

Article

Control Techniques for Gob-Side Entry Driving in an Extra-Thick Coal Seam with the Influence of Upper Residual Coal Pillar: A Case Study

Shengrong Xie *, Fangfang Guo and Yiyi Wu

School of Energy and Mining Engineering, China University of Mining and Technology-Beijing, Beijing 100083, China; gffever@163.com (F.G.); 15662045768@163.com (Y.W.)

* Correspondence: xsrxcq@163.com

Abstract: In multi-seam mining, the residual coal pillar (RCP) in the upper gob has an important influence on the layout of the roadway in the lower coal seam. At present, few papers have studied the characteristics of the surrounding rock of gob-side entry driving (GED) with different coal pillar widths under the influence of RCP. This research contributes to improving the recovery rate of the extra-thick coal seam under this condition. The main research contents were as follows: (1) The mechanical parameters of the rock and coal mass were obtained using laboratory experiments coupled with Roclab software. These parameters were substituted into the established main roof structure mechanics model to derive the breakage position of the main roof with the influence of RCP, and the rationality of the calculation results was verified by borehole-scoping. (2) Based on numerical simulation, the evolution laws of the lateral abutment stress in the lower working face at different relative distances to the RCP were studied. FLAC3D was used to study the whole space-time evolution law of deviatoric stress and plastic zone of GED during driving and retreating periods with various coal pillar widths under the influence of RCP. (3) The plasticization factor P was introduced to quantify the evolution of the plastic zone in different subdivisions of the roadway surrounding rock, so as to better evaluate the bearing performance of the surrounding rock, which enabled a more effective determination of the reasonable coal pillar width. The field application results showed that it was feasible to set up the gob-side entry with an 8 m coal pillar below the RCP. The targeted support techniques with an 8 m coal pillar could effectively control the surrounding rock deformation.

Keywords: residual coal pillar; gob-side entry driving; extra-thick coal seam; coal pillar size; surrounding rock control



Citation: Xie, S.; Guo, F.; Wu, Y. Control Techniques for Gob-Side Entry Driving in an Extra-Thick Coal Seam with the Influence of Upper Residual Coal Pillar: A Case Study. *Energies* **2022**, *15*, 3620. <https://doi.org/10.3390/en15103620>

Academic Editor: Sergey Zhironkin

Received: 18 April 2022

Accepted: 12 May 2022

Published: 15 May 2022

Publisher's Note: MDPI stays neutral with regard to jurisdictional claims in published maps and institutional affiliations.



Copyright: © 2022 by the authors. Licensee MDPI, Basel, Switzerland. This article is an open access article distributed under the terms and conditions of the Creative Commons Attribution (CC BY) license (<https://creativecommons.org/licenses/by/4.0/>).

1. Introduction

Gob-side entry driving (GED) with a small coal pillar (3~8 m) is widely used in China's mines due to the simple development processes, high resource recovery rate, and the ability of the coal pillar to isolate gangue, harmful gases, water, and fire in the adjacent gob [1,2]. China has many thick and extra-thick coal seams, and the GED has been successfully applied in many mines with extra-thick coal seams [3,4]. For the mining of extra-thick coal seams (>8 m), the characteristics mainly include [5–8]: (1) Strong-dynamic disturbance of abutment stress. Fully mechanized top coal caving mining is generally applied to the mining of extra-thick coal seams, with a large mining space and intense roof activity, resulting in severe abutment stress on the coal mass. (2) A thick top coal over the roadway. The roadway is usually developed along the floor line of the coal seam, the roof of the GED is composed of weak coal masses with a large thickness. The weak properties of the top coal mass seriously increase the difficulty of roadway control. (3) Disturbed by multiple mining-induced stresses. The GED is not only disturbed by the lateral abutment stress of the previous working face, but also advanced abutment stress of the present working face.

Therefore, it is of great significance to study the selection principles of reasonable coal pillar width and surrounding rock control techniques of the GED in extra-thick coal seams.

Presently, many scholars around the world have conducted extensive research on the design of coal pillar width using various methods such as theoretical analysis, numerical simulation, and similar simulation. UDEC can effectively study the macro-mechanical behavior and crack propagation law during the rock failure process, so it is frequently applied to the analysis of the roadway failure mechanism. Bai et al. [9] studied how the failure mechanisms of a 7 m coal pillar width during the formation process caused a large deformation. They investigated the propagation of shear and tensile cracks in the coal pillar of various widths and optimized the coal pillar width and support measures. Shi et al. [10] investigated the crack evolution mechanism of the gob-side entry for different conditions and proposed optimized-support parameters combined with roof-cutting measures. Gao [11–13] carried out a series of numerical simulations using the UDEC Trigon approach to focus on the roadway damage caused by squeezing failure and shear failure, and the effects of rock bolts in the roadway support were also evaluated. FLAC3D can simulate the mechanical behavior of geological materials and geotechnical engineering effectively and is widely used in underground coal mining activities [14–16]. He et al. [17–21] studied the stress distribution characteristics and plastic zone of roadway with different coal pillar widths in the process of coal mining and proposed a support technology for setting a reasonable width of coal pillars. The specific coal pillar width was designed and applied in engineering practices, which realized a good control effect of the roadway surrounding rock. Ma [22] conducted research on the stress distribution characteristics of the narrow coal pillar with different top coal heights of gob-side entry, and they believed that the thick top coal was not conducive to control of the roadway. Jiang [23] presented an approach for evaluating, designing, and optimizing EDG and yield pillar based on the results of numerical simulations and field practice. Han et al. [24–26] considered special geological situations to drive an entry along the gob, such as in an isolated island working face, in a deep soft-broken coal seam, or in a working face adopting roadway side sealing technology. The targeted control techniques of the surrounding rock were proposed and successfully applied in the field practices. These studies had made countless valuable explorations on the width of coal pillars and failure mechanisms and made certain innovations in the research methods. However, there were very few studies on the layout of the GED with the influence of a residual coal pillar (RCP) during the mining of multiple coal seams.

China has many multiple coal seam mining areas, such as Datong, Xinwen, Pingdingshan, etc. [27]. The mining of two adjacent coal seams tends to interact with each other due to the close distance [28,29]. The mining of the upper coal seam generally renders many section pillars, and lower coal seam roadways are usually designed with larger section coal pillars (usually over 20 m) to avoid safety hazards. That results in the waste of massive coal resources, especially for extra-thick coal seams. This paper focused on the layout selection of GED below the RCP. The lateral breakage position of the main roof under the influence of RCP was calculated by theoretical derivation and verified the rationality of the results using borehole-scoping. Under the influence of RCP, the whole space-time evolution laws of deviatoric stress and plastic zone of GED were studied during driving and retreating periods with different coal pillar widths using numerical simulation. The quantified index for the plastic zone of the surrounding rock of GED, plasticization factor P , was proposed. A coal pillar width of 8 m was finally determined. Based on the extent of the plastic zone of GED obtained by borehole-scoping, the targeted roadway support scheme was proposed. That provided a reference for promoting the coal recovery rate in similar geological conditions.

2. Project Overview

2.1. Geological Conditions

Nanyangpo coal mine is located in Shanyin County, Shanxi Province, China. The main mining seams are No. 4 and No. 6 coal seam with a spacing of 32 m. The average thickness

of the No. 4 coal seam is about 3.0 m, which has been mined out, but 15 m section coal pillars are left between working faces. The current main mining seam is the No. 6 coal seam with an inclination of $3\sim5^\circ$ and an average thickness of 9.6 m. The mining method is fully mechanized top coal caving mining of an extra-thick coal seam, the machine mining height is 3.5 m, and the caving height is 5~10 m. The geological column chart of the No. 6 coal seam is illustrated in Figure 1. The panel 26,102 is near the northern boundary of the mining field, and next to the panel 26,104 and 26,106 that have been mined out with a 20 m coal pillar left. To avoid wasting resources, Nanyangpo coal mine plans to develop the 26,102 tailgate along the gob with a small coal pillar. The layout of the panels is given in Figure 2.

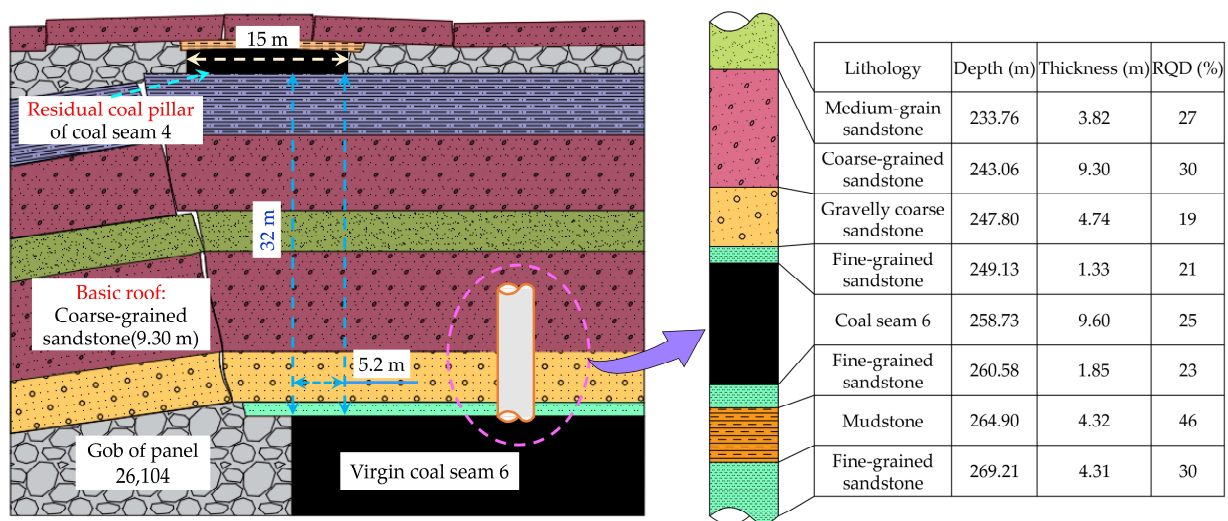


Figure 1. Generalized stratigraphic column of the test site with the panel layout.

