

Research Article

Research on the Technology of Small Coal Pillars of Gob-Side Entry Retained in Deep Mines Based on the Roof Cutting for Pressure Unloading in the Lower Key Stratum

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With the mining depth getting deeper and deeper, surrounding rock control technology has become a difficult issue in the development of modern mining, especially the mining roadways affected by mining. It is very difficult to control the surrounding rock due to the intense mining of the last working faces. In order to alleviate the impact of dynamic pressure on the gob-side entry retained, solve the ventilation difficulties caused by the commonly used method of the gob-side entry retained without leaving coal pillars, and solve the disadvantages of the inability to effectively isolate water and harmful gases, this paper proposes a small coal pillar gob-side entry retained technology based on the key stratum, that is, cutting the key stratum above the top plate, transferring lateral support pressure from the workings further up the seam to the depths of the coal seam, and then protecting gob-side entry retained method. In the final method of precracking and cutting joints by blasting with D-type polytunnel, we directionally cut the lower key stratum above the roadway in the designed direction and position. The theoretical analysis, numerical simulation and field test, and the method of the blasting broken roof pressure release effect are analyzed. Compared to the unloaded pressure method, the relatively complete direct roof is conducive to the maintenance of the surrounding rock on the gob-side entry retained. At the same time, the pressure relief caused by the fracture of the lower key stratum of the coal roof significantly improves the stress environment. The stress is no longer concentrated on the small coal pillars and the gob-side entry retained, and the required roadway support strength is greatly reduced and no longer needs to carry out the construction of roadside support system. This technology reduces labor intensity and production costs.

1. Introduction

When shallow coal resources become increasingly depleted, mining depths are increasing year on year. The coal seam conditions are becoming increasingly complex, and problems such as rock burst are a serious threat to mining safety. At the same time, the problem of mining succession tension is also becoming increasingly prominent, restricting the safe and efficient production of the mine. We have to adopt the gob-side entry retained to solve these problems, such as difficulties in ventilating the back hole in deep seam and high gas mines and the tension in the mining succession [1, 2].

The gob-side entry retained technology has the advantage of increasing the recovery rate of coal resources and achieving Y-shaped ventilation [3]. However, the current technique of the gob-side entry retained without coal pillar. It cannot effectively solve technical problems such as ventilation difficulties, air leakage from mining areas, and support difficulties in deep thick coal seam high gas mines. In recent years, academician He et al. proposed the roof cutting and pressure releasing automatically form a no-pillar entry mining technology [4–7]. But cutting technology of the gob-side entry retained requires superstrong support roadway, which is difficult and costly to support. And it is difficult to block

the harmful gas in the alley side barrier system. In addition, with the increase of mining depth, the roof cutting and pressure releasing automatically formed entry mining technology is influenced by high pressure. According to the statistical results, roof cutting and pressure releasing automatically formed entry technology is less applied under the condition that the depth of the well exceeds 650 m, and the top-cutting effect is not good [8].

Now, many experts and scholars at home and abroad on deep high stress entry surrounding rock deformation mechanism and stability problem have carried out a number of studies. Chen et al. [9] deep cut along the empty entry support difficult problem, by building control roof, lateral pressure, dynamic pressure resistance, and protection of coal for coordinated control. Yang et al. [10] for composite roof gob-side entry retained for filling body of inward problems build the mechanics model and obtained the displacement expression, and shear bolts are presented in this paper, as well as additional entry control scheme of the isolated column. Wu et al. [11] aiming at the gob-side entry retained the easy occurrence of delamination failure and divided the deformation and failure forms of the roof of deep well gob-side entry retained into four types; “more supporting structure” is put forward at the same time, the reasonable control of the roof abscission layer. Through practical engineering practice, Zhang et al. [12] and others found that the deformation of deep gob-side entry retained entry was mainly concentrated during mining and retained entry in working face and proposed to adopt the whole process support of roof support and retained entry with anchor, net, and cable combined with single hydraulic prop and build the wall by pumping and filling paste material, which can realize retained entry under the mining depth of kilometers. Zheng et al. [13] put forward subsection gob-side entry retained technology for deep high gas mines, aiming to solve the same problem and realize coal and gas comining. Hua et al. [14] took the deep well gob-side entry retained of Guqiao Coal Mine in Huainan as the engineering background and studied the evolution mechanism of floor heave of deep well gob-side entry retained. It is found that the phenomenon of entry floor heave is obvious during the first mining to the second mining. Xingen et al. [15] analyzed the stress state of hydraulic support and the deformation mechanism of roof of retreat channel according to statics theory and energy theory. Then, according to the geological conditions of the working face, the cutting design of the roof of the retracement trough is determined, and the cutting effect is analyzed by numerical simulation.

The No. 201 working face in Gaojiapu Coal Mine is a typical deep mining face with large mining height, which adopts double entry layout. In order to alleviate the problem of tight replacement, the headentry (No. 201 working face) is reserved as the tailentry in the No. 202 working face. Under the influence of high ground pressure and mining, the effect of retained entry is poor, and the entry roof fall and side slope appear. Under this background, a method of deep mine small coal pillar gob-side entry retained based on broken roof pressure relief is proposed; that is, the lower part key stratum above the entry roof is cut off while the direct roof is relatively intact. In this paper, the working principle

and test effect of this technology are studied and analyzed by various means, and the stress distribution state and surrounding rock failure law of broken roof pressure relief gob-side entry retained are obtained, thus verifying the feasibility of the gob-side entry retained method of small coal pillars in deep wells based on broken roof pressure relief key stratum. The research results can provide useful reference for entry stability control under similar conditions.

2. Based on the Principle of Small Coal Pillar Gob-Side Entry Retained in Deep Mine with Broken Roof and Pressure Relief in the Lower Part of Coal Seam Roof Key Stratum

Academician Song Zhenqi of Shandong University of Science and Technology put forward the theory of transfer rock beam, which regards each group of rock strata moving at the same time (or nearly moving at the same time) as a moving whole, called “transfer rock beam” or “transfer rock beam” for short; that is, when the working face is excavated, the supporting pressure in the stope roof is transferred in a relatively complete rock body, as shown in Figure 1. In this paper, the technical method of roof breaking and pressure relief is put forward; that is, presplitting drilling holes inclined to the solid coal side are drilled in the coal seam roof, explosives are installed in the drilling holes, and presplitting slits are formed in the lower part key stratum of the coal seam roof by explosive blasting, and the lower part key stratum is cut off so that it cannot transmit side abutment pressure caused by mining in the working face. In the process of mining in the working face, the gravity from the lower part key stratum of the coal seam roof to the surface rock layer is transferred from the thick and hard part key stratum above the lower part key stratum to the solid coal, as shown in Figure 2, so that the small coal pillars and gob-side entry retained are under lower supporting pressure, which is beneficial to the bearing stability of the small coal pillars and the maintenance of the gob-side entry retained.

3. Geological Conditions

3.1. Overview of No. 2 Panel. Gaojiapu Coalfield is located in the northwest of Binchang mining area in Shaanxi Province, China. The test working faces (No. 201 and No. 202) are in the second panel. The average thickness of coal seam is about 6.48 m, and the buried depth of working face is about 1000 m. The coal seam structure is relatively simple. The immediate roof is mainly composed of mudstone, with an average thickness of 3.66 m; the main roof is composed of fine sandstone, with an average thickness of 9.06 m. The rock conditions are shown in Table 1.

3.2. Overview of Gob-Side Retained in Working Face. Due to the large mining depth of Gaojiapu Coal Mine, the district entry arrangement is adopted in the way of double entry layout, but the entry protection with wide coal pillars easily induce rock burst. In order to solve the above technical problems, it is necessary to adopt the layout of gob-side

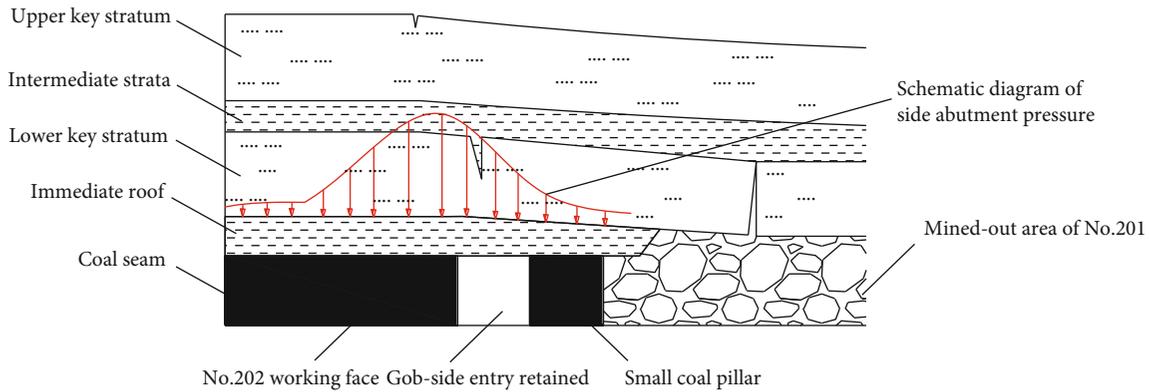


FIGURE 1: Distribution diagram of abutment pressure before roof cutting.

