INTRODUCTION

From the statistics, China's proven have over 1.6 trillion tons coal reserves. Among them, the percentage of thick coal seam accounted for about 50% and made up half of the total coal production in China. Obviously, the thick coal seam is particularly important for coal industry in China. Because of high-intensity mining and rigorous requirements of ventilation and transportation, the section area of main roadways is generally over 15 m. When the coal seam has relatively high strength, the driving roadway in ultrathick coal seam could not only increase coal production, but also dramatically improve the excavating rate. Therefore, this roadway driving method is widely used under permitting conditions.

In addition, the economic development in the last century led to a sharp increase in energy demand. Under this background, small coal mines in ultrathick coal seam were widespread. Their floor fracture zone and stress concentration zone had important effects on the stability of the roadway in the ultrathick coal seam. The roadways occurred roof leakage and rib extrusion which seriously affected safety production. The control of the roadway under these small abandoned mines in the same coal seam has become one of the key factors.

Many studies have been conducted on the support for large section roadways. For example, Kang et al put forward the technology of reproducing bearing stratum by grouting the loose and broken coal seams. This technology...
could increase the overall strength of roadway ribs. As a result, the deformation and failure of the roadways were well controlled; Yu et al.\textsuperscript{10} proposed the different support methods for the large section roadway in different positions and times. During the excavation, anchor bolt and cable are used for the large section roadway, and the bolt row spacing and the cable row spacing are $0.8 \times 0.8$ m and $2.4 \times 2.4$ m, respectively. In the mining stage, the single prop and $\pi$ girder are used, and the bolt row spacing and cable row spacing are reduced to $0.7 \times 0.7$ m and $1.4 \times 1.4$ m, respectively. In the advance section, the high water material is added for coal reinforcement. Based on the above control methods, the deformation of the large section roadway is obviously reduced; Zhan et al.\textsuperscript{11} proposed a support method of combining closed bracket of double-deck I-beam and grouting for the support of large section roadway under the high stress broken surrounding rock. The maximum deformation of roadway measured on site is not exceed 25 mm; Zhang et al.\textsuperscript{12} used UDEC polygonal methods to analyze the failure characteristics of the large section roadway and proposed a new scheme involving key area strengthening and high strength yielding bolts. The deformation of large section roadway is controlled at about 550 mm; Gao et al.\textsuperscript{13} analyzed the characteristics of the roof crack development of the large section roadway by the discrete element software. On this basis, the roadway roof control method based on the crack control is proposed, and the anchor cable support parameters are optimized. The separation of the roof of the roadway is about 15 mm; Liu et al.\textsuperscript{14,15} studied the mechanical properties of grouting materials and gave the function relationships among the grouting depth and rib displacement, and roof separation. Finally, they proposed control method for large section roadway by combining different length anchor cable and grouting; Wu et al.\textsuperscript{16} proposed that the time of the secondary support should be coordinated with the large section roadway deformation.

The abovementioned studies put forward different support schemes for the roadway. Next, a numerical software, Rocsciences Phase2D (RS2), was adopted to predict the deformation and failure characteristics of roadway under the proposed secondary support schemes. Finally, the industrial test was carried out in the roadway in Tashan Coal Mine.

2 | ENGINEERING BACKGROUND

2.1 | Geological conditions

The Carboniferous 3-5# coal seam, with the buried depth of 400-800 m, is currently mined in Tashan Coal Mine. As shown in Figure 1., two smalls coal mines, with an interval of 30 m, could be found 10.5 m above a main roadway. These abandoned mines, with a thickness of 3 m coal seam, were 120 m long and 30 m wide. The main roadway was on the floor of 3-5# coal seam. From the drill holes arranged near main roadway, it was observed that the thickness of 3-5# coal seam was 17.5 m, and the immediate roof is medium sandstone with 4.3 m thickness, while the floor is coarse sand with 3.0 m thickness.

2.2 | Design of the original support

The main roadway is used for material transportation. The size of the roadway is $6.5 \times 4.0$ m with a length of 2400 m.
The 1070 roadway adopts the support method combined bolt and anchor cable. The primary support scheme is shown in Figure 2.

1. The support scheme for roadway roof: The anchor bolt spacing was 0.9 m × 0.9 m, while the anchor cable spacing was 1.2 m × 1.8 m. The anchor bolt, in which pretightening was 69 kN, had a length of 2.7 m with a diameter of 22 mm. The length of anchor cable was 10 m with a diameter of 17.8 mm. It adopted 1 × 19 structure prestressed steel wire with a 79 kN pretightening force. There are two cables arranged in each row. The anchor cable tray was 300 mm × 300 mm with a 16 mm diameter. The size of rectangular metal mesh was 100 mm × 100 mm.

