INTRODUCTION

Energy and mineral resources play a key role in national economic development around the world. In China, coal accounts for approximately 70% of China’s primary energy production and consumption. In recent decades, with the depletion of shallow coal resources and advances in new technologies, deeper mining and excavation activities have been carried out.\(^1\) Compared with the shallow surrounding rock, the rock mass with a deep burial depth is subjected to high in situ stress and is vulnerable to strong mining disturbances.\(^2\) Therefore, the surrounding rock in deep rock engineering is often more fractured and shows creep and large long-term deformation characteristics.\(^3\) Because an increasing number of mines enter the deep mining stage, stability control of deep surrounding rock has become a significant challenge.

Research on the failure process and stability control technology in a deep roadway: Numerical simulation and field test

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Abstract

With the increase in mining depth and the deterioration of surrounding rock conditions, large deformation failure of deep roadway surrounding rock is still very common. To solve the support problem, it is necessary to understand the deformation and failure mechanism of the deep roadway. This paper presents a case study on the deformation failure behavior and support design of a deep roadway in the Tangyang mine by field tests and numerical simulations. The rock mass properties were first evaluated based on field data and the mechanical parameters of rock specimens. Then, the numerical model of the deep roadway surrounding rock was established based on the calibrated microparameters. The deformation features and cracking behavior of the roadway were investigated in detail. The results show that the cracking process of crack initiation, crack propagation, and rock separation in the surrounding rock of this kind of roadway is a gradual process developing from a shallow to deep depth. A combined support “bolt-cable-mesh-steel ladder” was proposed based on the failure characteristics of the roadway. Finally, a numerical simulation and field experiment were conducted to evaluate the rationality of the proposed support scheme, and the results show that the new support method can effectively control the surrounding rock. This study can provide valuable references for support design in deep underground engineering.

KEYWORDS

cracking evolution, numerical simulation, roadway deformation, support technology, underground excavation
that engineers must face more frequently. To maintain safe and steady mining activities, the stability of the surrounding rock must be ensured.

The instability mechanism, the deformation behaviors, and the support schemes have received more attention and have been extensively studied by various kinds of methods, such as theoretical analysis, field monitoring, physical model testing, and numerical simulation. Researchers proposed different constitutive models to theoretically predict the deformation of the surrounding rock. Various kinds of monitoring techniques, such as fiber Bragg grating-based monitoring systems, microseismic systems, extensometers, and stress meters, were developed to capture the activity of the roadway surrounding rock. These monitoring results provide the basis for further research of the failure mechanism and support design of the roadway. Geo-mechanical model tests are also an effective tool to investigate the failure mechanism of underground structures in a more intuitive way, and researchers have carried out a wide range of geo-mechanical model tests focusing on the surrounding rock mass stability problem of underground engineering. The experimental results are more intuitive and consistent than the results obtained from field experiments, but they consume a substantial amount of time and money. With the development of computer technology, numerical simulation plays an increasingly important role in investigating the mechanical behavior of deep rock engineering. Compared with the research methods mentioned above, numerical modeling can obtain more comprehensive information at a lower cost. Shen and Yang et al analyzed the failure mechanism of deep soft roadways based on Universal Discrete Element Code (UDEC) software. Chen et al presented a large deformation analysis of soft rock engineering at a great depth with 2D finite element software. Karampinos et al reproduced the buckling mechanism in hard rock mines with a three-dimensional distinct element code (3DEC) and investigated the influence of reinforcement in controlling squeezing failure. Based on engineering practice and the Fast Lagrangian Analysis of Continua (FLAC3D) simulation results, Li et al summarized the deformation failure laws in deep soft rock roadways.

