Research Article

Study on the Decoupled Charge Effect in Deep-Hole Cumulative Blasting of Coal Seam

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Five models of cumulative blasting are established by using ANSYS/LS-DYNA to study the effect of decoupling coefficient on cumulative blasting to improve coal seam permeability. The formation and migration process of the shaped energy jets with two kinds of decoupling coefficient are compared and analyzed; also, the propagation of explosive stress waves is represented. The result showed that the air in the blast hole is the key to the formation and migration of the condensing jet. The air in the hole also could reduce the attenuation of stress wave in a certain range. However, if the decoupling coefficient is too large, the air in the hole will consume excessive explosive energy, which is also not conducive to energy transfer. Therefore, there is an optimum decoupling coefficient which can minimize the coal crushing area, increase the coal fissure area, and improve the gas extraction rate. Besides, the cumulative blasting tests were carried out in a coal seam. The test results show that decoupling charge could effectively improve coal seam permeability, and the blasting effect was better when the decoupling coefficient is between 1.67 and 2.

1. Introduction

As a primary energy, coal has played a great role in promoting the development of human industrialization. In recent years, with the increase of mining depth, coal seams are mostly characterized by high gas and low permeability, and gas disasters are becoming more and more serious [1–3]. Scholars have put forward measures such as hydraulic fracturing, liquid CO₂ fracturing, and loose blasting for high gas and low permeability coal seams [4, 5]. Loose blasting is most widely used among them. The conventional blasting technology often encounters the difficult problem that the coal seam fracture development is not good or the coal pulverization is serious [6]. In order to improve the utilization rate of explosive, the cumulative blasting technology is proposed.

In the past years, a lot of scholars have carried out numerous research studies on cumulative blasting. Mu et al. have studied the crack propagation mechanisms in cumulative blasting through experiments and simulation [7]. Luo and Shen have studied the mechanism of crack initiation and

its expansion of directional fracture controlled blasting with shaped charges in rock. Their test results showed that the energy from blasting is directionally concentrated for the cumulative action [8]. Guo et al. have made a detailed study on the construction technology and mechanism of cumulative blasting and applied cumulative blasting to improve the permeability in high gas and low permeability coal seams [9–11]. A large number of literature show that radial decoupling coefficient seriously affects the blasting effect, but these results are mostly based on conventional blasting or slotted cartridge [12–16]. At present, scholars have seldom studied the effect of radial decoupling coefficient on the stress wave propagation and the crack propagation in cumulative blasting. In order to understand the mechanism of cumulative blasting more systematically, it is necessary to study the charge structure. Therefore, theoretical analysis combined with numerical simulation was adopted to study the effect of coupled charge and decoupled charge on cumulative blasting, and the effect of decoupling coefficient on stress wave propagation and stress state of coal element was discussed. At the same time, field tests have been carried out to study the

influence of decoupling coefficient on deep-hole cumulative blasting to improve coal seam permeability, and the optimum decoupling coefficient range is determined finally.

2. Fundamental Theories

Figure 1(a) shows the behavior of decoupled charge of conventional column charge, the explosion gas is isotropic adiabatic expansion in the blast hole, and the initial pressure of the explosion gas is very high. Due to the compressibility of the air, the explosion gas keeps expanding continuously and fills the whole hole finally, and the pressure decreases rapidly. The pressure on the wall of blast hole could be expressed as [17]

$$P = \lambda P_j \left(\frac{P_w}{P_j} \right)^{\gamma/k} \alpha^{-\gamma} \beta^{-2\gamma}, \quad (1)$$

where P_j is the critical pressure, P_w is the average detonation pressure of explosives, k is the isotropic exponent (k is usually 3), γ is the adiabatic exponent, α is the decoupling coefficient of axial charge, β is the radial decoupling coefficient ($\beta = d/dc$, d is the diameter of blast hole, and dc is the diameter of charge), and λ represents the pressure increase factor (λ is usually 8~11).

The derivative of β in the above equation is

$$\frac{\partial P}{\partial \beta} = -2\gamma \lambda P_j \left(\frac{P_w}{P_j} \right)^{\gamma/k} \alpha^{-\gamma} \beta^{-2\gamma-1}. \quad (2)$$

It can be obtained that the pressure of the hole wall is inversely proportional to the radial decoupling coefficient, that is, the smaller the radial decoupling coefficient is, the greater the initial pressure of the hole wall is, indicating that the radial decoupling coefficient seriously affects the blasting effect.

