Stress characteristics and control of coal and rock in steeply dipping coal seam mining under the goaf of a close coal seam group: a case study

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Abstract. Steeply dipping coal seams, which are widespread in western China, are considered to have complex mining conditions. The development of such a seam plays an important role in regional economic development. Some steeply dipping coal seams are distributed at close distances. For closely located coal seam groups, China generally adopts the downward mining method for mining. Aiming at the problems of the appearance of ground pressure such as coal wall spalling and roadway squeezing deformation of a fully inclined working face under a goaf in a close coal seam group, the mining of the 5th and 4th coal at the steeply dipping coal seam in Panbei Mine was established as the engineering background. The mechanical model of the coal zone plastic area in front of the 12124 working face under the goaf and the distribution characteristics of abutment pressure in front of the coal wall were obtained. The working surface layout and coal rock physical and mechanical parameters were derived based on field observation records and indoor rock mechanics experiments. A numerical analysis model based on the fast Lagrangian analysis of Continua 3D software was constructed, and the surrounding rock mechanics during the 12124 working face mining process was simulated. The stress distribution characteristics of the front and back of the 12124 working face under the goaf were obtained, magnitude and location of the stress peak were determined, and superposition mechanism of the support pressure evolution of the fully mechanized thick seam under the goaf was revealed. Through field measurement of the distribution of the working resistance of the supports, the maximum working resistance, position of the supports, and accuracy of the theoretical calculation and numerical simulation were verified within the error range. Furthermore, stability control technology of the surrounding rock of the stope, roof falling, and roadway under this condition was proposed and implemented. Good economic and technical effects were obtained, which ensured safe mining at these working faces.

1. Introduction
On the basis of research and experience, the mining of steeply dipping coal seams (SDCS) (35°–55°) is internationally recognized to present major problems in the mining industry [1]. In Sichuan, Chongqing, Yunnan, Xinjiang, Guizhou, Gansu, Ningxia, and other provinces in western China, SDCS with complex occurrence conditions is the main coal seam in many mine areas. In Shandong, Anhui,
Shanxi, and other provinces located in central and eastern China, some mining areas have conducted high-intensity mining in the early stage, leading to the exhaustion of coal resources that have simple occurrence conditions and can be mined easily. The SDCS and other complex occurrence conditions are being gradually mined to realize the effective utilization of coal resources and the sustainable development of the mining area economy [2].

After years of theoretical research and mining practice verification, research on ground control theory and technical system related to fully mechanized coal mining with a steeply dipping working face under single coal seam conditions has become relatively perfect [3-6]. These studies found that the ground pressure behavior of single SDCS shows typical characteristics of irregularity and asymmetry. The main reason for this finding is that the rock block formed after the roof caving in the stope will continue to slide down along the floor until it fills the lower and middle spaces of the working face. Affected by the non-uniform filling of gangue and the decomposition of the overburden strata’s own weight, the abutment pressure of the stope exhibits a spoon-like asymmetric stress arch shape.

However, as a result of the multiphase geological structure, many mining areas contain two or more layers of SDCSs located close to each other [7]. In multiple-seam mining, the floor of the upper working face (the roof of the lower working face) is damaged due to mining, resulting in a large difference between the structure of the overlying rock strata and the spatiotemporal evolution of mining-induced stress during mining of the lower working face [8-9]. Therefore, studying the law of the ground pressure behavior in SDCS mining under the goaf of a close coal seam group is of great practical significance.