The stability of the roadway surrounding rock depends on its mechanical properties and geological factors and on the rationality of the supporting scheme. Various support techniques were used to support the deep surrounding rock, including energy-absorbing bolts, concrete cribs, bolt-grouting, and concrete arches. For the control technology of deep roadways, traditional single bolt support can no longer achieve good supporting effects, and many engineering practices have proven that the combination of different support methods is an effective measure to control the stability of the roadway surrounding rock. To control the large deformation in deep soft rock roadways, Yang et al proposed a “bolt-cable-mesh-shotcrete + shell” combined support scheme. The “whole section anchor-grouting” reinforcement technology was used to support roadways with loose and fractured surrounding rock. A combined supporting system including foamed concrete and U-shaped steel is implemented in the practice of controlling the undergoing large deformation in deep roadways. The successful application of these technologies has produced a solid foundation for the effective control of roadway deformation.
Although there have been extensive studies on the stability problem in deep roadway engineering, deformation, failure, and support problems remain common for reasons related to performance, cost, and technology. To solve the support problems in the Tangyang coal mine, this paper presents field and numerical simulation research on the deformation failure behavior and support design of deep roadways. Based on the rock mass properties calibrated with the mechanical parameters of the rock specimens and the rock quality designation (RQD), a numerical model of the 2313 haulage roadway is established with the UDEC to investigate the deformation characteristics and cracking behavior of roadway surrounding rock. A combined support scheme is then proposed based on the failure characteristics of the unsupported roadway. Finally, field practices and simulations are conducted to evaluate the control effect.

2 | ENGINEERING BACKGROUND

2.1 | Engineering situation

The Tangyang coal mine is located in northern Jining city, Shandong Province, as shown in Figure 1. The thickness of the #3 coal seam is 4.8-5.8 m, the burial depth is approximately 658-685 m, and the coal seam dip angle is almost flat. The 2313 haulage roadway was developed along the #3 coal seam floor, and the rectangular roadway has a width of 3800 mm and a height of 3100 mm. The roof and floor of the #3 coal seam are mudstone and sandy mudstone, which are medium stable and relatively stable, respectively. The thickness of the top coal is approximately 3 m, and a detailed illustration of the 2313 haulage roadway is in Figure 2.

Rock bolting is used in the original support design of the roadway in the adjacent 2310 panel. However, severe deformation failure occurred in the roof and sidewall. Especially when encountered with a fault, roof falling, sidewall spalling, floor heave, and support structure failure appeared simultaneously. Therefore, the original support system was unable to control the stability of the surrounding rock, and roadway repair severely impacted productivity.

2.2 | Mechanical parameters of the surrounding rock

The properties of the surrounding rock have a large influence on the stability of rock engineering. To obtain a better understanding of the surrounding rock conditions, the uniaxial compression test, Brazilian splitting test, and shearing test were conducted on rock specimens obtained from the roof, coal, and floor of the roadway to obtain the rock properties. Figure 3 shows the test results for sandstone specimens obtained from the roadway roof, and the physical and mechanical properties of the surrounding rock specimens are listed in Table 1.

2.3 | Borehole televiewer detection

To further explore the failure characteristics of the surrounding rock above the roadway, borehole televiewer detection was employed to detect the fracture characteristics of the roof
The fracture characteristics obtained by the borehole televiewer detector are presented in Figure 5. The borehole televiewer detection results show that the top coal mass was relatively fractured, with intact and broken coal appearing alternately. Furthermore, some cracks extended from the coal mass to the upper mudstone and sandstone and gradually faded away with increasing depth. This phenomenon indicates that the current rock bolting support scheme cannot control the deformation of the surrounding rock, especially when the shallow top coal mass is seriously broken, which makes it difficult to maintain its stability.

2.4 Rock mass properties

In rock engineering, the surrounding rock mass is a complicated geological body that consists of intact rock and many discontinuities with various scales (such as joints, bedding planes, and faults), which is quite different from the mechanical properties of intact rock. Therefore, the rock specimen properties obtained from laboratory tests must be calibrated before being used in numerical simulations.

By analyzing the laboratory and field data, Zhang and Einstein presented a relationship between the rock classification index RQD and the ratio of the deformation modulus of intact rock \(E_r\) to that of a large-scale rock mass \(E_m\). The deformation modulus of the rock mass was evaluated as \(E_m = \left(10^{0.0186RQD - 1.91}\right) \cdot E_r\). Based on the compressive strength of the intact rock \(\sigma_n\), the compressive strength of the rock mass \(\sigma_m\) can be obtained from the empirical formula \(\sigma_m = \sigma_n \cdot \left(\frac{E_m}{E_r}\right)^n\), which was proposed by Singh and Seshagiri Rao, where \(n = 0.63\). The calibrated rock mass properties are summarized in Table 2.

