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Research Article

Failure Characteristics and Stability Control of Bolt Support in Thick-Coal-Seam Roadway of Three Typical Coal Mines in China

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Roadways in thick coal seams are widely distributed in China. However, due to the relatively developed cracks and brittleness of coal, the support failure of thick-coal-seam roadways frequently occurs. Therefore, the study of bolt failure characteristics and new anchoring technology is very important for the safety control of thick-coal-seam roadways. Based on field observations, the failure mechanism of selected roadway failures under distinct conditions at three representative coal mines in eastern and western China was analyzed. Recommendations are provided for roadway safety control. The results show that the strength and dimension of the anchoring structure in the coal roof of thick-coal-seam roadways are the decisive factors for the resistance of the roadway convergence and stress disturbance. The thick anchoring structure in the roof constructed by flexible long bolts can effectively solve the problem of support failure caused by insufficient support length of traditional rebar bolts under the condition of extra-thick coal roof and thick coal roof with weak interlayers. The concepts and techniques presented in the paper provide a reference for the design of roadway support under similar geological conditions and dynamic load.

1. Introduction

The rapid development of bolt support technology has become a critical component of efficient mining and offers significant economic and social benefits. However, the stability criteria of the surrounding rock in coal roadways under certain conditions, especially the roof, remain poorly understood. Uncertainties in the existing coal roadway bolt support system have therefore led to support failure and roof fall accidents [1–3]. Roof accidents owing to roadway support failure constitute a large proportion of coal mine accidents. For example, 9013 coal mine accidents occurred in China between 2008 and 2018 (Figure 1); among them, 42.8% were roof accidents that accounted for 31.7% of all mining-related fatalities. A better understanding of the support failure mechanism of coal roadways and bolt action mechanisms is therefore critical to achieve high levels of safety and quality in coal mining.

Several attempts have been made to improve control over surrounding rock stability by optimizing the supporting concept and design method. Yang [4] identified the instability mechanism of a deep and highly deformed soft rock roadway according to the results of field monitoring and numerical simulations, and proposed a combined support method of "anchor mesh spray + shell" to control deformation. Frith [5] proposed that the supporting effects of the roof anchoring system can be further enhanced by promoting or maintaining the specific way in which the horizontal layered roof rock layer loses its natural self-supporting ability. Reza [6] studied the energy dissipation ability of rockbolts under high-stress and dynamic loading conditions and recommended suitable bolt types for various

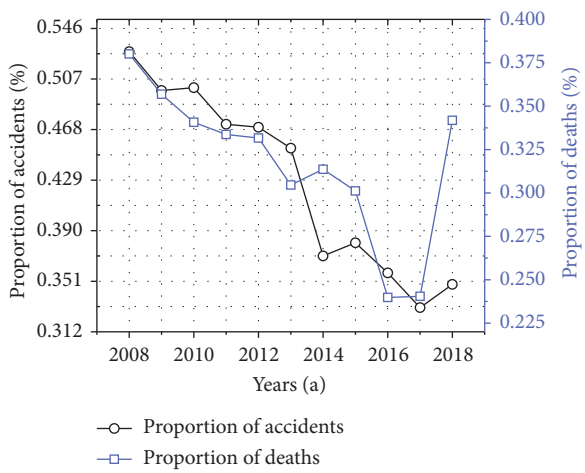


Figure 1: Proportion of coal mine roof accidents and deaths in China from 2008 to 2018.

ground energy requirements and deformation capacity ranges. Kaiser [7] distinguished the support design of a roadway prone to rock burst from a traditional rock roadway and proposed seven principles of rock burst support design, three important roles of rock burst support, and four design acceptance standards. Based on rock mass rating and rock tunneling quality index systems, Mohammad [8] proposed a mathematical model to develop the rock bolt support mechanism, including the bolt support coefficient, which can evaluate the ability of a specific rock body to be strengthened by anchor bolts.

However, in view of the uncertain geological conditions and concealment of the support system, potential risks remain difficult to accurately predict [9, 10]. This dilemma makes it difficult for mathematical analysis, numerical simulations, physical experiments, and other research methods to take into account the roadway destruction and deformation characteristics under real geological conditions [11–13]. Under such circumstances, field observations have received the most attention and remain the most original and effective method in mining engineering. Li [14] studied the failure characteristics of bolt support under high-stress conditions through field observations. Pells [15] provided the design method of bolt and shotcrete support in Sydney sandstone, as well as the calculation procedure of bolt length, density, and bearing capacity in this environment. Li [16] analyzed the failure of threaded steel bolt when soft rock deforms under high ground pressure and proposed that traditional steel bolts should be replaced by increased thread diameters or headed steel bolts. Compared with mathematical modeling, numerical simulations, physical experiments, and field observation can more intuitively observe roadway deformation and failure, as well as the mechanical properties of the tunnel supporting structure and its interaction with the surrounding rock [17, 18]. Moreover, based on precise consideration of various complex geological structures and engineering environments, field observation is more suitable for support failure [19, 20], which makes it the best tool for evaluating current support failures and further optimizing support solutions.

We selected three representative coal mines from the east and west of China for field observation of support failure, including the Jianxin coal mine and Wenjiapo coal mine in Shaanxi Province and Yaoqiao coal mine in Jiangsu Province. The failure modes of support under specific geological conditions are analyzed, and targeted support schemes and suggestions are provided. In particular, we discuss the increased bolt length, effectiveness of the pretension force in bolt support, application value of steel belts, and the relationship between heading advance and support failure. Related technologies and design concepts provide a reference for support design under similar geological conditions and dynamic load roadway to a certain extent to avoid unnecessary or invalid support schemes.

2. Support Failure of High-Stress and Intense Mining Roadway in Deep Mines

2.1. Support Scheme and Failure Description. These project cases are selected from Jianxin Coal Mine's 4203 ventilation roadway (VR 4203) located in Huangling County, Shaanxi Province, with an average burial depth of 810.5 m. The thickness of the coal seam is between 5.4 and 10 m, and the average thickness is 6.4 m. Table 1 shows the stratigraphy of VR 4203. The VR 4203 belongs to the gob-side entry and is separated from the goaf of the 4201 by 15 m large coal pillars, as shown in Figure 2(a). It is obviously affected by the side abutment stress from goaf, and the cracks are more developed. In Figure 2(b), the section of this roadway is rectangular, 5200 mm in width, and 3800 mm in height. The roof is supported by steel rockbolts (seven bolts in a row) whose diameter is 20 mm and length is 2500 mm. Besides, each row has four cables whose diameter is 21.8 mm and length is 8300 mm. The row spacing of bolts and cables both is 800 mm. Due to the serious imbalance between mining and excavation, the roadways of this coal mine are usually placed for 1 to 2 years after the drivage tunneling, which makes it difficult to control the deformation. The damage situation of the roadway taken by the camera is shown in Figure 2(c). The roadway shows clear evidence of asymmetric deformation and failure. The roof rock is fragmented and the amount of sagging in the middle of the roof exceeds 1500 mm. Anchorage failure phenomena are also observed, such as a broken steel belt, torn metal mesh, and broken anchor, and the resulting effects lead to unsatisfactory support.

To further understand the rock destruction inside the roadway roof, we used a borehole camera and corresponding analysis under the original support scheme, as shown in Figure 2(d). Cracks developed in the roof had penetrated the entire anchorage area of the bolts and cables, and annular cracks were also observed outside of the anchorage area with a maximum crack height of 8.58 m. The roof rock at 0–4.11 m depth was severely damaged and the integrity was extremely poor. Rock separation was observed at 1.07, 1.84, 2.83, and 4.11 m, which prevented the bolts from effectively transferring the tensile stress. Under these circumstances, the role of the anchor cable was then to suspend the shallow broken rock stratum in the upper stable rock, which greatly

Table 1: Rock properties and thickness.

