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Fracture evolution between blasting roof cutting holes in a mining stress environment

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Abstract Blasting roof cutting and pressure relief is an effective technical way to solve the problem of thick and hard roof. In order to solve this problem, it is necessary to carry out research on the evolution of cracks between the cut holes of the blasting roof. The univariate comparative analysis method is used to analyze the evolution law of the fissures between the cuts under different factors. Furthermore, it is concluded that the broken zone and fissure zone of the surrounding rock of the single-hole blasting hole wall are symmetrically distributed in the confining pressure environment, and the fissure zone and the surrounding rock fissure zone between the holes show an "X"-shaped continuous evolution. By analyzing the evolution law of cracks between blasting holes, the critical discriminant equation of penetration between blasting holes under mining stress environment is given, which is used to optimize the technical plan of blasting roof cutting. Engineering practice shows that the blasting roof cutting scheme has achieved a good seam effect, creating good initial conditions for the cutting of thick and hard roofs.

Keywords Thick and hard roof, Blasting crack evolution, Inter-hole crack penetration, Roof cutting and seam forming

1. Introduction

As coal mining depth increases, geological conditions in which coal mining face encounters thick and hard roofs become more common. The lateral and rear overhanging range of the working face, which forms during the mining process, is greater under thick and hard roof conditions. Strong mining pressure appears, which has a severe impact on the stability of roadway bearing capacity. According to relevant studies, using presplitting blasting roof cutting technology to achieve roof directional splitting reduces the lateral cantilever length of the roof and the mining pressure on the roadway and has achieved good application effects in many mines (Liang et al. 2021; Wei et al. 2021). At present, blasting roof cutting has become one of the effective technical ways to solve this engineering problem. The key to achieving the goal of cutting roof is to evolve and penetrate cracks between blasting roof cutting holes. As a result, thick and hard roofs can be cut off along the crack direction,

reducing the suspended roof size.

In recent years, many experts and scholars have conducted a considerable research on the pre-splitting blasting of thick and hard rock formations. Wang et al. (2013) used deep-hole pre-splitting blasting technology to control the roof cutting and collapse of a shallow coal seam face in the Shendong mining area to avoid or reduce large-scale roof pressure. Zong Qi (1994) analyzed the influence of explosion shock waves on the expansion range of a crack and posited a correction formula for the radius of the crack area in a deep-hole blasting surrounding rock. Gao Jinshi and Zhang Jichun (1989) comprehensively considered the influence of stress wave and blasting gas on rock fractures and obtained formulas for calculating the radius of a blasting hole wall surrounding the rock fragmentation zone and fissure zone. Wang Congcong et al. (2018) used numerical simulation to study the crack expansion range of deep blasting. Lv Pengfei et al. (2012) used numerical simulation to study the expansion law of coal seam deep-hole-shaped energy blasting fractures. Zhao Jiechao et al. (2019) studied the effective

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fracturing range of coal seam deep-hole energy-accumulating blasting via mathematical model calculations. Dai Jun(2001) comprehensively considered the strain rate effect of rock under the impact load and the stress state of the actual rock, and based on the Mises strength criterion, derived the calculation formula for the radius of the fracture zone and the fracture zone formed by the explosion of the columnar explosive in the rock. Far and Wang(2016) posited a probability prediction formula of the radius of blasting fracture zone through research. In addition, many researchers have used numerical simulation softwares, such as FLAC, AUTODYN, and LS-DYNA to explore the impact of explosive impact loads on crack propagation(Zhu Z M et al.2007;Zhu W C et al.2013;Dehghan et al.2012). Cai Feng et al.(2007) used LS-DYNA to simulate the process of coal blasting and discussed the propagation characteristics of the explosion stress waves and expansion of coal cracks during blasting. Yang Renshu et al.(2016) used LS-DYNA to discuss the effect of the explosion stress waves and explosion gas on the blasting medium during the blasting process. Zhou Shengcai et al.(2013) used LS-DYNA simulation to study the propagation law of the explosion stress waves with different hole spacings.