### Table 1: Physical and mechanical parameters of rock specimens from the Tangyang coal mine

| Lithology       | Compressive strength (MPa) | Tensile strength (MPa) | Elastic modulus (GPa) | Poisson’s ratio | Density (g/cm³) | Cohesion (MPa) | Internal friction (°) |
|-----------------|-----------------------------|------------------------|-----------------------|-----------------|----------------|----------------|---------------------|
| Sandstone       | 45.77                       | 9.90                   | 8.21                  | 0.22            | 2.56           | 10.19          | 39                  |
| Mudstone        | 12.95                       | 2.10                   | 3.14                  | 0.35            | 2.43           | 6.31           | 37                  |
| Coal            | 10.79                       | 1.28                   | 0.60                  | 0.24            | 1.67           | 3.51           | 42                  |
| Sandy mudstone  | 19.12                       | 1.60                   | 4.50                  | 0.25            | 2.32           | 4.75           | 38                  |

3 Numerical model establishment

3.1 UDEC approach

Considering that the surrounding rock of a deep roadway is composed of rock blocks and discontinuities, the distinct element method (DEM) (Cundall 1971) is better suited than the finite element method (FEM) and finite difference method (FDM) to solve the discontinuity problems of deep underground engineering. Since the broken rock mass is usually block-shaped, the discrete block elements in the UDEC model are more suitable for the discontinuum analysis of excavation in a jointed rock mass. In the UDEC model, the rock is regarded as aggregate blocks bonded together via contacts. In this study, the elastic constitutive model and Coulomb slip model are applied to the block and contact, respectively. Each block in the model is made elastic and will not fail. The contact behavior of a contact is simulated by a spring, and the force is divided into a normal stress and shear stress. The contact behavior depends on the stress state and the properties of the contact surface, and failure can occur in the contact when the shear stress or tensile stress acting on the contact surface exceeds its contact strength, as shown in Figure 6.
3.2 | Microparameter calibration

The mechanical behavior of the numerical model depends on the microparameters of the block and contact, as shown in Figure 7. However, these parameters cannot be obtained directly from the physical and mechanical characteristics of the rock mass. Therefore, to calibrate the microparameters in the UDEC, a series of uniaxial compression simulations were carried out and continued until the simulated compressive strength and elastic module were consistent with the rock mass properties listed in Table 1.

The bulk modulus \( (K) \) and shear modulus \( (G) \) of the blocks are computed by incorporating the elastic modulus, \( E \) and Poisson’s ratio into the following formulas:

\[
K = \frac{E}{3(1-2\mu)} \quad (1)
\]

\[
G = \frac{E}{2(1+\mu)} \quad (2)
\]

The joint stiffness of the contact is calculated using the following formulas:

\[
k_n = 10 \left[ \frac{K + \frac{3}{4}G}{\Delta z_{\text{min}}} \right] \quad (3)
\]

where \( \Delta z_{\text{min}} \) is the smallest width of the zone adjoining the contact in the normal direction.

The cohesion, friction angle, and tensile strength of the blocks were obtained from a series of simulated compression tests. The simulated stress-strain curves and failure modes are shown in Figure 7, and the final calibrated microparameters for the various specimens are summarized in Table 3.

3.3 | Numerical model and simulation scheme for the 2313 haulage roadway

Based on the geological data and calibrated microparameters, a 30 × 25.3 m numerical model was built with the UDEC software, as shown in Figure 8A. To improve the calculation efficiency, small blocks with an average length of 0.35 m were only generated in a range of 19.8 × 9.8 m within the influence area of the excavation. The rest of the blocks were rectangular blocks that are more suitable for simulating the rock strata. A vertical stress of 16.5 MPa was applied at the top of the model to simulate the overburden pressure. The horizontal displacement in the lateral boundary and vertical displacement in the bottom were fixed.

\[
k_s = 0.4k_n \quad (4)
\]

where \( k_s \) is the joint stiffness of the contact in the normal direction.

The cohesion, friction angle, and tensile strength of the blocks were obtained from a series of simulated compression tests. The simulated stress-strain curves and failure modes are shown in Figure 7, and the final calibrated microparameters for the various specimens are summarized in Table 3.