2. The support scheme for roadway ribs: The combined support of bolt and metal mesh was used to support the ribs. The related parameters were the same as that of the roof support.

3 | DEFORMATION AND FAILURE CHARACTERISTICS OF THE ROADWAY UNDER SMALL ABANDONED MINES IN THE SAME COAL SEAM

The deformation and failure characteristics of the roadway were determinants of the secondary support schemes. The fracture zone of roadway could determine the grouting filling zone. The roof separation of the roadway and lateral displacement of ribs could be the basis for the parameter selection of the anchors. Therefore, it is very necessary to reveal the roadway deformation and failure characteristics. To further reveal the deformation and failure laws of the roadway surrounding rock, the RS2 was used for the numerical simulation.

3.1 | Numerical model

3.1.1 | Strength criterion of rock mass at the stope

The mechanical parameters of the intact rock from the roadway were obtained by experiments, as shown in Table 1. Because of fractures in the rock, the strength of intact rock from the experiment was generally higher than that of the rock from the stope. To simulate the strength reduction of intact rock, the generalized Hoek-Brown strength criterion was used, as follows 17:

\[
\sigma_1 = \sigma_3 + \sigma_{ci} \left( \frac{m_b \sigma_3 + s}{\sigma_{ci}} \right)^a
\]

where \( \sigma_1 \) is the maximum principal stress and \( \sigma_3 \) is the minimum principal stress; \( m_b \) is a reduced value; \( s \) and \( a \) are constants; and \( \sigma_{ci} \) is the uniaxial compressive strength (UCS) of the intact rock pieces. The formulas of \( m_b, s, \) and \( a \) are as follows 17:

\[
m_b = m_i \exp \left( \frac{GSI - 100}{28 - 14D} \right)
\]

\[
s = \exp \left( \frac{GSI - 100}{9 - 3D} \right)
\]

\[
a = \frac{1}{2} + \frac{1}{6} \left( e^{-GSI/15} - e^{-20/3} \right)
\]

where \( GSI \) is the Geological Strength Index; \( D \) is a disturbance factor. The \( D \) is 0 in this study 18; \( m_i \) is a material

| Lithology            | Thickness, m | Elastic modulus, E, GPa | Poisson's ratio λ | \( \sigma_{ci} \), MPa | GSI     | \( m_i \) | \( m_b \) | \( s \) | \( a \) |
|----------------------|--------------|-------------------------|-------------------|----------------------|---------|---------|---------|---------|-------|
| Sandstone            | 5.1          | 41.2                    | 0.26              | 38.6                 | 87      | 17      | 10.69   | 0.24    | 0.50  |
| Siltstone            | 9.4          | 45.1                    | 0.27              | 55.5                 | 92      | 16      | 12.02   | 0.41    | 0.50  |
| Sandy mudstone       | 5.7          | 34.9                    | 0.30              | 33.3                 | 82      | 14      | 7.36    | 0.14    | 0.50  |
| Conglomerate         | 3.2          | 36.2                    | 0.31              | 23.6                 | 86      | 19      | 11.52   | 0.21    | 0.50  |
| Mudstone             | 4.4          | 15.8                    | 0.21              | 17.4                 | 76      | 10      | 4.24    | 0.07    | 0.50  |
| Medium sand          | 4.3          | 45.1                    | 0.28              | 35.2                 | 89      | 24      | 16.20   | 0.30    | 0.50  |
| 3-5# coal seam       | 17.5         | 6.8                     | 0.21              | 15.9                 | 65      | 7       | 2.01    | 0.02    | 0.50  |
| Coarse sand          | 3.0          | 34.2                    | 0.31              | 43.9                 | 90      | 23      | 16.09   | 0.30    | 0.50  |
| Medium sand          | 5.0          | 25.3                    | 0.29              | 23.6                 | 85      | 14      | 8.19    | 0.19    | 0.50  |
| Fine sandstone       | 11.2         | 47.6                    | 0.25              | 80.3                 | 95      | 25      | 20.91   | 0.57    | 0.50  |
constant for the intact rock. It was provided that GSI empirical parameters and empirical values of \( m_p \), \( s \), and \( a \), as shown in Table 1, could be calculated by a software.

### 3.1.2 Calculation of caving zone

It was analyzed that the heights of caving zones in a number of mines from China and USA and obtained the statistical regression formula for the height of caving zone, as follows:

\[
H = 100h/(c_1 h + c_2)
\]  

(5)

where \( h \) is the mining height, m; \( c_1 \) and \( c_2 \) are related parameters of roof lithological characters, as shown in Table 2.

