The impact of drilling and slitting construction on damage field: evolution characteristics in low-permeability coal seam

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ABSTRACT

The fracture field of coal and rock mass is the main channel for gas migration and accumulation. Exploring the evolution law of fracture field of coal and rock mass under the condition of drilling and slitting construction has important theoretical significance for guiding efficient gas drainage. The generation and evolution process of coal and rock fissures is also the development and accumulation process of its damage. Therefore, based on damage mechanics and finite element theory, the mathematical model is established. The damage variable of coal mass is defined by effective strain, the elastoplastic damage constitutive equation is established and the secondary development of finite element program is completed by FORTRAN language. Using this program, the numerical simulation of drilling and slitting construction of the 15-14120 mining face of Pingdingshan No. 8 Mine is carried out, and the effects of different single borehole diameters, different kerf widths and different kerf heights on the distribution area of surrounding coal fracture field and the degree of damage are studied quantitatively. These provide a theoretical basis for the reasonable determination of the slitting and drilling arrangement parameters at the engineering site.

KEYWORDS: damage, fracture field, drilling and slitting, finite element analysis

1. INTRODUCTION

China is one of the countries with the most serious coal and gas outburst disasters in the world. Forty-nine percent of the coal seams in state-owned coal mines belong to high-gas and -outburst coal seams, and high-blast mines account for 44% of the total number of mines in the country; gas disasters have become a “bottleneck” problem that seriously restricts coal mine safety production. In addition, gas is a non-renewable and efficient clean energy source; the efficient extraction of gas is an important way to achieve efficient mining of coal resources, green mining and energy security in China. However, China’s coal seams have the characteristics of high gas, low permeability and strong adsorption [1]. Gas drainage is very difficult, especially for a single high-gas and low-permeability coal seam, which has become a world problem [2]. The research shows that increasing the permeability of coal seam is one of the most economical and effective methods for gas extraction [3,4]. Therefore, it is particularly important to choose the right kind of coal seam pressure relief and permeability enhancement technology.

After long-term engineering practice, it is known that the stress concentration and redistribution of coal seam will be caused by the drilling and slitting of coal seam, so that the coal mass around the borehole and the kerf will be damaged and broken to form a crack, and the spatial distribution characteristics of the fracture have different shapes with the change of the drilling and slitting arrangement parameters. It can be seen that the drilling and slitting technology can well relieve pressure and increase the gas permeability of the coal seam, but the degree of pressure relief depends on whether the drilling and slitting parameters are reasonably arranged. At present, many scholars have studied the drilling and slitting technology. For example, Evans [5] presented a theory on the mechanics of coal cutting in 1962. Zhao et al. [6] analyzed the influence of hydraulic slitting and drilling technology on gas drainage, and established a multi-field coupling model to simulate the gas drainage process. Zou et al. [7,8] tested the selected seven coal samples and established a co-production model of coalbed methane through drilling and cutting integration technology. The changes of gas content, desorption performance and coal strength of coal seam after gas drainage were studied. Zhang et al. [9] proposed a slitting technique using self-oscillating pulsed water jets to improve the permeability and desorption rate of coal seams and shorten construction time. Liu et al. [10] analyzed the characteristics of self-excited oscillating water jet grooving technology (SOWJ), the effect of kerf and the permeability change of coal, indicating that SOWJ can significantly improve coalbed methane mining. Sharma et al. [11] optimized high-speed water jet cutting technology. Zou et al. [12] studied the coal samples with various slot inclination angles and borehole–slot ratios under the conditions of low-permeability coal seams to investigate the weakening effect of slot inclination angle and borehole–slot ratio on the mechanical property of the pre-cracked coal in recent research, respectively, which verifies the fact that the weakening effect on mechanical property of the slotted coal samples with small slot
inclination angles is more significant. In addition to coal seam construction, Xue et al. [13] found that increasing fracture permeability can increase the recovery rate of oil and gas reservoirs, which in turn can predict and analyze the production performance of hydrogen and carbon reservoirs. Liu et al. [14] studied the use of hydraulic fracturing and formation heat treatment in the process of shale gas recovery to enhance the permeability of shale gas reservoirs.