In recent years, many scholars and mine technicians have conducted extensive research on the law of ground pressure behavior manifestation and ground control theory in the mining of SDCS mining of a close coal seam group and achieved fruitful research results. Wu [10-11] discovered that the stress in surrounding rock and the migration characteristics of the SDCS mining of a close coal seam group are unusual. The disturbance of mining in the upper working face will change the stability of the “R-S-F” system in the lower working face and affect the normal mining of the face. Wu summarized the characteristics of roof failure in this type of stope, and the formation and evolution characteristics of the asymmetric low-level key strata structure of the roof of this type of stope are summarized. Periodic instability directly affects the stability of overlying strata, coal pillars, and supporting systems in the lower working face. Ren [12] studied the evolution law of surrounding rock stress, structural instability, and disaster mechanism in the working face of the SDCS mining of a close coal seam group. Wei [13] found the effect of working face position on the stress distribution and strata failure in surrounding rock of the SDCS mining of a close coal seam group and determined the arrangement of the working face with minimum mining disturbance. Chi [14] studied the law of roadway damage and the influence of the abutment pressure of the upper goaf on the roadway when rotary mining is adopted in the working face of the SDCS mining of a close coal seam group and analyzed the mechanism of the surrounding rock damage of the rotary mining roadway. Zhao [15] studied the reasonable layout of mining yards in the joint mining of thin coal seams with large inclination and analyzed the adaptability, stability, and technical requirements of fully mechanized mining equipment under this mining condition. Yang et al. [16-17] studied the supporting pressure superimposition and evolution mechanism during downward mining of a coal seam group with large inclination and close distance in the Huainain mining area. They then presented countermeasures for the mining pressure of the working face, studied the failure mechanism of the stope floor, and obtained the mining of the overlying working face during the process, the stress evolution law of the soleplate, and the maximum destruction depth and position of the floor.

On the basis of the above research results, current research mainly focuses on the layout of the working face of the SDCS mining of a close coal seam group, the law of overburden migration, and the adaptability of the support equipment. The failure characteristics of coal and rock masses and the behavior of ground pressure in the SDCS mining under the goaf of a close coal seam group still need to be understood further.
The main objectives of this study are to provide a strong basis for the development of mining pressure control measures in the lower working face and provide guiding significance to enable safe and efficient mining under similar conditions. The Panbei Mine’s fully mechanized coal mining face under a thick, inclined coal seam is used as the background. This paper uses theoretical analysis, numerical simulation, and on-site measured data analysis to reveal the characteristics of coal and rock mass destruction and the law of ground pressure behavior, and it proposes ground pressure control measures and technology to prevent roof falling and rib spalling.

2. Project overview

2.1. Geological conditions of the mine

The 12124 working face of Panbei Mine has a face length of 150 m and a total advancing length of 884 m. The elevation of the No. 4 coal seam is −519 m to −396 m. The coal seam thickness is 3.8 m with an average inclination angle of 38°. The elevation of the tail entry is −398 m, and the elevation of the head entry is −469 m. The roof is a sand–mudstone interlayer composite stratum with a thickness of 6.8–7.5 m. The phenomenon of mud cementation is obvious, and the bedding and fissures are developed. The hardness coefficient \( f \) is 0.9–1.5; the immediate floor is sandy mudstone with a thickness of 1.0–3.1 m. The hardness coefficient \( f \) of coal seam is 0.3–0.5, which is relatively soft.

The goaf of the 12125 working face is located above the 12124 working face at a vertical distance of 19.6 m, and it is parallel to the 12124 working face in the strike direction. The horizontal distance between the tail entry of the 12124 working face and the coal pillar of the 12125 goaf is 60 m on average, and the middle and lower parts of the 12124 working face are completely below the 12125 goaf. The spatial positions of the two working faces are shown in Figure 1.

![Figure 1 Layout of two working faces](image)

2.2. Characteristics of ground pressure behavior in the 12124 working face

Multiple-seam mining leads to overlapping of the effects of coal extraction, increasing the severity of ground pressure. After the upper coal seam is mined, the primary rock stress field is redistributed, which disturbs the primary rock stress state of the lower coal seam.

