Numerical simulation of the effect of water-decoupling charge blasting on reservoir permeability enhancement

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ABSTRACT
In the development of deep resources, blasting fracturing technology is one of the most effective means to improve the permeability of otherwise low-permeability reservoirs, while the pressure rise time of shock wave and the charge structure are the key factors affecting the blasting effect. Thus, this paper first deduces a formula for the pressure rise time, based on which, the blast-induced damage evolution is numerically simulated, and the numerical result is consistent with the existing studies, which verifies the feasibility of the formula. In addition, the influence of decoupling coefficients (K) of different types of explosives (i.e. TNT, emulsion, and ANFO explosives) on the damage range of reservoir blasting is studied. It is found that under the blasting of different types of explosives, the damage evolution law of the reservoir is different, and appropriate explosives should be selected in combination with rock stratum parameters in actual construction. Finally, the increase in permeability and the drainage effect of the reservoir after blasting are quantified by numerical simulations. It is demonstrated that the average permeability increment of the reservoir under the action of TNT explosive is the largest, which is 3.08 (K = 4), while that under the action of emulsion explosive and ANFO explosive are 1.49 (K = 2) and 1.17 (K = 3) respectively; and the changes in reservoir permeability and drainage volume are coherent with the range of damage; the greater the range of damage, the greater the reservoir permeability and the greater the drainage efficiency.

1. Introduction
In rock formations of relatively low permeability, the efficiency of extracting oil, geothermal energy, uranium resource, and natural gas is similarly low. At present, to improve the extraction efficiency of deep resources, the method of explosive blasting...
is often used to increase the permeability of the reservoir by producing fractures associated with the blast impulse.

Blast-induced fractures are a key factor in reservoir permeability enhancement, to model blast-induced fractures, various computational methods have been developed and applied (Gharehdash et al. 2021a, 2021b). For examples, Zhu and Zhao (2021) have presented a peridynamics-based computational approach for modelling blast-induced rock fractures, which is shown to capture reasonably well the plastic material failure surrounding the borehole as well as the tensile cracks in both radial and circumferential directions. Gharehdash et al. (2020a, 2020b) found that smoothed particle hydrodynamics (SPH) method can predict qualitatively and quantitatively the blast-induced fractures, and calculate the permeability of blasting rock, with convincing results. Baranowski et al. (2020) have used the Johnson-Holmquist II (JH-2) model with parameters for a dolomite rock to simulate blast-induced rock fragmentation and compared it with the experimental results, the JH-2 method has a good application prospect.

However, because rock incurs damage due to blasting rather quickly, it is extremely difficult to study the damage mechanism of rock blasting (Yilmaz and Unlu 2013). Two basic forms of energy are released when explosives explode: shock energy (shock wave) and gas energy (detonation gas). The enormous impact energy of the shock wave can crush and break the rock surrounding the blasthole, forming an initial crack zone. Simultaneously, the detonation gas enters the initial crack zone, causing the cracks to continue to expand until the gas pressure dissipates. Yuan et al. (2019a) combined numerical simulation to analyze the effect of the shock wave and detonation gas. The results showed that detonation gas can expand existing cracks, increasing both the size of the fractured region and the crack aperture during the formation of fracture networks. Yang et al. (2016) used a laboratory model test method with PMMA. By combining the numerical simulation of LS-DYNA, the action-effect on the medium under loading with an explosion stress wave and detonation gas was studied. The results showed that the effect of the shock wave mainly led to the formation of microcracks in the crack zone, such that the effect of detonation gas was the major impetus of fracture formation in the crack zone. However, some scholars believe that the development of rock cracks can be reasonably predicted only by simulating the effect of shock wave, while ignoring the effect of detonation gas (Ma and An 2008; Shin et al. 2011; Yuan et al. 2019b).

