Analysis and control of the mechanism of coal pillar sloughing in the shallow buried thick coal seam

Jie Zhang1,2 | Jianjun Wu1,2 | Sen Yang1,2 | Wenyong Bai1,3

1School of Energy Engineering, Xi’an University of Science and Technology, Xi’an, China
2Key Laboratory for Mining and Disaster Prevention in the West of the Ministry of Education, Xi’an, China
3School of Coal Engineering, Shanxi Datong University, Datong, Shanxi, China

Abstract
In northern Shaanxi province, the coal seam deposit conditions are generally characterized by large thickness and shallow depth of burial, the phenomenon of coal pillar sloughing is serious in the process of working face retrieval, which brings great potential danger to mine safety production. In this paper, by monitoring the stress and damage of the coal pillar in the field section, the stress changes and damage degree of the working face in different mine stages are determined. A new mechanical model of coal pillar in a thick coal seam is constructed and the stress changes and structural breakage law of each section are obtained, the stress transfers mechanism of overburden structure and the stress damage to coal pillar during the mining process of upper and lower section comprehensive mining working face is analyzed. The stress, microscopic damage, and the degree of sloughing in the coal pillar were significantly reduced by monitoring through the top plate hydraulic cracking pressure relief control program. The results of the study show that the extent of sloughing coal pillar and internal damage is closely related to the length of key block B of the masonry beam formed by the overlying rock layer and the position of the break line, by changing the orientation of the overlying masonry beam rock layer and the position and length of the tendency to break line, the sloughing coal pillar is effectively controlled and its stress and damage are reduced. We have achieved good results in control of the sloughing in the Hanjiawan coal mine and similar mines in northern Shaanxi.

Keywords
coal pillar, failure mechanism, field practice, pressure superposition, sloughing control, theoretical analysis

1 | INTRODUCTION

With the continuous development of coal mining science and technology, large-scale fully mechanized mining mode of thick and extra-thick coal seams has become the key means to achieving high efficiency and high yield in coal mines. The reserves of thick coal seams account for 44% of China’s total coal reserve and 45% of total coal production.1,2 In the process of fully mechanized mining in large mining heights and thick coal seams, the stability...
of coal pillar and mechanism sloughing cause huge security risks to mine safety production, transition supported to bring great economic burden to mine production. However, the coal seam thickness in the northwest mining areas in China is generally large and the geological structures, such as overlying strata are similar. Therefore, it is urgent to study the sloughing mechanism and control of coal pillars in the thick coal seam, which provides the basis for underground safety production.

Many scholars have done extensive research on the deformation and failure mechanism and control of gob side roadway and the optimization of coal pillar. The development of plastic zones in surrounding rocks is directly related to the configuration, stiffness, and strength of roadway support structures. The butterfly-shaped plastic zone theory of roadway surrounding rocks has been developed and the nonuniform evolution law of pressure and distribution characteristics of plastic zones in mining roadway have been analyzed. Based on the nonuniformity of pressure and surrounding rock failure, the mine pressure of roadway has been effectively studied. Especially for the large deformation problem of the gob side coal roadway, various effective research methods are also used. In view of the stratified fully mechanized mining face in extra-thick coal seam, the instability mechanism, and control technology of coal pillar under the direct roof of superthick coal seam are studied by theoretical analysis and field to test. In recent years, many working faces adopting double-roadway arrangements have appeared in the western mining areas in China, which makes coal pillar abutment pressure complex under repeated mining. In view of the stability and deformation characteristics of the mining roadway in the mining process of large mining height working faces, deep well working faces and fully mechanized cave working faces, the optimization of coal pillar and dynamic load mechanism are studied, and the asymmetric deformation mechanism and control of gob side roadway in fully mechanized cave mining are studied.

In summary, the research on the distribution law of abutment pressures of coal pillar and its size optimization has achieved remarkable results. However, there is less research on the analysis of the mechanism and support control of the sloughing caused by the different fracture structures of the overlying rock seam in the thick coal seam section, the thick coal seam with similar geological conditions is abundant in northern Shaanxi Province, China. Therefore, this paper takes the Hanjiawan Coal Mine of Shaanxi Coal Group as the main engineering background and adopts the field pressure and damage monitoring, theoretical analysis, and industrial practical support control to study the sloughing mechanism when the middle coal pillars are at the side of the gob-side entry and the side of gob in the mining process of 3107 and 3108 working faces. According to the sloughing mechanism, the control scheme is formulated and the control effect is verified by underground practice. It has been applied on site in mines with similar geological conditions in northern Shaanxi such as Shalang coal mine and Anshan coal mine, and the effect of sloughing control is obvious.

2 GEOLOGICAL BACKGROUND AND WORKING FACE PROFILE

The geological structure of 3–1 coal seams is a typical thin bedrock loose layer, which is the main coal seam in Hanjiaowan Coal Mine. Coal seam thickness 3.1–5.4 m, average 4.25 m. Buried depth +157.07 to +111.50 m, average buried depth 134 m, recoverable area 12.77 km². The main roof is composed of fine-grained sandstone with a thickness of 12.4 m. The immediate roof is composed of siltstone with an average thickness of 4.5 m. The direct bottom is composed of siltstone with an average thickness of 3.5 m.

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The width of 3107 fully mechanized working face is 265 m, the mined length is 2441 m, the thickness of the coal seam is 3–5.4 m, the average is about 4.2 m, the dip angle of the coal seam is about 1.5°, the average buried depth is 150 m. The south is 3108 working face; the north is 3106 working face gobs, and the east is the mine boundary. The width of 3108 fully mechanized working face is 262 m, the mined length is 2435 m, the thickness of the coal seam is 3.3–4.95 m, the average is about 4.15 m, the dip angle of the coal seam is about 1.5°, the average buried depth is 148 m, and the width of coal pillar between the two fully mechanized working faces is 15 m. A diagram of the working face position is shown in Figure 1. The coal seam geological column and mechanical parameters are shown in Figure 2.
3 | FIELD MONITORING

3.1 | Coal pillar stress monitoring

The stress changes of the coal pillar of 3107 working face and 3108 working face during mining were monitored. Four groups of stress boreholes were arranged in 3107 head entry and 3108 tail entry on both sides of the coal pillar. Two groups on each side were arranged, a total of 16. Each group has four boreholes, the drilling depth is 2, 4, 6, 8 m, the drilling diameter is 42 mm, the drilling spacing is 3 m, and the drilling distance is 1.5 m from the bottom. The specific location and parameters are shown in Figure 3. The stress change of the coal pillar in the mining process of working face is monitored continuously, a stress is collected every 5 min. The stress monitoring length of the coal pillar is 70 m, and the width of entry on both sides is 5.4 and 5.2 m, respectively. First, the stress of coal pillars at different distances from the station to the working face during the
mining of the 3107 working face is monitored. Secondly, when the end of the 3107 working face is the mined-out area, the stress of coal pillar in the same of 3108 working face is monitored at different distances from the station to the working face, the duration is long.