As shown in Figure 1(a), the explosion gas of conventional charge diffuses evenly around after detonation, and a large part of the explosion energy is consumed in coal body crushing. Ultimately, the crushing area is large, and the cracking area is small. Cumulative blasting realizes directional accumulation of blasting energy by changing the structure of charge and installing metal covers on both sides of charge, as shown in Figure 1(b). It weakens the crushing effect of blasting on coal and enlarges the fracture area of coal and rock. After initiation, the two wings of the metal cover are subjected to tremendous pressure of detonation wave which was intensely compressed and finally collide at high speed in the direction of the axis. Due to the characteristics of small compressibility and high density of metal cover, a metal jet with high speed, high energy, and high pressure is finally formed in the direction of the axis, which is called shaped charge jet [18]. Then the shaped charge jet penetrates the coal and forms the initial directional cracks, which is beneficial for the subsequent stress wave and explosion gas to expand the crack areas.

It could be seen from conventional blasting that radial decoupling coefficient is an important parameter of blasting, so it is of great significance to study the influence of decoupling coefficient on cumulative blasting.

3. Numerical Simulation

3.1. Model and Parameter. Five models of cumulative blasting are established by using ANSYS/LS-DYNA. The model consists of coal, explosive, metal cover, and air. The diameter of explosive is 4.5 cm, the thickness of metal cover is 0.2 cm, and the size of coal is 200 cm × 140 cm × 0.5 cm. The blast holes are located in the center of coal with diameters of 4.5 cm, 6 cm, 7.5 cm, 9 cm, and 11 cm, respectively; that is, the radial decoupling coefficients are $\beta = 1, 1.33, 1.67, 2$, and 2.5 . Due to the symmetry of the model, in order to reduce the computational workload, only 1/4 of the model can be established. Figure 2 shows the plane model of decoupling coefficient 2. The plane models of the other decoupling coefficients are similar. The ALE unit is used for explosive, concentrator, and air, and the multimaterial algorithm 11 is used. When coal is meshed, the mesh near the blast hole is denser, and the farther away from the blast hole, the more sparsely the coal mesh is divided. Z direction constraint is applied to the front and back of the model, Y direction constraint is applied to the lower boundary, and X direction constraint is applied to the right boundary. In order to eliminate the effect of reflection superposition of stress waves at the free surface on the fracture of coal, nonreflecting boundary conditions are imposed on the upper and left interfaces.

The material model of the explosive employed herein is MAT_HIGH_EXPLOSIVE_BURN, which corresponds to the JWL equation of state. The relationship between the pressure and the specific volume in the detonation process is given as [19]

$$P = A \left[1 - \frac{\omega}{R_1 V} \right] e^{-R_1 V} + B \left[1 - \frac{\omega}{R_2 V} \right] e^{-R_2 V} + \frac{\omega E_0}{V}, \quad (3)$$

where P is the pressure of detonation pressure, V represents the specific volume, and E_0 is the initial internal energy of explosives. A , B , R_1 , R_2 , and ω are all explosive constants, and the specific parameters are shown in Table 1.

The material model of the metal cover employed is MAT_JOHNSON_COOK, which corresponds to the EOS_GRUNEISEN equation of state. The material model of air employed is MAT_NULL, which corresponds to the EOS_LINEAR_POLY-NOMIAL equation. In the calculation process, the elastic-plastic properties of coal and the changing of strain rate should be considered, so MAT_PLASTIC_KINEMATIC material model is used to simulate the constitutive relationship of coal under blasting conditions.

3.2. Effect of Decoupling Coefficient on Stress Wave Propagation. An important part of cumulative blasting is that shaped charge jet penetrates into coal. Figures 3–7 show the formation and migration of shaped charge jets with decoupling coefficients 1, 1.33, 1.67, 2, and 2.5. The final shape of shaped charge jet varies with the decoupling coefficient, as shown in Figures 3(c)–7(c). The larger the uncoupling coefficient is, the later the jet begins to penetrate the coal. In this paper, the formation and migration of shaped jet with decoupling coefficients of 1 and 1.67 is analyzed in detail.

When decoupling coefficient is 1.67, the detonation wave begins to propagate from the detonation point of the

FIGURE 1: The principle of blasting: (a) conventional blasting; (b) cumulative blasting.

FIGURE 2: Numerical simulation model of cumulative blasting with decoupling coefficient two.

TABLE 1: Parameters of emulsion explosive.

