Study on the mechanism of crack formation and the behavior of crack propagation of directional tension blasting in deep and high stress coal mine

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Abstract. Roadway safety is related to the high productivity and efficiency of coal mining, roof cutting and pressure releasing technology is one of the effective measures for roadway stability control. Under deep mining conditions, the stress condition around the blasthole is complex, and the expansion of directional crack will be affected by high ground stress. In this paper, Based on the engineering background of the roof Cutting and Pressure Relief Technology of Non-pillar Automatic Roadway Formation in guotun coal mine, the crack propagation mechanism and propagation evolution of deep rock under directional tension blasting were investigated. Based on the stress model of blasthole under the deep condition, the mechanical conditions of crack propagation under the action of directional detonation wave and high surrounding rock binding force are obtained theoretically; The mechanical model of directional blasting under the condition of stress constraint is established by using LS-DYNA numerical calculation software, the evolution characteristics of effective stress and the law of crack growth are analyzed; Finally, through the drilling peep and other technical means, the field measurement of the presplitting effect is carried out, which verifies the previous research results and puts forward the appropriate roof presplitting scheme. The results show that when the deep roof strata of Guotun coal mine adopt four hole continuous blasting and per blasthole charge of 2800g, the roof strata can form a continuous seam, and the Roof presplitting effect is the best.

Keywords: roof cutting, high ground stress, numerical calculation, directional blasting

1. Introduction
Coal is one of the most important basic energy sources in the world, accounting for about 30% of the total global energy consumption[1]. With the massive exploitation of coal resources and the massive consumption of shallow resources, coal mining has gradually progressed to the deeper[2]. Most coal mines in China adopt the method of leaving coal pillars to maintain the roadway in the mining area. With the increase of mining depth, most mining areas adopt the method of increasing the width of coal pillars to control the deformation of deep surrounding rocks. High stress concentration will be formed, which can easily cause mine safety accidents.

The 110 mining method is an innovative non-pillar mining method. By cutting the roof on the side of the goaf, cutting off part of the roof pressure transmission of the mine, and filling the goaf with the
collapse and expansion characteristics of rock collapse[3, 5]. This new type of mining method not only improves the resource recovery rate, eases the tension of excavation and replacement, but also solves the problem of gas accumulation in the upper corner and eliminates disasters such as rock burst, coal and gas outburst [6,7]. The key to the successful implementation of the 110 construction method is the roof cutting effect. The roof cutting function should promote the separation of the roof of the roadway from the roof of the goaf, and promote the collapse of the rock. In view of this, the directional presplit cumulative blasting technology is proposed [8,9]. As shown in Figure 1, this technology is mainly achieved by installing an energy-accumulated tube device in the blast hole. The energy-accumulated tube is equipped with an energy-accumulated hole. During blasting, high-temperature, high-pressure, high-speed gas is produced, and a powerful gas is formed along the direction of the energy-accumulated tube, drive crack propagation [10,11]. The blasting load produces a uniform pressure in the non-focusing direction, and generates a concentrated tension in the energy-accumulated direction, so as to achieve directional cracks. Relevant scholars have done a lot of research and application on the directional presplit cumulative blasting technology. For example: Guo Pengfei obtained the principle of the directional presplit cumulative blasting through theoretical analysis of fracture mechanics, and obtained the best blasting parameters of the roof of Hecaogou coal mine through numerical simulation and field test[12]; Gao Yubing analyzed the initial stress field of the blasthole surrounding rock under the action of in-situ stress, and divided the process of energy gathering tension blasting on the rock into four stages, namely the stage of energy gathering flow eroding the rock mass and the effect of detonation shock wave. At the stage, stress wave action stage and detonation gas action stage, the mechanical model of directional presplit cumulative blasting was established[13]; Xue Haojie studied the field application effect of directional presplit cumulative blasting technology in Hongqinghe Coal Mine[14].