Many of the abovementioned documents are based on the research on the evolution of cracks in a single-hole blasting in a laboratory environment. However, there is limited research on the evolution of cracks between blasting top holes in a mining stress environment. Sensitive factors affecting the evolution of cracks between blasting roof cutting holes under mining stress and related issues, such as the critical penetration criterion, need to be further investigated. Therefore, this article combines specific engineering examples, adopts field investigation, simulation, mechanical analysis, and other research methods, analyzes the appearance of strong mining pressure caused by the hard roof of a mining site, and examines dynamic loading of the blasting under mining stress environment. In this study, we analyze the mechanical propagation characteristics of the surrounding rock medium of a hole, reveal the evolution characteristics of cracks under the dynamic load of the surrounding rock blasting, and use the univariate comparative analysis method to cut the top of blasting under the conditions of different apertures, hole spacings, confining pressures, and rock mass strength. The evolution law of cracks between holes is analyzed, and the critical penetration criterion of cracks between blasting roof cutting holes in a mining stress

environment is further derived using the mechanical analysis method, which serves as a reference for blasting and topping technology decision making under similar engineering conditions.

2. Engineering background analysis

The main mining 3-1 coal in the 113103 working face of Bojianghaizi CoalMine in Ordos has an average buried depth of 670 m, an average mining thickness of 4.64 m, and an average inclination of 3°. The roof is managed by the total caving method. The return airway of the working face is 5.2-m wide and 3.8-m high, the width of the coal pillar is 11 m, and the anchor beam provides net support. The columnar distribution of the structure of roof rock strata is shown in Fig.1.

Field investigations have revealed that the roof is composed of mudstone and fine sandstone composite thick and hard roof rock formations, and the length of the suspended roof on the goaf side is large, causing the return airway to be severely affected by mining. The bottom heave of the roadway reaches 1.5–2.0 m, and the roof is anchored. The de-anchoring phenomenon occurs, and the roadway section shrinkage rate near the exit of the working face is more than 50%, as shown in Fig.2. The roadway section is reduced, wind speed is high, dust is flying, and movement of pedestrians and transportation of materials are difficult, which severely affects the production safety of the working face. Therefore, given the above problems, during the mining period, the roof of the roadway is blasted to cut the top and relieve the pressure to ensure safe mining.

Affected by the mining of the previous working face, the top slab of this working face broke into a cantilever beam structure, where Block A was bent and deformed under the action of the overlying rock load. Due to the long cantilever, the mining roadway was under the mining pressure environment. Block A is broken by blasting and topping, reducing its cantilever length and the rotational force acting on the roadway. The key technology when blasting the roof is that each blast hole crack can expand along the line of the blast hole and form inter-hole cracks that evolve and penetrate, causing the entire Block A to be cut off along the strike.

Fig.1 Column chart of roof structure

(a) Floor bulging

(b) Smaller roadway section

Fig.2 Roadway deformation diagram

3. Analysis of the evolution characteristics of cracks between blasting roof cutting holes

Aiming at the key issue of crack penetration between blasting roof cutting holes, this section investigates the evolution characteristics of cracks between blasting roof cutting holes.

3.1 Evolution characteristics of surrounding rock fissures in single-hole blasting

Taking the cut-roof blasting of the return air tunnel in the 113103 working face as an engineering background, a single-hole rock explosive blasting model was developed using ANSYS/LS-DYNA. The initial conditions of the model are as follows: the diameter of the blast hole is 7 cm, confining pressure of the model is 20 MPa, and the rock mass material parameters and explosive state equation are shown in Tables 1 and 2. We established a numerical model consisting of explosives, rocks, and air, and adopted a fluid–solid coupling algorithm. At the same time, to simulate the blasting process in an infinite rock and eliminate the influence of the reflection and superposition of stress at the boundary of the developed model on crack propagation, the surrounding borders were all set as nonreflective borders (GUO Deyong et al.2018).

Table 1 Explosive and its equation of state parameters

Density / ($\text{g} \cdot \text{cm}^{-3}$)	Detonation velocity / ($\text{m} \cdot \text{s}^{-1}$)	Burst pressure (GPa)	A_1	B_1	R_1	R_2	ω	E_0 (GPa)
1600	6900	4.0	214.4	0.182	0.9	0.15	0.15	4.192

Table 2 Rock mass material parameters

Density / ($\text{g} \cdot \text{cm}^{-3}$)	Elastic modulus /(MPa)	Poisson's ratio μ	Yield stress /(MPa)	Tangent modulus/(GPa)	Hardening coefficient	Dynamic tensile strength /(MPa)
2700	47.7	0.37	75	17.4	1	3

The simulation effect of single-hole blasting is shown in Fig.3. The average radius of the broken zone is 21 cm and average radius of the crack zone is 103 cm. In addition, the broken zone and the crack zone are symmetrically distributed.

