Research Article

Study of the Stability Control of Rock Surrounding Longwall Recovery Roadways in Shallow Seams

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The rock pressure appearance of longwall faces in shallow seams is generally violent, and roofs and supports are susceptible to damage during equipment extraction. Stability control of the rock surrounding longwall recovery roadways allows safe and rapid equipment extraction. Herein, via theoretical analysis, numerical simulations, and field observations, the stability control of the rock surrounding recovery roadways is studied to ensure the release of the accumulated rock pressure on the roof, the working resistance of the supports and the reasonableness of the recovery roadway support design. Pressure-relief technology is introduced to release the accumulated rock pressure before equipment extraction, and a discriminative approach is proposed to determine the breaking and articulated forms of key strata and broken blocks, respectively. On this basis, mechanical models of roof instability are established based on four key stratum structures in the overburden of shallow seams. Methods for calculating a reasonable working resistance for supports are discussed. Finally, Liangshuijing Coal Mine and Fengjiata Coal Mine are taken as research objects to evaluate the roof stability of recovery roadways based on observations of weighting characteristics. The support working resistances and reasonable recovery roadway widths under three key stratum structures are determined. Considering the time effect of plastic zone development, the support design of recovery roadways is optimized. FLAC2D software simulates the surrounding rock control effect of two support designs, and roof subsidence curves are obtained. The results show that the key to equipment extraction in shallow seams is to ensure that supports have reasonable working resistances and to improve the support of recovery roadways. The results provide a reference for the selection and extraction of supports in shallow seam faces.

1. Introduction

Coal seams with shallow mining depths, thin bedrock, and thick unconsolidated strata are usually defined as shallow seams [1, 2]. Only a caving zone and a fracture zone exist above the longwall gob of shallow seams [3–6]. Many production practices show that the rock pressure in a shallow seam face is extremely high, and the roof is susceptible to bench convergence [7–11]. The plastic zone in the surrounding rock develops fully during the extraction of equipment. It is easy to break, rotate, and collapse the roof and crush the supports, and such events are likely to occur, as shown in Figure 1 [12]. For example, the roofs and supports in the 42107 panel of the Liangshuijing Coal Mine and

the 1202 panel of the Fengjiata Coal Mine were damaged during the extraction of equipment. Mining cracks can directly reach the surface, and air, water, and sediment can enter the recovery roadway through cracks, which seriously affects the extraction of equipment. Therefore, the key to realizing safe and efficient production in mines is to study the stability control of the rock surrounding longwall recovery roadways [13].

The extraction technologies for longwall faces include shearer-driven roadway technology and predriven roadway technology. The technology of predriven roadways has obvious shortcomings, such as relatively large work quantities, high maintenance costs, and a high likelihood of roof collapse when crossing the predriven roadway [14]. At

FIGURE 1: Roof and support damage during the extraction of equipment.

present, this technology is mainly used in mines with simple geological conditions such as the Shendong mining area. Most mines in China still apply shearer-driven roadway technology. The stability control of rock surrounding recovery roadways in shallow seams has been studied deeply at home and abroad. Numerous scholars have systematically studied support design for recovery roadways and the relationship between the support system, strata movement, and ground subsidence [14–16]. Some researchers have also proposed controlling the stability of rock surrounding recovery roadways by determining the applicability of an adapting roadway, evaluating potential accidents, and optimizing the locations of mesh installation [17, 18]. Other scholars have also studied the influence of mining-induced stress, the breaking locations of main roofs, the buried depth of the working face, the nature of rock strata, and mining heights on the stability of rock surrounding recovery roadways [19, 20]. However, the above research is mainly aimed at the stability control of rock surrounding predriven longwall recovery roadways in shallow seams. Few studies have focused on the stability control of rock surrounding shearer-driven longwall recovery roadways.

The key stratum structures of overlying strata in shallow seams can be divided into four types, as shown in Figure 2 [21]. There are significant differences in the breaking and migration laws of different key stratum structures. The overburden load of the recovery roadway is supported by the “coal wall-supporting structure-hydraulic support-gangue in gob” structure. Optimizing the bolting parameters of the recovery roadway can improve the supporting efficiency and reduce the development time of the plastic zone in the surrounding rock. Support with a reasonable working resistance can effectively protect the recovery roadway when the roof is unstable. Therefore, the bolting parameters of the recovery roadway and the reasonable working resistance of the support should be studied to realize the stability control of the surrounding rock.

This paper mainly studies the stability of the surrounding rock in shearer-driven recovery roadways. First, pressure-relief technology and its applicable conditions are introduced. Pressure-relief technology is an effective means of

releasing the accumulated rock pressure in the roof before the extraction of equipment. Subsequently, mechanical models of roof instability are established for different key stratum structures. The formulas for calculating the reasonable working resistances of supports under different roof conditions are deduced theoretically. The Liangshuijing Coal Mine and Fengjiata Coal Mine are taken as research objects. According to the actual geological conditions of each working face, the reasonable working resistance of the support and the reasonable width of the recovery roadway are calculated. Based on the above results, a numerical simulation method is used to verify the surrounding rock control effect of the improved support design for the recovery roadway. The research results are in line with the field practice, which realizes the safe and rapid extraction of the supports.

