Research Article

Failure Characteristics and Deformation Control Methods of the Bottom Drum of Roadways during Repeated Mining of Multiple Coal Seams

Changzheng Zhao, Shenggen Cao, Shuyu Du, Chiyuan Che, and Xingyao Wang

State Key Laboratory of Coal Resources & Safe Mining, School of Mines, China University of Mining & Technology, Xuzhou 221116, China

Correspondence should be addressed to Shenggen Cao; caoshenggen@126.com

Received 24 February 2022; Accepted 19 April 2022; Published 23 May 2022

1. Introduction

As the coal mining expands, the coal seams with complex conditions and high instability have increased proportion. Therefore, high ground stress, faults, weak surrounding rock, and mining influence have exacerbated the deformation and instability of roadways, increasing the difficulty of supporting and retaining the rock mass surrounding the roadways [1–4]. Thin and multiple coal seams, a large dip angle, and high instability contribute to the failure of the supporting structure, posing threats to mine safety [5–8]. Thus, the stabilization of the rock mass surrounding roadways during mining influence has become a hot topic of domestic and international scholars.

In terms of the stabilization of the rock mass surrounding roadways during mining influence, the experts have researched and achieved many results. For example, Liu et al. [9, 10] analyzed the stress evolution of surrounding rock of roadway during repeated mining. Besides, the asymmetric destruction mechanism was evaluated, and staged reinforcement and support methods were proposed. Jia et al. [11] et al. considered the butterfly plastic zone theory and analyzed the temporal and spatial distribution of the roadway’s confining pressure, ratio, and direction during mining to determine the key positions of the roadway’s deformation and failure. Xu et al. [12, 13] investigated the surrounding rock deformation and the roadway maintenance under strong dynamic pressure. The author proposed an anchor mesh cable+grouting reinforcement support system based on a bearing arch to control the surrounding rock damage. Xu et al. [14] performed theoretical derivation and numerical simulation to determine the causes of instability and the fracture of narrow coal pillar. The authors proposed a support scheme to stabilize the rock surrounding the roadway. Rong et al. [15] proposed a roadway support system for a high-stress area based on the Wudong coal mine, investigated the causes of roadway roof collapse, and developed an optimization method for the bolt support using an indoor test. Wang et al. [16] analyzed the influencing factors based on the roadway differential deformation on the inclined Pansan mine working face, and a reasonable support...
A scheme was proposed. Aiming at the problem of stress concentration in roadway due to the unreasonable design of coal pillar, Wu et al. [17] analyzed coal pillar stress and proposed corresponding roadway support measures of close distance coal seam. Li et al. [18] determined the influencing factors of roadway deformation after repeated mining of a coal seam using on-site monitoring and multiple linear regression analysis. These studies analyzed the failure mechanism of the rock surrounding roadways during mining and proposed various control measures. Mo et al. [19] staked the Glencore Bulga Underground Operations as the background, investigated and analyzed the main reasons for the floor heave in the roadway, and proposed the increase in mining height for the predicted floor heave domains based on the stress notch angles to control the floor heave. Mo et al. [20] analyzed the failure mechanism of the coal seam floor, and proposed a new floor classification system by quantifying the two main factors affecting the floor heave of the roadway, which helps to evaluate the possibility of the floor failure of the roadway.

In this study, we investigate the +980 m track roadway of the Faer coal mine passing through multiple layers of soft strata. Due to repeated mining of multiple coal seams, the roadway deformation and failure are characterized by non-uniformity and asymmetry. We analyze the deformation characteristics and mechanisms of the soft track roadway affected by repeated mining of multiple coal seams. We propose a combined support scheme consisting of bolts, anchor cables, grouting, and pressure relief grooves.

2. Engineering Background

2.1. Mining Conditions. The 1#, 3#, 5-2#, 5-3#, and 7# coal seams of Faer coal mine, located in Guizhou Province, are the primary coal seams. The Faer coal mine is characterized by coal seams with close proximity; the layer spacing between 1#, 3#, 5-2#, 5-3#, and 7# coal seams is 11.68 m, 10.33 m, 18.81 m, and 23.08 m, respectively, of which 7# coal is located under the +980 m track roadway; and the rest of the coal seams are located above the roadway. The buried depth of the +980 m track roadway is about 230 m, passing through multiple layers of soft coal and rock mass; roadway is close to multiple working faces, of which the 15021, 35203, and 31004 working faces are 50.9 m, 130 m, and 170 m, respectively, away from roadway, as shown in Figure 1. The serious bottom drum of roadway occurs between the 5-3# coal floor and the 7# coal roof, which is mainly made up of soft rock. Due to the complex geological conditions and the repeated mining of multiple coal seams, the strength of the surrounding rock is low, and the bottom drum of the roadway is serious.

