A study of the dynamic movement rule of overlying strata combinations using a short-wall continuous mining and full-caving method

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Abstract
At present in China, short-wall working face coal-mining methods are among the main methods used for the high-efficiency recoveries of boundary coal pillars and residual pillars. In traditional short-wall mining methods, overlying roofs are supported by coal pillars. However, issues such as the excessive number of coal pillars, low coal recovery efficiency, large areas of hanging roof arches, etc, have been encountered. In view of the aforementioned issues of the current short-wall mining technology, this study took the short-wall mining of the Yujialiang Mine in the Shendong mining area as the project background. The study then proposed a judgment method for the key stratum on the working face of the mine using a short-wall continuous mining and full-caving method. The proposed method was based on the theory of elastic plates and was verified using the theoretical analysis results of similar material simulation experiments. FLAC 3D software was adopted to analyze the stress and displacement evolution processes of the overlying rock strata during the advancement of the working face using a short-wall continuous mining and full-caving method. The results showed that the short-wall continuous mining and full-caving method not only had the ability to improve the recovery rate, but was also able to realize the effective control of the roof areas, thereby avoiding any large areas of hanging arches. The short-wall continuous mining and full-caving method is potentially of important significance for the improvement of coal recovery rates, as well as for the safe and highly efficient mining of boundary coal pillars and residual pillars in mines under special conditions.

Keywords
caving span, composite key stratum, numerical simulation, short-wall continuous mining and full caving, similar material simulation experiment
1 | INTRODUCTION

China is one of the largest coal-producing countries in the world, with its output of coal accounting for approximately 40% of the global total coal production. In addition, coal accounts for approximately 75% of the total production and consumption of primary energies in China. According to statistical data, long-wall working faces make up 60% of the existing coal resources in China. However, China’s coal resources have become gradually exhausted with the increasing intensity of the mining processes. The coal resources which have been applicable to long-wall mining have gradually decreased. However, coal reserves such as boundary coal pillars and residual pillars in mines which have resulted from long-wall mining have increased year by year. When considering the current resource coal-mining recovery rate in China, boundary coal pillars and residual pillars resulting from long-wall mining are known to account for nearly 20%, which causes the mines to scrap in advance, or to approach scrapping.

A short-wall mining method has the advantages of combined mining and driving processes, as well as high adaptability, etc. Therefore, short-wall mining methods have been applied to the mining of boundary coal pillars, and the recovery of residual pillars in mines. The main global mining countries have attached great importance to short-wall mining methods, and Chinese and international researchers have carried out thorough related studies. With respect to short-wall mining methods, Tan et al. adopted a catastrophe theory and damage constitutive equation in order to construct a catastrophic model for coal pillar instability, and obtained a calculation formula for the compression deformation of coal pillars. Zhang et al. started from the perspective of the coal pillar elastic-plastic zoning, and analyzed the plastic failure width of coal pillars. Then, the minimum critical elastic width for maintaining the stability of coal pillars was determined, after which the reasonable width criterion for coal pillars could be built. Jaiswal Ashok et al. regarded the coal bodies as Hoek-Brown strain softening material, and established a statistical expression for the coal pillar strength and postfailure modulus. For example, for the coal pillars with a width-to-height ratio of less than five, the coal pillar strength was found to be almost in a linear correlation with the width-to-height ratio of the coal body and nonlinear correlation with the uniaxial compressive strength of the coal body. Zhang et al. proposed a short-wall block backfill mining technique for the recovery of residual corner coal pillars and irregular blocks remaining after the exploitation of coal mines, and a solution is provided for the risks associated with gangue piling and the loss of water resources caused by coal mining. Sherrizadeh et al. examined the effects of different geological and mining factors on roof stability in underground coal mines, by combining field observations, laboratory testing, and numerical modeling. The research results showed that the bedding planes play an important role on the geomechanical behavior of roofs in underground excavations. Wilson et al. held that wider coal pillars have two zones along the width direction as follows: a “coal zone of the coal pillar,” and a “yield zone” outside the core zone. Wang et al. presented the calculation formula for the ultimate strengths of coal pillars unrestrained by mining geological conditions through the analysis of the influencing factors of the ultimate strength of coal pillars under a three-dimensional stress state. Bieniawski analyzed the dimension effect of coal pillar strength, and proposed that coal-rock masses have a critical dimension in which the coal pillar strength will not decrease become with decreases in dimension when the coal pillar dimensions are larger than the critical dimension. To determine the pillar stability, Waclawik et al. measured the vertical stress in two adjacent coal pillars which were diamond shaped and located within a row of pillars that formed the panel. The results of the stress-state and pillar displacement monitoring allowed the pillar loading and yielding characteristics to be described. Xu et al. built a time-dependent stability analysis model which considered the rigidity and softening of the roofs, along with the rheological characteristics of the coal pillars. In their study, the necessary conditions for the maintenance of the long-term stability of the coal pillars were obtained, as was the calculation formula for the minimum time of stability maintenance of the coal pillars. After considering the empirical data and technical parameters, Liu developed a shallow buried coal seam short-wall continuous mechanical mining method classification system, and established a shallow buried coal seam short-wall continuous mechanical mining technology system. Kushwaha and Banerjee conducted different working face advance numerical modeling studies using FLAC to assess the caving behavior of the overlying upper strata, and estimated the optimal obliquity of the face for faster and safe exploitation of the developed pillars by short-wall mining to improve the productivity. Nazarov et al. proposed a method to determine the rheological parameters of rocks based on the inverse problem solution with the use of data for the convergence in the room-and-pillar ore mining. Ghasemi et al. presented a multi-input/multi-output Mamdani fuzzy model by which to predict the pillar sizing in room and pillar coal mines. This model proved to be a suitable and practical technique that can be effectively used in the prediction of pillar sizing with a minor error. Guo et al. adopted a theoretical analysis and numerical simulation method in order to study the Wongawilli strip pillar mining technique. In said study, the stress, displacement, and plastic zone of a mine using the Wongawilli strip pillar mining technique for mining were achieved through the numerical simulation. The results showed that this method could effectively protect the surface structures from potential damages.
At present, the short-wall mining techniques still use the coal pillar method for roof control. Therefore, the main focus has been on the self-strength and long-term stability of coal pillars.\(^{19,25,32-35}\) However, there have been few studies conducted regarding the use of short-wall continuous mining and full-caving methods.