2. Pressure-Relief Technology

In the mid-1990s, the “Key Strata Theory” of strata control was put forward [22]. This theory provides in-depth information about the progressive caving behavior of strata and its impacts on longwall operations. According to this theory, the stratum that controls the movement of all or part of the overburden is defined as the key stratum (KS); that is, when the KS breaks, all or part of the overburden above the KS will subside simultaneously. To be more specific, the former is defined as the primary key stratum (PKS), whereas the latter is defined as a subordinate key stratum (SKS). There may be several subordinate key strata in the overburden of the stope, but the PKS is unique and located above all the subordinate key strata. Generally, the SKS closest to the stope in the overburden is called SKS1, and the other subordinate key strata are called SKS2, SKS3, etc. from bottom to top until reaching the PKS. The rock strata with synchronous subsidence and deformation with each SKS are named weak stratum 1 (WS1), weak stratum 2 (WS2), etc., and the strata controlled by the PKS are called load strata. The stable state of SKS1 directly determines whether periodic weighting occurs in the stope. Therefore, SKS1 should be kept as stable as possible during the extraction of

FIGURE 2: Types of key stratum structures in shallow seams: (a) thick and hard single-key stratum structure; (b) single-key stratum structure of a compound group; (c) multikey strata structure; (d) single-key stratum structure of an over-seam extracted group.

equipment. In general, the recovery roadway must be arranged according to the preset terminal line to maximize the exploitation of coal resources. If the law of weighting is not considered, periodic weighting may be necessary during the extraction of equipment, as shown in Figures 3(a)–3(c). Pressure-relief technology stops the advancement of the working face (usually 10 hours) before laying out the recovery roadway, which promotes the occurrence of periodic weighting according to fracture development. The working face continues to advance after periodic weighting. After the working face passes the weighting area, the recovery roadway can be arranged. The width of the weighting area is usually 2 to 3 times that of the regular circulation footage according to the statistical results of monitoring data of the rock pressure in Yushenfu Mining Area [16]. At this time, SKS1 can be ensured to be in as stable state as possible during the extraction of equipment, as shown in Figure 3(d).

Pressure-relief technology has certain applications. The technology should be applied reasonably according to roof conditions and the law of periodic weighting. For example, periodic weighting will not occur even if mining stops for a long time when the hanging length of SKS1 is less than 0.6 times the breaking interval or if there is a thick and hard roof above the working face. According to practical experience at Halagou Coal Mine and Xinhe Coal Mine, deep-hole presplitting blasting technology can be used to cut off the hanging roof [23, 24].

3. Determining the Reasonable Working Resistance of the Support

The shearer cuts the coal wall to arrange the recovery roadway after the rock pressure accumulated in the roof is

released. Formula (1) is the basis for evaluating the stability of SKS1 above the recovery roadway. If the working resistance of the support is low, then roof and support damage may occur due to the instability of SKS1:

$$l_1 = d_1 + d_2 + d_3 \geq kL_1, \quad (1)$$

where l_1 is the hanging length of SKS1 (m); d_1 is the width of the recovery roadway (m); d_2 is the length of the hydraulic support (m); d_3 is the reserve safe distance, which is generally 3 times the regular circulation footage according to engineering experience (m); L_1 is the periodic weighting interval of SKS1 (m); and k is the safety factor that ranges from 0.4 to 0.6. In this paper, the default value of k is 0.6.

Similarly, when the key stratum other than SKS1 in the overburden of the slope with a multikey strata structure is considered in formula (2), this stratum may break during the extraction of equipment. It is generally believed that SKS1 will break synchronously after the upper key stratum (UKS) is broken. Therefore, the stability of the UKS should be considered before the layout of the recovery roadway in the multikey strata structure slope:

$$l_m = d_1 + d_2 + d_3 + d_4 \geq kL_m, \quad (2)$$

where l_m is the hanging length of the UKS (m); d_4 is the horizontal distance from the latest-breaking position of the UKS to the latest-breaking position of SKS1 (m); and L_m is the periodic weighting interval of the UKS (m).

It is known that when the hanging length of the key stratum in the overburden meets the requirements of formulae (1) or (2), this stratum breaks easily during the extraction of equipment. Therefore, it should be ensured that the support can provide enough support resistance to

FIGURE 3: The position of periodic weighting during the extraction of equipment: (a) above the support; (b) above the recovery roadway; (c) along the coal wall; (d) above the gob.

FIGURE 4: Roof model of a stope with a multikey stratum structure.

control the bench convergence of the roof. Generally, theoretical calculations are used to determine the reasonable working resistance of the support. The method calculates the support load by establishing the cantilever beam, voussoir beam, and other mechanical models of the broken blocks of key strata based on the Key Strata Theory. It can be seen that the premise of applying the theoretical calculation method is to first distinguish the breaking forms of key strata and the articulated forms of broken blocks.

3.1. The Breaking Forms of the Key Strata and the Articulated Forms of Broken Blocks. The breaking forms of key strata can be roughly divided into two types: cantilever beam

and articulated beam. The difference between the two breaking forms is whether the absolute rotation of the broken block exceeds the maximum rotation of the articulated structure [25–27]. The breaking form of the UKS is influenced by the breaking form of the lower key stratum (LKS), as well as the thickness and bulking factor of the weak stratum between the two key strata. Because the breaking interval of the UKS is relatively large and the rotation space is limited, it must break in the form of an articulated beam when the LKS breaks in the form of an articulated beam. By combination with the above analysis, the discriminative approach of the structural form of the block after the key stratum is broken is obtained. (1) First,

the breaking form of each key stratum is determined based on SKS1. (2) It is known that there are two kinds of instability forms of voussoir beam structures, namely, sliding instability and rotation instability [25]. The voussoir beam structure is stable only when the sliding instability and rotation instability do not occur simultaneously. According to the “Sliding-Rotation (S-R)” stability theory of the voussoir beam structure, the specific articulated form (voussoir beam, unstable voussoir beam, or step beam) of a broken block is determined, as illustrated by the roof model shown in Figure 4, where M is the mining height (m), h_0 is the thickness of the immediate roof (m), $h_1 - h_k$ is the thickness from SKS1 to PKS (m), $\sum h_1 - \sum h_{k-1}$ is the thickness of the weak stratum controlled by each SKS (m), $l_1 - l_k$ is the breaking interval from SKS1 to PKS (m), $\theta_1 - \theta_k$ is the rotation angle of the broken block from SKS1 to PKS ($^\circ$), and $W_1 - W_k$ is the absolute rotation of the broken block from SKS1 to PKS (m).

