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Xiaoding Xu (✉ ding_xu@cumt.edu.cn)

Jilin University

Yubing Gao

China University of Mining and Technology

Manchao He

China University of Mining and Technology

Qiang Fu

China University of Mining and Technology

Xingjian Wei

China University of Mining and Technology

Research Article

Keywords: Mining pressure, artificial fault, concentrated stress, pressure transfer

Posted Date: March 23rd, 2021

DOI: <https://doi.org/10.21203/rs.3.rs-333799/v1>

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Study on Mining Pressure Control of Deep Coal Seam—— Based on Artificial Fault Technology

Xiaoding Xu^{a,b,*,†}, Yubing Gao^{b,*,†}, Manchao He^{a,b}, Qiang Fu^b, Xingjian Wei^b

^a College of Construction Engineering, Jilin University, Changchun 130026, China

^b State Key Laboratory for Geomechanics & Deep Underground Engineering Beijing, China University of Mining & Technology, Beijing 100083, China

* Correspondence: yubing.gao@yahoo.com(Y.b Gao), ding_xu@cumt.edu.cn (X.d Xu);

† All authors contributed equally to this work.

Abstract: Based on the pressure transfer principle and stress distribution characteristics around a fault, introducing artificial fault technology to control the propagation of abutment pressure, and a mechanical model of abutment pressure under the influence of artificial fault was established. This new mechanical model can well fit the distribution law of mining stress after roof cutting. The pressure transfer mechanism of prefabrication support of rock blasting was analyzed, and the transfer trend of pressure and the mining stress of rock top was determined. It is of great significance to guide the implementation of the pressure relief work at the top of the stope. The study shows that the total energy of the system is conserved, the integrity of rock layer is destroyed by blasting, and the deformation and damage of pressure relief zone absorb a large amount of energy. Thus, the accumulated strain energy of abutment pressure region is released, and the influencing range of abutment pressure is reduced. As the horizontal distance from the cutting surface is farther away from the working surface, the smaller the stress difference on both sides of the cutting top, the less obvious the blocking effect of mining pressure. When the cutting point is closer to the working surface, the higher the peak value of abutment pressure due to the superposition of peripheral stress concentration caused by the cutting and peak of abutment pressure caused by mining. Then, the numerical simulation analysis was carried out, the results show that the technology of forming artificial fault by cutting the top can cut off the influence range of the mining pressure. It can effectively control the deformation of dynamic pressure tunnel. Finally, a practice of rock blasting pressure relief engineering was carried out, and the influencing range of abutment pressure of working face before blasting pressure reduction was reduced by 1/3 compared with that before the pressure relief.

Keywords: Mining pressure; artificial fault; concentrated stress; pressure transfer.

1. Introduction

Mining pressure originates from the redistribution of stress caused by the movement of overlying strata in the mining face after coal seam mining. The key layer theory states that when the overlying strata are thicker and the rock stratum is harder, it can be overlaid [Qian *et al.*(2010), He *et al.*(2005), Guo (2018), Grady and Kipp (1987), Taylor *et al.* (1986), Kuszmaul (1987), Throne (1990), Gao (2013)]. In the rock stratum, the key layer plays the role of the main body. At this time, the influencing range of the abutment pressure is obviously increased. When the tunnel is close to the stoppage line and the coal pillar of retaining lane is insufficiently reserved, the abutment pressure and rock stratum are broken. The occurrence of Mining pressure will cause stress disturbance to the tunnel in front of the working face, leading to a large deformation of the tunnel [Li *et al.*(2012), Meng *et al.* (2016), Wang *et al.* (2008), Zhang *et al.* (2012), Zhang *et al.* (2016), Zhang *et al.* (2011), Jiang *et al.* (2016), Zhang *et al.* (2017), Xu *et al.* (2019)]. Therefore, it is especially important to control the range of dynamic pressure. In this study, a new method was developed to control the influencing range of mining pressure by cutting the top pressure relief structure.

Chinese scholars have also carried out extensive studies on stress distribution around the fractured zone. Liu and Zhang (2013) combined the theory of top-loading pressure relief with the actual production of underground mines to solve the problem of surrounding rock instability in the mining tunnel after the coal pillars are left behind. Good control of deformation instability of mining tunnel was achieved. Pu and Zhang (2014) used the top-loading pressure relief technology in the downhole to reduce the overburden pressure on the tunnel along the gob, so that the deformation of tunnel was well controlled. Wan *et al.* (2014) proposed precracking and reported that a combination of high-strength and high-preloading anchor cables could successfully strengthen the support to protect the tunnel along the road at the scene. Wang *et al.* (2014) and others conducted theoretical analysis and optimization of deep hole precracking and pressure-relief related construction of roof and solved the problems of large deformation of surrounding rock in a coal seam tunnel difficult to maintain and difficult to replace. Tang and Sheng (2015) and others solved problems in a special production environment of multi coal mining in coal mines when the deformation of surrounding rock of tunnel is large. They proposed to use presplitting and then cut the transmission between overburden pressures to reduce the pressure of tunnel. A test was carried

63 out on site, and the remaining tunnel was cut due to the pressure transmission mechanism, so that
64 the deformation amount was significantly reduced. Chen *et al.* (2015) used LS-DYNA simulation
65 software to theoretically deduce the effect of guide hole on a deep-hole precracked roof under the
66 condition of hard overburden in a thick coal seam. The test data surface was extracted from the hole
67 to the deep hole. The cracking effect is better and can be promoted appropriately. Based on the idea
68 of cutting off pressure transmission on the roof of coal seam, He *et al.* (2017) proposed the “110
69 method,” i.e., roof precracking was carried out at a certain angle by covering the rock layer in the
70 mining trough, and the constant resistance was high. The tightening anchor cable improves the
71 support strength of mining tunnel, successfully protects the mining and smoothing groove, and
72 shows the application of automatic coal pillar-free technology in the field.