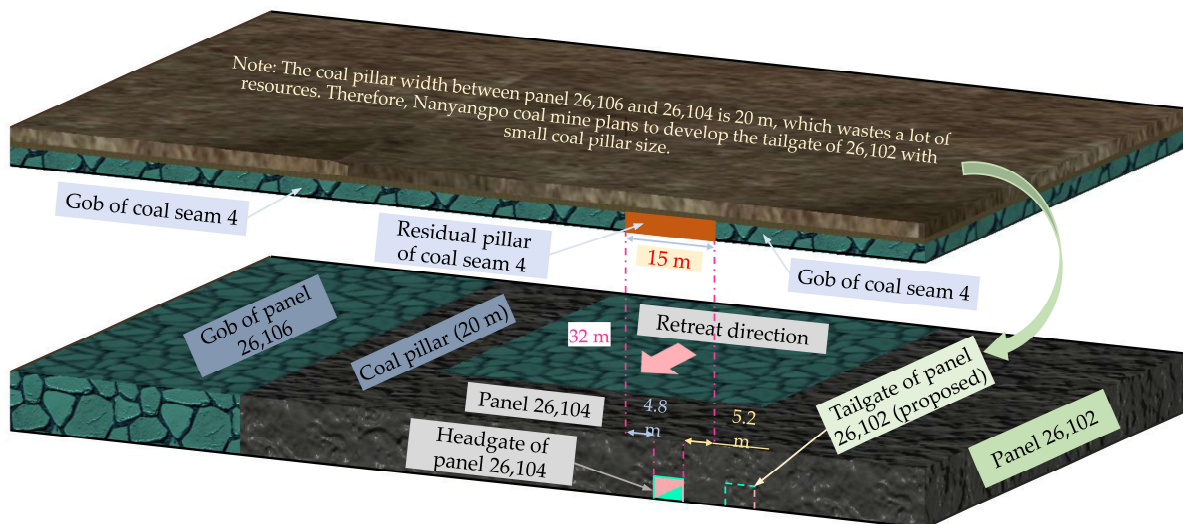


Figure 2. Detailed panel layout of the test site.

2.2. Rock Mass Properties

The physical and mechanical properties of rock mass are an important basis for the design of roadway support. The parameters obtained in the laboratory are also key data for further study on the theoretical model calculation and numerical simulation. The mechanical parameters of coal and rock mass in panel 26,102 are shown in Table 1.

Table 1. Properties of rock mass in panel 26,102.

Lithology	σ_{ci} (MPa)	σ_{ti} (MPa)	c_i (MPa)	φ_i (°)	E_i (GPa)	ν_i
Coal seam 4	26.37	1.02	7.63	26	1.53	0.39
Medium-grained sandstone	89.45	6.86	18.79	29	34.21	0.17
Coarse-grained sandstone	93.56	8.07	24.10	31	38.27	0.23
Gravelly coarse sandstone	47.63	3.34	14.62	25	32.04	0.27
Coal seam 6	34.35	1.27	8.14	24	1.72	0.34
Fine-grained sandstone	98.63	9.34	20.61	31	40.71	0.21
Mudstone	40.37	2.17	11.92	35	11.20	0.31

Note: σ_{ci} is the uniaxial compression strength, σ_{ti} is the tensile strength, c_i is cohesion, E_i is elastic modulus, φ_i is internal friction angle, and ν_i is Poisson's ratio.

It is difficult to get the mechanical parameters of coal and rock samples measured in the laboratory to reflect the actual physical and mechanical properties in engineering sites due to the absence of the original environment and structural characteristics of rock mass [30,31]. E. Hoek and E. T. Brown et al. [32,33] obtained the Hoek–Brown failure criterion by a large number of rock mechanics tests as well as field tests of rock masses after continuous revision and improvement. The parameters of coal and rock samples obtained in the laboratory were imported into Roclab software based on the Hoek–Brown strength criterion for calculation, and the revised parameters were obtained in Table 2, which were more in line with the engineering reality.

Table 2. Revised properties of rock mass by Hoek–Brown strength criterion.

Lithology	GSI	m_i	D	E_{rm} (GPa)	σ_c (MPa)	σ_t (MPa)	φ (°)	c (MPa)	ν
Coal seam 4	56	7	0.7	0.24	1.06	0.08	22.0	0.94	0.39
Medium-grained sandstone	73	18	0.7	13.06	12.59	0.44	38.0	6.26	0.17
Coarse-grained mudstone	77	16	0.7	16.77	17.63	0.74	38.8	7.04	0.23
Gravelly coarse sandstone	71	8	0.7	11.30	5.80	0.44	30.0	5.79	0.27
Coal seam 6	59	6	0.7	0.33	1.73	0.14	22.7	1.26	0.34
Fine-grained mudstone	78	17	0.7	18.39	19.98	0.80	39.8	7.71	0.21
Mudstone	62	7	0.7	2.52	2.54	0.19	25.1	1.68	0.31

Note: m_i is the constant of the intact rock, GSI is the constant evaluating the fractured rock mass, and D is the disturbance factor.

3. Breakage Position of the Main Roof

3.1. Influence of Upper Residual Coal Pillar

The stress redistribution in the roof strata after mining of the upper coal seam resulted in the RCP bearing a larger load. Thereby, a certain range of stress elevation area was formed at the floor of the RCP. That meant the different positions of the lower working face will make the GED in different stress environments. Based on numerical simulation, the evolution laws of the lateral abutment stress in the lower working face at different relative distances to the RCP were as shown in Figure 3.

Taking the centerline of the upper RCP as the base point, the peak values of lateral abutment stress were 23.2 MPa and 25.5 MPa when the edge of the working face (EWF) was -37.5 m and 37.5 m from the centerline of the RCP separately, with an increase of 0.9% and 10.9%. The peak positions of lateral abutment stress were both about 6 m from the gob. When EWF was -17.5 m and 17.5 m from the centerline of the RCP separately, the peak values of lateral abutment stress were 25.7 MPa and 24.7 MPa with an increase of 11.7% and 7.4%, and the peak positions of lateral abutment stress were both about 6 m from the

gob. When EWF was -7.5 m and 7.5 m from the centerline of the RCP separately, the peak values of lateral abutment stress were 29.9 MPa and 29.3 MPa with an increase of 30.0% and 27.4% , and the peak positions of lateral abutment stress were about 7 m and 6 m from the gob. When EWF was -2.5 m and 2.5 m from the centerline of the RCP separately, the peak values of lateral abutment stress were 28.3 MPa and 28.0 MPa with an increase of 23.0% and 21.7% , and the peak positions of lateral abutment stress were about 11 m and 7 m from the gob. The peak values of lateral abutment stress of working face showed an “M-shaped” tendency, and the peak depth first increased and then declined.

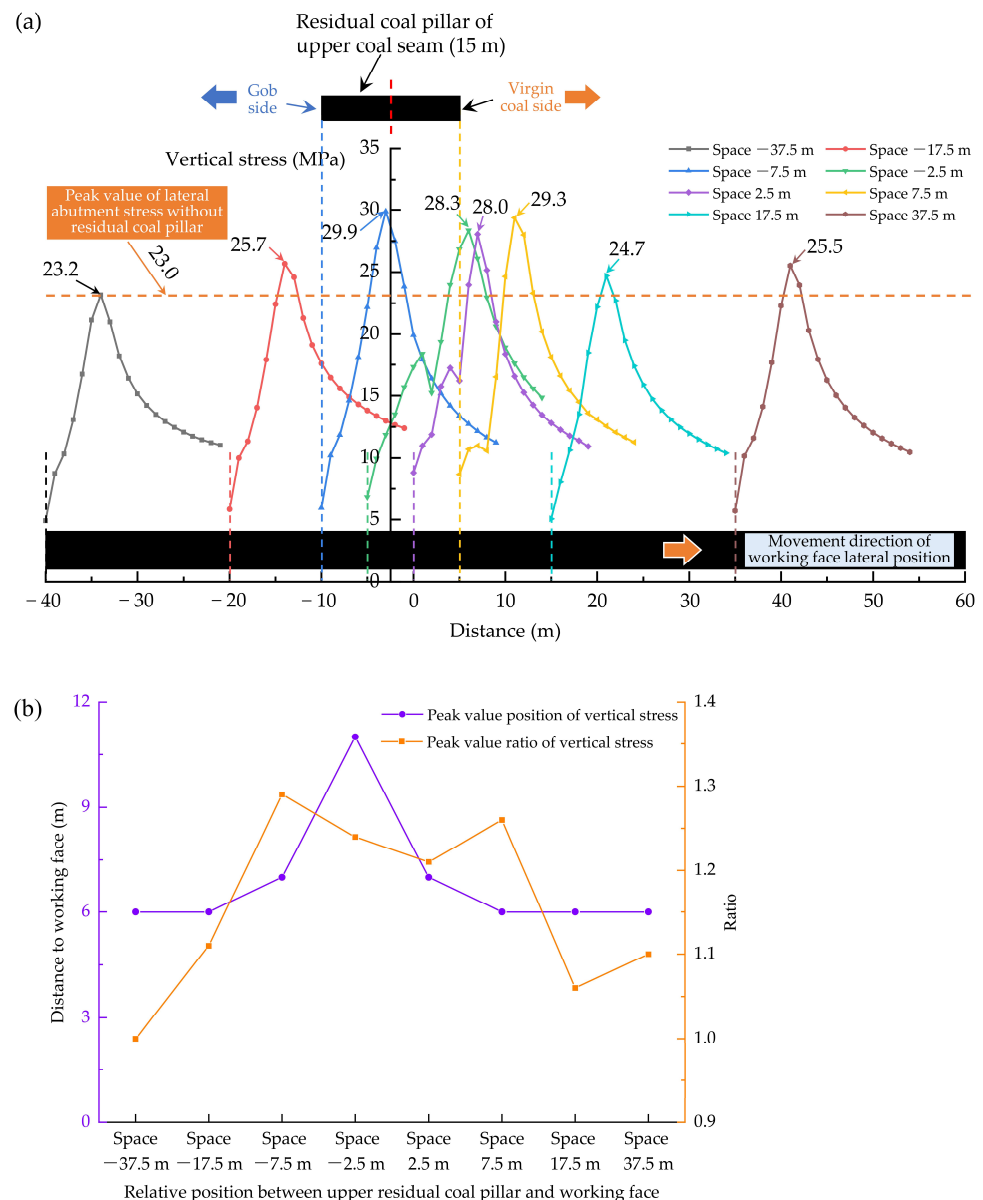


Figure 3. The evolution laws of the lateral abutment stress in lower working face at different relative distances to the RCP. (a) Overview under different distances. (b) Peak stress ratio and distribution of peak locations. (Note: the gob side and virgin coal side in the figure were both relative to the lower working face).

The above distribution laws indicated that the lateral abutment stress was most significantly affected when the edge of the lower working face was located directly below the residual coal pillar, and the peak growth coefficient was $1.20\sim 1.30$. Additionally, when EWF of the lower coal seam was 2.5 m away from the centerline of RCP, the peak growth coefficient of the lateral abutment stress was 1.22 . The influence of the stress disturbance

of RCP was relatively diminished when a small coal pillar was set up to drive the entry along the gob in such conditions. Consequently, the superimposed disturbance impact of RCP and mining-induced stress on GED should be fully considered to prevent the roadway from instability.

3.2. Mechanical Model of the Main Roof

The main roof controls the upper weak strata of the coal seam. The fracture morphology, the hinge status, and the stability of key blocks after main roof breakage greatly impact the stress distribution of the surrounding rock [34,35]. There is a 15 m coal pillar left in No. 4 coal seam at 32 m above the edge of 26,104 gob in Nanyangpo coal mine. The disturbance of RCP on the stress of floor will inevitably affect the load distribution pattern of the main roof in No. 6 coal seam. Based on the distribution regulation of the lateral abutment stress obtained by numerical simulation in lower working face under the influence of RCP, the mechanical model of the elastic foundation beam is established, as shown in Figure 3. The load curve shows the abutment stress before the lateral breaking of the main roof. The main roof above the virgin coal area is simplified as the elastic foundation beam under the pressure of overburden rock, and the hanging part of the main roof in gob is assumed as the cantilever beam structure.