In view of these issues, this study took the short-wall continuous mining of the Yujiashan Mine in the Shandong mine area as the project background. A judgment method for the key stratum on the working face using a short-wall continuous mining and full-caving method was put forward, which was based on the theoretical analysis of the structural mechanics of the overlying rock using a short-wall continuous mining and full-caving method. A similar material model and a FLAC 3D numerical model were built according to the engineering geological conditions. Then, the movement and fractures of the roof, and the strata pressure behavior rule of the working face of the mine during the advancement phase were analyzed.

2 | THEORETICAL ANALYSIS ON THE KEY STRATUM OF THE OVERLYING STRATA USING A SHORT-WALL CONTINUOUS MINING METHOD AND THE ULTIMATE HANGING ARCH AREA

2.1 | Judgment of the key stratum of the overlying strata combination using a short-wall continuous mining method

Due to the small scope of the short-wall continuous mining method, and the small differences between the lengths and widths, a theory of elastic plates was used for the calculation of the load and the ultimate step of the hanging arch, as well as the judgment of position of the key stratum.\(^{11,36}\)

As shown in Figure 1, taking one section using a short-wall continuous mining method as an example, \(L\) is the section length, \(W\) denotes the section width, and the section height was represented by the thickness \(H\) from the working seam to the ground. Then, by assuming that there were a total of \(m\) layers of overlying strata from the bottom to the top, the \(i\)th strata had a thickness of \(h_i\), density of \(\rho_i\), elastic modulus of \(E_i\), Poisson’s ratio of \(\nu_i\), tensile strength of \(\sigma_i\), and bending rigidity of \(D_i\). Next, a basic equation was built by regarding these rock strata as the study object. A coordinate system was established on the middle surface of the rock strata (position of the equal division of the thickness).\(^{37,38}\)

In accordance with the equilibrium equation, geometric equation, physical equation, and boundary conditions of the thin plate, the rock strata deflection equation was obtained as follows:

\[
w_i = C q_i / D_i
\]

\[i = 1, n\]

where \(w_i\) denotes the deflection of the \(i\)th rock strata, \(m\); \(q_i\) is the load suffered by the \(i\)th rock strata, MPa; \(C\) represents the coefficient determined by different boundary conditions, which was related to the advancement distance and length of the working face, and \(D_i\) is the bending rigidity of the rock strata, \(D_i = E_i h_i^3 / 12(1 - \nu_i^2)\).

2.1.1 | Deformation compatibility conditions

It is known that when a key stratum experiences subsidence deformation, its total or local overlying rock strata will exhibit synchronous and coordinated subsidence. Meanwhile, the deformation of its lower rock strata will display no coordinated deformation. Assuming that there are two thick and hard rock strata in the overlying \(m\) rock strata, the 1st layer will directly overlay the immediate roof, and the 2nd layer will be above the \(n\)th rock strata. The topsoil layer (\(m + 1\) layer) overlies the \(m\)th rock stratum, and actions the \(m\)th rock stratum in the form of load \(q\). For example, the 1st layer is hard strata, which has a synchronous and coordinated movement with the controlled 2~\(n\) strata to form a composite rock plate.\(^{39}\) Then, in order to ensure that there is no bed separation during the subsidence of each rock stratum, each rock stratum should have the equivalent subsidence deformation: \(w_1 = w_2 = w_3 = \ldots = w_n\).

Therefore, the load relationship of the overlying strata will be as follows:

\[
q_1 = D_1 q_1, \ldots, q_m = D_m q_1 / D_1
\]

\[2\]

The load prior to the fracture of the key stratum can be obtained by substituting \(D_i\) into the above-mentioned formula as follows:

\[
q_i = D_i \left[ \sum_{i=1}^{m} q_i / \sum_{i=1}^{m} D_i \right] \left[ \frac{(E_i h_i^3 / 1 - \nu_i^2) \sum_{i=1}^{m} \rho_i h_i}{(\sum_{i=1}^{m} E_i h_i^3 / 1 - \nu_i^2)} \right]
\]

\[3\]
where $\sum_{i=1}^{n} q_i$ is the sum of the load borne by the rock strata, MPa,

$$\sum_{i=1}^{n} q_i = \sum_{i=1}^{n} \rho_i g h_i,$$

Similarly, when the $(n + 1)$th layer is a hard rock stratum, the load of this layer can be obtained in accordance with its load relationship with its controlled overlying $(n + 2)$ rock strata as follows:

$$q_{n+1} = (E_{n+1} h_{n+1}^3 / 1 - \nu_{n+1}^2) \left[ \rho_{n+1} g h_{n+1} + \sum_{i=n+1}^{m} E_i h_i^3 / 1 - \nu_i^2 \right] \Bigg/ \left( \sum_{i=n+1}^{m} E_i h_i^3 / 1 - \nu_i^2 \right)$$

(4)

If the 1st layer is the key stratum, its controlled scope will be the $n$th layer, and the $(n + 1)$th layer will be the second key stratum. Then, $q_{n+1} < q_1$, and the deformation condition for the key stratum will be as follows:

$$\left( E_{n+1} h_{n+1}^3 / 1 - \nu_{n+1}^2 \right) \left( \sum_{i=1}^{n} E_i h_i^3 / 1 - \nu_i^2 \right) > \left( E_1 h_1^3 / 1 - \nu_1^2 \right) \sum_{i=1}^{n} \rho_i g h_i$$

(5)

### 2.1.2 Strength conditions

When $q_{n+1} < q_1$ and the working face width is $b$, the ultimate hanging arch distance of the 1st and $(n + 1)$th layer will be $L_1$ and $L_{n+1}$, respectively. The ultimate hanging arch areas will be $S_1$ and $S_{n+1}$, respectively. Then, by considering the geometrical morphological characteristics of the overlying rock strata fracture, the strength judgment condition of the key stratum will be that the ultimate hanging arch area of the lower hard rock strata will be smaller than that of the upper hard rock strata.

$$S_1 < S_{n+1} (j = 1 \ldots k)$$

(6)

However, if the $(n + 1)$th layer does not satisfy Formula (6), the total rock strata load controlled by the $(n + 1)$th layer should be applied to the 1st layer, and then the judgment can be continued after the ultimate hanging arch area of the 1st hard rock strata has been re-calculated.