73 Meng *et al.* (2006) used mechanical tests and numerical simulation methods to evaluate the
74 effect of normal faults on working face pressure. This study shows that with a close distance of
75 working face from the fault, the abutment pressure of working face is obviously increased, and the
76 support is supported. The stress peak is present between the working surface and fault. Yu and Xie
77 (1998) studied the effect of geometry of fault plane on its activation characteristics based on fractal
78 theory. A numerical software was used to analyze the stress distribution around the fault plane of
79 different fractal features. Li *et al.* (2010) established a mechanical model for fault and rockburst
80 induced prediction, studied the mechanism of fault-type rockburst occurrence, and analyzed the
81 effects of different stress states on the occurrence of fault-type rock pressure. Li *et al.* (1982)
82 conducted a ground stress test on the North China Sea faults by onsite measurement. It was found
83 that the principal stress directions remained unchanged at different distances, but the principal stress
84 values were quite different. Ding *et al.* (1981) measured the *in-situ* stress of a fault in Yunnan; the
85 results show that the principal stress direction around the fault is consistent with the fault strike. Su
86 (2002) used numerical simulation to study the effects of friction angle, cohesion, and rock stiffness
87 on the stress distribution around a fault. The study shows the effect of internal friction angle of rock
88 mass on the stress distribution around the fault.

89 The large deformation of tunnel has always been a big problem in mine support work. At
90 present, the main treatment methods for tunnel include the enhanced surrounding rock support
91 strength method, tunnel deep hole relief pressure tank method, tunnel surrounding rock blasting
92 pressure relief. These methods do not cut off the propagation path of mining load from the root
93 source, nor change the stress distribution law at the far end. Therefore, this study focuses on the

mechanism of rock mass blasting and topping pressure relief and determines the stress distribution of leading bearing after pressure relief.

2. Engineering background

The average depth of 14200 working face of Pingdingshan Coal Mine is 625 m; the average slope width is 173.5 m; the average coal thickness is 2.0 m; the area is 295175 m²; the bulk density is 1.4 tons/m³. The dynamic pressure affects the tunnel is located 100 m in front of the original stop line of 14200 working face and in the mudstone 15 m above the coal seam. When the coal seam is redistributed after the mining, the weight of overburden layer in the original coal seam shifts to the periphery, thus increasing the abutment pressure above the working face. The overall trend is that the abutment pressure of working face is small, and the pressure is within the distance of working face. When the abutment pressure reaches its peak value, the elevated zone decreases rapidly and reaches the original rock stress as the distance increases, as shown in Fig. 1. Because of the effect of main key layer of 30 m thick at 80 m above the coal seam, the effect of abutment pressure on the working face is 100 m, which causes the tunnel to deform greatly, as shown in Fig. 2. .

Fig.1. General information of the working face

Fig.2. Tunnel deformation under dynamic pressure

The key to the large deformation problem of tunnel in stope is to find the root cause of disturbance of tunnel. Therefore, the formulation of a governing method is the key to analyze the basis of stress distribution law of stope. To control the deformation of tunnel from the root source, this study proposes a deep hole blasting top-loading pressure relief technology for rock formation, which uses the cavity formed by blasting and the deformation of fracture zone to absorb the high stress energy to reduce the influencing range of abutment pressure, thereby reducing the stress concentration around the tunnel.

3. Analysis of principle of pressure transfer in blasting and cutting

After rock blasting, the integrity of rock stratum is destroyed, and a blasting cavity and fracture zone are formed in the rock stratum. The effect is similar to that of nonfalling fault. Because of the fault rupture zone, the coal rock mass is relatively broken with the effect of stress relief and fracture. The stress unloading at the belt causes the stress distribution of leading working face to shift, thereby acting as a pressure relief to reduce the influencing range of stress. The pressure transfer mechanism can be analyzed using energy theory. Because the total energy of rock mass is composed of rock mass fracture energy and strain energy is accumulated in the rock body, according to the law of conservation of power:

$$f = W_c + W_d \quad (1)$$

where f is the total energy of system; W_c is the strain energy reaccumulated by the rock mass after the coal seam is opened; W_d is the strain energy absorbed by various forms such as deformation and failure of rock mass.

Eq. (1) shows that the abutment pressure at each point of rock layer can be reduced by

reducing the accumulated strain energy W_c at each point. Because the total energy of the system is constant, to lower the W_c value, the W_d value should be increased. Increasing W_d means increasing the deformation and damage of rock mass within a certain range. The principle of blasting pressure relief is to form some broken and cracked areas in the overburden species, so that the deformation absorbs a large amount of energy and increases the W_d .