The rock-beam of the overhanging part in the gob is taken for forces analysis as in Figure 4b. According to $\sum F_y = 0$:

$$F = \int_0^{l_1} \left(\frac{(q_1 - q_2)x}{l_1} + q_2 \right) dx + \int_{l_1}^{l_2} \left(q_1 - \frac{(q_1 - q_c)x}{l_2 - l_1} \right) dx + \int_{l_2}^{l_3} q_c dx \quad (1)$$

Then, the shear force Q_0 and the bending moment M_0 at $x = 0$ are

$$\left. \begin{aligned} Q_0 &= F = \int_0^{l_1} \left(\frac{(q_1 - q_2)x}{l_1} + q_2 \right) dx + \int_{l_1}^{l_2} \left(q_1 - \frac{(q_1 - q_c)x}{l_2 - l_1} \right) dx + \int_{l_2}^{l_3} q_c dx \\ M_0 &= \int_0^{l_1} \left(\frac{(q_1 - q_2)x}{l_1} + q_2 \right) x dx + \int_{l_1}^{l_2} \left(q_1 - \frac{(q_1 - q_c)x}{l_2 - l_1} \right) x dx + \frac{q_c(l_3 - l_2)(l_3 + l_2)}{2} \end{aligned} \right\} \quad (2)$$

where l_1 is the horizontal distance from the center of RCP to the edge of the lower virgin coal; l_2 is the influence range of RCP on the stress distribution of the main roof; and l_3 is the hanging length of the main roof in the gob, which is approximately equal to the periodic weighting step of the working face.

$q_1 = K_1 \gamma H$, $q_2 = K_2 \gamma H$, where K_1 and K_2 are the stress increase coefficient, H is the buried depth of the main roof stratum. q_c is the uniform load on the overhanging part of the main roof, the expression is as follows:

$$q_c = \frac{E_1 h_1^3 (\gamma_1 h_1 + \gamma_2 h_2 + \dots + \gamma_n h_n)}{E_1 h_1^3 + E_2 h_2^3 + \dots + E_n h_n^3} \quad (3)$$

The beam structure model with immediate floor, virgin coal, and immediate roof as the elastic foundation is shown in Figure 4c. Taking the edge of virgin coal as the origin, the relationship between subsidence y and stress p is as follows:

$$p = -ky \quad (4)$$

$$k = \frac{E_i}{(1 - \nu_i^2)h_i}, \quad \frac{1}{k} = \sum_i^n \frac{1}{k_i} \quad (5)$$

where k is the foundation stiffness coefficient, which is related to Poisson's ratio ν_i and thickness h_i of the weak rock layers below the main roof.

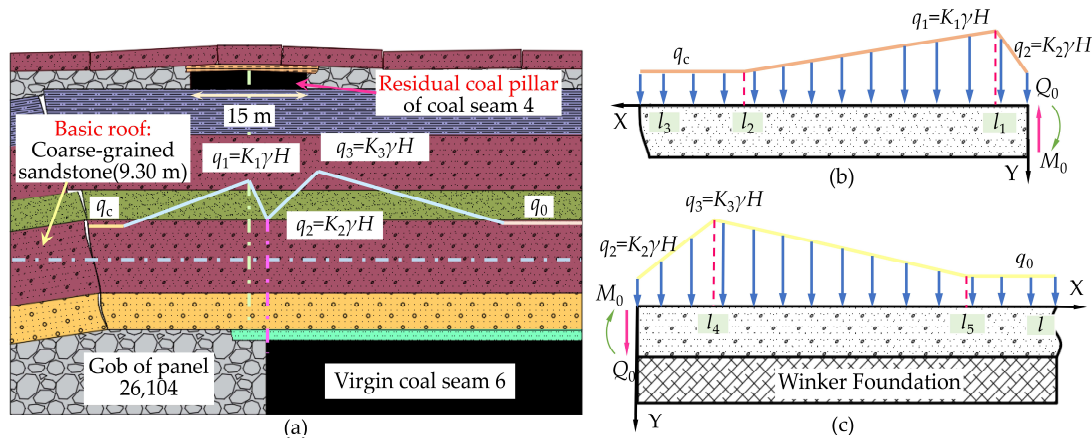


Figure 4. Mechanical model of elastic foundation beam. (a) Structural diagram before basic roof breaking. (b) Analysis of rock-beam of the overhanging part. (c) Analysis of rock-beam with the elastic foundation.

The elastic foundation beam differential equation is

$$\frac{d^4 y}{dx^4} + \frac{ky}{EI} = \frac{q(x)}{EI}, \quad \beta = \sqrt[4]{\frac{k}{4EI}} = \frac{1}{L} \quad (6)$$

where L is the characteristic length of the beam and EI is the bending rigidity of the main roof rock mass.

Then, the differential equation of elastic foundation beam is

$$\frac{d^4 y}{dx^4} + 4\beta^4 y = \frac{q(x)}{EI} \quad (7)$$

According to the calculation of elastic foundation beams [36], the general solution of the deflection equation is

$$y = y_0 \phi_1(\beta x) + \theta_0 \frac{1}{\beta} \phi_2(\beta x) - M_0 \frac{1}{EI\beta^2} \phi_3(\beta x) - Q_0 \frac{1}{EI\beta^3} \phi_4(\beta x) + \langle g(x) \rangle_t \quad (8)$$

where the Kralof function, defined to simplify the calculation, is as follows:

$$\left. \begin{aligned} \phi_1 &= ch\beta x \cos \beta x \\ \phi_2 &= \frac{1}{2}(ch\beta x \sin \beta x + \sin \beta x \cos \beta x) \\ \phi_3 &= \frac{1}{2}sh\beta x \sin \beta x \\ \phi_4 &= \frac{1}{4}(ch\beta x \sin \beta x - \sin \beta x \cos \beta x) \\ ch\beta x &= \frac{e^{\beta x} + e^{-\beta x}}{2} \\ sh\beta x &= \frac{e^{\beta x} - e^{-\beta x}}{2} \end{aligned} \right\} \quad (9)$$

where y_0, θ_0, M_0, Q_0 are the deflection, angle of rotation, bending moment, and shear force at $x = 0$; $\langle g(x) \rangle_t$ denotes the correction term that should be added when $x > t$.

For the solution of the elastic foundation beam under the loading condition as shown in Figure 4c, the deflection equation of the elastic foundation beam with a correction term section can be given according to the superposition principle:

$$\begin{aligned}
g &= \frac{q_2}{k} + \left(\frac{q_3 - q_2}{kl_1} - \frac{q_2}{k} \right) \frac{1}{\beta} \phi_2(\beta x) - \frac{q_3 - q_2}{kl_1} x \\
&+ \left\langle \left[\frac{q_2 - q_3}{kl_4} + \frac{(q_3 - q_0)}{(l_5 - l_4)k} \right] x + \frac{2q_2 - q_3}{k} \phi_1[\beta(x - l_4)] \right. \\
&+ \left[\frac{(q_0 - q_3)l_4}{(l_5 - l_4)k} - \frac{q_2}{k} \right] - \left[\frac{q_2 - q_3}{kl_4} + \frac{(q_3 - q_0)}{(l_5 - l_4)k} \right] \frac{1}{\beta} \phi_2[\beta(x - l_4)] \Bigg\rangle_{l_4} \\
&\left\langle \frac{(q_0 - q_3)l_5}{(l_5 - l_4)k} - \frac{(q_0 - q_3)}{(l_5 - l_4)k} x + \frac{(q_0 - q_3)}{(l_5 - l_4)k} \frac{1}{\beta} \phi_2[\beta(x - l_5)] \right\rangle_{l_5}
\end{aligned} \quad (10)$$

where the load on the main roof at l_5 length in virgin coal approximately equals the original rock stress, that is $q_0 = \gamma H$, $q_3 = K_3 \gamma H$, which is approximately considered that the peak value of the abutment stress of the main roof above the virgin coal with a length of l_4 from the edge of gob. K_3 is the stress disturbance coefficient factor.

The equation for the deflection, angle of rotation, bending moment, and shear force of the main roof before breakage are given:

$$\left. \begin{aligned}
y &= y_0 \phi_1(\beta x) + \theta_0 \frac{1}{\beta} \phi_2(\beta x) - M_0 \frac{1}{EI\beta^2} \phi_3(\beta x) \\
&\quad - Q_0 \frac{1}{EI\beta^3} \phi_4(\beta x) + g \\
\theta &= -y_0 \beta \phi_4 + \theta_0 \phi_1 - M_0 \frac{1}{EI\beta} \phi_2 - Q_0 \frac{1}{EI\beta^2} \phi_3 + g' \\
M &= y_0 \cdot 4EI\beta^2 \phi_3 + \theta_0 \cdot 4EI\beta \phi_4 + M_0 \cdot \phi_1 + Q_0 \frac{1}{\beta} \phi_2 + g'' \\
Q &= y_0 \cdot 4EI\beta^3 \phi_2 + \theta_0 \cdot 4EI\beta^2 \phi_3 - M_0 \cdot 4\beta \phi_4 + Q_0 \phi_1 + g'''
\end{aligned} \right\} \quad (11)$$

The boundary conditions are:

$$\left. \begin{aligned}
M_0 &= [q_1(4l_1^2 + 5l_2^2 + 5l_1l_2) + q_2l_1^2 \\
&\quad + q_c(l_1^2 - 5l_2^2 + 3l_3^2 + l_1l_2)]/6 \\
Q_0 &= [q_1(l_2 - l_1) + 2q_2l_1 + q_c(2l_3 - l_1 - l_2)]/2 \\
y|_{x=l} &= 0, \theta|_{x=l} = 0
\end{aligned} \right\} \quad (12)$$

According to the numerical simulation results and the geological conditions of the mine, the engineering parameters are given as follows: the depth of the 26,104 working face is about 250 m; the dip length of the working face is 240 m; and the periodic weighting step is 20~25 m.

The immediate roof, coal seam, and immediate floor are considered as the elastic foundation of the main roof beam. The foundation coefficient k is 0.0380 GPa; the elastic modulus, bending modulus, and bending stiffness of main roof are 21.78 GPa, 67.03 m⁴, and 1460 GN·m², respectively; $l_1 = 2.3$ m, $l_2 \approx l_3 \approx 20\sim 25$ m; $l_4 \approx 6$ m, l_5 approximately equals to 22 m, $l_5 \geq 3L$, taken as $3L = 60$ m; K_1, K_2, K_3 were approximated to 1.2, 0.7, 1.5; $q_0 = 5.85$ MPa, $q_1 = 7.02$ MPa, $q_2 = 4.10$ MPa, $q_3 = 8.78$ MPa.