With the mining of 3107 working faces, the stress meter data on both sides of the coal pillar are monitored and recorded, and the vertical stress changes of the coal pillar at different distances between the station and the working face are analyzed. Figure 4a shows the variation curve of the vertical stress in the sidewall of the coal pillar along the tail entry with the mining of the working face. The variation in the sidewall stress in the range of 90 m in the length of the coal pillar is monitored when the depth of the coal pillar is 2, 4, 6, and 8 m, respectively. The variation trend of vertical stress of coal pillars at different depths can be seen through the stress variation curve. The distance between the station and the working face is 5–15 m, and the vertical stress of the coal pillar at different depths reaches the maximum value, which is about 5.4 MPa at 2 and 4 m depths, about 3.75 MPa at 6 and 8 m depths. The measuring station lags behind the working face 0–20 m, the vertical stress of the coal pillar at the depth of 2 m is obviously relieved, and the stress is about 2.8 MPa. The vertical stress of the coal pillar at the depth of 4, 6, and 8 m is about 3.5 MPa. The station is 30–70 m away from the working face. The vertical stress at different depths of the coal pillar is basically in the original rock stress area, the stress is about 3.3 MPa. It indicates that the advance stress of 3107 working faces has little effect on the coal pillar of the tail entry side section, but the difference in the sloughing degree is obvious.

Figure 4B shows the variation of the vertical stress in the sidewall of the coal pillar along the head entry with the mining of the working face. The stress changes in the range of 70 m in the length of the coal pillar at the depths of 2, 4, 6, and 8 m were monitored. It can be seen from the stress curve that the vertical stress changes at different depths of the coal pillar are quite different. Similarly, when the station is 5–15 m away from the working face, the vertical stress at different depths of the coal pillar reaches the maximum value. The stress jump at the depth of 2 m is relatively large, with a maximum of about 5.1 MPa, a minimum of about 2.2 MPa, a depth of 4, 6, and 8 m of about 6.8 MPa. The measuring station is 0–5 m away from the working face, the stress at the depth of 2 m is significantly smaller than at other depths, and the stress at 6 and 8 m is the largest, indicating that the more obvious the crushing degree of the coal pillar side is, the smaller the stress value is, the more obvious the stress relief is. The measuring station is 15–40 m away from the working face, and the stress of the coal pillar is gradually decreasing. The station is 40–70 m away from the working face, the vertical stress at different depths of the coal pillar is basically in the original rock stress area, the stress is about 3.5 MPa.

After the mining of 3107 working faces, the coal pillar in the middle section forms a gob on the side of the head entry. During the mining of the 3108 working face, with the continuous change of the distance between the station and the working face, the vertical stress at different drilling depths of the coal pillar in the side section of the tail entry is monitored. The vertical stress curve is shown in Figure 5. When the measuring station is 5–10 m away from 3108 working faces, the vertical stress of different borehole depths of the coal pillar is counted, and the histogram of average vertical stress is obtained, as shown in Figure 6.
The gob is formed after the return to 3107 working faces. During the mining process of 3108 working faces, the vertical stress at different depths of the coal pillar of the side section of the tail entry is monitored. The vertical stress of the coal pillar increases significantly, and the maximum vertical stress at different drilling depths increases by about one-third as a whole, indicating that the stress concentration is formed above the coal pillar by breaking the key block B of the overburden rock along the gob side and breaking the key block B of the overburden rock along the working face. When the station distance is 0–5 m from 3108 working faces, the vertical stress at different depths of the coal pillar increases by 1.5–2.5 MPa. The station distance is 5–15 m from the 3108 working face, and the vertical stress at different depths of the coal pillar increases by 2.5–4.5 MPa. The station distance is 15–40 m from the 3108 working face, and the vertical stress at different depths of the coal pillar increases by 1.5–3 MPa. The station distance is 40–70 m from 3108 working faces, the vertical stress at different depths of the coal pillar increases by 0–1 MPa, and the stress of 2 m depths of the coal pillar varies greatly. It shows that the key block B of masonry beams formed by strike and dip rock plays a key role, which causes the jump change of stress value of the shallow borehole in coal pillar.

When the 3108 working face is mined to 5–15 m from the station, the stress jump variation on boreholes at different depths is the largest, so the vertical stress of coal pillars at different depths is counted. The variation range of vertical stress at the depth of 2 m of the coal pillar is large, the average stress is about 5.05 MPa. The variation range of vertical stress at 4 m depths of coal pillar is obviously reduced, the average stress is about 8.62 MPa. The vertical stress of a coal pillar at 6 m depths is about 10.50 MPa, and the vertical stress of a coal pillar at 8 m depths is about 8.85 MPa. According to the statistical histogram of vertical stress at different depths of coal pillar, under the influence of advanced stress of working face, the vertical stress at 6 m depth of coal pillar reaches the maximum value, indicating that the fracture zone and plastic zone of coal pillar are located from the side of coal pillar to 6 m depths of coal pillar, the elastic zone with stress increase is located at a depth greater than 6 m. The vertical stress of the side wall of the coal pillar shows a parabolic trend. If it is symmetrically distributed, the overall stress of the coal pillar is “double hump” type, indicating that the overlying strata structure of the coal pillar is redistributed due to rotation collapse, and the masonry beam structure and its key blocks are gradually moved up.