Fig.3 Simulated crack propagation diagram of single-hole blasting

The total energy change curve of single-hole blasting is shown in Fig.4. After the main explosive is detonated, the accumulated detonation energy acts on the surrounding rock of the hole wall to form a high-strength blasting dynamic load and rapidly diffuses from the hole wall to the deep part of the surrounding rock. The detonation energy decays rapidly during the propagation process.

The dynamic load propagation and radial crack formation process of single-hole blasting are shown in Fig.5. Under the action of the dynamic load of blasting, the blast hole diameter is broken toward the shallow surrounding rock as shown in Fig.5 (a, b, c), and the hole diameter forms fissures toward the deep surrounding rock, as shown in Fig.5 (d, e, f). At the same

time, the explosive detonation energy quickly rushes into the fissures of the surrounding rock and produces violent dynamic load expansion on the surrounding rock media in the fissures, thereby forming a “divergent” blasting aperture to the fissure distribution characteristics, as shown in Fig.5 (g, h, i).

Fig.4 Energy variation curve of single-hole blasting

Through the simulation of the radial fissure evolution of single-hole blasting, the following is observed: (1) the dynamic load of blasting is severely attenuated during the diffusion and propagation process, (2) scale of blasting dynamic load fracturing is limited, (3) detonation energy can easily flow into the fissure space of the surrounding rock and aggravate the expansion of the fissure, and (4) blast fracture gap extends and evolves from the hole wall to the deep part of the surrounding rock. From this analysis, it is concluded that the key to achieving the goal of blasting cutting and perforating the joint is to make full use of the limited blasting dynamic load to form an effective inter-hole fissure penetration on the surrounding rock of the hole wall.

Fig.5 Dynamic load propagation process of single-hole blasting

3.2 The evolution law of cracks between the top holes of blasting

To further reveal the sensitive influencing factors and laws of cut-roof blasting in the mining stress environment, the univariate comparative analysis method is used to analyze the influence of different apertures, hole spacings, confining pressure, surrounding rock strength, and other factors on the evolution of perforation cracks in cut-top blasting. We established a numerical model of blasting top cutting (size: $6000 \times 3000 \times 2$ cm), as shown in Fig.6. The five holes have no

explosives in the middle hole, which is used as a peephole to observe the penetration effect of the burst rupture gap. Confining pressure is uniformly applied to the surrounding nonreflective boundary. The univariate comparative analysis method is used to design the simulation comparison plan, as shown in Table 3.

Fig.6 Numerical simulation model

Table 3 Simulation scheme of single quantity comparative analysis

	Blast hole diameter	Hole spacing	Confining pressure	Tension-compression ratio
Option 1	30 mm,	0.8 m	X, Y: 20 MPa	0.04
	50 mm			
	70 mm,			
	90 mm			
Option 2	70 mm	0.6 m,	X, Y: 20 MPa	0.04
		0.8 m		
		1.0 m,		
		1.2 m		
Option 3	70 mm	0.8 m	X, Y: 20 MPa,	0.04
			30 MPa	
			60 MPa,	
			90 MPa	
Option 4	70 mm	0.8 m	X, Y: 20 MPa	0.04
				0.0625
				0.125
				0.25

3.2.1 Influence of aperture on the evolution of cracks between roof cutting blasting holes

The simulation results of the blasting rupture gap penetration with different apertures are shown in Table 4 and Fig.7. The crack between the two blasting holes with a diameter of 30 mm is not penetrated, and the propagation distance of the edge hole crack is only 43 cm. Cracks between the two blasting holes with a diameter of 50 mm are penetrated, but the number of cracks is relatively small. The influence range of the crack propagation of the edge blasting hole is increased to 75 cm. An

aperture of 70-mm blasting is reserved for peephole cracks to penetrate, and edge blasting hole cracks spread up to 104 cm. Both sides of the peephole reserved for blasting with a diameter of 90 mm are broken through, and the cracks of the edge blasting hole spread the farthest, i.e., 107 cm. In addition, the blasting gap expansion of the upper and lower parts of the two blasting holes with a diameter of 70 mm is better than other blasting effects. According to the analysis, the larger the aperture, the more explosives are required in the blast hole, and the higher is the blasting dynamic load generated after blasting, which in turn will form a larger rupture gap evolution scale. Only when sufficient detonation energy is achieved can the blast fracture between holes be formed, and the pores can evolve and penetrate.