The above research and evaluation on the effect of pressure relief and permeability enhancement of drilling and slitting construction are based on experimental and numerical simulation methods. Although the experimental research can simulate the more realistic engineering construction background, and obtain more intuitive and reliable gas emission data, it lacks effective theoretical guidance for the mechanism of pressure relief and permeability enhancement of borehole and kerf. In addition, the numerical simulation methods are based on the traditional study of the stress and displacement fields of coal and rock mass around the borehole and kerf to evaluate the effectiveness of pressure relief and permeability enhancement. However, the spatial distribution characteristics of the fissure field and degree of rupture of the surrounding coal mass are not clear, and it is difficult to give a scientific and quantitative evaluation of the mechanism and effect of permeability enhancement. It is not possible to effectively guide the reasonable location of drilling and slitting construction and related parameters.

In order to effectively guide coal mine gas drainage, it is necessary to accurately and precisely describe the distribution, shape, rupture degree and evolution law of coal and rock fracture zones around boreholes and slits. In this paper, the damage variable of coal and rock mass is defined according to the effective stress calculation model, the elastoplastic damage constitutive equation is established and the secondary development of finite element program is completed by FORTRAN language. Using this program, the numerical simulation of drilling and slitting construction of the 15-14120 mining face of Pingdingshan No. 8 Mine is carried out, and the effects of different single borehole diameters, different kerf widths and different kerfheights on the distribution area of surrounding coal fracture field and the degree of rupture are studied quantitatively. These provide a theoretical basis for the reasonable determination of the slitting and drilling arrangement parameters at the engineering site.

2. MATHEMATICAL MODEL

2.1 Damage evolution equation

In order to describe the dynamic evolution process of damage in coal and rock mass, the strain-related damage variable is defined, and its expression under the unidirectional stress state is given as follows [15]:

\[
D = \begin{cases} 
0, & 0 < \varepsilon < \varepsilon_f, \\
\frac{\varepsilon - \varepsilon_f}{\varepsilon_u - \varepsilon_f}, & \varepsilon_f < \varepsilon < \varepsilon_u, 
\end{cases}
\]

where \( \omega \) is the damage variable, \( \varepsilon_f \) is the damage evolution threshold value of the materials under unidirectional stress state and \( \varepsilon_u \) is the ultimate strain.

For the three-directional stress state [16], let the three principal strains are \( \varepsilon_1, \varepsilon_2 \) and \( \varepsilon_3 \), respectively. The equivalent total strain can be expressed as \( \varepsilon = \sqrt{\varepsilon_1^2 + \varepsilon_2^2 + \varepsilon_3^2} \). The equivalent tensile strain is \( \varepsilon_t = \sqrt{\sum \varepsilon_i^2} (\varepsilon_i > 0) \) and the equivalent compressive strain is \( \varepsilon_c = \sqrt{\sum \varepsilon_i^2} (\varepsilon_i < 0) \). Then, the total damage of the three-dimensional object can be written as

\[
D = \alpha_t D_t + \alpha_c D_c, \tag{2}
\]

where \( D_t \) and \( D_c \) are determined by \( \varepsilon_t \) and \( \varepsilon_c \) according to Eq. (1), respectively, and \( \alpha_t = (\varepsilon_t / \varepsilon_f)^2 \) and \( \alpha_c = (\varepsilon_c / \varepsilon_f)^2 \).

2.2 Elastoplastic damage constitutive equation of coal and rock mass

Damage is a sign of material deterioration. The macroscopic fracture damage of rock is closely related to the development and accumulation of internal microcracks (i.e. internal damage accumulates gradually). In this paper, the rock mass is assumed to be the homogeneous medium, and the damage is mainly caused by the deviatoric stress; then, the constitutive equation of rock mass damage can be given as follows [17]:

\[
\sigma_{ij} = (1 - D)E_{ijkl}\varepsilon_{kl}^e + \frac{D}{3}\delta_{ij}E_{ijkl}\varepsilon_{kl}^c, \tag{3}
\]

where \( E_{ijkl} \) denotes the material parameters of coal and rock mass, \( \varepsilon_{kl}^e \) is the elastic strain and \( \delta_{ij} \) is the Kronecker symbol. Equation (3) is written as a tensor type:

\[
\sigma_{ij} = \tilde{E}_{ijkl}\varepsilon_{kl}^e, \tag{4}
\]

where \( \sigma_{ij} \) is the stress array, \( \varepsilon_{kl}^e \) is the elastic strain array and \( \tilde{E}_{ijkl} \) is related to the damage variable \( D \) and the matrix elastic constant \( E_{ijkl} \).

