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Research and Engineering Practice of Coupled Unloading and Supporting Prevention Rock Burst Technology of Hard Coal Seams in Ordos Region

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Abstract: Hard coal seams are important geological feature of rockburst in the Ordos region. In response to the poor pressure relief effect of conventional drilling in hard coal seams in the Ordos region, as well as the problems of high-density and multi round pressure relief and support failure, a non equal pressure stress field model for drilling was established. The mechanical analysis of the plastic zone radius in drilling was derived. This analysis revealed how the coal seam strength and drilling diameter affect the pressure relief effect of drilling. A "shallow support and deep pressure relief" hard coal seam unloading support coupling and expanding pressure relief technology was proposed. A 70m comparative industrial test was conducted on site, and the pressure relief and support effect was tested using methods such as coal powder monitoring, coal gun monitoring, stress monitoring, and surrounding rock deformation monitoring. The conclusion was drawn that: (1) The radius of the plastic zone in drilling is primarily influenced by coal seam strength and drilling deformation. The drilling diameter affects the strength of the coal seam. A smaller plastic zone radius is associated with a larger drilling diameter, while a larger plastic zone radius corresponds to a larger diameter as well. However, the increase in pressure relief effect diminishes as the diameter increases. Therefore, simply increasing the drilling diameter cannot indefinitely improve the pressure relief effect. (2) In terms of the pressure relief effect, the amount of coal powder produced in the expanded pressure relief zone has increased by 3.1 times compared to conventional drilling. The "coal cannon" events are concentrated in the 7-14m section of the coal wall, which can significantly release the accumulated elastic energy inside the coal body; After unloading, the peak stress of the coal body decreases, and the recovery rate becomes slower, which has a significant unloading effect on the concentrated static load area. (3) In terms of the support effect, during and after the pressure relief period of borehole expansion, the stress on the roof of the roadway and the anchor cables on the front wall is stable, and the stress on the anchor cables is lower than that of conventional drilling; In the course of the expansion and pressure relief phase, the maximum observed reduction in the lateral movement of the tunnel's two sides was 63%, while the maximum reduction in the vertical movement of the top and bottom plates reached 51%. The efficacy of this method in mitigating the deformation of the surrounding rock in the tunnel significantly surpasses that achieved through traditional large-diameter drilling techniques. (4) We have developed a complete set of anti-erosion technology and equipment based on mechanical drilling and pressure relief, forming an engineering demonstration. The research results can provide theoretical basis and engineering reference for mines with similar conditions in the Ordos region.

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Keywords: Ordos region; Hard coal seam; Impact ground pressure; Unloading support coupling; Mechanical reaming

1 Introduction

Rock burst is a typical dynamic disaster that occurs during the coal mining process. [1-3]. In recent years, there has been a notable annual increase in the incidence of rock bursts in mines located within the Ordos region. The mining operations at certain sites have exhibited significant deformation of the surrounding rock in the roadways, accompanied by pronounced dynamic impact phenomena. These issues pose substantial challenges to the safe operation of the mines and hinder the optimization of production capacity. The research on rock burst in this area has become the focus of academic and engineering circles [4-7].

The hard coal seam represents a significant geological characteristic associated with rock bursts in the Ordos region. The presence of the hard coal seam leads to a substantial accumulation of elastic energy in the vicinity of the roadway, resulting in pronounced dynamic effects. Notably, approximately 86.8% of rock bursts are observed to occur within the roadway. [8-9]. In addressing the challenges associated with the control of surrounding rock in roadways within hard coal seams, numerous researchers have engaged in pertinent investigations. Recognizing the complexities involved in managing surrounding rock stability in these environments, a significant body of scholarly work has been dedicated to this issue. Han et al. [7] posited that the presence of a hard roof and a hard coal seam are significant geological features associated with this particular type of rock burst. They further identified the overall slippage of the coal body along the roadway side as the primary characteristic of its impact failure. Pan Yishan et al. [10] established the three-level support theory and technical system of coal mine rock burst roadway. Pan Junfeng et al. [11] analyzed and summarized the engineering technical problems of high and one low in deep rock burst prevention and control, and put forward the local targeted pressure relief prevention and control technology. Liu Guolei et al. [12] proposed the mechanism of stress difference gradient of surrounding rock in coal roadway and introduced the anti-impact method of active support, passive support, and coal seam transformation. Zhu et al. [13] elucidated the coordinated control mechanism governing rock bursts and significant deformations within thick coal seams. Their research indicates that by optimizing the support processes and pressure relief parameters, it is possible to mitigate the issue of substantial deformation in gob-side entries. Wu Yongzheng and colleagues [14] introduced a collaborative approach for the prevention and control of deep rock burst roadways, which effectively mitigated the overall deformation of the surrounding rock in the roadway. This approach is based on the strong and weak structural model of rock burst roadways. Gao Mingshi et al. [15] put forward the branch-outside unloading combined rock burst prevention technology. Xie Shengrong [16] investigated the collaborative control technology involving external anchoring and internal unloading of surrounding rock within large-section coal roadways. This approach has proven effective in mitigating significant deformations of the surrounding rock on either side of the expansive roadway. In the specific anti-impact practice, the team of Academician Kang Hongpu [17] proposed a variety of surrounding rock control technologies such as hydraulic fracturing pressure relief and high toughness bolts, which significantly improved the stability of the roadway. Tan Yunliang et al. [18] constructed the collaborative control theory and technology of rock burst disaster,

which improved the coupling energy absorption level of the roadway anchoring support system and surrounding rock structure. The integrated pressure relief technology was proposed by [19] et al., which significantly reduced the risk of rock burst in gob-side entry.

In the aforementioned research, it is evident that the academic and engineering communities predominantly adhere to two distinct control concepts regarding the surrounding rock of roadways: 'pressure relief' and 'support'. 'The 'pressure relief' focuses on weakening the strength of surrounding rock and releasing the accumulated elastic energy to the greatest extent, while the 'support' focuses on improving the strength of surrounding rock on the roadway surface, thereby enhancing the impact resistance of the roadway. For rock burst mines, maintaining the overall support strength of the roadway while implementing high-intensity pressure relief is a challenging aspect of surrounding rock control and is also a focal point of attention in the industry. To address the inherent contradiction between pressure relief and support within hard coal seams, and to investigate a suitable control method for the surrounding rock in roadways specific to the Ordos region, this study employs both theoretical analysis and numerical simulation. These methodologies are utilized to systematically elucidate the mechanisms underlying the coupling of pressure relief and support, as well as the anti-scour mechanisms present in hard coal seams. Furthermore, a pressure relief technology for hard coal seams, which is predicated on mechanical reaming, is proposed. Engineering practice is carried out in Menkeqing Coal Mine, and a multi-means test of pressure relief and support effect is conducted to verify the anti-scour effect. The research results can provide a theoretical basis and engineering reference for mines with similar conditions in the Ordos area.

2 Characteristics of Borehole Pressure Relief in Hard Coal Seam

The uniaxial compressive strength serves as the most direct indicator of coal hardness. The identification reports of the burst tendency of the main coal seams of 40 coal mines in the main rock burst mining areas, such as Ludong, Luzhong, Luxi, Henan Yima, Shaanxi Binchang, and Ordos in China, were collected, and the uniaxial compressive strength index of the coal seam was counted. The statistical results are shown in Table 1 and Figure 1. Through analysis, it is found that the uniaxial compressive strength of the main coal seam in Ordos area is generally between 25 ~ 35 MPa, which is higher than that of rock burst mines in Shandong, Henan and Shaanxi, and ranks first in the main rock burst mines in China.

