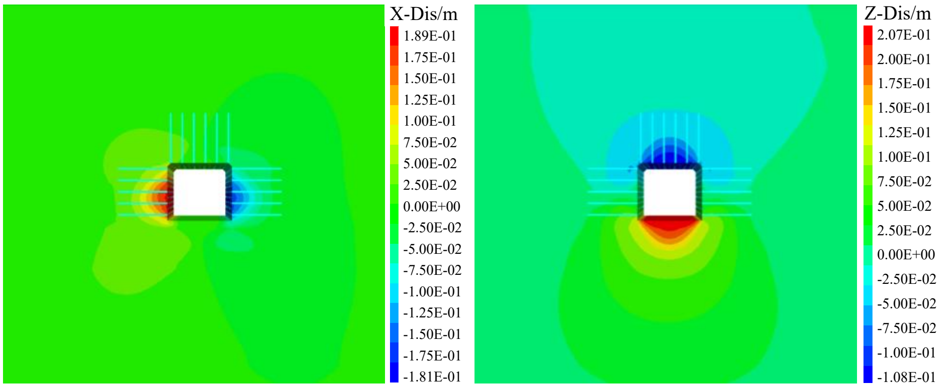
Figure 9 shows the nephogram of roadway surrounding rock displacement under different pretightening forces applied to anchor cables. From the analysis of the relationship trend between surrounding rock displacement and pretightening force, the reduction amplitude of surrounding rock displacement is no longer significant when the pretightening force exceeds 300 kN, so the pretightening force is determined to be 300 kN. Under the pretightening force of 300 kN, the roof subsidence, convergence of the two roadway sides, and floor heave are 101 mm, 339 mm, and 200 mm respectively, with a safety redundancy of 49 mm, 111 mm, and 50 mm from the corresponding safety thresholds, and all surrounding rock displacements are within the safe and controllable range. When the prestress is increased to 300 kN, further increase cannot significantly improve the control effect, and no additional investment is required. The theoretical breaking force of the ϕ28.6 mm high-strength anchor cable is 990 kN, and the pretightening force of 300 kN only accounts for 30.3% of its breaking force, which complies with the provisions of the safety limit. In summary, the optimal pretightening force applied to the anchor cables is selected as 300 kN.



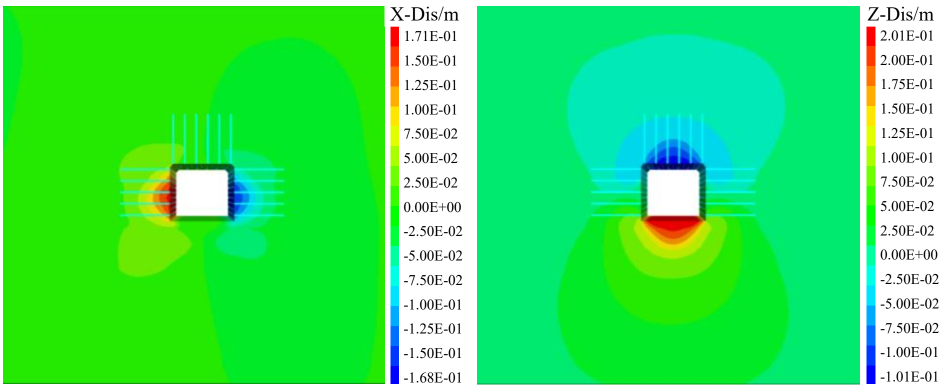
(a) 150kN



(b) 200kN



(c) 250kN



(d) 300kN

Figure 9 Nephogram of Surrounding Rock Displacement Under Different Pretightening Forces

4.3. Numerical Simulation Comparison of Support Effects

The numerical simulation results show that the optimized hierarchical coupling support scheme has significant advantages over the original support system:

(1) Displacement control effect: Under the optimized scheme, the convergence of the two roadway sides is 227 mm, the roof subsidence is 80 mm, and the floor heave is 106 mm, which are 47.9%, 46.2% and 40.1% lower than those of the original support system respectively.

(2) Plastic zone control effect: Under the original support system, the range of the plastic zone at the two roadway sides reaches 8–9 m, which is coalesced with the plastic zone of the adjacent roadways. Under the optimized scheme, the plastic zone at the two roadway sides is reduced to 3–4 m, the roof plastic zone is 4–5 m, and the floor plastic zone is 3–4 m, with no coalescence with the adjacent roadways.

(3) Stress field evolution: The optimized scheme can effectively improve the stress state of surrounding rock, reduce the degree of stress concentration, make the stress distribution of surrounding rock more uniform, and give full play to the self-bearing capacity of surrounding rock.

5. CONCLUSIONS

(1) This paper reveals the hierarchical coupling control mechanism of "large-diameter high-strength anchor cables + hierarchical anchoring with long and short anchor cables + deep and shallow grouting" for deep soft rock roadways under strong dynamic mining pressure. Through the three-level composite load-bearing system, the collaborative control of the shallow fractured zone and the deep plastic zone is realized, which solves the problems of insufficient strength and mismatched anchoring depth of conventional support systems.