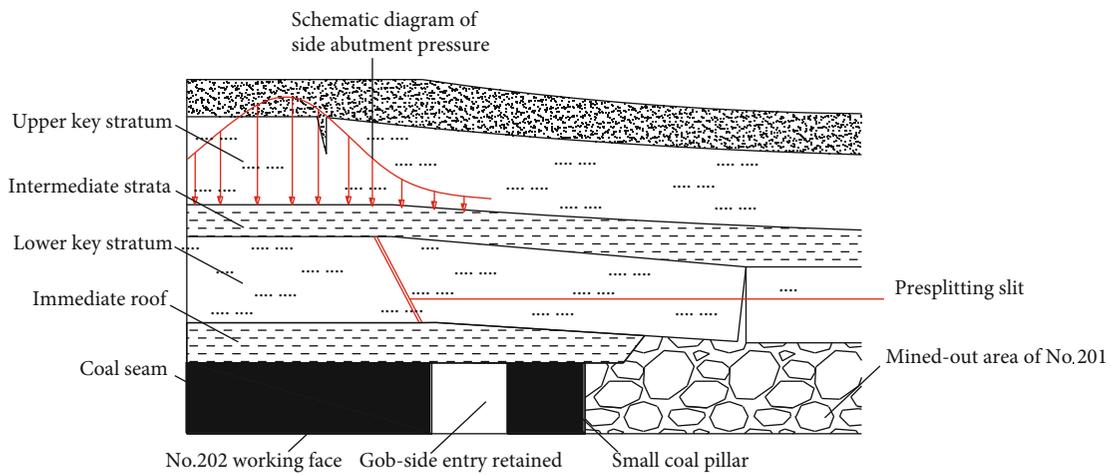


FIGURE 2: Distribution diagram of abutment pressure after roof cutting.

TABLE 1: Illustrative table of support sections of the gob-side entry retained.

Cross section	17.88 m ²	Anchor name	Rebar resin anchors	Quantity of anchor rods	16.9 (m)
Net cross section	16.69 m ²	Specification for roof anchors	MSGLW-500 22/2200 mm	Anchor ropes	SKP18 – 1/1860 mm × 7000 mm Spacing: 1900 mm
Shape	Trapezoidal	Gang anchor specifications	MSGLD-500 20/2200 mm	Anchor ropes pallets	250 mm × 250 mm × 20 mm
Slope	Design slope	Row spacing	950 mm	Roof metal mesh	LW50/4-SZ
Steel belts	T-shaped	Anchor rods pallets	150 mm × 150 mm × 10 mm	Anchoring agent	Roof: MSCKb2350, MSCK2370 Gangs: MSCKb2370 Top anchor cable: MSCKb2350, MSCK2370

entry. The headentry and the tailentry of No. 201 working face were excavated in solid coal. The tailentry excavation of No. 202 working face lags behind the small coal pillar during the mining of No. 201 working face. The construction layout diagram is shown in Figure 3.

3.3. Determination of the Support Method for the Test Section of Roadway. The original 201 working face headentry and tailentry were designed as trapezoidal roadways. The width

of the trapezoidal roadways is 4.4 m at the top and 5.0 m at the bottom, with a clear height of 3.5 m and a net area of 16.45 m². Therefore, the test section of the gob-side entry retained (202 working face tailentry) in this manuscript took the same roadway shape and dimensions and used the same support parameters. The aim of this manuscript is to highlight the comparison of channel deformation variables before and after top cutting, so the same support parameters are used both before and after the roof cutting. The test

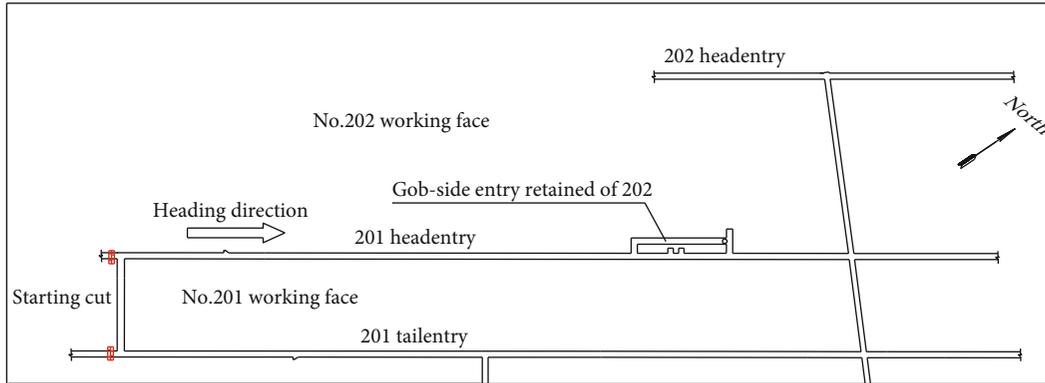


FIGURE 3: Schematic diagram of 202 gob-side entry retained test section.

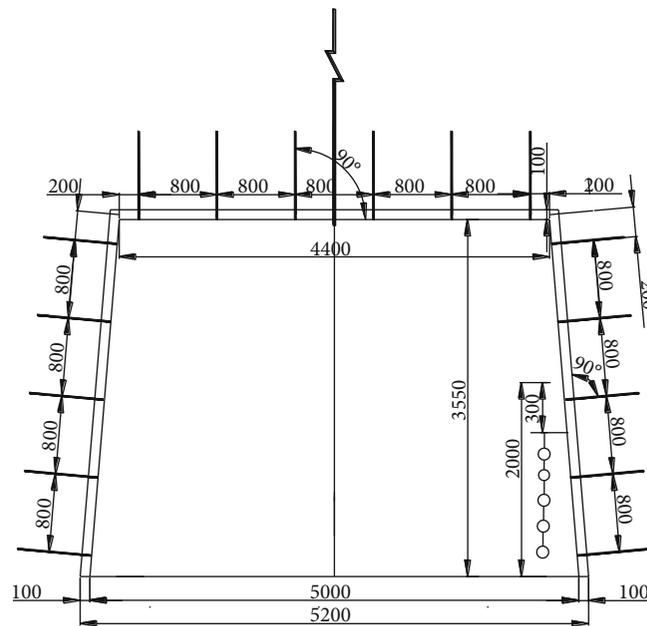


FIGURE 4: Roadway cross sections of the gob-side entry retained.

control variable is only whether or not the roof is broken, which is used to determine the effectiveness of the pressure relief from the broken roof. The roadway cross section is shown in Figure 4, and the support parameters are shown in Table 1.

3.4. Coal Seam Roof Conditions. It is very important for the technology to analyze the roof conditions of coal seam and determine the key stratum closest to the coal seam in the roof, namely, the lower key stratum. The lower key stratum is the rock layer with large thickness and hard rock that is closest to the immediate roof above the immediate roof and can transmit the stress of the overlying strata. According to Table 2, the immediate roof of No. 4 coal seam is mudstone (3.66 m), and the main roof is fine sandstone (9.9 m). There are two layers of thick mudstone and coarse sandstone above the main roof, and the position of the key stratum is not obvious. Therefore, according to the geological data of Gaojiapu Coal Mine, the FLAC^{3D} numerical calculation

model is established. The numerical calculation model takes 345 m in the x direction (dip), 300 m in the y direction (strike), and 72 m in the z direction. The simulation direction of the coal seam is horizontal, and the buried depth is 800 m. The horizontal displacement constraint is applied to the four boundaries, and the vertical displacement of the bottom is limited. The equivalent uniform load of the average 800 m buried depth is applied to the top boundary of the model by 20 MPa. The Mohr-Coulomb constitutive model is used for the contact model. The numerical model is shown in Figure 5, and the physical and mechanical parameters of each rock layer are shown in Table 2.

The calculation process of the numerical model is as follows: model establishment \rightarrow parameter assignment and initial balance calculation \rightarrow No. 201 working face headentry excavation and gob-side entry retained test simulation \rightarrow calculation result output and analysis. After the above procedures, the distribution law and range of side

TABLE 2: Rock mechanics parameters.

No.	Lithology	Bulk modulus GPa	Shear modulus GPa	Cohesion MPa	Internal friction angle °	Tension strength MPa	Density kg·m ⁻³	Height m
1	Mudstone	0.3	0.35	0.8	30	0.1	2400	14
2	Aluminous argillite	0.8	1.62	3.8	35	0.4	2450	2
3	4# coal seam	0.3	0.39	3.64	31	0.9	1390	8
4	Mudstone	0.3	0.35	0.8	30	0.1	2400	4
5	Fine sandstone	4.84	3.26	7.8	32	6.5	2650	10
6	Mudstone	0.3	0.35	0.8	30	0.1	2400	17
7	Coarse sandstone	6.28	5.09	9.9	34	5.6	2600	17

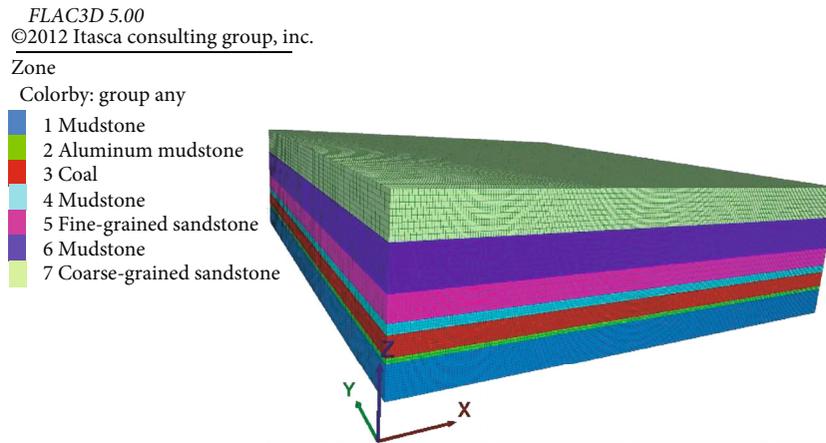


FIGURE 5: Numerical model diagram.