	Rock stratum	Thickness (m)
Roof	Fine sandstone	8.6
	Siltstone	3.0
	Carbon mudstone	0.79
Coal	4-2# coal	6.4
Floor	Fine sandstone	0.5
	Sandy mudstone	1.67

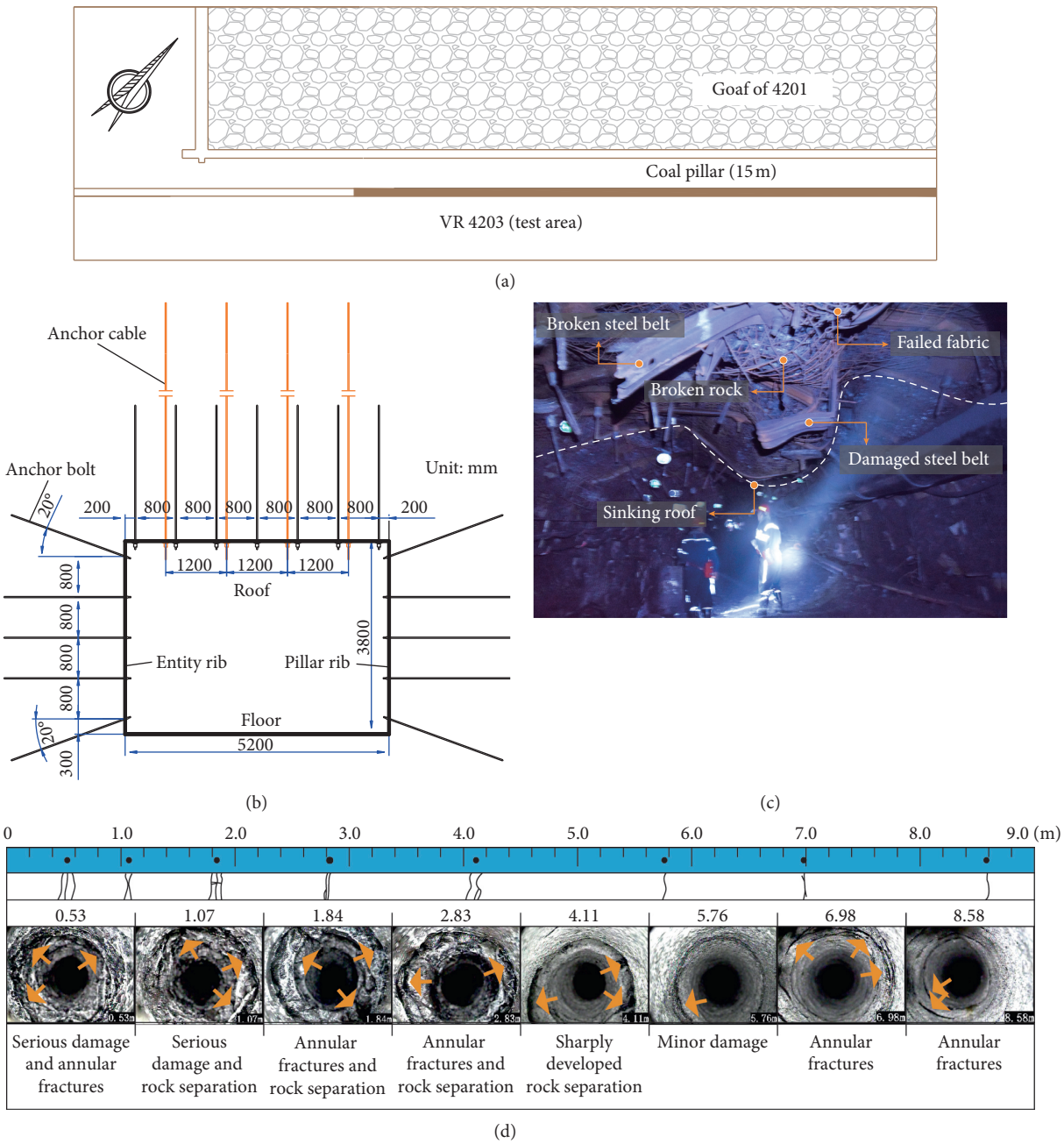


Figure 2: The original support scheme and the serious damage of roadway: (a) layout of the test areas; (b) original support scheme; (c) photos of the roadway; (d) borehole camera images and corresponding analysis.

weakened the active support of the bolt and cable. Crack propagation was not significantly suppressed and the rock mass continued to deform under the high in situ stress and mining pressure, leading to an exceedingly high risk of overall roof collapse. Therefore, in this case, it is urgent to increase the anchoring depth and enhance the support strength of the roof to ensure an appropriate thickness and effectiveness of the anchoring structure.

Based on the above understanding, the engineering staff has made changes to the existing support process in order to improve the support quality and reduce the cost. In the new support scheme, the diameter of the roof bolt is 22 mm and the length is 3000 mm. The pretightening torque is changed from 200 N m to 240 N m. The length of the cable is changed to 6500 mm, and the pretension force is changed from 100 kN to 150 kN because most engineers and technicians set reliable bolt parameters, including bolt length, diameter, and preload, according to their previous experience [21]. Other parameters are the same as the original scheme. Although various considerations and changes have been made, unfortunately, the objective of support effect was not reached.

2.2. Refined Characterization of Support Failure. In order to grasp the detailed deformation and stress rules of the roadway during experiment, field observations were carried out as shown in Figure 3. Figure 3(a) records the results of the axial load of the bolts (B1 and B2) and the anchor cables (C1 and C2) which were measured by digital dynamometer. The pretension force of B1, B2, C1, and C2 is 21 kN, 18 kN, 45 kN, and 43 kN, and the axial load is stabilized finally at 57 kN, 51 kN, 95 kN, and 93 kN, respectively.

Figure 3(b) shows the monitoring results of surface displacement. The roadway displacement mainly occurs within the distance of 20 m to the excavation face, where the roof deformation and rib convergence increase sharply by 82% and 87.5%, respectively. The final sagging of the roof is 251 mm and the rib convergence is 280 mm.

The monitoring results of borehole camera in the roof are shown in Figure 3(c). Similar to the original support, the maximum fracture depth in the roof reaches 7.31 m, which exceeds the anchorage area of cables. At the depth of 0.47 m, 1.31 m, and 3.42 m in the roof, the developed rock separation can be observed, and the shear and tensile failure of the rock at 3.42 m can be clearly observed which blocks the effective transmission of bolt tensile stress to deep strata. Compared with the original support, the stability of the surrounding rock is not well controlled (even if measures such as strengthening the strength, increasing the length, and increasing the pretension force have been applied to the bolt), and the risk of roof fall is still very serious.

2.3. Support Failure Mechanism. According to the specific engineering geological conditions of Jianxin Coal Mine and the abovementioned field observation analysis, the support failure mechanism can be considered from the following aspects.

2.3.1. High-Stress Field Environment. The stability of the rock surrounding the mining roadway must consider the influence of mining depth and the disturbance that mining activity imposes on the original rock stress field over a certain range. This is one of the important principles of coal mine roadway support design. Roadway VR 4203 has a burial depth of 810 m, which presents a control problem that is typical for deep high-stress roadways. This depth exposes the roadway to rock deformation across the brittle-plastic transformation, associated rheology, and long-term expansion characteristics [22]. The VR 4203 roadway is separated from the #4201 goaf by 15 m of coal pillars (Figure 2(a)), which strongly affects the residual bearing pressure. The amount of floor heave during roadway excavation reaches 600–700 mm. The superimposed effects of high in situ stress and mining stress exacerbate the stress concentration phenomena in this roadway, which makes the support work particularly challenging.

2.3.2. Lower Pretension of Bolts and Cables. The monitoring results of the axial load on the bolt rods and cables (Figure 3(a)) show that the pretension force is dangerously low. The different geological conditions make sure the roadway support parameters are also different. The damage of the support structure under the high-stress environment is greater than the low-stress in the shallow part. Therefore, there should be more strict requirements on the preloading force loaded on the anchor bolt and anchor cable, to ensure the safety of the roadway roof. To simplify operation and construction, the engineering staff did not install an anti-friction gasket, which reduces the pretension force of the bolt and its conversion factor. We checked the tension of the anchor cable tensioner and found that the designed 150 kN can actually only reach 84 kN. Further prestress loss occurs during the transmission of pallets, steel belts, and broken rock mass, which lowers the effective pretension of the anchor cable to only 43–45 kN. The small pretension force causes the bolts and cables to exert lower axial restraint force on the rock mass, and an effective pressure arch structure cannot form in the roof. The stability of an artificial pressure arch is a crucial factor for controlling roadway deformation [23, 24]. As shown in Figure 4(a), when the artificial pressure arch does not provide an effective axial binding force to the roof rock, it cannot control the early rock deformation after roadway excavation, and therefore results in the continuous expansion of small original cracks in the rock that eventually form large fractures and lead to rock separation. The roadway also exhibits large deformation and failure, as shown in Figure 4(b).