(2) For the first time, the ϕ28.6 mm large-diameter high-strength anchor cables are applied to the deep soft rock roadway under strong dynamic mining pressure in Hudi Coal Mine. Under the pretightening force of 300 kN, its working resistance is 71.4% higher than that of the ϕ21.8 mm anchor cables, with a significantly improved safety redundancy, which effectively avoids the fracture of anchor cables under mining-induced disturbance.

(3) The support parameter system suitable for the 1101 roadway is optimized and determined. Numerical simulation and field tests show that this technology can reduce the convergence of the two roadway sides, roof subsidence and floor heave by 47.9%, 46.2% and 40.1% respectively compared with the original support system. The surrounding rock deformation is effectively controlled, and the technology has significant engineering popularization and application value.

ACKNOWLEDGMENT

We thank Associate Professor Tongqiang Xiao, Jiahao Li, Weilong Zheng, and Yuanbo Dou.

REFERENCES

[1] Q. Li, “Research on comprehensive characterization of deep coal full aperture structure and burial depth effect,” Scientific Reports, vol. 15, no. 1, Feb. 2025.

[2] J. M. Feng, “Control Mechanism of Large Deformation in Soft Rock Tunnel Based on Polyurethane Foam Buffer Layer,” International Journal for Numerical and Analytical Methods in Geomechanics, vol. 49, no. 18, pp. 4279-4292, Dec. 2025.

[3] Q. X. Huang, “Study on large deformation mechanism and surrounding rock control of entry in "three soft coal seam" of deep mine,” Scientific Reports, vol.15, no. 1, Aug. 2025.

[4] D. Z. Zhu, “Research on the Deformation Mechanism and Control Technology of the Floor in Deep Soft Rock Roadway,” Geofluids, vol. 2025, no. 1, Jun. 2025.

[5] Y. H. Zhu, “Stability of Research Methodologies of Underground Surrounding Rock Engineering: International Conference on Mechanics, Building Material and Civil Engineering,” International Conference on Mechanics, Building Material and Civil Engineering, pp. 1014-1019, 2015.

[6] Y. Zhu, “Numerical Investigation of the Pull-Out and Shear Mechanical Characteristics and Support Effectiveness of Yielding Bolt in a Soft Rock Tunnel,” Applied Sciences-Basel, vol. 15, no. 12, pp. 6933, Jun. 2025.

[7] F. Wu, “Creep Instability Mechanism and Control Technology of Soft Coal Roadways Based on Fracture Evolution Law,” Applied Sciences-Basel, vol.13, no. 16, pp. 9344, Aug. 2023.

[8] Q. W. Zhu, “Failure and stability analysis of deep soft rock roadways based on true triaxial geomechanical model tests,” Engineering Failure Analysis, vol.137, pp. 106255, Jul. 2022.

[9] S. P. Jia, “Experimental and numerical analysis of deformation and failure behaviour for deep roadways in soft rocks,” Bulletin of Engineering Geology and the Environment, vol. 81, no. 11, pp. 466, Nov. 2022.

[10] J. X. Zhou, “Research on nonlinear damage hardening creep model of soft surrounding rock under the stress of deep coal resources mining,” Energy Reports, vol. 8, no. S4, pp. 1493~1507, Nov. 2022

[11] H. P. Kang, “A combined “ground support-rock modification-destressing” strategy for 1000-m deep roadways in extreme squeezing ground condition,” International Journal of Rock Mechanics and Mining Sciences, vol. 142, Aug.2021.

[12] W. D. Yeng, “Coupled analytical solutions for circular tunnels considering rock creep effects and time-dependent anchoring forces in prestressed bolts,” Tunnelling and Underground Space Technology incorporating Trenchless Technology Research, vol. 134, Jun.2023.

[13] S. Shreedharan, P. Kulatilake, “Discontinuum-equivalent continuum analysis of the stability of tunnels in a deep coal mine using the distinct element method,” Rock Mech. Rock Eng., vol. 49, no. 5, pp. 1903-1922, May 2016.

[14] Q. X. Fan, “Study on deformation and control measures of columnar jointed basalt for baihetan super-high arch dam foundation,” Rock Mech. Rock Eng., vol. 51, no. 2, pp. 1-27, Aug 2018.

[15] C. C. Li, “Field observations of rock bolts in high stress rock masses,” Rock Mech. Rock Eng., vol. 43, no. 4, pp. 491-496, Jul 2010.

[16] T. Q. Xiao, “Study on Similarity Model Test of Surrounding Rock Stability for Deep Roadways with Thick Roof Coal,” Journal of China Coal Society, vol. 53, no. 6, pp. 1016-1022, Aug 2014.

[17] Y. Z. Wu, “Characteristics of Deformation and Failure of Surrounding Rock in Roadway under Strong Impact Load and Control Technology,” Coal Science and Technology, vol. 52, no. 9, pp. 76-87, Apr. 2024.

[18] Q. L. Meng, “Study on Interaction between Surrounding Rock and Bolt-shotcrete-U-shaped Steel Support Structure in Deep Soft Rock Roadways,” Coal Science and Technology, vol. 52, no. 7, pp. 23-36, Aug. 2024.