Relevant scholars also studied blasting under different confining pressure conditions. Bai Yu simulated the blasting under different in-situ stress conditions by loading stress waves. The results showed that the initial in-situ stress field affected the blasting cracking effect. When the lateral pressure coefficients are the same, the greater the burial depth, the smaller the crack propagation length, the smaller the number of cracks, the smaller the crack area, and the worse the blasting effect[15]. Yang Zhonghao used numerical simulation software LS-Dyna to simulate the blasting under different in-situ stress conditions, and concluded that the initial confining pressure had a certain restrictive effect on the development of blasting cracks[16].

However, related scholars' research on directional presplit cumulative blasting technology is mostly concentrated in low-stress and shallow buried coal seams. Whether the directional presplit cumulative blasting technology is applicable under high stress conditions has yet to be verified. Therefore, we need to study the directional presplit cumulative blasting technology under high stress conditions.

2. Engineering background
Guotun Coal Mine is located in Yuncheng County, Heze City, Shandong Province, China. The coal mine has an annual output value of 2.4 million tons, a coal field area of 69.3293 km², 4306 mining face width of 208 meters and a length of 788 meters, with an average inclination angle of less than 5°, as shown in Figure 2. The working face is mainly mined with 3 lower coal seams. The average thickness of the coal seam is 2.7 m and the average depth of the coal seam is 845 m. The overlying rock layer of 3 coal seam is mainly composed of sandstone, and the roof of 3 coal seam is mainly composed of fine sandstone. The total thickness of the roof aquifer is about 24 m. Figure 3 shows the vertical geological column at the...
study site. The rock strata above the coal seam are composed of fine sandstone (10.68 m thick), siltstone (14.27 m thick), mudstone (1.85 m thick), and fine sandstone (13 m thick) in ascending order. Below the coal seam, the floor is composed of siltstone and mudstone, with average thicknesses of 2.5 m and 7.73 m, respectively.

Figure 2. Mining panel and roadway layout.

Figure 3. Stratigraphic column and geological description.

3. The Principle of directional presplit cumulative blasting and Analysis of Force on Blast Hole

3.1. The Principle of directional presplit cumulative blasting
The key problem of the 110 mining method is the stability control of the roof. The roof presplit technology can change the compact structure of the original roof by cutting part of the roof, so that the roof breaks from the long-arm suspension structure to the broken wall breaking mode. The traditional blasting method will cause greater damage to the roof structure, which have an adverse effect on the stability of the roadway. In view of this, the directional presplit cumulative blasting technology is proposed. In the traditional blasting method, the blasting product and blasting energy spread to all sides, the pressure effect is relatively uniform, and a large part of the energy is dissipated on the broken rock; Taking advantage of the rock's weak tensile capacity, the directional presplit cumulative blasting technology was developed. The blasting energy is shown in Figure 4. During the implementation of this
technology, the blasting energy flows in the artificially set direction, forming a strong gas wedge force acting on the artificially set direction. When the tensile force in the crack is greater than its compressive strength, the crack is generated, forming a slit line.

**Figure 4.** Energy-gathered effects of bilateral cumulative tensile explosion technology. (a) The energy is not controlled in conditional blasting. (b) The energy is centered in the desired directions in bilateral cumulative tensile explosion technology.

The energy of driectional presplit cumulative blasting is mainly controlled by the energy-accumulated tube, as shown in Figure 5; The material of the energy-accumulated tube is a special PVC material. Two rows of energy-accumulated holes are arranged on the surface of the device, which are 180 directions from each other, and the distance is 8mm. The inner diameter of the energy-accumulated tube is 42mm, and the length is 1.5m. First, put the explosive into the energy-accumulated tube, then push the explosive together with the energy-accumulated tube into the blast hole, and finally block it tightly with gun mud. The biggest difference between the driectional presplit cumulative blasting technology and the traditional split blasting is that the energy-accumulated tube can make the blasting energy concentrate in the direction of gathering energy. Directional shaped energy blasting technology can reduce the large area of blasting damage to the surrounding rock of the roadway, and can also cut off the connection between the roof of the roadway and the roof of the goaf, making it easier for the roof of the goaf to slide along the rupture surface and form gravel to fill the goaf.

**Figure 5.** Energy-accumulated tube used for roof splitting in directional bilateral cumulative tensile explosion technology.