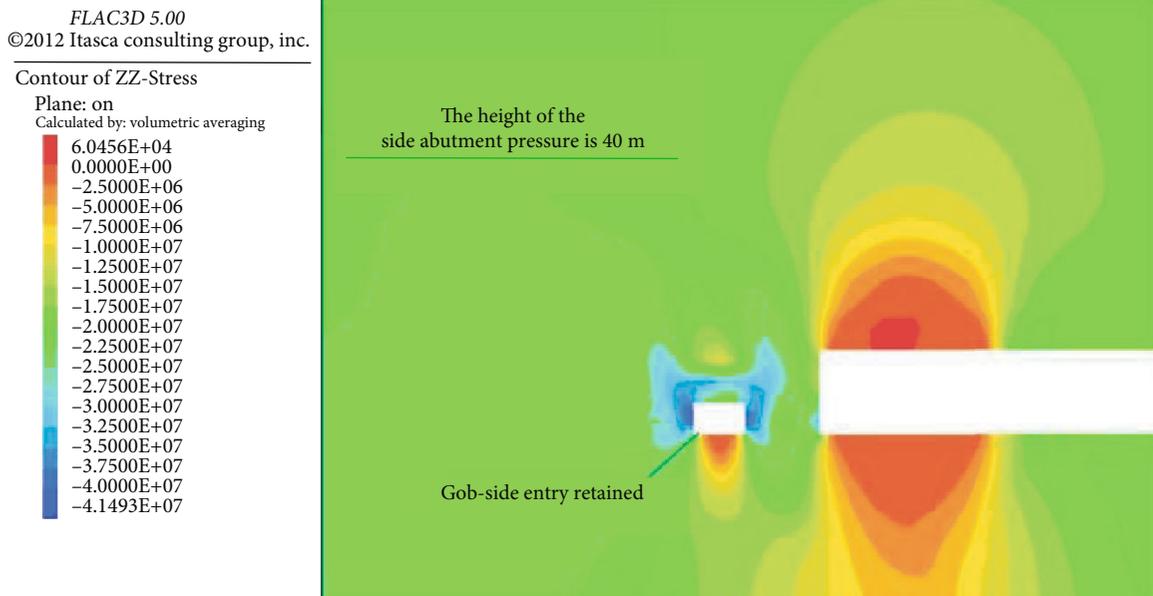


FIGURE 6: Nephogram of side abutment pressure distribution in the working face.

abutment pressure after mining in the No. 201 working face are analyzed to determine the position of the lower key stratum. Numerical simulation results are shown in Figure 6.

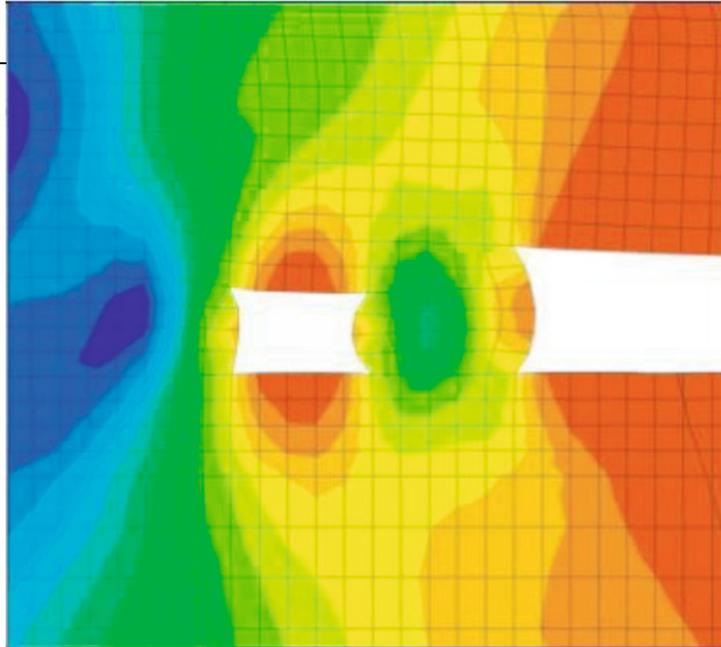
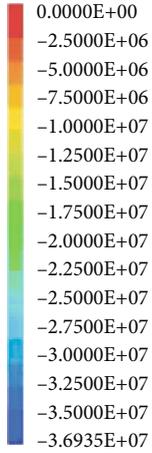
It can be seen from Figure 6 that the side abutment pressure generated after the mining of the No. 201 working face has obvious stress concentration phenomenon in the test

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Contour of ZZ-Stress

Plane: on

Calculated by: volumetric averaging



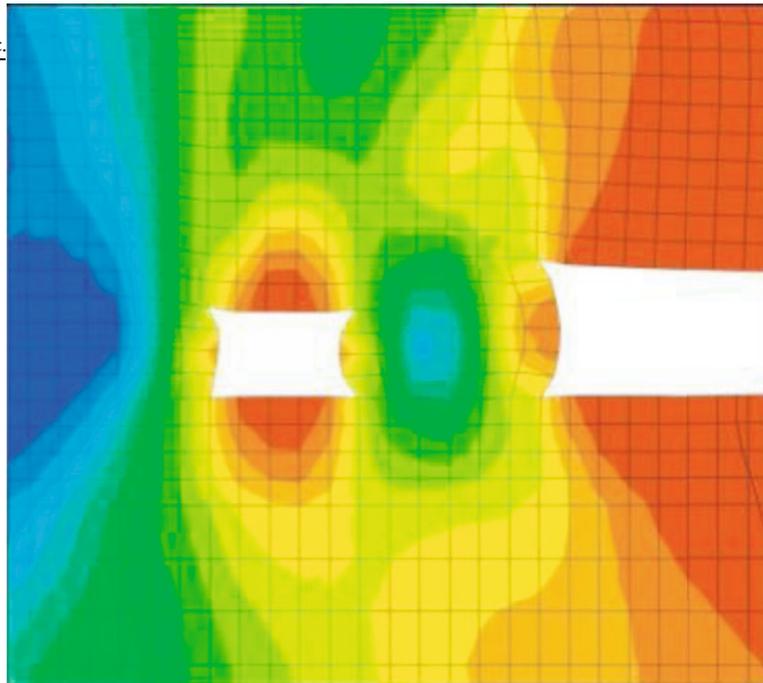
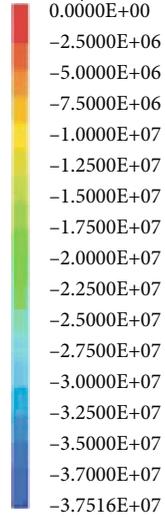
(a) 5 m coal pillar

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Contour of ZZ-Stress

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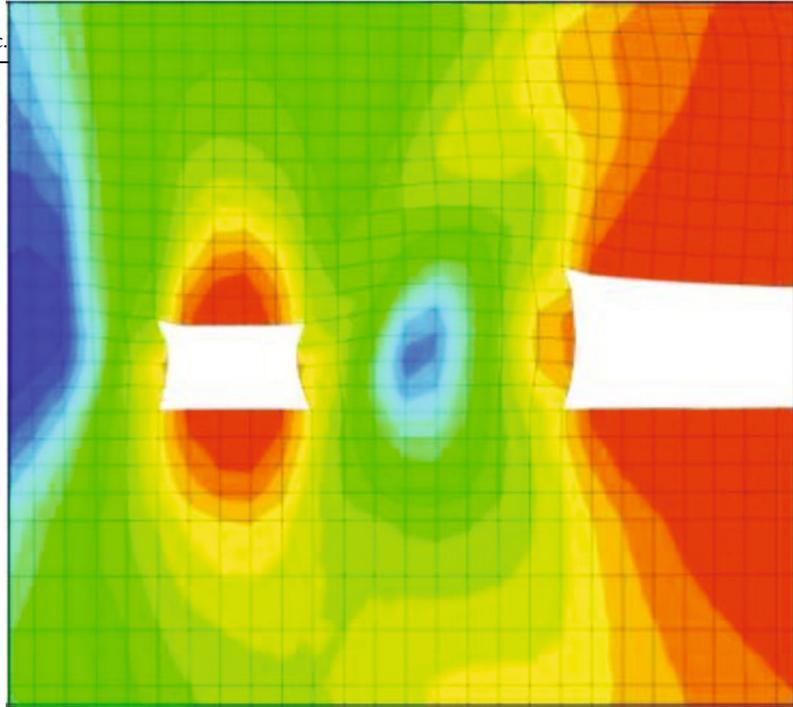
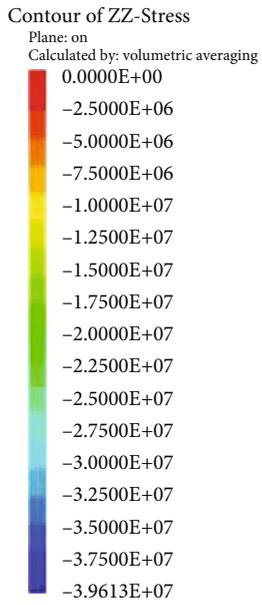
Calculated by: volumetric averaging