2.3.3. Insufficient Thickness of the Shallow Anchoring Structure. The average coal seam thickness in the test area of the VR 4203 roadway is 6.4 m, the roadway height is 3.8 m, and the thickness of the coal seam above the roadway is 2.6 m. The bolt length in the design of a new support scheme is 3 m or an effective length of 2.85 m when excluding the exposed tail length. This shows that the end of the bolt was 0.25 m anchored in the carbonaceous mudstone above the

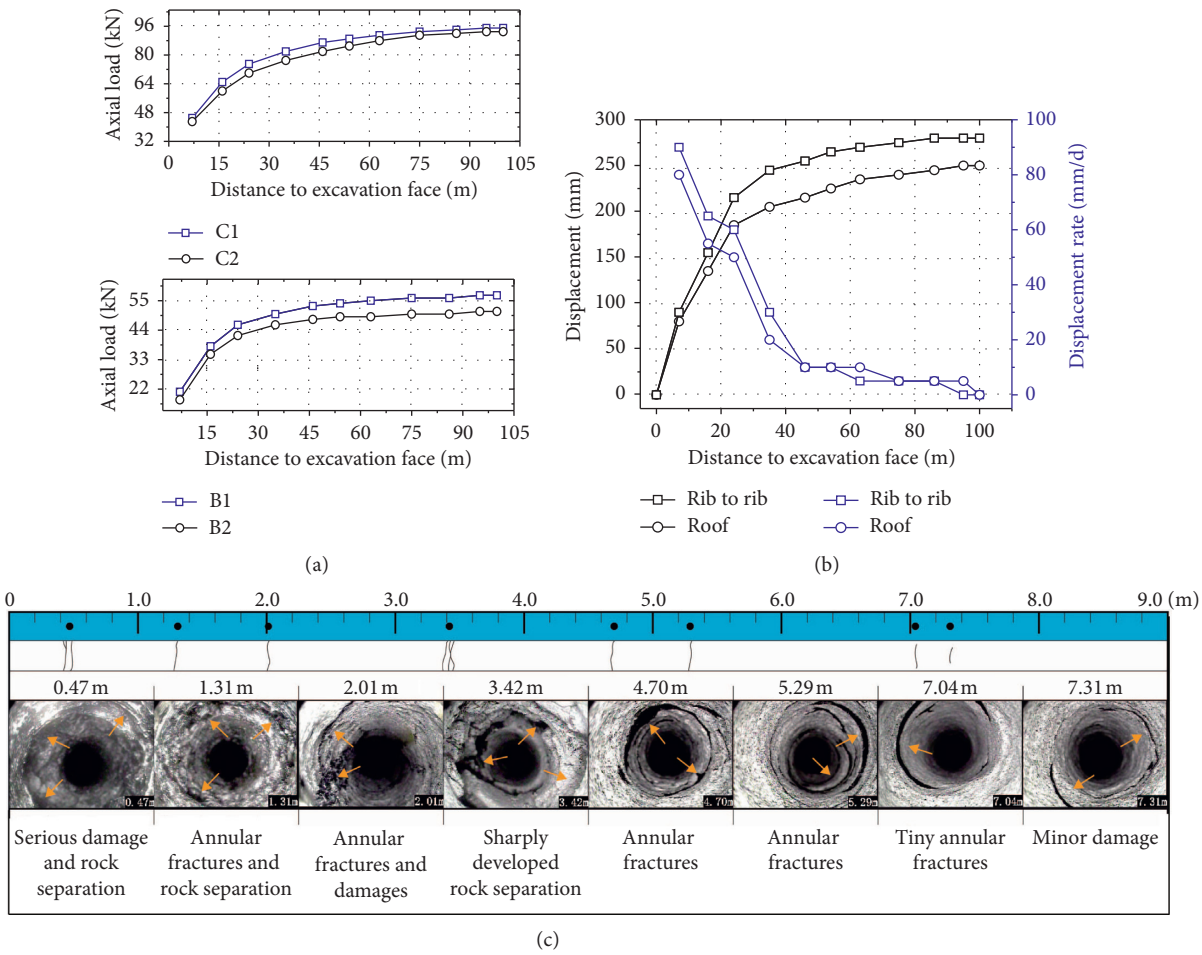


Figure 3: The results of the roadway monitoring: (a) the monitoring results of surface displacement in the roadway; (b) the monitoring results of axial load of flexible bolts and cable; (c) borehole camera images and corresponding analysis.

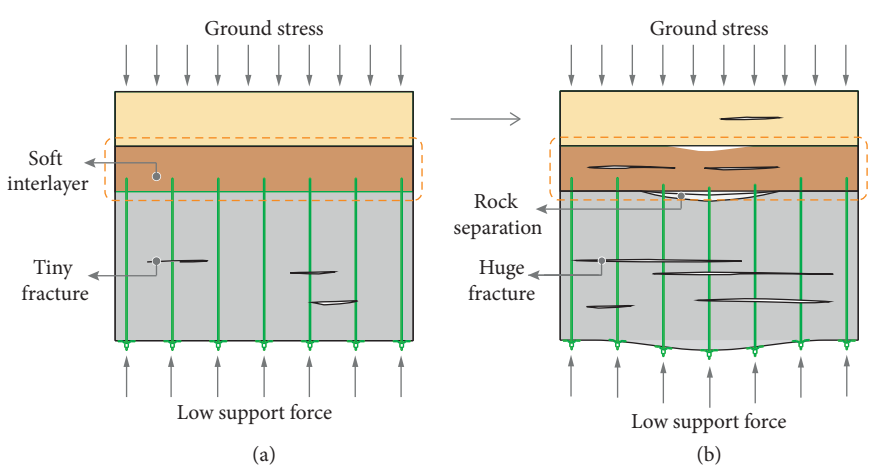


Figure 4: Support failure mechanism: (a) initial small deformation of rock mass; (b) rock deformation and rock separation failure.

coal seam, as shown in Figure 4. This type of rock is not conducive to the transmission of axial force to the bolt in end-anchoring mode [25]. The low strength of the carbonaceous mudstone causes its destruction first, which when stressed leads to further damage of the rock at the anchoring edge of the bolt. The bond strength at the interface between the bolt and rock mass is therefore reduced and an effective pressure arch structure within the reinforcement range of the bolt cannot form. The lack of pretension force further aggravates this situation. Significant rock separation

eventually occurs within the rock mass, as shown in Figures 3(c) and 4(b).

2.3.4. Lack of Early Warning Mechanism. Roof accidents are often highly concealed, sudden, and dangerous. When faced with complex engineering geology (e.g., high-stress fields, extremely broken rock mass), roadway support failure and roof fall hazards remain prominent even when a reasonable roadway layout and support design are applied [20]. The most effective method is therefore to ensure that the supporting materials fully contribute in their supporting role to prevent the occurrence of support failure, which requires the establishment of a complete monitoring system of early warning response mechanisms.

Mathematical prediction models of support failure induced by the separation of the roof rock under the given geological conditions should be established, and the deformation process of surrounding rock cracks and expansion of the separation layer should be monitored. When the deformation speed or total deformation exceeds the limit, corresponding local reinforcement measures should be taken. Another aspect is a monitoring plan for maintaining a high pretension force of bolts and cables. The stability of the rock surrounding a roadway during the entire maintenance period decreases even when the most advanced and reliable support methods are used. This implies a fluctuation of the axial force of the bolt. A portion of the pretension force of the bolt will be lost during fragmentation and destruction of a shallow rock mass, which reduces the strength of the artificial pressure arch. The maintenance of bolt and cable pretension should therefore also be added to the monitoring plan. When loosening of the pretension force is detected, the bolt should be retorted and the anchor cable should be pretensioned to ensure an effective pretension force.

3. Cost-Effective Support Scheme for Roadways in Deep and Extra-Thick Coal Seams

3.1. Engineering Problem Description. The project case presented in this section is based on the 4106 ventilation roadway (VR 4106) of Wenjiapo Coal Mine in Binzhou City, Shaanxi Province. The burial depth of the coal seam where the roadway is located is 730 m, and the average thickness is 11.2 m, which has the characteristics of a typical extra-thick coal roadway, and the supporting structures of the bolts and cables are located in the coal seam, as shown in Figure 5. The width of the roadway is 5500 mm and the height is 3850 mm. The roof is supported by eight bolts in each row whose diameter is 22 mm and length is 2500 mm. The pretension force of each bolt is 30 kN and row spacing is 800 mm. Besides, each row has five cable bolts whose diameter is 21.8 mm and length is 7100 mm. The pretension force of each cable bolt is 60 kN and row space is 800 mm. In addition, bolts and cables are used with W-steel tape and I-beam, respectively. Similar to Section 2, the roadway was emplaced 1–2 years after excavation, which makes the long-term maintenance problem exceptionally prominent. Engineers