Table 1. Statistical table of uniaxial compressive strength of coal seams in major rockburst mining areas in China

Mining Area	Mine	Main coal seam	Uniaxial Compressive Strength / MPa
Ludong Mining Area	Longyun Coal Mine	3 Coal	10.3
	Gucheng Coal Mine	3 Coal	18.5
	Wanglou Coal Mine	3 Upper Coal	15
	Guotun Coal Mine	3 Coal	16.4
	Pengzhuang Coal Mine	3 Lower Coal	26.32
Luzhong Mining Area	Xiezhang Coal Mine	2 Coal	27.11
	Suncun Coal Mine	2 Coal	21.31
	Huafeng Coal Mine	11 Coal	9.8
	Xin Julong Coal Mine	3 Coal	17.4
	Liangbaosi Coal Mine	3 Coal	18
	Chenmanzhuang Coal Mine	3 Coal	1.56
Luxi Mining Area	Tangkou Coal Mine	3 Upper Coal	19.25

	Dongtan Coal Mine	3 Upper Coal	7.361
	Jining No. 3 Well	3 Lower Coal	25
	Xinglongzhuang Coal Mine	3 Coal	19.36
	Baodian Coal Mine	3 Lower Coal	25
	Jining No. 2 Well	3 Upper Coal	16.74
	Nantun Coal Mine	3 Lower Coal	11.16
	Zhaolou Coal Mine	3 Coal	8.55
	Yunhe Coal Mine	3 Coal	11.09
	Xingcun Coal Mine	3 Coal	10.4
	Anju Coal Mine	3 Coal	13.79
Henan Yima Mining Area	Changcun Coal Mine	2 ⁻¹ Coal	10.81
	Yuejin Coal Mine	2 ⁻¹ Coal	21.46
	Gengcun Coal Mine	2 ⁻³ Coal	16
	Qianqiu Coal Mine	2 ⁻¹ Coal	15.2
Shaanxi Binchang Mining Area	Jiangjiahe Coal Mine	4 Coal	16.61
	Tingnan Coal Mine	4 Coal	16.68
	Mengcun Coal Mine	4 Coal	19.37
	Gaojiapu Coal Mine	4 Coal	19.32
	Yadian Coal Mine	4 Coal	13.43
	Hujiahe Coal Mine	4 Coal	24.31
	Xiaozhuang Coal Mine	4 Coal	17.32
	Tangjiahe Coal Mine	8 ⁻¹ , 8 ⁻² Coal	19.55
Ordos Mining Area	Hulusu Coal Mine	2 ⁻¹ Coal	28.52
	Menkeqing Coal Mine	3 ⁻¹ Coal	30.77
	Muduchaideng Coal Mine	3 ⁻¹ Coal	35.27
	Nalinhe No. 2 Mine	3 ⁻¹ Coal	26.01
	Hongqinghe Coal Mine	3 ⁻¹ Coal	29.56
	Shilawu Su Coal Mine	2 Coal	26.26

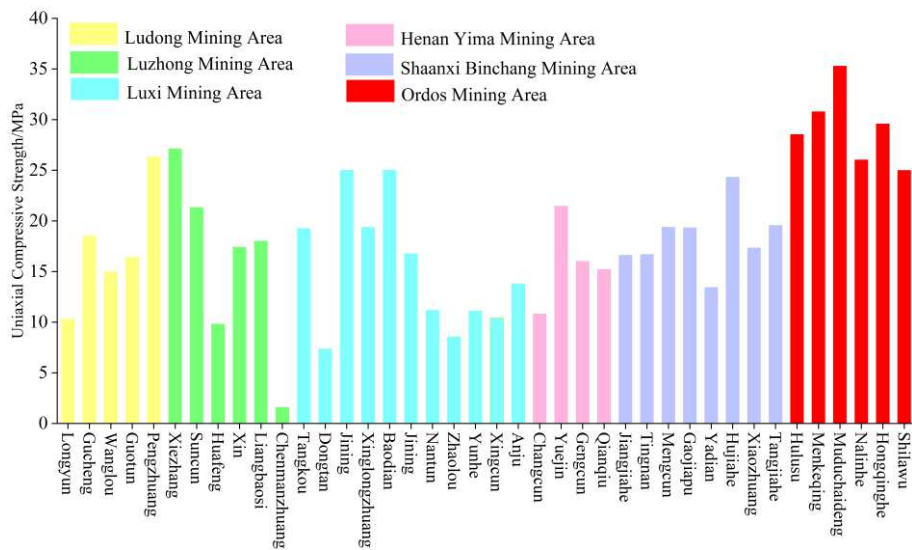


Figure 1. Statistical analysis of uniaxial compressive strength of coal seams in major rock burst mining areas in China.

Borehole pressure relief in coal seams is the most commonly used pressure relief method in the Ordos area. The area is generally constructed according to the national standard of 'Rock Burst Measurement, Monitoring and Prevention Methods (GB/T 25217.10)'. The borehole exhibits a diameter ranging from 150 to 200 mm, with a depth spanning from 15 to 25 m. In the context of a hard coal seam, the characteristics of drilling pressure relief are primarily manifested in the following ways:

(1) The drilling outside the advanced influence zone is better. Borehole collapse is rarely observed beyond 70 m ahead of the working face, and the borehole shape is complete, as shown in Fig. 2(a).

(2) The borehole is prone to shear failure after entering the advanced influence area. As shown in Figure 2(b), the boreholes near 1 m inside the orifice are transversely split, and there is a separation between the boreholes and the surface coal of the roadway.

(3) The damaged area of the roadway surface is large after a borehole collapse. The borehole often exhibits sudden deformation and failure after entering the advanced influence area, which is often accompanied by an obvious side drum. Therefore, the sudden collapse of the hole can easily damage the surface protection ability of the roadway, as shown in Fig. 2 (c).

4) Repeated pressure relief is required. Due to the large strength of the coal body and the difficulty of pressure relief, the mine often needs to carry out high-density, multi-round large-diameter drilling pressure relief work. Repeated pressure relief is likely to cause roadway support failure, as shown in Figure 2(d).

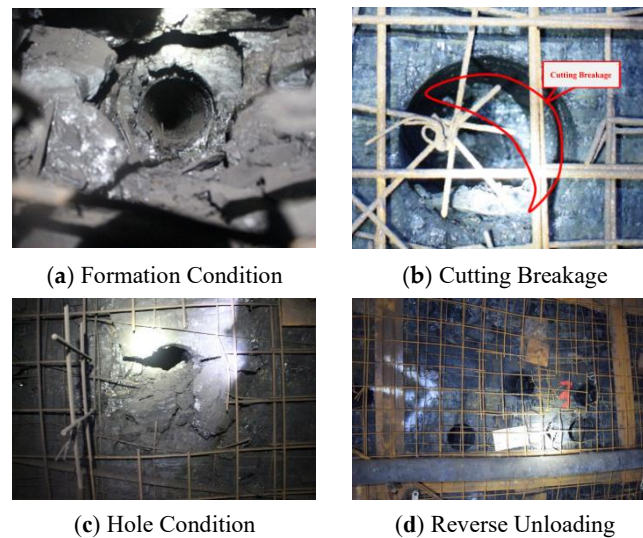


Figure 2. Characteristics of drilling pressure relief in hard coal seam

In conclusion, the analysis indicates that while pressure relief techniques for coal seam drilling in the Ordos region are extensively implemented, their effectiveness remains suboptimal. Therefore, enhancing the pressure relief outcomes of drilling in hard coal seams and reducing the detrimental impact of drilling on foundational support constitutes a significant area for further investigation.

3 Theoretical Analysis of Unloading-Support Coupling Anti-Scour Mechanism in Hard Coal Seams

3.1. Radius Analysis of the Plastic Zone Around the Borehole

In accordance with the principles of classical elastic mechanics, a drilling mechanics model has been developed for a non-uniform pressure stress field. It is posited that the coal matrix adheres

to conditions of continuity, uniformity, and isotropy, while also conforming to the Mohr-Coulomb strength criterion. The rock mass at an infinite distance around the R_0 borehole is in a two-way unequal pressure state, which is subjected to vertical stress P , horizontal stress kP , and plastic zone radius R_p , as shown in Figure 3.

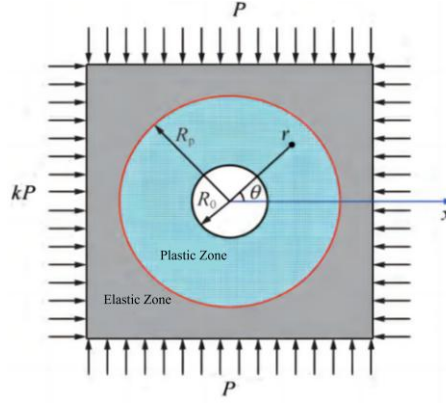


Figure 3. Mechanical model of non-isobaric stress field drilling

The expression for the stress surrounding the borehole in polar coordinates is presented in Equation (1), and the stress at any point in the model can be determined by r and θ .

$$\begin{cases} \sigma_r = \frac{1}{2}P \left[(1+k) \left(1 - \frac{R_0^2}{r^2} \right) + (1-k) \times \left(1 - 4\frac{R_0^2}{r^2} + 3\frac{R_0^4}{r^4} \right) \cos 2\theta \right] \\ \sigma_\theta = \frac{1}{2}P \left[(1+k) \left(1 + \frac{R_0^2}{r^2} \right) - (1-k) \times \left(1 + 3\frac{R_0^4}{r^4} \right) \cos 2\theta \right] \\ \tau_{r\theta} = \frac{1}{2}P \left[-(1-k) \times \left(1 + 2\frac{R_0^2}{r^2} - 3\frac{R_0^4}{r^4} \right) \sin 2\theta \right] \end{cases} \quad (1)$$

In the formula: p is the initial in-situ stress; K represents the lateral pressure coefficient associated with the specific location of the coal body; R_0 is the borehole radius; σ_r is the radial stress of coal at point (r, θ) ; σ_θ is the tangential stress of coal at point (r, θ) ; $\tau_{r\theta}$ is the shear force of coal at point (r, θ) .