Table 4 Propagation table of blasting through cracks with different diameters

Aperture	Penetration of reserved peepholes	Farthest propagation distance of edge hole crack
30 mm	Not penetrated	43.7 cm
50 mm	Single fracture penetration	75.1 cm
70 mm	Multi fracture penetration	104.0 cm
90 mm	Crushing penetration	107.6 cm

(a) Aperture 30mm

(b) Aperture 50mm

(c) Aperture 70mm

(d) Aperture 90mm

Fig.7 Evolution and distribution characteristics of cracks between blasting roof cutting holes with different hole diameters

3.2.2 Influence of hole spacing on the evolution of blasting perforation cracks

As shown in Table 5 and Fig.8, when the hole spacing is 0.6 m and 0.8 m, the peephole and the blasting holes on both sides form a gap between the holes and penetrate closely. When the hole spacing is 1.0 m, only the cracks between the peepholes and the blasting holes on both sides have the potential to approach and penetrate. When the hole spacing is 1.2 m, the peephole and blasting holes on both sides fail to form a crack between the holes to penetrate closely. The analysis shows that the larger the hole spacing, the greater is the blasting dynamic load transfer and attenuation stroke, and the greater is the difficulty in forming the crack penetration in the center of the hole spacing. In addition, even a sufficiently large distance between the holes cannot realize the effective blasting and perforation crack evolution. Combined with the analysis of the hole diameter of the cut-top blasting hole, the hole diameter determines the scale of the blasting dynamic load and the range of surrounding rock fracturing, whereas the hole spacing is the effective stroke of cracks between holes. The hole diameter and spacing need to be designed in tandem to make full use of the limited dynamic load of blasting to achieve the expected goal of cutting the top blasting and piercing into a seam.

Table 5 Propagation of blasting through cracks with different hole spacings

Hole spacing	Penetration of reserved peepholes	Farthest propagation distance of edge hole crack
0.6 m	Crushing penetration	107.5 cm
0.8 m	Fissure penetration	104.0 cm
1.0 m	Not penetrated	96.4 cm
1.2 m	Not penetrated	103.4 cm

(a) Hole spacing 0.6m

(b) Hole spacing 0.8m

(d) Hole spacing 1.0m

(d) Hole spacing 1.2m

Fig.8 Evolution and distribution characteristics of cracks between blasting roof cutting holes under different hole spacings

3.2.3 Influence of different confining pressures on the evolution of blasting perforation cracks

As shown in Table 6 and Fig.9, under different confining pressure conditions, the reserved peepholes have crack penetrations. Confining pressures of 20–30 MPa have minimal effect on the blasting effect. When the confining pressure increases to 60 MPa, the upper and lower rock cracks near the midpoint of the two blasting holes decrease. When the confining pressure increases to 90 MPa, the range of the broken zone between the two blast holes becomes smaller, expansion of the cracks around the single-blast hole is suppressed, and the length and number of cracks are notably reduced. The simulation results show that the greater the confining pressure, the greater is the impact of the rock mass on the cracking resistance of the blasting dynamic load, resulting in greater energy loss during the transmission of the blasting dynamic load. Moreover, a longer distance crack expansion effect cannot be produced. However, the confining pressure has minimal effect on the perforation effect of cut-roof blasting within a certain range. When the confining pressure increases to 90 MPa, the fracture area of the blast hole is remarkably reduced. Therefore, when blasting the roof cutting, the stress environment of the roadway roof should be grasped in time. In the case of stress concentration areas, timely change the blasting roof cutting plan to increase the explosive amount or reduce the hole spacing to improve the roof cutting blasting effect.