At the same time, after rock blasting, the integrity of rock stratum is destroyed, and a blasting cavity and fracture zone are formed in the rock stratum. The effect is similar to artificial fault. Therefore, the blasting can be analyzed using the prestressed stress distribution law under the effect of faults. The distribution law of abutment pressure after the topping was determined, as shown in Fig. 3. The fault fracture zone concentrates the surrounding stress, concentrates the high-end stress on both sides of fault, and reduces the distal stress, so that the influencing range of abutment pressure is shortened.

Fig.3. Diagram of stress distribution under the influence of faults

4. Pressure distribution model of strata with artificial faults

4.1 Mechanical model of stress distribution of abutment pressure before cutting

With reference to the solution to the plane problems as to the action of concentrated force on semi-infinite body in elasticity theory, a calculation model of vertical stress formed by load in a semi elastomer can be established, as shown in Fig. 4.

Fig.4 Arbitrary point force calculation model

Where ds is the unit length s away from the O point, then the force per unit length can be approximated as the concentrated force, $dF = q(s)dx$. According to the knowledge of elastic mechanics, the stress at any point of the semi-infinite body under the concentrated force can be solved, and combined with the mathematical integration algorithm, the stress at any point in the semi-infinite body under the load $q(x)$ can be obtained as follows:

$$\begin{cases} \sigma_y = \frac{2}{\pi} \int_0^a \frac{q(x)y^3}{[y^2 + (x-s)^2]^2} ds \\ \sigma_x = \frac{2}{\pi} \int_0^a \frac{q(x)(x-s)^2 y}{[y^2 + (x-s)^2]^2} ds \\ \tau_{zx} = \frac{2}{\pi} \int_0^a \frac{q(x)(x-s)y^2}{[y^2 + (x-s)^2]^2} ds \end{cases} \quad (2)$$

Model assumptions:

- 1) The overlying rock mass of coal seam is a uniform elastic isotropic material.
- 2) Longwall mining is considered as a plane strain problem along the direction of advancement of working face.

According to the characteristics and assumptions of prestressed bearing stress distribution curve of mining face, a calculation model was established for the bearing stress arbitrary point, as shown in Fig. 5. According to the simplified abutment pressure curve, it can be divided into three sections: ab section, bc section, and cd section, and the upper stress function of each section can be obtained using the force geometric similarity theory.

Fig.5. The calculation model of abutment pressure transferring

The pressure functions can be expressed as follows:

Section ab :

$$q_{ab}(x) = \gamma h \quad (3)$$

Section bc :

$$q_{bc}(x) = \frac{(K-1)\gamma h}{x_c - x_b}(x - x_b) + \gamma h \quad (4)$$

Section cd :

$$q_{cd}(x) = \frac{K\gamma h}{x_d - x_c}(x_d - x) \quad (5)$$

The vertical stress of each point on half-face can be obtained according to the theory of arbitrary point force calculation of elastic mechanics.

When $s \in (-\infty, x_b)$, the vertical stress of ab segment load at any point of hemisphere can be expressed as follows:

$$\sigma_{ab}(x, y) = \frac{\gamma h y (x_b - x)}{\pi [y^2 + (x - x_b)^2]} + \frac{\gamma h}{\pi} \arctan \left(\frac{x_b - x}{y} \right) + \frac{1}{2} \gamma h \quad (6)$$

When $s \in (x_b, x_c)$, the vertical stress of bc segment load at any point of hemisphere can be expressed as follows:

$$\sigma_{bc}(x, y) = \left\{ \frac{\gamma h y \left[(K(x - x_b) + x_c - x)s + (x_b(K-1) - x_c + x_b)x - (K-1)(x^2 + y^2) \right]}{\pi(x_c - x_b)(y^2 + (x - s)^2)} \right\} \Bigg|_{x_b}^{x_c} + \frac{\gamma h y}{\pi(x_c - x_b)} (K(x - x_b) + x_c - x) \arctan \left(\frac{s - x}{y} \right) \quad (7)$$

When $s \in (x_c, x_d)$, the vertical stress of cd segment load at any point of hemisphere can be expressed as follows:

$$\sigma_{cd}(x, y) = \left\{ \frac{K\gamma hy(x^2 + y^2 - xs + (s-x)x_d)}{\pi(x_d - x_c)(y^2 + (x-s)^2)} + \frac{K\gamma hy(x_d - x)}{\pi(x_d - x_c)} \arctan\left(\frac{s-x}{y}\right) \right\} \Bigg|_{x_c}^{x_d} \quad (8)$$