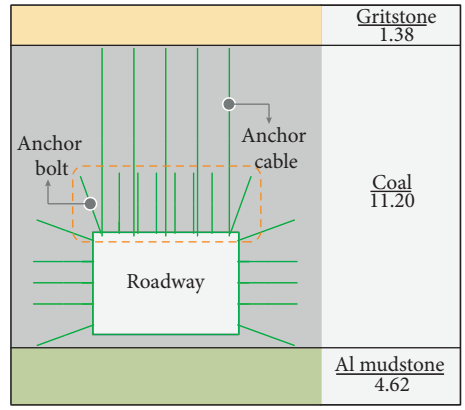
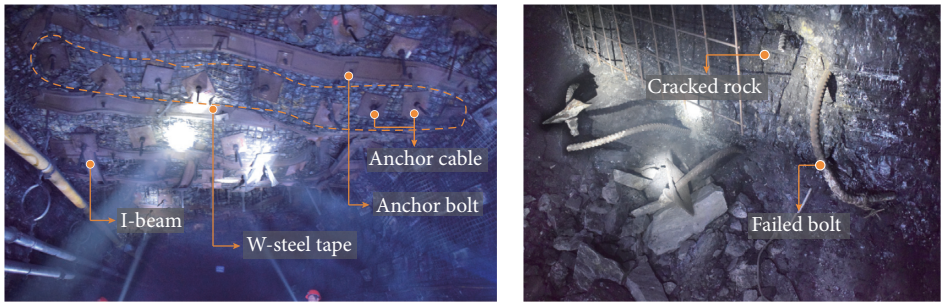


Figure 5: The original support scheme of the roadway.

usually perform multiple reinforcements on the basis of the original support plan before mining to meet the roof support strength. This leads to a very high final roadway support density, as recorded in Figure 6(a). Anchor cables are added in the middle of each row of bolts with an average of seven per row and up to nine anchor cables at most. However, even if the roof adopts such a large supporting density, the average amount of roof sagging after the drivage was more than 200 mm. Under the long-term influence of the high-stress environment, it is often necessary to repair the part of largely deformed roadway to meet requirements, as shown in Figure 6(b).

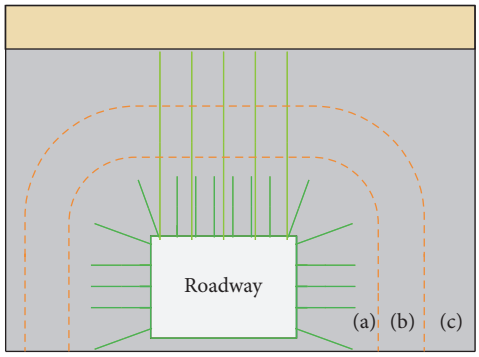
3.2. Failure Mechanism Investigation of the Original Scheme. For extra-thick (>10 m) coal seams, the coal seam thickness above the roadway roof usually exceeds 6 m. The roadway support structure in these cases is arranged in the coal seam, and the anchor cable cannot be anchored through the top coal to the stable rock layer above the coal seam. However, the shear strength of coal is lower than that of other rocks and primary cracks are abundant. When the top coal is affected by excavation, it microscopically shows as a large number of joint cracks that develop, expand, and distort within the coal body. This is manifested as delamination failure in the thick roof coal seam [26], which extends and deforms so that the size of the plastic failure zone of the rock surrounding the roadway roof is substantially larger than in conventional roadways. The design of this type of roadway that relies on an engineering analogy to conventional roadway design methods undoubtedly increases the risk of support failure.

Roadway excavation and unloading cause a redistribution of rock mass stress. The surrounding rock is divided into a fracture zone, plastic zone, and intact zone from the inside outward according to the fracture state of the surrounding rock [27], as shown in Figure 7. In this example, a 2.5 m steel rockbolts designed by the engineering analogy cannot come through the surrounding rock fracture area, so it is impossible to reach the required pretension force in the bolt. Insufficient rock support will increase the width of the fracture zone, especially in very thick coal seams. Therefore, the lack of bolt support will affect the width of the fracture zone. When the crack area becomes larger, the quality of bolt



(a) (b)

Figure 6: Photos of support failure: (a) high-density cable support for roadway roof; (b) the site of the enlarged roadway section.



(a) Fracture zone
(b) Plastic zone
(c) Intact zone

Figure 7: Roadway roof failure zone in thick coal seam.

support will also be reduced. This results in a large number of internal and external separation cracks in the bolt anchorage area, and the integrity of the roof rock mass is severely damaged. The strength and stability of the shallow bearing structure constructed by the bolt are greatly reduced. When joint cracks within the range of the bolt support extensively expand, the pretension force applied by the anchor cable cannot be eliminated and the shallow bearing structural strength and rock mass integrity cannot be enhanced. By this time, the role of the anchor cable is more aimed to suspend the shallow broken rock mass, and the role of active reinforcement is not exerted. Large deformation of the roadway therefore remains inevitable even if the onsite engineering staff apply extensive supporting material and repeatedly carry out anchor cable reinforcement support. The key to controlling the surrounding rock in this case is to extend the length of the bolt, which can eliminate the adverse impacts of the ruptured rock mass on the shallow bearing structure and improve its strength by using a thick rock-bearing structure to resist its own deformation and other engineering disturbances.

3.3. Evaluation of Solutions and Effects Based on Flexible Bolts. Thread steel rockbolts are widely used because of their convenient processing and use. Restricted by roadway height and construction difficulty, the length of the conventional threaded steel anchor is 2–2.5 m. 3-meter bolts were used in the Jianxin coal mine owing to the roadway height; however, a threaded steel anchor rod cannot be used if the rod length exceeds 3.5 m to form a thicker anchoring structure.

We developed a new type of flexible long bolt to increase the anchoring length. As shown in Figure 8, a flexible long bolt is composed of a steel stranded cable, a threaded tube, a self-aligning nut, a large arched tray, and a small antitwist tray. The flexible steel strand cable is used to make the bolt bendable, and the length is not limited by the roadway height, which can effectively increase the thickness of the artificial reinforcement arch in the shallow roadway layer and strengthen the support effect. The threaded sleeve and self-aligning nut serve as the anchor during rapid installation of the flexible steel strands, which effectively increase the speed of support and are conducive for the detection and maintenance of long-term pretension. The antitwist small pallet is in contact and is fixed with the large arched pallet and rock wall used to limit twisting of the steel strand during installation and rebound after twisting, thereby enhancing the support quality. The difference between flexible long bolt and the anchor cable is mainly reflected in the loading mode of its preload force. It is mainly different from traditional bolts with regard to the flexibility and tail structure of the rod. By using a flexible cable, the length of the strand bolt can be increased to 4 m. This can effectively improve the deflection and load performance of the base anchor structure.

The flexible long bolt is used to replace the threaded steel bolt, and the torque amplifier is used to increase the initial pretension of the flexible long bolt (as shown in Figure 9, the role of the torque amplifier is equivalent to the wrench). In the new scheme, the roof is supported by eight flexible long bolts in each row whose diameter is 17.8 mm, length is 4000 mm, and pretightening torque is not less than 320 N m. Resin capsules are used to bond steel strand to the rock mass. The size of the cable is the same as the original scheme, but row spacing is increased to 1600 mm. Both flexible long bolt and cable are used with a square large arch-pallet (size: 300 mm × 300 mm × 12 mm), and the W-steel tape and I-beam are eliminated.

Field observation results are shown in Figure 10. Figure 10(a) shows the results of axial load of flexible long bolts (F1 and F2) and cable bolts (C1 and C2). As is shown in

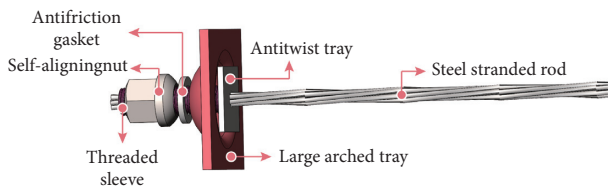


Figure 8: Structure of flexible cable strand bolt.

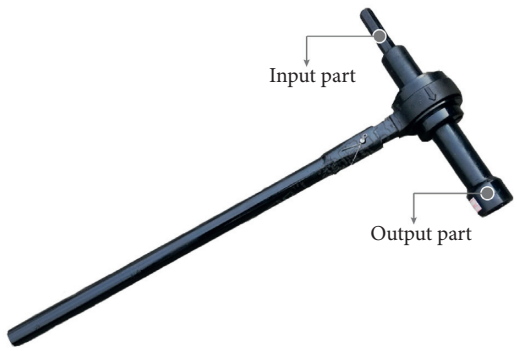


Figure 9: Torque amplifier.