3.2 Coal pillar damages monitoring

In the mining process of 3107 working faces, the uneven sloughing phenomenon occurs to the coal pillar side of the head entry, the failure place in the advanced pressure stage is monitored. Therefore, the damage peep at the coal pillar is monitored. Through monitoring, it can be found that the damage degree is the most serious at the depth of 0–1.25 m in the coal pillar sidewall. Broken coal seam is generated under the action of pressure in the hole. The radial and axial cracks in the hole are fully developed, the longitudinal and transverse cracks form fine flake-shaped broken blocks, indicating that the broken area of coal pillar is at the depth of 1.25 m in the coal pillar sidewall. At the depth of 1.25–3.35 m in the side of the coal pillar, the transverse and longitudinal cracks in the coal wall in the damage peep hole are significantly reduced, the number and width of cracks is less than the broken zone, indicating that the plastic zone of the coal pillar is 1.25–3.35 m. With the increase of the damaged peep depth of the coal pillar, when the depth is...
greater than 3.35 m, the coal structure of the side of the damaged borehole is complete, there are basically no longitudinal and transverse cracks. It shows that the elastic zone of the coal pillar is greater than 3.35 m, some damaged boreholes are shown in Figure 7.

The mined-out area is formed after the mining of 3107 working faces, the damage monitoring is carried out on the serious sloughing of the coal pillar on the side of the tail entry during the mining of 3108 working faces. The damage at different depths of the sloughing coal pillar is shown in Figure 8. In the range of 0–1.75 m in depths of the coal pillar sidewall, the damage to the borehole wall of the damage peeping borehole is the most serious, the fragmentation degree decreases from the orifice to the inside of the coal pillar, indicating that the range of the coal pillar fragmentation area is about 1.75 m. The damage drilling depth is from 1.75 m to 4.25 m, the number and width of longitudinal and transverse fractures in the sidewall of the hole is gradually decreasing, the fractures are mainly radial fractures. When the depth of the coal pillar is greater than 4.25 m, the cracks at the borehole side disappear. It indicates that the plastic zone of the coal pillar in the mining process of 3108 working faces is 1.75–4.25 m. The borehole depth of the sidewall is greater than 4.25 m, the coal pillar is basically in the elastic zone.

As shown in Figure 9, sloughing occurs to the mining process of the coal pillar in the two working faces. Through field monitoring, it can be seen that sloughing occurs in the coal pillar during the mining process of 3107 and 3108 working faces. When the sloughing degree and internal damage of the coal pillar during the mining process of 3107 working face are less than 3108 working face, the block rib of the coal pillar on the side of the head entry appears to crisscross, a small net pocket is formed under the action of the steel mesh support. During the mining of 3108 working faces, the sloughing of the coal pillar is serious. The surface of the coal pillar

![Figure 7](image1.png)

**Figure 7** The 3107 working face mining section coal pillar damage

![Figure 8](image2.png)

**Figure 8** The 3108 working face mining section coal pillar damage diagram
forms a small block fracture due to the longitudinal and transverse cracks. Under the action of steel mesh support, the rib appears prominent large net pocket. The roof of entry has different degrees of subsidence. When the degree of coal pillar sloughing is large, the roof subsidence and failure of entry increase at the same time, which is highly consistent with the coal pillar sloughing. However, the overall sloughing and roof subsidence damage of 3108 working face is greater than that of 3107 working face. Through field monitoring, it can be concluded that intermittent sloughing occurs to the side of coal pillar during the mining process of one side and two sides of working face.

FIGURE 9 Coal pillar local sloughing monitoring

FIGURE 10 Diagram of overlying strata structure during working face mining

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4 | PRESSURE MECHANISM ANALYSIS OF COAL PILLAR

4.1 | Load transfer mechanisms of coal pillar

With the gradual increase of mined-out area in the working face, the “O–X” periodic fracture occurs to the main roof of the working face, and the “masonry beams” structure is formed into the entry along the gob and above the working face. The main roof is a solid support side of the solid coal, the fracture line of the coal side is located in the coal wall, forming a large structure of the overlying rock mass dominated by the main roof strata. As shown in Figure 10, during the mining process of the working face, key block A, key block B, and the key block C are basically broken along the inclined top, in which the key block B collapses above the gob-side entry and the coal pillar to form an arc triangle block B, rotates downward with the fracture line above the coal seam as the axis. Along the strike, under the superposition of the advanced abutment pressure and the lateral abutment pressure, the coal mass and the immediate roof below the key block A are compressed and sunk. The key block C is compressed and sunk after contacting with the waste rock in the gob, the key block B is rotated and sunk. Therefore, during the mining process of the working face, the double “masonry beam” structure formed by the overlying strata in the toward and inclined, the load transmitted by the overlying strata through the change of the double “masonry beam” structure has a serious impact on the pressure change of the coal pillar.

There are some differences between the rock structure of the entry and the rock structure of the stope. The rock structure of the stope is broken along the toward, The rock structure of the entry is broken along the inclination. The pressure concentration and failure of the coal pillar is caused by the superposition of the advance pressure of the toward broken rock and the pressure of the inclined broken rock. The fracture of the rock is controlled by the main roof, which is the most obvious factor affecting the fracture and movement toward the rock structure. Therefore, when the caving waste rock is filled with the gob or the mining height of the working face is small, the key block C is supported by the waste rock, and a stable “masonry beams” structure is formed. Therefore, the “masonry beam” structure is essential to the pressure and sloughing of the coal pillar. The breaking position of the key block B and the hinge
degree of the key blocks A and C will directly affect the pressure state and sloughing degree of the coal pillar, as shown in Figure 11.

According to the general situation of mine geology, the Hanjiawan Coal Mine belongs to the typical geology of thick loose layer with shallow buried depth and thin bedrock. Therefore, with the mining surface of the working face, the bending subsidence zone cannot be formed, only the quasi-bending subsidence zone composed of loose layers can be formed. With the gradual increase of gob space in working face, due to the large thickness of coal seam and the surface loose layer is not easy to form a stable structure, under the action of load gravity, the original “masonry beam” structure instability moves up to form a new “masonry beam,” until the rock block C in the middle of the gob and the waste rock compaction in the caving zone are in a stable state, forming a new caving zone and the fracture zone with joint development tends to be stable. The key block B above the coal pillar side loses the horizontal extrusion and friction resistance of key block C, thus forming a cantilever beam structure. The load on it is transmitted to the coal pillar and the solid coal through the cantilever beam. The cantilever beam is supported by the coal pillar, the cantilever beam is broken and destroyed into a short cantilever beam structure under the load of the overlying strata. With the mining of the working face until the stability of the gob, the short cantilever beam structure with stepped superposition will be formed due to the influence of the lithology of the weathered rock, a new “masonry beam” structure will be formed into the upper part. The key block B acts on the short cantilever beam structure. The short cantilever block has no horizontal thrust in the direction of the gob. It is affected by the self-weight and the key block B to turn and sink in the direction of the gob. The overburden load is further transferred to the deep coal and rock mass, resulting under pressure concentration. Its rock structure is shown in Figure 12.