(b) 6 m coal pillar

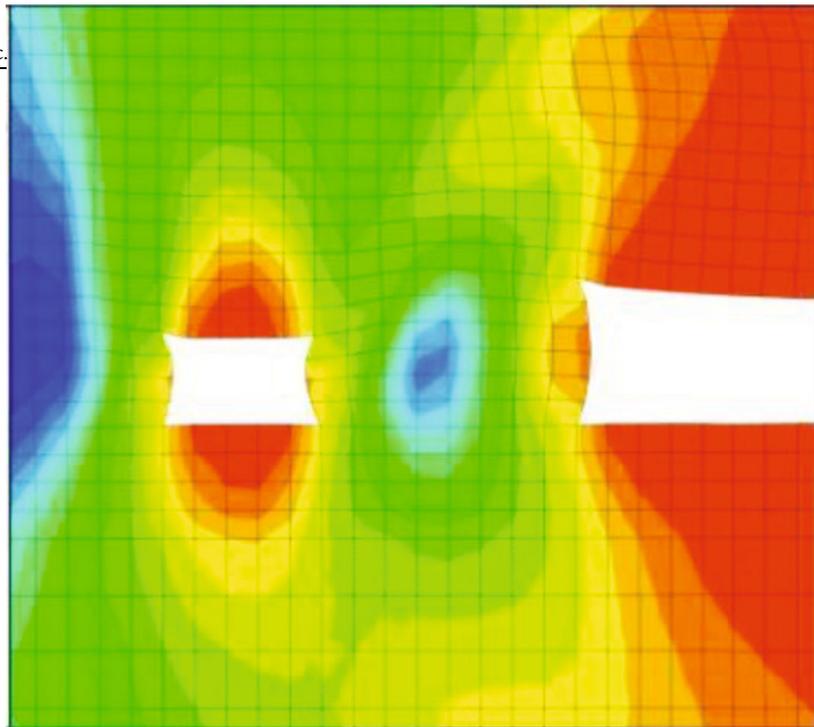
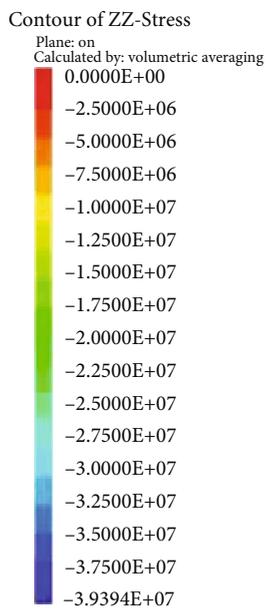
FIGURE 7: Continued.

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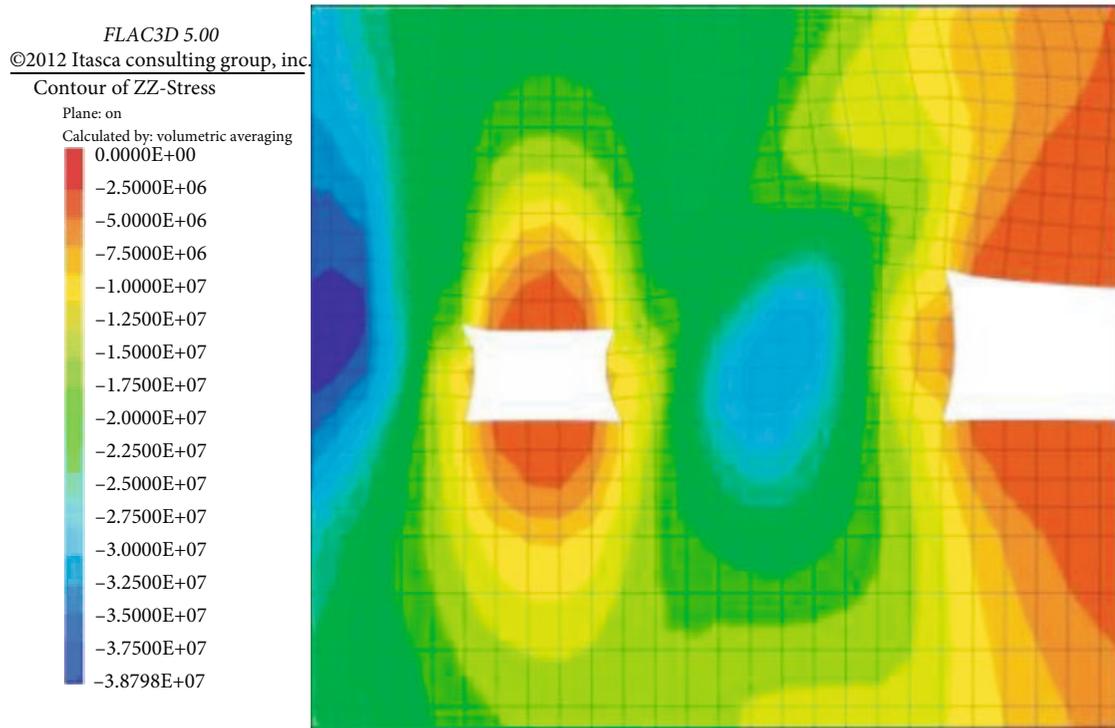
(c) 7 m coal pillar

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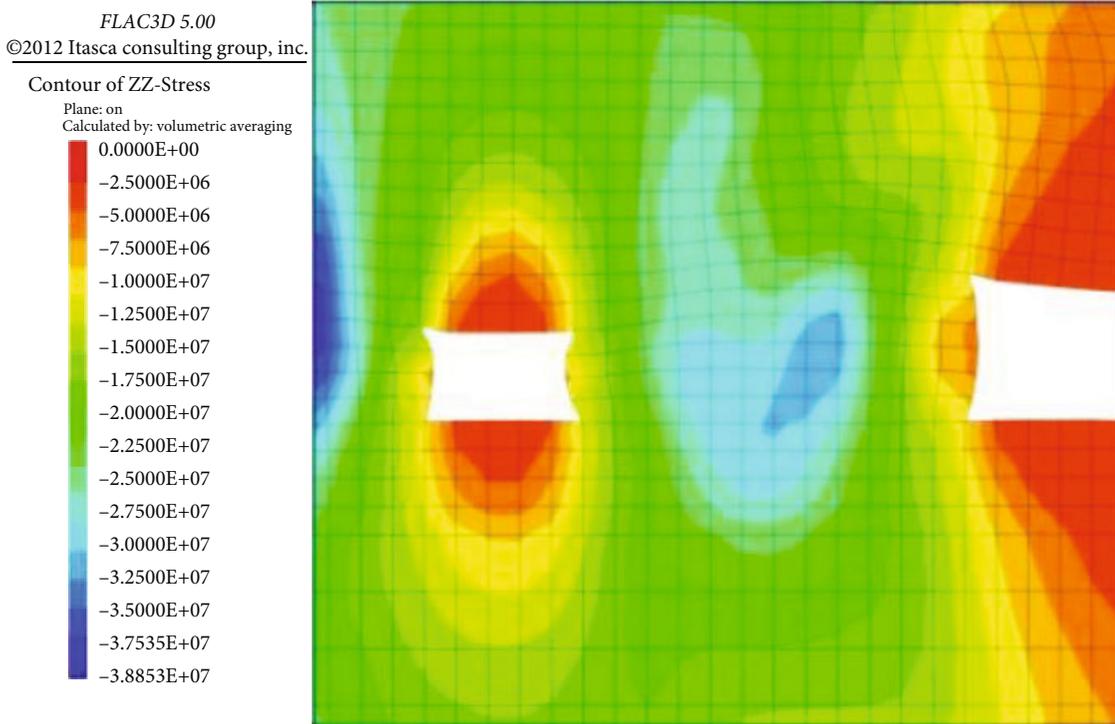


(d) 8 m coal pillar

FIGURE 7: Continued.



(e) 10 m coal pillar



(f) 12 m coal pillar

FIGURE 7: Cloud map of vertical stress distribution of different coal pillar widths under the effect of one mining.

gob-side entry retained section, and the vertical stress is butterfly-shaped distribution. The maximum influence height of the abutment pressure on the left side of the gob-side entry retained is 40 m. From Figure 6 and Table 2, it can be seen that the height of the fifth group of fine sand-

stone in the model is about 38 m, and the fine sandstone with a thickness of 10 m can still transfer stress after the mining of the No. 201 working face. Therefore, the key stratum at the bottom of the entry roof is the fine sandstone with a thickness of 10 m.

3.5. Determination of Reasonable Size of Small Coal Pillar.

Determining the reasonable width of small coal pillar is one of the key factors to ensure the success of gob-side entry retained. For Gaojiapu Coal Mine, in view of the problems of large mining depth, obvious impact tendency, and serious connection tension, the reasonable position and width of small coal pillar are determined. The entry arrangement in the low stress area can ensure that the coal pillar has a stable bearing area, which is an important condition to ensure the success of the gob-side entry test.

According to the limit equilibrium theory, the yield zone width R of near-horizontal coal seam [16, 17] is

$$R = \frac{mA}{2 \tan \phi_0} \ln \left[\frac{\lambda \gamma H + (C_0 / \tan \phi_0)}{(C_0 / \tan \phi_0) + (P_x / A)} \right]. \quad (1)$$

In these equations, m is coal pillar height (m); A is lateral pressure coefficient; ϕ_0 is internal friction angle ($^\circ$); C_0 is cohesion (MPa); λ is stress concentration coefficient; γ is average rock bulk density (kN/m^3); H is coal pillar depth (m); and P_x is the support resistance of P_x -support to the entry side (kN).

Substituting the data into equation (1), the yield width of coal pillar is 4.32 m.