As a result, the 12124 working face in the lower part of the 12125 goaf and the roof of the recovery roadway are soft and broken, thereby posing great difficulties to the surrounding rock control, roadway support, and maintenance of the 12124 working face. This condition causes the occurrence of rock pressure problems such as coal wall flank, roof fall, and U-steel shed leg breakage in the working face.
3. Establishment of mechanics and numerical model

3.1. Mechanical model

3.1.1. Influence of abutment pressure on the failure of coal wall and rock mass

After the coal seam is excavated, the initial equilibrium state of the stope is broken, and the primary rock stress is redistributed. At this time, the abutment pressure in front of the coal body is several times greater than the primary rock stress. With the advance of the working face, the abutment pressure gradually transfers to the deep part of the coal body. In a certain range, the front abutment pressure and the bearing capacity of the coal body are in a state of limit equilibrium. The coal body is repeatedly loaded by the mining-induced pressure; as a result, a plastic failure area is formed near the coal wall, while the deep part of the coal body is in the elastic state and the primary rock stress state. The distribution of the deformation area in front of the coal body is shown in Figure 3.

![Figure 2 Ground pressure behavior of the 12124 working face: (a) rib spalling; (b) roadway support failure (c) rib spalling statistics](image)
In the figure, zone ① is the failure area, zone ② is the plastic area, zone ③ is the elastic area, and zone ④ is the primary rock stress area.

In the process of coal mining, rib spalling occurs easily in the plastic area of the coal wall. A large width of the plastic zone corresponds to a high probability of spalling. The width of the plastic zone is deduced by von Mises criterion as follows:[18]:

\[
L = \frac{M}{2\xi f} \ln \left( \frac{K\gamma H + \frac{C_0}{\tan \varphi_0}}{\xi (p + \frac{C_0}{\tan \varphi_0})} \right)
\]

where \( M \) is the mining height, \( C_0 \) is the cohesion, \( \varphi_0 \) is the internal friction angle, \( K \) is the stress concentration coefficient of the abutment pressure in front of the coal wall, \( H \) is the buried depth of the working face, \( f \) is the friction coefficient of the coal seam and the roof and floor, \( \gamma \) is the triaxial stress coefficient, \( \xi = (1 + \sin \varphi_0) / (1 - \sin \varphi_0) \), and \( P \) is the support force of the hydraulic support to the coal wall.

Formula (1) shows that when the physical and mechanical parameters of the coal seam in the working face are determined, the support force of the hydraulic support to the coal wall is derived. The width of the plastic zone in front of the working face is proportional to the height of the working face, the buried depth of the working face, and the abutment pressure in front of the working face.

### 3.1.2. Influence of goaf on abutment pressure

The 12124 working face is located below the 12125 goaf and its upper coal pillars. During the mining process, the 12124 working face is significantly affected by the short-distance goaf and its coal pillars. Therefore, combined with the engineering background, a coal wall stress model of 12124 mining process, the 12124 working face is significantly affected by the short-distance goaf, which is not affected by the stress concentration of the coal pillar in the working face, and the abutment pressure in front of the working face was analyzed and studied.

**Figure 4** Mechanical model of coal wall in the 12124 working face

The mechanical model shows that the front abutment pressure of the coal wall of the 12124 working face changes when it overlaps with the stress of the 12125 goaf. According to the calculation formula of Reference[19], the width of the plastic zone of the coal pillar on the upper side of the 12125 goaf is 7.01 M, while the length of the 12124 working face under the coal pillar is 60 m. Therefore, the coal body in zone ⑤ of the upper part of the 12124 working face is under the primary rock stress of the No. 5 coal seam, which is not affected by the stress concentration of the coal pillar in the 12125 goaf; the coal bodies in zones ⑥ and ⑦ are under the stress concentration area of the coal pillar in the upper part of the 12125 goaf, which is affected by the stress concentration; the stress concentration of the coal pillar overlaps with the supporting pressure in front of the coal wall, and the supporting pressure becomes larger; the coal body in zone ⑧ of the middle and lower parts of the 12124 working...
face is larger under the 12125 goaf. This part of the coal body is in the upper goaf stress release area, and the abutment pressure becomes smaller. Therefore, the order of stress in front of the coal wall of the 12124 working face is $K3 > K2 > K1 > K4 > K5$. Suppose that the expression formula of the SE section’s abutment pressure in zone (5) is $q(\zeta) = C\zeta^2 + D$. A straight line theorem is determined according to the two points, the coordinates of the two points $((h-L), K_1 H_1)$, $((-f-g), K_2 H_2)$ are integrated, and the expression of the SE section abutment pressure is obtained. \[ q(\zeta) = \frac{K_1 H_1 - K_2 H_2}{e} \zeta + K_2 H_2 + \frac{K_1 H_1}{(h-L)} - \frac{K_2 H_2}{(h-L)} \zeta \in [(h-L),(f-g)] \] (2)