A section of the +980 m track roadway is shown in Figure 2. The straight wall has a 4800 mm width and 4300 mm height semicircular arch. The original permanent support consists of anchor bolts and metal mesh; the rebar bolts have \( \Phi 20 \times 2000 \) mm dimensions and \( 800 \times 800 \) mm row spacing. The deformation and damage were extensive after roadway was constructed. Although the mining company repaired and reinforced roadway and added anchor cables and shotcrete, the anchor cables have \( \Phi 15 \times 3500 \) mm dimensions and \( 800 \times 800 \) mm row spacing, the shotcrete thickness is 100 mm, and the roadway continued to deform. Some anchor mesh cables failed, and the bottom drum of roadway was extensive, adversely affecting the coal mine operation.

2.2. The Reasons Analysis of Roadway Deformation. By drilling cores at different positions of roadway, the lithology of roadway is analyzed by XRD (X-Ray Diffraction) rock component test, using CHK7.2 (B) mine borehole imager to detect and analyze the internal failure and deformation of roadway, and surveying the failure characteristics and existing support measures, the monitoring results are shown in Figure 3. The primary reasons were the following:

1. The poor lithology of rock surrounding roadway. The +980 m track roadway is a crosslayer roadway. The drilling cores at different locations of roadway indicate that roadway lithology is mainly composed of soft rocks, which contains a large amount of quartz and...
chlorite, accompanied by a small amount of dolomite. Feldspar and chlorite are clay minerals with low hardness, plasticity, and volume expansion when exposed to water, and dolomite has relatively low strength, and the rock composed of the above parts has low strength. The clay mineral composition is high; thus, the soil rapidly expands when wet and has a low bearing capacity, resulting in roadway deformation.

2. The repeated mining of multiple coal seams. The Faer coal mine has closely spaced vertical mining coal seams. As the mining depth increases, stress redistribution occurs in upper coal seam goaf and leads to stress concentration at weak positions of roadway, decreasing roadway stability.

3. The developed fissures in surrounding rock of roadway. Through the on-site TV detection results, it is found that the internal fissures of the surrounding rock have a high degree of development and a wide distribution range. The surrounding rock is seriously deformed and damaged, and roadway damage is obviously asymmetric. The plastic zone has reached more than 4.2 m, which belongs to the large loose circle. Due to the high degree of development of fracture in roadway, the supporting bolts and cables are both in the plastic zone. Thus, the surrounding rock deformation cannot be controlled.

4. The low support strength. The existing support structure did not adequately consider the site conditions. Thus, the support structure could not prevent the strong deformation of roadway.

3. Composite Fracture Mechanism of the Overburden Caving in Multi-Coal Seam Mining

The roof strata bend, break, and collapse to goaf due to the secondary stress field with the mining, which is redistributed [21–23]. The overburden adjusts to resist uneven deformation and creates pressure arch to bear the load. The stress around the mining area is concentrated around this arched load-bearing structure, and transmission path is deflected through this arched load-bearing structure, thereby forming an arch stress distribution zone, which includes into stress-increased and stress-reduced zone around the mining area, corresponding to the arch bearing structure and the fractured loose rock. The essence of this arched load-bearing structure is “pressure arch.” The stress around the stope is concentrated around the arch, and the stress is deflected through the arch to transfer the weight of the rock stratum from the arch to coal pillar. According to the degree of stress on coal pillar, structure formed by overburden caving in the stope is divided into two types. One is the surrounding rock bearing the overlying strata weight, i.e., the GPA (Goaf Pressure Arch), and the other is, in the stress reduction area, the hinged structure bearing the strata weight, which is the CPPA (Coal Pillar Pressure Arch) under GPA [24, 25], as shown in Figure 4.