The above analysis shows that the reasonable position of gob-side entry retained driving is to arrange the gob-side entry in the "internal stress field," so that it is in a low stress environment, which is conducive to maintaining the overall stability of the entry. Therefore, it is necessary to calculate the distribution range of the abutment pressure "internal stress field" on the coal side of the working face in theory. According to the theory of "internal stress field," the vertical abutment pressure distributed in the range of internal stress field around the gob is equal to the weight of the main roof rock beam before the first weighting of the working face [18]:

$$\frac{1}{2} S_0 \frac{k \gamma h S_0}{S_3} = \frac{LC \sum_1^n M_i \gamma}{2(L+C)}. \quad (2)$$

In these equations, S_0 is internal stress field width (m); M_i is the main roof rock beam thickness (m); C is first weighting interval of main roof (m); L is length of the working face (m); S_3 is bearing pressure which has obvious influence range (m); k is 2.2~2.8; γ is density (kN/m^3); and n is number of main roof rock beams.

Substituting the data into equation (2), $S_0 = 12.86$ m can be obtained.

According to the above equation (1), the yield width of coal pillar is 4.32 m, so the entry should be arranged within 4.32~12.86 m from the working face in theory. In order to ensure that the coal pillar is arranged in the low stress zone, so as to reduce the deformation of the coal pillar and ensure the internal integrity of the coal pillar, which is helpful for the bolt to effectively play a supporting role, so as to maintain the overall stability of the entry, six numerical simulation schemes with different coal pillar sizes are proposed: 5 m, 6 m, 7 m, 8 m, 10 m, and 12 m, and the numerical simu-

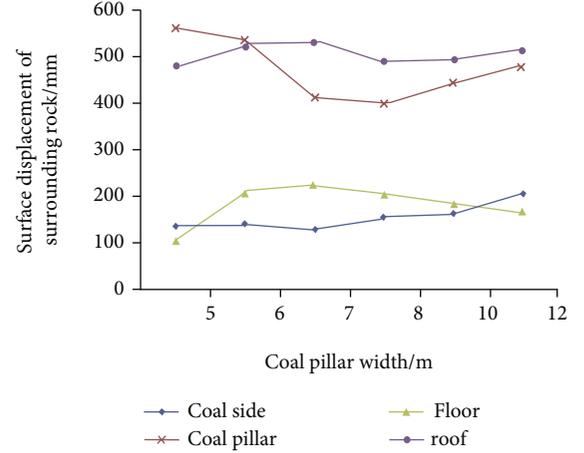


FIGURE 8: Deformation curve of surrounding rock of coal pillar with different widths.

lation comparative analysis of coal pillars with different sizes during a mining period is carried out; the numerical simulation results are shown in Figure 7.

From the numerical simulation results and Figure 8, during the working face mining, due to the roof rotation and subsidence of the roof, the stress of the surrounding rock of the entry is redistributed, and the peak value of the vertical stress in the coal pillar increases first and then decreases. When the coal pillar size is 5~7 m, it has good bearing capacity, and the side abutment pressure affected by mining is small, the stress concentration coefficient is small, and the entry is in a safe environment. When the coal pillar size is 7~12 m, the peak stress begins to decrease, and the coal pillar still maintains good bearing capacity. However, the peak stress of the side coal seam is higher, and the stress concentration coefficient is large, which is not conducive to the stability and maintenance of the entry. After comprehensive consideration, under the specific geological conditions of Gaojiapu Coal Mine, the reasonable coal pillar size of the test section of small coal pillar gob-side entry retained should be 7 m.

3.6. Analysis of the Mechanical State of the Coal Pillar Gob-Side Entry Retained. The second rock layer (fine-grained sandstone) above the coal seam is identified from Subsection 3.4 as the lower key layer. According to the Academician Qian Minggao's "critical stratum theory," after the working face was mined, a small structure consisting of key blocks A, B, and C was formed together, as shown in Figure 9.

A mechanical analysis of the coal pillars of gob-side entry retained was performed. With reference to literature [19], we analyze the coal pillars to obtain the following equation:

$$u = (m + n - kn) \frac{(l_1 + l_2 + l_3)}{l}. \quad (3)$$

In these equations, u is the amount of sinking of key block B (m); m is the thickness of the coal seam (m); n is the thickness of the immediate roof (m); k is the immediate

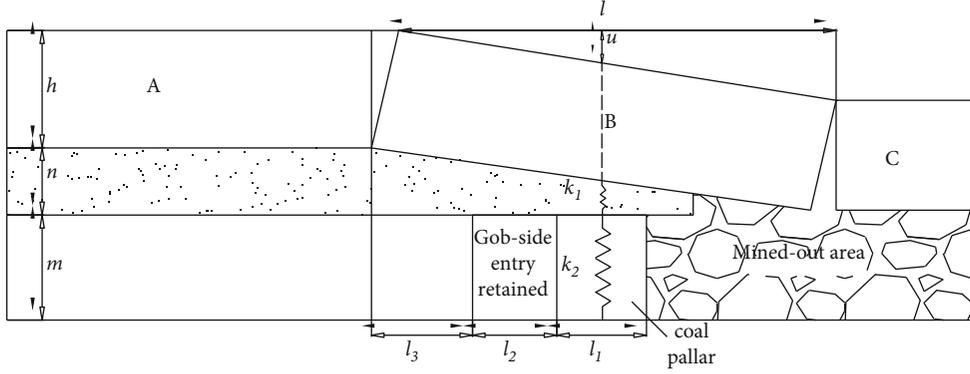


FIGURE 9: Mechanical model diagram of coal pillar.

roof breaking expansion factor; l_1 is the width of the coal pillar (m); l_2 is the width of the gob-side entry retained (m); l_3 is the length of the key block B fracture location from the roadway (m); and l is the length of the key block B (m), where the key block B length l is equal to the distance of periodic roof weightings.

The l_3 is obtained from the following equation:

$$l_3 = \frac{MA}{2 \tan \varphi} \ln \frac{k\gamma H + (C_0/2 \tan \varphi)}{(C_0/\tan \varphi) + (P_x/A)}. \quad (4)$$

In these equations, A is the lateral pressure coefficient; φ is the angle of internal friction; γ is the gravitational density of the rock formation (kN/m^3); H is the mining depth (m); C_0 is the cohesion of the coal seam at the junction with the roof and floor; P_x is the preload force of the anchor against

the roof (kN); and M is the mining height (m). In this manuscript, the coal thickness is equal to the mining height ($M = m$).

The pressure p acting on the coal column is given by the following equation:

$$p = \frac{k_1 k_2}{k_1 + k_2} u = \frac{E_1 E_2 l_1}{E_1 m + E_2 n} u. \quad (5)$$

In these equations, p is the pressure acting on the coal pillar (MPa); k_1 and k_2 are the stiffness factor in the vertical direction of the direct roof and coal pillar; and E_1 and E_2 are the modulus of elasticity of the direct top and coal pillar, respectively (MPa). Substituting equation (4) into equation (3) yields u ; then, substitute u in equation (5) to obtain p as follows:

$$p = \frac{(m + n - kn)(E_1 E_2 l_1)(l_1 + l_2 + (mA/2 \tan \varphi) \ln ((k\gamma H + (C_0/2 \tan \varphi))/((C_0/\tan \varphi) + (P_x/A))))}{l(E_1 m + E_2 n)}. \quad (6)$$

In this manuscript, the pressure p before and after the top break is related to the location of the key block B break, l_3 , and the amount of sinking of key block B- u . When the tops are manually forced off, the l_3 decreases and the u follows, resulting in fewer p .

3.7. Determination of Roof Breaking Method and Parameters

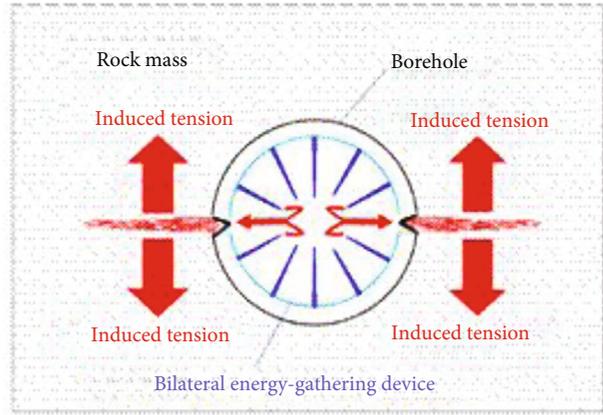
3.7.1. Determination of Roof Breaking Method. In the past 20 years, in order to solve the occurrence of rock burst in deep mining, many researchers have developed a series of pressure relief methods, among which hydraulic fracturing and blasting pressure relief are the most common. The hydraulic fracturing method was originally used for the development and utilization of natural gas and petroleum gas and then was used for the pressure relief method in deep mining. However, the hydraulic fracturing itself is greatly affected by the rock strata, and the rock hardness

and fracture development degree affect the effect of hydraulic fracturing to a large extent. Therefore, this paper adopts the blasting pressure relief method of D-type [20] polymer tube, which is more commonly used in the pressure relief method.

As shown in Figure 10, the conventional explosive is loaded into the D-shaped tube in the predrilling. When the charge explosive explodes in the device, the explosion energy is collected to form an energy flow. The energy flow is released along the D-tube structure, and the release direction is the planned crack direction. The initial crack is first caused by explosion shock wave and stress wave. Because the rock is a brittle material, the subsequent local failure is easily caused by the stress concentration in the damaged area. When the borehole is arranged in a straight line at a certain interval and the charge in the energy gathering pipe is reasonably designed, the rock roof can be cut along the direction of energy accumulation, while the rock in other



(a) D-shaped energy collecting groove



(b) The mechanical effect of energy accumulation

FIGURE 10: Mechanical model of double-sided energy-accumulating tension blasting.

directions is intact due to the protection of the energy gathering pipe.