After superimposing the vertical stresses of each load using the principle of stress superposition, The vertical stress at any point produced by the whole load can be obtained as follows:

$$\begin{aligned} \sigma_y(x, y) = & \frac{\gamma hy(x_b - x)}{\pi[y^2 + (x - x_b)^2]} + \frac{\gamma h}{\pi} \arctan\left(\frac{x_b - x}{y}\right) + \frac{1}{2}\gamma h \\ & + \left\{ \frac{\gamma hy \left[(K(x - x_b) + x_c - x)s + (x_b(K-1) - x_c + x_b)x - (K-1)(x^2 + y^2) \right]}{\pi(x_c - x_b)(y^2 + (x-s)^2)} \right. \\ & \left. + \frac{\gamma hy}{\pi(x_c - x_b)} (K(x - x_b) + x_c - x) \arctan\left(\frac{s-x}{y}\right) \right\} \Bigg|_{x_b}^{x_c} \\ & + \left\{ \frac{K\gamma hy(x^2 + y^2 - xs + (s-x)x_d)}{\pi(x_d - x_c)(y^2 + (x-s)^2)} + \frac{K\gamma hy(x_d - x)}{\pi(x_d - x_c)} \arctan\left(\frac{s-x}{y}\right) \right\} \Bigg|_{x_c}^{x_d} \end{aligned} \quad (9)$$

Because of the rupture of rock mass after the blasting pressure, the elastic half-plane theory cannot be used directly to solve the stress state of rock layer after blasting. However, the fracture zone obtained after blasting pressure relief is similar to the fault-free fault, so the theory of elastic hemisphere can be used. Combined with the stress distribution theory of fault zone and the principle of stress superposition, the distribution of abutment pressure after blasting pressure relief can be solved.

4.2 Mechanical model of stress distribution in abutment pressure after cutting

Based on the stress distribution characteristics on both sides of fault-free fault zone, a stress distribution mechanical model was constructed on both sides of pressure-relief belt (artificial fault). Let k be the stress attenuation coefficient at the lowest point of stress reduction zone, k' be the concentration coefficient of peak stress in the stress-increasing zone, and q is the original stress. The rock stress and stress of each section are linearly distributed. The center point of pressure relief zone is the minimum stress kq of the stress reduction zone, and the peak stress of stress zone is $k'q$, as shown in Fig. 6.

Fig.6. Mechanical model of stress distribution on both sides of artificial fault

Model assumptions:

- 1) The overlying rock mass of coal seam is a uniform elastic isotropic material.
- 2) The medium in pressure relief belt is a completely elastic material.
- 3) The width of pressure relief belt is negligible compared to a large range of rock mass.

Fig.7. Calculation model of arbitrary point force of bearing stress with artificial fault

Based on the stress redistribution characteristics on both sides of artificial fault and the analytical solution of abutment pressure under the unloading state, the distribution law of lead abutment pressure after pressure relief can be obtained. As shown in Fig. 7, the overburden after demolition and blasting can be divided into two large areas: the blasting pressure relief zone ($e'emn$ zone) and the nonblasting pressure relief zone. The influence area of blasting pressure relief is affected by the stress zones in Fig. 4 and Fig. 5 at the same time, so it can be divided into four zones, namely, $c-f$ zone (Zone 1), $c-f'$ zone (Zone 2), $b-e$ zone (Zone 3), and $b'-e'$ zone (Zone 4). The stress of four regions can be solved by equation (9) in combination with Fig. 6.

The distributed stress in the $c-f$ zone (Zone 1) of blasting pressure relief zone can be calculated

235 as follows:

$$236 \quad \begin{cases} \sigma_1(x, y) = k\sigma_y + \frac{\sigma(k'-k)(x-x_f)}{x_c - x_f} \\ x \in (x_f, x_c), y \in (y_m, y_n) \end{cases} \quad (10)$$

237 Blasting pressure relief affects the distribution stress calculation in Zone 2 as follows:

$$238 \quad \begin{cases} \sigma_2(x, y) = \sigma_y + \frac{\sigma_y(k'-1)(x-x_e)}{x_f - x_e} \\ x \in (x_e, x_f), y \in (y_m, y_n) \end{cases} \quad (11)$$

239 Blasting pressure relief affects the distribution stress calculation in Zone 3 as follows:

$$240 \quad \begin{cases} \sigma_3(x, y) = k'\sigma_y + \frac{\sigma_y(k'-k)(x-x_{f'})}{x_{f'} - x_{e'}} \\ x \in (x_c, x_{f'}), y \in (y_m, y_n) \end{cases} \quad (12)$$

241 Blasting pressure relief affects the distribution stress calculation in Zone 4 as follows:

$$242 \quad \begin{cases} \sigma_4(x, y) = k'\sigma_y + \frac{\sigma_y(k'-1)(x-x_{f'})}{x_{f'} - x_{e'}} \\ x \in (x_{f'}, x_{e'}), y \in (y_m, y_n) \end{cases} \quad (13)$$

243 In summary, an analytical formula for the distribution after the demolition of rock blasting can
244 be expressed as follows:

$$245 \quad \begin{cases} \sigma_1(x, y) = k\sigma_y + \frac{\sigma(k'-k)(x-x_f)}{x_c - x_f} & x \in (x_f, x_c) \\ \sigma_2(x, y) = \sigma_y + \frac{\sigma_y(k'-1)(x-x_e)}{x_f - x_e} & x \in (x_e, x_f) \\ \sigma_3(x, y) = k'\sigma_y + \frac{\sigma_y(k'-k)(x-x_{f'})}{x_{f'} - x_{e'}} & x \in (x_c, x_{f'}) \\ \sigma_4(x, y) = k'\sigma_y + \frac{\sigma_y(k'-1)(x-x_{f'})}{x_{f'} - x_{e'}} & x \in (x_{f'}, x_{e'}) \end{cases} \quad (14)$$