### 2.2 Ultimate hanging arch area of the key stratum

When the hard rock stratum is in an ultimate hanging position under the condition of the clamped plate, the maximal main moment value $M_a$ will occur in the middle section of the longer side. Then, the calculation can be obtained according to the improved solution of Marcus as follows:

$$|M_a| = qa^2 \left( 1 - \nu^2 \right) \left( 1 + \omega \nu^2 \right) / 12 \left( 1 + \omega^2 \right)$$

(7)

where $M_a$ is the maximum main moment, $q$ denotes the load borne by the hard rock strata, $a$ is the advancement distance of working face, $b$ denotes the width of the working face, $\nu$ is the Poisson’s ratio of the hard rock stratum, and $\omega$ represents the geometric shape coefficient of the goaf, $\omega = a/b$.

The relational expression for the first breaking of the hard rock stratum is as follows, according to $M_a = h^2 \sigma_i / 6$:

$$a = (h / \sqrt{1 - \nu^2}) \sqrt{2\sigma_i (1 + \omega^2) / q (1 + \nu \omega^2)}$$

(8)

where $h$ denotes the thickness of hard rock stratum, and $\sigma_i$ represents the tensile strength of the hard rock stratum.

The formula for the ultimate span $l_m$ of the clamped plate is as follows:

$$l_m = \left( h / \sqrt{1 - \nu^2} \right) \sqrt{2 \sigma_i / q}$$

(9)

where $l_m$ is the ultimate span of the clamped plate, which is referred to as the span criterion of the hard rock stratum.

Then, the ultimate hanging arch distance $a$ of the key stratum can be obtained as follows after substituting $\omega = a/b$ and Formula (9) into Formula (8):

$$a = \begin{cases} b \sqrt{\nu^2 b^2 + 4\sigma_i^2 (b^2 - l_m^2) - \nu b^2} / \sqrt{2 (b^2 - l_m^2)} & (l_m < b < \sqrt{2/1 + \nu l_m}) \\ b \sqrt{b^2 - \sqrt{b^2 - 4\sigma_i^2 (l_m^2 - \nu b^2)}} / \sqrt{2 (l_m^2 - \nu b^2)} & (b \geq \sqrt{2/1 + \nu l_m}) \end{cases}$$

(10)

When $b < l_m$, the upper hard rock strata of the coal seam will be stable and without caving. However, when $b > 3l_m$, the breaking span of the hard rock strata will be close to the span criterion. For example, the width of the working face will have only a very small influence on the size of breaking span. Therefore, the influence of the width of the working face on the breaking span of hard rock strata may not be considered if the width of the working face is large enough, and the thin plate model can be replaced by a beam model.

In this study, the ultimate hanging arch area was $S = a \times b$. Therefore, the formula for the ultimate hanging arch area $S$ of the key stratum was as follows:

$$S = \begin{cases} \frac{b^2}{2} \sqrt{\nu^2 b^4 + 4\sigma_i^2 (b^2 - l_m^2) - \nu b^2} / \sqrt{2 (b^2 - l_m^2)} & (l_m < b < \sqrt{2/1 + \nu l_m}) \\ \frac{b^2}{2} \sqrt{b^2 - \sqrt{b^2 - 4\sigma_i^2 (l_m^2 - \nu b^2)}} / \sqrt{2 (l_m^2 - \nu b^2)} & (b \geq \sqrt{2/1 + \nu l_m}) \end{cases}$$

(11)

In this study, the position of the key stratum and ultimate hanging arch area was determined layer by layer from the bottom to the top according to Formulas (5) and (11).
2.3 Project examples

This study selected the 42 209 short-wall continuous mining working face in the Yujialiang Mine as an example. This area was found to have a simple geological structure, and was without faults, folds, etc. The working face had a dip angle of 1° to 3°, and buried depth of 60 to 140 m. The thickness of the loose layer was 40 to 120 m, and the average thickness was 80 m. The thickness of the overlying bedrock was 20 m. The in situ stress was dominated by vertical stress, and the original rock stress was approximately 1.7 MPa. The immediate roof of the working face was mudstone, which displayed a horizontal and undulated bedding and good integrity, with a thickness of 6.1 m. The main roof consisted of siltstone, which displayed a horizontal and wavelike bedding, with a thickness of 3.69 m. The parameters of the overlying rock strata on this working face are shown in Table 1.

The calculation was implemented by regarding the Siltstone 1 as the 1st layer, and the following calculations could be obtained according to Formula (3): \( q_{\text{siltstone 1}} = 0.087 \) MPa, \( q_{\text{mudstone interbedding}} = 0.12 \) MPa, \( q_{\text{siltstone 2}} = 0.06 \) MPa. Due to the fact that \( q_{\text{siltstone 1}} < q_{\text{mudstone interbedding}} > q_{\text{siltstone 2}} \), it was determined that the Siltstone 1 (main roof) and Siltstone 2 were the Key Stratum 1 and Key Stratum 2, respectively. The ultimate hanging arch distance \( (a) \) and ultimate hanging arch area \( (S) \) of the key strata were calculated using Formulas (10) and (11), as shown in Tables 2 and 3, along with Figures 2 and 3. The ultimate hanging arch area of the Key Stratum 2 was found to be smaller than that of the Key Stratum 1 (main roof). Furthermore, the Key Stratum 1 was observed to move with the Key Stratum 2 by following a consistent movement step. Therefore, the Key Strata 1 and 2, along with the middle rock strata, jointly formed a composite key stratum.