According to formula (5) and the types of immediate roof, the height of the caving zone was calculated to be 13.5 m. Then, the caving zone of the small abandoned mines could be obtained by the angle and height of caving zone.

### 3.1.3 The constitutive model of the caving zone

To simulate the compaction characteristics of the broken rock in the goaf by the finite element software, Salamon proposed the broken rock mass compaction theory for the caving zone. That is, the stress-strain relationship of the rock mass was given. The specific formula is as follows:

\[
s_i = E_0 \varepsilon/(1 - \varepsilon/\varepsilon_m)
\]  

(6)

where \( s_i \) is the vertical stress in the goaf, MPa; \( E_0 \) is the initial tangential modulus of the rock mass in the caving zone, MPa; \( \varepsilon \) is the current vertical strain; and \( \varepsilon_m \) is the maximum vertical stress.

\( \varepsilon_m \) could be determined by formula (7) as follows:

\[
\varepsilon_m = (b-1)/b
\]  

(7)

Table 2: The height coefficients of the caving zone

| Types of immediate roof | Uniaxial compressive strength, MPa | Coefficients | \( c_1 \) | \( c_2 \) |
|-------------------------|-----------------------------------|--------------|--------|--------|
| Strong                  | >40                               | 2.1          | 16     |
| Moderate                | 20-40                             | 4.7          | 19     |
| Weak                    | <20                               | 6.2          | 32     |

where \( b \) is the comprehensive coefficient of rock mass expansion in the caving zone.

\( b \) could be determined by formula (8) as follows:

\[
b = 1 + 0.01(c_1 h + c_2)
\]  

(8)

where \( h \) is the mining height, m; \( c_1 \) and \( c_2 \) are related parameters of roof lithological characters, as shown in Table 2.

\( E_0 \) could be determined by formula (9) as follows:

\[
E_0 = 10.39\sigma_c^{1.042}/b^{7.7}
\]  

(9)

where \( \sigma_c \) is the uniaxial compressive strength.

The \( \varepsilon_m \) and \( b \) of caving zone after the mining are 0.18 and 1.22, respectively, by formulas (7) and (8). Then, the stress-strain relationship of the rock mass of caving zone could be obtained by substituting \( \varepsilon_m \) and \( E_0 \) into formula (6), respectively.

The double-yield model could describe the stress-strain relationship in the formula. The relationship was used for the caving zone of the small coal mine and a two-dimensional axisymmetric model of \( \Phi 50 \text{ mm} \times 100 \text{ mm} \) was established, as shown in Figure 3. The right side of the model constrained the horizontal displacement and the bottom of the model was constrained by the vertical displacement. A load of \( 1.0 \times 10^{-5} \text{ m/s} \) was applied in the roof. It was extracted that the stress-strain relationship in the process of compression from numerical simulation. Then, the stress-strain relationship of the formula (6) was added into Figure 3.

By comparison, the numerical simulation results were very consistent with the Salamon model results, thus verifying the reliability of the model.

### 3.1.4 Model establishment

As shown in Figure 4, a 2D numerical model was established with a size of \( 190 \text{ m} \times 68.7 \text{ m} \). The generalized Hoek-Brown criteria were for rock. The caving zone adopted the double-yield Model. The right and left side of the model were constrained by the horizontal displacement. A vertical load of 10 MPa was applied to the top of the model, and the initial ground stress was also applied. The mesh size of the model was between 0.2 m and 1.0 m. To simulate the fracture range of the main roadway, the Voronoi elements were divided around the roadway. The model excavation process was in the following. First, the small coal mine on the left was extracted and the double-yield Model was used for the caving zone. Second, the same step was applied on the right small mine. Third, the bolt, anchor cable, and metal mesh were applied.
3.2 | Deformation and failure characteristics of the roadway

The deformation and failure characteristics of roadway surrounding rocks could be effectively revealed by monitoring roof separation amount, lateral displacement of ribs, vertical stress of roadway ribs, and the fracture area of the roadway.

3.2.1 | The monitoring scheme

As shown in Figure 5, two measuring lines were set in the numerical model. Line I lied in the middle of roadway roof with a length of 10 m and is used to monitor roof separation amount. Line II lied in the middle of roadway ribs with a length of 20 m and was used to monitor the vertical stress and lateral displacement on ribs of the roadway.

3.2.2 | Roof separation amount

By the data in measurement line I, Figure 6 showed relationship between the roof separation amount and the distance from roadway roof. When the distance from the roof was between 6 m and 10 m, the roof separation amount was small and changed slowly with the range from 0.18 to 0.29 m. When the distance was between 0 m and 6 m, the roof separation amount increased rapidly and the values were 0.29-1.18 m, which was about two to three times of the former. It was indicated that the roadway roof suffered serious damage when the distance from roof was 0-6 m.