246 4.3 Analysis of engineering examples

247 Based on the geological conditions of 14200 working face of Pingdingshan No. 8 Mine, the
248 distribution of abutment pressure of coal seam working face under different cutting conditions was
249 calculated. The horizontal distance between the peak point of fracture zone and working face is 10

250 m, and the stresses on both sides of peak point are symmetrically distributed. The height of blasting
 251 cutting roof is 30 m, and the ranges of stress reduction zone and stress rise zone caused by blasting
 252 chopping are both 5 m. The calculation parameters are shown in Table 1.

253

254

255

Tab.1. Working surface abutment pressure calculation parameters

K	k	k'	$\gamma/\text{kN/m}^3$	h/m	y/m
1.5	0.6	1.1	25	600	30

256

257 To study the distribution law of abutment pressure at different pressure relief locations, five
 258 calculation schemes were set: horizontal distances of blasting pressure relief belt from the working
 259 surface of 10 m, 20 m, 30 m, 40 m, and 50 m. The parameters were substituted into Formula (14).
 260 The magnitude of abutment pressure formed by the overburden load in the coal seam can be
 261 obtained, and the abutment pressure curve can be obtained after fitting, as shown in Fig. 8.

262

Fig.8. Abutment pressure curves at different horizontal cutting positions from the working face

264 The buried depth of coal seam is 625 m, and the average weight of overburden is 25 kN/m³.
 265 The vertical stress of coal seam was calculated to be 15.6 MPa when not excavated. According to
 266 the relevant Mining pressure disturbance theory, it was determined whether it is a significant
 267 influencing area by 1.05 times of the stress under the original rock stress state, i.e., 16.4 MPa. If the
 268 vertical stress is higher than 16.4 MPa, it is considered to be at the working face. Within the scope
 269 of influence and vice versa, there is no effect, so a horizontal straight line of 16.4 MPa was added in

270 Fig. 8 to measure the dynamic pressure influencing range under different pressure relief positions.
271 Fig. 8 shows that when the cutting top is 10 m away from the working surface, the stress difference
272 between the two sides of cutting top is 6 MPa, and the effect of lead abutment pressure is 80 m.
273 When the cutting point is 20 m away from the working surface, the stress difference between the
274 two sides of cutting top is 3 MPa, and the influencing range of abutment pressure is also ~80 m.
275 When the cutting point is 30 m away from the working surface, the stress difference between the
276 two sides of cutting roof is 3 MPa, and the effect of abutment pressure is 90 m. When the cutting
277 point is 40 m away from the working surface, the stress difference between the two sides of cutting
278 roof is ~2.5 MPa, and the effect of abutment pressure is 105 m. When the cutting point is 50 m
279 away from the working surface, the stress difference between the two sides of cutting top is ~2 MPa,
280 and the effect of abutment pressure is 110 m. In a certain range, the distance from the cutting
281 surface to the working surface is farther, and the top is cut into two halves. The smaller the lateral
282 stress difference, the less obvious the effect of mining pressure blocking. However, from the
283 analysis of abutment pressure peak of working face, the closer the cutting top to the working
284 surface, the higher the peak value of abutment pressure, mainly because of the cutting area. The
285 distortion effect of edges results in peripheral stress concentration.

286 Based on the above analysis, the mechanical model can be well fitted to the distribution curve
287 of abutment pressure after blasting and cutting, showing the law of pressure transfer of abutment
288 pressure after blasting and cutting and controlling the influencing range of abutment pressure.
289

290 **5. Numerical simulation analysis of the pressure relief of the roof**

291 *5.1 Establishment of the numerical model*

292 A numerical model is established based on the actual geological and production conditions of
293 the 14200-working face in the Pingba Mine using Flac3D[(Mehravar *et al.*(2017), Kabwe *et al.*
294 (2020)],. The coal seam depth is 620m, and the coal seam is 100 meters from the surface. A
295 30m-thick medium-grain sandstone is the key layer approximately 80m above the coal seam. The
296 top interface of the model is applied with the equivalent compressive stress of the overburden layer
297 is $P=13\text{Mpa}$. The overall model size is $440\text{m (X)} \times 320\text{m (Y)} \times 153\text{m (Z)}$, a total of 2076800
298 grids and 2134860 nodes, as shown in Fig. 9.
299

Figure 9 Numerical model of the strata

The rock masses in this simulation are all coal-bearing strata which generally exhibits yield behavior under the shear stress. The Mohr-Coulomb model is selected for the simulation, and the mechanical parameters of the rock mass used in the numerical simulation are shown in Table 2.