### 3.2. Analysis of Force on Blast Hole

In the process of a nonpillar mining method with entry automatically retained, the length of the blast hole is much larger than its diameter. Therefore, the stress of the blast hole can be simplified to analyze the plane strain problem. The roof of the test area is near horizontal. In order to reduce the impact of the roof collapse of the goaf on the roof of the roadway, the axial direction of the slit hole has a certain angle β to the vertical direction. The initial stress of the drilled rock mass at the cutting angle is shown in Figure 6(b). Among them, q0 is the vertical stress ρgH, λq0 is equal to the horizontal stress, and q′is the
component of the vertical and horizontal ground stress in the hole cross section direction after considering the cutting angle, which can be approximately expressed as \( q_0 \sin \beta + \lambda q_0 \cos \beta \). When the slit hole direction is perpendicular to the roof of the roadway, \( q' = \lambda q_0 \); when the slit hole direction is parallel to the roof, \( q' = q_0 \). Through the above analysis, we assume that the circular borehole with radius \( r \) is subjected to biaxial compressive stress on the infinite plane, as shown in figure 6(a), and the Kirsch equation is used to determine the stress distribution around the hole:

\[
\begin{align*}
\sigma_\theta &= \frac{\sin \beta + \lambda \cos \beta + \lambda}{2} \left(1 - \frac{a^2}{r^2}\right)q_0 - \frac{\sin \beta + \lambda \cos \beta - \lambda}{2} \left(1 - 4 \frac{a^2}{r^2} + 3 \frac{a^4}{r^4}\right)q_0 \cos 2\theta \\
\sigma_{\theta0} &= \frac{\sin \beta + \lambda \cos \beta + \lambda}{2} \left(1 + \frac{a^2}{r^2}\right)q_0 + \frac{\sin \beta + \lambda \cos \beta - \lambda}{2} \left(1 + 3 \frac{a^4}{r^4}\right)q_0 \cos 2\theta \\
\tau_{\theta\phi} &= \frac{\sin \beta + \lambda \cos \beta - \lambda}{2} \left(1 + 2 \frac{a^2}{r^2} - 3 \frac{a^4}{r^4}\right)q_0 \sin 2\theta
\end{align*}
\]

Where: \( \lambda \) is the lateral pressure coefficient; \( \rho \) is the average density of the overlying strata on the roof; \( H \) is the burial depth of the mining face.

According to the actual situation of Guotun Coal Mine, the 4306 mining face has an average buried depth of 865 m, a roof cutting angle of 15°, and a hole depth of 8.5 m. The vertical stress is 22.4 Mpa, and the lateral pressure coefficient is taken as 1.46. After calculation, the horizontal stress is 32.7 Mpa.

![Figure 6](image)

Figure 6. Analysis of initial stress around blasthole. (a) Biaxial compressive stress of the circular drilling hole in the infinite plane. (b) Angle between blasthole and vertical direction.

4. Directional presplit cumulative blasting crack propagation

The energy-accumulated hole will cause a blasting stress to form an energy-accumulated jet in the specified direction. At this time, the hole wall first cracks in the direction of the energy-accumulated hole, and then expands forward, upward, and downward[17,18].

When the blasthole wall cracks in the direction of the energy-accumulating hole, the continuous action of the energy-accumulating jet causes the crack to propagate forward, and also expands along the axis of the blasthole. Because the velocity of the explosion gas wedging into the fracture is less than the velocity of fracture propagation [19], and the velocity of the shaped jet is much greater than the velocity of fracture propagation.

When the blasthole wall cracks in the direction of the charge hole, the continuous expansion of the charge jet causes the crack to continue to expand, and it also expands along the axis of the hole. Because the velocity of the explosion gas wedging into the fracture is less than the velocity of fracture propagation[20], and the velocity of the shaped jet is much greater than the velocity of fracture propagation. This shows that the cracking process is still dominated by the shaped jet in the early stage of expansion, and at the same time, the pressure of the blasting gas creates favorable conditions for the expansion of the crack.