Similarly, the stress expression in zones (6), (7), and (8) can be obtained as follows:

\[ q_1(\zeta) = K_5 H_5 - K_2 H_5 \zeta + K_2 H_5 \zeta \in [-f,-g] \] (3)

\[ q_2(\zeta) = K_6 H_6 - K_2 H_6 \zeta + K_2 H_6 \zeta \in [0,h] \]

where $H_1$ is the burial depth of the upper end of the 12124 working face, $H_2 = H_1 + \sin a$, $H_3 = H_2 + \sin a$, and $H_4$ is the burial depth of the lower end of the 12124 working face.

By integrating formulas (2) and (3) into formula (1), we can obtain the expression of the plastic zone range of the DE, EF, FG, and GH sections as follows:

- **DE section**:
  \[
  L_{DE} = \frac{M}{2\pi f} \ln \frac{q_1(\zeta) + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})} = \frac{M}{2\pi f} \ln \frac{C\zeta^2 + D + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})}
  \]
  \[
  C = K_2 H_2 - K_1 H_1, \quad D = K_1 H_1 + \frac{K_1 H_1}{(h-L)} - \frac{K_2 H_2}{(h-L)} \zeta \in [(h-L),(-f-g)]
  \]
  (4)

- **EF section**:
  \[
  L_{EF} = \frac{M}{2\pi f} \ln \frac{q_2(\zeta) + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})} = \frac{M}{2\pi f} \ln \frac{E\zeta + F + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})}
  \]
  \[
  E = K_3 H_3 - K_2 H_2, \quad F = K_3 H_3 + \frac{-K_2 H_2 g + K_2 H_2 g}{f} \zeta \in [(-f-g),-g]
  \]
  (5)

- **FG section**:
  \[
  L_{FG} = \frac{M}{2\pi f} \ln \frac{q_2(\zeta) + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})} = \frac{M}{2\pi f} \ln \frac{G\zeta + H + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})}
  \]
  \[
  G = K_4 H_4 - K_2 H_2, \quad H = K_4 H_4 \zeta \in [-g,0]
  \]
  (6)

- **HG section**:
  \[
  L_{HG} = \frac{M}{2\pi f} \ln \frac{q_1(\zeta) + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})} = \frac{M}{2\pi f} \ln \frac{I\zeta + J + \frac{C_o}{\tan \varphi_0}}{\zeta(p + \frac{C_o}{\tan \varphi_0})}
  \]
  \[
  I = K_4 H_4 - K_4 H_4 \zeta \in [0,h]
  \]
  (7)
3.2. Numerical model

According to the actual parameters of the 12124 working face in Panbei Mine, the size of the calculation model is determined. The Mohr–Coulomb strength criterion is used in the model. The geometric parameters of the model are 336 m strike length, 320 m dip length, and 360 m model height. The four sides of the model are constrained in the normal direction, the top is the free boundary of stress and displacement, and the bottom is constrained by horizontal and vertical. The average dip angle of the simulated 12124 and 12125 working faces is 38°, the length of the 12124 working face is 140 m, and the length of the working face is 130 m. The numerical model is shown in Figure 5. The whole model simulates 18 layers of strata, which accurately reflects the occurrence of coal strata. The physical and mechanical parameters of each layer are shown in Figure 6. The load applied in the vertical direction above the model is equal to the rock weight of the non-simulated rock stratum. For the goaf of the 12125 working face, weakening material is used to refill the working face to simulate collapsed rock, and the upper part of the 12124 working face is weakened by mining disturbance.