3.2.3 | Lateral displacement on ribs of the roadway

The lateral displacement on ribs of the roadway caused plastic failure and the loss of bearing capacity. This ultimately
made the arch foot lose for roadway and further exacerbated roof subsidence. Figure 7 shows the lateral displacements on the ribs of roadway.

In Figure 7, the maximum lateral displacements are 0.39 m and 1.09 m on left rib and right rib of the roadway, respectively. The lateral displacement changed vastly near the ribs. The movement range on the right rib was within 12.9 m, and that on the left rib was within 4.2 m. In addition, the maximum lateral displacement on the right rib was 0.7 m larger than that on the left rib. It was about three times of the maximum lateral displacement of the left rib.

### 3.2.4 Vertical stress on both ribs of the roadway

The vertical stress on ribs is an important indicator of the stress states of the roadway surrounding rock. Figure 8 shows the data of measurement line II. The abutment stress peak on the left rib was 11.9 MPa, and the stress concentration factor was 1.2. The abutment stress peak on the right rib was 18.0 MPa and the stress concentration factor was 1.8, which was obviously larger than that on the left rib. In addition, the depth of the maximum vertical stress on the left rib was 1.9 m, while the depth of the maximum vertical stress on the right rib was 9.8 m. It could be seen that the failure depth on the right rib was larger.

### 3.2.5 Fracture range of the roadway surrounding rock

The failure range of the roadway surrounding rock is the key to determine the grouting reinforcement zone of the roadway. As shown in Figure 9(A), there are three reasons for the surrounding rock failure of 1070 transportation roadway. First, many fractures in the floor of small abandoned mines were formed after mining and the floor strength decreased significantly. The roadway roof was in the fractured zone of the floor and its integrity was damaged. Second, the plastic failure occurred on the right rib of the roadway affected, which caused the roadway roof to lose the bearing arch foot and further intensified roof failure. Third, the length of the bolt was basically smaller than the fracture depth of roadway surrounding rock. The tensile fracture could be observed in the bolts and anchor cable.

Under the influences of the abovementioned three factors, the roadway occurred right rib extrusion and roof leakage. Then, the Voronoi elements were applied to calculate the failure depth of the roadway roof, floor, left rib, and right rib. They were 5.0 m, 3.0 m, 4.0 m, and 4.0 m, respectively. From the field observation, serious roof leakage and spalling occurred in the roadway, as shown in Figure 9(B). It showed that the main roadway surrounding rock occurred serious deformation and failure with roof leakage and right rib extrusion. The roof separation amount reached 1.30 m, and the large area of roof leakage could be observed. The lateral displacement occurred on right rib and the amount was 1.10 m. The field measurement results are in good agreement with the numerical simulation.
4 | THE SECONDARY SUPPORT SCHEMES FOR THE LARGE SECTION ROADWAY

4.1 | The way of selecting secondary support

According to the damage range of the roadway surrounding rock, the different failure depths could be found in the roof, floor, left rib and right rib of the roadway. Therefore, the grouting technology was necessary to improve the coal seam, strength of the fractured zone. The new network skeleton structure after the grout consolidation could improve the rock mass deformation modulus and greatly weaken the stress concentration at the ends of fractures. \(^{32}\) It also could change the cracks’ original propagation and fracture mechanisms and remarkably increase the bearing capacity, antideformation and antifailure capacities.

According to the lateral displacement and vertical stress characteristics on both ribs of the roadway, the surrounding rock on the right side of the roadway had larger vertical stress and occurred more serious deformation and failure. Therefore, the single prop was adopted near the right rib and it transferred roof loads to the floor and avoided too large vertical stress on the right rib.

According to the lateral displacement on both ribs and roof separation, the metal support was preferred after the grouting. \(^{33}\) This was because the metal shed was easy for operation and formation. Moreover, with larger supporting strength and strong antideformation capacity, it was helpful for roadway reinforcement. The bolt, anchor cable, and metal mesh could also be used. Finally, considering the tensile failure of the bolt and anchor cable, the high-strength bolt and anchor cable were adopted and their pretightening forces were increased. Based on the abovementioned supporting principles, after the grouting for surrounding rock, two support methods were proposed. One is the support method combined I-shape support and anchor bolt (hereinafter referred to as scheme I), another is the support method combined U-shaped support and anchor bolt and cable (hereinafter referred to as scheme II).