Tab.2 Numerical simulation of physical and mechanical parameters

Rock group category	$\rho/\text{kg} \cdot \text{m}^{-3}$	E/GPa	G/GPa	c/MPa	$\phi/^\circ$	σ_v/Mpa
Coal	1400	1.67	1.02	1.15	28	1.12
Mudstone	2200	5.00	3.00	1.52	30	1.56
Sandy mudstone	2320	9.63	6.35	3.86	32	2.36
Purple spot mudstone	2350	12.65	8.66	4.32	35	2.98
Fine sandstone	2550	1.68	1.23	2.21	36	5.28
Middle sandstone	2650	3.16	1.95	3.68	38	2.65

In the simulation process, setting of weakening parameters of the blasting zone with reference to existing research, the weakening parameters of the blasting zone are: blasting cavity width is 0.2m; the blasting cavity is set as the center, and an area with a radius of 2 meters is the blasting weakening area (Table 3).

314

Tab.3 Calculation parameters for the blasting weakened area

$\rho/\text{kg} \cdot \text{m}^{-3}$	E/GPa	G/GPa	c/Mpa	$\varphi/^\circ$	σ_t/Mpa
1300	0.30	0.18	0.06	6	0.15

315

316 *5.2 Numerical simulation scheme and result analysis*

317 In order to analyze the distribution of the mining stress in the face of the work and the
318 deformation of the tunnel, we choose two working conditions for comparative analysis, which are
319 under the condition of no pressure relief and under the condition of roof cutting pressure relief. The
320 simulation solution process is as follows: establish the overall model, calculate the original rock
321 stress balance, form the original in-situ stress field, and open the pressure relief cut seam (pressure
322 relief position: the horizontal distance from the stope line is 15m, the height of the pressure relief
323 belt is 30m, and the width of the pressure relief belt is 4m). The working face is mined once with
324 25m as the circulation ring, and excavated four times with a total of 100m to make the overlying
325 strata in the goaf fully move, to solve and obtain the vertical stress nephogram in front of the stope
326 line and the vertical stress curve of the coal seam, so as to analyze the influence of the pressure
327 relief position of the roof on the vertical stress in the coal and rock body and the deformation of the
328 tunnel.

329

331

332

Fig. 10 Abutment pressure curve before and after pressure relief

333 The red horizontal line in Fig.10. is the original rock stress in the case of no mining, so the
334 cross coordinate of the intersection of the abutment pressure curve and the original rock stress line
335 is the influence range of the mining pressure. The comparison between the two figures shows that

the influence range of mining pressure before pressure relief is about 120m, the influence range of mining pressure after pressure relief is about 75m, and the pressure relief shortens the influence range of mining pressure by 45m, indicating the effectiveness of cutting top pressure relief for blocking load transmission path and shortening the influence range of mining. The result is consistent with the trend of theoretical calculation.

Fig. 11 Vertical stress cloud diagram before and after pressure relief

It can be seen from Fig.11. that, compared with the scheme before pressure relief, the peak stress area in front of the coal wall of the working face disappears after pressure relief; after pressure relief, the load transfers to the vicinity of the pressure relief area, and stress concentration occurs on both sides of the pressure relief area. After the pressure relief, the influence range of the advance bearing stress is obviously reduced. Before the pressure relief, the mining advance bearing stress overlaps with the concentrated stress produced by the tunnel excavation. The stress concentration of the two sides of the tunnel is serious. After the pressure relief, the propagation of the advance bearing stress is cut off. The influence range of the advance bearing stress is greatly reduced compared with that before the pressure relief, and the stress concentration around the tunnel disappears.

Fig.12 tunnel displacement cloud map before and after blasting pressure relief

It can be seen from Fig.12. that before the pressure relief, the deformation of the tunnel is large, the roof sinks 32.7cm, the floor heave is 55.8cm, the roof and floor displacement reaches 88.4cm; the deformation of the left side is 50.4cm, the deformation of the right side is 53.8cm, and the displacement of the two sides reaches 104.2cm. After blasting, the deformation of surrounding rock in the tunnel was well controlled. The roof subsidence was 10cm, the floor heave was 19.7cm, the roof and floor displacement was 29.7cm, 66.4% less than before; the left side deformation was 13.5cm, the right side deformation was 17.9cm, the two sides displacement was 31.4cm, 69.8% less than before. Therefore, the deformation of the tunnel in front of the work was effectively controlled by pressure relief.

6. Rock layer blasting and cutting project practice

6.1 Blasting pressure relief area and parameters

The rationality of blasting pressure relief position directly affects the pressure relief effect. To reduce the influencing range of abutment pressure, the peak value of abutment pressure can be controlled, and the effect on the working surface is reduced. The theoretical analysis result is combined with the cutting position. It should be 10~20 m away from the stop line. Finally, the pressure relief blasting should be carried out 15 m ahead of the stop line of working face. The blasting drilling arrangement is shown in Fig. 13.

Fig.13. Layout of blastholes

According to the geological conditions of working area, the lateral length of working face, the

position of rock formation where the drill is located, the radius of influence of drilled blasting, the spacing of drilling holes, and the height of pressure relief, the drilling of working face can be designed as follows: In the noodle machine lane and wind lane, a set of oblique upward-penetrating boreholes are arranged 15 m away from the stoppage line, which is fan-shaped, as shown in Fig. 14. The borehole lengths are 85 m, 65 m, 86 m, 70 m, 58 m, 48 m, 40 m, 35 m, and 30 m, and the angles with the coal seam are 10°, 15°, 20°, 26°, 30°, 38°, 45°, 57°, and 69°, a total of nine holes; three green holes among them are explosive-free control holes.