With roof strata in goaf break and fall to goaf, the surrounding rock stress decreases, and the stress zone in goaf redistributes to achieve a new balance. The rock strata in different layers experience different degrees of fracture-balance-fracture process as the mining process occurs closer to the coal face, and the number of coal seams increases. The coal seams that have been mined are affected by multiple mining, and the different layers of rock experience different degrees of fracture, the bearing structure changes continuously, and the fissures gradually expand upwards to the surface. The overburden and stress field distribution of the multiple coal seams change dynamically because the length and structure of the broken rock blocks change during the mining process. The reason for this dynamic change is that the overlying rock of each coal seam working face is broken several times, which makes the length and structure of the broken rock gradually shorten and disordered, which reduces the possibility of forming the structure [26].
4. Numerical Simulation of Overburden Caving and Roadway Deformation in Multiseam Mining

We used the UDEC 3.0 numerical simulation software to analyze the overburden caving and roadway deformation in the Guizhou Faer coal mine. We investigated the stress evolution of the rock near roadway resulting from the overburden caving. The model was divided into 19 layers. We used the Mohr-Coulomb criterion, and physical and mechanical parameters of rock mass of each formation are shown in Table 1. Each coal seam had an inclination of 14°, the model size is 720 m length and 280 m height, and the model was divided into 8582 blocks. Fixed constraints were applied at the bottom, and horizontal displacement constraints were set at the left, right, front, and back. A 50 m coal pillar was placed at this model boundary to eliminate the boundary influence; the numerical simulation model is shown in Figure 5. The simulated excavation conditions were the same as the actual ones. The working face length on both sides of the roadway was about 400 m and 100 m, respectively, and the mining methods were inclined longwall mining and strike longwall mining, respectively.

Vertically downward mining occurred in the mining process. Besides, excavating the entire length and step-by-step excavation were used on coal seams, respectively. The 400 m long working face was excavated in 8 steps.

4.1. The Caving Structure and Migration Characteristics of Overburden in Multi-Coal Seam Mining

4.1.1. Strike Longwall Mining Working Face. The structure and migration characteristics of overburden in strike longwall mining working faces are shown in Figure 6. After mining 1# coal, immediate roof of 1# coal completely collapsed. The overlying rock of central area gradually collapsed and compacted goaf, and the rock stratum in the inclined lower area failed to collapse completely due to the coal seam inclination. Goaf floor did not bear pressure during mining. Due to vertical and horizontal stresses on goaf, goaf floor migrated towards the overhead direction of goaf, resulting in numerous buckling failures. Similar to single coal seam mining, the unbroken strata bore the weight of the overlying strata by forming GPA.

Because the distance between 1# coal and 3# coal is close, the roof bent and dropped after the working face had been mined. CPPA formed by the hinged rock formations fractured and rotated. The roof separation intensified; the overburden fissures expanded upward.

Affected by mining of upper 1# and 3# coal, the roof is basically broken rock blocks after mining 5-2# coal, which are connected with the collapsed roof structure, forming a broken roof-group structure. The hinged structure formed by mining 1# and 3# coal is further broken, deformed, and...
rotated, and the overlying rock formed by the mining of coal 1 and 3 is further broken. The load-bearing structure is gradually moved up.

Since the distance between the 5-2# and 5-3# coal seams was large, the roof integrity of 5-3# coal seam was high. After mining the working face, the roof strata cave entirely, forming a hinge structure at the coal wall, which bear the weight of the roof and the overlying broken strata. Due to the fracture of the main roof, the overburden bearing structure weakened, and an articulated structure formed at the coal wall of each overlying coal seam. This structure rotated, significantly reducing the bearing capacity. The 7# coal seam was located below the roadway floor. During mining, stable structure of overlying coal seams became unstable, fractured, and compacted due to repeated mining, resulting in loose rock. Due to the structure of GPA damaging, the stress concentration of the arch structure on coal pillar decreased.

From the analysis, before the overburden failed at the surface, the scope of GPA expanded with the coals mining. When the fissures reached the surface, GPA was damaged, and the stress concentration of arch structure decreased. The caving rock moved to the coal pillar, and its weight was borne by CPPA and transmitted to coal pillar.