Substituting $\theta=0^\circ$ into Eq. (1), the tangential stress of the elastic surrounding rock on the horizontal axis is

$$\sigma_\theta = \frac{P_0}{2} \left[2 + (k+1)\frac{R_0^2}{r^2} + 3(1-k)\frac{R_0^4}{r^4} \right] \quad (2)$$

When $k \neq 1$, the total load of elastic surrounding rock on the horizontal axis is:

$$Q^e = \lim_{L \rightarrow \infty} \int_{R_0}^L \sigma_\theta dr = PL \quad (3)$$

In the event that the borehole undergoes plastic deformation, the stress distribution will display characteristics indicative of the simultaneous presence of both elastic and plastic states. When $k \leq 1$, Figure 4 illustrates the elastic-plastic tangential stress and the elastic tangential stress of the surrounding rock in the borehole at an angle of $\theta=0^\circ$.

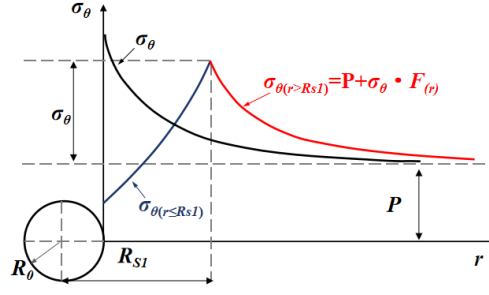


Figure 4. Stress analysis diagram of surrounding rock ($k \leq 1$)

The cumulative load exerted by the surrounding rock of the borehole along the horizontal axis can be documented as:

$$Q^P = \int_{R_0}^{R_{sI}} \sigma_{\theta(r \leq R_{sI})} dr + \lim_{L \rightarrow \infty} \int_{R_{sI}}^L \sigma_{\theta(r > R_{sI})} dr \quad (4)$$

In the formula: $\sigma_{\theta(r \leq R_{sI})}$ and $\sigma_{\theta(r > R_{sI})}$ are the tangential stress of the plastic zone and the elastic zone around the borehole on the horizontal axis, MPa.

Studies have shown that [20], Under non-isobaric conditions, the stress distribution along the horizontal axis of the plastic zone remains uniform.

$$\sigma_{\theta(r \leq R_{sI})} = M(1-A)\left(\frac{r}{R_0}\right)^{A-1} \quad (5)$$

For the stress in the elastic zone, the tangential stress $\sigma_{\theta(r \leq R_{sI})}$ of the elastic zone of the surrounding rock at $\theta=0^\circ$ still obeys formula (2), that is,

$$\sigma_{\theta(r > R_{sI})} = \frac{P}{2} \left[2 + (k+1) \frac{R_{sI}^2}{r^2} + 3(1-k) \frac{R_{sI}^4}{r^4} \right] \quad (6)$$

The stress at the elastic-plastic interface must satisfy the yield criterion, while the shear stress of the surrounding rock in the horizontal axis direction is assumed to be zero. Therefore, $r=R_{sI}$ can be substituted into the above parallel $M-C$ criterion equation to obtain:

$$\sigma_r \big|_{r=R_{sI}} = \frac{(3k-1)P-B}{A+1} \quad (7)$$

In order to describe the $\sigma_{\theta-r}$ relation of surrounding rock in the elastic zone, the formula (6) is rewritten as

$$\sigma_{\theta(r > R_{sI})} = P + \sigma_{\theta} \cdot F(r) \quad (8)$$

In the formula, $F(r)$ is a function of r ; the physical meaning of σ_{θ} is shown in Fig. 4, which is the stress difference between the stress at the elastic-plastic junction of the surrounding rock on the horizontal axis and P , MPa.

By combining Figure 4 and Formula (8), it can be seen that when $r=R_{sI}$, $F(r)=1$. Therefore, $r=R_{sI}$ is substituted into Formula (6), and $F(r)=1$ is substituted into Formula (8), and then the two formulas are solved simultaneously:

$$\sigma_{\theta} = (2-k)P - \sigma_r \big|_{r=R_{sI}} = (2-k - \frac{\sigma_r \big|_{r=R_{sI}}}{P})P \quad (9)$$

Combining Eq. (6), Eq. (7), and Eq. (8), the expression for $F(r)$ can be obtained as:

$$F(r) = \left(\frac{1+k}{2(2k-1-\frac{\sigma_r|_{r=R_s}}{P})} - \frac{\sigma_r|_{r=R_s}}{(2k-1-\frac{\sigma_r|_{r=R_s}}{P})P} \right) \frac{R_{s1}^2}{r^2} + \frac{3(1-k)}{2(2k-1-\frac{\sigma_r|_{r=R_s}}{P})} \frac{R_{s1}^4}{r^4} \quad (10)$$

Therefore, the tangential stress in the elastic zone of surrounding rock at $\theta=0^\circ$ can be expressed as

$$\sigma_{\theta(r>R_s)} = P + (\sigma_r|_{r=R_s} - P) \left[\left(\frac{1+k}{2(2k-1-\frac{\sigma_r|_{r=R_s}}{P})} - \frac{\sigma_r|_{r=R_s}}{(2k-1-\frac{\sigma_r|_{r=R_s}}{P})P} \right) \frac{R_0^2}{r^2} - \frac{3(1-k)}{2(2k-1-\frac{\sigma_r|_{r=R_s}}{P})} \frac{R_{s1}^4}{r^4} \right] \quad (11)$$

In the formula, $\sigma_{\theta}|_{r=R_s} = R_s$ is the tangential stress at the elastic-plastic interface on the vertical axis of the surrounding rock, in MPa. Its expression can also be solved according to the condition of a uniform stress field:

$$\sigma_r|_{r=R_s} = M(1-A)\left(\frac{R_s}{R_0}\right)^{A-1} \quad (12)$$

Finally, the total load of elastic-plastic surrounding rock on the vertical axis can be obtained by substituting formula (5), formula (11), and formula (12) into formula (4). According to the law that the total load is constant, the total load formula (3) of elastic surrounding rock is solved; that is, when $k \leq 1$, the approximate solution of R_{s1} is.

$$R_{s1} = R_0 \left[\frac{(2k-1-\frac{\sigma_r|_{r=R_s}}{P})(P-M) + \left(k - \frac{\sigma_r|_{r=R_s}}{P}\right)(P-M)}{-M(2k-1-\frac{\sigma_r|_{r=R_s}}{P}) - AM\left(1 - \frac{\sigma_r|_{r=R_s}}{P}\right)} \right]^{\frac{1}{k-1}} \quad (13)$$

Similarly, when plastic deformation occurs in the borehole under the condition of $k > 1$, as shown in Fig. 5, the radius of the plastic zone on the vertical axis, R_{s2} , must not be 0, and the tangential stress of the surrounding rock on the vertical axis will have elastic and plastic stress states. R_{s2} can be derived by using the law of constant total load on the vertical axis and the stress formula of the plastic zone in a uniform stress field.