Formula (4) is rewritten in matrix form, which can be expressed as

\[
\sigma = \tilde{E}\varepsilon^e, \tag{5}
\]

where \( \sigma \) is the stress array and \( \tilde{E} \) is the damage stiffness matrix.

In the finite element calculation process, the element stiffness matrix needs to be calculated to form the overall stiffness matrix of the structure. The damage stiffness matrix can be written according to Eq. (3) as follows:

\[
\tilde{E} = \begin{bmatrix}
B & A & A & 0 & 0 & 0 \\
A & B & A & 0 & 0 & 0 \\
A & A & B & 0 & 0 & 0 \\
0 & 0 & 0 & C & 0 & 0 \\
0 & 0 & 0 & 0 & C & 0 \\
0 & 0 & 0 & 0 & 0 & C
\end{bmatrix}, \tag{6}
\]

where

\[
A = \frac{E\mu}{(1 + \mu)(1 - 2\mu)} + \frac{E}{1 + \mu}, \quad B = \frac{E}{1 + \mu}, \quad C = \frac{(1 - D)E}{2(1 + \mu)}, \quad D = \frac{E(1 - \mu)}{(1 + \mu)(1 - 2\mu)}. \]
2.3 Basic equation of elastic damage finite element method under mining conditions

In the actual coal seam mining process, the loading method and mining conditions are complex and variable, and the full-quantity finite element method may not be able to solve effectively; the most ideal method is to solve in incremental form. Therefore, the stress of formula (4) is differentiated, and the elastic damage constitutive equation of coal and rock mass in incremental form can be obtained as follows:

$$d\sigma_{ij} = \dot{E}_{ijkl} d\varepsilon_{kl} + \varepsilon_{kl} \dot{E}_{ijkl} d\varepsilon_{kl} + \varepsilon_{kl} \frac{d\tilde{E}_{ijkl}}{dD} dD.$$  (7)

For the kth incremental step, formula (7) can be rewritten as an incremental matrix as follows:

$$\Delta \sigma_k = \dot{E}_k \Delta \varepsilon_k + \Delta \dot{E}_k \varepsilon_k,$$  (8)

where $\Delta \sigma_k$ is a stress increment array, $\Delta \varepsilon_k$ is an elastic strain increment array and $\Delta \dot{E}_k$ is an equivalent modulus increment matrix.

For the ith element,

$$(\Delta \dot{E}_i) = \frac{\partial(\dot{E}_i)}{\partial(D_i)} (\Delta D_i), \quad (i = 1, \ldots, n),$$  (9)

where $(\Delta D_i) = (D_i) - (D_{i-1})$, $D_i$ is determined by the damage evolution equation and $n$ is the total number of elements.

According to Eq. (6), the equivalent modulus increment matrix can be solved as follows:

$$\Delta \dot{E} = \frac{\partial\dot{E}}{\partial D} \Delta D = \begin{bmatrix} B' & A' & A' & 0 & 0 & 0 \\ A' & B' & A' & 0 & 0 & 0 \\ A' & A' & B' & 0 & 0 & 0 \\ 0 & 0 & 0 & C' & 0 & 0 \\ 0 & 0 & 0 & 0 & C' & 0 \\ 0 & 0 & 0 & 0 & 0 & C' \end{bmatrix} \Delta D,$$  (10)

where $A' = E/3(1 + \mu) + 2E/3(1 + \mu)$, $B' = -2E/3(1 + \mu)$ and $C' = -G = -E/2(1 + \mu)$.

For the kth incremental step, the incremental equilibrium equation of the system can be obtained from the principle of virtual work as follows [18]:

$$\sum \int_{\Omega} B_{k}^T \Delta \sigma_k d\Omega - \Delta f_k = 0,$$  (11)

where $B_{k}$, $\Delta \sigma_k$ and $\Delta f_k$ are the strain matrix, the stress increment array and the load increment array in the kth incremental step, respectively, and $\Omega$ is the computational domain.