4.2 | Support parameter optimization

In order to optimize the supporting parameters of bolt and cable of the roadway, as shown in Figure 10, FLAC numerical calculation software is used to analyze the additional stress field of bolt and cable under different lengths, spacing, pretension, and combinations of anchor and cable. According to additional stress field, reasonable bolt and cable support parameters are determined, and the specific optimization results are given in the secondary support schemes.
4.3 | **The method of secondary support**

4.3.1 | **The scheme I**

As shown in Figure 11, the scheme I was adopted. First, the roof leakage area was filled. Then, the grouting was conducted in the fractured zone of roof, ribs and floor. Second, I-shape support was used with 4.0 m in height, 6.2 m in length and 1.0 m in row spacing. Third, the single hydraulic prop was added on the near right rib. Fourth, the bolt was applied in the grouting reinforcement and filling area. The support parameters are listed as follows:

The support for the roof: The bolt was a high-intensity left spiral steel. Its yield strength was 600 MPa, and strength limit was 800 MPa. Its diameter was 22 mm and the length was 2.4 m. The spacing of the bolt was 0.8 m × 1.0 m. The pretension for the bolt was 100 kN. The anchorage angle of the bolt near two ribs of the roadway was 10°. The W steel strip was 4 mm thick and 250 mm width. The rhombus metal mesh was added on the roof.

The support for on both ribs of the roadway was same with the roof.

4.3.2 | **The scheme II**

As shown in Figure 12, the scheme II is as follows: First, the roof leakage area was filled. Then, the grouting was performed for the fracture zones of roof, ribs, and floor. Second, U29-shaped support was used with 0.5 m interval. The radius of U-shaped support was 2.75 m. The height was 1.49 m. Wood was applied on the back of U-shaped support. Third, bolt and anchor cable were conducted.

The support for the roof: The anchor cable was a high-strength left spiral steel. Its yield strength was 600 MPa and the strength limit was 800 MPa, respectively. Its diameter was 22 mm with 2.4 m length. The bolt spacing was 0.8 m × 0.8 m. The anchor cable had a diameter of 17.8 mm and a length of 7.8 m. Its pretightening force was 150 kN with a total of 5. The anchorage length was 2.5 m. The high-strength tray was adopted with the specification of 300 mm × 300 mm × 16 mm. The W steel strip was 4 mm thick and 250 mm wide. The rhombus metal mesh was added.

The support parameters were the same as those of the roof.

5 | **DEFORMATION AND FAILURE CHARACTERISTICS OF THE ROADWAY**

After the secondary supporting, to reveal the deformation and failure characteristics of the roadway and predict the secondary supporting effects, RS2 was used to simulate the roof separation, rib displacement, vertical stress on the ribs and failure zone. During the numerical simulation, the grouting reinforcement and filling area were determined according to the roadway surrounding rock fractured zone. Then, it was necessary to get the mechanical properties of the rock at the stope, the caving zone and its constitutive model, etc. Finally, according to the secondary support schemes, constraints were applied such as I-shape steel support, U-shaped support, bolt, and anchor cable.

5.1 | **Filling area and numerical model establishment**

According to the fractured range of the roadway surrounding rocks in Section 3.2.5, Figure 13 shows the grouting filling and reinforcement area. Based on the characteristics of filling materials, the strength of the surrounding rock in the filling area should be increased by 20%.14

Similar to the numerical modeling process in Section 3.1, the numerical models of scheme I and II were established. The material strength was increased by 20% in the filling area. Moreover, the I-shaped steel support, U steel support, single hydraulic prop, anchor bolt, anchor cable, W steel strip, and metal mesh were applied, as shown in Figure 14.

5.2 | **Deformation and failure characteristics of the roadway**

After the construction of the secondary support, the monitoring scheme for the roadway deformation and failure was the same as that in Section 3.2.1. It formed a comparison group with the simulation results of the original support in Section
3.2. The surrounding rock deformation and failure and vertical stress laws were analyzed before and after the secondary support, which were reflected by roof separation amount, vertical stress on the ribs, lateral displacement on the ribs, and failure zone of the roadway.

5.2.1 | Roof separation amounts

As shown in Figure 15, the roof separation amounts within a 10 m distance from the roof were extracted. The characteristics of different supporting schemes were compared and analyzed.

In Figure 15, the maximum roof separation amounts were 1.18 m, 0.10 m, and 0.02 m in the original scheme, scheme I, and scheme II, respectively. The roof separation ranges were 0-10 m, 0-2.0 m, and 0-0.5 m in the original scheme, scheme I, and scheme II, respectively. It could be seen that the roof separation amounts and ranges decreased remarkably in scheme I and scheme II, indicating that the roadway roof separation was well controlled.