$$R_{s2} = R_0 \left[\frac{(2k-1-\frac{\sigma_r|_{r=R_s}}{P})(kP-M) + \left(k - \frac{\sigma_r|_{r=R_s}}{P}\right)(kP-M)}{-M(2k-1-\frac{\sigma_r|_{r=R_s}}{P}) - AM\left(1 - \frac{\sigma_r|_{r=R_s}}{P}\right)} \right]^{\frac{1}{k-1}} \quad (14)$$

$$\begin{aligned}
f(r, \theta) = & \left[P(k+1) \frac{R_0^2}{r^2} - P(k-1) \cos 2\theta \left(1 + 3 \frac{R_0^4}{r^4} - 2 \frac{R_0^2}{r^2} \right) \right]^2 \\
& + \left[P(k-1) \sin 2\theta \left(1 + 3 \frac{R_0^4}{r^4} + 2 \frac{R_0^2}{r^2} \right) \right]^2 \\
& - \frac{(1 - \cos 2\phi)}{2} \left\{ \left[P(k+1) \frac{R_0^2}{r^2} - 2 \cos 2\theta \cdot P(k-1) \frac{R_0^2}{r^2} \right]^2 - 4c \right\} \\
& - 2c \sin 2\phi \left[P(k+1) - 2 \cos 2\theta \cdot P(k-1) \frac{R_0^2}{r^2} \right] - 4c^2
\end{aligned} \tag{16}$$

The boundary equation that delineates the transition between the elastic zone and the plastic zone of the surrounding rock near the borehole can be derived when $f(r, \theta)$ equals zero. From Eq. (16), it is known that the main factors affecting the plastic zone are the lateral pressure coefficient, coal strength, and borehole diameter. This study examines the influence of coal strength and borehole diameter on the extent of the plastic zone for a specific roadway, assuming that the lateral pressure coefficient remains constant.

3.2.1 Influence of Coal Strength

Through the test of mechanical parameters such as uniaxial compression and splitting tests of coal specimens with different strengths, the mechanical parameters such as cohesion, internal friction angle, compressive strength, and shear strength of six different coal samples are obtained, see Table 2. Assuming that the six types of coal are subjected to identical geological conditions, the radius of the plastic zone surrounding the borehole is calculated using Equation (16). Additionally, the relationship between the strength characteristics of coal and the dimensions of the plastic zone is established, as shown in Fig.7.

Table 2. Summary of mechanical parameters of different coal bodies

Serial Number	Cohesion (MPa)	Internal Friction Angle (°)	Compressive Strength (MPa)	Shear Strength (MPa)
C1	0.2	30.6	9.36	5.72
C2	0.56	32.4	13.69	9.25
C3	1.24	32.6	18.66	11.93
C4	1.88	27.2	24.56	15.60
C5	2.0	25.1	32.68	18.03
C6	3.21	28.6	35.12	23.85

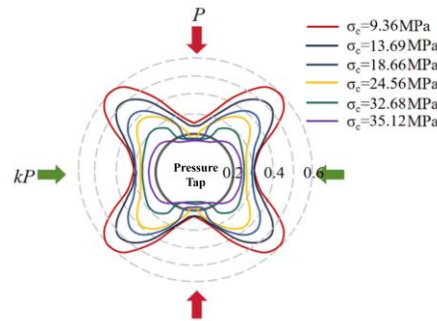


Figure 7. Relationship between uniaxial compressive strength and plastic zone.

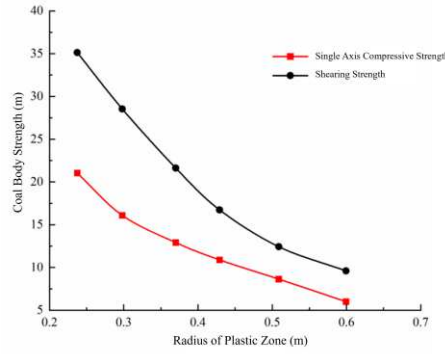


Figure 8. Relationship between coal strength and plastic zone radius

Figure 8 illustrates a negative correlation between the radius of the borehole plastic zone and both the uniaxial compressive strength and shear strength of coal and rock masses. This relationship indicates that as the compressive and shear strengths of these materials increase, the radius of the borehole plastic zone decreases, signifying an enhanced capacity of the surrounding rock to resist failure. Furthermore, it is noteworthy that once the compressive strength of coal exceeds 13.69 MPa, the slope of the curve shows a significant reduction. This observation suggests that the plastic zone radius in hard coal is relatively small, which correlates with increased stability of the roadway. Additionally, it implies that the energy accumulated during failure events is greater, potentially leading to more intense rock bursts.

3.2.2 The Influence of Borehole Diameter

In a similar manner, by establishing the parameters $P = 16$ MPa, $k = 0.4$, $c = 2$ MPa, and $\varphi = 25^\circ$, the correlation between the borehole diameter (R_0) and the plastic zone of the adjacent rock is determined by varying the borehole diameter. This relationship is illustrated in Figure 9.

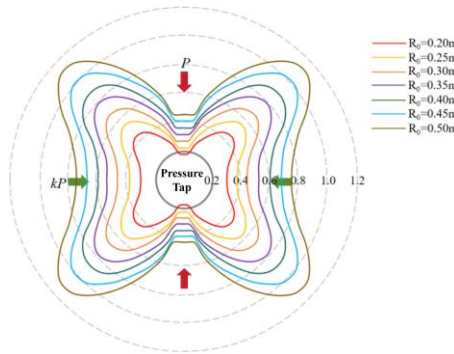


Figure 9. The correlation between the diameter of the pressure relief aperture and the extent of the plastic zone.

As illustrated in Figure 9, an increase in borehole diameter corresponds to a gradual expansion of the plastic zone radius; however, the morphological characteristics of the plastic zone remain consistent. The radial stress curves of boreholes with $R_0 = 0.2$ m, 0.3 m, and 0.4 m were extracted and analyzed, as shown in Fig. 10.

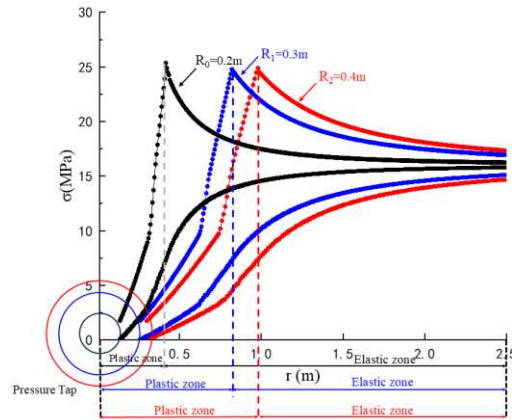


Figure 10. The relationship between the diameter of pressure relief hole and plastic zone

As illustrated in Figure 10, increasing the borehole diameter from 0.2 meters to 0.3 meters results in an approximate increase of 0.43 meters in the radius of the plastic zone. Conversely, when the borehole diameter is increased from 0.3 meters to 0.4 meters, the radius of the plastic zone experiences a smaller increase of only 0.26 meters. This observation suggests that the effectiveness of pressure relief associated with borehole expansion diminishes as the diameter increases. Consequently, it can be inferred that merely enlarging the borehole diameter is insufficient to indefinitely enhance the pressure relief effect.

Based on the above analysis, it can be seen that with the increase in coal strength, the self-stress of the coal body is less obvious for the cracking effect of the borehole; the more difficult it is for the borehole to be destroyed, the worse the pressure relief effect becomes. The most effective method for improving this situation is to increase the diameter of the borehole. However, it is important to note that simply enlarging the borehole diameter does not infinitely enhance the pressure relief effect. In fact, such an increase may have adverse consequences for the bolt and cable support systems within the roadway, thereby impeding the effective management of the surrounding rock formations.

3.3 Unloading-Supporting Coupling Anti-Impact Method for Hard Coal Seam

To ensure the stability of the surrounding rock within the roadway, it is essential to dissipate the elastic energy accumulated in the coal mass while also preserving the load-bearing capacity of the support structure. This necessitates establishing a balance between pressure relief and support mechanisms. The advancement of technologies related to the coupling of unloading support for hard coal seams, which incorporates reaming pressure relief and anti-scour techniques, can be summarized in the support and deep pressure relief anti-scour methodology, as illustrated in Figure 11.