Substituting Eq. (8) into the system incremental balance equation (11), the basic equation of elastic damage finite element method of coal and rock mass can be obtained as follows:

$$\mathbf{K}_k \Delta \mathbf{a}_k = \Delta \mathbf{f}_k + \Delta \mathbf{f}^k,$$  (12)

where $\mathbf{K}_k$ is the overall damage stiffness matrix, $\Delta \mathbf{a}_k$ is the displacement increment array and $\Delta \mathbf{f}^k$ is the additional force caused by the accumulation of coal and rock damage, which is called the damage evolution additional force. The specific calculation formulas are as follows:

$$\mathbf{K}_k = \sum \int_{\Omega} B_{k}^T \mathbf{D}_k B_{k} d\Omega,$$  (13)

$$\Delta \mathbf{f}^k = -\sum \int_{\Omega} B_{k}^T \Delta \mathbf{D}_k \varepsilon_k d\Omega.$$  (14)

2.4 Basic equation of elastoplastic damage finite element method under mining conditions

When the coal and rock material enters the plastic stage, its stress–strain relationship no longer obeys Hooke’s law, so the constitutive relationship between stress–strain increments can only be established. For any small stress increment $d\sigma$, it is assumed that the total strain increment is composed of elastic strain increment $d\varepsilon$ and plastic strain increment $d\varepsilon^p$, which is

$$d\varepsilon = d\varepsilon^e + d\varepsilon^p.$$  (15)

From the elastic damage constitutive equation (7), it can be obtained as follows:

$$d\sigma = \dot{E} d\varepsilon^e + \frac{\partial\dot{E}}{\partial D} d\varepsilon^e dD.$$  (16)

The plastic strain can be obtained from Eq. (15) as follows:

$$d\varepsilon^p = d\varepsilon - d\varepsilon^p.$$  (17)

Substituting Eq. (17) into Eq. (16), an elastoplastic damage constitutive equation in incremental form can be obtained as follows:

$$d\sigma = \dot{E} (d\varepsilon - d\varepsilon^p) + \frac{\partial\dot{E}}{\partial D} d\varepsilon^e dD.$$  (18)

For the plastic strain increment $d\varepsilon^p$, it is related not only to the stress increment $d\sigma$ but also to the stress state, stress path and loading history. Its relationship with the plastic potential function is

$$d\varepsilon^p = d\lambda \frac{\partial Q}{\partial \sigma},$$  (19)

where $d\lambda$ is a proportional constant, which is called plastic factor. Formula (19) controls plastic flow after yielding occurs.

The plasticity factor $d\lambda$ can be obtained by the following method, and the yield condition is expressed by the following formula:

$$F(\sigma, H_\sigma, D) = 0.$$  (20)

According to the general differential rule, the following formula is established:

$$dF = \frac{\partial F}{\partial \sigma} d\sigma + \frac{\partial F}{\partial H_\sigma} dH_\sigma + \frac{\partial F}{\partial D} dD = 0.$$  (21)

Equation (21) can also be expressed as

$$\left(\frac{\partial F}{\partial \sigma}\right)^T d\sigma + \frac{\partial F}{\partial H_\sigma} dH_\sigma + \frac{\partial F}{\partial D} dD = 0.$$  (22)

Let

$$A = -\frac{\partial F}{\partial H_\sigma} \frac{1}{d\lambda}.$$  (23)
Then, the above formula becomes
\[
\left( \frac{\partial F}{\partial \sigma} \right)^T d\sigma - A d\lambda + \frac{\partial F}{\partial D} dD = 0.
\] (24)

Multiplying both sides of Eq. (15) by \( (\partial F/\partial \sigma)^T \) gives the following formula:
\[
\left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} de = \left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} d\epsilon^e + \left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} d\lambda \frac{\partial Q}{\partial \sigma} = 0.
\] (25)

Then,
\[
\left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} de^e = \left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} de - \left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} \frac{\partial Q}{\partial \sigma} d\lambda.
\] (26)

Substituting formula (16) for formula (24), we can obtain the following formula:
\[
\left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} de^e + \frac{\partial E}{\partial \epsilon^e} dD - A d\lambda + \frac{\partial F}{\partial D} dD = 0.
\] (27)

Substituting Eq. (26) into the above equation, the following equation can be obtained:
\[
\left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} de - \left( \frac{\partial F}{\partial \sigma} \right)^T \dot{E} \frac{\partial Q}{\partial \sigma} d\lambda + \left( \frac{\partial F}{\partial \sigma} \right)^T \frac{\partial \dot{E}}{\partial \epsilon^e} dD
\]
\[
- A d\lambda + \frac{\partial F}{\partial D} dD = 0.
\] (28)