In accordance with the above-mentioned figures and tables, the easily caving stratum on the examined working face was the Key Stratum 2. Its minimum ultimate caving area was determined to be 949.44 m², and its minimum ultimate hanging arch distance was 24.7 m. The maximum ultimate hanging arch distance of the composite key stratum was 72.57 m, and the maximum ultimate area was 6906.26 m². At the same time, the ultimate hanging arch distance was observed to gradually decrease with the increases in the mining width, and finally tended to be a constant when the mining width was larger than the lowest ultimate hanging arch distance. Meanwhile, the ultimate area was observed to constantly increase with the increases in the mining width.

3 SIMILAR MATERIAL SIMULATION STUDY OF THE MOVEMENT RULE OF THE OVERLYING ROCK USING A SHORT-WALL CONTINUOUS MINING METHOD

Similar material simulation has been a widely used research method in the engineering field, and is able to simulate specific projects relatively accurately. The success of a similar material simulation experiment depends on the satisfaction degree between the model and the actual conditions. In a similar material simulation experiment, the corresponding results of the prototype structure could be deduced from the model experiment results only if the model and the actual conditions remained similar. A similar material simulation experiment used materials that were similar in physical properties to the actual rock strata. The physical model was created according to a certain proportion, and the mining work was simulated. The deformation, failure, and rock strata movement of the model were observed and recorded.

3.1 Similar material simulation study

The three theorems of similarity were the theoretical basis for similar material simulation experiments, which expressed the basic properties of similar phenomena, and illuminated the
necessary conditions for the similarity of physical phenomena. In similar material simulation experiments, the similarity ratios that were set included geometric similarity ratio, bulk density similarity ratio and strength similarity ratio. Notably, all of the similarity ratios commonly followed the equations, but were not subjectively set:

\[
C_l = \frac{l_p}{l_m} \quad (12)
\]

\[
C_\gamma = \frac{\gamma_p}{\gamma_m} \quad (13)
\]

\[
\frac{C_\sigma}{C_l \cdot C_\gamma} = 1 \quad (14)
\]

where \(C_l\), \(C_\gamma\), and \(C_\sigma\) are the geometric similarity ratio, the bulk density similarity ratio and the strength similarity ratio, respectively, \(l_p\) denotes prototype size, \(l_m\) represents model size, \(\gamma_p\) is prototype bulk density, and \(\gamma_m\) denotes model bulk density.

By adopting a method of two-dimensional similar material simulation, and regarding the short-wall continuous mining working face in Yujialiang Mine 42 209 as the geological prototype, this study simulated the movement rule of the roof, along with the strain distribution state in front of the coal wall along the coal seam strike after mining, using a short-wall continuous mining and full-caving method. The experimental bench had a length, width, and height of 1.90 m × 0.22 m × 1.80 m, respectively. The model design adopted the geometric similarity ratio of \(C_l = 60\), the bulk density similarity ratio of \(C_\gamma = 1.6\), and the strength similarity ratio of \(C_\sigma = 96\).

According to the basic data of the prototype stratum, the model was laid with sand as the aggregate, and with gypsum and calcium carbonate as the bonding materials. The ratio of similar materials was selected on the basis of the compressive strength of the laid rock strata. The respective amounts of sand, gypsum, calcium carbonate, and water were then calculated based on the size of the model and the thickness of the rock stratum. The amount of each layered material in the model could be calculated by the following formula:

\[
Q = l \times j \times f \times \rho_m \times k
\]

where \(l\) is the length of the model, \(m; j\) represents the width of the model, \(m; f\) denotes the layer thickness of the model, \(m; \rho_m\) is the mass density of similar materials, kg/m³; and \(k\) is the necessary conditions for the similarity of physical phenomena. In similar material simulation experiments, the similarity ratios that were set included geometric similarity ratio, bulk density similarity ratio and strength similarity ratio. Notably, all of the similarity ratios commonly followed the equations, but were not subjectively set:

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\]

Where \(l\) is the length of the model, \(m; j\) represents the width of the model, \(m; f\) denotes the layer thickness of the model, \(m; \rho_m\) is the mass density of similar materials, kg/m³; and \(k\) is the
is the material loss coefficient. The geomechanical properties of the coal rock are shown in Table 4, and the model proportioning material and laying hierarchy are detailed in Table 5.

3.2 | Construction of the model

In this study, the dimension line of coal rock strata was drawn, and the required quantities of material for each layer were successively weighed out. A mixer was used to mix the material, and then the materials were laid out in sequence until the layering was completed. In order to guarantee the homogeneity of the material and achieve a good caving effect, the simulated layered strata had a minimum thickness of no <0.01 m. The material was laid out in layers when the thickness was larger than 0.03 m, and layers of mica powder were evenly laid between the layers. A stress sensor was buried in the coal seam according to the designed location during the laying process. For the 0.40 m surface soil layer, which had not been laid due to the limited model size, a simulation was made using the applied load.

A rock strata displacement monitoring grid with a length × width of 0.10 m × 0.10 m was arranged on the surface of the model. The subsidence and displacement of the rock strata were observed using a total station. Stress measurement points were arranged in the middle of the coal strata intervals of 0.10 m, and these measurement points were numbered No. 1 to No. 13 starting from the left along the coal seam strike. This was completed in order to measure the changes in the stress of the surrounding rock during the excavation. The layout of the measurement points and the experimental model are shown in Figure 4.