![Figure 5 Numerical model](image)

![Figure 6 Physical and mechanical parameters of coal and rock](image)

4. Result analysis

4.1. Calculation results of the mechanical model

The burial depth of the upper end of the 12124 working face is 400 m, the burial depth of the lower end of working face \( H_s \) is 470 m, the length of working face \( L \) is 140 m, and the mining height of the working face is 3.8 m. In this work, \( g = 8 \text{m}, f = 3, g = 24 \text{m}, h = 80 \text{m}, e = 60 \sim 24 \sim 8 = 28 \text{m}, H_v = H_1 + \sin \alpha = 400 + 28 \times \sin 30 = 414 \text{m}, H_3 = H_2 + f \sin \alpha = 414 + 24 \times \sin 30 = 426 \text{m}, H_5 = H_3 + g \sin \alpha = 426 + 8 \times \sin 30 = 430 \text{m}, \) because \( K_5 > K_2 > K_1 > K_2 = K_5, \) taking \( K_2 = 1.2, K_1 = 1.4, K_2 = 3.2, K_2 = 4, \gamma = 2.5 \text{kN/m}^3. \) The dip angle of coal seam \( \alpha = 38°, \phi_v = 35°, C_0 = 2.15 \text{MPa}, P = 0.3, \bar{c} = (1 + \sin \phi_v)/(1 - \sin \phi_v) = 3.7, f = \tan \phi_v = 0.175. \) By substituting the above parameters into expressions (4)–(7), we can obtain the range of the plastic zone in front of the coal wall during the recovery of the 12124 working face. The results are shown in Figure 7.
The calculation shows that during the mining period of the 12124 working face, the average plastic area range of the upper part (0–28 m) of the working face is 13.3 m, while the average plastic area range of the upper part (28–60 m) of the working face is 15.1 m, and the average plastic area range of the lower part (60–140 m) of the working face is 12.6 m. Therefore, the 12125 goaf and the upper coal pillar have an impact on the mining of the 12124 working face.

![Figure 7 Plastic zone of the coal wall in the working face](image)

**Figure 7** Plastic zone of the coal wall in the working face

4.2. *Calculation results of the numerical model*

(1) The primary rock stress curve of the 12125 working face shows that when the working face is not excavated, the stress of the stope presents a linear distribution, and the stress value of the lower part of the working face is large.

(2) After the excavation of the 12125 working face, the No. 4 coal seam is affected by the mining stress of the No. 5 coal seam, and the stress is redistributed. A relief area is present under the goaf of the 12125 working face. The stress value of this part of the goaf is lower than the original rock stress, and the stress value is about 5 MPa, while the stress concentration area is formed under the upper coal pillar of the 12125 working face. The peak value of the upper concentration stress is about 19 MPa, the stress concentration coefficient is 1.65, the lower concentration stress is about 19 MPa, the peak stress is about 22 MPa, and the stress concentration coefficient is 1.8.

(3) During the excavation of the 12124 working face, abutment pressure areas are present within 40 m in front of the coal wall, and 5–10 m is the peak area of abutment pressure.

(4) In the initial stage of 12124 working face excavation, the peak value of the surrounding rock stress increases gradually. When the excavation depth is 20 m, the peak value of stress is about 35 MPa. When the excavation depth is 100 m, the stress value of the surrounding rock tends to be stable, and the peak value of stress reaches about 40 MPa.