5.2.2 | Lateral displacement on both ribs of the roadway

As shown in Figure 16, the lateral displacements of ribs within a distance of 20 m were obtained. The lateral displacements under different supporting parameters were compared and analyzed. The lateral displacements on the left rib were 0.39 m, 0.04 m, and 0.003 m in the original scheme, scheme I, and scheme II, respectively, while maximum lateral displacements on the right rib were 1.090 m, 0.140 m, and 0.023 m in the original scheme, scheme I, and scheme II, respectively. Obviously, the lateral displacements on the right rib were larger than those on the left rib. Compared with the original scheme, the lateral displacements under scheme I and II were reduced significantly.

In addition, the lateral displacement of the ribs near roadway sides changed very rapidly. The left rib movement ranges were 4.2 m, 1.8 m and 1.7 m in the original scheme, scheme I, and scheme II, respectively. The right movement ranges were 12.9 m, 8.6 m, and 3.9 m, respectively. Compared with the original scheme, the movement ranges on ribs of the roadway under scheme I and scheme II decreased effectively.

The above analyses indicated that the lateral displacement on both rib of the roadway could be well controlled under scheme I and scheme II.
5.2.3 | Vertical stress on both ribs of the roadway

As shown in Figure 17, the vertical stresses on the ribs within a distance of 20 m were obtained. In Figure 17, the abutment stress peaks on the left rib of the roadway were 11.9, 16.4, and 12.2 MPa in the original scheme, scheme I, and scheme II, respectively. The stress concentration factors were 1.2, 1.2, and 1.6, respectively. The abutment stress peaks on the right rib of the roadway were 18.0, 22.0, and 20.1 MPa, respectively. The corresponding stress concentration factors were 1.8, 2.2 and 2.0, respectively. In addition, the depth of the maximum stress on the left rib were 1.9 m, 3.6 m, and 4.0 m in the original scheme, scheme I, and scheme II, respectively. The depth of the maximum stress on the right rib was 9.8 m, 2.7 m, and 12.0 minutes the original scheme, scheme I, and scheme II, respectively. Compared with scheme II, the abutment stress peak on the right rib was larger under the scheme I due to the influence of the single hydraulic support.

According to the abovementioned laws, the bearing capacity and bearing range of the surrounding rock increased to a certain degree, indicating that grouting reinforcement and filling areas, I-shape steel support, U steel support, and single hydraulic prop had sound supporting effects on the roadway.

5.2.4 | Fractured areas of the roadway surrounding rock

After grouting filling and reinforcement, the bearing capacity and antideformation capacity of the roadway surrounding rock increased greatly. Figure 18 shows the shear failure and tensile failure areas of the roadway surrounding rocks under scheme I and II.
Compared with the original supporting scheme, the shear and tensile failure areas of the roadway decreased significantly under scheme I and scheme II and the roadway failure was controlled effectively. In addition, no failure occurred on the roof near right rib of the roadway supported by the single hydraulic prop, proving that the single hydraulic prop was effective to control the deformation and failure. Compared with scheme I, scheme II had better effects in controlling the roadway surrounding rock failure and deformation.
6 | ENGINEERING APPLICATION

6.1 | Engineering background

The scheme I and scheme II were effective in controlling the roadway surrounding rock deformation, and scheme II had better effects from Section 5. At different subsection of the roadway in the Tashan Coal Mine, the support schemes were used according to its deformation and failure states. As shown in Figure 19, scheme I and scheme II were used at the different subsection of the roadway. The monitoring scheme was used to analyze roof separation amount, vertical stress,

![FIGURE 19](image-url) Secondary support positions and schemes

![FIGURE 20](image-url) Monitoring equipment

![TABLE 3](image-url) Deformation and stress of the roadway under secondary support

| Roof separation, m | Distance from roof, m | 0 | 1 | 2 | 3 | 6 | 10 |
|--------------------|-----------------------|---|---|---|---|---|----|
| Scheme I           |                       | 0.120 | 0.06 | 0.043 | 0.03 | 0.028 | 0.025 |
| Scheme II          |                       | 0.020 | 0.021 | 0.028 | 0.027 | 0.025 | 0.022 |

| Lateral displacement, m | Distance into rib, m | 0 | 2 | 4 | 6 | 8 | 10 |
|------------------------|----------------------|---|---|---|---|---|----|
| Scheme I               |                       | 0.160 | 0.090 | 0.060 | 0.045 | 0.020 | 0.010 |
| Scheme II              |                       | 0.025 | 0.022 | 0.018 | 0.008 | 0.007 | 0.007 |

| Vertical stress, MPa | Distance into left rib, m | 0 | 2 | 4 | 6 | 8 | 10 |
|---------------------|----------------------------|---|---|---|---|---|----|
| Scheme I            |                           | 0 | 6.4 | 9 | 17 | 10 | 8  |
| Scheme II           |                           | 0 | 15  | 23 | 16 | 15 | 14.5 |
| Distance into right rib, m |                   | 0 | 2  | 4 | 6 | 8 | 10 |
| Scheme I            |                           | 0 | 11.1 | 12.1 | 10.5 | 8.5 | 8.4 |
| Scheme II           |                           | 0 | 12.0 | 14.0 | 15 | 15.8 | 17.1 |
and lateral displacement on both ribs, as well as evaluate the supporting effects.