3.3 | Plan and simulation results of the experiment

In order to avoid the influence of the boundary effect, the excavation of the model was initiated from a location 0.20 m

| Stratum number | Bulk modulus (MPa) | Shear modulus (MPa) | Cohesion (MPa) | Friction angle (°) | Tensile strength (MPa) | Thickness (m) |
|----------------|--------------------|---------------------|----------------|-------------------|-----------------------|--------------|
| Roof 5         | 70                 | 0.2                 | 0.03           | 26                | 0.01                  | 80           |
| Roof 4         | 4500               | 2.8                 | 1.4            | 32                | 10.8                  | 6            |
| Roof 3         | 4000               | 2.4                 | 1.2            | 33                | 7.2                   | 3            |
| Roof 2         | 1200               | 0.8                 | 1.1            | 35                | 10.7                  | 4            |
| Roof 1         | 1000               | 0.6                 | 0.8            | 30                | 6.1                   | 6            |
| Coal           | 500                | 0.28                | 0.6            | 25                | 1.1                   | 3.45         |
| Floor 2        | 6000               | 3.9                 | 1.5            | 35                | 2.7                   | 10           |
| Floor 1        | 6800               | 3.4                 | 2.0            | 38                | 6.1                   | 10           |
The roof movement rule was analyzed at the following phases: initial excavation period of the working face, first caving of the immediate roof, first breaking of the combination of the key stratum of the overlying strata, and the periodic weighting of the roof.

After the excavation of the working face, and prior to the first caving of the immediate roof (excavation distance < 0.50 m), the stress peak of the surrounding rock was found basically at the location of the coal wall. The sensor only began to display changes in the stress conditions when the working face had advanced to a location approximately 1 to 0.02 m from the stress sensor. The stress increased gradually with the progress of the excavation. The maximum stress peak was determined to be 0.068 MPa (in situ stress of the model: 0.0175 MPa), and the stress concentration factor was 3.88. The stress change curve at Measurement Point 2 (0.20 m from the open-off cut) is shown in Figure 5.

As can be seen in Figure 5, the short-wall continuous mining working face displayed a complete coal seam structure without a plastic failure area during the initial production period. The coal body was in an elastic compression state, and the peak stress of the surrounding rock had occurred near the coal wall without shifting toward the deeper sections of the coal body.

It was observed that the mudstone R2 (the immediate roof) had undergone a bed separation when the working face had advanced to the 0.40 m location (the prototype was 24 m in length). As the working face continued to advance toward the 0.50 m location, the bed separation became significantly larger, and a longitudinal crack occurred on the coal wall at both ends and in the middle part of the roof within the goaf. At that time, the siltstone R4 (the main roof) also began to move and incurred an obvious separation phenomenon. It was also observed that the end section experienced a longitudinal crack, as shown in Figure 6.

The stress sensor on Measurement Point 5 at that point in time began to display stress changes when it was 0.02 to 0.03 m from the coal wall, which indicated that the stress peak had already shifted toward the deeper locations of the coal wall compared with the preceding measurement points. The stress peak reached 0.065 MPa, and the stress concentration factor was 3.71. Figure 7 shows the relationship between the stress changes of the first five measurement points, along with the excavation distances. It can be seen from the figure that the peak value of the supporting pressure at the location of coal wall began to shift toward the deeper sections of the coal body with the increases in distances of the unsupported roof.

It was found that when the mining distance of the working face increased to 0.53 m, the siltstone R2 (the immediate roof) underwent its first caving, and the height from the caving rock stratum to the coal seam was 0.064 m. The mudstone...
R3 and siltstone R4 above the roof of the coal seam incurred bed separation, and developed a longitudinal crack, as shown in Figure 8.

The siltstone R3 (the immediate roof) experienced its first caving when the working face advanced to the 0.60 m location. Only the lower layer of the siltstone R4 (the main roof) was observed to have dropped down due to the layered laying of the siltstone R4. However, there was still a relationship of transfer force observed along its horizontal direction, as shown in Figure 9.

The initial display of the stress sensor at Measurement Point 6 was determined to be 0.03 MPa when the working face had advanced to the 0.70 m location. Then, the excavation continued until the stress peak reached 0.07 MPa before caving occurred in the immediate roof area. The distance from the peak point to the working face was between 0.05 and 0.07 m when the stress peak was reached, which indicated that the 3 m location of the advanced working face in the coal body was the point at which the stress of the surrounding rock had reached the maximal value. After that, sensor No. 6 was observed to fluctuate within a range of 0.01 to 0.02 MPa, and then the pressure became stabilized at 0.02 MPa. It was determined that the siltstone of the immediate roof had fractured at that time. The stress change curve of Measurement Point 6 is shown in Figure 10.

As the working face continued to advance, the cracks in the upper bed separation further developed along the

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FIGURE 4  Similar material simulation experiment model

FIGURE 5  Stress change curve for Measurement Point 2

FIGURE 6  Bed separation and crack in the middle and end sections of the overlying rock when the working face had advanced to the 0.50 m location
When the working face had advanced to the 0.90 m location, the siltstone R4 (the main roof) caved together with the upper soft rock strata, whereas the siltstone R6 still maintained a hanging arch state, as detailed in Figure 11. After the lower layer of the siltstone R4 caves, the immediate roof behind the coal wall was still in an intact state and formed a hanging arch structure. The stress concentration degree was found to be high within a certain scope within the coal wall. It was observed that a stress peak had occurred at the location of the No. 8 sensor in the front of the coal wall. The stress peak reached 0.09 MPa when the distance to the coal wall was 0.10 to 0.12 m, and the stress concentration factor reached 5.14.