In addition to shock wave and detonation gas, the charge structure of explosives also affects the formation of rock fracture networks. The installation method of explosives is called the charge structure, and the water-decoupling charge structure denotes that the diameter of the explosive is smaller than that of the blasthole, and the gap between the cartridge and the blasthole wall is filled with water. As the medium of energy transmission of explosives, water has the advantages of uniform energy transmission and reduced energy dissipation, which is conducive to the development of fracture networks. (Xahykaeb 1980; Zong and Meng 2003; Cui et al. 2010; Yuan et al. 2018). Numerous studies have shown that water-decoupling blasting is a critical stimulation technique in reservoir engineering. For examples, Zhu et al. (2008) used the AUTODYN code to discuss the effect of three coupling media (air, sandstone, and water) on rock fracturing. The results show that water coupling produces the
most extensive fracture zones because water serves as an excellent medium for the transmission of shock waves. Zong et al. (2011) used ANSYS/LS-DYNA to simulate the explosion stress field of rocks with different charge structures. The results indicate that water-decoupling blasting has higher energy utilization. Wang et al. (2018) verified through physical model tests that the role of water medium as an elastic buffer layer between explosive detonation products and rock reduces the energy dissipation caused by crushing the blasthole wall rock, increases energy transmission, and increases the scope of explosion.

It is generally believed that the shock wave, quasi-static effects of detonation gas, and decoupling charge structure are the most important factors affecting the outcomes of rock blasting. However, it is easy to ignore that the pressure rise time of the blasthole wall can also affect the formation of fractures (the pressure rise time refers to the time required for the pressure to reach the peak value when the shock wave reaches the blasthole wall). This phenomenon is difficult to monitor because the pressure rise time is very short. Therefore, Yilmaz and Unlu (2013), Ma and An (2008), Cho and Kaneko (2004), and Zhu et al. (2013) assumed pressure rise time values between 10 $\mu$s and 1,000 $\mu$s. In these studies, the authors held the peak pressure was constant to study the influence of different pressure rise times on the blasting fracture networks. While the pressure rise time can be controlled by the decoupling coefficient. When the decoupling coefficient increases, the peak pressure decreases, and the pressure rise time increases. It is easy to see that the above studies suffer from the following shortcomings: first, the pressure rise time of the blasthole wall is not constant in reality, instead, it decreases with the increase in the decoupling coefficient. Second, there is no theoretical formula for calculating the pressure rise time.

In summary, given the shortcomings of existing studies, this paper calculated the initial rock parameters based on the condition that the shock wave is continuous at the interface, and then, by incorporating the impulse and momentum conservation laws of the shock wave, this work deduced the theoretical formula for calculating the pressure rise time. For the same rock material, the higher the detonation velocity of the explosive is, the higher the peak pressure of the blasthole wall. Thus, the formation of rock fracture networks by explosives with different detonation velocities will be different. In this study, the charge density was held constant, and the explosive type was changed. Based on the deduced formula of the pressure rise time, the influence of the decoupling coefficient on the permeability enhancement effect of the reservoir under explosions with different explosive types was studied.

2. Basic parameters of numerical simulation

2.1. Basic parameters of explosives and rocks

According to the performance indicators of explosives introduced in the ‘Handbook of Blasting’ (Wang 2010), three kinds of explosives with different detonation velocities were selected in this study, namely, TNT explosive ($D_e = 5100$ m/s), emulsion explosive ($D_e = 3580$ m/s), and ANFO explosive ($D_e = 2800$ m/s). The charge radius and charge density are 0.05 m and 1000 kg/m$^3$, respectively. The peak value of the detonation wave ($P_d$) can be expressed as (Henrych 1979):
\[ P_d = \frac{\rho_e D_e^2}{(1 + \kappa)} \]  \hspace{1cm} (1)

where \( \rho_e \) is the charge density; \( D_e \) is the velocity of the detonation wave; and \( \kappa \) is the isentropic exponent of the detonation products, \( \kappa = 3 \). The calculation results are \( P_d (\text{TNT}) = 6.50 \times 10^9 \text{Pa} \), \( P_d (\text{Emulsion}) = 3.20 \times 10^9 \text{Pa} \), \( P_d (\text{ANFO}) = 1.96 \times 10^9 \text{Pa} \).

Bashibulake deposit is a very important sandstone-type deposit, which is located in Kashgar Sag in the west of China, Xinjiang Autonomous Regions. Geological survey results show that the main target ore bed is buried at a depth of 400 m. To obtain the physical property parameters of the ore bed, cylindrical rock samples are used for mechanical tests and wave velocity tests. The resulting physical property parameters are listed in Table 1.