Combining (9)–(12) and substituting the data, the breaking position of main roof is 4.4~5.8 m from the edge of the gob.

3.3. Borehole-Scoping in the Main Roof

Borehole-scoping can accurately visualize the lithology, thickness, delamination, cracks, and fractures of the overlying rock strata. To verify the theoretical derivation result of the lateral breakage position of the main roof after panel 26,104 retreated, the borehole-peeping stations were arranged in the test section of 26,102 tailgate to observe the cracks propagation in the roof strata. A total of 23 holes were drilled in 4 groups with a total depth of over 640 m. The site construction is shown in Figure 5.

The obvious crack was defined by a width over 5 mm and a length over 10 mm, and the remarkable crack was defined by a width over 10 mm and a length over 100 mm. The distribution of cracks propagation in the roof strata of 26,102 tailgate is shown in Figure 6, indicated that: (1) most of the cracks were distributed between 10.55 m and 21.43 m above the coal pillar, that is, the range of main roof thickness. (2) Remarkable longitudinal cracks with intersecting circumferential cracks occurred at depths of 19.49 m in BII and 13.93 m in BII, indicating that the main floor had been fractured. The borehole wall within 1 m of the

two cracks sites was relatively intact again, so it could be deduced that the fracture line of the main roof was located above the coal pillar. (3) Based on the angles and depths of the boreholes where the two remarkable cracks were observed, the position of fracture line in the main roof was 5~6 m away from the edge of the gob, which was consistent with the theoretical calculation result.

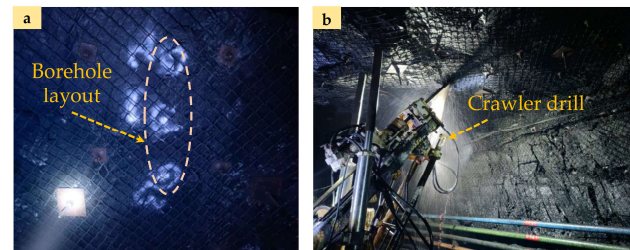


Figure 5. Site construction. (a) Layout of part boreholes in roof. (b) Crawler drill.

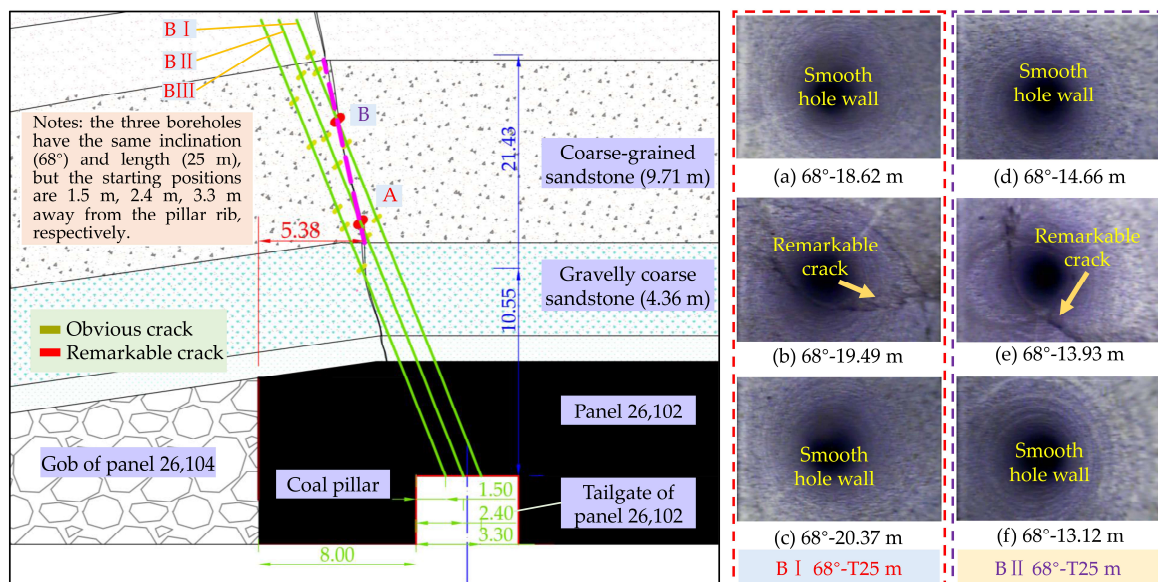


Figure 6. The distribution of roof strata cracks in 26,102 tailgate. Note: taking the representation of 45°-T35 m as an example, 45° denotes the borehole angle, and T35 m represents the full length of the borehole; taking the description of (a) 68°-19.62 m as an example, (a) denotes the serial number of the camera image of borehole, 68° denotes the borehole angle, and 19.62 m represents the detection depth.

4. Discussion and Analysis of the Simulation Results

4.1. Global Model for the Pillar Width

The model selects the 26,102 tailgate as the test roadway, which is adjacent to the gob of panel 26,104. The X-axis is 180 m in the length direction of working face, the Y-axis is 120 m in the advancing direction and the Z-axis is 110 m in the vertical direction. The roadway section is rectangular, and the dimension is 5.2 m × 3.5 m (width × height). The boundary displacement of the model is constrained in horizontal and bottom. The upper of the model is subjected to a stress of 4.05 MPa equivalent to the self-weight of the overburdened rock, and the lateral pressure coefficient is 1.2. The model is calculated using the Mohr-Coulomb model, while the double-yield model is used for the gob, and the mechanical parameters of each rock formation are taken from Table 2.

4.2. Double-Yield Model of the Gob

With the retreating of the working face, the broken blocks are backfilled in the gob after the main roof periodic fracture. The modulus coefficient of the gangue in the gob will

increase significantly after compacted. The densely compacted gob can bear part of the abutment pressure, which effectively weakens the stress concentration in the coal pillar. Therefore, the double-yield model in FLAC3D can be used to simulate the compaction and hardening process of waste in the gob [37]. The overburden pressure parameters and mechanical parameters of gangue in the gob can be obtained by the Salamon formula, expressed as follows [38]:

$$\sigma = E_0 \varepsilon / \left(1 - \frac{\varepsilon}{\varepsilon_{\max}} \right) \quad (13)$$

where E_0 is the initial tangential modulus, GPa; σ is the stress of gangue in the gob, MPa; ε is the bulk strain of the compressed gangue in the gob; $\varepsilon_{\max}$ is the maximum bulk strain. The values are given:

$$\left. \begin{aligned} \varepsilon_{\max} &= (b - 1) / b \\ E_0 &= \frac{1.039 \sigma_c^{1.042}}{b^{7.7}} \\ b &= (h + h_c) / h_c \end{aligned} \right\} \quad (14)$$

where b is the crushing expansion coefficient of gangue; σ_c is the gangue compressive strength; h is the mining height of the coal seam; h_c is the height of the roof caving zone in the gob.

According to the engineering situations of Nanyangpo coal mine, the average mining height of coal seam during retreating period is 9.6 m, and the height of the roof caving zone in the gob is about 34.6 m. The values can be substituted into Equations (13) and (14) to derive the double-yield model parameters of the gangue in the gob as shown in Table 3.

Table 3. Double-yield model parameters of gangue in the gob.

Strain	Stress (MPa)	Strain	Stress (MPa)	Strain	Stress (MPa)
0.01	0.10	0.08	1.23	0.15	4.67
0.02	0.22	0.09	1.50	0.16	5.83
0.03	0.34	0.10	1.80	0.17	7.46
0.04	0.48	0.13	2.17	0.18	9.93
0.05	0.63	0.14	2.60	0.19	14.12
0.06	0.81	0.13	3.14	0.20	22.77
0.07	1.01	0.14	3.81	0.21	60.00

The specification of the established unit sub-model of the gob is $1 \text{ m} \times 1 \text{ m} \times 1 \text{ m}$, and a constant velocity of 10^{-5} m/s is applied to the upper surface of the model to determine the mechanical properties of the gob by trial-and-error method. when the parameters are set to the gangue with a density of 1000 kg/m^3 , bulk modulus of 11.12 GPa, shear modulus of 5.20 GPa, internal friction angle of 5° , the stress–strain curve of the numerical simulation matches well with that of the theoretical calculation, as shown in Figure 7.

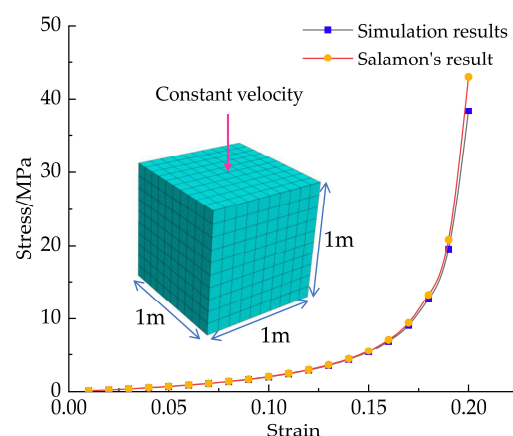


Figure 7. Numerical simulation inversion of goaf parameters.

4.3. Discussion and Analysis of the Simulation Results

4.3.1. Results with Various Coal Pillar Widths

Deviatoric stress is the synthesis of horizontal, vertical, and tangential stresses, and represents the distribution of shear stress in the material subjected to loads, revealing that the essential force of rock failure is mainly caused by shear stress. This stress index is gradually adopted by an increasing number of papers [39,40].

The size of the coal pillar affects the state of deviatoric stress distribution, the failure range of the plastic zone and the deformation extent of the roadway surrounding rock in GED. In this paper, we define the plasticization factor P as a parameter to characterize damage of the surrounding rock in GED, which is given as

$$P_1 = \frac{S_1}{S_p}, P_2 = \frac{S_2}{S_r}, P_3 = \frac{S_3}{S_e}, P_4 = \frac{S_4}{S_e} \quad (15)$$

where P_1, P_2, P_3 , and P_4 are the plasticization factors of coal pillar subdivision (I), top coal subdivision (II), virgin coal upper corner subdivision (III), and virgin coal subdivision (IV) in GED, respectively. S_1, S_2, S_3 , and S_4 are the areas of plastic zone in I, II, III, and IV. S_p, S_r , and S_e are the cross-section area of I, II, and gob-side entry, respectively.

As shown in Figures 8 and 9, with the increase of coal pillar width, the peak zone of deviatoric stress in the virgin coal area of GED tended to be gradually reduced, which contrasted with the coal pillar area. With the width of coal pillar being 4~6 m, the surrounding rock of gob-side entry was in a lower stress environment. While the plastic zone in II was coalesced with that of the overlying rock and the high stress of the surrounding rock was mostly concentrated in the virgin coal area, indicating that the bearing capacity of coal pillar was poor in such conditions. The range of plastic zone of GED decreased rapidly with a coal pillar of 8 m. The peak zone of deviatoric stress in coal pillar progressively enlarged, while virgin coal side continued to diminish up to similar peak values on both sides. It indicates that the bearing capacity of the coal pillar was enhanced and initiated to sustain the overburden stress in concert with the virgin coal. The coal pillar was in a high stress state with a width of 10~15 m. Additionally, the main bearing body of the overlying load gradually transformed from the virgin coal side to the coal pillar. At this time, the plastic zone of GED was small with a high stability of the surrounding rock, while a significant amount of the coal resources were wasted.