Density (g/cm^{-3})	Detonation velocity ($\text{cm}/\mu\text{s}^3$)	Detonation pressure/(GPa)	A/(GPa)	B/(GPa)	R_1	R_2	ω	E_0 /(GPa)
1.14	3.2	2.918	246.1	10.46	7.177	2.401	0.069	4.19

FIGURE 3: Process of shaped charge jet formation and migration under cumulative blasting with decoupling coefficient 1: (a) $t = 3.94 \mu\text{s}$; (b) $t = 8.99 \mu\text{s}$; (c) $t = 12.99 \mu\text{s}$.

explosive, and then the detonation wave begins to compress the metal cover, as shown in Figure 5(a). The metal cover is heavily compressed under the intense pressure of the detonation wave and moves along the central axis, as shown in Figure 5(b). Finally, the two wings of the metal cover collide at a high speed in the direction of the central axis and then move forward at an ultrahigh speed, eventually

forming a high-speed, high-pressure metal jet in the axial direction to penetrate the coal, as shown in Figure 5(c). When decoupling coefficient is 1, the metal cover is also strongly compressed by the detonation wave, but it is found that the metal cover does not have enough space to migrate, as shown in Figure 3(b), and penetrates into the coal directly, as shown in Figure 3(c). Finally, it cannot form the

FIGURE 4: Process of shaped charge jet formation and migration under cumulative blasting with decoupling coefficient 1.33: (a) $t = 4.96 \mu s$; (b) $t = 7.92 \mu s$; (c) $t = 17.29 \mu s$.

FIGURE 5: Process of shaped charge jet formation and migration under cumulative blasting with decoupling coefficient 1.67 m: (a) $t = 4.97 \mu s$; (b) $t = 12.98 \mu s$; (c) $t = 32.99 \mu s$.

FIGURE 6: Process of shaped charge jet formation and migration under cumulative blasting with decoupling coefficient 2: (a) $t = 4.20 \mu s$; (b) $t = 20.96 \mu s$; (c) $t = 35.79 \mu s$.

shaped charge jet as shown in Figure 5(c) and ultimately cannot achieve a good penetration effect. From the above analysis, it could be seen that the air between the explosive and the hole can provide a certain space for the migration of the shaped charge hood, which is the key to the formation of shaped charge jet.

After the shaped charge jet penetrates the coal, the explosion stress wave continues to spread in the coal. The stress wave propagation process with different uncoupling coefficients is similar. The stress wave propagation process of decoupling coefficient 1.67 is analyzed here, as shown in Figure 8. The detonation energy is highly concentrated, as

FIGURE 7: Process of shaped charge jet formation and migration under cumulative blasting with decoupling coefficient 2.5: (a) $t = 4.28 \mu s$; (b) $t = 21.96 \mu s$; (c) $t = 45.99 \mu s$.

FIGURE 8: Propagation of stress wave under cumulative blasting with decoupling coefficient of two: (a) $t = 60.96 \mu s$; (b) $t = 86.95 \mu s$; (c) $t = 126.98 \mu s$.

shown in Figure 8(a). The coal in the direction of concentrating energy is destroyed preferentially by the stress wave relative to the coal in other directions, as shown in Figure 8(b). The coal element in the direction of concentrating energy shows obvious stress concentration, which promotes the development of coal cracks in the direction of shaped jet.

The coal in the direction of nonconcentrated is affected by the energy-concentrating structure, and the energy transmitted to the hole wall is reduced, thus reducing the crushing range of the coal near the hole wall by the stress wave. It can be seen that cumulative blasting can effectively reduce the coal crushing area and increase the coal crack area.

3.3. Effect of Decoupling Coefficient on the Stress of Coal Element. It can be seen from the above and literature [20] that the cumulative blasting is mainly to increase the coal crack area in the direction of concentrating energy. The stress of coal element at the hole wall varies with different decoupling coefficients. Figure 9 shows the curve of stress of coal element in direction of concentrating energy at the hole wall varying with the time.

It is found that the peak stress of coal element in the case of coupling charge is 697 MPa, which is far less than the peak

FIGURE 9: The curve of stress of coal element in direction of concentrating energy at the wall varying with the time (β is the radial decoupling coefficient).

stress of coal element in the case of decoupling charge. There is not enough space between the explosive and the borehole to form a shaped charge jet after the detonation wave compressing the metal cover under the condition of coupling charge. It leads to the stress of the coal unit in the direction of concentrating energy being too small, weakens the advantage of the cumulative blasting, and finally, ultimately penetrates coal with poor effect. The peak stress of the coal unit in the borehole wall increases when the decoupling coefficient is between 1 and 1.67, and the energy carried higher when the decoupling coefficient is 1.67, which indicates that the air in the borehole is the key to the formation and migration of the jet. The air within a certain range in the borehole can increase the initial stress of the coal unit. However, as the decoupling coefficient continues to increase, the stress of coal unit begins to decrease gradually, which indicates that the air in the borehole consumes too much energy and reduces the energy transfer. The excessive decoupling coefficient also weakens the advantage of cumulative blasting.