6.2 Deformation and stress of roadway surrounding rock under scheme I and scheme II

In order to monitor the vertical stress of the two ribs of the roadway, the stress monitoring meters are installed at the depth of 2, 4, 6, 8, and 10 m of the two ribs of the roadway. Similarly, the multipoint displacement meters are installed at the above positions. In order to measure the roof separation, multipoint displacement meters are installed at the roof depth of 1, 2, 3, 6, and 10 m respectively, and the monitoring equipment is shown in Figure 20.

The roof separation amounts, vertical stresses, and lateral displacements on both ribs of the roadway were monitored, see Table 3 in detail.

It was indicated that (a) the maximum roof separation amounts were 0.120 m and 0.028 m in scheme I and scheme II, respectively. (b) The maximum roof separation amount was 1.30 m in the original scheme. Therefore, the roof separation amounts under scheme I and scheme II decreased 90.7% and 97.8%, respectively. (c) The maximum lateral displacements on the right rib were 0.160 m and 0.025 m in scheme I and scheme II, respectively. Therefore, lateral displacements dropped 85.5% and 97.7%. (d) Compared with the original scheme, the vertical stress peaks increased from 12.3 MPa to 17.0 MPa and 12.9 MPa in scheme I and scheme II, respectively. The corresponding stress concentration factors increased from 1.2 to 1.7 and 1.3. The vertical stress peaks on the right rib increased from 17.3 MPa to 23.0 MPa and 17.8 MPa in scheme I and scheme II, respectively. The corresponding stress concentration factors increased from 1.7 to 2.3 and 1.8.

Therefore, the filling, single hydraulic prop, I-shape steel support, and U steel support were effective in strengthening the support. They were useful to enhance the bearing capacity on both ribs of the roadway and the antideformation capacity of roadway.

According to the abovementioned deformation and stress states, the bearing capacity of 1070 auxiliary transportation roadway increased and the roadway deformation was well controlled under scheme I and scheme II.

7 CONCLUSIONS

1. The roadway, under the small coal mines in the same coal seam, is seriously damaged by the impact of the floor fracture zone and the concentration area of coal pillars. The roof separation and lateral displacements reached 1.18 m and 1.09 m, respectively, and the stress concentration coefficient is 1.9.
2. RS software is used to reveal the stress distribution characteristics and fracture development law of the roadway. From the above laws, two support schemes are proposed. That is, after grouting for roadway, I-shaped support combined with anchor bolts (scheme I) and U-shaped support combined with anchor bolt and cable (scheme II). The numerical simulation results show that the roof separation and lateral displacement of scheme I are reduced by 91.5% and 87.2%, respectively, while the roof separation and lateral displacements of scheme II are reduced by 98.3% and 98.9%, respectively.
3. From field measurement, roof separation and lateral displacement of scheme I were reduced by 90.7% and 85.5%, respectively, while roof separation and lateral displacement of scheme II were reduced by 97.8% and 97.7%, respectively. The results of field measurement and numerical simulation are consistent, and the control effect of the roadway is good.
4. The paper only focuses on the roadway support scheme. However, it does not consider the location of the roadway, which needs further discussion.

ACKNOWLEDGMENTS

This work was funded by the State Key Research Development Program of China (2018YFC0604500) and Talents Project of Liaoning Revitalization (XLYC201807219). The authors gratefully acknowledge the financial support from the organization.