Simplifying Eq. (28), the following equation can be obtained:
\[
d\lambda = \frac{\left( \frac{\partial E}{\partial \epsilon^e} \right)^T \dot{E} de + \left[ \left( \frac{\partial F}{\partial \sigma} \right)^T \frac{\partial \dot{E}}{\partial \epsilon^e} \right] dD}{A + \left( \frac{\partial F}{\partial \sigma} \right)^T \frac{\partial \dot{E}}{\partial \epsilon^e} \frac{\partial \sigma}{\partial \sigma}}.
\] (29)

If the associated flow rule is used, then \( Q = F \) in Eq. (29), where the above equation can be written as
\[
d\lambda = \frac{\left( \frac{\partial E}{\partial \epsilon^e} \right)^T \dot{E} de + \left[ \left( \frac{\partial F}{\partial \sigma} \right)^T \frac{\partial \dot{E}}{\partial \epsilon^e} \right] dD}{A + \left( \frac{\partial F}{\partial \sigma} \right)^T \frac{\partial \dot{E}}{\partial \epsilon^e} \frac{\partial \sigma}{\partial \sigma}}.
\] (30)

where \( A \) is the hardening parameter of the material.

According to the elastoplastic damage constitutive equation in the incremental form of coal and rock mass, the basic elastoplastic damage finite element equation of coal and rock mass can be established. For the kth incremental step, Eq. (18) is rewritten in incremental matrix form by the incremental method as follows:
\[
\Delta \sigma_k = \dot{E}_k \left( \Delta \epsilon^e - \Delta \epsilon^p_k \right) + \Delta \dot{E}_k \epsilon^p_k.
\] (31)

At this time, the incremental equilibrium equation of the system is still as follows:
\[
\sum \int_{\Omega} B_k^T \Delta \sigma_k d\Omega - \Delta f_k = 0,
\] (32)

where \( B_k \) is the strain matrix in the kth incremental step, \( \Delta \sigma_k \) is the stress delta array, \( \Delta f_k \) is the load delta array and \( \Omega \) is the computational domain.

Substituting Eq. (31) into the system incremental balance equation, the basic elastoplastic damage finite element equation of coal and rock mass can be obtained as follows:
\[
\dot{K}_k \Delta a_k = \Delta f_k + \Delta f_k^p + \Delta f_k^q.
\] (33)

where \( \dot{K}_k \) is the total damage stiffness matrix, \( \Delta a_k \) is the displacement increment array and \( \Delta f_k^p \) is the damage evolution additional force, and its specific calculation form can be seen in Eqs (13) and (14). \( \Delta f_k^q \) is the additional force caused by the plastic strain increment, which is calculated as
\[
\Delta f_k^q = \sum \int_{\Omega} B_k^T \Delta \epsilon^e d\Omega.
\] (34)

3. ACOUTIC EMISSION EXPERIMENT OF COAL SAMPLE UNDER UNIAXIAL COMPRESSION

According to the basic theory of damage finite element method, the damage value of each element can be determined by numerical calculation, but the fracture form and rupture degree of the corresponding element cannot be intuitively and clearly understood. In order to determine the micro-damage failure, macro-fracture penetration failure or broken damage of coal and rock mass in different damage stages, the acoustic emission experiment will be used to reflect the fracture degree of coal mass in different damage stages.

3.1 Coal sample collection and processing

The coal samples were all collected from the No. 8 Mine in Pingdingshan Mining Area. In the upper slot beyond 100 m ahead of the working surface, the eye-catching machine is used to intensively drill the eye to obtain a representative coal block with good integrity and no weathering. Standard samples were made by cutting large coal samples in the laboratory, as shown in Fig. 1.
3.2 Test equipment and solutions

The rock uniaxial test machine TAW-2000 of China University of Mining and Technology is used in the test, as shown in Fig. 2. The axial maximum compression load of the experimental machine is 2000 kN, the axial deformation range is 8 mm, the circumferential deformation range is 4 mm and the precision is 1%. The acoustic emission test uses PCI-2 type acoustic emission system produced by American Acoustics Physics Company. This system adopts 18-bit A/D conversion technology, which can realize real-time acquisition of acoustic emission signals, and can also collect and store waveform signals in real time. The test adopts the displacement control mode, and the loading rate is 0.02 mm/min for static uniaxial loading of coal and rock mass. At the same time, the acoustic emission sensor is attached to the center of the coal and rock specimen in the length direction, and fixed firmly with tape. In order to ensure the synchronization of data acquisition between the loading system and the acoustic emission system, the loading system and the acoustic emission system simultaneously start recording.