The maximum rotation of the broken block of SKS1, which can form an articulated structure, is $\Delta_{1\max}$. According to the mechanical model of deformation and the instability of the voussoir beam structure, $\Delta_{1\max}$ can be determined by formula (3) [25–29]:

$$\Delta_{1\max} = h_1 - \sqrt{\frac{2q_1 l_1^2}{\sigma_{1c}}}, \quad (3)$$

where q_1 is the load on SKS1 (MPa) and σ_{1c} is the compressive strength of SKS1 (MPa).

The formula for calculating the absolute rotation W_1 is as follows:

$$W_1 = M - (K_p - 1)h_0, \quad (4)$$

where K_p is the bulking factor of the immediate roof and is generally taken as 1.3.

By combining formulas (3) and (4), the criteria of SKS1 breaking in the forms of a cantilever beam and an articulated beam are obtained, respectively, and given by formulas (5) and (6):

$$M - (K_p - 1)h_0 > h_1 - \sqrt{\frac{2q_1 l_1^2}{\sigma_{1c}}}, \quad (5)$$

$$M - (K_p - 1)h_0 \leq h_1 - \sqrt{\frac{2q_1 l_1^2}{\sigma_{1c}}}. \quad (6)$$

Field observations show that the bulking factor of WS1 is basically the same as K_p when SKS1 is broken in the form of a cantilever beam. The bulking factor of WS1 is represented by K_{p0} when SKS1 is broken in the form of an articulated beam,

and K_{p0} is generally taken as 1.05. Therefore, the absolute rotation W_2 of the broken block of SKS2 is

$$W_2 = M - (K_p - 1)h_0 - (K_x - 1) \sum h_1, \quad (7)$$

where the value of K_x is K_{p0} or K_p , and the specific value is determined by the breaking form of SKS1. According to formula (6), the criterion indicating that the broken block of SKS2 can form an articulated beam is obtained:

$$M - (K_p - 1)h_0 - (K_x - 1) \sum h_1 \leq h_2 - \sqrt{\frac{2q_2 l_2^2}{\sigma_{2c}}}, \quad (8)$$

where q_2 is the load on SKS2 (MPa) and σ_{2c} is the compressive strength of SKS2 (MPa). By analogy, the general discriminant formula describing the broken block of the key stratum forming an articulated beam is obtained:

$$W_m = M - (K_p - 1)h_0 - (K_x - 1) \sum h_1 - \dots - (K_x - 1) \cdot \sum h_{m-1} \leq h_m - \sqrt{\frac{2q_m l_m^2}{\sigma_{mc}}}, \quad (9)$$

where the value range of m is from 1 to k .

The specific articulated form of the broken blocks can be further discriminated according to the S-R stability theory of a voussoir beam structure. Sliding instability and rotating instability will occur when the articulated beam structure cannot meet the requirements of formulas (10) and (11), respectively. In the two formulas, φ is the internal friction angle of the broken block, i_m is the lumpiness of the broken block (the ratio of height to length), and θ_m is determined by equation (12):

$$h_m + \sum h_m \leq \frac{\sigma_{mc}}{30\gamma} \left(\tan \varphi + \frac{3}{4} \sin \theta_m \right)^2, \quad (10)$$

$$h_m + \sum h_m \leq \frac{0.15\sigma_{mc}}{\gamma} \left(i_m^2 - \frac{3}{2} i_m \sin \theta_m + \frac{1}{2} \sin^2 \theta_m \right), \quad (11)$$

$$\theta_m = \arcsin \left(\frac{W_m}{l_m} \right). \quad (12)$$

Research shows [25] that the lumpiness of the broken block of the key stratum in the overburden is large during the mining of shallow coal seams. The broken block will slide and lose stability behind the support when formula (13) is satisfied. At this time, the key stratum will break periodically in the form of a step beam. In summary, the discriminant conditions of the breaking form of key strata and the articulated form of broken blocks are obtained, as given by formula (14).

$$i_m > \frac{2 \cos \theta_m + 3 \sin \theta_m}{4(1 - \sin \theta_m)}, \quad (13)$$

$$\left\{ \begin{array}{l} W_m > h_m - \sqrt{\frac{2q_m l_m^2}{\sigma_{mc}}}, \text{ Cantilever beam,} \\ \\ W_m \leq h_m - \sqrt{\frac{2q_m l_m^2}{\sigma_{mc}}} \left\{ \begin{array}{l} \left\{ \begin{array}{l} h_m + \sum h_m \leq \frac{\sigma_{mc}}{30\gamma} \left(\tan \varphi + \frac{3}{4} \sin \theta_m \right)^2 \\ h_m + \sum h_m \leq \frac{0.15\sigma_{mc}}{\gamma} \left(i_m^2 - \frac{3}{2} i_m \sin \theta_m + \frac{1}{2} \sin^2 \theta_m \right) \end{array} \right\}, \text{ Voussoir beam,} \\ \left\{ \begin{array}{l} h_m + \sum h_m > \frac{0.15\sigma_{mc}}{\gamma} \left(i_m^2 - \frac{3}{2} i_m \sin \theta_m + \frac{1}{2} \sin^2 \theta_m \right), \text{ Unstable voussoir beam,} \\ \left\{ \begin{array}{l} h_m + \sum h_m > \frac{\sigma_{mc}}{30\gamma} \left(\tan \varphi + \frac{3}{4} \sin \theta_m \right)^2, \text{ Unstable voussoir beam,} \\ i_m > \frac{2 \cos \theta_m + 3 \sin \theta_m}{4(1 - \sin \theta_m)}, \text{ Step beam.} \end{array} \right\} \end{array} \right\} \end{array} \right. \quad (14)$$

It is known that load transfer coefficients C_1 , C_2 and C_3 of a cantilever beam (unstable voussoir beam), voussoir beam, and step beam are, respectively [25],

$$C_x = \begin{cases} 1, & x = 1, \\ 2 - \frac{l_m \tan(\varphi - \beta)}{2(h_m - W_m)}, & x = 2, \\ 1 - \tan \varphi \cdot \frac{(h_m/\sin \alpha) \cos(\alpha - \theta_m) + (l_m/2) \cos \theta_m}{(h_m/\sin \alpha) \sin(\alpha - \theta_m) - W_m - 0.5a}, & x = 3, \end{cases} \quad (15)$$

where α is the breaking angle of the rock stratum ($^\circ$); β is the angle between the fracture line and the vertical direction ($^\circ$); and a is the height of the extrusion contact surface at the end angle of the block (m), which is determined by formula (16).