Table 6 Penetration propagation of blasting cracks under different confining pressures

Confining pressure	Penetration of reserved peephole	Farthest propagation distance of edge hole crack
20 MPa	Fissure penetration	104.0 cm
30 MPa	Fissure penetration	102.3 cm
60 MPa	Fissure penetration	93.1 cm
90 MPa	Fissure penetration	86.4 cm

(a) Confining pressure 20MPa

(b) Confining pressure 30MPa

(c) Confining pressure 60MPa

(d) Confining pressure 90MPa

Fig.9 Evolution and distribution characteristics of cracks between blasting roof cutting holes under different confining pressures

3.2.4 Influence of rock mass strength on the evolution of blasting perforation cracks

As shown in Table 7 and Fig.10, the cracks between the rock blasting holes of different strengths can penetrate the reserved peepholes. The blasting effect is not considerably different when the tension–compression ratio is between 0.04 and 0.125; only the single-hole blasting gap evolution scale is slightly reduced. When the tension–compression ratio increases

to 0.25, the expansion range of the single-hole blasting fracture gap decreases rapidly, and the rock fissure near the midpoint of the two blasting holes disappears. The cracks near the blasting hole are arranged in parallel, which is different from the previous irregular free extension, as shown in Fig.11(d). The simulation results show that the strength of the rock mass has minimal influence on the effect of blasting cutting and perforation. However, the expansion of the blast fracture is caused by the micro-cracks in the rock mass under the action of blasting dynamic load. The larger the size, the stronger is the detonation pressure tension it can withstand, resulting in fewer burst rupture gaps and reduced propagation distance. Combined with blasting roof cutting, the rock mass strength determines the difficulty of explosive blasting dynamic load on rock mass breaking and propagation. Designing a reasonable blasting roof cutting scheme under different strengths can effectively increase the crack propagation range and improve the success rate of roof cutting.

Table 7 Penetration propagation of blasting cracks with different tension–compression ratios

Tension–compression ratio	Penetration of reserved peeppholes	Farthest propagation distance of edge hole crack
0.04	Fissure penetration	104.0 cm
0.0625	Fissure penetration	103.6 cm
0.125	Fissure penetration	101.6 cm
0.25	Fissure penetration	90.8 cm

(a) Tension compression ratio 0.04

(b) Tension compression ratio 0.0625

(c) Tension compression ratio 0.125

(d) Tension compression ratio 0.25

Fig.10 Evolution and distribution characteristics of cracks between blasting roof cutting holes under different tension–pressure ratios

4. Mechanical analysis of crack penetration between blasting cut holes in mining stress environment

According to the evolution characteristics of cracks between blasting roof cutting holes revealed in the previous section, this section establishes the fracture forming criterion of blasting roof cutting perforation in a mining stress environment using a mechanical analysis method and obtains the critical breakdown spacing criterion to assist in optimizing the technical scheme of roof cutting blasting.

After the explosive is detonated, the blasting dynamic load of high temperature and pressure impacts the hole wall on both sides of the hole, and its peak stress is remarkably greater than the compressive strength of the rock mass. The energy of the shock wave decreases rapidly while crushing the rock until the energy is exhausted. Therefore, the explosion impact area can

be roughly divided into blasting crushing zone and blasting fissure zone (Guo Deyong et al.2019), that is, the stress and failure distribution of the surrounding rock of the blast hole wall under blasting dynamic load is shown in Fig.11 (Xu Zhilun et al.2016).

Fig.11 Schematic of blasting analysis

In Fig.11, the inner circle is the range of the crushing area, with a radius of a ; outer circle is infinite, with a radius of m ; detonation pressure at the edge of the crushing area transmitted by the dynamic load of explosive blasting is P' (P represents the size of the external uniformly distributed load), and the radius of the fracture area is set as b .

Under engineering blasting conditions, P_1 represents the initial impact pressure on the hole wall (Guo Deyong et al.2019):

$$P_1 = n_1 \cdot \frac{\rho_0 V_0}{8} \cdot \left(\frac{d_c}{d_b} \right)^6 \cdot \left(\frac{l_c}{l_b} \right)^2 \quad (1)$$

In the formula, n_1 represents the explosion impact pressure coefficient, generally $n_1 = 8-9$; ρ_0 represents explosive density; V_0 represents explosive detonation velocity; d_c and d_b are roll and blast hole diameters, respectively; and l_c and l_b are the axial charge and axial chamber lengths, respectively.