3.7.2. *Determination of Blasting Parameters.* The factors affecting the presplitting blasting effect involve the charge mass, charge structure, sealing length, and borehole spacing.

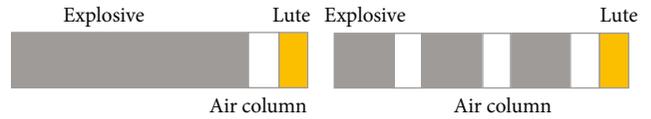


FIGURE 11: Axial decoupling charge method.

- (1) Charge mass. The charge mass determines the cohesive blasting energy produced by single hole charge explosion, which affects the cutting effect. Small charge mass and insufficient crack propagation cannot effectively cut off the key stratum. When the charge mass is large, the energy generated by blasting will destroy the roof and other supporting materials. Especially in this technical research, the purpose is to cut off the lower key stratum and protect the integrity of the immediate roof as far as possible. Therefore, the charge should be based on the strength of rock mass
- (2) Charge structure. There are many charging methods in the hole in the roof presplitting blasting, but in order to facilitate the charging or meet the needs of the blasting engineering in the actual blasting engineering, no matter the radial and axial direction, the noncoupling charging method is adopted, which is mainly divided into two ways: one section of the air column and several sections of the air column, as shown in Figure 11
- (3) Sealing length. When doing column explosive blasting, the antiexplosion ability of rock mass increases with the increase of hole depth, and the blasting ability of explosive is related to the length of sealing hole and the antiexplosion strength of rock mass
- (4) Borehole spacing. When the roof is presplitting blasting by deep hole blasting, the rock is around the explosive explosion, and there is no free surface. The stress wave generated by blasting can be seen as

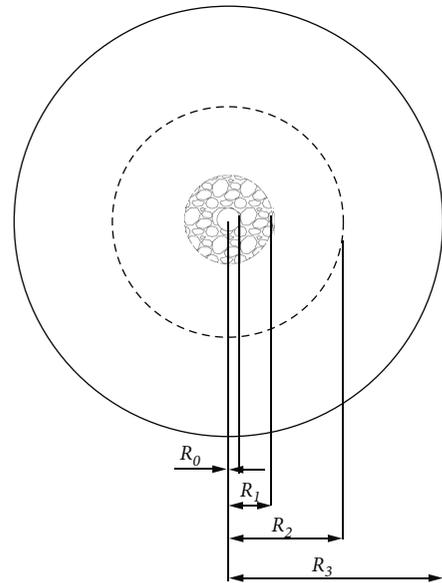


FIGURE 12: Internal effect of blasting.

uniform diffusion around the rock mass. After explosion, the borehole radius of R_0 , the crushing zone of R_1 , the rupture zone of R_2 , and the vibration zone of R_3 will be generated with the borehole as the center in the rock mass, as shown in Figure 12

After the explosive is detonated in the borehole, the resulting stress wave causes the rock mass to be crushed

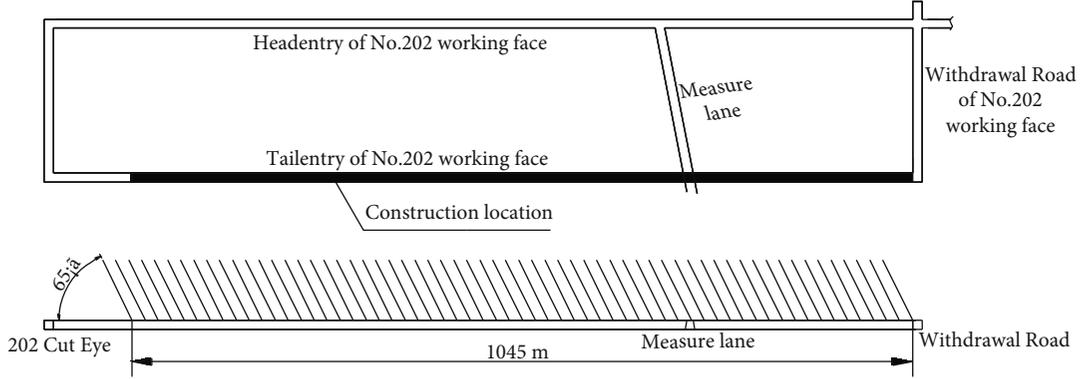


FIGURE 13: Drilling plan layout.

and destroyed, thus forming the crushed zone. The radius calculation formula of the crushed zone can be used in the following equation:

$$R_1 = \left(\frac{\rho_m C_p^2}{5\sigma_c} \right)^{1/2} R_k. \quad (7)$$

In these equations, R_1 is radius of crushing zone; ρ_m is rock initial density (kg/m^3); C_p is wave velocity of rock mass (m/s); R_k is cavity radius limit value is calculated by equation (4); and σ_c is rock uniaxial compressive strength (MPa).

$$R_k = \left(\frac{P_1}{\sigma_0} \right)^{1/4} r_b. \quad (8)$$

In these equations, r_b is drilling radius (mm); P_1 is explosive explosion pressure (MPa); it can be calculated by equation (9); σ_0 is rock strength under multidirectional stress (MPa); it can be calculated by equation (10).

$$P_1 = \frac{1}{8} \rho_0 D^2, \quad (9)$$

$$\sigma_0 = \sigma_c \left(\frac{\rho_m C_p^2}{\sigma_c} \right)^{1/4}. \quad (10)$$

Since the explosion of explosives, the force time of the shock wave on the coal is very short, and its effect is rapidly attenuated, and the radius of the crushing zone is relatively small. Therefore, the radius calculation of the fracture zone is crucial to the study of the borehole layout parameters. At present, the radius of the fracture zone is mainly determined by the stress wave, so the radius of the fracture zone is solved by equation (11) of the stress wave:

$$R_2 = \left[\frac{bP_2}{S_T} \right]^{1/\alpha} r_b. \quad (11)$$

In these equations, R_2 is rupture radius (mm); b is the ratio of radial stress to tangential stress which can be calculated by equation (12); α is the attenuation coefficient of

α -stress wave which can be calculated by equation (13); P_2 is the peak shock wave stress under the uncoupled charge factor, which can be calculated by equation (14); r_b is drilling radius (mm); and S_T is rock uniaxial tensile strength (MPa).

$$b = \frac{\mu}{1 - \mu}. \quad (12)$$

In these equations, μ is the Poisson ratio.

$$\alpha = 2 - b, \quad (13)$$

$$P_2 = \frac{1}{8} \rho_0 D^2 \left(\frac{r_c}{r_b} \right)^6 n. \quad (14)$$

In these equations, ρ_0 is explosive density (kg/m^3); D is explosion velocity (m/s); r_c is charge radius (mm); and n is stress increase multiple, generally 8~11.

The rock mechanics data of Gaojiapu are substituted into the above equation and coupled with the explosive charge to calculate and solve radius of crushing zone R_1 is 0.7 m, and the radius of fracture zone R_2 is 5.3 m. The lower key stratum thickness is 10 m. Through theoretical analysis and field investigation, it is known that when the coal seam conditions are certain, the smaller the borehole spacing is, the more conducive to the formation of precracking surface of fracture propagation, and the better the unloading effect is. However, with the decrease of borehole spacing, the amount of field engineering increases and the construction cost increases. Therefore, according to the specific geological conditions and construction conditions of Gaojiapu No. 202 working face, the borehole spacing is determined to be 2 m.

3.7.3. Layout of Boreholes for Roof Breaking Presplitting Blasting. The arrangement of blasting boreholes is based on the thickness of the main roof, the degree of joint fracture development, the first weighting and periodic weighting interval of the working face, the explosive performance of explosives, and other factors. The drilling arrangement of presplitting blasting in advance broken roof is divided into a unidirectional drilling method and bidirectional drilling method. The one-way drilling method is to drill a deep hole at the other end of the tailentry or headentry to ensure the overall stability of the roadway. The two-way drilling

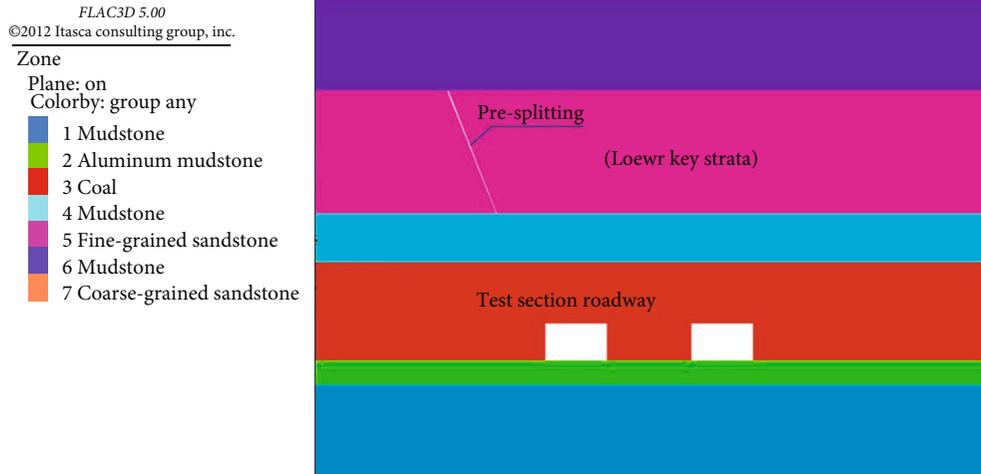


FIGURE 14: Simulated diagram of roof-breaking presplitting blasting.