Fig.14. A-A section

Coal mine three-stage emulsion explosives (45*400 mm/volume, 440 g/volume) were used. A millisecond delay electric detonator was used to detonate the explosives. Single-hole detonators were connected in series, and the holes were connected in series with the holes. Positive blasting was used, the bottom of hole was charged, and the outer mouth was sealed with yellow mud. The length of sealing mud is not less than one third of length of hole.

Tab.4 Charge parameters of blasthole

Number	Hole depth / m	Angle / °	Explosive length / m	Sealing length / m
A1	85	10	30	55
A2	65	15	30	35
A3	86	20	30	56
A4	70	26	0	3
A5	58	30	30	28
A6	48	38	0	3
A7	40	45	25	15
A8	35	57	0	3
A9	30	69	15	15
B1	85	10	30	55
B2	65	15	30	35
B3	86	20	30	56
B4	70	26	0	3

B5	58	30	30	28
B6	48	38	0	3
B7	40	45	25	15
B8	35	57	0	3
B9	30	69	15	15

397 6.2 Monitoring plan and analysis

398 6.2.1 Monitoring plan

399 (1) Monitoring of artificial fault

400 The method of ultrasonic detection is used to monitor the artificial fault formed by blasting,
401 assisting in the side of the dynamic pressure roadway, receiving the same, and doing reflection wave
402 exploration, the monitoring instrument is shown in Fig.15. The monitoring spots were set on the
403 side of the roadway. There were 12 receiving spots set up with a spacing of 12 m, and the excitation
404 point was located in the middle of the receiving point; the total length of the monitoring line was
405 approximately 144 m. The monitoring point arrangement is shown in Fig.16.

407 **Fig.15. Ultrasonic detection system**

Fig.16. Schematic diagram of station layout

(2) Monitoring of coal stress

Combined with the specific conditions of mine, a total of two Coal stress measuring stations are arranged on the 14200 working face, as shown in Fig.16: Station 1 is located on the left side of blasting borehole, 25 m away from the design stop line, and Station 2 is located on the right side of blasting borehole, 25 m away from the design stop line (10 m away from the pressure relief position). Each measuring station was arranged with three measuring points. The horizontal distance between the three measuring points was controlled within 1 m. The measuring depth of each measuring station is 5 m, 10 m, and 15 m, and the measuring hole diameter is 42 mm.

6.2.2 Analysis of ultrasonic detection results

The direction of the seismic wave exploration in the mine was the bedding direction. When the integrity of rock strata is good, the change of wave velocity is small. When there is a fault fracture zone in the detection area, the wave velocity will show abnormal changes, and the original data is shown in Fig. 17.

Fig.17. Original waveform of ultrasonic detection

After normalizing the original amplitude data, the artificial fault formed by blasting is

426 represented by dark bright spots, as shown in Fig. 18. The width of the artificial fault is about 2.5m,
 427 the middle part is dark red, and the color on both sides gradually becomes lighter. This is because
 428 the middle part of the blasting area, the rock damage is more serious, the color is dark red. The two
 429 sides are fracture areas, where the rock damage is very small, so the color becomes lighter.

430
 431 **Fig.18. Nephogram of ultrasonic detection**

432 6.2.3 Analysis of coal stress results

433 When the working surface is recovered to a range of ~ 10 m from the station, the data shown in
 434 the pressure gauge were collected using a data acquisition instrument. According to the measured
 435 data, the data of each measuring point were analyzed separately. Combined with the progress
 436 schedule of 14200 working face, the working surface was promoted by 4.5 m per day. The distance
 437 from the working station to the measuring station was calculated every day. A scatter plot of
 438 pressure gauge pressure over time was obtained, as shown in Fig.19. The influence range of
 439 abutment pressure can be calculated by the number of days when the slope of the pressure curve
 440 suddenly increases and the advancing rate of working face.

Fig.19. Pressure curve of each station

Fig.19 (a) shows the pressure change of Station 1. When a stress monitoring system was arranged, the reading of pressure gauge fluctuated slightly and then stabilized. After the stress monitoring system was arranged for three days, the reading of pressure gauge increased slightly. This indicates that the inner wall of borehole was completely in contact with the pressure gauge at this time, and the pressure on the pressure gauge slightly increased. At this time, the working surface advanced for three days. The working face is 116 m away from the Station 1. When the working surface was advanced for seven days, the increase in pressure gauge reading is slightly increased compared with the increase after three days. This indicates that the drilling hole entered the influencing range of leading stress, and the drilling is caused by the pressure of coal body at the drilling hole. The deformation increased gradually, so the pressure of pressure gauge increased synchronously until the 16th day after the pressure gauge was placed. The reading of pressure gauge started to increase slowly, indicating that the deformation of borehole became relatively stable. When the working surface was advanced for 25 days, the pressure of pressure gauge started to increase linearly. This indicates that the drilling hole is located near the peak of abutment pressure of working face, and the deformation of drilling hole continued to increase. At this time, the measuring station is 17 m away from the working surface. In the case of curve, the number of days of pressure increase is 25 days, and the daily footage of working face is 4.5 m. The influencing range of abutment pressure of working face is 129.5 m.