3.3 Test results and analysis

The stress–strain curve of coal sample in the process of conventional uniaxial compression failure is an important means to describe the mechanical and deformation characteristics of coal mass. The relationship between strain and stress of coal sample can be obtained after data processing as shown in Figs 3 and 4. Figure 3 reflects that the coal sample used in the test has good plasticity. Therefore, the basic theory of elastoplastic mechanics should be adopted for the construction analysis of Pingdingshan No. 8 Mine. It can be seen from Fig. 4 that when the axial stress reaches ~6.4 MPa, the volume compaction point is reached, and the circumferential strain of the coal sample increases rapidly, indicating that the coal sample is about to be destroyed.

According to the relationship curve between the acoustic emission information and strain obtained by the experiment, we use damage variables to describe the dynamic evolution process of damage in the rock, assuming that strain is the main factor controlling the damage evolution, so as to define the strain-related damage variables. The damage or degradation of material is a cumulative process, and it is impossible to judge when it started from a macroscopic perspective. The acoustic emission ringing or energy count can reflect the rupture of the material to a certain extent. Therefore, the strain when the acoustic emission energy counting rate first has a larger value in the coal and rock compaction process is taken as the damage evolution threshold, and the damage variation law of coal samples is determined, as shown in Fig. 5.

The acoustic emission phenomenon of coal sample can well reflect the deformation characteristics and damage development of coal. By comparing the acoustic emission signal with the stress–strain curve of coal sample, the acoustic emission law of coal sample can be roughly divided into the following five stages: initial closed stage, slow rising stage, front peak
stage, peak section and post-peak section. The acoustic emission characteristics of each stage correspond to the damage evolution of the coal sample: when the damage increases, the acoustic emission signal increases, and when the damage decreases, the acoustic emission signal also decreases. According to the acoustic emission information of Fig. 6, when the damage degree is $<0.6$, the acoustic emission information is weak, and the energy counting rate and the ringing counting rate are both small, indicating that the internal microcracks of the coal sample start to develop continuously at this stage, but the expansion is slow, and no macroscopic main fracture is formed. Therefore, it is defined as the micro-damage stage. When the damage degree is $>0.6$, the acoustic emission activity is severe, and the maximum energy count rate and the ringing count rate appear in the process of failure, indicating that the internal crack of the coal sample rapidly expands, penetrates and forms the main crack. Therefore, it is considered that the macroscopic fractures of the coal samples are interpenetrated in this damage stage, and the coal sample will be broken to a large extent. Obviously, the quantitative description of the damage variable can clearly reflect the degree of coal rupture.

4. NUMERICAL CALCULATION ANALYSIS

In program design, based on the solid structure finite element source program written in FORTRAN language, the two developments of the corresponding program are carried out according to the theoretical foundation of the elastoplastic damage and deformation analysis of coal mass given by part 2; the program is further developed. In the process of elastoplastic damage and deformation calculation of coal mass, the nonlinear incremental solution method is applied. At the same time, the impact of borehole and slotted construction disturbances is transformed to the corresponding additional nodal force, which improves the efficiency of program calculation. Therefore, the program can solve complex three-dimensional problems. In addition, the computational program uses the large finite element software ANSYS as a pre-processing tool to establish numerical model and divide the units. Then, APDL language is used to output relevant information such as elements, nodes and materials of the numerical model, so as to form a text file and be read into the finite element program written to calculate. Then, we use Surfer, Tecplot and other software to carry out the post-processing of the related results.
Table 1 Mechanical parameters of each rock stratum.

| Strata formation name | Thickness, $h$ (m) | Elastic modulus, $E$ (GPa) | Poisson ratio, $\mu$ | Internal cohesion (MPa) | Internal friction angle (°) | Tensile strength (MPa) | Density (N/m$^3$) |
|-----------------------|-------------------|---------------------------|---------------------|-------------------------|---------------------------|----------------------|------------------|
| Roof                  | 5                 | 5.04                      | 0.22                | 1.68                    | 32                        | 3.02                 | 22 710           |
| Coal seam             | 3.6               | 4.92                      | 0.27                | 2.67                    | 44                        | 2.57                 | 12 950           |
| Floor                 | 3.4               | 5.04                      | 0.22                | 1.68                    | 32                        | 3.02                 | 22 710           |