4.2 Pressure analysis of coal pillar

In the mining process of working face, the movement state of rock determines the distribution characteristics of rock structure and has an important influence on the deformation and pressure distribution of the coal pillar. The pressure state of the coal pillar is mainly divided into two categories, one is the force of the coal pillar on the side of the gob, the other is the force of coal pillar along the gob under the action of advanced pressure of working face. On the one hand, the pressure generated by the high loose layer disturbed by mining under gravity load is finally transferred to the coal through its lower supporting rock mass. On the other hand, the coal pillar supports the transfer pressure of the incomplete caving rock above the unilateral and bilateral gobs at the same time as the support of the rock pillar. The pressure sources of these two parts are different, they jointly act...
on the coal pillar. Under the action of the advance pressure of the working face, the pressure of the coal pillar on the side of the gob-side entry is affected by the gravity of the overlying strata and the loose layer above the coal pillar. On the other hand, the advance pressure formed by the mining of the working face is transmitted to the coal pillar of the overlying strata, and jointly acts on the middle section coal pillar.

4.2.1 Pressure analysis of coal pillar on gob side

The pressure of the coal pillar on the gob side section is mainly derived from the gravity of the high loose layer and the transfer pressure of the completely collapsed rock layer. The solid coal area and its bedded rock that has not yet been mined are regarded as rigid bodies, the coal pillar is simplified as an equivalent spring, the high loose layer bedded rock at the working face is regarded as the uniform load on the rigid rock at both ends.

According to the coal pillar and its overlying strata structure in Figure 12, the pressure analysis of the unit length coal pillar along the working face is carried out. Among them, \( F_g \) is the gravity pressure of the overlying strata in the coal pillar, \( F_d \) is the friction force on the coal pillar during the subsidence of the roof strata, \( F_b \) is the force of the short cantilever beam structure on the coal pillar, \( F_n \) is the horizontal force on the coal pillar after the compaction of the waste rock in the lateral gob of the coal pillar. The above forces directly affect the coal pillar. The pressure state of the coal pillar is shown in Figure 13.

\( F_g \) and \( F_d \) are the fixed mine pressure of the roof to the coal pillar, the pressure do not change from the movement toward the mining face, \( F_g = \gamma H \), \( \gamma \) is the volume force of the overlying strata, \( H \) is the thickness of the strata; \( f_d = \tan \varphi \), \( \varphi \) which is the internal friction angle of coal. \( F_b \) is the horizontal force on the coal pillar after the compaction of the waste rock in the lateral gob of the coal pillar. The above forces directly affect the coal pillar. The pressure state of the coal pillar is shown in Figure 13.

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\[ F_i = \left( H_1 \cot \alpha + \frac{L}{2}\right) \gamma H_i, \]  

where \( F_i \) is the concentrated force generated at the fulcrum of the overlying loose bed erosion rock; \( L \) is the length of working face, m; \( H_1 \) is upper loose layer thickness, m; \( H_i \) is the height of a loose layer from coal seam floor, m; \( \alpha \) is the angle of rock movement.

(2) Load of key block B on the short cantilever beam

The “hinged rock beams” structure of the coal pillar along the empty side mainly applies concentrated pressure to the short cantilever beam below the inclined block B. Based on this, the mechanical model of the key block B is established, which is shown in Figure 15.

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**Figure 13** Coal pillar pressure model. (A) Unilateral mining force model, (B) bilateral mining force model—no superposition, (C) bilateral mining force model—no pressure core, and (D) bilateral mining force model—superposition.
In Figure 15, $F_i$ is the uniform load acting on the loose layer to the key block B; $P_b$ is the weight of rock block B; $F_{AB}$ is the support force of the lower short cantilever beam; $T_{CB}$ and $T_{AB}$ are the horizontal thrust between the key blocks; $f$ friction at the hinge; $\theta$ is the horizontal angle of key block B; $\Delta s$ is the subsidence of key block B; $L_B$ is the breaking length of the key block.

According to the equilibrium conditions, the vertical force $F_{AB}$ of hinged key block B is:

$$F_{AB} = \frac{F_i L_B \sin \theta + P_b (L_B + h \tan \theta)}{2L_B + 2h \tan \theta},$$

(2)

$L_B$ is the lateral fracture length of the key block B, which can be given as follows:

$$L_B = \frac{2L'}{17} \sqrt{\left(\frac{10L'}{L} \right)^2 + 102 - 10 \frac{L'}{L}}.$$

(3)

In the formula, $L$ is the length of the working face, $L = 265m$; $L'$ is the fracture length of the key block B along the advancing direction, which can be regarded as the periodic weighting step. Based on the field data, $L' = 16m$.

(3) Concentrated load of short cantilever beams.

The mechanical model of the short cantilever beam is shown in Figure 16. In the figure, $P_i$ is the weight of the broken rock mass of the main roof, $P_2-P_n$ is the load of each rock layer that is coordinated with the main roof. $l_m$ is the length of the fracture line on the coal pillar, $l_1$ is the length of the broken rock block of the first layered main roof, $l_2-l_n$ is the length of the strata with the coordinated movement of the main roof; $h_1$ is the thickness of the first layered main roof fault block, $h_2-h_n$ is the incremental thickness of the overly caving zone; $\beta$ is rock breaking angle; $\theta$ is the horizontal angle of the short cantilever beam; $F_{AB}$ is the downward force of hinged rock block B.