$$a = 0.5(h_m - l_m \sin \theta_m). \quad (16)$$

Figure 5 is used as an example to discuss the calculation methods for the reasonable working resistance of a support based on different key stratum structures. Because the thick and hard single-key stratum structure is similar to the single-key stratum structure of a compound group, this paper provides an explanation in a unified way. In Figure 5, H_z is the thickness of the load strata (m), L_k is the hanging length of the immediate roof (m), R_0 is the weight of the immediate roof above the recovery roadway and top beam (kN), $R_1 - R_k$ is the force acting on the substructure due to the broken block of each key stratum (kN), $P_1 - P_k$ is the sum of the

deadweight of the broken block of each key stratum and the overlying load (kN), P_l is the load imposed on the lower structure by the load strata (kN), and P_H is the reasonable working resistance of the support (kN).

3.2. Reasonable Working Resistance of the Support in the Stope with a Single-Key Stratum Structure. A single-key stratum structure means that there is only one key stratum in the roof of a shallow seam, which is the PKS. The instability mechanics model of a single-key stratum structure is shown in Figure 5(a).

The formula for calculating the support load P_z is as follows:

$$P_z = R_0 + R_1. \quad (17)$$

R_0 and R_1 can be calculated by formulas (18) and (19),

FIGURE 5: Mechanical models of the roof structure during periodic weighting: (a) instability mechanics model of a single-key stratum structure; (b) instability mechanics model of a multikey strata structure; (c) instability mechanics model of a single-key stratum structure of an over-seam extracted group.

respectively [16, 30, 31]:

$$R_0 = (L_k + 0.5h_0 \cot \alpha)bh_0\gamma, \quad (18)$$

$$R_1 = C_x \cdot P_1 = C_x \cdot (l_1bh_1\gamma + P_l), \quad (19)$$

where b is the width of the hydraulic support and γ is the average volume-weight of the stratum, which is generally taken as 25 kN/m^3 .

P_z can be calculated by formula (20) [25, 30]:

$$P_l = K_G l_1 b H_z \gamma, \quad (20)$$

where K_G is the load transfer coefficient of the load strata, and the calculation formula is

$$K_G = \frac{0.8l_1}{2H_z(1 - \sin \varphi_1) \tan \varphi_1}, \quad (21)$$

where φ_1 is the internal friction angle of the load strata ($^\circ$).

The calculation formula of the support resistance P_s is obtained based on the above analysis:

$$P_z = (L_k + 0.5h_0 \cot \alpha)bh_0\gamma + C_x \cdot \left[l_1bh_1\gamma + \frac{0.8l_1}{2H_z(1 - \sin \varphi_1) \tan \varphi_1} \cdot l_1bH_z\gamma \right]. \quad (22)$$

Formula (22) is also used to calculate the support load of the stope with a single-key stratum structure of a compound group. However, in this case, h_1 is the total thickness of the single-key stratum of the compound group. The actual support efficiency of the support is 0.9 according to the research results of related literatures [16, 30, 32]. The reasonable working resistance P_H of the support is determined by formula (23):

$$P_H = \frac{P_z}{0.9}. \quad (23)$$

3.3. Reasonable Working Resistance of the Support in a Stope with a Multikey Strata Structure. A multikey strata structure has multiple key strata above the coal seam. It is generally believed that when the distance between two key strata is less than 41 m, there will be mutual influence [30]. An obvious nonuniform periodic weighting phenomenon occurs in a stope with a multikey strata structure. The stope will appear as weak periodic weighting when the LKS is broken and the UKS is stable. At this time, the strength of periodic weighting is relatively low because the UKS obstructs the transmission of the overburden load. The instability load acts on the LKS when the UKS is broken, which causes two key strata to break simultaneously. Strong periodic weighting will occur in the stope at this time [33]. Strong periodic weighting should be guaranteed to occur in the multikey strata before the recovery roadway is laid out. However, the key stratum in the overburden of a stope with a multikey strata structure may still break during the extraction of the equipment when the hanging length of the key stratum meets formula (1) or (2). The instability mechanics model of the multikey strata structure is shown in Figure 5(b). In this case, R_1 is calculated by formula (24) [34–36]:

$$\begin{aligned} R_1 &= C_x \cdot P_1 = C_x \cdot [l_1 b \gamma (h_1 + \sum h_i) + R_2] \\ &= C_x \cdot [l_1 b \gamma (h_1 + \sum h_i) + C_x \cdot (\dots + C_x \cdot P_k)]. \end{aligned} \quad (24)$$

Therefore, the calculation formula of the support resistance P_z is

$$\begin{aligned} P_z &= (L_k + 0.5h_0 \cot \alpha) b h_0 \gamma + C_x \\ &\cdot [l_1 b \gamma (h_1 + \sum h_i) + C_x \cdot (\dots + C_x \cdot P_k)]. \end{aligned} \quad (25)$$

3.4. Reasonable Working Resistance of the Support in a Stope with a Single-Key Stratum Structure under the Gob. The key stratum in the overburden is broken after the upper seam is mined. Therefore, SKS1 will act as the PKS, as shown in Figure 5(c). However, there is still a connection of the horizontal forces between the broken blocks, and an articulated structure can still be formed due to the effect of mining the lower coal seam. It is difficult to accurately determine the migration law of the broken blocks in the overburden of the upper coal seam during the mining of the lower coal seam. Therefore, the support resistance P_z should be considered in the case of SKS1 synchronous breaking caused by the instability and rotation of the broken blocks. Additionally, the compacted rock mass in the caving zone of the upper coal seam can be regarded as a weak stratum. The calculation formula of the support resistance P_z is the same as formula (25).