In order to analyze the influence of decoupling coefficient on the attenuation characteristics of stress wave propagation, five units are selected for each charge case, and the unit number is recorded as 1~5. Among them, the first unit is 10 cm away from the explosive center, and the adjacent units are 5 cm away. The peak values of the pressure time history curves of these units are obtained, as shown in Figure 10. It can be seen from Figure 10 that the coal elements' stress is the lowest, which indicates that the explosive utilization rate is the lowest and the energy transfer is the worst in the case of coupling charge. It is found that the stress wave attenuates faster when the decoupling coefficient is 1.33. Comparing the peak stress value of coal element with decoupling coefficient 1.67 and 2, it is found that the peak stress value of coal element with decoupling coefficient 1.67 in initial stage is larger than that of coal element with decoupling coefficient 2. However, with the increase of propagation distance, the peak stress value of coal elements with decoupling coefficient 2 decreases slowly, and the values are gradually larger than that of coal elements with decoupling coefficient 1.67 at the same distance. It is because with the increase of the decoupling coefficient, the time of the hole wall under the impact pressure increases, and the stress wave attenuates slowly, which showed that air increases the time of stress wave action in a certain range. When the decoupling coefficient is 2.5, the air in the hole will consume too much explosive energy, which is not conducive to the energy transfer.

4. Field Experiment

4.1. Engineering Situation. Experiments have been carried out at 15, 16-24130 working face of Pingmei No.10 Coal Mine. The strike length, dipping length, and elevation of the working face are 187 m, -540 m~-590 m, 850 m~940 m, respectively. According to the results of the coal seam gas pressure and gas content test, the coal seam gas pressure is 1.7~2.2 MPa and the maximum gas content is 12.5 m³·t.

FIGURE 10: Stress-time curves of coal elements at different distances (β is the radial decoupling coefficient).

The permeability of coal seam is usually represented by the permeability coefficient. The greater the permeability coefficient is, the easier the gas flow is. There are plenty of research studies about how to determine permeability coefficient. Liu Mingju proposed an algorithm to optimize the permeability coefficient of coal seam, which improves the radial flow method proposed by Zhou Shining, and it has been widely used at present.

According to the algorithm, the dimensionless flow criterion Y and time criterion F_0 of gas radial flow can be expressed as

$$Y = \frac{A}{\lambda} = \frac{qr}{\lambda(p_0^2 - p_1^2)}, \quad (4)$$

$$F_0 = B\lambda = \frac{4\lambda t p_0^{1.5}}{\alpha r^2},$$

where $A = q/\lambda(p_0^2 - p_1^2)$ and $B = 4t p_0^{1.5}/\alpha r^2$, which can be calculated by the actual parameters measured, then permeability coefficient can be calculated by A, B directly, and λ is permeability coefficient. P_0 is original coal seam gas pressure; P_1 is gas pressure when drilling a hole to drain gas; α is coal seam gas content coefficient. r is the borehole radius. q is gas discharge per unit area at time t ; it can be expressed as $q = Q/2\pi rL$, where Q is borehole flux at time t and L is the coal length.

Through calculation, the permeability coefficient of coal seam fluctuates between 0.052 and 0.076 m² (MPa·d⁻¹). It shows that this coal seam belongs to the lower permeability coal seam.

4.2. Experimental Program. According to the field geology condition, two stages of deep-hole cumulative blasting tests have been carried out. Five groups of specific test schemes are arranged along the dip of coal seam as follows in each stage of the experiment. The borehole and sealing of the hole

FIGURE 11: Structure of cumulative blasting and sketch of boreholes in the investigated area. (a) Structure of cumulative blasting in hole. (b) Profile of I-I. (c) Layout of blasting and gas extraction holes in field test.

are, respectively, 45 meters and 10 meters deep. The quantity of charged powder per hole is 16 kg, and the charge diameter is 45 mm. The diameter of the hole is, respectively, 48 mm (near to coupled charge), 75 mm, 90 mm, and 110 mm. The gas extraction inspection holes are 2 m apart from adjacent blasting holes. The charge structure of cumulative blasting and specific blasting drilling program is shown in Figure 11.

The crack length cannot be measured directly in blasting engineering. The permeability improvement effect of coal can be quantitatively reflected by comparing the gas volume fraction of each extraction hole. The construction of each gas extraction hole is completed in advance. We continuously measure the change of gas volume fraction in the gas extraction hole before and after blasting.