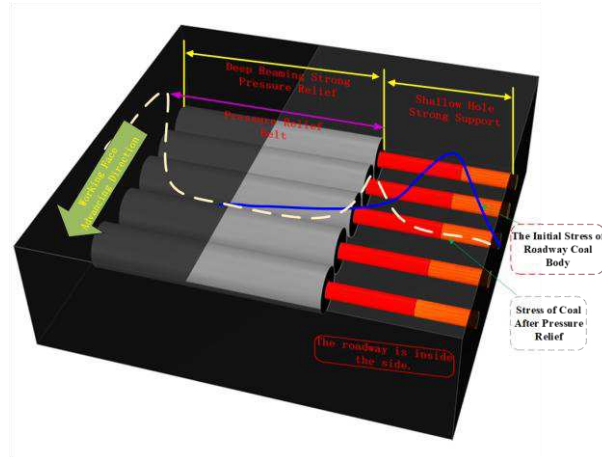


Figure 11. Coupled unloading and support technology of expansion borehole pressure relief in hard coal seams

(1) Shallow support is defined as a circular borehole that encompasses the entire roadway, with the surrounding coal body divided into three distinct zones: a fracture zone, a plastic softening zone, and an elastic zone. The primary focus of the support system is on the fracture zone and the plastic softening zone, while the peak stress region is not emphasized. Additionally, the drilling diameter and density are not prioritized. As a result, only conventional large-diameter drilling techniques are utilized to maintain the integrity of the shallow support structure of the roadway.

(2) Deep pressure relief involves modifying the 'elastic core' region of the coal body that is subjected to peak stress. This is achieved by enlarging the borehole diameter through reaming and pressure relief techniques, which facilitates the transition of a larger section of the coal body from the elastic zone to the plastic zone. This process results in a significant release of elastic energy, effectively reducing the total elastic energy stored within the 'elastic core' area of the hard coal body. The ultimate objective of this intervention is to achieve pressure relief and mitigate the risk of erosion.

The implementation of shallow hole support, deep hole expansion, and pressure relief techniques effectively reduces the level of stress concentration at the initial stress peak. This process also facilitates the relocation of the stress peak to deeper regions of the solid coal, thereby achieving the objectives of cutting and pressure redistribution. Concurrently, by maintaining the support strength of the surrounding rock, the deformation of the roadway's surrounding rock is minimized, which enhances the overall impact resistance of the roadway.

4. Numerical Simulation of Unloading-Support Coupling Anti-Scour in Hard Coal Seam

4.1 Releasing-Supporting Coupling Anti-Impact Mechanism of Reaming Pressure Relief

To elucidate the unloading-support coupling anti-scour mechanism associated with hard coal seams and to assess the impact of critical parameters such as borehole diameter, reaming diameter, and reaming point position on anti-scour efficacy, this study utilizes the engineering geological conditions of the Menkeqing 3106 working face as its research context. Layered modeling is conducted in accordance with the physical and mechanical properties of various rock strata, and FLAC^{3D} is employed to simulate the changes in the stress field and displacement field of the surrounding rock in the roadway during the pressure relief process. A three-dimensional model that reflects the actual working conditions is presented in Figure 12.

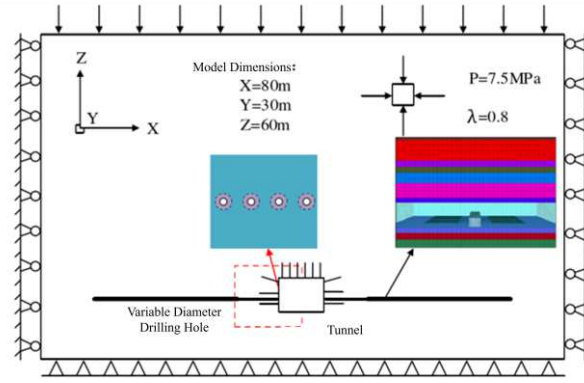


Figure 12. Numerical model structure

The dimensions of the model are defined as follows: length (Y) $\times$ width (X) $\times$ height (Z) = 45 m $\times$ 8 m $\times$ 15 m. A borehole is drilled in the positive direction of the Y-axis, utilizing the Mohr-Coulomb plastic strength criterion for analysis. A brick element is employed to represent both the coal seam and the surrounding rock. Horizontal movement is constrained on both lateral sides of the model, while vertical movement is restricted at the base. A vertical stress of 17 MPa is applied to the upper section, corresponding to the burial depth of the coal seam, and a lateral pressure coefficient of 0.8 is implemented based on the results of ground stress testing. The physical and mechanical properties of the surrounding rock, determined through laboratory analysis, inform the parameters for the coal and rock mass. The specific physical and mechanical parameters utilized in the simulation calculations and analyses are detailed in Table 3.

Following the initial calibration of the model, simulations were conducted to analyze the stress and deformation characteristics of the surrounding rock during roadway excavation, conventional drilling, and unloading support coupling reaming excavation. The dimensions of the roadway were established at 5 m $\times$ 5 m. After the excavation process, a cable bolt unit was incorporated into the roadway, characterized by a bolt diameter of $\Phi 22$ mm and a length of 2500 mm, with full anchorage. The arrangement consisted of a row spacing of 1 m $\times$ 1 m, with 12 bolt units allocated per row, and an initial anchoring force set at 60 kN. Following the installation of the bolts, simulations were performed for both conventional borehole excavation and unloading-branch coupling reaming excavation. The conventional borehole was designed with a diameter of 150 mm, while the reaming borehole had a diameter of 350 mm, positioned 6 m from the coal wall, with a borehole spacing of 1 m. The vertical stress distribution and the expansion of the plastic zone within the coal seam are illustrated in Figure 13.

Table 3. values for physical and mechanical parameters of coal rock.

Sequence Number	Rock Layer	Layer Thickness (m)	Density (kg $\cdot$ m ³)	Uniaxial Compressive Strength (MPa)	Cutting Modulus (GPa)	Volume Modulus (GPa)	Cohesion (MPa)	Internal Friction Angle(°)
1	Sandstone	5	2450	2.4	7.3	7.5	7.2	24
2	3-1 Coal	5	1400	0.6	1.5	2.3	3.75	35
3	Sandstone	5	2200	2.3	5.2	5.6	4.3	21

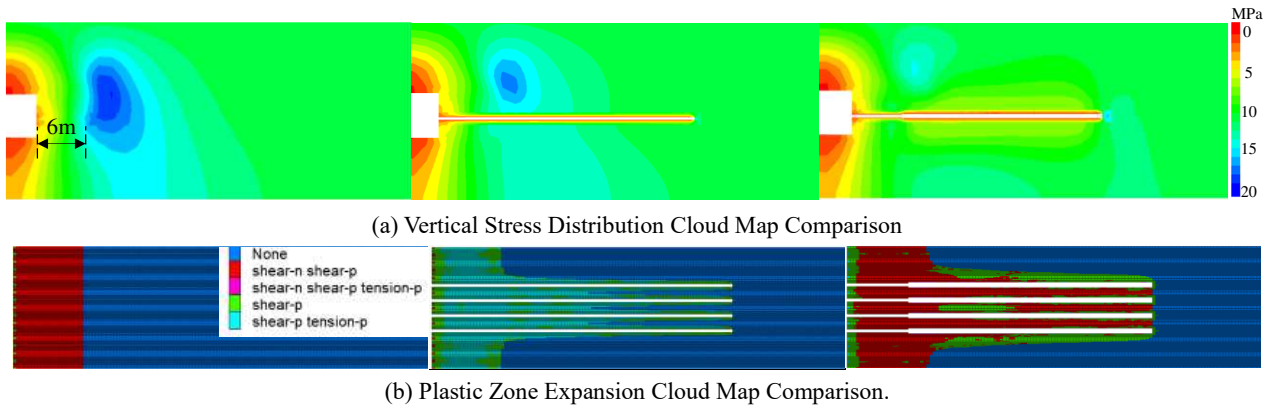
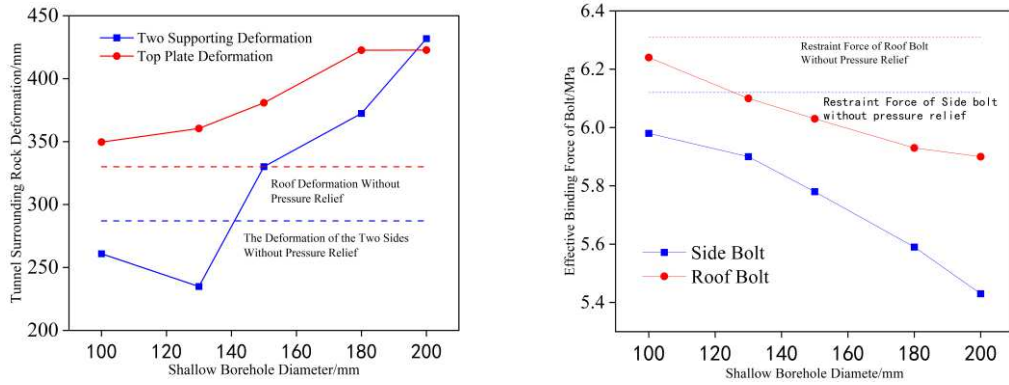