4. Engineering Background

The Liangshuijing Coal Mine and Fengjiata Coal Mine are both located in the Yushenfu Mining Area. The geographical locations of the two mines are shown in Figure 6. Accidents due to roof and support damage have occurred in both mines during equipment extraction.

4.1. #3-1 Coal Seam in the Liangshuijing Coal Mine. The approved production capacity of the Liangshuijing Coal Mine is 8 Mt/a. The #3-1 coal seam is the main mining seam in this mine. The occurrence characteristics of the coal strata are shown in Figure 7. The #3-1 coal seam and its roof and floor were drilled and sampled, and then the samples were processed into standard test pieces in the laboratory. A mechanical testing and simulation (MTS) universal testing machine is applied to test the mechanical parameters of the coal strata [37, 38]; see Table 1 for details. The overlying strata are determined to be a single-key stratum structure according to the discrimination conditions of the key stratum [39, 40]. Fine sandstone with a thickness of 9 m is the PKS, according to the definition of the PKS. The regular circulation footage is 0.8 m, and the gob roof is treated by a caving method. The working face is equipped with ZY8000/17.5/35-type supports.

Field practice shows that when the width of the recovery roadway is half the length of the support, the supports can be successfully extracted. Therefore, the reasonable width of the recovery roadway is determined according to the length of the support, the regular circulation footage, and the safe distance of a certain width. It is known that the length and width of the ZY8000/17.5/35-type support are 6 m and 1.66 m, respectively. The reasonable width of the recovery roadway was set to 3.2 m by considering that the regular circulation footage is 0.8 m. According to the measured rock pressure data in the #3-1 coal seam, the working resistance curve of the support was drawn as shown in Figure 8. A total of 13 periodic weightings are monitored. The average interval of the periodic weighting is approximately 16.7 m. By substituting the values of each parameter into formula (1), we can obtain $l_1 = 11.6$ m. Thus, SKS1 is easy to break during the extraction of equipment.

By referencing the related literature [16, 25, 27], the following are assumed: $\varphi = 39.5^\circ$, $\alpha = 90^\circ$, and $\beta = 0^\circ$. L_k is the sum of the width of the recovery roadway and the length of the top beam. It is known that the length of the top beam is 4.1 m, and $L_k = 7.3$ m is determined after calculation. The compressive strength of fine sandstone, $\sigma_{1c} = 50.5$ MPa, is derived from laboratory test data. According to the mining conditions of #3-1 coal seam and the previous statement, the following are determined: $l_1 = 11.6$ m, $b = 1.66$ m, $\varphi_1 = 33^\circ$, and $M = 3$ m. After calculation by formulas (3) and (4), $\Delta_{1\max} = 5.76$ m and $W_1 = 1.8$ m are obtained. According to formula (14), the broken block of SKS1 will form a step-beam structure during the extraction of equipment. Therefore, formula (26) is used to calculate the support load P_z . The result shows that $P_z = 5220$ kN; that is, $P_H = 5800$ kN.

FIGURE 6: Geographical locations of the two mines.

FIGURE 7: Bore hole columnar section of the coal strata in the Liangshuijing coal mine.

TABLE 1: Physical and mechanical parameters of the coal strata in the Liangshuijing coal mine.

Rock formations	Bulk modulus (GPa)	Shear modulus (GPa)	Density (kg·m ⁻³)	Friction angle (°)	Cohesion (MPa)	Tensile strength (MPa)
Aeolian	0.0125	0.0071	1760	23	0.019	
Fine sand	0.0242	0.0136	1810	20	0.003	
Loess	0.0383	0.0177	1860	27	0.0016	
Clay	0.28	0.093	1960	25	0.85	
Siltstone	4.3	3.36	2730	32.9	4.1	3.2
Mudstone	2.86	1.4	2540	39	2.8	2.48
Fine sandstone	16.04	12.02	2800	43	9.2	8.5
Siltstone	5	3.8	2630	35	3.8	2.75
#3-1 coal seam	2.8	1.2	1420	27	0.6	0.4
Siltstone	6.25	3.57	2570	31.4	9.2	8.5
Fine sandstone	21.01	13.52	2873	42	3.2	1.29

$$P_z = (L_k + 0.5h_0 \cot \alpha)bh_0\gamma + \left[1 - \tan \varphi \cdot \frac{(h_1/\sin \alpha)\cos(\alpha - \theta_1) + (l_1/2)\cos \theta_1}{(h_1/\sin \alpha)\sin(\alpha - \theta_1) - W_1 - 0.5a} \right] \cdot \left[l_1bh_1\gamma + \frac{0.8l_1}{2H_z(1 - \sin \varphi_1)\tan \varphi_1} \cdot l_1bH_z\gamma \right]. \quad (26)$$

This result shows that the working resistance of the selected supports in the #3-1 coal seam is low. Therefore, the selected supports in the #3-1 coal seam can meet the requirements of the roof support during the extraction of equipment.

4.2. #2 Coal Seam in the Fengjiata Coal Mine. The Fengjiata Coal Mine is located in the eastern part of the Fugu Mining Area, and its approved production capacity is 6 Mt/a. The #2 and #4 coal seams are mined at the same time in this mine. The occurrence characteristics of the coal strata are shown in

FIGURE 8: Working resistance curve of the support during the mining of the #3-1 coal seam.

Figure 9. The mechanical parameters of the coal strata are determined by mechanical tests, as shown in Table 2. A multikey strata structure exists in the roof of the #2 coal seam. According to the definitions of PKS and SKS, a medium coarse sandstone stratum with a thickness of 7.5 m is the SKS closest to the stope, and a medium sandstone stratum with a thickness of 10.82 m is the PKS. The regular circulation footage of the working face is 0.8 m. The working face is equipped with ZY8500/20/42-type supports.