According to the pressure of the roof, the torque balance at point A can be obtained as follows:

$$F_{B} l_m = P_1 \left(\frac{1}{2} l_1 + \frac{1}{2} h_1 \cot \beta\right) + P_2 \left(\frac{1}{2} l_2 + \frac{1}{2} h_2 \cot \beta + h_1 \cot \beta\right) + \cdots + P_n \left(\frac{1}{2} l_n + \frac{1}{2} h_n \cot \beta + \sum_{j=1}^{n-1} h_j \cot \beta\right) + F_{AB} \left(\frac{1}{2} l_n + \sum_{j=1}^{n} h_j \cot \beta \cos \theta\right).$$

(4)

In the above formula, i serial numbers for each short cantilever rock. The concentrated load of short cantilever beam in coal pillar is

$$F_B = \frac{1}{2l_m} \left[\sum_{i=1}^{n} P_1 l_i + \sum_{i=1}^{n} P_2 h_i \cot \beta + 2 \sum_{j=2}^{n} \sum_{i=1}^{n-j} P_j h_i \cot \beta + F_{AB} \left(l_n + \sum_{j=1}^{n} h_j \cot \beta \cos \theta\right)\right].$$

(5)
It can be seen from the above formula that the concentrated load at point B is related to the fracture angle, the horizontal rotation angle of the rock stratum, the geometric shape of the rock stratum, the self-weight of the rock stratum and the load of the high rock stratum. Therefore, the failure height of the coating rock on the mining field has a great influence on the pressure of the coal pillar in the mining process of the thick coal seam. Combined with the mechanical model, it can be obtained that when the 3107 working surface is mined, the coal pillar on the side of the gob side of the head entry is subject to the concentrated load of the short cantilever beam, the stress of the key block B, the force of the overlying loose layer, the stress of the coal pillar on the side of the head entry will be greater than that of the tail entry side, the coal pillar on the side of the tail entry will only be stress transfer because of the roadway excavating, the force of the overlying loose layer on it, there is no short cantilever beam concentrated load, the key block B transmits stress on it. According to the coal pillar on-site stress monitoring Figure 4, it can be obtained that the stress of the coal pillar on the side of the head entry is greater than the stress of the coal pillar on the side of the tail entry when unilateral mining, the stress distribution on both sides of the coal pillar are similar. This paper shows that the mechanical model of the coal pillar constructed in this section is reasonable, which fully reveals the similarity of the stress distribution and the difference of stress on both sides of the coal pillar at the site, determines the mechanism of the change of the stress of the coal pillar on the gob side.

4.2.2 Pressure analysis of coal pillar along the gob-side entry

During the mining process of the coal pillar in the side of the gob-side entry, the overlying strata showed “O–X” fracture along the working face, the advance pressures to increase area in front of the coal wall of the working face and the pressure decrease area in the gob behind the working face. Therefore, the pressure of the coal pillar in the gob-side entry mainly comes from the gravitating rock above the coal pillar and the advanced pressure superposition formed by the mining of the working face. The gravity of the rock above the coal pillar depends on the burial depth of the coal seam and the bulk density of the gravitating rock. The influence of the advanced pressure on the coal pillar mainly depends on the mining height, the lithology of the gravitating rock, the gob and the burial depth, which further affects the advanced pressure formed by the breaking structure of the gravitating rock.

The plane diagram of the advanced pressure of coal pillar is shown in Figure 17.

The mechanical model of coal pillar along gob-side entry under the action of advanced pressure in the mining process of working face is shown in Figure 18. $L$ is the length of the coal pillar along the toward of the working face, $M$ is the height of the coal pillar, namely, the thickness of the coal seam, $F$ is the support force of the floor to the coal pillar, $F_g$ is the gravity of the overlying strata on the coal pillar, $F_c$ is the advance pressure acting on the coal pillar in the mining process of the working face.

The mechanical equilibrium can be obtained at 0 point of $y$ axis

$$
\sum F_y = 0.
$$

According to the mechanical model of coal pillar in the side section of gob-side entry, it can be obtained:

$$
F - F_g - F_c = 0,
$$

$$
F = F_g + F_c = BLyH + K(BLH)y.
$$

$B$ is the width of coal pillar, $m$; $L$ is the toward length of coal pillar in advance pressure section, $m$; $H$ is the depth of coal pillar, $m$; $y$ is the average volume force of overlying strata, kN/m$^3$; $K$ is the pressure concentration factor, 1.6–4.

Through the analysis of the mechanical model of the coal pillar along the gob side entry, it can be obtained that the pressure $F_g$ of the rock layer on coal pillar is not related to the mining of the working face, but depends on the buried depth of the coal pillar and the lithology of the rock layer on
the coal pillar, it is a constant value of the calculation process. The advance pressure \( F_c \) on the coal pillar in the mining process of the working face is closely related to the lithology of the coating rock, the mining height of the working face, and the internal friction angle. Along the working face, the coal wall, the hydraulic support, and the gob form the masonry beams structure in front of the working face. The length of the key block \( B \) in the masonry beams and the position of the fracture line in front of the coal wall forms different advance pressures, ultimately form different coating rock fracture structures that affect the pressure of the coal pillar.

According to the analysis of the mechanical model of the coal pillar along the empty alley side, when the coal pillar along the empty alley side is smaller than the working surface distance, the coal column is not only subjected to the stress of the inclined overburden structure, but also by the advanced stress of the working surface toward the overburden structure. Therefore, when the 3107 working surface is mined, the coal pillar stress along the empty alley side near the working surface is greater than the coal pillar stress on the side of the gob area, the 3108 working surface is mined, the coal pillar stress along the empty alley side is greater than the stress when the 3107 working surface is mined. Combined with the on-site monitoring results of the stress of coal pillars in Figures 5 and 6, it can be seen that the stress changes curve, stress statistics, and the analyzing results of the constructed mechanical model of the coal pillar are consistent when the 3108 working surface is mined, the rationality of the mechanical model of the coal pillar along the empty side is verified by the on-site stress results, the stress mechanism of the coal pillar along the tail entry is revealed.

### 4.3 Pressure state of coal pillar

When the width of the coal pillar is large enough, according to the elastic–plastic mechanics model, pressure broken zone, plastic zone, elastic zone, and original rock pressure zone can be formed in the coal pillar. The pressure state of the coal seam in the section can be simplified as plane strain treatment. It is approximately considered that the coal–rock mass is an isotropic and homogeneous continuous medium, the coal–rock mass conforms to the ideal elastic-plastic mechanical model. The mechanical model of the coal pillar is shown in Figure 19A. When the section \( x_1 \) is the broken zone, \( x_2 \) is the plastic zone, \( x_0 \) is the limit equilibrium zone, \( x_3 \) is the elastic zone; \( k\gamma H \) is the peak pressure increment on the interface between section coal pillar and roof; \( q_1 \) is the load of caving rock above the coal pillar.