(5) During the excavation of the 12124 working face, front abutment pressure is formed in front of the coal wall of the working face. The stress value in the 5 m area in front of the working face is higher than that in the original rock, while the stress concentration in the lower part of the 12124 working face is lower than that in the middle and upper parts of the working face due to the influence of the 12125 goaf. The peak value of stress in the coal body of the 12124 working face is located near the upper boundary coal pillar of the 12125 goaf. The stress concentration is formed in the solid coal and pillar on both sides of the roadway in the 12124 working face. The stress value in the solid coal on both sides of the roadway within 5–10 m in front of the coal wall is higher than that in the pillar, while the stress value in the solid coal on both sides of the roadway beyond 10 m is lower than that in the pillar.
5. Analysis of field monitoring data

A KJ345–F2 mine intrinsic safety pressure monitoring substation is installed to monitor the characteristics of periodic weighting during the mining of the 12124 working face. Twenty supports of a working face are installed with an abutment pressure gauge, and the data of four supports (20#, 40#, 60#, and 80#) are taken from the lower, middle, and upper parts of the working face for analysis. The data of these pressure gauges are representative. During the observation, the advance distance of the upper and lower parts of the working face is 29.2 and 23.5 m, respectively. The working resistance curve of the four hydraulic supports is shown in Figure 9.
Thus, the upper coal seam, which is greatly affected by mining in this area is higher than that of coal seam 40#. The average terminal resistance when the roof pressure is applied, and the dynamic load coefficient is 1.59. The average weighted resistance during the non-pressure period is 2898 kN, the average time-weighted resistance in the roof pressure period is 4002 kN, and the average dynamic load coefficient is 1.38.

(2) For support 40#, the average periodic weighting interval is 8.1 m, the average end resistance of the cycle during the non-roof pressure period is 3889 kN, the average end resistance of the cycle during the pressure period is 4955 kN, and the dynamic load coefficient is 1.27. The average time-weighted resistance during the non-roof pressure period is 3403 kN, the average time-weighted resistance during the roof pressure period is 4151 kN, and the average dynamic load coefficient is 1.22.

(3) For support 60#, the average periodic weighting interval is 8.6 m, the average terminal resistance is 3983 kN when the pressure is not applied, the average terminal resistance is 5343 kN when the pressure is applied, and the dynamic load coefficient is 1.34. The average weighted resistance is 3403 kN when the pressure is not applied, the average weighted resistance is 4525 kN when the roof pressure is applied, and the average dynamic load coefficient is 1.33.

(4) For support 80#, the average periodic weighting interval is 8.8 m, the average terminal resistance is 369 0kN when the roof pressure is not applied, the average terminal resistance is 5236 kN when the roof pressure is applied, and the dynamic load coefficient is 1.59. The average weighted resistance is 3029 kN when the roof pressure is not applied, the average weighted resistance is 4264 kN when the roof pressure is applied, and the average dynamic load coefficient is 1.41.

In summary, the 12124 working face is greatly affected by the residual stress caused by the upper coal seam mining, especially support 60# and its nearby area. The working resistance of the support in this area is higher than that of supports 40# and 80#. Support 60# is located under the coal pillar of the upper coal seam, which is greatly affected by mining and has a high degree of stress concentration; thus, the working resistance of the support is large. Support 80# on the working face is relatively less affected by mining, and supports 20# and 40# are below the goaf and are least affected by mining.

6. Control technology of surrounding rock in working face

6.1. Control technology of coal wall spalling and roof falling
(1) A diamond metal mesh is laid on the roof of the whole working face. At the same time, 68 pieces of downward broken roof are connected with a double-layer anchor chain mesh, and the I-steel is connected with the diamond metal mesh on the top beam of the support near support 60# to strengthen the management of the broken roof under the remaining coal pillars. The roof support of the working face is shown in Figure 10.