4.1 Mechanical model

There is high gas content and low permeability of coal seam in the 15-14120 mining face of No. 8 Coal Mine of Pingmei Coal Mine. According to the data of China Pingmei Shenma Group, the 15-14120 mining face of Pingmei No. 8 Mine is located in the west wing of the Jier Shangshan mining area, from the mining area to the east, to the north wind shaft of the No. 12 Mine in the west, to the protection coal pillar line of the Ji group and adjacent to the 15-14100 mining face to the north, not yet developed. The mining face is buried at 580–705 m, the east and west can be 864 m long and the north–south slope is 190 m wide. According to the geological conditions of the mining face, consider a single borehole length of 70 m, and cut the seam in the middle of the borehole. The mechanical model is established as shown in Fig. 6. The coal seam tends to take 100 m, the strike is 30 m and the vertical direction is 24.12 m. The coal seam has an inclination of 22° and a thickness of 3.6 m, and there is a layer of 0.1–0.5 m coal line ~0.8 m from the top of the coal seam, which is easy to fall along with mining. The direct bottom is a thin layer of mudstone, ~2.8–6.0 m thick. Before the coal seam mining, the initial stress field is formed according to the self-weight of the overlying 600-m rock layer. Table 1 shows the mechanical parameters of each rock layer.

4.2 Distribution and evolution characteristics of coal damage field under the drilling and slitting construction

4.2.1 Impact of different borehole diameters on the distribution and evolution characteristics of coal seam damage field

In the coal seam bedding drilling construction, different mining areas often use different drilling diameters to extract gas. In order to study the influence of different borehole diameters on the mechanism of pressure relief and permeability enhancement of coal seam, the borehole diameters of 50, 75 and 100 mm are selected to study the influence of borehole diameters on the fissure distribution and damage degree of surrounding coal and rock mass. Figure 7 shows the distribution of the damage field of the coal mass around different borehole diameters.

It can be seen from Fig. 8 that after the coal seam is drilled, the stress concentration around the drilling hole resulted in the damage and destruction of the coal mass around the drilling hole. The most severely damaged coal bodies are all around the borehole. The distribution of the entire damage field is “elliptical” and is distributed symmetrically along the borehole. When the diameter of the borehole is 50 mm, the impact field of the damage field is 2.9 m, and the damage degree of the coal mass $>0.55$ occurs within 1.1 m around the borehole. When the diameter of the borehole is 75 mm, the impact field of the damage field is 3.1 m, and the degree of damage of the coal mass $>0.55$ occurs within 1.2 m around the borehole. When the diameter of the borehole is 100 mm, the impact field of the damage field is 3.3 m, and the same degree of damage of the coal mass $>0.55$ occurs within 1.3 m around the borehole. It can be seen that as the diameter of the borehole increases, the range of damage and destruction of the coal mass increases, but the increase of the area with serious damage is not obvious. At the same time, stress concentration around borehole is not conducive to gas permeation, so it is of little significance to increase borehole diameter only to extract gas. In addition, the coal mass around 1 m of borehole will be damaged to a large extent. Therefore, the borehole spacing can be considered to be ~2 m when using simple borehole for gas extraction.

4.2.2 Effects of slitting on damage field distribution in coal seam

From the analysis of the previous section, it can be seen that increasing the borehole diameter has little effect on increasing the permeability of coal seam. Therefore, it is necessary to...
consider cutting around the borehole to increase the permeability of coal seam. In this paper, the effects of different slot widths and heights on the spatial distribution and rupture degree of the damage fields in the surrounding coal seam are studied.

4.2.2.1 Impact of different slitting widths on the distribution of coal seam damage fields

In order to investigate the influence of different widths of single slit on the distribution and rupture degree of coal seam damage field, when the height of the slit is 1000 mm and the diameter
Figure 9 Distribution of the coal seam damage field on section $\alpha$ under different slit heights: (a1, b1) distribution of the damage field in coal seam with the slit height of 600 mm and in local area around the slit; (a2, b2) distribution of the damage field in coal seam with the slit height of 1000 mm and in local area around the slit; and (a3, b3) distribution of the damage field in coal seam with the slit height of 1500 mm and in local area around the slit.
of the borehole is 75 mm, the spatial distribution and rupture degree of coal seam damage field are calculated and analyzed for slit widths of 50, 80 and 100 mm, as shown in Fig. 8.