4.1.2. Inclined Longwall Mining Working Face. The structure and migration characteristics of overburden in inclined longwall mining working faces are shown in Figure 7. During 1# coal mining, the overlying strata began to deform and become unstable and showed a gradually increasing arch structure. After 42 m of 1# coal excavation, the immediate roof is completely broken, the roof fissures and upper coal seam floor are completely connected. During the mining process, the range of GPA continued to expand, and the development height of the fissures gradually moved upward. During the mining

| Number | Lithology     | \( \rho \) (kg/m\(^3\)) | \( E \) (GPa) | \( \sigma_b \) (MPa) | \( c \) (MPa) | \( \mu \) | \( \varphi \) (°) |
|--------|---------------|-------------------|-------------|-----------------|--------------|-------|---------|
| 1      | Siltstone     | 2500              | 17.0        | 8.9             | 6.0          | 0.22  | 39.5    |
| 2      | Silty mudstone| 2400              | 8.5         | 3.6             | 3.0          | 0.25  | 36      |
| 3      | 1# coal       | 1400              | 5.0         | 1.5             | 1.5          | 0.28  | 28.5    |
| 4      | Fine-sandstone| 2550              | 10.0        | 11.4            | 6.5          | 0.18  | 37.5    |
| 5      | 3# coal       | 1400              | 5.0         | 1.5             | 1.5          | 0.28  | 28.5    |
| 6      | Silty mudstone| 2400              | 8.5         | 3.6             | 3.0          | 0.25  | 36      |
| 7      | 5-2# coal     | 1400              | 5.0         | 1.5             | 1.5          | 0.28  | 28.5    |
| 8      | Mudstone      | 2100              | 6.5         | 3.0             | 2.1          | 0.27  | 30.0    |
| 9      | Silty mudstone| 2400              | 8.5         | 3.6             | 3.0          | 0.25  | 36      |
| 10     | 5-3# coal     | 1400              | 5.0         | 1.5             | 1.5          | 0.28  | 28.5    |
| 11     | Mudstone      | 2100              | 6.5         | 3.0             | 2.1          | 0.27  | 30.0    |
| 12     | Fine-sandstone| 2550              | 10.0        | 11.4            | 6.5          | 0.18  | 37.5    |
| 13     | 7# coal       | 1400              | 5.0         | 1.5             | 1.5          | 0.28  | 28.5    |
| 14     | Fine-sandstone| 2550              | 10.0        | 11.4            | 6.5          | 0.18  | 37.5    |
| 15     | Mudstone      | 2100              | 6.5         | 3.0             | 2.1          | 0.27  | 30.0    |
process of 5-2#, 5-3#, and 7# coal, the bearing structure of the overlying strata was gradually destroyed until the surface, GPA was damaged in a large range, and the stress concentration of arch structure formed in coal pillar was further weakened.

Through the analysis of the above results, the stress concentration mainly comes from GPA and CPPA. Before the overburden damage develops to the surface, the range of GPA gradually expands with mining depth. As the fissures develop to the surface, the stress concentration gradually decreases, which is transmitted to coal pillar in the form of pressure arch.

4.2. Stress Evolution Characteristics of Roadway Floor in Multiseam Mining. The evolution of roadway floor stress was measured on roadway floor during mining. The vertical stress evolution of roadway floor was analyzed. The results are shown in Figure 8.

After roadway is excavated, plastic failure occurs within a certain range of the floor, and the stress is significantly reduced. When 1# coal advanced to 192 m, the stress of roadway floor stabilized around 6.13 MPa, indicating that the mining had a negligible influence on roadway. After completing mining of 1# coal, the stress of roadway floor increased to 7.75 MPa, representing an increase of 26.4%. In the single coal seam mining process, the goaf expands with coal seams mining, and caving rock forms a stress arch above goaf to carry overlying rock layer, and it is transmitted to roadway through the coal pillar, causing the stress balance of roadway to be broken.

As 3# coal advanced to 231 m, the stress of roadway floor decreased initially and then increased. After completing of 3# coal, the stress increased from 9.46 MPa to 10.5 MPa, an increase of 11.0%. The overburden caving structure and migration characteristics of multi-coal seam mining are more complex. Due to the close distance between 1# coal
Figure 8: Continued.
Figure 8: Continued.
and 3# coal, 3# coal roof has been damaged during mining, which relieves pressure partly. With the advancement of 3# coal, the roof continued to break, and overlying fissures continued to develop upward. The range of the GPA continues to expand and gradually moves to roadway, which eventually leads to a large stress concentration on roadway floor.