As shown in Figure 19A, the pressure of semi-infinite section coal pillar under concentrated force is derived from elastic–plastic mechanics, the pressure component at any point \( M(x, y) \) of section coal pillar along the gob side under the action of upward load can be obtained by superposition principle. The microsegment \( dF \) is taken at \( x = \xi \), the microconcentrated forces \( dF = qd\xi \) within the microsegment range. The pressure component at point \( M(x, y) \) is as follows:

\[
\begin{align*}
\sigma_x &= -\frac{2q_1(x)d\xi}{\pi} \cdot \frac{x^2y}{(x^2 + y^2)^2} \\
\sigma_y &= -\frac{2q_1(x)d\xi}{\pi} \cdot \frac{y^3}{(x^2 + y^2)^2} \\
\sigma_{xy} &= -\frac{2q_1(x)d\xi}{\pi} \cdot \frac{x^2y}{(x^2 + y^2)^2}
\end{align*}
\]

When the mining height of a thick coal seam or working face is large, the main pressure state of the coal pillar in the actual section can be divided into the following two kinds:

1. Pressure superposition type of an elastic zone in coal pillar

---

**Figure 19** Coal pillar pressure mechanics model. (A) Mechanical model without pressure superposition, (B) mechanical Model of Part Superposition of pressure, and (C) mechanical model of entire pressure superposition.
At $x_0 < \frac{1}{2}x$, time, there is a pressure superposition zone in the middle of the coal pillar. The pressure superposition mechanical model of the elastic zone of the wide coal pillar is established, as shown in Figure 19B. The load borne by the coal pillar in the pressure superposition zone is $Q = \sum_{i=1}^{n} \frac{E_ih_i}{\rho_i}$, the coordinate of the left intersection of the elastic zone is $(\frac{q-b_2}{k_2}, Q)$.

The three-dimensional pressure expression of any point in the coal pillar can be obtained from the above formula (9):

$$
\begin{align*}
\sigma_x &= \Delta \sigma_{x_1} + \Delta \sigma_{x_2} - \gamma H = \int_{x_0}^{x} \frac{2(k_1 \xi + b_1) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi + \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \\
\sigma_y &= \Delta \sigma_{y_1} + \Delta \sigma_{y_2} - \gamma H = \int_{x_0}^{x} \frac{2(k_1 \xi + b_1) \cdot y \cdot (x - \xi)^3}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi + \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^3}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \\
\tau_{xy} &= \Delta \tau_{(x) \xi} + \Delta \tau_{(y) \xi} - \gamma H = \int_{x_0}^{x} \frac{2(k_1 \xi + b_1) \cdot y \cdot (x - \xi)^3}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi + \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^3}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \\
&\quad + \left. \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \right| \frac{x}{2} - \frac{Q-b_2}{k_2} - \gamma H \\
&+ \left. \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \right| \frac{x}{2} - \frac{Q-b_2}{k_2} - \gamma H \\
&+ \left. \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \right| \frac{x}{2} - \frac{Q-b_2}{k_2} - \gamma H \\
&+ \left. \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \right| \frac{x}{2} - \frac{Q-b_2}{k_2} - \gamma H \\
&+ \left. \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \right| \frac{x}{2} - \frac{Q-b_2}{k_2} - \gamma H \\
&+ \left. \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) \cdot y \cdot (x - \xi)^2}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi \right| \frac{x}{2} - \frac{Q-b_2}{k_2} - \gamma H
\end{align*}
$$

Among them, $k_1 = \frac{H - \lambda \alpha}{x_0}$, $k_2 = \frac{H(1.1 - K)}{x_0}$, $b_1 = q_1 - \lambda H$, $b_2 = K H - \frac{H}{x_0} - \frac{H}{x_0} (1.1 - K)$.

$$x_0 = \frac{M(1 - \sin \varphi)}{2 \tan \varphi (1 + \sin \varphi)} \left[ \frac{Ky H + C / \tan \varphi}{(C / \tan \varphi + P_2 \cdot \frac{1 + \sin \varphi}{1 - \sin \varphi})} \right].$$

In the formula, $M$ is the mining height, $m$; $C$ is the cohesion of coal, MPa; $\varphi$ is the internal friction angle, $^\circ$; $K$ is the pressure concentration coefficient; $\lambda$ is the lateral pressure coefficient; $H$ is the depth of entry, $m$; $P_2$ is support strength, MPa.

(2) Peak pressures superposition type of coal pillar.

When $x < 2x_0$, the peak pressure on both sides of the coal pillar is superimposed, and the pressure distribution of the coal pillar is shown in Figure 13D. Based on this, the mechanical model of peak pressure superposition of the section coal pillar is established in Figure 19C. The coal pillar is in the limit equilibrium zone, the coal pillar is only affected by vertical pressure.

Similarly, the vertical pressure expression of any point in the coal pillar can be obtained by formula (9):

$$
\sigma_y = \Delta \sigma_{y_1} + \Delta \sigma_{y_2} - \gamma H = \int_{0}^{x} \frac{2(k_1 \xi + b_1) y^3}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi + \int_{x_0}^{\frac{q-b_2}{k_2}} \frac{2(k_2 \xi + b_2) y^3}{\pi [(x - \xi)^2 + y^2]^2} \, d\xi
$$

Among them, $k_1 = \frac{H - \lambda \alpha}{x_0}$, $k_2 = \frac{H(1.1 - K)}{x_0}$, $b_1 = q_1 - \lambda H$, $b_2 = \lambda H (2K + 1) - q_1$.

The above calculus equation can be solved by MATLAB, the calculation process will not be repeated here because the final expansion of the analytical formula is long. Through the above mechanical model and calculation process, it can be seen that the pressure of any point in the coal pillar is related to the physical and mechanical properties and occurrence conditions of the coal seam, it is mainly affected by the size of the coal pillar, the pressure concentration coefficient of the overlying strata and the rock load of the caving zone.

In summary, according to the mechanical model of the stress state of the coal pillar, the stress of the coal pillar tends to show a zoning phenomenon, when the width of the coal pillar is large or the one-sided working surface is mined, as shown in Figure 19A, there is no stress superposition and elastic region superposition, the model theory calculates the maximum of 6.2 MPa. When the width of the coal pillar is small or double-sided mining, as shown in Figure 19B,C, the superposition of elastic regions and the peak stress of the coal pillar occurs, the theoretical calculation of the model are 10.3 and 12.8 MPa, respectively. Combined with the
change of the stress of the coal pillar on-site monitoring in the second chapter, the maximum stress of the coal pillar tends to be 6.8 MPa when the 3107 working surface is mined on one side, and the degree of damage to the coal pillar is small. When the 3108 working surface is mined, the maximum stress of the coal pillar stress monitoring site is 10.5 MPa, indicating that the elastic region superposition and stress superposition phenomenon of the coal pillar has not reached the peak stress superposition. According to the comparative analysis of the calculation of the stress state theoretical model and the stress of the on-site monitoring coal pillar, it is explained that the theoretical model of stress distribution of the inclined coal pillar is reasonable and can reveal the true stress state of the coal pillar.