It was found that the upper bed separation and longitudinal crack were extremely developed, and the siltstone R6 also had started to bend and sink. According to this study's measurements, the caving height of the mudstone R2 was 0.07 m at that time, and the caving height of the mudstone R3 was 0.069 m. The siltstone R4 moved at the same time as the siltstone interbedding R5, and the caving height was also determined to be 0.064 m. The height between the siltstone interbedding R5 and sandstone R6 was 0.06 m. When the working face had advanced to the 0.94 m location, the siltstone R6 underwent cracking, as shown in Figure 12. Meanwhile, the first periodic weighting phenomenon occurred in the immediate roof area. It could be seen that the composite key stratum above the working face displayed a batched caving phenomenon. However, Key Stratum 1 (siltstone R4) displayed no major differences from Key Stratum 2 (siltstone R6) in the caving span (model 0.04 m). Therefore, it was basically assumed that Key Stratum 1 and 2 had almost synchronous motion with the middle soft rock strata. The key strata were closely related to the lower continuous mining working face, which played an important role in controlling the motion of the overlying rock strata within the goaf, and also in managing the roof areas when the full-caving method was used.

When the working face had advanced to the 1.11 m location, secondary periodic caving of the main roof occurred. The caving rock strata still maintained a relationship of transfer force along the horizontal direction. The cracks of the bed separation developed to a certain extent along the advancement direction of the working face during the mining processes. Sensor No. 9, which was located at the 1.10 m location of the coal body, began to exhibit a stress manifestation after the first periodic weighting, and the stress value was observed to constantly increase with the forward advancement of the working face. However, when compared with the stress measured by sensor No. 7 during the previous first caving of the main roof, it was determined that its pressure peak and weighting manifestation degree had become decreased to a certain extent. Under the effects of the overlying rock strata and transfer rock beam, it was found that the stress manifestation still occurred at approximately the 0.10 m location inside the coal body. The simulation sections and conditions of the overlying rock strata following the completion of this study's experiment are shown in Figure 13.

The working face started to advance from the open-off cut. When the advance distance was 0.4 m (the prototype was 24 m), the mudstone R2 (the immediate roof) underwent bed separation. Following the advance distance was 0.53 m (Figure 8), R2 occurred caving, R3 and R4 incurred bed separation. As the working face continued to
advance, the bed separation of the main roof constantly expanded and developed to the upper rock strata. The siltstone R4 (the main roof) experienced its first caving when the working face advanced to 0.90 m. With the advance of the working face, the bed separation of the overlying strata continued to expand upward, and the upper strata began to bend and sink. When the working face had advanced to 1.11 m, the bed separation expanded upward to the middle position of the model, and the bed separation the lower rock strata began to close under the action of the upper rock strata (Figure 12). After the immediate roof away from the working face was in contact with the floor, the lower rock strata were in a relatively stable state, and this stable state moved forward with the advance of the working face. When the working face advanced to 1.5 m, that is to say, the simulated working face had been fully advanced (Figure 13), the bed separation of the overlying strata continued to expand to the upper part of the model. The deformation and failure of overlying rock strata reached the maximum degree, and there was obvious bending and subsidence in the middle of the model.

As shown in Figure 14, the vertical displacement of the overlying strata increased gradually with the continuous advance of the working face, and the maximum vertical displacement was located in the middle of the working face advance length. When the working face advance distance was 0.7 m, the maximum vertical displacement of the first row measurement point was 0.0705 m, and only a small displacement (the maximum value was 0.006 m) occurred in the fourth row measuring point, which indicated that the mining had little effect on the rock strata above the main roof. When the working face advance distance was 1.3 m (the stopping line position of the similar simulation working face), the vertical displacement of the overlying rock strata reached the maximum value. The maximum vertical displacements of the first row and seventh row measuring points were 0.0716 and 0.0078 m, respectively. Through the similar material simulation, it could be seen that the roof caving occurs periodically with the advance of the working face and the deformation of the lower rock strata was large. The effect of mining on the overlying rock strata continued to the upper part of the model.

3.4 Analysis of the results

In accordance with this study's similar material simulation experiment, and under the condition of no artificial interventions on the roof areas, the rock strata motion rules which were considered according to the natural caving and stress distribution in the coal bodies were as follows:

1. The mudstone R2 directly overlying the coal seam first experienced bed separation and caving, and its first caving span was 31.8 m after the geometric conversion. The mudstone R3 became separated from the upper rock strata after the caving of the mudstone R2, and that separation further increased with the mining scope of the working face. The caving phenomenon continued, with the first caving span measuring 36 m.
2. The key stratum of the overlying rock consisted of the siltstone R4, mudstone interbedding R5, and siltstone R6. The experimental results showed that its motion damage rule was as follows: First, the R4 and R5 jointly experienced subsidence and bending until the fracture occurred, and the fracture span was 42 m. Then, the R6 quickly displayed bed separation and fracturing over a short period of time, and the fracturing span was 44.4 m. Therefore, it was basically believed that the key stratum which was jointly composed of the R4, R5, and R6 underwent simultaneous motion.

**FIGURE 9** Secondary caving of the immediate roof area when the working face had advanced to the 0.60 m location

**FIGURE 10** Relationship between the stress of Measurement Point 6 and the advancement distance of the working face
3. The surface topsoil layer had developed a large number of cracks after the fracturing of the key stratum, and a large scope of layer separation phenomenon had occurred within the scope of 42 m above the coal seam. Also, the surface displayed an obvious subsidence phenomenon.

4. With the advancement of the working face, the influence scope of the advance supporting pressure during the first caving stage of the immediate roof was between 3 and 5 m, and the stress concentration factor was between 2 and 4. The influence scope of the advance supporting pressure during the first fracture stage of the key stratum was from 6 to 9 m, and the stress concentration factor was from 5 to 7. This study’s experiment included five periodic weightings which began from the onset of the excavation to its completion, and the experimental phenomena were basically similar. Also, according to the statistics, the periodic weighting length was between 1/4 and 1/3 of the first weighting length.