The trend of roadway floor stress for the 5-2# and 5-3# coals was similar to that of 3# coal. After completing mining of two coal seams, the maximum stress of roadway floor was 11.5 MPa and 12.2 MPa, representing increases of 6.4% and 4.2%, respectively. As the advancing distance and mining depth increase, the stable structure of the overlying rock is damaged again, and GPA moves continuously around roadway. The greater the advancing distance and the mining depth, the higher the vertical stress concentration of roadway floor. Meanwhile, the fractured rock forms CPPA above the coal pillar and transfers the weight of the overlying fractured rock to the surrounding roadway, resulting in a relatively small decrease in the stress increase around roadway.

These results indicate that the stress of roadway floor increased; the overburden bearing structure, i.e., the GPA, moved upward with mining depth. The overburden of each coal seam goaf was relatively loose after being broken many times, and the rate of increase of the stress of roadway floor decreased.

4.3. Failure Characteristics of Roadway Floor in Multiseam Mining. The structure of goaf roof rock constantly changed during mining process, resulting in a change in the lateral support stress around roadway and the coal stratum of the coal pillar floor. Roadway repeatedly experienced stages of failure, stability, and refailure during each coal seam excavation. By arranging measuring points around the roadway, the deformation characteristics of the roadway is monitored.

In Figure 9, as each coal seam was mined, roadway deformation increased, and the bottom drum of the roadway was particularly affected after the 7# coal was excavated. As shown in Figure 10, after 1# coal was excavated, the rock mass of the roadway floor began to deform, local bending deformation occurred, and the maximum bottom drum was only 0.05 m. During the 3# coal excavation, the bottom drum of roadway suddenly increased to 0.2 m, and the whole roadway was substantially deformed. During the subsidence of both sides and roof of the roadway, the floor rock stratum was significantly damaged, resulting in high instability. During the 5-2# coal excavation, the bottom drum increased to 0.24 m, and the floor rock layer flexed upward, resulting in separation phenomenon. During the 5-3# coal excavation, the fissures around the roadway were compacted and roadway section was reduced due to the interaction of horizontal
and vertical stress around the roadway. After the 7# coal was excavated, the deformation of roadway floor increases rapidly, and the maximum bottom drum increased to 0.28 m. These results show that roadway floor exhibited upward flexural deformation. The rock broke when rock of roadway floor reaches the yield strength. During the following repeated mining, the broken rock blocks in roadway floor were moved in the free direction of roadway due to the high stress on both sides, and the bottom drum increased significantly. Corresponding to the mining of 1# coal, the roadway vertical stress increases greatly, but after mining 1# coal, the roadway vertical stress is insufficient to cause a large deformation of roadway floor. As each coal seam is mined subsequently, the vertical stress gradually increases. The 3#, 5-2#, and 7# coal mining influences the stability of roadway floor, and the 1# coal and 5-3# coal mining has a small effect on roadway floor stability.

5. Field Test and Results

5.1. Repair and Reinforcement of Roadway. Through the above analysis, we chose a combination of bolts, anchor cables, grouting, and pressure relief grooves. Roadway support scheme is shown in Figure 11.

5.1.1. Support Parameters. The anchor rod had a dimension of $\Phi20\text{mm} \times L\text{3500 mm}$. A resin capsule (CK2570) was selected as the anchoring agent. Its length was a minimum
of 1000 mm, and the pretension force of the bolts was 300 N·m. The anchor cable dimension was $\Phi 25 \text{ mm} \times L 8500 \text{ mm}$, with a row spacing of 3200 mm $\times 1600 \text{ mm}$. Similarly, a resin capsule was used as the anchoring agent. Its length was a minimum of 2000 mm, and the pretension force of the anchor cable was 150 KN. After the anchor bolt (cable) support was installed, the initial shotcrete thickness was 90 mm.

5.1.2. Grouting Parameters. The grouting material was ordinary cement slurry, and the water-cement ratio was 0.7 : 1. The grouting pressure was 2 MPa. It was adjusted according to the roof condition. The grouting speed for a single hole was 4 m$^3$/h, and the maximum grouting volume was 2.13 t. The grouting sequence was from the roadway floor to roadway roof. The spacing between the grouting holes is 1600 mm, and two of those near the bottom plate formed an angle of 20° with the vertical direction of the bottom plate, and they were arranged obliquely on both sides between the anchor bolts (cables). After the grouting pipe was installed, concrete was poured with a thickness of 110 mm to prevent slurry leakage.