### Table 2: Properties of intact rock and a rock mass

| Lithology         | Rock specimen | Rock mass |
|-------------------|---------------|-----------|
|                   | \( E_r \) (GPa) | \( \sigma_r \) (MPa) | RQD | \( E_m \) (GPa) | \( \sigma_m \) (MPa) |
| Sandstone         | 8.21          | 45.8      | 84  | 3.69          | 27.64 |
| Mudstone          | 3.14          | 13.0      | 52  | 0.36          | 3.30  |
| Coal              | 0.67          | 10.8      | 48  | 0.06          | 2.47  |
| Sandy mudstone    | 4.56          | 19.1      | 72  | 1.23          | 8.35  |
As the primary support scheme cannot effectively control the deformation of the roadway, several uneven deformations occurred in the roadway in the form of roof subsidence and side shrinkage. Therefore, the excavation of roadways without supports is first simulated to reveal the failure mechanism of the deep roadway surrounding rock. The numerical model was first equilibrated to produce the initial stress state, and then, the blocks in the designated area were deleted to simulate the excavation of the roadway. In addition, a gradually decreasing stress was applied on the roadway surface (Figure 8B) to simulate the process of the change in the stress in real excavation engineering. The stress release path is also shown in Figure 8C.

4 | NUMERICAL RESULTS

4.1 | Cracking characteristics

Many rock engineering accidents show that the failure of the surrounding rock is often caused by internal crack propagation, which gradually weakens the bearing capacity of the rock mass. To further reveal the failure mechanism of the roadway surrounding rock, the cracking process of the roof and sidewall is analyzed in this section.

Figure 9 shows the failure process of the roadway roof captured in the UDEC model. Note that the cracks are marked with blue lines. Under the effect of high in situ stress and excavation pressure relief, the surrounding rock experiences high stress, which leads to crack initiation, propagation, and coalescence. The cracking process indicates that the failure of the roof is a gradual process. Owing to the poor force state of the right angle, the crack first appears at the roadway corner due to the stress concentration. With increasing deviatoric stress, an increasing number of cracks appear in the shallow surrounding rock, and the cracks in the corner propagate to a greater depth and coalesce with each other. In this process, the shallow rock mass becomes fragmented, and the bearing capacity of the rock mass in the fractured zone is decreased significantly. The bearing structure transfers into the depths while cracks in the roof propagate deeper into the top coal, and the shallow coal mass is detached from the deep coal mass while the cracks coalesce. Finally, the separated coal mass falls due to its own gravity, and thus, roof failure occurs.

Figure 10 shows the gradual cracking process of the roadway sidewall. Similar to the cracking process of the roof, the cracks are also initiated from the roadway corner and propagate into the deep surrounding rock in an arch shape pattern. As the cracks continue to propagate and coalesce within the coal mass, the shallow coal begins to separate from the surrounding rock under the action of the internal extrusion force, which will finally result in rib spalling failure if sufficient support resistance is not applied.

Figure 11 shows the comparison of the roadway failure pattern between the simulation results and the field observations. It can be seen that the UDEC is able to simulate the main failure pattern in the field well, including roof subsidence and side shrinkage. The good agreement between the simulated results and field observations also proves that the calibrated microparameters of the rock strata in the numerical model are rational.
After excavation, the rock mass around the roadway will be damaged and will then be further fractured because it is subjected to unloading, and stress redistribution will occur. The stress distribution of the roadway surrounding rock is shown in Figure 12. According to Figure 12A, a large area of stress relaxation appeared around the roadway, mainly including the fractured surrounding rock area shown in Figure 10. Meanwhile, the shallow rock was in tension (Figure 12B), which could result in fragmentation, swelling, and separation of the surrounding rock.3