The radius r_1 of the crushing zone of the available cartridge blasting is as follows:

$$r_1 = r_0 \left[\frac{n_1 \left[(1+\beta)^2 + (1+\beta^2) - \right]^{1/2} \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt{\varepsilon}} \cdot \left(\frac{d_c}{d_b} \right)^6 \cdot \left(\frac{l_c}{l_b} \right)^3 \right]^{2 + \frac{\mu_d}{1-\mu_d}} \quad (2)$$

In the formula: ε is the loading strain rate. In engineering blasting, the rock loading rate ε is between 10 and 10^5s^{-1} (Shan Renliang et al.1997). In the crushing zone, the loading rate is higher, which can be taken as $\varepsilon = 10^2 \sim 10^4 \text{s}^{-1}$; in the fracture zone, the loading rate is further reduced, which can be taken as $\varepsilon = 10^0 \sim 10^3 \text{s}^{-1}$, β is the lateral stress coefficient, $\beta = \mu_d(1-\mu_d)$; σ_c is the static compressive strength of the rock mass; r_0 is the radius of the blasthole; μ is the Poisson's ratio of the rock mass, and μ_d is the dynamic Poisson's ratio. Under engineering blasting conditions (Dai J 2001), generally $\mu_d = 0.8\mu$.

Because the blasting effect is affected by numerous factors and the research on this problem is insufficiently comprehensive, the coefficients λ_1 and λ_2 are introduced. According to the relevant research (Guo Deyong et al.2016; Yang Y Q et al.1995), $\lambda_1 = 0.7-0.9$ is used in the crushing zone.

After introducing λ_1 , the radius a of the crushing zone can be obtained as follows:

$$a = \lambda_1 \cdot r_1 \quad (3)$$

According to elastic mechanics, the force at any point in polar coordinates can be expressed as follows (Shan Renliang 1997):

$$\begin{cases} \sigma_r = \frac{\partial^2 \phi}{\partial y^2} = \frac{1}{r} \frac{\partial \phi}{\partial r} + \frac{1}{r^2} \frac{\partial^2 \phi}{\partial z^2} \\ \sigma_\theta = \frac{\partial^2 \phi}{\partial x^2} = \frac{\partial^2 \phi}{\partial r^2} \\ \tau_{rz} = -\frac{\partial^2 \phi}{\partial x \partial y} = -\frac{\partial}{\partial r} \left(\frac{1}{r} \frac{\partial \phi}{\partial z} \right) \end{cases} \quad (4)$$

Under axisymmetric conditions, the Airy stress equation of the surrounding rock of the blast hole wall is

$$\phi = \phi(r) \quad (5)$$

Under axisymmetric conditions, the stress component equations of the surrounding rock of the hole wall are as follows:

$$\begin{cases} \sigma_r = \frac{1}{r} \frac{\partial \phi}{\partial r} \\ \sigma_\theta = \frac{\partial^2 \phi}{\partial r^2} \\ \tau_{rz} = \tau_{rz} = 0 \end{cases} \quad (6)$$

Therefore, the compatible equation for the problem between axisymmetric holes is

$$\left(\frac{d^2}{dr^2} + \frac{1}{r} \frac{d}{dr}\right)\phi = 0 \quad (7)$$

The general solution is:

$$\phi = A \ln r + B r^2 \ln r + C r^2 + D \quad (8)$$

In the formula, A, B, C, and D are undetermined coefficients.

Here, formula (4) can be expressed as follows:

$$\begin{cases} \sigma_r = \frac{A}{r^2} + B(1 + 2 \ln r) + 2C \\ \sigma_\theta = -\frac{A}{r^2} + B(3 + 2 \ln r) + 2C \\ \tau_{rz} = \tau_{zr} = 0 \end{cases} \quad (9)$$

From the literature (Guo Deyong et al.2019), the dynamic load P' of blasting at the edge of the crushing zone can be obtained as follows:

$$P' = P_1 / \bar{r}^\alpha \quad (10)$$

In the formula, $\bar{r}$ represents the comparison distance, $\bar{r} = r_i / r_0$, r_i represents the distance from any point to the blast hole center, r_0 represents the blast hole radius, α represents the stress wave attenuation coefficient, and $\alpha = 2 \pm \mu_d / (1 - \mu_d)$, take positive in the shock wave action area and negative in the compressive stress wave action area.