With the increase of coal pillar width, the plasticization factor in four subdivisions of the entry gradually declined. Correspondingly, the peak value of deviatoric stress in the coal pillar area grew swiftly to a stable state, while the value in virgin coal area continued to get smaller. When the width of coal pillar is 8 m, the P were not more than 80% in I and II, and 60% in III and IV. The peak deviatoric stress in coal pillar area was roughly coincident with that of the virgin coal area.

4.3.2. Results with Disturbance of Panel Retreating

Mining-induced stress exerted an essential influence on the stability of the surrounding rock in GED. Exploring the distribution characteristics of the deviatoric stress and plastic zone of the surrounding rock advanced of the working face could provide the necessary basis for the surrounding rock control.

As shown in Figures 10 and 11:

- (1) In the vicinity of the working face, the peak and range of deviatoric stress in coal pillar were much larger than that in virgin coal area, and the peak ratio was 1.41, indicating that the stress in the surrounding rock was mainly sustained by the coal pillar. P were more than 90% in I and II, and 100% in III and IV, representing that extensive damage occurred in the surrounding rock of GED, that meant reinforced-supported measures should be implemented to avoid destabilization of the coal pillar.

- (2) At 7 m ahead of the working face, the peak deviatoric stress in the coal pillar reached the maximum value and was much smaller than that of virgin coal area with a peak ratio of 0.68, and the mining-induced stress of the working face was mainly undertaken by virgin coal area. P were still more than 90% in I and II, and 100% in III and IV. P dropped rapidly by 15% in II, and the value in I did not vary significantly compared to that of the vicinity of the working face.
- (3) At 30 m ahead of the working face, the discrepancies of peak deviatoric stress on both sides of entry were further reduced with a peak ratio of 0.81. It stated that the mining-induced stress was progressively performed by virgin coal area in cooperation with coal pillar. P in III and IV diminished promptly, and P were virtually invariable in I and II, indicating that the stress disturbance of the working face was fading quickly.
- (4) At 30 m ahead of the working face, the peak ratio of deviatoric stress between coal pillar and virgin coal area was 0.84. P in III and IV were further depressed by 9% and 7%, and P in I and II kept constant, indicating that the extent of damage to the roadway had steadily stabilized.

The above analysis suggested that the plasticization factor P in III and IV were more significantly troubled by mining-induced stress of the working face. P firstly dropped from both much more than 100% to 40% and 60% in III and IV, respectively. The deviatoric stress in coal pillar and virgin coal area first increased rapidly to the peak value, and then gradually tended to be stable. That indicated that the main bearing body of overburden stress was “the coal pillar → the virgin coal area → the collaboration of coal pillar and virgin coal area”.

4.3.3. Coal Pillar Width Determination

Based on the theoretical calculation and field measurement results, the fracture position of the main roof is located in the range of 5~6 m above the virgin coal from edge of the gob. That means the fracture line of the main roof is above the coal pillar when the width is 8 m, which contributes to maintaining the stability of the surrounding rock of the gob-side entry. The overlying strata load of GED is performed by virgin coal area in cooperation with coal pillar. The plasticization factors of coal pillar subdivision and top coal subdivision are both less than 80%, which are within the control of support. The surrounding rock in GED, advanced 60 m of the working face, is obviously affected after the panel retreated, and the influence on the stress and plastic zone of the surrounding rock tends to be stable. Therefore, it is feasible to set up an 8 m coal pillar with targeted support techniques to maintain the stability of the surrounding rock of GED.

5. Surrounding Rock Control Techniques

5.1. Cracks Distribution of the Coal Body

The results of borehole-scoping in 26,102 tailgate are shown in Figure 12:

- (1) On the coal pillar side of GED. The borehole wall within 2.08 m was relatively broken from the horizontal distance of the coal pillar rib of the roadway, indicating that the integrity of coal body was poor. The integrity of the coal body was enhanced within 2.08~6.07 m from the coal pillar rib with certain cracks existing. When the horizontal distance from the gob edge of coal pillar was less than 1.93 m, the borehole wall was seriously damaged or even collapsed, making it impracticable for further observation.
- (2) On the virgin coal side of GED. The borehole wall within 1.84 m from the virgin coal rib of the roadway was relatively broken, and the borehole wall integrity was improved with increasing borehole depth. Slightly developed cracks existed in the borehole within 1.84~3.76 m from virgin coal rib. Borehole wall was gradually smooth at a horizontal distance of over 3.79 m from virgin coal rib, that indicated good coal body integrity.

- (3) In the roof of GED. The coal body was severely damaged at a depth of 1.00 m from upper corner of coal pillar rib, 1.95 m from the roof of the roadway, and 1.76 m from the upper corner of virgin pillar rib. The coal body gradually turned intact with the distances exceeding 1.76 m, 4.72 m, and 3.97 m, respectively.

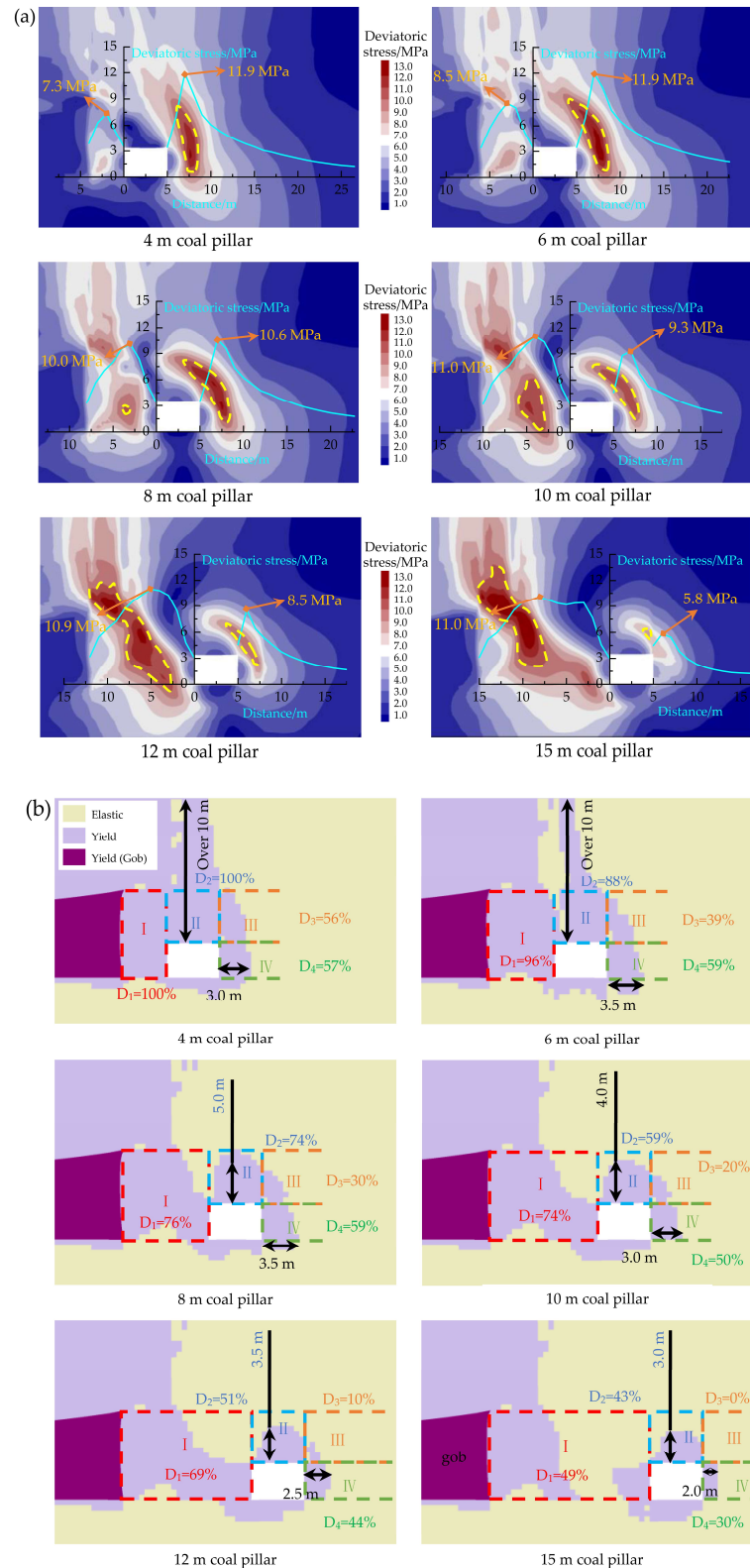


Figure 8. Characteristics of the surrounding rock with various coal pillar widths. (a) Deviatoric stress. (b) The plastic zone.

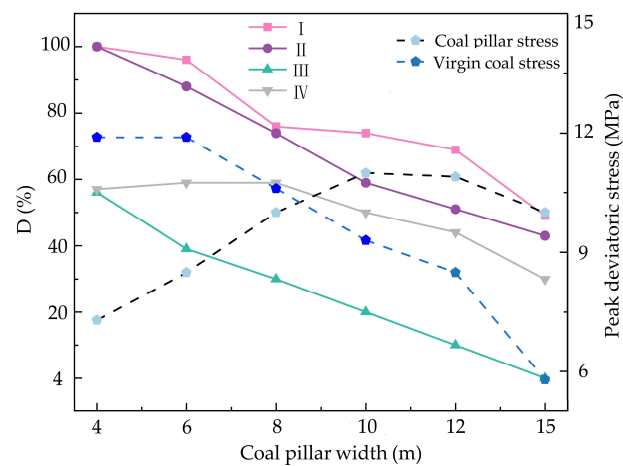


Figure 9. Distribution of the plasticization factor in subdivisions.