### 2.2. Calculation of the initial parameters of rock with water-decoupling charge

Decoupling charge blasting can slow the propagation speed of shock wave and effectively improve the utilization rate of explosive energy. Many studies show that compared with the air decoupling charge structure, the water decoupling charge structure has a better buffering effect on the shock wave and a high energy utilization rate and is beneficial for improving the crushing effect. In previous studies, elastic wave theory was generally used to calculate the initial pressure of a transmitted wave \( P_x \) and its formula is (Zong and Meng 2003; Yuan et al. 2019c):

\[ P_x = \frac{2\rho_r C_r}{\rho_r C_r + \rho_w D_w} P_d \]  \hspace{1cm} (2)

where \( C_r \) is the P-wave velocity of the rock; \( D_w \) represents the propagation velocity of the shock wave in water; \( P_d \) represents the peak value of the detonation wave; \( \rho_r \) and \( \rho_w \) represent the densities of the rock and water, respectively.

The above computational method is established based on elastic wave motion theory, in which the whole rock is regarded as an elastic medium and the shock wave in the rock is deemed an elastic stress wave. However, because the shock wave can crush the rock around the blasthole and dissipate the energy quickly, it is unreasonable to calculate the initial pressure of the transmitted wave using elastic wave theory (Xahykaeb 1980; Yuan et al. 2019c).

**Figure 1** is a schematic diagram of shock wave propagation in a water medium. The blasting wave is transmitted to the water medium and then to the blasthole wall, forming the initial pressure of the blasthole wall, where \( P_w \), \( \rho_w \), \( u_w \), and \( N_w \) respectively, represent the interface pressure, density, particle velocity, and wave front velocity when the shock wave propagates a distance of \( x \) in the blasthole. \( P_0 \), \( \rho_0 \), and \( u_0 \) represent the initial pressure, initial density, and initial particle movement velocity of

### Table 1. Physical and mechanical parameters of sandstone.

| Density \((\text{kg} \cdot \text{m}^{-3})\) | Compressive strength (MPa) | Poisson’s ratio | Elasticity modulus (MPa) | P-wave velocity \((\text{m} \cdot \text{s}^{-1})\) | GSI | m, |
|-----------------|-----------------|----------------|-----------------------|-------------------|------|-----|
| 2760            | 88.1            | 0.09           | 12.9                  | 2359              | 100  | 18  |
water without disturbance in the water medium, respectively. When the water is in a static state, $P_0 \approx 0$, $u_0 = 0$, and $\rho_0 = 1000 \text{ kg/m}^3$. When the shock wave transmission distance is $R$, the initial parameters on the interface between water and rock are $P_x$, $\rho_x$, $u_x$, and $N_x$, representing the transmission pressure of the blasthole wall, the density, particle velocity, and wave front velocity after the compression of the blasthole wall, respectively.

This section does not use elastic wave theory, but according to the interface continuity condition of the shock wave, the wave equation of the energy conservation law, the law of mass conservation, the law of momentum conservation, and the state equation of rock and water, the initial parameters of the explosion shock wave propagating to the blasthole wall through the water medium are obtained. Henrych (1979) gave integral forms of the conservation laws for any shock wave front:

\[ \text{Momentum conservation law} : u_x = \left(1 - \frac{\rho_0}{\rho_x}\right)N_x \]  
\[ \text{Mass conservation law} : P_x - P_0 = \rho_0 u_x N_x \]  
\[ \text{Energy conservation law} : Q_0 = E_x - E_0 + \frac{P_x + P_0}{2} \left(\frac{1}{\rho_x} - \frac{1}{\rho_0}\right) \]  

where $E_0$ and $E_x$ are initial specific internal energy and specific internal energy after blasting of explosive. $Q_0$ is the specific energy of explosives, if the medium does not release energy, $Q_0 = 0$.  

Figure 1. Schematic diagram of blasting shock wave propagation in a water medium.
According to Eqs. (3) and (4), the expression of the particle velocity of the shock wave front in any medium can be obtained as:

\[ u_x = \sqrt{(P_x - P_0) \left( \frac{1}{\rho_0} - \frac{1}{\rho_x} \right)} \]  

(6)

where \( P_0 \) is considerably less than \( P_x \), and can be ignored in the calculation.