However, the test results confirm that the fracture length formed by the shaped jet is much shorter.
than the final length of the fracture. This indicates that after the effect of the shaped charge jet disappears, due to the quasi-static pressure effect of the blasting gas on the wall of the blasthole, the rock mass is further fractured and destroyed, providing power for the continued expansion of the fracture until the pressure of the blasting gas is insufficient to cause the expansion of the fracture. As shown in Figure 7. Suppose $a$ is the radius of the crack tip, $\alpha$ is the angle between the blasting crack growth direction and the main stress, $\beta$ is the slit angle, $q_0$ is the vertical stress, the effective force under the detonation gas is $P$. At the same time, the crack propagation caused by the pressure of the detonation gas will be affected by the high ground stress. When the tensile stress generated by the detonation gas is less than the ground stress, the crack development will be suppressed.

According to the above analysis, the crack growth is affected by the combined action of the blasting load and in-situ stress load. Next, we will get the crack growth length of the conventional blasting and the cumulative blasting respectively through the theoretical analysis.

Assuming that the explosive gas only produces a stable static pressure field that does not change with time, which can used the static method to analysis. In the action stage of blasting stress wave, it is known from elastic mechanics, the peak value of hoop tensile stress of rock element can be expressed as follows [21]:

$$ (\sigma_\theta)_m = P \frac{b}{r} r^{-\alpha} \tag{2} $$

Where $b$ is the ratio coefficient; $P$ is the peak value of compressive stress in the stage of stress wave action.

Make $(\sigma_\theta)_m = \tau$, obtained:

$$ r = (bP/\tau)^{{\frac{\theta}{\tau}}} a \tag{3} $$

Considering the defects of the rock mass, the damage factor $D$ is introduced to obtain the fracture development range under the non cumulative blasting:

$$ r = \frac{(bP/\tau)^{{\frac{\theta}{\tau}}} a}{1-D} \tag{4} $$

In the direction of energy accumulation, due to the penetration of shock wave, the scope of comminution area is expanded, so the energy consumed through comminution area is reduced, the energy of energy accumulation direction is increased, and the range of fracture expansion is increased. The crack growth length can be obtained by introducing the energy accumulation coefficient $\zeta$:

$$ r = \frac{(bP\zeta/\tau)^{{\frac{\theta}{\tau}}} a}{1-D} \tag{5} $$

Obtain the crack growth radius under the concentrated energy blasting:

$$ \frac{(bP/\tau)^{1+\mu/2+3\mu}}{1-D} a \leq R \leq \frac{(bP\zeta/\tau)^{1+\mu/2+3\mu}}{1-D} a \tag{6} $$

From the above formula: crack propagation radius is mainly related to the hoop tensile stress of rock element and the peak value of compressive stress in the stage of stress wave action. The hoop tensile stress of rock element is mainly related to the confining pressure around the rock mass. Formula (4,6) shows that the larger the confining pressure around the rock mass, the smaller the expansion radius of the rock mass crack.
5. Model

5.1. Model building

In order to study the evolution of blasting detonation waves and the distribution characteristics of pressure field under high stress conditions, a numerical model was established using ls-dyna software. The model size is 120 cm and 120 cm rectangular. The size of the energy-accumulated tube in the model is the same as the actual size. The diameter of the blast hole is 4.8 cm. The inner diameter and outer diameter are 3.6 cm and 4.2 cm, respectively, and the energy-accumulated hole width is 0.4 cm, as shown in Figure 8. Rock columnar cartridges are longer, and the damage to the section perpendicular to the longitudinal direction of the columnar cartridges is similar, simplifying the blasting model to a plane strain model. Considering the characteristics of deep-hole blasting, only one section perpendicular to the length of the explosive is used for blasting analysis. The problem is simplified as a plane strain model. In order to simulate the effect of ground stress on the rock, we use the dynamic relaxation method to simulate the influence of confining pressure. The dynamic relaxation method mainly calculates the influence of prestress before the blasting force. The dynamic relaxation method is a method introduced in LS-DYNA in order to solve the implicit problem, adding artificial damping to approximately solve the statics problem. In order to ensure the accuracy of the calculation and the convergence of the explicit central difference, this simulation uses a variable step integration method, and the time step selection is determined by the smallest element in the grid. Calculate the limit time step of each unit first, and take the minimum value of the next time step $\Delta t$. It can be found that the unit with the smallest geometric size controls the time step, and the appropriate time step can be achieved by adjusting the unit density and using quality scaling.
5.2. Model material