From Fig. 8a, it can be seen that as the width of the slit increases from 50 to 80 and 100 mm, the maximum influence range of coal seam damage field increases from 4.9 to 5.1 and 5.3 m, respectively, and the influence range of coal mass with damage degree > 0.55 increases from 1.75 to 1.9 and 2 m. According to the distribution chart of the coal seam damage field in the local area around the slit in Fig. 8b, the damage degree of coal mass in a large area around the slit is maintained above 0.95. With the increase of the slit width from 50 to 100 mm, the length of the circular area only increases from 4 to 4.3 m, which shows that the influence of the slit width is small. Therefore, it is not preferable to further expand the coal seam cracks by increasing the width of the slit. However, in order to avoid the closure of both sides of the slit, it is necessary to ensure that the slit has a certain width in engineering construction. If the distribution of coal seam damage field under a single borehole is compared with that under a single slit, the maximum impact range of the damage field will increase from 3.1 to ~5 m, and the distribution shape of damage field will also evolve from a flat "ellipse" to a "dumbbell" shape. Moreover, the coal damage degree in a large area around the slot is >0.95, indicating that the coal seam fissure development in this area is better. It can be seen that the stress concentration is released due to the existence of the slit, and the coal damage degree in the pressure relief area near the slit is higher. It can be seen that the slitting can only fully relieve the pressure on the surrounding coal mass in a local area, promote the further expansion of the cracks of the coal mass and improve the permeability of the coal seam rapidly. Therefore, in order to further increase the permeability of the coal seam around the whole borehole, it is possible to produce a better pressure relief effect by the rational arrangement of the slits and the multi-slit cutting.

4.2.2.2 Impact of different slitting heights on the distribution of coal seam damage fields

In order to study the influence of different heights of single slit on the spatial distribution and rupture degree of coal seam damage field, when the width of the slit is 50 mm and the diameter of the borehole is 75 mm, the spatial distribution and rupture degree of coal seam damage field are calculated and analyzed for slit heights of 600, 1000 and 1500 mm, as shown in Fig. 9.

From Fig. 9, it can be seen that with the increase of slit height from 600 to 1000 and 1500 mm, the maximum impact range of damage field of coal seam increases from 4.5 to 4.9 and 5.3 m. It is also known that slit height has less influence on distant coal seam. Therefore, this section focuses on the impact of cutting height on the damage and destruction of the surrounding coal. From Fig. 9, it also can be seen that with the increase of the slot height, the length of the circular region with damage degree >0.95 around the slot increases from 2.6 to 4 and 6.2 m, respectively, which indicates that the range of serious damage and rupture around the slot obviously enlarges. It can be seen that with the increase of the height of the slit, a larger area of the pressure relief zone appears around the slot, and the cracks in the coal seam are continuously diffused and penetrated, which increases the gas permeability of the coal seam and is beneficial to the pressure relief of the gas and extraction. Therefore, in engineering construction, increasing the slot height can be adopted to crack the coal around the borehole, so as to increase the effective area of gas extraction.

5. CONCLUSION

(1) For a single borehole, when the diameter of the borehole increases from 50 to 100 mm, the extent of the damage of the coal mass around the borehole that is >0.55 increases from 1.1 to 1.3 m. It can be seen that increasing the diameter of the borehole cannot effectively enhance the penetration of the coal around the borehole.

(2) Comparing the spatial distribution of the damage field of a single slit and a single borehole, it can be seen that the distribution pattern of the damage field evolves from "ellipse" to "dumbbell", and the degree of coal damage within a large area around the slot is >0.95; the effect of pressure relief is obvious. According to the influence range of a single slit, the slit can be arranged reasonably, and the multi-slit cutting can provide better effect of pressure relief. In addition, according to the influence range of the damage field after multi-slit cutting, the drilling spacing can be enlarged to save a lot of manpower and material resources.

(3) Different widths of slit have little effect on the spatial distribution and damage degree of the coal mass damage field around the borehole. As the width of the slit increases from 50 to 100 mm, the length of the circular region with damage >0.95 only increases from 4 to 4.3 m, and the effect of antireflection is not obvious. However, in the course of construction, in order to avoid the closure of the slit, it is also necessary to ensure that the slit has a sufficient width.

(4) The height of different slots has a great influence on the spatial distribution and degree of the damage field around the borehole. As the height of the slit increases from 600 to 1500 mm, the length of the circular region with damage >0.95 increases from 2.6 to 6.2 m; the effect of cracking and permeation is better. Therefore, the height of the kerf can be increased to enhance the permeability of the coal surrounding the borehole to increase the effective area of the gas drainage.

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