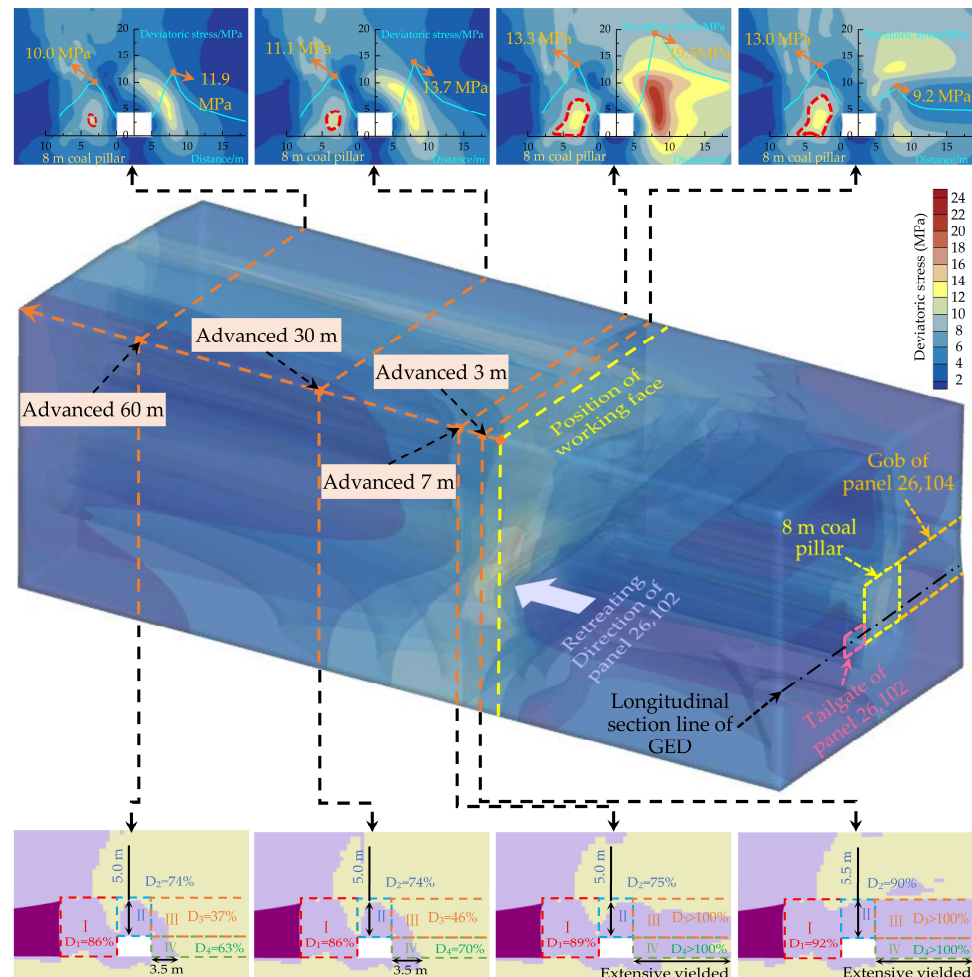


Figure 10. The distribution characteristics of surrounding rock advanced of the working face.

5.2. Support Principles

Based on the crack distribution of the surrounding rock, the compressive stress zone of bolts should cover the fracture zone, and the length of anchor cables should be greater than the depth of plastic zone, which played an effective role in improving the stress situations of the surrounding rock. The control techniques of the surrounding rock with the cooperative bearing of bolts and anchor cables were proposed, and the support principles are shown in Figure 13. The main contents were as follows: (1) Shallow bearing area of bolts. The 2.4 m

prestressed bolts were used in the shallow part of the surrounding rock, supplemented by high strength plates and W-shaped steel straps, to provide combined compressive stress for the fractured zone, which contributed to preventing support components from failure caused by the surrounding rock falling of the fractured zone. (2) Deep bearing zone of anchor cables. The 8.0 m high prestressed anchor cables were used in the roof and were embedded in the stable strata by passing through the top coal of about 6 m thick in the roof. The 5.0 m high prestressed anchor cables were used in both ribs to pass through the fracture zone and anchored into the relatively intact coal body. The high pre-stressed anchor cable could increase the shear resistance of the coal body, making the surrounding rock of the roadway to reinstate the state of three-dimensional stress in a certain extent, and exert the bearing capacity of the deep surrounding rock of the roadway.

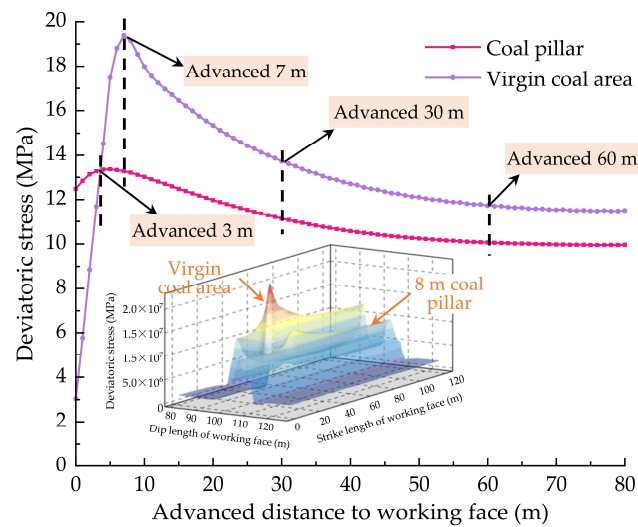


Figure 11. The stress distribution advanced of the working face in GED.

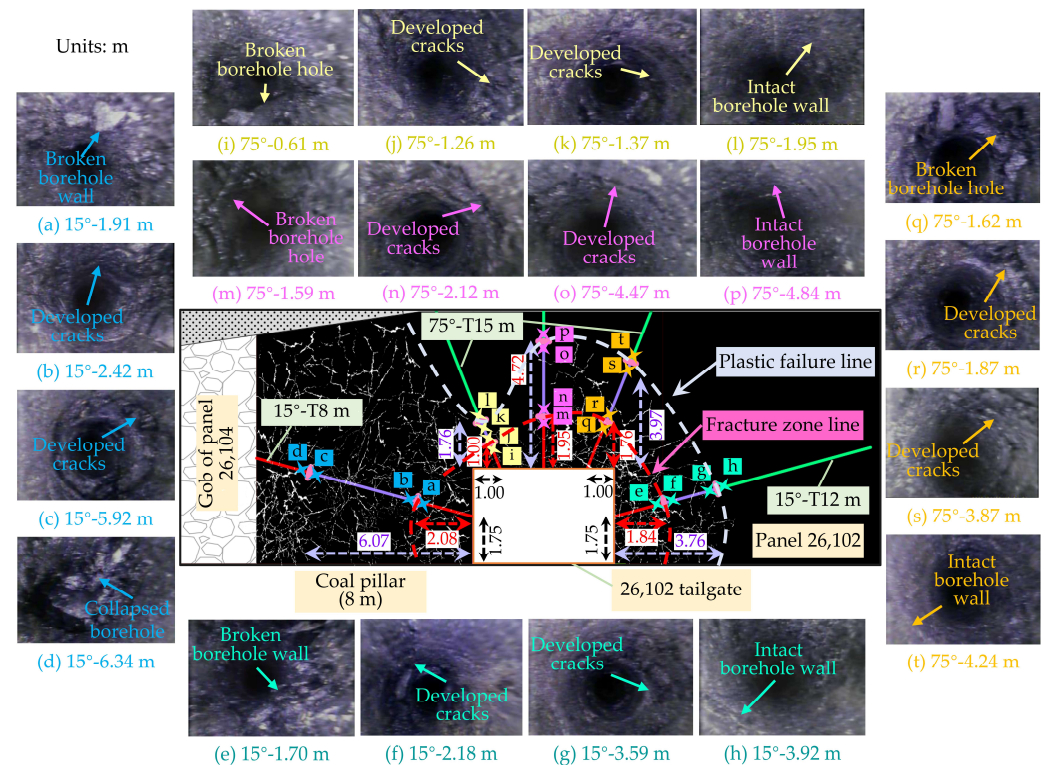


Figure 12. Borehole-scoping for the coal body in 26,102 tailgate.

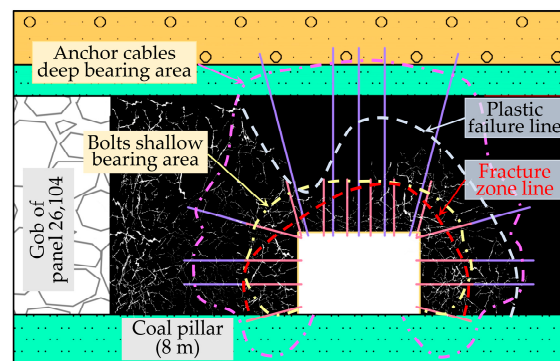


Figure 13. Cooperative bearing structure of bolts and anchor cables.

5.3. Technical Measures

Based on the field geological conditions, the support scheme of 26,102 tailgate is shown in Figure 14. The specific support parameters are as follows:

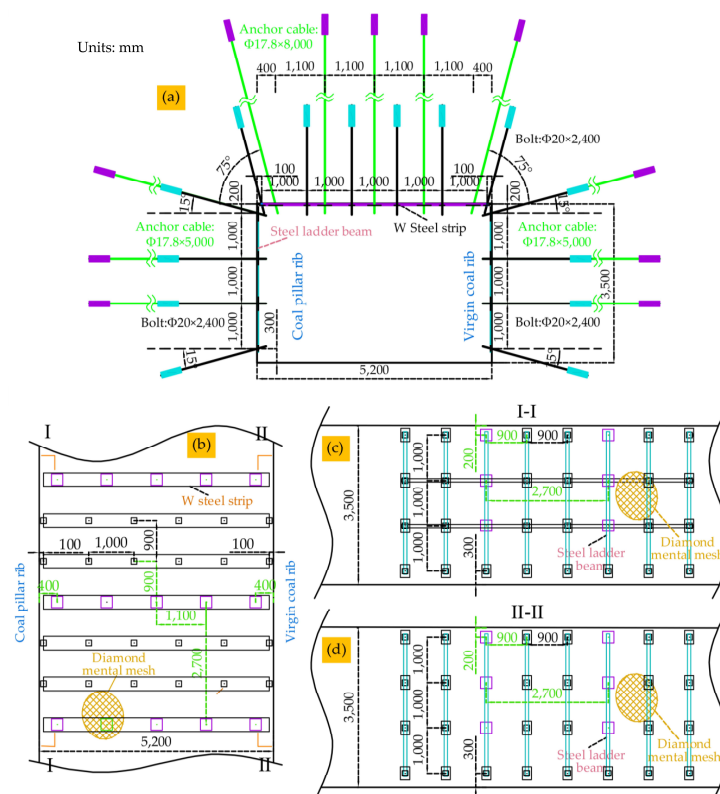


Figure 14. Detailed support parameters. (a) Entry section. (b) Support pattern in roof. (c) Support pattern in coal pillar rib. (d) Support pattern in virgin coal rib.

The roof and two ribs were supported by high-strength steel bolts 20 mm in diameter and 2400 mm in length. The inter-row spacing of the bolts was 1000 mm × 900 mm. A row of anchor cables is arranged for every two rows of bolts. The anchor cables 17.8 mm in diameter and were used for roadway support with a length of 8000 mm in roof and 5000 mm in two ribs. The inter-row spacing for the cables was 1100 mm × 2700 mm. The bottom one of each row of anchor cables was replaced by a high-strength steel bolt 20 mm in diameter and 2400 mm in length for both ribs. The bolts and anchor cables were connected with steel ladder beams made of 16 mm round steel in both ribs and with W-shaped steel straps with length × width × thickness of 5000 mm × 300 mm × 3 mm in the roof. Bolts in the roof and rib corners were installed at a 15° incline. Bolt plates of W-shaped steel straps

were selected for the bolts in both ribs. A metal mesh 4 mm in diameter was used in the roadway section to prevent broken coal body from falling.