Therefore, the boundary conditions are determined as follows:

At boundary $r = a$,

$$(\sigma_r)_{r=a} = -P' \quad \tau = 0 \quad (11)$$

At boundary $r = m$,

$$(\sigma_r)_{r=m} = -P \quad \tau = 0 \quad (12)$$

Combined with the displacement condition, $B = 0$ can be obtained, and A and 2C can be obtained as follows:

$$\begin{cases} A = \frac{\left(\lambda_1 \cdot r_0 \left(\frac{n_1[(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b}\right)^6 \cdot \left(\frac{l_c}{l_b}\right)^3 \right)^{\frac{1}{2+\frac{\mu_d}{1-\mu_d}}} \right)^2}{m^2 - \left(\lambda_1 \cdot r_0 \left(\frac{n_1[(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b}\right)^6 \cdot \left(\frac{l_c}{l_b}\right)^3 \right)^{\frac{1}{2+\frac{\mu_d}{1-\mu_d}}} \right)^2} \\ 2C = \frac{P' \left(\lambda_1 \cdot r_0 \left(\frac{n_1[(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b}\right)^6 \cdot \left(\frac{l_c}{l_b}\right)^3 \right)^{\frac{1}{2+\frac{\mu_d}{1-\mu_d}}} \right)^2 - P m^2}{m^2 - \left(\lambda_1 \cdot r_0 \left(\frac{n_1[(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b}\right)^6 \cdot \left(\frac{l_c}{l_b}\right)^3 \right)^{\frac{1}{2+\frac{\mu_d}{1-\mu_d}}} \right)^2} \end{cases} \quad (13)$$

According to the problem of a circular hole with an infinite boundary, the radial and circumferential stress of the surrounding rock of the blasting hole are as follows:

$$\begin{cases} \sigma_r = -P' - \left(1 - \frac{1}{r^2} \right) \left(\lambda_1 \cdot r_0 \left(\frac{n_1[(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b}\right)^6 \cdot \left(\frac{l_c}{l_b}\right)^3 \right)^{\frac{1}{2+\frac{\mu_d}{1-\mu_d}}} \right)^2 P \\ \sigma_\theta = P' - \left(1 + \frac{1}{r^2} \right) \left(\lambda_1 \cdot r_0 \left(\frac{n_1[(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b}\right)^6 \cdot \left(\frac{l_c}{l_b}\right)^3 \right)^{\frac{1}{2+\frac{\mu_d}{1-\mu_d}}} \right)^2 P \end{cases} \quad (14)$$

By substituting (14) into the Griffith strength theory $k\sigma_{td} = -\frac{(\sigma_1 - \sigma_3)^2}{4(\sigma_1 + \sigma_3)}$ (Li Tonglin 1991), the equation of fracture zone

radius b can be obtained as follows:

$$k\sigma_{td} = \frac{1}{8P} \left(-2P' + \frac{2P}{b^2} \left(\lambda_1 \cdot r_0 \left(\frac{n_1 [(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b} \right)^6 \cdot \left(\frac{l_c}{l_b} \right)^3 \right)^{2+\frac{1}{1-\mu_d}} \right)^2 \right)^2 \quad (15)$$

In the formula, k represents the strength change coefficient of the rock mass under dynamic load, generally taken as 1.3 (Gong Fengqiang 2010).

Therefore, the critical criterion $[d]$ for penetrating the blasting roof crack under the mining stress environment is

$$\begin{cases} [d] \leq 2nR \\ k\sigma_{td} = \frac{1}{8P} \left(-2P' + \frac{2P}{b^2} \left(\lambda_1 \cdot r_0 \left(\frac{n_1 [(1+\beta)^2 + (1+\beta^2) - 2\mu_d(1-\mu_d)(1-\beta)^2] \rho_0 V_0^2}{8\sqrt{2}\sigma_c \sqrt[3]{\varepsilon}} \cdot \left(\frac{d_c}{d_b} \right)^6 \cdot \left(\frac{l_c}{l_b} \right)^3 \right)^{2+\frac{1}{1-\mu_d}} \right)^2 \right)^2 \end{cases} \quad (16)$$

In the formula, $[d]$ represents the critical criterion for the penetration of the blasting head gap and n is the effective safety factor, which is taken as 0.8 in the text.