Figure 13. Comparison of vertical stress distribution and plastic zone cloud maps in coal seams under different drilling methods

It can be seen from Fig. 13 (a) that after the excavation of the roadway, the 'elastic core' area is generated due to the stress concentration at the position of 6 m in the coal wall. After the conventional drilling pressure relief, the area of the 'elastic core' is reduced, but the 'elastic core' area between the borehole and the roof still exists, and the degree of stress concentration is not significantly reduced. After the excavation of the unloading-support coupling reaming, the area of the 'elastic core' is reduced to nearly disappear, and the degree of stress concentration is significantly reduced. From Fig. 13 (b), it can be seen that the plastic zone between the boreholes is obviously expanded after the pressure relief of the unloading-support coupling reaming, which indicates that the pressure relief effect of the borehole is enhanced, while the plastic zone of the roadway side is about 1 m. Compared with the conventional boreholes, there is no obvious change, and the coal wall still has a certain support strength, indicating that the unloading-support coupling pressure relief has a relatively small disturbance to the roadway surface support structure.

4.2 Influence of Borehole Diameter on the Effectiveness of Unloading-Support Anti-Scour

The findings from the numerical simulations indicate that the unloading-support coupling reaming pressure relief method demonstrates a significant relationship between the diameter of shallow boreholes and the effectiveness of roadway support, while the diameter of deep boreholes primarily influences coal pressure relief. To quantitatively assess the impact of both deep and shallow borehole diameters on roadway support and pressure relief, two sets of numerical simulations were conducted using the control variable method. In the first scenario, the diameter of the deep borehole was maintained at 350 mm, while the diameter of the shallow borehole was varied from 100 mm to 200 mm. In the second scenario, the diameter of the shallow borehole was fixed at 150 mm, and the diameter of the deep borehole was adjusted from 200 mm to 400 mm. The resulting curves depicting the deformation of the surrounding rock and the effective binding force of the bolt were extracted for the varying shallow borehole diameters, as illustrated in Figure 14. Additionally, the curves representing vertical stress and peak stress of the coal body corresponding to the changes in deep borehole diameter were extracted, as shown in Figure 15.

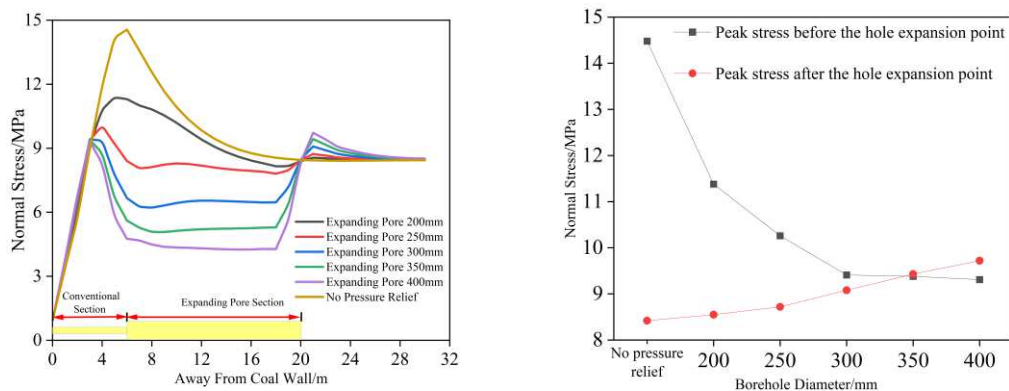


(a) Deformation curve of surrounding rock

(b) Effective binding force curve of bolt

Figure 14. The influence of shallow drilling diameter on support effectiveness

The analysis of Figure 14(a) reveals that as the diameter of the shallow borehole increases, there is a corresponding gradual increase in the deformation of both the roof and floor of the roadway. Conversely, the deformation of the lateral sides initially decreases before subsequently increasing. Specifically, when the borehole diameter ranges from 100 to 130 mm, the deformation of the roadway's lateral sides is slightly lower compared to scenarios without pressure relief. This phenomenon can be attributed to the pressure relief effect of the borehole, which alleviates stress concentration within the coal body and subsequently reduces the deformation of the surrounding rock in the roadway. However, once the diameter of the shallow borehole exceeds 130 mm, the deformation of the lateral sides escalates rapidly. For instance, at a borehole diameter of 200 mm, the deformation of the lateral sides can reach 429.3 mm, representing an increase of 49.3% compared to conditions without pressure relief. Furthermore, the analysis presented in Figure 14(b) reveals that as the diameter of the shallow borehole increases, the effective binding force between the lateral sides and the roof bolt progressively diminishes, with the rate of decline accelerating in tandem with the borehole diameter. This analysis underscores that once the diameter of the shallow borehole exceeds 130 mm, larger diameters exacerbate damage to the roadway support structure, thereby complicating the control of roadway deformation.



(a) vertical stress change curve

(b) stress peak change curve

Figure 15. The influence of deep drilling diameter on pressure relief effect

The analysis of Figure 15(a) indicates that as the diameter of the deep borehole increases, the vertical stress distribution within the coal body maintains a consistent pattern. Specifically, the peak stress observed on the shallow side of the roadway exhibits a gradual decline, while the peak stress on the deep side shows a corresponding increase. As illustrated in Figure 15(b), increasing the

borehole diameter from 200 mm to 400 mm results in a reduction of the peak stress in the shallow region from 14.48 MPa to 9.31 MPa, representing a decrease of 5.17 MPa. Conversely, the peak stress in the deep region rises from 8.42 MPa to 9.72 MPa, reflecting an increase of only 1.3 MPa. This analysis suggests that, in terms of differential peak stress, the benefits of borehole pressure relief become more pronounced with the enlargement of the deep borehole diameter.

5. Unloading-Supporting Coupling Anti-Impact Engineering Practice of Hard Coal Seam

5.1 Engineering Test Situation

The 3106-working face of the Menkeqing Coal Mine is characterized as a side-free face. The primary coal seam, designated as 3-1, exhibits a significant tendency for impact, with a uniaxial compressive strength of 33.54 MPa. During the excavation and mining operations of the 3106-return airway, numerous substantial energy events exceeding 105 J have been recorded, along with frequent occurrences of coal gun phenomena. Notably, there is pronounced deformation of the roadway side drum, which increases the risk of impact. Conventional large-diameter boreholes have proven ineffective in achieving adequate pressure relief in this context. To enhance the pressure relief efficacy of the coal body, an industrial test involving unloading-support coupling anti-scour techniques was conducted at the 3106 working face.

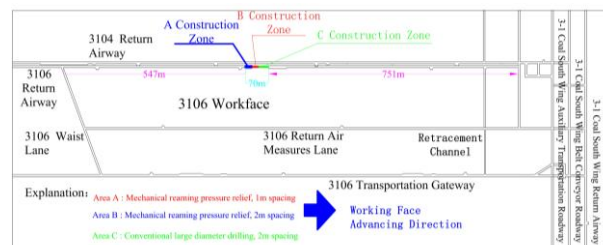
As shown in Figure 16 (a), the test site is selected at a distance of 579 m to 649 m away from the retracement channel of the 3106-return airway, and the mining position of the advanced working face is 700 m, so as to avoid the influence of the advanced abutment pressure. During the excavation in this area, no borehole pressure relief was carried out on both sides of the roadway, the support was not damaged, and the integrity of the coal wall was good. A total of three different pressure relief parameters of ABC were set up for comparative tests, as follows:

A area: construction range 20 m, mechanical reaming pressure relief, drilling spacing 1 m, hole depth 20 m, using segmented reaming, 6 m at the beginning of reaming, each reaming 3 m, leaving 1 m, shallow drilling diameter 150 mm, deep drilling diameter 350 mm.

Area B: construction range of 20 m, mechanical reaming and pressure relief, drilling spacing of 2 m, the remaining parameters unchanged.

C area: construction range is 30 m, conventional large diameter drilling pressure relief, drilling spacing is 2 m, drilling depth is 20 m, drilling diameter is 150 mm.