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REFERENCES

1. Yu B, Zhang R, Gao M-Z, Li G, Zhang Z-T, Liu Q-Y. Numerical approach to the top coal caving process under different coal seam thicknesses. *Thermal Science*. 2015;19(4):1423-1428.
2. Shealy M, Dorian JP. Growing Chinese coal use: dramatic resource and environmental implications. *Energy Pol*. 2010;38(5):2116-2122.
3. Yu B, Zhao J, Kuang T, Meng X. In situ investigations into overburden failures of a super-thick coal seam for longwall top coal caving. *Int J Rock Mech Min Sci*. 2015;78:155-162.
4. Gibeová B, Hudeček V, Zubíček V, Brenek R, Vrublova D. SGEM. Longwall top coal caving in the centrum mine in the Czech republic. In: 16th International Multidisciplinary Scientific GeoConference, Sgem 2016: Science and Technologies in Geology, Exploration and Mining, Vol II. 2016:535-542.
5. Shimada H, Matsui K, Anwar H. Control of hard-to-collapse massive roofs in longwall faces using a hydraulic fracturing technique 1998.
6. Lu G, Wang C, Jiang Y, Wang H. Roadway supporting technology in fully mechanized workface with large mining height of specially thick coal seam in datong mining area. In: Yang WJ, Li QS (eds.).
1. Bai QS, Tu SH, Chen M, Zhang C. Numerical modeling of coal wall spall in a longwall face. *Int J Rock Mech Min Sci*. 2016;88(8):242-253.

2. Bertuzzi R, Douglas K, Mostyn G. An approach to model the strength of coal pillars. *Int J Rock Mech Min Sci*. 2016;89(11):165-175.

3. Zhang Q, Jiang B, Lv H. Analytical solution for a circular opening in a rock mass obeying a three-stage stress-strain curve. *Int J Rock Mech Min Sci*. 2016;86(4):16-22.

4. Yao Q, Li X, Sun B, et al. Numerical investigation of the effects of coal seam dip angle on coal wall stability. *Int J Rock Mech Min Sci*. 2017;100(10):298-309.

5. Wang SL, Hao SP, Chen Y, Bai JB, Wang XY, Xu Y. Numerical investigation of coal pillar failure under simultaneous static and dynamic loading. *Int J Rock Mech Min Sci*. 2016;84(4):59-68.

6. Shen B. Coal mine roadway stability in soft rock: a case study. *Rock Mech Rock Eng*. 2013;47(6):2225-2238.

7. Ju MH, Li XH, Yao QL, Li DW, Chong ZH, Zhou J. Numerical investigation into effect of rear barrier pillar on stress distribution around a longwall face. *Journal of Central South University*. 2015;22(11):4372-4384.

8. Gao F, Stead D, Kang H. Numerical Simulation of Squeezing Failure in a Coal Mine Roadway due to Mining-Induced Stresses. *Rock Mech Rock Eng*. 2014;48(4):1635-1645.

9. Wang SL, Hao SP, Chen Y, Bai JB, Wang XY, Xu Y. Numerical investigation of coal pillar failure under simultaneous static and dynamic loading. *Int J Rock Mech Min Sci*. 2016;84(4):59-68.

10. Shen B. Coal mine roadway stability in soft rock: a case study. *Rock Mech Rock Eng*. 2013;47(6):2225-2238.

11. Liu Y, Kang Z, Li Q, Chen S. Study on Numerical Simulation and Bolt Grouting Mechanism of Mine Fracture rock masses. *Appl Sci*. 2019;9(18):3891.

12. Gao F, Stead D, Kang H. Numerical Simulation of Squeezing Failure in a Coal Mine Roadway due to Mining-Induced Stresses. *Rock Mech Rock Eng*. 2014;48(4):1635-1645.

13. Kang H, Fan M, Gao F, Zhang H. Deformation and support of rock roadway at depth more than 1000 meters. *Chin J Rock Mech Eng*. 2015;34(11):2227-2241.

14. Jiang LS, Wang P, Zhang PP, Zheng PQ, Xu B. Numerical analysis of the effects induced by normal faults and dip angles on rock bursts. *CR Mec*. 2017;345(10):690-705.

15. Gao FQ, Stead D. The application of a modified Voronoi logic to brittle fracture modelling at the laboratory and field scale. *Int J Rock Mech Min Sci*. 2014;68(4):1-14.

16. Gao FQ, Stead D, Kang HP. Numerical investigation of the scale effect and anisotropy in the strength and deformability of coal. *Int J Coal Geol*. 2014;136(10):25-37.

17. Gao FQ, Stead D, Kang HP. Numerical investigation of the scale effect and anisotropy in the strength and deformability of coal. *Int J Coal Geol*. 2014;136(10):25-37.

18. Jin YY, Song L, Wang XZ, Adoko AC. Improvement of the U-shaped steel sets for supporting the roadways in loose thick coal seam. *Int J Rock Mech Min Sci*. 2013;60(2):19-25.

How to cite this article: Tai Y, Xia H, Liu H, Ma Z, Zhang Y. Control for the large section roadway under small abandoned mines in the same coal seam by secondary support. *Energy Sci Eng*. 2020;8:3476–3489. [https://doi.org/10.1002/ese.3758](https://doi.org/10.1002/ese.3758)