In accordance with the results of this study’s similar material simulation experiment, the statistics of the motion span, along with the ultimate area of the overlying rock on the roof areas, are shown in Table 6.
NUMERICAL SIMULATION STUDY OF THE MIGRATION AND STRESS FIELD EVOLUTION RULE OF THE OVERLYING ROCK USING A SHORT-WALL CONTINUOUS MINING AND FULL-CAVING METHOD

4.1 Model building and excavation scheme

In this research study, a numerical model was built by regarding the geological conditions of a short-wall working face in Yujialiang Mine 42 209 as the background. The aim was to examine the migration rule and stress distribution state of the roof area of the short-wall working face during the mining process. The simulated mining height was 3.45 m, and neither side of the working face was mined. The three-dimensional model was 190 × 180 × 120 (x × y × z) m. In total, the model consisted of 71,136 units and 79,712 nodes. The Mohr-Coulomb elastoplastic constitutive relation was used to determine the mechanical behavior of rock strata in the numerical calculation model. The initial stress field along the vertical direction was generated according to the self-weight of the overlying rock. The initial stress field along the horizontal direction was generated according to Formula (16), and the lateral pressure coefficient was \( \lambda = 0.5 \). The lithologic parameters of the model are shown in Table 4. The lateral boundary of model was limited in the horizontal displacement and the bottom limits of the vertical displacement.\(^4\) Also, the self-weight stress of the overlying rock was applied to the upper section as follows:

\[
\sigma_z = \sigma_y = \lambda \sigma_z
\]

(16)

The mining sequence of the short-wall working face was the same as an actual designed mining sequence. For example, blocks 1 to 9, coal pillars in the left transportation roadway, blocks 10 to 13, and coal pillars in the right-hand transportation roadway. The stress and displacement of surrounding rock in the stope were studied with the mining of the short-wall working face. The schematic diagram for the established model is shown in Figure 15.

4.2 Analysis of the results

As can be seen in Figure 16, obvious stress concentrations were caused at the lower ends of blocks 4, 5, and 6 following the mining of blocks 1, 2, and 3. The maximum concentrated stress was determined to be 7.6 MPa, the initial rock stress was 3 MPa, and the stress concentration factor was 2.53. The mining of blocks 1, 2, and 3 was found to basically have no influence on blocks 7, 8, and 9, or on the coal pillars in the upper transportation roadway. Also, the stress concentration phenomenon of the coal body at the right side was not obvious. On the OYZ profile, the influence scope of the displacement following the mining of blocks 1, 2, and 3 was observed to be basically within the scope of 30 m of blocks 4, 5, and 6. The maximum subsidence displacement within the goaf was 0.048 m, and the subsidence displacements at the left and right ends of blocks 4, 5, and 6 were 0.005 m and 0.03 m, respectively.
As shown in Figure 17, obvious stress concentrations occurred at the lower ends of blocks 7, 8, and 9 following the mining of blocks 4, 5, and 6. The maximum concentrated stress was determined to reach 10 MPa, and the stress concentration factor was 3.33. The mining of blocks 1 to 6 was found to have basically no influence on the coal pillars in the upper transportation roadway. Also, the stress concentration phenomenon of the coal body at the right side was not obvious. On the OYZ profile, the displacement influence scope following the mining of blocks 4, 5, and 6 was basically within the scope of 25 m at the left sides of blocks 7, 8, and 9. The maximum subsidence displacement within the goaf was 0.17 m, and the subsidence displacements at the left and right ends of blocks 7, 8, and 9 were 0.02 m and 0.12 m, respectively.

It was found that an obvious stress concentration phenomenon had occurred in the middle of the residual coal pillars and central coal pillars of the transportation roadway of blocks 7, 8, and 9 following the mining of blocks 7, 8, and 9, as detailed in Figure 18. The maximum concentrated stress reached 12.8 MPa, and the stress concentration factor was 4.27. The mining of blocks 1 to 9 was determined to have already influenced the coal pillars in the upper transportation roadway.

On the OYZ profile, the displacement influence scope following the mining of blocks 7, 8, and 9 was determined to be basically within the scope of 20 m at the left sides of blocks 7, 8, and 9. Therefore, it was observed that the displacement influence area had gradually shrunk with the gradual increases in the mining area. The maximum subsidence displacement in the goaf reached 0.45 m, whereas the average subsidence displacements of the residual and transportation roadway coal pillars were found to be 0.17 m and 0.075 m, respectively.

As detailed in Figure 19, the left side of the goaf had the highest stress concentration degree following the mining of the coal pillars in the left transportation roadway. The maximum concentrated stress was 16.3 MPa, and the stress concentration factor was 5.43. It was observed that the concentrated stress at the right side of the goaf was approximately 14 MPa, and the stress concentration factor was 4.67. The stress concentration factor at the upper and lower sides of the goaf was approximately 4.7.

Following the mining of the coal pillars in the left transportation roadway, the influence scope of the displacements on the left and right sides along the X direction was further reduced to 3 m on both sides of the goaf, and the maximum subsidence displacement within the goaf reached 1.70 m.
**FIGURE 17** Stress and displacement nephograms following the mining of blocks 4, 5 and 6. (A) Stress nephogram on the OXY Profile ($z = 4m$) following the mining of blocks 4, 5 and 6. (B) Displacement nephogram on the OYZ Profile ($X = 71 m$; middle part of the goaf) following the mining of blocks 4, 5 and 6.

**FIGURE 18** Stress and displacement nephograms following the mining of blocks 7, 8 and 9. (A) Stress nephogram on the OXY Profile ($z = 4m$) following the mining of blocks 7, 8 and 9. (B) Displacement nephogram on the OYZ Profile ($X = 71 m$; middle part of the goaf) following the mining of blocks 7, 8 and 9.
As can be seen in Figure 20, an obvious stress concentration occurred in the middle lower part of block 12 following the mining of blocks 10 and 11. The maximum concentrated stress reached 28.9 MPa, and the stress concentration factor was 9.63. The left blocks of the coal pillars in the transportation roadway also underwent a stress concentration phenomenon.

On the OYZ profile, the displacement influence scope was determined to be basically within 30 m on the left sides of blocks 10 and 11. The subsidence displacement was observed to be large and had developed from 40 to 1.00 m from left to right. The maximum subsidence displacement within the goaf had reached up to 1.79 m.