2.2.1. Initial parameters of the interface between water and detonation products

(1) When the decoupling medium is water and \( \bar{\tau} = 1 \), \( \bar{\tau} \) is the relative distance, \( \bar{\tau} = R/R_0 \), where \( R_0 \) is the radius of the cartridge and \( R \) is the distance from the explosion centre. For the water-decoupling structure, the continuity condition of the velocity on the interface between the water and detonation products is (Henrych 1979; Wang et al. 2008; Yuan et al. 2019c):

\[ u_w = u_D + u_R \]  

(7)

where \( u_w \) is the velocity of the detonation product on the water interface; \( u_D \) is the velocity of the detonation wave and \( u_R \) is the particle velocity of the reflected wave, which is an expansion. \( u_D \) and \( u_R \) can be expressed as (Henrych 1979; Wang et al. 2008; Yuan et al. 2019c):

\[ u_D = \frac{D}{\kappa + 1} \]  

(8)

\[ u_R = \frac{2\kappa D}{\kappa^2 - 1} \left[ 1 - \left( \frac{P_w}{P_d} \right)^{(\kappa-1)/2\kappa} \right] \]  

(9)

When Eqs. (8) and (9) are substituted into Eq. (7), \( u_w \) can be expressed as:

\[ u_w = \frac{D}{\kappa + 1} \left\{ 1 + \frac{2\kappa}{\kappa - 1} \left[ 1 - \left( \frac{P_w}{P_d} \right)^{(\kappa-1)/2\kappa} \right] \right\} \]  

(10)

where \( \kappa \) is the isentropic exponent of the detonation products, \( \kappa = 3 \);

The equation of the state of water can be expressed as (Xahykaeb 1980):

\[ P_w = A' \left( \frac{\rho_w}{\rho_0} \right)^{\kappa'} - 1 \]  

(11)

where \( \kappa' \) is the isentropic exponent of water and \( A' \) is a constant. When \( P_w > 2.5 \times 10^9 \) Pa, \( A' = 425 \times 10^6 \) Pa and \( \kappa' = 6.29 \); when \( P_w < 2.5 \times 10^9 \) Pa, \( A' = 304.7 \times 10^6 \) Pa and \( \kappa' = 7.15 \). For water, \( \rho_0 \) is equal to 1000 kg/m³.
From Eq. (3), the propagation velocity of the shock wave front is given as follows:

\[ N_w = \frac{u_w}{1 - \rho_0/\rho_w} \] (12)

Thus, according to Eq. (6) and Eqs. (10)–(12) the initial parameters of the interface between water and detonation products can be obtained, and the data are listed in Tables 2–4.

(2) When the relative distance \( \bar{r} > 1 \), the shock wave propagates into the water and gradually decays. Wang et al. (2008) proposed a method to evaluate the attenuation law of the shock pressure in water for a column charge:

\[ \Delta P = \begin{cases} A_1/\bar{r}^{2.49} ; & 1 \leq \bar{r} \leq 2 \\ A_2/\bar{r}^{1.45} ; & 2 < \bar{r} \leq 5 \\ A_3/\bar{r}^{0.63} ; & 5 < \bar{r} \leq 240 \end{cases} \] (13)

where \( \Delta P \) is the overpressure induced by the shock wave in the water and \( A_1, A_2, \) and \( A_3 \) are the overpressures at the origins of the three sections of this continuous piecewise function. Take TNT explosive as an example, when \( \bar{r} = 1 \), \( A_1 = 5.80 \times 10^9 \) Pa, and when \( \bar{r} = 2 \), \( A_2 = A_1 \times 2^{1.45}/2^{2.49} = 2.82 \times 10^9 \) Pa. According to Eq. (13), the shock pressures of the water at the interface of the water and blasthole wall for \( K = 2, 3, \) and \( 4 \) are \( 1.03 \times 10^9 \) Pa, \( 5.73 \times 10^8 \) Pa, and \( 3.78 \times 10^8 \) Pa, respectively. \( \rho_w \), \( u_w \), and \( N_w \) can be obtained from simultaneous Eq. (6) and Eqs. (10)–(12). The calculated data are listed in Tables 2–4.

### 2.2.2. Initial parameter calculation of the blasthole wall

1. For the coupling charge, the continuity condition of the velocity on the interface between the rock and detonation products can be expressed as:

\[ u_x = u_D - u_S \] (14)

where \( u_S \) is the particle velocity of the reflected wave, which is a compressive wave; According to Eq. (6), the interface between the detonation wave and the rock can be expressed as (Wang et al. 2008):

\[ u_S = \sqrt{(P_x - P_d) \left( \frac{1}{\rho_d} - \frac{1}{\rho_x} \right)} \] (15)
When Eqs. (8) and (15) are substituted into Eq. (14), the particle velocity at the interface between the detonation product and the rock is:

\[ u_x = \frac{D}{\kappa + 1} \left[ 1 - \sqrt{2\kappa - \frac{P_x/P_d - 1}{\sqrt{(\kappa + 1)P_x/P_d + (\kappa - 1)}}} \right] \]  