5.1.3. Parameters of the Pressure Relief Grooves. Pressure relief grooves were excavated at the both sides of roadway. They were 2000 mm deep and 800 mm wide. The grooves were filled with 1800 mm gangue, and 200 mm of concrete was poured for sealing. The construction sequence was as follows: undercover $\rightarrow$ excavation of pressure relief grooves $\rightarrow$ filling of gangue $\rightarrow$ pouring of drainage ditch $\rightarrow$ bottom angle anchor bolt $\rightarrow$ grouting.

5.2. The Performance of Roadway Repair Scheme

5.2.1. Deformation of Roadway. Surface displacement of roadway was monitored for 55 d after the repair and reinforcement. The monitoring data included bottom drum, roof, floor, and both sides of roadway. The displacement and deformation of the roadway during monitoring period are shown in Figure 12. The results indicated that the deformation of roadway in the unrepaired and reinforced section continued to increase, and there was no stable trend during the 55-day monitoring period. Nevertheless, the growth of surrounding rock deformation tends to be stable after roadway repair and reinforcement. Within 30 d of roadway deformation monitoring, roadway roof and two sides exhibited substantial deformation. Roadway roof showed high subsidence, and roadway floor deformation increased slightly. Roadway surrounding rock deformation stabilized after 50 d. The relative displacement was 0.026 m on both sides of the roadway, 0.082 m for the roof and floor, and 0.056 m for the bottom drum, indicating that the surrounding rock deformation was within a reasonable range.

5.2.2. Grouting Performance. After the grouting reinforcement of the +980 m track roadway was installed, the internal fissures and the integrity were monitored using borehole video data. The results are shown in Figure 13. The grouting performance was good, and the rock mass fissures of the roadway had been filled with slurry using external pressure. This technique cemented the fractured coal and rock mass into a single mass, enhancing the mechanical properties of rock. The observation results demonstrated the excellent performance of the grouting reinforcement of roadway to stabilize the surrounding rock.
Figure 12: Displacement curves of surrounding rock: (a) unrepaired reinforcement section; (b) repaired reinforcement section.
6. Conclusion

(1) The dominant factors causing serious bottom drum of roadway floor during the mining of multiple coal seams were determined by analyzing the surrounding rock composition and borehole video data. These factors included the soft and broken rock surrounding roadway, developed fissures in surrounding rock, repeated mining of multiple coal seams, and an inadequate existing support structure.

(2) The composite fracture mechanism of the overburden structure was theoretically analyzed. The caving structure and migration characteristics of the overburden were investigated using numerical simulation in UDEC 3.0. The results indicated that the caving rock weight affected the coal pillar, forming a stress arch. During the mining process, the increase in the stress arch range of the coal seams is the main reason contributing to the serious bottom drum of roadway. The stress evolution and failure characteristics of roadway floor indicated that roadway bottom drum continued to occur during mining each coal seam, although the rate of increase in roadway floor stress decreased.

(3) According to the characteristics of the overburden caving and movement and roadway floor failure during multi-coal seam mining, we proposed a combination of bolts, anchor cables, grouting, and pressure relief grooves for support. The field measurements showed that the surrounding rock deformation remained unchanged 15 d after roadway repair and reinforcement. The maximum deformation of roadway roof, floor, and both sides was only 26, 56, and 26 mm, respectively, 50 d after roadway repair and reinforcement. The fissures in the surrounding rock were filled with slurry, preventing the surrounding rock from further deformation.

Data Availability

The data used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

The authors declare have no conflict interest.

Acknowledgments

This study was supported by the National Natural Science Foundation of China (51874284) and Priority Academic Program Development of Jiangsu Higher Education Institutions.

References

[1] D. J. Liu, J. P. Zuo, J. Wang, T. L. Zhang, and H. Y. Liu, "Large deformation mechanism and concrete-filled steel tubular support control technology of soft rock roadway-a case study," *Engineering Failure Analysis*, vol. 116, article 104721, 19 pages, 2020.

[2] L. J. Zheng, Y. J. Zuo, Y. F. Hu, and W. Wu, "Deformation mechanism and support technology of deep and high-stress soft rock roadway," *Advances in Civil Engineering*, vol. 2021, Article ID 6634299, 14 pages, 2021.

[3] X. L. Guo, Y. Q. Zhu, Z. S. Tan, L. Li, A. Li, and Y. T. Yan, "Research on support method in soft rock tunnel considering the rheological characteristics of rock," *Arabian Journal of Geosciences*, vol. 14, no. 2703, pp. 1–14, 2021.