5 | COAL PILLAR SLOUGHING ANALYSIS

During the mining process of the working face, the sloughing degree and internal damage of the coal pillar, the pressure of the coal pillar, the support resistance of the working face, and the periodic weighting step were monitored on-site. There are changes in the degree and range of coal pillar sloughing. The fragmentation degree of the coal pillar is large, the internal damage and pressure peak value of the coal pillar increase, and the unloading pressure in the broken zone of the side decreases significantly. Through the monitoring of the support resistance of the working face and the calculation of the periodic weighting step distance, it is known that the periodic weighting step distance varies from 16 to 23 m. The smaller the periodic weighting step distance is, the smaller the peak value of the support resistance of the working face is, the larger the periodic weighting step distance is, the greater the peak value of the support resistance of the working face is. When the periodic weighting step distance and the support resistance of the working face are large, the coal pillar sloughing along the side section of the entry is serious.

Through the on-site monitoring of the sloughing degree, pressure, support resistance, and periodic weighting interval of the coal pillar, combined with the analysis of the mechanical model of the coal pillar on the gob side and the gob-side entry side of the coal pillar in the mining process of the working face, it can be seen that the direct influencing factors of the different sloughing degree of the coal pillar are the difference of the working face toward between inclined breaking coating rock structure. The breaking length of the key block B of the toward and inclined breaking coating rock, the position of the breaking line of the key block A and B of the coating rock on the coal pillar, so the pressure and load directly affects the sloughing of the coal pillar are the length of the key block B in the masonry beam and the position of the breaking line.

**FIGURE 20**  Position of key block B breaking line in masonry beam. (A) Position of key block B breaking line above the solid coal seam. (B) Position of key block B breaking line above the entry. (C) Position of key block B breaking line above coal pillar. (D) Position of key block B breaking line above gob.
In the process of working face mining, the breaking length and breaking line position of key blocks along the entry side and the square rock on the working face are different, which makes the pressure acting on the coal pillar change, finally leading to destruction and sloughing of the coal pillar. Figure 20A shows the broken direct roof fully fills the gob, the key block breaking line occurs to the solid coal seam, and the key block breaking length is large. At this time, the coal pillar sloughing is small, the pressure is mainly concentrated in the key blocks supported by solid coal and waste rock. It can be seen from Figure 20B,C that when the filling of gob with a broken immediate roof is poor, the fracture line of key block occurs above the entry and section coal pillar. The fracture length of key block is small, the transfer pressure of overlying strata is mainly concentrated on the coal pillar, so the sloughing is the most serious at this time. It can be seen from Figure 20D that the key block broken in Figure 20 along the entry side and the working face, the section coal pillar appears uneven and sloughing.

Through the above analysis, it can be seen that the broken lines’ position and key block length of the key block directly affect the degree of coal pillar sloughing and the pressure. Therefore, to determine the quantitative influence of the key block of the masonry beam on the coal pillar, the key influencing factors of the key block B are analyzed. The rotation subsidence model of the key block B is shown in Figure 21.

According to the state of key block B of the masonry beam after rotation of Figure 21, it can be obtained:

$$T(h - a - \Delta) = \frac{1}{2}ql^2.$$  \hspace{1cm} (12)

Considering the pressure state of the key block contact point, the action point of horizontal pressure $T$ is $a/2$, $\Delta$ approximately equal to $\Delta = l \sin \alpha$

$$a = \frac{1}{2}(h - l \sin \alpha), \hspace{1cm} (13)$$

$$T = \frac{ql^2}{h - l \sin \alpha}. \hspace{1cm} (14)$$

The contact pressure of the key block is $\sigma_p$

$$\sigma_p = \frac{T}{a} = \frac{2ql^2}{(h - \sin \alpha)^2}. \hspace{1cm} (15)$$

It is assumed that the ratio of compressive strength $[\sigma_c]$ to pressure strength $\sigma_p$ is $K$, therefore, the allowable load $q$ is

$$q = \frac{(l - \sin \alpha)^2[\sigma_c]}{2l^2K}. \hspace{1cm} (16)$$

When the beam breaks (reaching the limit span), the relationship between load $q$ and tensile strength $\sigma_t$ of rock beam is as follows:

$$\sigma_t = Kq \frac{6l^2}{h^2}, \hspace{1cm} (17)$$

where $K$ is generally taken as $1/2$–$1/3$, thus

$$q = \frac{\sigma_t h^2}{6Kl^2}. \hspace{1cm} (18)$$

In general rock, the ratio of compressive strength $[\sigma_c]$ to tensile strength $\sigma_t$ is $n$, so

$$\sigma_t = [\sigma_c]/n, \hspace{1cm} (19)$$

---

**Figure 21** Subsidence model of key block B in masonry beam
\sin \alpha = \frac{h}{l} \left( 1 - \frac{K}{3nK} \right)^\frac{1}{2} \quad (20)

And \( \Delta = l \sin \alpha \), so

\Delta = h \left( 1 - \frac{K}{3nK} \right) \quad (21)

Therefore, it can be obtained that when the middle subsidence reached \( \Delta \) after the rock beams break and joined each other, deformation and instability of the rock block structure, as shown in Figure 22D, the whole main roof structure was cut down along the coal wall, therefore, the degree of concentrated force in some areas of entry was greatly reduced, deformation and failure tend to be eased.

After the coal seam was mined, the immediate roof strata collapsed and fill the gob, the gap \( \Delta_1 \) between the filling and main roof is:

\[ \Delta_1 = \sum h + M - Kp \sum h = M - \sum h (Kp - 1), \quad (22) \]

where \( \sum h \) is the immediate roof thickness, \( m \); \( M \) is the coal seam thickness, \( m \); \( Kp \) is the rock expansion coefficient, taking 1.4.

So when \( \Delta_1 > \Delta \), the main roof subsidence is greater than the limit subsidence, there is the possibility of deformation and instability.