4.3. Effect of Decoupling on the Degree of Permeability Increase.

According to the above experimental program, we detect the gas content of each extraction hole for 10 consecutive days, in which the first three days are the gas content before blasting and the last seven days are the gas content after blasting.

After the first stage of deep-hole cumulative blasting test in coal seam, the gas volume fraction of each extraction hole has been measured. A set of data with decoupling coefficient 1.67 shows that the gas volume fraction increases little after blasting. It is found that punching phenomenon occurred due to the poor quality of hole sealing, resulting in blasting gas escape. Therefore, the quality of hole sealing directly determines the effect of cumulative blasting on coal seam fracture.

Finally, the average values of the remaining groups of experimental data have been obtained. The average gas volume fraction of each extraction hole is shown in Figure 12(a).

As shown in Figure 12(a), the average volume fraction of gas in each extraction hole is about 26% before cumulative blasting. It is proved that the degree of fracture development of coal body near each extraction hole is basically the same before cumulative blasting. After blasting, it is found that the gas volume fraction of each extraction hole is strengthened to different degrees. The minimum gas volume fraction of extraction hole A is 37.3% after blasting. It shows the coal penetration effect is not ideal finally. It is because blasting hole A is approximately coupled charge. As a result, there is not enough space between blast holes to form a shaped charge jet, which reduced the advantage of the cumulative blasting. The average gas volume fraction of extraction hole D is 47.2%; the coal penetration effect is also not ideal. It indicates that when the decoupling coefficient is 2.45, the air consumes too much energy, which reduces energy transfer. It can be seen that too large decoupling coefficient is not conducive to cumulative blasting. Gas volume fraction of extraction holes B and C is, respectively, 68.2% and 70.3%, which is about 2.7 times the volume fraction of gas before blasting, and the effect of coal seam cracking is good.

As shown in Figure 12(b), three days before cumulative blasting, the average volume fraction of gas in each extraction hole is about 10%. It is proved that the degree of fracture development of coal body near each extraction hole is basically the same before cumulative blasting. After blasting, the average gas volume fraction in each inspected hole has changed greatly. The average gas volume fraction of

FIGURE 12: The average volume fraction of gas before and after cumulative blasting. (a) The first stage. (b) The second stage.

extraction hole A increased from 10% to 28.2% and by 2.82 times. The average gas volume fraction of extraction hole D is 47.2% after blasting and increased by 3.92 times. The average gas volume fraction of extraction holes B and C is, respectively, 64.57% and 61.5%, and the effect of coal seam cracking is the best in the same way.

Compared with the first stage, the gas volume fraction of each gas extraction hole is smaller than the first stage before the cumulative blasting. It is because the exposed time of coal seam in the test face is longer and the residual gas content is smaller. But the gas volume fraction in the hole caused by crack expansion is more remarkable after blasting.

The field test shows that the degree of permeability increase of decoupled charge is much better than that of coupled charge. The decoupling coefficient seriously affects the effect of cumulative blasting on coal seam fracture. The reasonable decoupling coefficient can greatly improve the permeability of coal seam and improve the gas drainage rate.

5. Conclusions

- (1) The model of cumulative blasting is established, and the numerical analysis shows that the air in the hole is the key to the formation of shaped charge jet and can provide space for the migration of shaped charge jet. The detonation energy is highly concentrated after blasting, and the coal in the direction of concentrating energy is destroyed preferentially relative to the coal in other directions. The coal element in the direction of concentrating energy shows obvious stress concentration, which promotes the development of coal cracks.
- (2) The influence of decoupling coefficient on the propagation characteristics of stress wave in cumulative blasting is analyzed. With the increase of decoupling coefficient, the stress on the hole wall

increases first and then decreases. Unlike the conventional blasting, the hole wall stress decreases with the increase of decoupling coefficient [15]. In the case of coupling charge, the energy transfer is the worst and the explosive utilization rate is the lowest. The air in the borehole can slow down the attenuation of stress wave in a certain range. But if the coupling coefficient is too large, the air in the borehole will consume too much explosive energy, which is also not conducive to the energy transfer.

- (3) The field tests of cumulative blasting with different decoupling coefficients are carried out. It is found that the average concentration of extracted gas is the lowest when it is coupling charge. There is an optimal decoupling coefficient in cumulative blasting, and the blasting effect is better when the decoupling coefficient is between 1.67 and 2 under this test condition. At this time, explosive energy can be fully utilized to improve gas extraction efficiency.

Data Availability

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

Conflicts of Interest

The authors declare that they have no conflicts of interest.

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