The rock boundary condition is set to the non-reflection boundary. In order to prevent the calculation interruption caused by the excessive deformation of the element, the ALE algorithm is used to solve the problem of large deformation of the grid. Since the rock is anisotropic, the elastoplastic model is more suitable for the rock constitutive model. The Lagrangian algorithm is used for the energy collection tube and the rock. The fluid-solid coupling relationship is established between explosive, air, energy gathering tube and rock. Using the MAT_HIGH_EXPLOSIVE_BURN material model and the EOS-JWL equation of state to simulate the relationship between the pressure and volume after the explosive exploded. In order to better reflect the damage of the rock, we introduced the MAT_ADD_EROSION failure parameter. The selection of simulated material parameters are all measured data in laboratory experiments. The material parameters in the numerical simulation are shown in Table 1.

| Type                      | Variable | Parameter                  | Unit  | Value   |
|---------------------------|----------|----------------------------|-------|---------|
| Roof rock                 | \( f_c \) | Uniaxial compressive strength | MPa   | 84      |
|                           | \( E \)  | Young’s modulus            | GPa   | 36      |
|                           | \( f_t \) | Uniaxial tensile strength  | MPa   | 5.3     |
|                           | \( v \)  | Poisson’s ratio             |       | 0.23    |
|                           | \( E_t \) | Tangent modulus            | GPa   | 13.1    |
|                           | \( \rho_s \) | Mass density               | kg/m³ | 2600    |
|                           | \( \varphi \) | Internal friction angle    | °      | 27      |
| Blasting                  | \( R \)  | Constant value of ideal gas| J/(mol·K) | 8.22   |
|                           | \( M_g \) | Relative molecular mass of gas| g/mol | 43      |
|                           | \( D \)  | Detonation velocity        | m/s   | 4250    |
|                           | \( R_0 \) | Mass density               | kg/m³ | 1200    |
|                           | \( P_c \) | Chapman-jouget pressure    | GPa   | 3.3     |
|                           | \( T \)  | Gas temperature            | K      | 348     |
|                           | \( Q \)  | Amount of explosive        | kg     | 2.8     |
| Energy-accumulated tube   | \( \rho_p \) | Density                    | kg/m³ | 1410    |
|                           | \( E_p \) | Elasticity modulus         | GPa   | 8.4     |
|                           | \( f_{st} \) | Tensile strength           | MPa   | 63      |
|                           | \( f_y \) | Yield strength             | MPa   | 74      |
|                           | \( \nu_p \) | Poisson’s ratio            |       | 0.31    |
5.3. Model boundary conditions
In order to simulate the high stress environment around the rock mass, we apply different confining pressure loads on the model boundary. The specific confining pressure application scheme is shown in Table 1.

![Table 2. different confining pressure loads](image)

6. Numerical simulation results

6.1. Blasting results under different confining pressure
The result of free blasting under different confining pressure is shown in the figure 9. Under different values of unidirectional confining pressure of 30μs, the crack propagation and stress cloud diagram, at 30μs, the stress wave basically propagates to the model boundary, which also shows that different initial stresses do not affect the propagation speed of the stress wave. In the following five figures, the range of the crushing zone formed by the explosive impact load is basically the same. However, the range of the blast fracture zone varies greatly. With the increase of confining pressure, the fracture range of the model keeps decreasing. Therefore, we know that the confining pressure has a certain restrictive effect on the development of burst lines.

![Figure 9. Free blasting under different confining pressure](image)
along the direction of energy accumulation. There were only a few cracks in the direction of non-energy accumulation. However, shaped energy blasting and free blasting are both affected by confining pressure. When the stress wave is transmitted to the boundary of the model, as the confining pressure increases, the length of the energy-concentrating crack propagation decreases continuously.