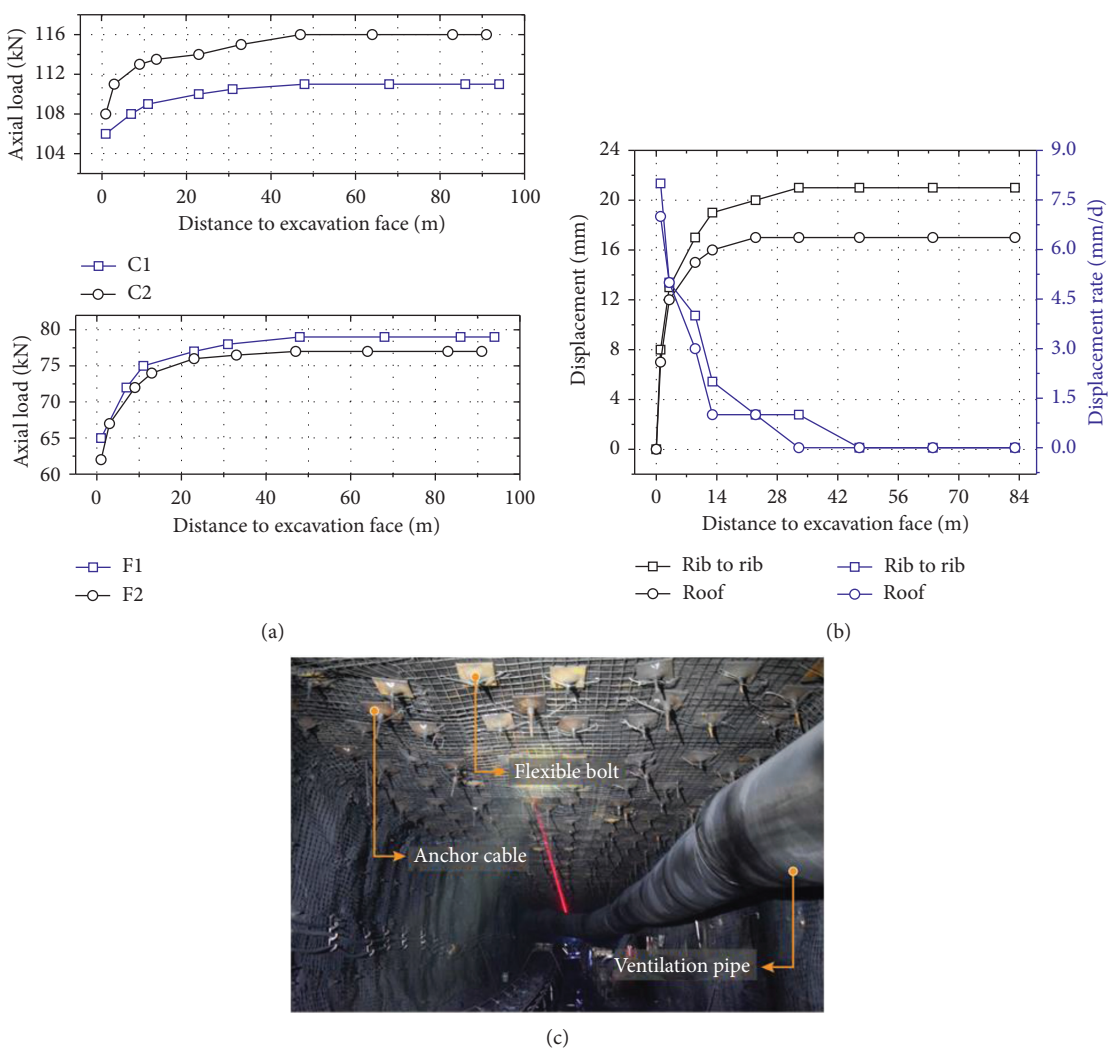


Figure 10: Field observation results and maintenance effects of the roadway: (a) monitoring results of the surface displacement in the roadway; (b) monitoring results of the axial load of the flexible bolts and anchor cables; (c) field photo of test roadway.

the dynamometers F1 and F2, the pretension force of flexible long bolts is 65 kN and 62 kN, and it is fixed finally at 79 kN and 77 kN, respectively. As is shown in the dynamometers C1 and C2, the pretension force of cables is 106 kN and 108 kN, and it is fixed finally at 111 kN and 116 kN, respectively. Since the flexible long bolts and the cables are in a high-stress linear elastic stage, they will come to function rapidly in response to any deformation in the surrounding rock of the roof, thus making the overall axial load variation range smaller and bringing the early roadway deformation under control [28]. Within the distance of 23 m to the excavation face, the axial load of the flexible long bolts increased by 85.7% and 93%, the axial load of the cables increased by 80% and 75%, respectively, and basically stabilized at 33 m to the excavation face.

Figure 10(b) shows the monitoring results of surface displacement in the roadway. Within a distance of 20 m to the excavation face, the surface displacement rises sharply, where the roof deformation and rib convergence increases by 81% and 88%, respectively. Then it experiences a mild increase and finally stabilizes at 23 m to the excavation face. The final sagging of the roof is 17 mm and the rib convergence is 21 mm.

The amount of roof delamination from the beginning to the end of the construction was monitored for two weeks. The average delamination of the roof in the depth of 0–4 m is 1.5 mm, and the average delamination of the roof in the depth of 4–8 m is 2 mm. The further expansion of the rock cracks in the fracture zone is well controlled by the flexible long bolt, which makes the roof deformation smaller. With the supporting of flexible long bolts, the control effect on the roadway surrounding rock has been improved remarkably, as shown in Figure 10(c) (the pictures were taken at the site).

A comparison of the total drilling length of the original scheme with the new scheme shows that the drilling length per meter of roof in the original scheme is 83 m, whereas that in the new scheme is 59.75 m, a reduction of 28% (the length of the bolt and the cable minus the length of the tail leakage after the support and the remaining distance is the length of the drilling). The amount of supporting materials (e.g., bolts, cables) also decreases when using fewer boreholes. This shows that the new scheme achieves safe, efficient, and economical results.

3.4. Mechanism Revelation and Experience Learned. The rock surrounding an ultrathick coal roadway is characterized by low strength and crack development, and the destruction depth of the surrounding rock is much deeper than that of conventional rock. Thread steel bolts designed by engineering analogy are limited by the roadway height, and the design length is usually 2.0–2.5 m and cannot pass through the fracture zone.

In this example, the thickness and stability of the shallow rock roof bearing structure are effectively improved using 4-meter flexible strand cable bolts. The strand cable bolts were anchored through the fracture zone (where the joint fractures of the surrounding rock are developed) to the deep rock mass with smaller crack openings and less engineering disturbance. The strength of the weakly deformed rock mass in the deep of the surrounding rock can therefore be used to control the largely deformed rock mass at shallow depth. A torque amplifier is used to quickly increase the initial pretension force of the flexible anchor (with an initial pretension force of 65 and 62 kN, respectively), which puts the bolt in a high-stress linear elastic stage that can effectively and quickly suppress the crack destruction features in coal such as opening, delamination, sliding, expansion, and loosening. This allows the initial roof deformation to be suppressed and the long-term roof stability to be effectively controlled.

The thickness and deflection of the bolt anchorage body in the rock layer are significantly improved by increasing the bolt length and pretension. The rock layer's ability to resist its own expansion and deformation is enhanced and it also has the ability to resist stronger engineering disturbances. This is also the reason why even if the amount of supporting material is reduced by 28%, the steel belt and I-beam are simultaneously eliminated and roadway deformation can still be controlled at 17 mm (roof) and 23 m (rib to rib).

4. Efficient Control of the Weak Interlayer Roof with Flexible Long Bolts

4.1. Engineering Geology and Original Support Scheme. The project case selected in this section is the 7704 haulage entry in Yaoqiao Coal Mine in Xuzhou City, Jiangsu Province. The burial depth of coal seam is 211–298 m, and the average thickness is 5.89 m. The immediate roof of roadway is a 0.94 m thick layer of sandy mudstone with low strength. Above the sandy mudstone is a key stratum composed of fine sandstone with an 8.52 m thick layer. The immediate floor is a 3.95 m layer of mudstone, which is easy to swell in water. Therefore, the floor of the roadway is built with 500 mm of coal to overcome the influence of mudstone, as shown in Figure 11.

The width of the roadway is 4800 mm and the height is 3300 mm. The roof is supported by six thread steel bolts in each row whose diameter is 20 mm and length is 2000 mm. The pretension force of each bolt is 40 kN and row spacing is 900 mm. Moreover, each row has two cables whose diameter is 17.8 mm and length is 7200 mm. The pretension force of each cable is 100 kN and row spacing is 2700 mm.