6. Application and Analysis

Field measurements of ground response can effectively and comprehensively reflect the working status of the support system and verify the effect of the roadway support scheme, which contributes to the stability of the roadway support. Three stations were set up in the 200 m test section of 26,102 tailgate, and each station included one group deformation observation of GED and one group of bolts and anchor cables forces monitoring. The station layout is shown in Figure 15.

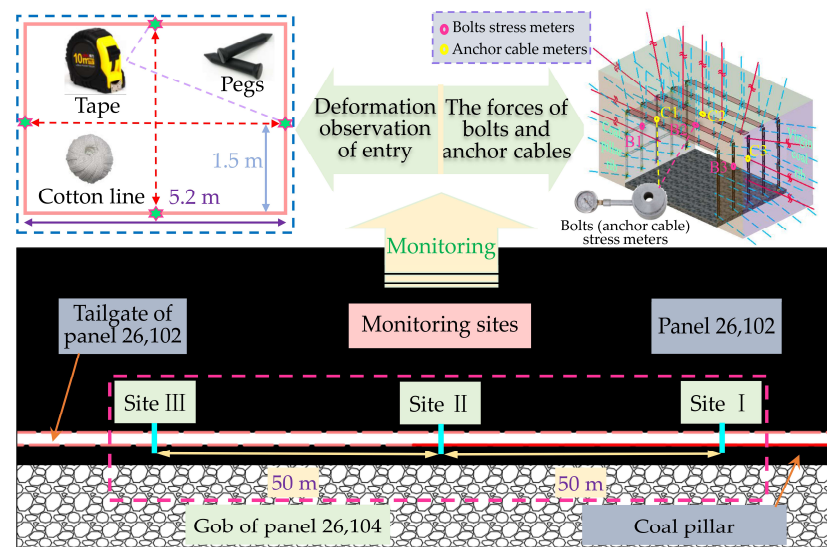


Figure 15. Layout of the field measurements.

Taking the monitoring results of the typical station II as an example, the surface deformation of the roadway surrounding rock is shown in Figure 16. With the combined support of high-strength prestressed bolts and anchor cables, there was no considerable deformation and damage in GED during roadway driving and retreating period. The surrounding rock deformation in the roof, virgin coal rib and coal pillar rib finally stabilized by 140 mm, 96 mm, and 105 mm separately after 25 days of roadway development. During the driving period, the maximum deformation of roadway in roof, virgin coal rib, and coal pillar rib advanced of the working face was 296 mm, 272 mm, and 251 mm, respectively.

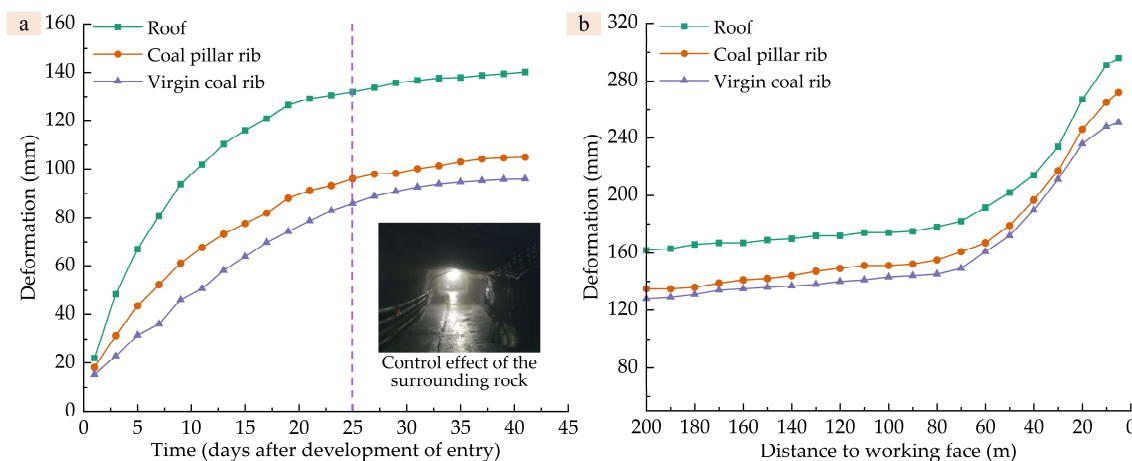


Figure 16. Deformation observation of the surrounding rock. (a) During roadway development. (b) During working face retreating.

At each monitoring section, the forces of bolts and anchor cables were monitored in the roof and two ribs, respectively. The measured bolts and anchor cables of the coal pillar rib, roof and virgin coal rib were numbered B1, B2, B3 and C1, C2, C3, respectively. The initial prestressing forces of bolts and anchor cables were 72~80 kN and 175~190 kN, which were 40~45% and 34~37% of the breaking load separately. The monitoring results are shown in Figure 17.

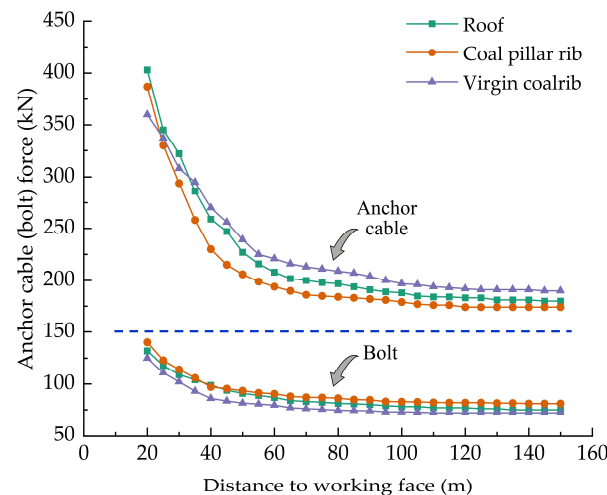


Figure 17. Anchor cable (bolt) force at different distances from the working face.

When the monitoring section was 60 m away from the working face, the forces of B1, B2, and B3 bolts were about 51%, 49%, and 45% of their breaking load (179 kN), and the forces of C1, C2, and C3 anchor cables were about 37%, 40%, and 43% of their breaking load (520 kN). The growth of bolts and anchor cables forces was less than 10%, indicating that the roadway section was weakly affected by the mining-induced stress at a distance of over 60 m from the working face. The disturbance impact of abutment stress was rapidly intensified once the monitoring section was less than 60 m from the working face. The forces of B1, B2, and B3 bolts were about 78%, 74%, and 70% of their breaking load, and the forces of C1, C2, and C3 anchor cables were about 75%, 78%, and 69% of their breaking load with 20 m from the working face. The forces of bolts and anchor cables were far less than its upper breaking limit, indicating that they were in good working condition. The 20 m area of GED advanced of the working face was supported by single hydraulic props, which contributed to avoiding the instability of the surrounding rock caused by intense disturbance of mining-induced stress. Therefore, the monitoring of mining pressure in this area could not be highlighted. Field measurements of ground response showed that the combined control techniques of bolts and anchor cables with an 8 m coal pillar achieved effective control of the roadway surrounding rock under the influence of upper residual coal pillar.

7. Conclusions

Based on theoretical calculation, numerical simulation, and field measurements, the evolution laws of the lateral abutment stress in lower working face at different relative distances to the RCP were studied, as well as the whole space-time evolution law of deviatoric stress and plastic zone of GED during driving and retreating periods with various coal pillar widths under the influence of RCP. The targeted support techniques with an 8 m coal pillar were proposed. The conclusions were as follows:

- (1) The above distribution laws indicated that the lateral abutment stress was most significantly affected when the edge of the lower working face was located directly below the RCP, and the peak stress growth coefficient was 1.20~1.30. When the EWF of the lower coal seam was 2.5 m away from the centerline of RCP, the peak growth coefficient of the lateral abutment stress was 1.22. The influence of stress disturbance of RCP was relatively diminished when a small coal pillar was set up to drive the entry along the gob in such conditions. Consequently, the superimposed disturbance impact of RCP and mining-induced stress on GED should be fully considered to prevent the roadway from instability.
- (2) The mechanical parameters of the rock and coal mass were obtained using laboratory experiments coupled with Roclab software. These parameters were substituted into the established main roof structure mechanics model to derive the breakage position of the main roof with the influence of RCP, which was verified the rationality of the calculation results using borehole-scoping.
- (3) FLAC3D was used to study the whole space-time evolution law of deviatoric stress and plastic zone of GED during driving and retreating periods with different coal pillar widths under the influence of RCP, and the plasticization factor P was introduced to quantify the evolution of the plastic zone in four subdivisions of the roadway surrounding rock. The results showed that when the width of the coal pillar was 8 m, P were not more than 80% in I and II, and 60% in III and IV. The peak ratio of deviatoric stress between coal pillar and virgin coal area was 0.84. Additionally, the plasticization factor P in the virgin coal upper corner subdivision and virgin coal subdivision were more significantly troubled by mining-induced stress of the working face. P firstly dropped from both being much more than 100% to 40% and 60% in III and IV.
- (4) Field measurements of ground response showed that the combined control techniques of bolts and anchor cables with an 8 m coal pillar achieved effective control of the roadway surrounding rock under the influence of the residual coal pillar. The maximum deformation of roadway in the roof, virgin coal rib and coal pillar rib advanced of the working face was 296 mm, 272 mm, and 251 mm separately during the retreating period. The forces of bolts and anchor cables were 70%~78% and 69%~75% of their breaking load during the retreating period, and the supporting components were all in good working conditions, which realized a valid effect of surrounding rock control. This provided a reference for promoting the coal recovery rate in similar geological conditions.

Author Contributions: Conceptualization, S.X. and F.G.; methodology, F.G. and Y.W.; software, F.G. and Y.W.; data curation, F.G. and Y.W.; writing—original draft preparation, F.G.; writing—review and editing, S.X. and F.G.; supervision, S.X.; project administration, S.X., F.G. and Y.W. All authors have read and agreed to the published version of the manuscript.

Funding: This work is financially supported by the National Natural Science Foundation of China (Grant No. 52074296), the Fundamental Research Funds for the Central Universities (Grant No. 2022YJSNY18), the National Natural Science Foundation of China (Grant No. 52004286), the Fundamental Research Funds for the Central Universities (Grant No. 2022XJNY02), and the China Postdoctoral Science Foundation (Grant No. 2020T130701, 2019M650895), all of which are gratefully acknowledged.

Conflicts of Interest: The authors declare no conflict of interest.