6.2. Blasting results under high stress conditions
In order to simulate the actual situation on site, the vertical load of 22.4 Mpa and the horizontal load of 32.7 Mpa are applied at the model boundary. The blasting results are shown in the figure 11, the development of cracks is also affected by high stress. From step 20, the growth of directional cracks has become very slow. The final propagation length of directional cracks is more reduced than the ordinary conditions. The final expansion length of one side of the directional crack is about 50cm.
It can be seen from the numerical simulation results that after the blasting of the blasthole constrained by high stress, the blasting crack pattern expands slowly, and no longer spreads divergently around, but is suppressed to a smaller range. Directional cracks are also affected by high confining pressure, the cracks grow slowly, and the final penetration length is less than 50cm. Therefore, the parameters of shaped charge blasting under normal conditions are not suitable for high-stress roadways, and the charge amount and blast hole spacing must be readjusted.

7. Engineering experiment
The roof of 4306 track roadway was directional pre-split blasting before mining, and after blasting, a continuous crack was formed along the roadway direction. According to the rock expansion coefficient and the thickness of the coal seam, the drilling depth is determined to be 8.5m. Through the research results of Gao Yubing and the numerical simulation results in this paper, combined with the actual production conditions on site, the final design of the blast hole is 200 mm from the side of the tunnel, the drilling direction is at an angle of 17° to the vertical line, and the spacing is set to 500 mm. In the blasting test, mine-use secondary emulsified explosives were used. The specifications of the explosives were Φ24 mm×300 mm/roll, and each roll of explosives was 200 g. The specific parameters of explosives are shown in Table 3. The energy gathering device uses a special energy gathering tube with an outer diameter of 42 mm, an inner diameter of 36.5 mm, and a tube length of 1500 mm, as shown in Figure 12a. The charge structure adopts the ”4+3+3+3+1” method, the charge amount is 2800g, and the blasting method uses continuous hole blasting. The specific charge structure is shown in Figure 12b.

Table 3. Emulsion explosive parameters

| Mass density (Kg/m³) | Detonation velocity (m/s) | Chapman-jouget pressure (Gpa) | Gas temperature (k) | Amount of explosive (kg) |
|----------------------|---------------------------|-------------------------------|---------------------|------------------------|
| 1200                 | 4250                      | 3.3                           | 348                 | 2.8                    |
Figure 12. Distribution structure of blasthole explosives. (a) energy-accumulate tube. (b) Charge structure.

The field pre-splitting blasting test uses the shaped charge blasting mode, and the test process is shown in the figure 13.

Figure 13. Field test procedure

After the blasting is completed, we can see a continuous crack appearing outside the blast hole, but in order to further check the crack distribution rate in the hole, we use a special instrument to view the situation in the hole. The image in the hole is shown in Figure 14. It can be seen that there are two parallel cracks appearing on the side of the hole wall, and cracks appeared from the top to the bottom of the hole, and the crack occurrence rate was close to 100%.
**Conclusion**

The 110 construction method is an innovative non-coal pillar mining process, and the directional presplit cumulative blasting technology is one of his key technologies. Because the 110 construction method is mostly used in shallow burial environments and is rarely used in deep conditions, we urgently need to obtain the shaped blasting parameters under high stress conditions. In this article, the following conclusions are drawn through theoretical analysis, numerical simulation and field test:

1) Through the analysis of related formulas, it is known that the expansion length of blasting cracks is mainly related to blasting load, ground stress and rock damage coefficient.

2) By comparing the numerical simulation results, it is found that the directional presplit cumulative blasting technology has a good energy gathering effect and is also applicable under high stress conditions and conventional conditions. However, both energy-accumulated blasting and non-energy-accumulated blasting are affected by high ground stress. Burst cracks develop slowly, and crack development is inhibited by high ground stress.

3) Field test results and numerical simulation results show that the charging structure adopts the method of "4 + 3 + 3 + 3 + 1", the charging amount is 2800g, and the blasting method adopts the continuous hole blasting have a good results.

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