**Table 3**  Microparameters for the numerical model

| Lithology        | Density (kg/m³) | Bulk modulus (GPa) | Shear modulus (GPa) | Normal stiffness (GPa/m) | Shear stiffness (GPa/m) | Friction angle (°) | Bond strength (MPa) | Tensile strength (MPa) |
|------------------|-----------------|--------------------|---------------------|--------------------------|-------------------------|--------------------|---------------------|-----------------------|
| Sandstone        | 2560            | 2.20               | 1.51                | 188.4                    | 75.4                    | 39                 | 9.0                 | 0.9                   |
| Mudstone         | 2430            | 0.40               | 0.13                | 28.3                     | 11.3                    | 37                 | 1.2                 | 0.12                  |
| Coal             | 1670            | 0.38               | 0.02                | 3.2                      | 1.3                     | 42                 | 0.9                 | 0.09                  |
| Sandy mudstone   | 2320            | 0.82               | 0.492               | 67.3                     | 26.9                    | 38                 | 3.0                 | 0.3                   |

**Figure 8** Simulation scheme. A. The numerical model of the roadway surrounding rock. B, Virtual force applied on the surface of the roadway. C, Diagram of the stress release path.

**Figure 9** Simulated failure process of the roadway roof.
4.2 Displacement analysis

Figure 13 shows the simulated displacement vector map of the roadway surrounding rock. As shown in Figure 13, several uneven deformations occurred in the roadway, particularly rib spalling and the roof falling. Additionally, an asymmetrical deformation pattern was observed in the surrounding rock, while the deformation in the right side was slightly higher than that of the left sidewall, which was due to the uneven distribution of the blocks in the UDEC.

Two monitoring lines were set in the middle of the roof and right side to record the displacement of the surrounding rock, the spacing interval between the monitoring points was 1 m, and the length of the monitoring line was 8 and 6 m, respectively. Figure 14 shows the layout diagram of the measuring points and the monitoring results. The displacement inside the roadway continued to increase after the
excavation, but the growth rate varied from stage to stage. In the initial nine stages of stress release, the deformation rate was slow and stable; after the virtual support stress was completely released, the deformation rate increased significantly. However, there was an obvious difference in the displacement rule at varied depths in the roadway surrounding rock. For the surrounding rock in the roof, the range of the main deformation areas was within 3 m, while the deformation in the sidewall was more concentrated in a smaller range within 1 m.

5 | CONTROL TECHNOLOGY

5.1 | Support parameters

From the analysis of the numerical results given above, it can be seen that the major failure modes for the 2313 haulage roadway are roof subsidence and side shrinking. This is because the shallow surrounding rock is severely broken, which is mainly affected by excavation unloading, and the simple rock bolting support system cannot effectively control the large deformation. Thus, to effectively control the deformation of the surrounding rock of the roadway in the Tangyang mine, a combined control technology that uses a...
strong anchor bolt (cable) support, metal mesh, and steel ladder reinforcement is proposed. The detailed support parameters are described below.

Five φ18 × 2200 mm and four φ18 × 2200 mm superstrong thread steel bolts are used to support the roof and each sidewall, respectively. The row and line space of the roof and sidewall bolts are both determined to be 850 × 900 mm; two φ17.8 × 6000 mm steel cables are used to strengthen the roof support with a distance between the cable and roof midline of 950 mm, and the row and line space of the cables are set as 1900 × 1800 mm. Every bolt and cable is installed together with a steel anchor plate with sizes of 150 × 150 × 8 mm and 300 × 300 × 12 mm, respectively. A metal net was used to cover the coal. A steel ladder with a size of 3600 × 100 × 14 mm was designed to connect the bolts and cables in the roof. Prestressing forces of 50 and 80 kN are applied to the bolts and cables, respectively. The final support diagram is shown in Figure 15.

5.2 | Simulation verification

The combined support scheme is adopted in the UDEC to verify the control effect on the surrounding rock mass. The parameters of the support elements used in UDEC are listed in Table 4.

Figure 16 shows the displacement vector map of the roadway with the combined support system. We can see that the deformation of the surrounding rock tends to be more uniform, and there is no obvious side spalling or failure of the roof falling. The monitoring results in Figure 17 show that the deformation slowed down and gradually stabilized after the excavation. Compared with that in the roadway without the support system, the roof substance was well controlled.
with no failure of the roof falling, while the displacement of the sidewall decreased by 71% to 0.06 m. The resulting good control verified the rationality of the support system.