4.2. Failure Characteristics. The in situ stress under geological conditions at the Yaoqiao coal mine is small and the coal seam is relatively stable. The soft sandy mudstone above the coal seam seriously affects the stability control of the roadway. As shown in Figure 12(a), owing to the presence of sandy mudstone above the coal seam, there is a weak interlayer outside the anchoring end of the bolt, which is destroyed by the stress applied to the surrounding rock. The tensile force generated by the end of the bolt will further worsen this interlayer failure. In this case, increasing the bolt pretension is not ideal because the anchoring force of the bolt cannot be transmitted outside the anchoring structure. The key to this problem is therefore to increase the

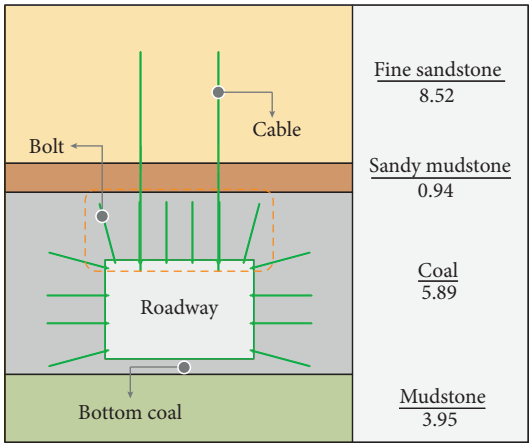


Figure 11: The original support scheme.

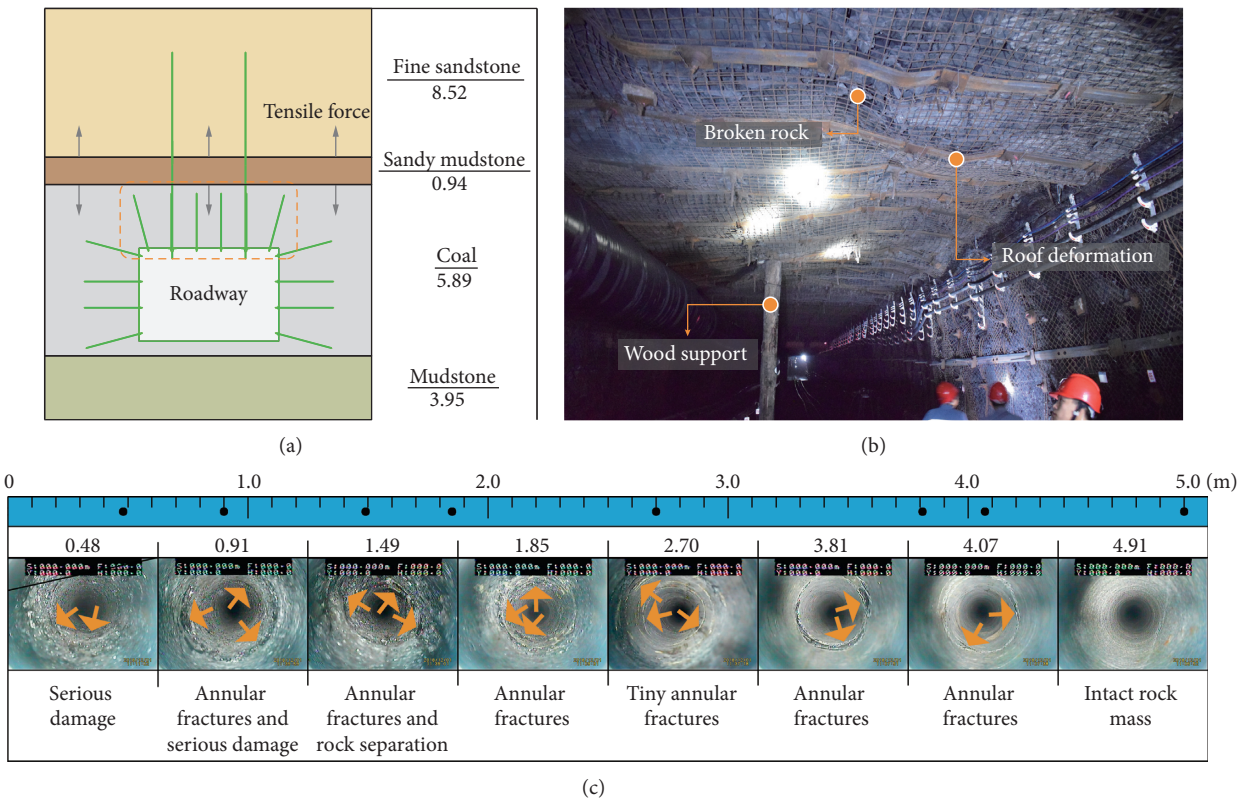


Figure 12: The failure characteristics of the original support scheme: (a) failure mechanism; (b) failure of supporting structure; (c) borehole camera images and corresponding analysis.

anchoring structure length. The length of the traditional steel rockbolts is limited by the roadway height and cannot be anchored into more stable, upper fine-grained sandstone by increasing the length. The engineering staff therefore adopted a 7.2 m anchor cable, but the supporting effect was not particularly strong.

As shown in Figure 12(b), support failure mainly manifests as shallow roof rock fragments, rock cracking in the rib, and irregular roadway deformation. To ensure further roof stability, the engineering staff even used wooden piles to support the roof, which did not play an effective supporting role and can be further clarified by borehole camera images.

Figure 12(c) shows the borehole camera images and corresponding analysis of the original support scheme. The maximum crack development height of the roof rock is 4.07 m, and the rock is severely damaged in the range of 0–1.85 m. The fragmentation is severe, a large number of circumferential cracks clearly develop, and the rock separation is found to be 1.47 m, which shows that crack

propagation in the roof failure area under the original support scheme has not been effectively suppressed, and the support effect is poor.

At the same time, a higher number of cables increase the support workload. The engineers used handheld drilling equipment to drill holes. The operation time of one cycle under the scheme construction was 67 minutes, the support time was 41 minutes, and the support construction required 61.2% of the total construction time. This creates a prominent imbalance between mining and excavation. Two or three excavation teams are required to complete the amount of excavation to ensure the normal mining of a working face, which doubles the mining cost.

A bolt length of 2 m is not functional in this case owing to a weak rock layer at the anchoring edge of the roof bolt. Limited by the roadway height, even if the bolt length is increased to 2.5 m, the influence of the weak interlayer on the anchoring end cannot be eliminated. The main anchoring structure of the roof can pass through the weak interlayer if the number of 7.2 m anchor cables is increased, which greatly improves the anchoring quality. However, this will further aggravate the imbalance between mining and excavation, and the massive use of anchor cables under such simple geological conditions is also wasteful. A method to reduce the weakening effect of the roof weak interlayer on the anchoring end of the bolt and reduce the amount of anchor cables by revising the supporting process is therefore essential for safe and efficient mining at the Yaoqiao coal mine.

4.3. Support Scheme Improvement and Result Evaluation. In order to eliminate the adverse effect of weak interlayer on the anchor end of the bolt, flexible long strand bolts in Section 3 were used to support the roof. Moreover, the cable bolts and the wooden pillars were eliminated. The roof was supported solely by strand bolts whose diameter was 17.8 mm and length was 4000 mm. As shown in Figure 13, four flexible strand bolts are arranged in a row on the roof. The row spacing is 1100 mm and the pretightening torque is not less than 300 N m.

The field observation results of the new support scheme are shown in Figure 14. It can be seen from Figure 14(a) that the initial surface displacement of the roadway increased rapidly. Within the distance of 35 m to the excavation face, the roof deformation and rib convergence increases by 76.5% and 77.3%, respectively. The final sagging of the roof is 17 mm and the rib convergence is 22 mm.

Figure 14(b) shows the results of axial load of flexible long bolts (F1 and F2). The initial pretensions of the flexible long bolts F1 and F2 were 79.5 kN and 74 kN, respectively, and they finally reached 109 kN and 102 kN, respectively. Because a large initial pretension force is applied to the flexible long bolts, the axial load increases rapidly. Within the distance of 42 m to the excavation face, the axial load of flexible long bolts increased by 88.1% and 85.7% and then slowly increased and basically stabilized at 100 m to the excavation face.