The mechanical reaming tool is used in the field. The tool adopts hydraulic control. It automatically opens under high-pressure hydraulic action, closes under low pressure, and retreats during rotary cutting reaming, as illustrated in Fig. 16b. The location and field photographs of the reaming experiment are presented in Figures 16a and 16c.



(a) Engineering Test Site



(b) Mechanical Reaming Tool

(c) Reaming Pressure Relief Site

Figure 16. Engineering testing location and on-site situation

5.2 Inspection of Pressure Relief Effect of Reaming

In order to test the effect of the mechanical reaming method on the release of elastic energy accumulated in the coal body, the amount of drilling powder produced per meter during drilling in each borehole in the three regions of A, B, and C, as well as the number of 'coal gun' events produced per meter, were counted respectively, as shown in Figure 17 and Figure 18.

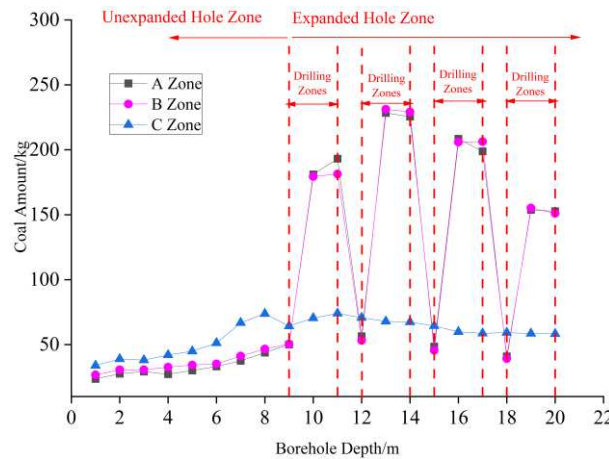


Figure 17. Statistics of average drilling powder.

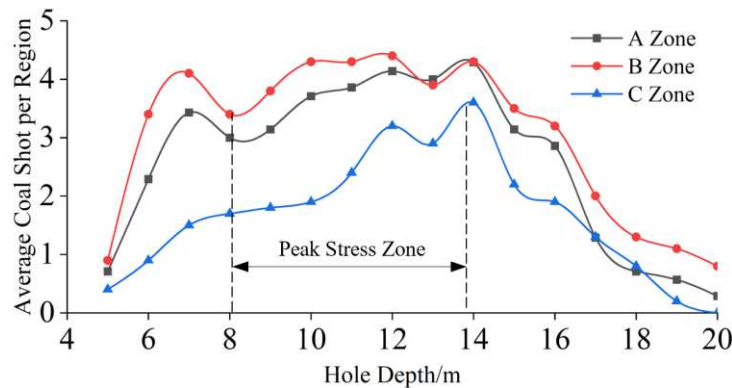


Figure 18. Statistics on the average number of "coal cannons" produced.

As illustrated in Figure 17, the quantity of pulverized coal generated in the reaming section (6-20 m) significantly exceeds that observed in conventional drilling, reaching a level of 225 kg/m, which is 3.1 times greater than the output from traditional methods. Furthermore, Figure 18 indicates that the 7-14 m segment of the coal wall represents the peak stress region within the coal body. In the context of conventional drilling for pressure relief, only the 12-14 m section experiences an increase in 'coal gun' events. Conversely, when mechanical reaming is employed for pressure relief, the entire stress concentration area exhibits an average of approximately four 'coal gun' events. This

finding suggests that mechanical reaming effectively alleviates the elastic energy accumulated within the coal body, thereby enhancing the efficacy of pressure relief in hard coal.

In order to test the pressure relief effect of the mechanical reaming method on the static load in the coal body, three sets of 7-order deep and shallow hole stress gauges were constructed on the front side of the 3106 return airway. The depths of the stress gauges were 2m, 4m, 6m, 8m, 10m, 12m, and 14m, and a total of 21 stress gauges were installed. Three groups of second-order deep and shallow holes were constructed on the auxiliary side, and the depths of the stress gauges were 2m and 3m, respectively. A total of 6 stress gauges were installed, as shown in Figure 19.

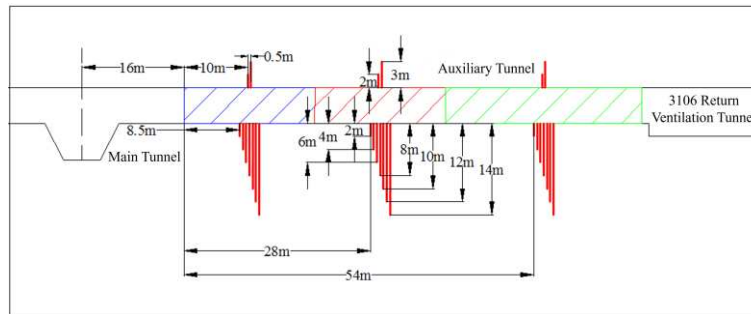
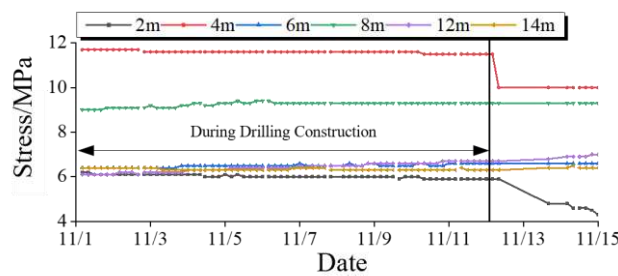
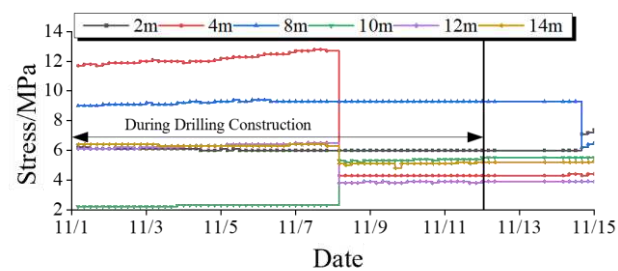


Figure 19. Monitoring position of coal seam stress.

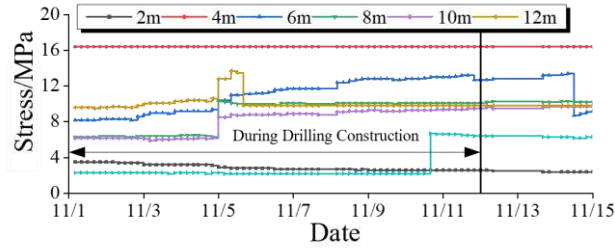
Figure 20 is the monitoring curve of the coal stress meter in the three areas of ABC before and after the drilling construction. The analysis shows that when the mechanical reaming pressure relief is carried out in the A and B areas, the stress level of the coal body is stable as a whole, and there is no stress fluctuation or sudden increase (the roof of the B area exploded on November 8, and some stress meters have a sharp rise and fall). During the pressure relief of the conventional large-diameter drilling in the C area, the 6 m stress meter in the coal wall showed a gradual upward trend, and the 8 m, 10 m, and 12 m stress meters all showed irregular sudden increases, with the highest increases of 3.8 MPa, 3.1 MPa, and 3.9 MPa, respectively, indicating that the conventional large-diameter drilling after pressure relief results in high stress in the peak area and a fast stress recovery speed. The pressure relief effect of reaming pressure relief on the concentrated static load area of the coal seam is obvious.



(a) The Stress Monitoring Curve of the Coal Body at Varying Depths in Area A



(b) The Stress Monitoring Curve of the Coal Body at Varying Depths in Area B



(c) The Stress Monitoring Curve of the Coal Body at Varying Depths in Area C

Figure 20. Stress monitoring curves of coal seams in different regions and depths.

5.3 Surrounding Rock Control Effect Test

To evaluate the impact of various pressure relief methods on roadway support damage and the deformation of surrounding rock, five newly constructed anchor cables, each measuring $\Phi 21.8 \times 4300$ mm, were installed in the roof and both sides of three designated areas (A, B, and C). The pre-tensioning force applied to each anchor cable was maintained at a minimum of 150 kN. Following the completion of the anchor cable installation, a self-storage anchor cable stress meter was affixed to the end of each cable. This device is programmed to store data for a period exceeding two weeks, with a monitoring frequency set at every 10 minutes. Concurrently, the 'cross point method' was employed to assess the deformation of the surrounding rock within the test area. A total of nine displacement monitoring stations were established along the roadway surface in the 3106-return airway. The displacements of the roadway's sides, roof, and floor were recorded both prior to and following the construction of the pressure relief borehole. The configuration of the measurement points is illustrated in Figure 21.