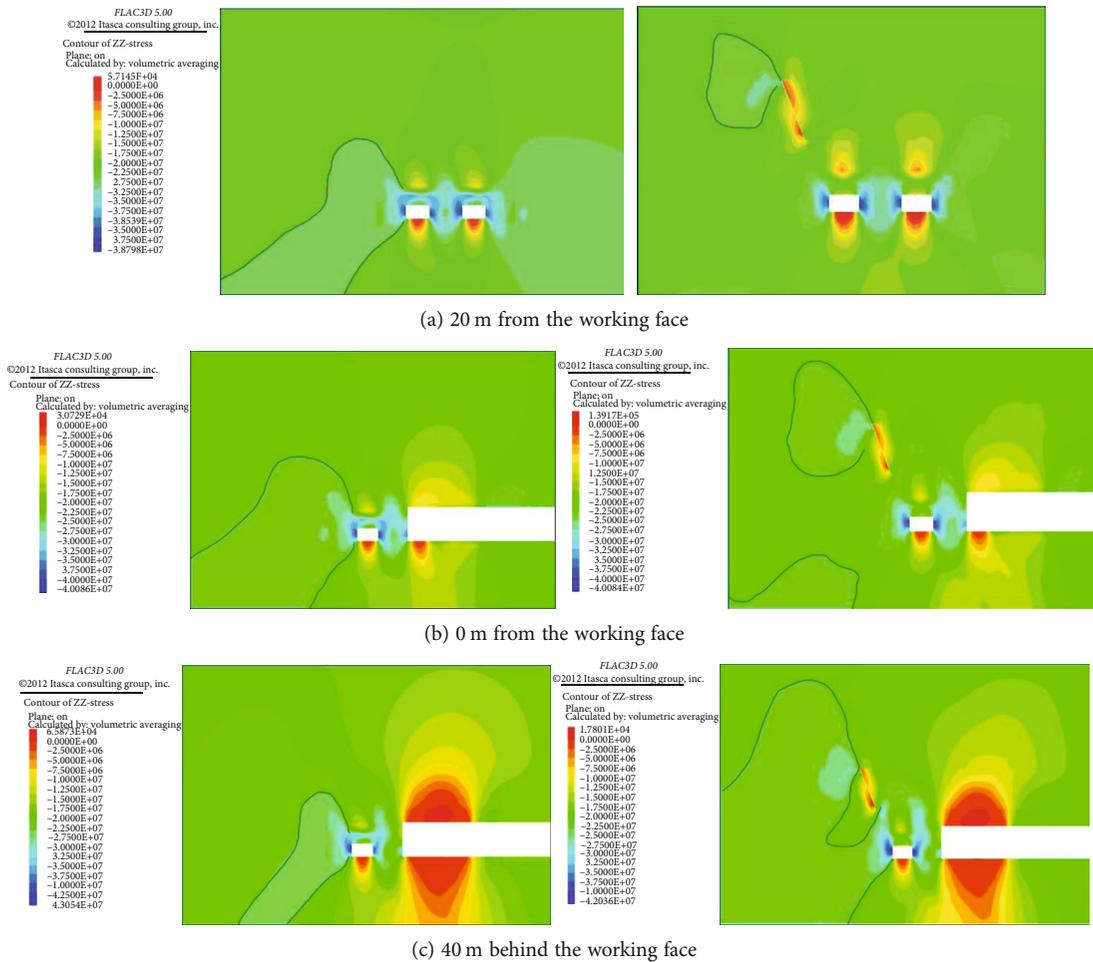


FIGURE 15: Vertical stress distribution nephogram.

method is to simultaneously drill deep and oblique holes into the coal body in the tailentry or headentry [21, 22].

Through the above theoretical calculation and field investigation and analysis, in view of the characteristics of high ground stress in the No. 202 working face of Gaojiapu

Coal Mine, the layout mode of unidirectional roof breaking presplitting blasting boreholes with 75 mm borehole diameter, 60 mm charge diameter, 1.25 uncoupling coefficient, 65° borehole inclination angle, 20 m borehole depth, 2 m borehole spacing, and 6 m sealing length is proposed. The

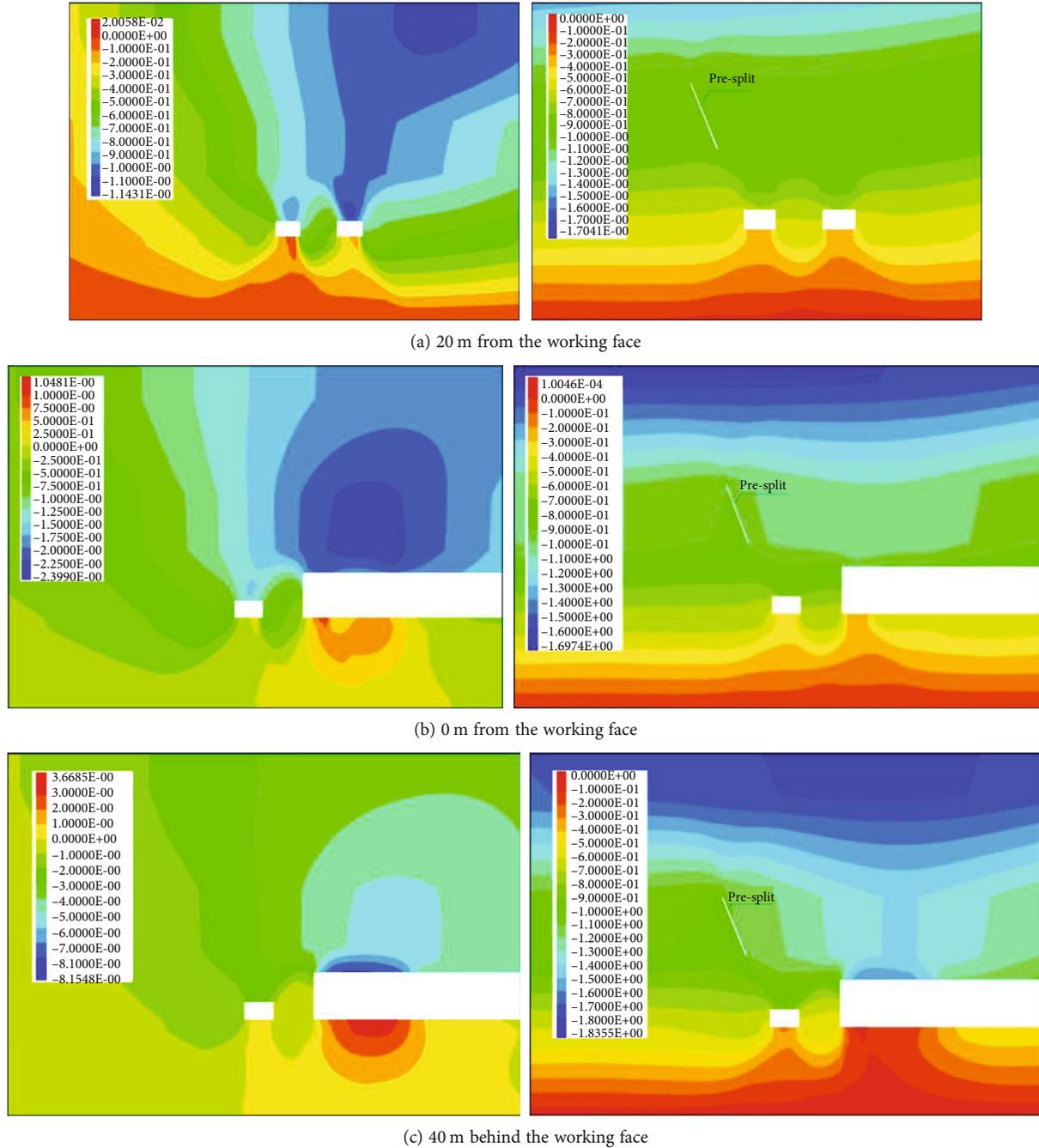


FIGURE 16: Horizontal displacement nephogram of entry.

presplitting blasting time of roof breaking in the No. 202 working face is set before the mining after the breakthrough of tailentry, and the scheme layout diagram is shown in Figure 13.

4. Numerical Simulation Analysis of Presplit Blasting of Broken Roof

In order to facilitate the overall coherence analysis and simulation, this chapter used the numerical simulation of mechanical model mentioned above, to continue using FLAC^{3D} finite difference software to continue the simula-

tion, and analyzed the roof cutting and pressure releasing and not roof cutting and pressure releasing under the condition of two kinds of working condition of small coal pillar along the gob-side entry retained for the surrounding rock stress variation of the arrangement. The deformation of surrounding rock along the test section of the gob-side entry retained is monitored under two working conditions. Through the method of comparative analysis, it is verified that the parameters designed by the presplitting blasting of interrupted roof in the stopping process of the No. 202 working face can achieve good results in maintaining the gob-side entry retained.

According to the above theoretical calculation, it can be concluded that the lower key stratum is the fine-grained sandstone directly above the immediate roof of the coal seam, with a thickness of 10 m. In this section, fine-grained sandstone 10 m above the entry in the test section is simulated to simulate the unloading effect of top-breaking presplitting blasting, and then, the deformation amount of the remaining small coal pillar and the transfer law of surrounding rock stress are analyzed through numerical simulation calculation. The blasting area is shown in Figure 14.