Figure 10 Control technology of coal wall spalling and roof falling

(2) To prevent a blank area of support between supports, the gap between the supports of the working face does not exceed 200 mm, resulting in partial roof leakage. When the working face encounters the soft coal wall, which is prone to coal wall spalling and partial roof falling, the expansion beam of the support is extended in time to support the roof, the front guard board is extended at the same time close to the coal wall, and the pressure perpendicular to the coal wall is applied. In the process of mining, the frame should be moved in time to support the roof.

(3) In the process of moving supports, the top beam and the base of the support are perpendicular to the roof and floor of the coal seam, thereby meeting the requirements of initial bearing capacity. The staggered stubble of the two adjacent side guard boards is not more than two-thirds of the side guard boards.

6.2. Control technology roadway surrounding rock
In the mining process of the 12124 working face, the stress concentration near the tail entry was high, which leads to a large amount of roadway deformation. At this time, the U-shaped shed was seriously deformed, and even the U-shaped shed was broken. To solve these problems, we removed the U-shaped shed of the roadway within 10 m in front of the tail entry. At the same time, the bottom of the tunnel was excavated again to expand the tunnel to the original designed cross-sectional area, and the combination of anchor, mesh, and cable was used for support. The support design scheme is shown in Figure 11. Both sides and the roof are provided with Φ 20 mm × 12000 mm high-strength bolt and Φ 22 mm × L6500 mm cable, and 8# wire mesh is used for joint support, in which the roof bolt array pitch is 850 mm, the two bolt array pitch is 640 mm, the roof cable array pitch is 1200 mm, the two cable array pitch is 1000 mm, and the vertical direction of the two sides of the roadway is equipped with an M3 steel belt.
The stress of the two sides of the coal body of the roadway is concentrated, and the roadway is seriously deformed due to the influence of mining in front of the coal wall of the working face. To meet the production demand, the wood shed or I-beam is used to replace the U-steel shed within 10 m ahead of the coal wall of the upper and lower tail entry of the working face, supported by the HDJA-1200 metal shingle roof beam or 11# mining I-beam, as shown in Figure 12. When the length of the inclined shed is less than or equal to 3.5 m, two rows of shed are changed up and another two rows are changed down. When the length of the inclined shed is more than 3.5 m and less than or equal to 4 m, three rows of the shed are changed up and two rows of the shed are changed down. When the length of the inclined shed is more than 4 m and less than or equal to 4.5 m, three rows of the shed are changed up and another three rows are changed down. The length and diameter of the round wood in the shed are more than 3.2 and 0.18 m, respectively.

**Figure 12** Plan and section of advanced support of roadway

7. Conclusion
(1) The theoretical analysis is basically consistent with the numerical simulation results. The basic law of the range of the plastic zone of the working face during the mining of SDCS under the goaf of a close coal seam group is middle–upper part > upper part > lower part. This finding also shows that the front abutment pressure in the middle–upper part is the largest, followed by the upper part and the lower part. The peak of abutment pressure is about 40 MPa, which appears in the range of 5–10 m in front of the coal wall in the middle–upper part of the 12124 working face.
(2) Field observation shows that the periodic weighting interval and the working resistance of the supports also show the trend of the upper part being the largest, the middle part being the second largest, and the lower part being the smallest. From the bottom to the top, the periodic weighting interval is 7.9–8.6 m; the maximum working resistance at the end of the support cycle is 5343 kN, which is 74.2% of the rated working resistance; and the dynamic load factor is 1.27–1.59.
(3) This work proposed and implemented stability control technology for the surrounding rock of the stope and roadway in the SDCS mining of a close coal seam group. This technology includes laying a metal mesh on the roof of the working face and connecting the I-beam to the frame head to strengthen the roof management; controlling the surrounding rock deformation of the roadway by using the combined support of an anchor net and a cable; and using a U-shaped shed to strengthen the support in the roadway 10 m ahead of the working face. Practice has proven that the implementation of the
above-mentioned control measures for the surrounding rock of the stope guarantees the safe mining of the working face.

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