The length and width of the ZY8500/20/42-type support are 6.6 m and 1.75 m, respectively. The reasonable width of the recovery roadway is set to 4 m by considering that the regular circulation footage is 0.8 m. Figure 10 plots the periodic weighting law observed during the last mining period of the #2 coal seam. The figure shows that a non-uniform periodic weighting phenomenon occurred in the working face. The strength of the strong-period weighting is approximately 1.22 times that of the weak-period weighting. The average interval of the strong-period weighting is approximately 36.2 m and that of the weak-period weighting is approximately 18 m. By substituting the values of each

parameter into formula (1), we obtain $l_1 = 13$ m. Thus, SKS2 breaks easily during the extraction of equipment. When pressure-relief technology is applied in advance to release the accumulated rock pressure in the PKS, according to formula (2), the PKS will be in a steady state during the extraction of equipment.

By combining working-face mining conditions and field-monitoring data, the following are determined: $l_2 = 13$ m, $b = 1.75$ m, and $M = 3$ m. It is known that the length of the top beam is 4.5 m, and $L_k = 8.5$ m is determined after calculation. The compressive strength of medium coarse sandstone, $\sigma_{2c} = 68.4$ MPa, is derived from laboratory test data. The values of α , β , φ , and φ_1 are the same as those provided above. After calculation by formulas (3) and (4), $\Delta_{2\max} = 5.3$ m and $W_2 = 2.8$ m are obtained. According to formula (14), the broken block of SKS2 will form a voussoir beam structure during the extraction of equipment. Therefore, formula (27) is used to calculate the support load P_z . The result shows that $P_z = 8247$ kN. This result shows that the selected support in the #2 coal seam can basically meet the requirements of the roof support.

$$P_s = (L_k + 0.5h_0 \cot \alpha)bh_0\gamma + \left[2 - \frac{l_2 \tan(\varphi - \beta)}{2(h_2 - W_2)} \right] \cdot (L_2 b \gamma h_2 + L_2 b \gamma \sum h_2). \quad (27)$$

4.3. #4 Coal Seam in the Fengjiata Coal Mine. The average distance between the #2 coal seam and the #4 coal seam is only 12.4 m. SKS1 will act as the PKS during the mining of the #4 coal seam. The working face is also equipped with ZY8500/20/42-type supports. The reasonable width of the recovery roadway that meets the extraction requirements is 4 m, which is consistent with that of the #2 coal seam. Figure 11 shows the periodic weighting law observed during the last mining period of the #4 coal seam. The strength of

the periodic weighting varies greatly. Low-strength periodic weighting is caused by the breakage of the SKS. High-strength periodic weighting occurs because the rotational instability of the broken block of the PKS causes SKS1 to break. The average interval of periodic weighting is approximately 10.7 m. Therefore, it can be determined that SKS1 must break during the extraction of equipment.

The reasonable working resistance of the support is considered in the case of synchronous instability of each key

FIGURE 9: Bore hole columnar section of the coal strata in the Fengjiata coal mine.

TABLE 2: Physical and mechanical parameters of the coal strata in the Fengjiata coal mine.

Rock formations	Bulk modulus (GPa)	Shear modulus (GPa)	Density (kg·m ⁻³)	Friction angle (°)	Cohesion (MPa)	Tensile strength (MPa)
Loess	0.040	0.019	1880	27	0.0018	
Subclay	0.15	0.054	1840	23	0.52	
Clay	0.28	0.093	1960	25	0.85	
Sandy mudstone	2.56	2.36	2510	40	2.45	2.01
Pelitic siltstone	2.2	1.05	2650	42	1.88	1.77
Coarse sandstone	15.29	8.31	2890	30.6	12.4	13.1
Sandy mudstone	4.9	3.2	2520	35	1.18	1.8
Fine sandstone	6.25	3.57	2570	31.4	9.2	8.5
Pelitic siltstone	4.2	2.65	2490	39	3.2	1.29
Medium coarse sandstone	18.7	12	2550	37.5	11.5	7.8
Sandy mudstone	7.5	4.2	2500	29	4.5	2.5
#2 coal seam	5	2.3	1380	28	1	0.5
Sandy mudstone	4.3	2.8	2545	37	3.2	2.25
#3 coal seam	5.6	2.3	1390	28	1	0.5
Siltstone	10.83	8.13	2460	38	3.75	1.84
Pelitic siltstone	4.3	3.36	2730	32.9	6	2.5
#4 coal seam	2.5	1.72	1420	29.5	2.11	2.6
Pelitic siltstone	6.27	5.19	2665	44.5	11.5	7.8

stratum. It is known that $L_k = 8.5$ m, $L_1 = 10.7$ m, $L_2 = 18$ m, $L_3 = 36.2$ m, $b = 1.75$ m, $M = 3.3$ m, and $\sigma_{2c} = 68.4$ MPa. The results of mechanical tests show that $\sigma_{1c} = 49.5$ MPa and $\sigma_{3c} = 53.5$ MPa. According to formula (14), the form of the key strata in the overburden is step beam-voussoir beam-

unstable voussoir beam. Therefore, formula (28) is used to calculate the support load P_z . The result shows that $P_z = 9580$ kN; that is, $P_H = 10644$ kN. This result shows that the yield load of the selected support in the #4 coal seam is low.

FIGURE 10: Working resistance curve of the support during the mining of the #2 coal seam.