5. Engineering practice

According to the site working conditions, Poisson's ratio $\mu = 0.25$, $n = 9$, explosive density $\rho_0 = 1600 \text{ g/cm}^3$, explosion velocity $V_0 = 6900 \text{ m/s}^{-1}$, loading rate $\varepsilon = 10^3 \text{ s}^{-1}$, compressive strength of rock mass $\sigma_c = 68.68 \text{ MPa}$, cartridge diameter $d_c = 35 \text{ mm}$, blast hole diameter $d_b = 70 \text{ mm}$, axial charge length $l_c = 1.2 \text{ m}$, axial chamber length $l_b = 1.5 \text{ m}$, and blast hole radius $r_0 = 35 \text{ mm}$ λ_1 for 0.9. The radius of the crushing zone ($a = 20.76 \text{ cm}$) can be obtained by substituting formulas (2) and (3). Then, the explosive explosion pressure $P_I = 4000 \text{ MPa}$, confining pressure $P = 20 \text{ MPa}$, radius of crushing area $a = 20.76 \text{ cm}$, tensile strength of rock mass $\sigma_{td} = 2.75 \text{ MPa}$, $k = 1.3$. The value of blasting fracture zone b under confining pressure of 20 MPa can be obtained by substituting formulas (10) and (16). Through calculation, the range of the blasting crack area under 20 MPa is $b = 100.58 \text{ cm}$.

Due to the thick hard fine sandstone with strong integrity of roadway roof, along with numerical simulation, theoretical calculation and analysis, and economic factors, the designed roof cutting angle is 75° , drilling diameter is 70 mm, drilling depth is 11–22 m, and the spacing of blasting holes is 0.8 m, as shown in Fig. 12.

According to the actual situation of the project, the buried depth of the coal seam is 670 m, and the confining pressure of 17 Mpa is applied. The simulated blasting results are shown in Fig.13. The blasting holes are broken and penetrated, reserved

peephole is penetrated by cracks, maximum range of cracks in edge holes is 105 cm, and crack penetration effect between blasting top cutting holes is good.

(a) Drilling construction layout

(b) Top view of borehole layout

Fig.12 Drilling construction layout

Fig.13 Engineering Blasting simulation diagram

According to the technical scheme of blasting roof cutting, the on-site industrial test is conducted, and the reserved peepholes are selected at different positions for peeping. The on-site peeping results are shown in Fig.14. Obvious cracks can be seen in the peepholes, and the surrounding rocks in some holes collapse locally, indicating that the blasting roof cutting effect is good, and the roof can be completely cut off.

Fig.14 Image result diagram of speculum

The measured results of roadway surrounding rock deformation before and after roof cutting are shown in Fig.15. The influence of advanced mining is more than 180 m, 80 m in front of the work is affected by mining, and the severe influence range is 60 m in front of the work. After roof cutting, the average approach of the roof and floor 40 m in front of the working face decreases from 1537 to 1047 cm, average

approach of two sides decreases from 1325 to 837 cm, and deformation of roadway surrounding rock evidently decreases. After the roof cutting, the deformation of the roadway 80–200 m behind the adjacent working face gradually tends to be stable, and the integrity of the surrounding rock of the roadway can meet the production and safety requirements. By analyzing the measured results of roadway deformation, it can be seen that blasting roof cutting is conducive to reducing roadway deformation and increasing the stability of roadway and surrounding rock.

Fig.15 Comparison of roadway deformation before and after roof cutting

6 Conclusion

(1) In a confining pressure environment, the fracture zone of the surrounding rock on the hole wall of single-hole blasting is symmetrically distributed with the fracture zone, and the diameter of the fracture zone is 5–7 times that of the blasting hole, and the diameter of the fracture zone is approximately 5 times that of the fracture zone. During blasting top cutting, the broken area of the surrounding rock between holes and the fracture area show “X” type penetration evolution. The blasting dynamic load between adjacent holes forms a superposition effect in the transmission process to enhance the blasting effect of fracture penetration between holes.

(2) When blasting roof cutting is implemented, the hole diameter has the most obvious impact on the expansion scale of blasting crack. The larger the hole diameter, the stronger is the superposition effect of blasting dynamic load, and the larger is the evolution scale of crack expansion. The hole spacing has a particularly key impact on the effect of top cutting and crack formation. Moreover, the confining pressure and rock mass strength have an inhibitory effect on the development of blasting crack expansion, and the impact effect is small in a certain range.

(3) Through mechanical analysis, the critical criterion of blasting top clearance penetration in the mining stress

environment is deduced. Along with the field peeping results, it is concluded that the key to blasting top clearance pressure relief is that through cracks can be formed between blast holes after blasting, and the entire thick and hard roofs can be cut off along the cracks to achieve pressure relief.

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