From the above analysis, the breakage length of the key block of the overburden above, the breakage line position, and the sinking amount of the key block determine the degree of coal pillar sloughing. When the breakage length of the key block is larger than the width of the coal pillar, \( \Delta_1 < \Delta \), the breakage line position is in Figure 20A–C, the extent of coal pillar sloughing is less, the overburden above the coal pillar forms the intersection structure, the vertical stress transfer above is interrupted. When the position of the broken line is in Figure 20D, \( \Delta_1 > \Delta \), the overlying rocks have not formed the articulated structure to completely unload the pressure, at this time, the coal pillar of the section is not affected basically does not occur sloughing, this situation is the best solution for coal pillar sloughing control. This is the best solution for the control of coal pillar sloughing. The control of sloughing caused by hydraulic pressure on site should follow this principle, the pre-fracture of overlying rock structure above the coal pillar in the section to unload pressure, so as to control the breaking length of the key block, breaking line position, and sinking amount of key block, then achieve the control of coal pillar sloughing.

6 | FIELD PRACTICE OF SLOUGHING CONTROL SCHEME

According to the analysis of coal pillar pressure and sloughing mechanism on the gob side and entry side, the underground sloughing control scheme is formulated. During the mining process of 3107 working faces and 3108 working faces, hydraulic fracturing relief pressure was carried out on the roof of the side coal pillar of gob-side entry and the gap roof of hydraulic support in the working face, respectively. The hydraulic fracturing relief pressure to the roof of gob-side entry can reduce the influence of erosion rock breaking for the direction of

![Figure 22](image)
working face on coal pillar, the hydraulic fracturing relief pressure to hydraulic support frames can reduce the influence of advance pressure formed by erosion rock breaking for the direction of working face on coal pillar. The hydraulic fracturing relief pressure to the side roof of the gob-side entry adopts the continuous hydraulic fracturing relief pressure hole. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure. The hydraulic fracturing relief pressure hole is arranged in a single row along the toward, 30 cm from the coal pillar, the distance between the two hydraulic fracturing relief pressure holes is 5 m. The hole spacing mainly depends on the radius formed by the hydraulic fracturing relief pressure.

According to the hydraulic fracturing pressure relief scheme for along the gob side and the roof of the working face, the roof is continuous drilling for the entry side, and the hydraulic fracturing pressure relief is carried out on the roof of the 30 m range of the advanced working face and the lagging working face, the fracture length, and the position of the fracture line of the upper inclined rock are controlled, the pressure of the coal pillar is reduced by changing the structure of the rock. The roof of the working face adopts the interval hydraulic fracturing pressure relief to change the length of the upper strata along the toward and the breaking line position of the key block of the masonry beam, the breaking position of the upper strata is behind the hydraulic support to form a short cantilever beam structure, reduce the influence of the advanced pressure caused by the mining of the working face on the coal pillar. Through the hydraulic fracturing and pressure relief of the overlying strata on the inclined and toward, the structure of the overlying strata breaking section and the position of the breaking line is changed, and the influence on the pressure of the coal pillar is reduced, thus effectively controlled the sloughing of the coal pillar.

According to the hydraulic fracturing pressure relief control scheme, Figure 23 is the sloughing condition after the coal pillar treatment, the specific construction process of hydraulic fracturing pressure relief is no longer described here. Through the roof hydraulic fracturing pressure relief can be obtained, during the mining process of 3107 working face, continuous monitoring of coal pillar 1000 m sloughing damage, section coal pillar overall sloughing degree is significantly reduced, the net pocket

![Coal pillar hydraulic fracturing pressure relief treatment](image-url)
and the original sloughing phenomenon is completely eliminated. The mined-out area is formed after the mining of 3107 working faces. During the mining of 3108 working faces, the damage degree of coal pillar sides is significantly greater than that of one side. The support scheme of section coal pillar sides remains unchanged, but the uneven sloughing phenomenon disappears after hydraulic fracturing and pressure relief, which is significantly smaller than the overall sloughing degree before treatment. Through the on-site monitoring of the coal pillar sloughing during the mining of one side and two sides of the working face, the sloughing of the coal pillar is well controlled. It shows that the hydraulic fracturing pressure relief changes the breaking length and the position of the breaking line of the overlying strata structure, then changes the pressure of the overlying strata on the coal pillar, which plays a good control in the sloughing phenomenon in the underground. Similarly, the hydraulic fracturing pressure relief control scheme has achieved good results in the management of mine sloughing and disasters in similar geological conditions in the northern Shaanxi mining area, which will not be repeated here.

7 | CONCLUSIONS

(1) When the overburden of the coal pillar is easy to form the overburden structure of “short cantilever beam + masonry beam + overburden loose layer,” the location and breakage length of the key block of the overburden masonry are the main causes of the coal pillar sloughing, the mechanism of the coal pillar sloughing studied in this paper is mainly for the thick and huge thick coal seam in the northern Shaanxi mining area, it is not applicable if the general coal seam does not form such overburden structure.

(2) By analyzing the breaking conditions, breaking line position, and breaking degree of key block B in the overlying rock structure, it is known that $\Delta_1 \geq \Delta$ at that time, key blocks B are easy to break and deformation slip destabilization. According to the location of the breaking line and the breaking length key block B can be divided into four cases. When the breaking line is located above the mining area, it has the least influence on the coal pillar sloughing. On the basis of this study, we can analyze the stress transfer mechanism between overburdened rocks and study the mechanism and control method of coal pillar sloughing in the section under different storage conditions in the future.

(3) It has made a plan to control the sloughing of coal pillar, and carried out hydraulic cracking to unload the top plate along the side roof plate of the empty lane and the gap roof plate of the hydraulic support at the working face, respectively, which changed the position and length of the key block breaking line in the overlying rock layer, effectively controlling the sloughing of coal pillar and significantly reducing the degree of sloughing.

(4) The study on the mechanism and control of the sloughing provides ideas for the stress change and disaster management of the coal pillar under different geological conditions, different mining methods, and different retrieval processes. The overlying rock structures and breakage form of the coal pillar have a key influence on it, the stress of the coal pillar can be reduced and a disaster contingency plan can be made by changing the breaking position and breakage length of the key block of the overlying rock of the coal pillar.

ORCID
Jianjun Wu http://orcid.org/0000-0003-0310-6655

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