FIGURE 11: Working resistance curve of the support during the mining of the #4 coal seam.

$$P_s = (L_k + 0.5h_0 \cot \alpha)bh_0\gamma + \left[1 - \tan \varphi \cdot \frac{(h_1/\sin \alpha)\cos(\alpha - \theta_1) + (l_1/2)\cos \theta_1}{(h_1/\sin \alpha)\sin(\alpha - \theta_1) - W_1 - 0.5a} \right] \cdot \left\{ L_1b\gamma(h_1 + \sum h_1) + \left[2 - \frac{L_2 \tan(\varphi - \beta)}{2(h_2 - W_2)} \right] \cdot \left(L_2b\gamma(h_2 + \sum h_2) + L_3b\gamma h_3 + \frac{0.8L_3}{2H_z(1 - \sin \varphi_1)\tan \varphi_1} \cdot L_3bH_z\gamma \right) \right\}. \quad (28)$$

5. Numerical Simulation of the Recovery Roadway Support Design

Based on the actual geological conditions of the Liangshuijing Coal Mine and Fengjiata Coal Mine, FLAC2D software is applied to establish numerical models to study the reasonable support parameters of the recovery roadway. The numerical model of the recovery roadway of Liangshuijing coal mine shown in Figure 12 was taken as an

example to illustrate the modeling process. The Mohr-Coulomb constitutive model was applied to all models [41]. A uniform load was applied in the vertical direction to simulate the gravity of the overlying strata. We used the stress-relief method to investigate the in situ stress in the Liangshuijing Coal Mine. Field measurements showed that the vertical stress was 3.55 MPa; the minimum principal stress was parallel to the longwall face advancing direction, with a value of 1.81 MPa; and the maximum principal stress

FIGURE 12: Numerical model of the recovery roadway in the Liangshuijing coal mine.

was 4.14 MPa. According to the test results of the in situ stress, the horizontal lateral pressure coefficient was determined to be 0.5. The upper boundary was a free surface, the bottom boundary was restricted to move vertically, and the side boundaries around the model restricted horizontal movement [42]. The double-yield constitutive model of the compacted rock mass in the gob was established. The numerical simulation parameters of the compacted rock mass are shown in Table 3, according to the compaction theory in the gob [43–46].

5.1. Optimization of the Recovery Roadway Support Design in the #3-1 Coal Seam. In the past, the Liangshuijing Coal Mine used an engineering analogy method to formulate the support design of recovery roadways. The support parameters are shown in Figure 13.

The steel beam is made of round steel with dimensions of $\Phi 16 \text{ mm} \times 6,000 \text{ mm}$. Each steel beam connects six rockbolts. JDPET200 * 200MS-type resin mesh was selected. The actual construction of the initial support design was time-consuming. Adequate development time was given for the plastic zone in the surrounding rock. The deformation of the rock surrounding the recovery roadway was large, which affected the normal operation of extraction. The actual supporting efficiency of the roof rockbolts near the gob was low and could not meet the requirements of rapid extraction. The initial support design was optimized based on the above problems, as shown in Figure 14. (1) Three wire ropes were laid instead of 3 rows of rockbolts near the gob. The diameter of the wire rope was 21.5 mm, and the spacing was 800 mm. (2) Adjacent rockbolts were replaced with anchor cables.

The numerical simulation method was applied to compare the surrounding rock control effects of the two support designs. Refer to Table 1 to determine the physical and mechanical parameters of each rock stratum in the model. The setting load of the support was set to 6400 kN, and the yield load was set to 8000 kN. The mechanical parameters of the rockbolts and cables in the model were obtained according to field test data and the data provided by the manufacturer; see Table 4 for details [47]. The anchoring

force of the rockbolt was ensured to be greater than 105 kN. The pretightening force of the anchor cable was not less than 100 kN, and the anchoring force was not less than 150 kN. The grout properties were consistent with laboratory data obtained by performing pull-out tests on the rockbolts used in the physical model [36]. In this model, 10,000 time steps were calculated for each meter that the working face advanced to simulate the influence of mining on the overburden. After each rockbolt was installed, 1,000 time steps were calculated. After the recovery roadway support was completed, 100,000 time steps were run. The wire rope was not simulated in the model because it was used to fix the resin mesh.

Figure 15 shows the distribution of the plastic zone after the two support designs were simulated.

Figure 15 shows the following. (1) The lithology of the immediate roof was weak, and plastic failure occurred during the simulation support. Plastic failure also occurred in the PKS adjacent to the gob. (2) The plastic zone in the roof was divided into a shear failure zone, a plastic failure zone, and a tensile-failure zone. The development range of the plastic zone in each section was larger than that achieved with the improved support design because the initial support design ran more steps. (3) Because of the low support strength behind the support in the improved support design, the range of the tensile-failure zone was larger than that in the initial support design. (4) The deformation of the surrounding rock behind the support in the improved support design was larger than that in the initial support design because of the low tensile strength of the rock. However, this effect would not have a significant impact on the normal extraction of equipment.

The roof subsidence curves of each section for the two support designs are shown in Figure 16.

As shown in Figure 16, (1) the roof subsidence increased monotonically from the terminal line to the rear of the support; (2) the roof subsidence slowed near the rockbolt; and (3) the roof subsidence of each section in the initial support design was significantly larger than that in the improved support design. The results show that the surrounding rock control effect of the improved support design is better than that of the initial support design.

TABLE 3: Numerical simulation parameters of the compacted rock mass in the gob.

Bulk modulus (GPa)	Shear modulus (GPa)	Density ($\text{kg}\cdot\text{m}^{-3}$)	Friction angle ($^{\circ}$)	Cohesion (MPa)	Tensile strength (MPa)
0.497	0.298	1800	5	0.001	0

FIGURE 13: Initial support design of the recovery roadway in the #3-1 coal seam.

FIGURE 14: Improved support design of the recovery roadway in the #3-1 coal seam.

TABLE 4: Mechanical parameters of the rockbolts and cables.

Supporting materials	Elastic modulus (GPa)	Cross-sectional area (m^2)	Tensile strength (MPa)	Stiffness of anchorage agent (GPa)	Cohesion of anchorage agent (mN/m)
Rockbolts	20	0.00025	250	5.35	0.42
Flank rockbolts	20	0.00025	200	5.35	0.42
Cables	20000	0.00018	1900	5.35	0.42

FIGURE 15: Plastic zone distribution of the two support designs.

FIGURE 16: Roof subsidence curves of each section for the two support designs.

5.2. Optimization of the Recovery Roadway Support Design in the #2 Coal Seam. The initial support design of the recovery roadway in the Fengjiata Coal Mine is shown in Figure 17.