An obvious stress concentration had occurred in the middle part of the residual coal and transportation roadway coal pillars following the mining of blocks 12 and 13, as shown in Figure 21. The maximum concentrated stress reached 28.6 MPa, and the stress concentration factor was determined to be 9.53.

On the OYZ profile, the displacement influence scope was observed to be basically within 40 m on the left sides of blocks 12 and 13. It was found that the subsidence displacement was large and had developed from 0.02 to 1.00 m from the left to the right. The maximum subsidence displacement within the goaf reached up to 1.82 m.

In accordance with Figure 22, it could be seen that the maximum stress concentration had occurred at the left side of the area following the completion of the mining of the coal seam blocks in the area. The maximum concentrated stress was determined to be 25.7 MPa, and the stress concentration factor was 8.57. The maximum concentrated stress of the right coal mass was approximately 18 MPa, and the stress concentration factor was 6.

In this study, it was found that the displacement influence scope had been further reduced following the total mining of the coal bodies. Moreover, the external displacement of the goaf was basically zero, and the maximum subsidence displacement within the goaf had reached up to 1.80 m.

The mining sequence in the entire mining area was from the bottom left corner to the upper right corner. Therefore, this study selected a stress monitoring point from the upper right corner of the goaf. Its coordinates were x, y, z (140, 136, 4), and it was located in the middle of the rightmost block of the coal pillars in the transportation roadway, as shown in Figure 23. The stress change curve for this monitoring point with the increases in the mining area is detailed in Figure 24.

In this research study, it was determined that the selected monitoring point had experienced no major stress changes.
FIGURE 20  Stress and displacement nephograms following the mining of blocks 10 and 11. (A) Stress nephogram on the OXY Profile (z = 4 m) following the mining of blocks 10 and 11. (B) Displacement nephogram on the OYZ Profile (z = 122 m) following the mining of blocks 10 and 11.

FIGURE 21  Stress and displacement nephograms following the mining of blocks 12 and 13. (A) Stress nephogram on the OXY Profile (z = 4 m) following the mining of blocks 12 and 13. (B) Displacement nephogram on the OYZ Profile (X = 122 m) following the mining of blocks 12 and 13.
These findings indicated that the mining processes on the left side of the area had only a minor influence on the stress of the surrounding rock on the right side. It was observed that during the mining of the coal pillars on the left side of the transportation roadway, the stress of the monitoring point had begun to display a sharp increase. The maximum stress had reached up to 15 MPa, and the stress concentration factor was up to 5. The selected monitoring point was within the scope of the goaf following the mining of the coal pillars in the right transportation roadway. Therefore, the stress of the surrounding rock had displayed a sharp decrease.

As can be seen in Figure 25, with the continuous mining of the blocks, the maximum subsidence displacement position of the roof continuously developed in the direction of the working face advance. The maximum subsidence displacements of the roof caused by the mining of blocks 1-9 were 0.048 m, 0.17 m, and 0.45 m, respectively. As the working face advanced continuously, the roof subsidence displacement gradually increased. The roof subsidence displacement increased to 1.7 m after the coal pillars in the left transportation roadway were mined. Following the mining of blocks 10-11, the maximum subsidence displacement of the roof was 1.79 m. The roof subsidence displacement caused by the mining of blocks 12-13 reached the maximum value of 1.82 m. When the coal pillars in the right transportation roadway were mined, the maximum subsidence displacement of the roof remained at 1.8 m. This, combined with Figure 24, showed that there was a certain positive correlation between the roof subsidence displacement and stress of the monitoring points.

### CONCLUSIONS

1. This study proposed a judgment method for determining the key stratum of working faces which uses short-wall
continuous mining and full-caving methods based on elastic plate theory. As a result, the key strata 1 and 2 of the short-wall continuous mining working face in Yujialiang Mine 42 209 were determined to be Siltstone 1 (the main roof), and Siltstone 2, respectively. Therefore, the key strata 1 and 2 and the soft rock strata in the middle were determined to be a jointly composed key stratum.

2. This similar material simulation experiment verified that the composite key stratum of the roof in the Yujialiang Mine consisted of two layers of siltstone and a soft rock stratum in the middle. During the process of the simulation, the stress concentration factor gradually increased from 2.86 to 5.14, and shifted toward to the deeper sections of the coal wall until the rock strata caved in. The immediate roof caved in by layers, and the caving spans were 31.8 m and 36 m. Meanwhile, the first fracture span of the key stratum was 45 m approximately, which proves that the key stratum was almost in a synchronous movement phase with the immediate roof.

3. This study analyzed the numerical simulation results of the stress and displacement evolution processes of the overlying rock during the advancement of the short-wall continuous mining working face. When the mining of blocks 1 to 9 and the coal pillars at the left side of the transportation roadway was completed, the peak stress at the left side of the mining area was determined in the coal wall within 5 m, and the coal body on the right side had a core elastic area. The coal pillars were observed to be stable, and the remaining blocks could be continually mined without preserving the coal pillars in these sections. Until the mining had been completed, the maximum concentrated stress was observed on the left side of this mined area, which had no influence on the mining safety. The displacement influenced zone was gradually reduced during the mining process, and the goaf sinking displacement increased from 0.048 m and then stabilized at 1.80 m. These observations verified the feasibility of the short-wall continuous mining and full-caving method in this mining area.

4. The proposed full-caving short-wall continuous mining method optimized the mining process, avoided the construction of coal pillars in and in-between mining sections. As a result, it created a continuous mining method without encountering pillars during the backstopping process. Meanwhile, the regular caving behavior of the roof during the mining processes not only avoided the occurrence of large hanging roof arches, but also improved the recovery rate. Therefore, this study can be a potential safe and highly efficient instruction of boundary coal and residual coal pillars mining. Ultimately, this study provides important references for improving the resource recovery rates in Shendong mine area, and even in similar mining areas throughout China.

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CONFLICTS OF INTEREST

The authors declare no conflict of interest.

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