4.1. Comparative Analysis of Surrounding Rock Vertical Stress. Through numerical simulation under two working conditions before and after the roof cutting of 202 tailentry, we compare and analyze the law of stress change of small coal pillars of gob-side entry retained when excavating 202 working face. We intercepted three sections (20 m, 0 m, and 40 m from the working face) to reflect the comparative changes of stresses under the two working conditions from near to far from the working face. Final comprehensive evaluation of the vertical stress distribution law before and after interruption of top decompression by top precracking blasting in the mining process of thick coal seams in deep wells was performed. The vertical stress distribution cloud diagram is shown in Figure 15.

Through comparative analysis of the stress distribution law of the three sections, it is found that without the roof off the top handling pressure state, The side abutment pressure concentration zone of the roadway was always in the position close to the roadway during the retrieval process of the 202 working face. The maximum stress value reached 43 MPa, and the stress concentration factor reached 1.9. During this period, the roadway is always under the influence of high bearing pressure, which can easily lead to coal wall flakes, serious deformation of the roadway, and even instability. And the maximum range of side abutment pressure is about 40 m above the left side of the entry, indicating that the side abutment pressure is transmitted by 10 m fine-grained sandstone at this time. After the roof cutting and pressure relief, the precracking surface setup during the simulation calculation cuts off the stress transfer path between the lower key stratum. And it can be seen from Figures 15(a) and 15(b) that there is an obvious unloading pressure zone in a certain range above the roadway, and the lateral support pressure side abutment pressure range above the roadway is located above the cut, and the height is about 55 m. And the abutment pressure increases in the left side of the fracture line after the roof cutting, forming a stress concentration area. This indicates that the lower key layer is not transferring pressure and that pressure is being transferred farther up the line from the higher rock stratum. As can be seen from the previous section, the abutment pressure is transferred in the mudstone layer of 17 m above the fine-grained sandstone at this time. In this state, the entry itself is in the range of in situ stress zone, which is conducive to the stability of the entry itself and the performance of bolt support.

4.2. Comparative Analysis of Entry Roof Subsidence. Through the actual investigation of Gaojiapu Coal Mine, it is found

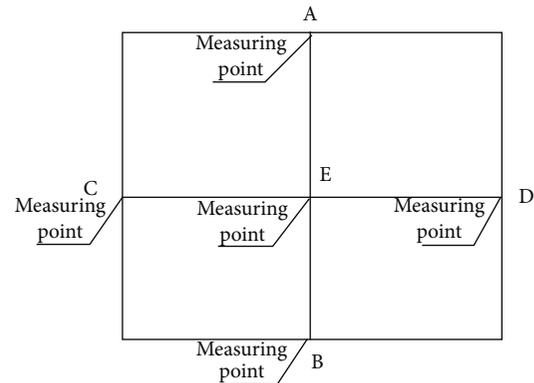


FIGURE 17: Schematic map of measuring point arrangement.

that the bolt (cable) support of mining entry in one panel area is not ideal, and the roof subsidence is serious, which is manifested as roof separation subsidence and support structure bending fracture. In order to ensure the effect of presplit blasting with broken roof and provide feasible suggestions for later support, the roof subsidence of mining entry under two working conditions of broken roof unloading and unbroken roof unloading was compared and analyzed.

According to the overall analysis of Figure 16, it can be seen that the maximum subsidence of entry is located in the middle of the roof, so the support design of the middle of the roof should be strengthened when considering entry support. After the implementation of top-breaking presplitting blasting on the working face, the amount of roof subsidence is significantly reduced in the range affected by advanced mining, and the final settlement value is reduced from 2000 mm to 1100 mm, 900 mm less. It can be seen that under the condition of roof breaking and presplitting blasting without considering the support, the roof subsidence of the test section becomes smaller as a whole because the concentrated stress zone in the coal body of the entry is obviously weakened and unloading zone appears in the entry roof.

In conclusion, the stress and vertical displacement of surrounding rock of the 202 tailentry have changed significantly after the roof presplitting blasting. It can be seen that the stress concentration area moves to a higher and farther place obviously after the presplitting blasting. The influence of vertical stress on entry is obviously reduced in the lateral coal body and the unloading zone appears at the top of entry. The roof subsidence of the entry in the test section is significantly reduced, the overall deformation of the entry in the test section is reduced, and the bearing capacity is increased, which is more conducive to the stability of the entry in the test section.

5. Actual Entry Ore Pressure Monitoring

5.1. Coal and Rock Surface Displacement Monitoring

5.1.1. Station Layout. The No. 1 station was selected as the measuring point to monitor the surrounding rock deformation

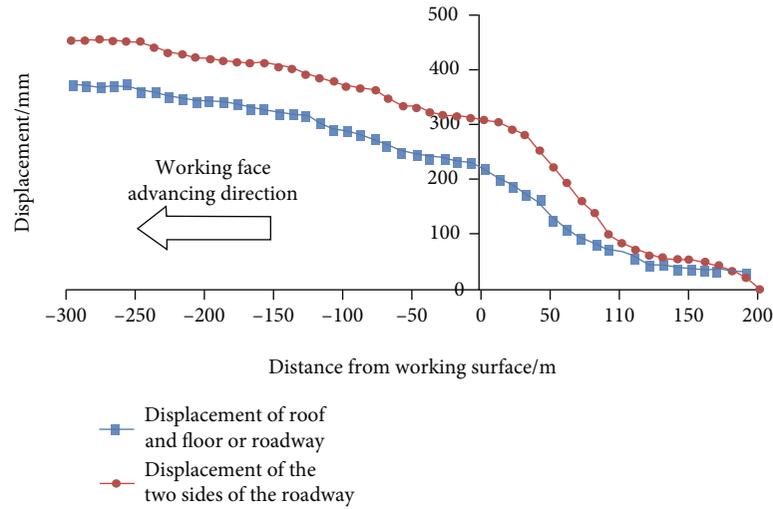


FIGURE 18: Approaching quantity of roof and floor and two sides of entry during the mining period of the No. 201 working face.



FIGURE 19: Effect picture of entry staying along goaf before and after roof failure pressure relief.

of the entry, mainly monitoring the surrounding rock deformation before and after the mining of the No. 201 working face. The monitoring data were sorted out, and the deformation curves of the roof and floor and two sides of the entry were drawn.

5.1.2. Layout and Monitoring Requirements of Artificial Observation Points for Surrounding Rock Deformation. The layout of measuring points of the monitoring section of entry deformation in the test section is shown in Figure 17. The stations are arranged by crosspoint method.

5.2. Monitoring of Entry Surrounding Rock Deformation during Mining of the No. 201 Working Face. During the stopping of working face 201, the deformation of surrounding rock of entry along the test section of goaf retaining entry was monitored. The change rule of entry surrounding rock surface displacement at the measuring station with the progress of stopping the working face is shown in Figure 18.

It can be seen from Figure 18 that when the station is 110m away from the working face, the moving amount of the roof, floor, and two sides of the entry begins to increase, indicating that the influence range of advanced abutment pressure of working face 201 is 110 m in front of the working face. In the range of 50~80 m from the working face to the

station, the moving rate of the roof, floor, and two sides is accelerated, indicating that the leading abutment pressure has a severe influence on the surrounding rock of entry in this range. After the working face pushes over the measuring point, the deformation of surrounding rock of entry continues to increase under the influence of the movement of the basic roof rock beam. After 250m stopping of the working face, the deformation of entry tends to be stable, indicating that the movement of the basic roof rock beam basically ends at this time and the entry will no longer have obvious deformation. The actual engineering effect is shown in Figure 19.

By comprehensive analysis of the above observation results, it can be seen that when small coal pillars are used in the goaf retaining test in No. 2 panel area of Gaojiapu Coal Mine, the supporting system can always play an active supporting effect and maintain the stability of entry surrounding rock under the action of a mining disturbance. From the analysis of surrounding rock deformation, the surrounding rock deformation of entry is always in a controllable range, and the overall deformation of entry is relatively small, which can meet the needs of safety production and continue to serve the mining of the No. 202 working face. It can be seen that the entry retention test was successful, but it is still necessary to continue monitoring the entry in

the test section during the stopping of the No. 202 working face to test the overall effect of entry retention in the test section.

6. Conclusion

In this paper, a method of roof cutting and pressure releasing of small coal pillar is proposed based on the lower key layer of coal seam roof. According to the characteristics of high in situ stress in Gaojiapu Coal Mine, the roof conditions of coal seam are analyzed through theoretical calculation and the lower key layer above the coal seam is determined through field investigation and actual construction conditions. The arrangement mode and blasting parameters of presplit blasting borehole for one-way roof breakage were designed to cut off the lower key stratum, and FLAC^{3D} software was used to simulate the unloading effect of the lower key stratum after fracture.

The results show that the presplitting surface formed by blasting can effectively cut off the transmission of raw rock stress between the lower key stratum, and the stress concentration area in the side coal body of entry is obviously transmitted to the higher and farther inside the solid coal, and part of the unloading area appears above. The gob-side retaining entry can adopt the same support parameters as the transport entry, so that superstrong support is no longer needed. Besides, the retention of small coal pillars reduces the consumption of the entry side retaining rod system, reduces the cost of coal mining, and improves the safety of coal mining.

Data Availability

The data used to support the findings of this study are included within the article.

Conflicts of Interest

We declare that we have no conflict of interest.

Acknowledgments

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