The initial support design was improved, as shown in Figure 18, by considering the development of the plastic zone in the surrounding rock. Four wire ropes were laid

FIGURE 17: Initial support design of the recovery roadway in the Fengjiata coal mine.

FIGURE 18: Improved support design of the recovery roadway in the Fengjiata coal mine.

instead of 5 rows of rockbolts near the gob. The diameter of the wire rope was 21.5 mm, and the spacing was 1000 mm.

The processes of excavation and the calculation of the model were consistent with those mentioned above. The setting load of the support was set at 6800 kN, and the yield load was set at 8500 kN in the simulation of the two support designs. Figure 19 shows the distribution of the plastic zone after the two support designs were simulated.

Figure 19 shows the following: (1) the roof directly above the recovery roadway underwent integral plastic failure with the two support designs; (2) plastic failure occurred in SKS2 above the support; and (3) the development range of the plastic zone in the surrounding rock of the improved support design was smaller than that of the initial support design.

The roof subsidence curves of each section in the two support designs are shown in Figure 20.

The roof subsidence of the improved support design was significantly lower than that of the initial support design. Therefore, a decision was made to apply the improved support design in the recovery roadway of the #2 coal seam.

5.3. Optimization of the Recovery Roadway Support Design in the #4 Coal Seam. The initial support design and improved support design of the recovery roadway in the #4 coal seam were consistent with those in the #2 coal seam. The yield load of the support was 8500 kN in the simulation of the initial support design, and the yield load of the support was 9600 kN in the simulation of the

FIGURE 19: Plastic zone distribution of the two support designs.

FIGURE 20: Roof subsidence curves of each section in the two support designs.

improved support design. Figure 21 shows the distributions of the plastic zone after the two support designs were simulated.

Figure 21 shows the following: (1) the roof directly above the recovery roadway underwent integral plastic failure with the two kinds of support designs; (2) plastic failure occurred

FIGURE 21: Plastic zone distributions of the two support designs.

in SKS1 above the recovery roadway; and (3) the development range of the plastic zone in the surrounding rock with the improved support design was smaller than that with the initial support design.

The roof subsidence curves of each section in the two support designs are shown in Figure 22.

The roof subsidence of the improved support design was significantly lower than that of the initial support design.

The results showed that the key to the safe and rapid extraction of equipment in shallow seams is to ensure that the support has a reasonable working resistance and to improve the supporting efficiency of the recovery roadway.

6. Field Application and Discussion

6.1. Field Application. Based on a combination of theoretical analysis and numerical simulation, the improved support design was adopted to support the recovery roadway during the extraction of equipment in two mines. Because the yield load of the selected support is on the low side, individual hydraulic props were installed on both sides of the support as reinforcement during the extraction of equipment in the #4 coal seam of the Fengjiata Coal Mine. Table 5 lists the parameters of the individual hydraulic props. The support resistance of the support was raised to approximately 9700 kN. Figure 23 shows the construction sites of the recovery roadway in the Liangshuijing Coal Mine and Fengjiata Coal Mine.

The field application results show that the rock pressure appearance and surrounding rock deformation of the recovery roadway can meet the requirements of safe and rapid extraction. Moreover, no roof or support damage occurred during the extraction of equipment. The Liangshuijing Coal Mine and Fengjiata Coal Mine completed the extractions 20 and 14 days ahead of schedule, respectively. The safe and rapid extraction of the longwall face in shallow seams was realized. This success provides a reference for the extraction of equipment under similar geological conditions.

6.2. Discussion. This paper mainly studies the stability control of the rock surrounding recovery roadways to ensure that the supports have reasonable working resistance and to improve the support of the recovery roadways. However, the yield load of the support is generally fixed. When the support cannot meet the roof support requirements, the support resistance of a single hydraulic prop is very limited. If necessary, the roof can be controlled by arranging wooden cribs.

The plastic failure of the roof is inevitable during the extraction of equipment because of the weak lithology. The role of roof rockbolts near the gob is limited, and the rockbolts should be replaced by wire ropes to fix the resin mesh. Therefore, while ensuring the safe extraction of equipment, the construction time can be shortened and the high production and high efficiency of the mine can be realized.

FIGURE 22: Roof subsidence curves of each section in the two support designs.

TABLE 5: Technical parameters of the individual hydraulic props.

Type	Height (m)	Setting load (kN)	Yield load (kN)
DZ35-30/110Q	2.7~3.5 m	144~188	300

FIGURE 23: Construction sites of the recovery roadways: (a) #3-1 coal seam in the Liangshuijing coal mine; (b) #4 coal seam in the Fengjiata coal mine.

7. Conclusions

This paper mainly studied the stability control of the rock surrounding longwall recovery roadways in shallow seams. The main conclusions are as follows:

- (1) Pressure-relief technology can effectively release the accumulated rock pressure in the roof before the extraction of equipment. However, pressure-relief technology is applicable only under certain conditions. The technology should be applied reasonably according to the roof conditions and the law of periodic weighting.
- (2) The discriminative approach was proposed for discriminating the breaking form of key strata and the articulated form of broken blocks according to the formation conditions of cantilever-type breaking and the “S-R” stability theory of the voussoir beam structure. On this basis, mechanical models of roof instability were established based on four key stratum structures in the overburden of shallow seams. The calculation methods for the reasonable working resistance of the support were discussed.
- (3) According to the actual geological conditions of the two mines, the reasonable working resistance of the support and the reasonable width of the recovery roadway were calculated. Then, the support designs of the recovery roadways were improved based on the development of the plastic zone. The numerical simulation results showed that the development range of the plastic zone in the surrounding rock and the roof subsidence were significantly reduced after the improved support design was applied.
- (4) The field application effect of the research results is ideal and can help realize the safe and rapid extraction of longwall faces in shallow seams. The key to the stability control of the rock surrounding longwall recovery roadways in shallow seams is ensuring that the support has a reasonable working resistance, enhancing the supporting efficiency of the roadway and speeding up the extraction of equipment.

Data Availability

Part of the data used to support the findings of this study are included within the article. Rest of the data used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

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