References

1. Zhang, Z.Z.; Bai, J.B.; Chen, Y.; Yan, S. An innovative approach for gob-side entry retaining in highly gassy full-mechanized longwall top-coal caving. *Int. J. Rock Mech. Min. Sci.* **2015**, *80*, 1–11. [[CrossRef](#)]
2. Wu, R.; He, Q.; Oh, J.; Li, Z.; Zhang, C. A new gob-side entry layout method for two-entry longwall systems. *Energies* **2018**, *11*, 2084. [[CrossRef](#)]
3. An, Y.P.; Zhang, N.; Zhao, Y.M.; Xie, Z.Z. Field and numerical investigation on roof failure and fracture control of thick coal seam roadway. *Eng. Fail. Anal.* **2021**, *128*, 105594. [[CrossRef](#)]

4. Wang, J.C.; Yang, S.L.; Wei, W.J.; Zhang, J.W.; Song, Z.Y. Drawing mechanisms for top coal in longwall top coal caving (LTCC): A review of two decades of literature. *Int. J. Coal Sci. Technol.* **2021**, *8*, 1171–1196. [\[CrossRef\]](#)
5. Zhang, G.C.; Wen, Z.J.; Liang, S.J.; Tan, Y.L.; Tian, L.; Zhao, Y.Q.; Zhao, D.S. Ground Response of a Gob-side Entry in a Longwall Panel Extracting 17 m-Thick Coal Seam: A Case Study. *Rock Mech. Rock Eng.* **2020**, *53*, 497–516. [\[CrossRef\]](#)
6. Yan, H.; Zhang, J.X.; Feng, R.; Wang, W.; Lan, Y.W.; Xu, Z.J. Surrounding rock failure analysis of retreating roadways and the control technique for extra-thick coal seams under fully-mechanized top caving and intensive mining conditions: A case study. *Tunn. Undergr. Space Technol.* **2020**, *97*, 103241. [\[CrossRef\]](#)
7. Yan, H.; Zhang, J.X.; Ding, Z.W.; Huang, Y.L. Surrounding rock deformation mechanism of roadways with extra-thick coal seam. *Disaster Adv.* **2013**, *6*, 226–233.
8. Jiang, B.Y.; Gu, S.T.; Wang, L.G.; Zhang, G.C.; Li, W.S. Strain burst process of marble in tunnel-excavation-induced stress path considering intermediate principal stress. *J. Cent. South Univ.* **2019**, *26*, 984–999. [\[CrossRef\]](#)
9. Wu, W.D.; Bai, J.B.; Wang, X.Y.; Shuai, S.; Wu, S.X. Numerical study of failure mechanisms and control techniques for a gob-side yield pillar in the Sijiazhuang coal mine, China. *Rock Mech. Rock Eng.* **2019**, *52*, 1231–1241. [\[CrossRef\]](#)
10. Shi, X.S.; Jing, H.W.; Zhao, Z.L.; Gao, Y.; Zhang, Y.C.; Bu, R.D. Physical experiment and numerical modeling on the failure mechanism of gob-side entry driven in thick coal seam. *Energies* **2020**, *13*, 5425. [\[CrossRef\]](#)
11. Gao, F.Q.; Stead, D.; Kang, H.P.; Wu, Y.Z. Discrete element modelling of deformation and damage of a roadway driven along an unstable goaf—A case study. *Int. J. Coal Geol.* **2014**, *127*, 100–110. [\[CrossRef\]](#)
12. Gao, F.Q.; Stead, D.; Kang, H.P. Simulation of roof shear failure in coal mine roadways using an innovative UDEC Trigon approach. *Comput. Geotech.* **2014**, *61*, 33–41. [\[CrossRef\]](#)
13. Gao, F.Q.; Stead, D.; Kang, H.P. Numerical simulation of squeezing failure in a coal mine roadway due to mining-induced stresses. *Rock Mech. Rock Eng.* **2015**, *48*, 1635–1645. [\[CrossRef\]](#)
14. Liu, X.; He, M.; Wang, J.; Ma, Z. Research on non-pillar coal mining for thick and hard conglomerate roof. *Energies* **2021**, *14*, 299. [\[CrossRef\]](#)
15. Wang, K.; Huang, Y.; Gao, H.; Zhai, W.; Qiao, Y.; Li, J.; Ouyang, S.; Li, W. Recovery technology of bottom coal in the gob-side entry of thick coal seam based on floor heave induced by narrow coal pillar. *Energies* **2020**, *13*, 3368. [\[CrossRef\]](#)
16. Batugin, A.; Wang, Z.Q.; Su, Z.H.; Sidikovna, S.S. Combined support mechanism of rock bolts and anchor cables for adjacent roadways in the external staggered split-level panel layout. *Int. J. Coal Sci. Technol.* **2021**, *8*, 659–673. [\[CrossRef\]](#)
17. He, W.R.; He, F.L.; Zhao, Y.Q. Field and simulation study of the rational coal pillar width in extra-thick coal seams. *Energy Sci. Eng.* **2020**, *8*, 627–646. [\[CrossRef\]](#)
18. Li, X.B.; Zhao, Y.Q.; He, W.R.; Li, L.; He, F.L. Study on coal pillar width and surrounding rock control of gob-side entry in extra-thick coal seam. *Geotech. Geol. Eng.* **2020**, *38*, 1–14. [\[CrossRef\]](#)
19. Yang, R.S.; Zhe, Y.; Li, Y.L.; Li, W.Y.; Lin, H. Coal pillar size design and surrounding rock control techniques in deep longwall entry. *Arab. J. Geosci.* **2020**, *13*, 1–14. [\[CrossRef\]](#)
20. Qi, W.Y.; Zhang, Q.; Zhang, J.X.; Zhang, J.G.; Zhu, C.L. Design of coal pillars of gob-side entry between mining faces with large differences in mining height in deep mine. *Energy Sources Part A Recovery Util. Environ. Eff.* **2020**, *42*, 2648–2663. [\[CrossRef\]](#)
21. Wang, Q.; Gao, H.K.; Jiang, B.; Li, S.; He, M.C.; Wang, D.C.; Li, W.; Gao, S.; Yang, H.C. Research on reasonable coal pillar width of roadway driven along goaf in deep mine. *Arab. J. Geosci.* **2017**, *10*, 466. [\[CrossRef\]](#)
22. Ma, Z.; Chen, C.; Liang, X.; Chen, A.; Song, W. Field and numerical investigation on the stability of coal pillars of gob-side entry driving with top coal. *Arab. J. Geosci.* **2020**, *13*, 1–11. [\[CrossRef\]](#)
23. Jiang, L.S.; Zhang, P.P.; Chen, L.J.; Hao, Z.; Sainoki, A.; Wang, Q.B. Numerical Approach for Goaf-Side Entry Layout and Yield Pillar Design in Fractured Ground Conditions. *Rock Mech. Rock Eng.* **2017**, *50*, 3049–3071. [\[CrossRef\]](#)
24. Han, C.L.; Yang, H.Q.; Zhang, N.; Deng, R.J.; Guo, Y.X. Zoning control technology of gob-side roadway driving with small coal pillar facing mining in a special isolated island working face: A case study. *Appl. Sci.* **2021**, *11*, 10744. [\[CrossRef\]](#)
25. Wang, E.; Xie, S.R. Determination of coal pillar width for gob-side entry driving in isolated coal face and its control in deep soft-broken coal seam: A case study. *Energy Sci. Eng.* **2022**, 1–12. [\[CrossRef\]](#)
26. Zha, W.H.; Shi, H.; Liu, S.; Kang, C.H. Surrounding rock control of gob-side entry driving with narrow coal pillar and roadway side sealing technology in Yangliu Coal Mine. *Int. J. Min. Sci. Technol.* **2017**, *27*, 819–823. [\[CrossRef\]](#)
27. Xie, S.R.; Pan, H.; Chen, D.D.; Gao, M.M.; He, S.S.; Song, B.H. The Structure and activity laws of interlayer equivalent basic roof in mult-seam mining. *J. China Univ. Min. Technol.* **2016**, *46*, 1218–1225.
28. Zhang, C.L.; Zhang, Y.; Zuo, J.P.; Gao, S.Y. Coal seam mining in a physical model vis-à-vis MATLAB analysis and geological radar. *Min. Metall. Explor.* **2021**, *38*, 897–911.
29. Xie, S.R.; Wu, X.Y.; Chen, D.D.; Sun, Y.H.; Wang, E.; Wu, X.W.; Duan, X.B.; Jiang, Z.S.; Qi, P.Y.; Shi, S.H. Failure mechanism and control technology of thick and soft coal fully mechanized caving roadway under double gobs in close coal seams. *Shock Vib.* **2020**, *2020*, 8846014. [\[CrossRef\]](#)
30. Wang, M.; Bai, J.B.; Li, W.F.; Wang, X.Y.; Cao, S.G. Failure mechanism and control of deep gob-side entry. *Arab. J. Geosci.* **2015**, *8*, 9117–9131. [\[CrossRef\]](#)
31. Yan, S.; Bai, J.B.; Wang, X.Y.; Huo, L.J. An innovative approach for gateroad layout in highly gassy longwall top coal caving. *Int. J. Rock Mech. Min. Sci.* **2013**, *59*, 33–41. [\[CrossRef\]](#)
32. Hoek, E.; Brown, E.T. Practical estimates of rock mass strength. *Int. J. Rock Mech. Min. Sci.* **1997**, *34*, 1165–1186. [\[CrossRef\]](#)

33. Hoek, E.; Carranza-Torres, C.T.; Corkum, B. Hoek-Brown failure criterion–2002 edition. In Proceedings of the Fifth North American Rock Mechanics Symposium, Toronto, ON, Canada, 7–10 July 2002; pp. 18–22.
34. Zhang, G.C.; He, F.L.; Jiang, L.S. Analytical analysis and field observation of break line in the main roof over the goaf edge of longwall coal mines. *Math. Prob. Eng.* **2016**, *2016*, 4720867. [[CrossRef](#)]
35. Zhu, C.L.; Zhang, J.X.; Li, M.; He, Z.W.; Wang, Y.Y.; Lan, Y.W. Effect mechanism of strata breakage evolution on stope deformation in extra-thick coal seams. *Alex. Eng. J.* **2022**, *61*, 5003–5020. [[CrossRef](#)]
36. Long, Y.Q. *Calculation of Elastic Foundation Beam*; People's Education Press: Beijing, China, 1981.
37. Wang, M.; Li, W.F. Analysis on the loading mechanism of caving gangue in goaf based on Salamon model. *Coal Sci. Tech.* **2015**, *41*, 50–54. [[CrossRef](#)]
38. Salamon, M. Mechanism of caving in longwall coal mining. Rock mechanics contributions and challenges. In Proceedings of the 31st US Symposium. on Rock Mechanics, Golden, CO, USA, 18–20 June 1990; pp. 161–168.
39. Liu, R.; Zhu, Q.J. An analysis of the impact of deviatoric stress and spherical stress on the stability of surrounding rocks in roadway. *Geofluids* **2021**, *2021*, 3095690. [[CrossRef](#)]
40. Chen, D.D.; Wang, E.; Xie, S.R.; He, F.L.; Wang, L.; Zhang, Q.; Wu, X.Y.; Jiang, Z.S.; Li, Y.B.; Shi, S.H. Roadway surrounding rock under multi-coal-seam mining: Deviatoric stress evolution and control technology. *Adv. Civ. Eng.* **2020**, *2020*, 9891825. [[CrossRef](#)]