It can be seen from Figure 14(c) that there exists no rock separation in the anchoring area, and a serious damage of the

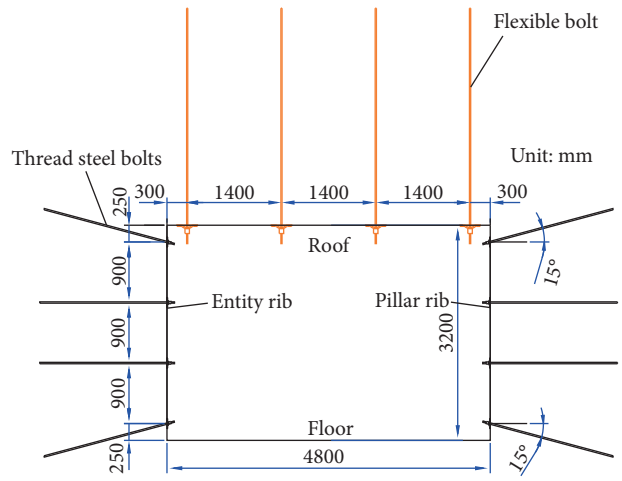


Figure 13: The supporting design flexible bolt.

surrounding rock was observed at a depth of 0.16 m in the roof. Within the depth of 3.1 m in the roof, there exist four annular fractures and two minor damages, and no fractures were found beyond the depth of 3.1 m. All fissures are limited to the anchorage area, and the flexible long bolt with increased length has a better control effect on weak interlayer in the roof, which can also be verified from Figure 14(d).

According to the statistics in the face, the construction time of the new support changed to 17 minutes, and it was reduced by 58.5% compared to the original scheme. The average daily excavation advance of 150-meter experimental roadway was 19.8 m, so the theoretical monthly excavation length would exceed 500 m, which greatly eases the imbalance between mining and excavation of mine.

4.4. Function Mechanism and Engineering Insights. In this case, owing to the weak interlayer above the anchoring zone of the bolt, the bolt pretension force does not produce effective compressive stress between the coal seam and sandy mudstone. As shown in Figure 12(a), when the stress environment inside the rock mass changes, the weak rock mass is destroyed first, which leads to substantial damage around the interface between the weak rock layer and coal layer. At this time, the strength of the anchoring structure is completely dependent on the shear strength of the anchored rock mass, but it cannot resist the weight of the entire anchoring structure and will inevitably lead to serious damage and separation in the anchoring area [29]. The construction of an anchor cable does not eliminate the weakening effect of the weak rock in the bolt anchorage area, so the weak rock layer position must be controlled within by increasing the bolt length, as shown in Figure 15.

By applying a large initial pretension force, the rock inside the anchorage area tends to limit the equilibrium state under three-way stress, forming a higher-strength anchoring structure. The cohesive force and internal friction angle of the weak rock layers increase under the clamping of the upper and lower rock layers. The expansion of cracks within the rock mass is suppressed, the strength of the weak rock

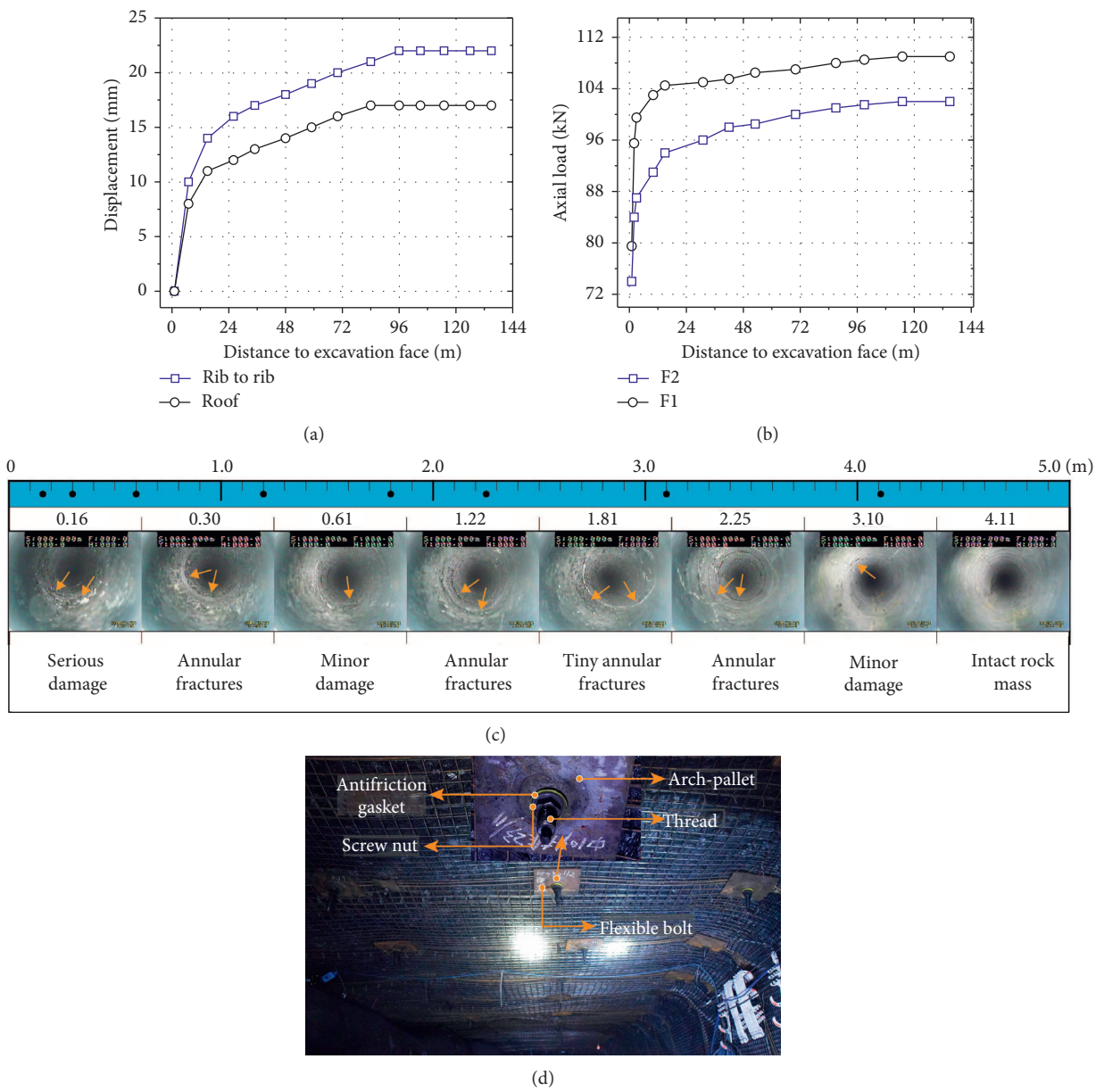


Figure 14: The field observation results: (a) the monitoring results of surface displacement in the roadway; (b) the monitoring results of axial load of flexible bolts; (c) borehole camera images and corresponding analysis; (d) field photo of test area.

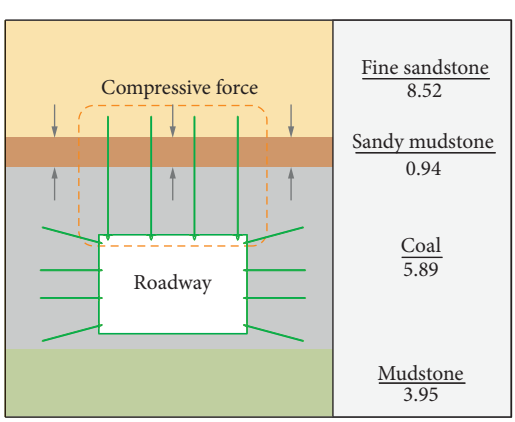


Figure 15: Action mechanism of flexible bolt.

layer is enhanced, and the stability of the anchor structure is effectively improved. On the premise of ensuring the overall strength of the rock mass, even if the anchor cable is cancelled, the number of bolts is reduced and the displacement of the anchor rod is enlarged from 900 to 1100 mm. The roadway roof deformation was only 17 mm and the horizontal convergence was 22 mm. To a large extent, the roadway safety is ensured and the speed of heading drivage increases.

5. Discussion

5.1. Analysis of Support Failure in Coal Roadway. The anchoring structure and in situ stress of the roadway roof are critical components of the evolution of surrounding rock

cracks after roadway excavation, which directly determine the early deformation and later stability control of the roadway [30, 31]. Coal measure strata in Chinese coal mines are complex and variable, which limits the applicability of supporting structures of bolts and cables under specific geological conditions, and support failure is common [2, 20]. In the Jianxin coal mine, the multiple adverse effects of the high-stress environment, weak interlayers, and low pretension force result in significant separation of the roadway roof rock mass. Even if the threaded steel rock bolt length is extended, the generation of delamination cracks in and outside the anchorage area cannot be effectively controlled. The key to roadway deformation control lies in the strength and size of the anchor structure in the rock mass. In the Wenjiapo coal mine, the depth and extra-thick coal seam is the key influencing factor. Conventional length limited steel rock bolts cannot pass through the rock fracture zone, so the bolts are anchored in the broken rock. The key control technology is therefore to increase the bolt length through the fracture zone while applying a high pretension force to build a stable and thick bearing structure. In the Yaoqiao coal mine, the geological conditions are relatively simple. The weak interlayer outside the end of the anchorage zone is the main problem to ensure roadway stability, so the key control technique is to pass through the weak interlayer. Increasing the pretension force enhances the strength of the weak interlayer, which reduces shear failure within and between layers. Under safe and reliable support, the tunnel support structure can be simplified and the driving speed can be effectively improved.