Figure 18 shows the comparison of the crack distribution of the roadway with the combined support scheme between the borehole televiewer and UDEC simulation. As shown in Figure 18, the fractured zone of the surrounding rock around the roadway decreased dramatically, especially the cracks in the top coal that did not exceed the mudstone strata. This indicates that the cable's high bearing capacity enhanced the stability of the top coal by increasing the resistance of the deeper surrounding rock. Figure 18 also shows the axial force state of the support elements, and we can see that all the elements are in the tension state under the swelling deformation of the surrounding rock. Note that the bolt in the corner is vulnerable to shear failure.

5.3 Engineering practice

The field test was also conducted to evaluate the effectiveness of the new support scheme, and two monitoring stations were set up in the conveying roadway of the 2313 working face to monitor the deformation features of the surrounding

**FIGURE 17** Simulated displacement evolution of the supported roadway

**FIGURE 18** Simulated fracture characteristics and force state of the support element
rock. The monitoring content includes the roadway surface displacement, roof separation, and the force of the bolt and cable. All the monitoring equipment was installed immediately after excavation to obtain the whole process of the surrounding rock deformation.

5.3.1 Force of the anchor monitoring results

During the excavation of the 2313 roadway, force sensors were installed between the bolt (cable) trays and nuts. In the roadway cross-section, one cable and two rock bolts were chosen for force monitoring, and the layout of the force sensors is shown in Figure 19A. The monitoring results of the bolt (cable) force are shown in Figure 19B. When driving occurred on the roadway, the bolt force increased steadily and then became stable. The bolt force increased from the initial pretightening force value to approximately 86 kN at a rate of approximately 3.8 kN/d. Compared with that of the bolts, the force increase rate of the cable was much faster. The cable force increased from the initial pretightening force value to 297 kN at a rate of approximately 24 kN/d, which indicated that the cable can improve the anchoring effect of the bolts by hanging the weak lower strata from the high strong rock strata with a high bearing capacity.

5.3.2 Displacement monitoring results

Figure 20 shows the monitoring data of the surface displacement of the 2313 haulage roadway, which was monitored by the cross-point method. We can see that the roadway displacement grew at a relatively fast rate, then slowed down, and finally stabilized. The final displacements of the roof, floor, and sidewalls were 110, 40, and 100 mm, respectively.

The inside-body displacement of the roadway was also monitored by the multipoint displacement monitor. The two base points in the roof were located 2 and 8 m above the roof, whereas the base points in the sidewalls were located 1.5 and 6 m away from the roadway sidewall. The monitoring results shown in Figure 21 indicate that the displacement of the surrounding rock inside the roof and sides started to increase faster and then slowed down and finally remained nearly constant after 12 and 5 days, respectively. The displacement inside the surrounding rock also showed a difference between the roof and sidewalls. The separation in the roof was mainly concentrated above 2 m from the roof, while the separation in the sidewalls was mainly concentrated within 1.5 m, which indicates that the damage was concentrated in the shallow surrounding rock. The deformation law inside the coal mass
has the same changing tendency as the simulated results shown in Figure 17, which also verifies the feasibility of the UDEC simulation.

6  CONCLUSION

To control the stability of the surrounding rock of deep roadways, a case study on the failure behavior and stability control technology of a deep roadway in the Tangyang coal mine is presented in this paper. Based on the field observation data and laboratory test results, the rock mass properties were first evaluated. Then, the microparameters of the rock mass used in the simulation were carefully calibrated. A detailed numerical model based on the geological situation of the 2313 haulage roadway was established in the UDEC model. The excavation was simulated by the stress release process.

Due to the unloading process caused by the excavation, the deviatoric stress was concentrated in the shallow surrounding rock, which resulted in rock failure. The crack initiated from the roadway corner, propagated into the deep surrounding rock, and finally coalesced with the other cracks. The shallow broken coal began to separate from the surrounding rock and finally led to the instability of the roadway. The simulated displacement vector map of the unsupported surrounding rock showed that several uneven deformations occurred around the roadway, particularly rib spalling and roof falling.

According to the failure characteristics of the roadway surrounding rock, a combined support scheme of “bolt-cable-mesh” was proposed and evaluated. The numerical simulation and field monitoring results showed that the new support could effectively control the deformation of the roadway surrounding rock.

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