Fig.19 (b) shows the pressure change of Station 2. After the stress monitoring system was arranged for 10 days, the reading of pressure gauge started to increase, indicating that the inner wall of borehole is in full contact with the pressure gauge. At the same time, the pressure on pressure

gauge slightly increased. At this time, the working surface is 80 m away from Station 2. After that, the pressure curve continued to rise, and at 21 days after loading the table (the working surface is 30 m away from Station 2), the pressure increased significantly, indicating that the stress peak occurred at this time. Because the working face is stopped at a distance of 25 m from Station 2, Station 2 failed to measure the overall stress of abutment pressure after blasting. However, according to the above analysis, the effect of abutment pressure after blasting is 80 m.

7. Conclusion

This study analyzed the distribution law of stope stress, constructed a new mechanical model for rock mass excavation and pressure relief, elucidated the pressure relief mechanism of rock blasting, and carried out engineering practice. The following conclusions can be drawn:

(1) Using the theory of energy conservation to analyze the principle of pressure transfer after cutting the top of rock, it was found that the total energy of system is constant, composed of rock energy and the accumulated strain energy. The blasting cut causes the fracture zone to deform and destroy a large amount of energy. The accumulated strain energy in the abutment pressure region is released, which in turn reduces the influencing range of abutment pressure.

(2) Based on the pressure transfer principle and stress distribution characteristics around the fault zone, and introducing the coefficient of stress change K , k and k' , a new pressure transfer mechanical model was constructed after blasting and cutting is. The stress mechanism of abutment pressure was analyzed, and an analytical formula was deduced for the mining stress after the crest of rock formation. The results show that the smaller the distance between the cut-off and top of cut-off, the smaller the stress difference on both sides of cut-off within a certain range, but the effect of stop pressure on the working face is less obvious. When the cutting point is closer to the working surface, the higher peak value of abutment pressure is mainly due to the superposition of peripheral stress concentration caused by the cutting and the peak of abutment pressure caused by the mining.

(3) The distribution of the abutment pressure pressure and the deformation of the tunnel before and after the pressure relief were analyzed by numerical simulation. The results show that after the pressure relief, the peak stress area in front of the coal wall disappears and transfers to the area near the pressure relief area, and the stress concentration appears on both sides of the blasting pressure relief area. After blasting and pressure relief, the influence range of the advance bearing stress

495 decreased obviously, and the stress concentration around the tunnel was not obvious. The
496 displacement of roof and floor is 29.7cm, 66.4% less than that before pressure relief, and the
497 displacement of two sides is 31.4cm, 69.8% less than that before pressure relief. It can be seen that
498 blasting pressure relief can effectively control the deformation of the tunnel in front of the work.

499 (4) The results of ultrasonic detection show that the artificial fault zone was well formed, the
500 width of the fault zone was about 2.5m, the damage in the middle was serious, and the damage on
501 both sides is reduced. The results of coal stress test show that the influencing range of abutment
502 pressure of working face before blasting pressure relief is 129.5 m. The influencing range of
503 abutment pressure after blasting pressure relief is 80 m. This is 49.5 m shorter than the pressure
504 before the pressure relief, which is reduced by nearly 1/3, shortened from the previous 150 m to 90
505 m. Therefore, rock blasting pressure relief can effectively shorten the influencing range of abutment
506 pressure of working face. The application of blasting pressure relief tunnel technology has great
507 significance to reduce the coal pillar size of tunnel and improve the mine recovery rate and dynamic
508 pressure tunnel deformation control.

509

510 **Compliance with ethical standards**

511 This work is supported by the State Key Laboratory for Geomechanics and Deep Underground
512 Engineering, China University of Mining & Technology, Beijing (No. SKLGDUEK1928), the
513 National Natural Science Foundation of China (No. 51674265) and Project funded by China
514 Postdoctoral Science Foundation (No. 2020T130702), which are gratefully acknowledged.

515

516 **Conflict of interest**

517 The authors declare that they have no conflict of interest.

518 **Data Availability:**

519 The manuscript data used to support the findings of this study are available from the corresponding
520 author upon request.

521

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592

Figures

Figure 1

General information of the working face

Figure 2

Tunnel deformation under dynamic pressure

Figure 3

Diagram of stress distribution under the influence of faults

Figure 4

Figure 5
The calculation model of abutment pressure transferring

Figure 6
Mechanical model of stress distribution on both sides of artificial fault

Figure 7

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Figure 8

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Figure 9

Numerical model of the strata

Figure 10

Abutment pressure curve before and after pressure relief

Figure 11

Vertical stress cloud diagram before and after pressure relief

Figure 12

tunnel displacement cloud map before and after blasting pressure relief

(a) 3D diagram

(b) 2D diagram

Figure 13

Layout of blastholes

Figure 14

A-A section

Figure 15

Ultrasonic detection system

Figure 16

Schematic diagram of station layout

Figure 17

Original waveform of ultrasonic detection

Figure 18

Nephogram of ultrasonic detection

(a)station 1

(b)station 2

Figure 19

Pressure curve of each station