5.2. Roadway Support Form and Increased Bolt Length. Restrained by the size constraints of traditional thread steel bolts, previous studies have developed an alternative to anchor cables. In most cases, support is therefore carried out in the form of a combination of bolt, cable, mesh, and steel belt [32]. The high pretension and sufficient length make the anchor cable appear reliable in improving the roof support strength, which makes most of the roadway support parameters more fixed. This also presents a reasonable justification for regional reinforcement of a largely deformed roadway. However, according to the three case studies presented here, a support structure that relatively fixes a short bolt and long anchor cable cannot be adapted to a deformation control problem under specific conditions and the hidden safety risks remain severe.

We suggest that the safety control of the roof does not depend on the form and density of the support material but on the size and reliability of the deformation resistance of the anchored rock mass structure. After roadway excavation, cracks, delamination, sliding, and crack propagation appear in the rock mass in a short period of time with notably brittle characteristics [33, 34], and ultimately show characteristics of the circle layer shown in Figure 7. Larger macrofailures occur in shallow parts and deep rock masses are in a microfracture state under three-dimensional stress.

As shown in Figure 16, when the anchoring area of the bolt cannot pass through the fracture area, it exacerbates the fissure of the rock mass at the end of the bolt. This creates substantial cracks both inside and outside of the anchorage area (Figures 2(d), 3(c), and 12(c)). This is particularly prominent in the problems of extra-thick coal seams and weak interlayer roofs, and is also a significant shortcoming of threaded steel anchors. On the one hand, increased bolt length leads to an increased range of the internal anchorage area and thickness of the artificially reinforced arch. On the other hand, an elongated bolt can penetrate the fracture zone (or weak interlayer) and anchor into the rock mass with smaller crack openings, is less influenced by engineering disturbances, and has higher integrity. The damage and fissure area of the anchoring end is gradually moved upward, reduced, or even eliminated, and the risk of delamination outside the anchoring area is also removed. The anchoring structure is more stable, so it also has the ability to resist stronger engineering disturbances.

At the same time, for the support of rock burst roadway, it usually requires high support resistance and certain pressure release performance [35]. The support strength of the flexible bolt has been demonstrated above. The flexible bolt increases the length of the structure, which is equivalent to the length of the free section of the bolt, which improves the pressure release capacity of the bolt (because the elongation of the material is fixed). As the material used is steel strand, the shear resistance of the rod body has been improved, and the anchor rod is not easy to be cut (most of the rockbolts around a roadway are cut by rock mass), which shows that the support design concept of a flexible long bolt is also applicable to the support of the impact roadway.

5.3. Effectiveness of Pretension Force. The early high pretension force makes bolts to play a role quickly to control interlayer movement and rock separation as early as possible to improve the integrity of the surrounding rock. Does higher pretension force improve the control of surrounding rock deformation? The mainstream view [36, 37] believes that the working load of prestressed bolts is mainly composed of an initial pretension force and axial load generated by the deformation of the surrounding rock. When the working load is constant, a greater initial pretension force is associated with a smaller deformation load of the surrounding rock. Based on the insights obtained here, we suggest that the physical properties (e.g., material, diameter) of the bolt and crushing and expansion characteristics of the surrounding rock should also be considered. Owing to the complexity, and the long-term nature of the engineering environment of the bolt, the interaction between ultra-high prestress and engineering environment (especially groundwater) should be avoided, which can cause significant stress corrosion during the bolt lifetime to ensure long-term mechanical properties. The rock mass also has crushing and expansion characteristics so that an increase of the pretension force reduces the controlling effect of the rock mass to a certain extent. When exceeding a certain range, the effect of the pretension force will not be as apparent as the initial deformation control (generally, the preload of the anchor and anchor cable does not exceed 50% of its breaking

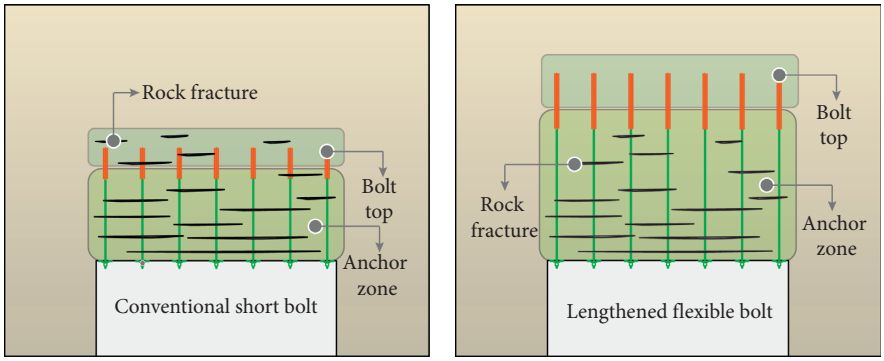


Figure 16: Stability characteristics of anchor structure: (a) conventional short bolt; (b) lengthened flexible bolt.

load, and is not less than 20% of the break load). In general, a reduction of the same deformation may require a larger amount of force, which poses few economic considerations. In fact, only a proper amount of pretension is required to quickly limit the initial deformation and their speed after the drivage, which is highly beneficial for later maintenance. The deformation amount during the whole roadway process can be controlled to a very small value, as shown in Figures 10(b) and 14(a).

The pretension force will attenuate over an extended service period owing to the long-term stress and deformation characteristics of the rock mass and supporting members. A certain degree of attenuation of the pretension force reduces the strength of the artificially reinforced arch structure and increases the amount of deformation in the roadway. This is also one of the reasons for the repeated construction of anchor cables. A plan should therefore be made to detect pretension of the bolt and cable. When the reduction of the pretension exceeding a certain range is detected, it will be necessary to perform secondary twisting of the bolt or secondary tension of the cable to ensure the effectiveness of the pretension over a long service period.

6. Conclusion

(1) The main reason for roadway support failure in the Jianxin coal mine is the discrepancy between the strength and size of the high-stressed rock mass and anchoring structure. The key to control the over-stressed rocks is the thickness of the anchoring area and application and maintenance of high pretension force. In the Wenjiapo coal mine, the control principle of a roadway in an extra-thick coal seam in a high-stress field environment is to improve the reliability of the anchoring structure of the bolt. The flexible bolt anchorage across the fracture zone to the small deformed rock mass can significantly improve roadway stability. In the Yaoqiao coal mine, the weak rocks outside the anchored section of the bolt is the main reason for its failure. The control method is to increase the bolt length, control the weak rock layer in the anchoring area, and increase the pretension to increase the shear resistance of the weak rock strength.

(2) The shallow artificial pressure arch structure formed by the anchor rod in the rock mass is very important for the stability of the support system. By adopting flexible bolts, the length of the bolt is not restricted by roadway height. On the one hand, the range of the compressive stress area and thickness of the pressure arch structure in the rock mass increase so that rock deformation over a larger range can be controlled by a flexible bolt. On the other hand, a flexible bolt can penetrate the fracture area and anchor into the deeper, small deformed rock mass so that the fractured area of the anchoring end damage gradually moves up, decreases, or even disappears. The risk of delamination of the anchored rock mass is eliminated and the stability is improved, which also demonstrates a stronger resistance to engineering disturbances. And, the flexible anchor can also be adapted to roadway support under rock explosion conditions.

(3) The early pretension force after excavation can provide early axial stress to the surrounding rock and make the bolt quickly play a role so that it can control interlayer displacement and delamination as early as possible. The setting of the pretension force must consider the characteristics of the bolt and crushing and expansion characteristics of the surrounding rock. Blindly increasing the pretension force will increase stress corrosion during the bolt service period and increase costs. Maintenance of the pretension force of the bolts and cables of coal mine roadways should be added to the engineering monitoring plan to ensure the effectiveness of the pretension force for a longer service period.

(4) By constructing a safe and reliable thick anchoring structure in the rock mass and using single flexible bolts to simplify the roof support, the support density of the roadway and time required for support from the source can be effectively reduced. The roadway driving speed can therefore be substantially

improved by ensuring the safety of the roof, which can effectively alleviate the abnormal and outstanding problem of imbalance between mining and excavation under complex geological and low mechanization conditions.

Data Availability

The data used to support the findings of the study are included in the article.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

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