[4] Y. Xiong, D. Kong, Z. Cheng et al., "Instability control of roadway surrounding rock in close-distance coal seam groups under repeated mining," *Energies*, vol. 14, no. 16, pp. 5193–5219, 2021.

[5] H. Yan, J. Zhang, R. Feng et al., "Surrounding rock failure analysis of retreation roadways and the control technique for extra-thick coal seams under fully-mechanized top caving and intensive mining conditions: A case study," *Tunnelling and Underground Space Technology*, vol. 97, article 103241, 2020.
[6] H. Yan, M. Y. Weng, R. M. Feng, and W. K. Li, “Layout and support design of a coal roadway in ultra-close multiple-seams,” Journal of Central South University, vol. 22, no. 11, pp. 4385–4395, 2015.

[7] R. K. Pan, Z. H. Ma, M. G. Yu, and S. D. Wu, “Research on the deformation characteristics and support technology of a bottom gas extraction roadway under repeated interference,” Advances in Civil Engineering, vol. 2019, no. 4, Article ID 1413568, p. 14, 2019.

[8] I. B. Tulu, G. S. Esterhuizen, T. Klemetti, M. M. Murphy, J. Sumner, and M. Sloan, “A case study of multi-seam coal mine entry stability analysis with strength reduction method,” International Journal of Mining Science and Technology, vol. 26, no. 2, pp. 193–198, 2016.

[9] H. H. Liu, X. Y. Wu, Z. Hao, X. D. Zhao, and X. F. Guo, “Evolution law and stability control of plastic zones of retained entry of working face with double roadways layout,” Journal of Mining & Safety Engineering, vol. 34, no. 4, pp. 689–697, 2017.

[10] X. Y. Wu, H. H. Liu, J. W. Li, X. F. Guo, K. Lv, and J. Y. Wang, “Space-time evolutionary regularity of plastic zone and stability control in repetitive mining roadway,” Journal of China Coal Society, vol. 45, no. 10, pp. 3389–3400, 2020.

[11] H. X. Jia, G. S. Li, L. Y. Wang, and A. Z. Qiao, “Characteristics of stress-field environment and roof falling mechanism of mining influenced roadway,” Journal of Mining & Safety Engineering, vol. 34, no. 4, pp. 707–714, 2017.

[12] Y. L. Xu, M. T. Xu, and L. X. Cheng, “Control mechanism and experimental study on renewable bearing arch in soft and mudding roadway under dynamical pressure impact,” Journal of Mining & Safety Engineering, vol. 35, no. 6, pp. 1135–1141, 2018.

[13] Y. L. Xu and H. Zhang, “Research on reinforcement and treatment technology for soft rock roadway under dynamic pressure,” Coal Science and Technology, vol. 46, no. 1, pp. 68–73 +111, 2018.

[14] Q. Y. Xu, Q. G. Huang, and G. C. Zhang, “Fracture and instability mechanism and control technology of a narrow coal pillar in an entry in fully mechanized,” Journal of Mining & Safety Engineering, vol. 36, no. 4, pp. 941–948, 2019.

[15] H. Rong, L. T. Pan, X. Y. Li et al., “Analysis on influence factors of roadway instability in high-stress, steeply inclined extrathick coal seam,” Advances in Civil Engineering, vol. 2021, Article ID 4676685, 17 pages, 2021.

[16] P. Wang, N. Zhang, J. G. Kan, B. Wang, and X. L. Xu, “Stabilization of rock roadway under obliquely straddling working face,” Energies, vol. 14, no. 5, pp. 1–23, 2021.

[17] X. Y. Wu, S. Wang, C. Tian, C. X. Ji, and J. Y. Wang, “Failure mechanism and stability control of surrounding rock of docking roadway under multiple dynamic pressures in extrathick coal seam,” Geofluids, vol. 2020, no. 9, Article ID 8871925, 16 pages, 2020.

[18] Q. Li, D. Z. Kong, G. Y. Wu, Z. J. Wen, and Y. Q. Shang, “Influence factors’ analysis of the face-end roof leaks exposed to repeated mining based on multiple linear regression,” Advances in Materials Science and Engineering, vol. 2021, Article ID 5755055, 15 pages, 2021.

[19] S. Mo, K. Tutuk, and S. Saydam, “Management of floor heave at Bulga underground operations - a case study,” International Journal of Mining Science and Technology, vol. 29, no. 1, pp. 73–78, 2019.