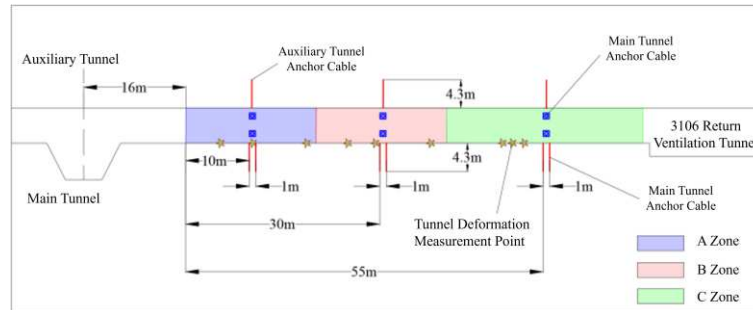
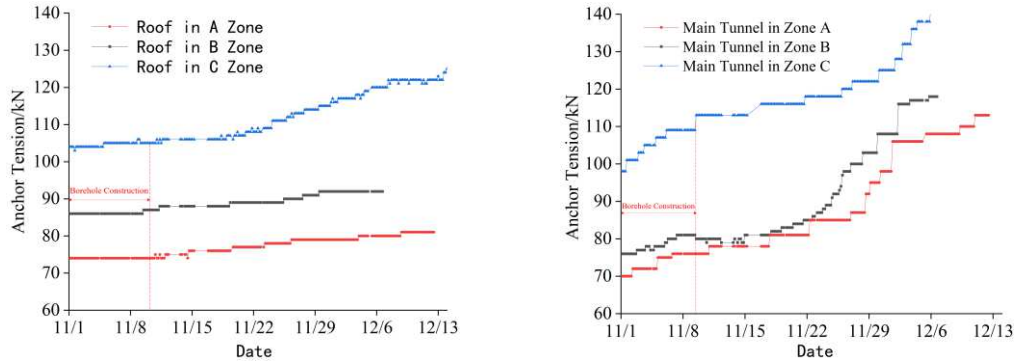


Figure 21. Schematic of monitoring plan.

Figure 22 (a) and 22 (b) are the monitoring curves of the anchor cable stress gauges on the main side and the roof in the three regions, respectively. It can be seen from the diagram: (1) During the drilling construction, the roof anchor cable in the mechanical reaming area is evenly stressed, and there is no sudden rise and fall of the force; (2) After the completion of drilling construction, the force of the roof anchor cable is C area > B area > A area. The absolute value and increase of anchor cable tension in the conventional drilling area are greater than those in the mechanical reaming area, which indicates that mechanical reaming will not have a significant impact on the stability of roof support. (3) During the drilling construction, the force of the anchor cable in the conventional drilling area and the mechanical reaming area increased step by step. The increase of the tension in the conventional drilling area can reach 16 kN, which is about twice the increase of the tension in the mechanical reaming area. Considering that the supporting area is 0 to 4.3 m on the side of the roadway, this shows that mechanical reaming is helpful in reducing the stress of the supporting structure on the side of the roadway, which is conducive to maintaining the stability of

the roadway surface support. (4) After the completion of the drilling construction, the force of the anchor cable in the conventional drilling area and the mechanical reaming area is still rising step by step. However, within one month after the completion of the construction, the force of the anchor cable on the side still shows the trend of C area > B area > A area, which indicates that mechanical reaming has a positive effect on improving the force of the supporting structure of the roadway.

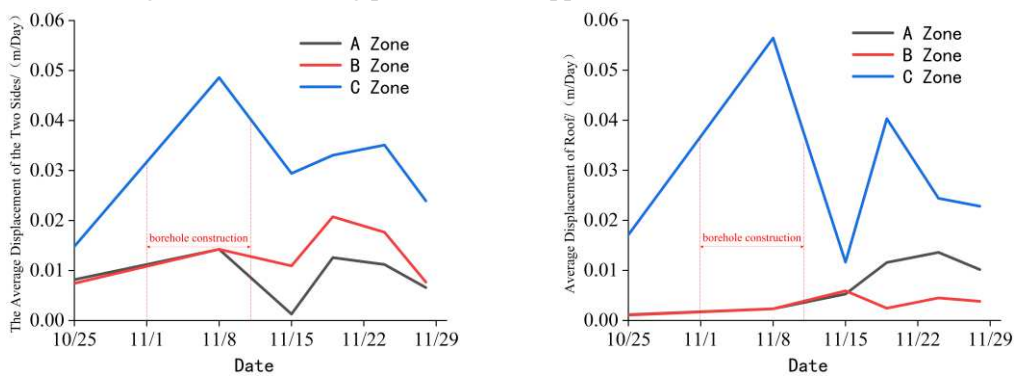


(a) Stress monitoring of anchor cable on the front side of roof

(b) Force monitoring of positive side anchor cable

Figure 22. Anchor force monitoring curve

Figures 23 (a) and 23 (b) present the monitoring curves depicting the average displacements of the two sides, as well as the average displacements of the roof and floor, calculated as the mean values from three sets of measurement points within each region. The data illustrated in these figures indicate that during the drilling operations, there is an observable increase in the displacements of both the roof and the lateral sides of the roadway. Notably, the deformation observed on the lateral sides within the conventional drilling construction area is significantly greater, reaching a maximum of 51 mm/day. In contrast, the lateral displacements in the mechanical reaming pressure relief area can be reduced by as much as 63%, while the displacements of the roof and floor can be diminished by up to 51%. These findings suggest that the reaming pressure relief method exhibits a substantially superior control effect on the deformation of the surrounding rock in the roadway compared to the conventional large diameter drilling pressure relief approach.



(a) The average displacement monitoring of the two sides

(b) Surrounding rock deformation monitoring curve of roadway

Figure 23. Monitoring curve for deformation of tunnel surrounding rock

6. Conclusions

(1) A non-isostatic stress field model for coal seam drilling has been developed, considering the specific conditions of the coal seams in the Ordos region. A mechanical analysis was conducted to evaluate the radius of the plastic zone surrounding the drill. The findings indicate that the radius

of the plastic zone is primarily influenced by the lateral pressure coefficient, the strength of the coal seam, and the diameter of the drill. Notably, an increase in the strength of the coal seam is associated with a reduction in the radius of the plastic zone. Conversely, an increase in the diameter of the drill leads to an expansion of the plastic zone radius; however, the rate of this expansion decreases over time. This suggests that merely increasing the drill diameter may not significantly enhance the pressure relief effect beyond a certain threshold.

(2) In light of the inadequate pressure relief effectiveness of conventional boreholes in hard coal seams in the Ordos area, as well as the challenges posed by high density and multiple instances of pressure relief failure, a comprehensive set of anti-impact technologies and equipment was developed. This innovation is grounded in the principles of support, deep pressure relief, and balance of relief support. deep large hole is utilized to release the elastic energy accumulated within the coal body, while a shallow small hole is employed to safeguard the bearing capacity of the support structure, thereby achieving a balance between pressure relief and support.

(3) A comparative industrial assessment of a 70-meter unloading-support coupling anti-scour control system was conducted at the 3106 working face of Menkeqing Coal Mine, focusing on the independent evaluation of pressure relief and support mechanisms. The findings indicate that:

1. The amount of pulverized coal produced by the reaming section is 3.1 times higher than that of conventional drilling, and the 'coal gun' events in the 7-14 m section are concentrated, which greatly releases the elastic energy accumulated inside the coal body.

2. Compared with the conventional borehole pressure relief peak area, the stress is lower, the recovery speed is slower, and the pressure relief effect is obvious in the concentrated static load area of the coal seam.

3. During the pressure relief period of reaming, the stress of the roof and main anchor cable is stable, and there is no loss of axial force and no stress on the anchor cable. After the construction is completed, the stress on the anchor cable is less than that of conventional drilling.

4. In the mechanical reaming pressure relief area, the displacement of the two sides of the roadway is reduced by up to 63%, and the displacement of the roof and floor is reduced by up to 51%. The control effect on the deformation of the surrounding rock of the roadway is much better than that of conventional large-diameter drilling. The research results have strong engineering reference significance for mines with similar conditions in the Ordos area.

Data availability

The datasets used and/or analysed during the current study available from the corresponding author on reasonable request.

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