(16)

Based on the state equation for rock-like material, \( P_x \) can be expressed as (Xahykaeb 1980):

\[ P_x = A'' \left( \frac{P_x}{\rho_0} \right)^{\kappa''} - 1 = \frac{\rho_0 C_0^2}{n} \left( \frac{P_x}{\rho_0} \right)^{\kappa''} - 1 \]  

(17)

where \( \rho_0 \) is the initial density of the rock; \( \kappa'' \) is the isentropic exponent of the rock; \( C_0 \) is the \( P \)-wave velocity of the rock, and \( n \) is a constant. When \( P_x > 4 \times 10^9 \text{ Pa} \), \( \kappa'' = 5 \) and \( n = 5.5 \). When \( P_x < 4 \times 10^9 \text{ Pa} \), \( \kappa'' = 3 \) and \( n = 3 \). According to Eqs. (3), (6), (16), and (17), the associated parameters of the coupling structure on the blasthole wall are listed in Tables 5–7.

1. In the case of decoupling charge, the shock wave energy is transferred from water to rock. According to Eq. (16), the particle velocity \( u_x \) on the interface between water and rock can be deduced as:

\[ u_x = \frac{N_w}{\kappa' + 1} \left[ 1 - \sqrt{2\kappa'} - \frac{P_x/P_w - 1}{\sqrt{(\kappa + 1)P_x/P_w + (\kappa - 1)}} \right] \]  

(18)

Similarly, according to Eqs. (3), (6), (17), and (18), the associated parameters of the rock of the blasthole wall can be acquired for different water-decoupling structures. In summary, the parameters of the shock wave at the wall of the blasthole (\( P_x, \rho_x, u_x, N_x \)) for \( K = 1, 2, 3, \) and \( 4 \) are all shown in Tables 5–7.
2.3. Calculation of pressure rise time

The simulation of blasting load can apply a radial pressure on the surface of the blasthole, but due to the transient nature of the explosion, it is difficult to measure the change in the blasthole wall pressure using instruments. This can usually be calculated using the following impulse function (Cho and Kaneko 2004; Zhu et al. 2013):

\[ P_t = \frac{P_x \zeta}{t_r} \left( e^{-\alpha t} - e^{-\beta t} \right) \]  

(19)

where \( P_t \) is the shock pressure at time \( t \); \( P_x \) represents the peak pressure of the blasthole; \( \alpha \) and \( \beta \) are constants. For the convenience of expressing the rising and falling phases, \( t_r \) and \( \zeta \) (\( \beta/\alpha = 1.5 \)) are expressed by the following formulas:

\[ t_r = \frac{1}{\beta - \alpha} \ln \left( \frac{\beta}{\alpha} \right) \]  

(20)

\[ \zeta = \frac{1}{e^{-\alpha t_r} - e^{-\beta t_r}} \]

(21)

where \( t_r \) is the pressure rise time. Many published books cite a pressure rise time of a few milliseconds, but there are also measurements of the blasthole wall that show a pressure rise time between 20 \( \mu \)s and 150 \( \mu \)s (Saharan et al. 2006). Current studies cannot provide accurate methods to obtain the pressure rise time, and many scholars use the assumed pressure rise time to study its influence on the blasting effect.

Based on the study of Yuan et al. (2019c), this paper obtained the calculation formula of pressure rise time based on the initial parameters \( (P_x, \rho_x, u_x, N_x) \) of the shock wave incident on the rock and the principles of impulse and momentum.
conservation of the shock wave. Figure 2 is a schematic diagram for calculating the pressure rise time of the shock wave on the blasthole wall, where $r$ represents the blasthole radius, $\rho_0$ and $\rho_x$ represent the initial density and compressive density of the rock, respectively, $P_x$ is the peak pressure of the shock wave, $u_x$ is the maximum particle velocity in the blasthole wall, $N_x$ is the velocity of the wavefront, $C_p$ represents the P-wave velocity of the rock, and the shock wave travels a distance $\Delta$ within time $t_r$. To simplify the calculation, suppose that the particle velocity decays linearly to zero. During the explosion, the shock wave first propagates at velocity $N_x$, which is greater than the P-wave velocity of rock, and then decays into an elastic wave, which propagates at the P-wave velocity of the rock (Xahykaeb 1980). Then, the propagation distance $\Delta$ within $t_r$ can be approximated using the average value of $[(N_x + N_x)/2]t_r$, and the density of rock within $t_r$ can be approximated by $[(\rho_x + \rho_0)/2]t_r$. The derivation process is as follows:

The impulse, $I_p$, resulting from a shock wave within $t_r$ can be expressed as:

$$I_p = 2\pi r \times 1 \times P_x t_r = 2\pi r P_x t_r$$  \hspace{1cm} (22)

The propagation distance, $\Delta$, of the shock wave in rock within $t_r$ can be expressed as:

$$\Delta = \left(\frac{N_x + C_p}{2}\right) t_r$$  \hspace{1cm} (23)

The velocity boundary conditions in time $t_r$ can be expressed as:

$$u(r) = u_x, u(r + \Delta) = 0$$  \hspace{1cm} (24)

Assuming that the particle velocity decays linearly, $x$ represents the distance from the hole. Therefore, the particle velocity at any point in the range $r \leq x \leq r + \Delta$ can be expressed as:
The momentum of the particle in the range of \( r \leq x \leq r + \Delta \) can be expressed as:

\[
M_p = \int_r^{r+\Delta} \left[ \frac{\rho_x + \rho_0}{2} \right] (2\pi x) \cdot u(x) \, dx
\]  \hspace{1cm} (26)

According to \( I_p = M_p \), the expression of \( t_r \) is finally shown as:

\[
t_r = \frac{48P_x r - 6(\rho_x + \rho_0)u_x r(N_x + C_p)}{(\rho_x + \rho_0)u_x (N_x + C_p)^2}
\]  \hspace{1cm} (27)

According to Tables 5–7 and Eq. (27), the pressure rise time of the TNT explosive, emulsion explosive, and ANFO explosive when the decoupling coefficient is \( K = 1, 2, 3, \) and 4 can be calculated, and the results are shown in Table 8.

Table 8 shows that the pressure rise time of the three types of explosives increases with an increasing decoupling coefficient. The TNT explosive with a high detonation velocity has the shortest pressure rise time, while the ANFO explosive with a low detonation velocity has the longest pressure rise time, and the emulsion explosive falls in the middle. The pressure rise time calculated in Table 8 ranges from approximately 50 \( \mu s \) to 300 \( \mu s \), which is closely related to the explosive detonation velocity, charge density, charge diameter, and blasthole diameter. Combined with previous research results, the assumed pressure rise time is between 10 \( \mu s \) and 1000 \( \mu s \), which is consistent with the results calculated in this paper. Using Eqs. (19)–(21) and (27), the pressure of the blasthole wall \( (P_t) \) at different times can be calculated. There are 12 groups of pressure–time history curves of the blasthole wall for three types of explosives and four kinds of decoupling coefficients, as shown in Figure 3.

### 3. Numerical simulations

In this study, the finite difference software \( FLAC^{3D} \) was used to simulate the effect of decoupling charge coefficients on reservoir permeability enhancement for different explosives. The model size is \( 1 \times 8 \times 8 \) m (Length \times Width \times Height), located at a depth of 400 m below the surface. And the blasthole radii corresponding to decoupling coefficients \( K = 1, 2, 3, \) and 4 are 0.05 m, 0.1 m, 0.15 m, and 0.2 m, respectively. The whole model is meshed as 49,920 elements, with a fixed normal displacement at the bottom of the model, and the remaining five surfaces are constrained by stress. The vertical stress \( (\sigma_z) \) applied on the top of the model is approximately equal to the
gravity of the overlying strata, i.e. \( \sigma_x = \sigma_y = \sigma_z = 2760 \times 10 \times 400 = 11.04 \) MPa, as shown in Figure 4. During the dynamic calculation, the displacement fixed boundary condition of the model is changed to a viscoelastic boundary condition to absorb the external traveling wave from the blasthole explosion to the model boundary.

### 3.1. Failure criterion for the rock

The Hoek–Brown failure criterion is widely used to describe the nonlinear failure characteristics of the rock. After years of research and improvement, this criterion has become one of the most widely used failure criteria in rock engineering (Yuan et al. 2020). The ‘generalized’ Hoek–Brown criterion, adopting the convention of positive compressive stress, is:

\[
\sigma_1 = \sigma_3 + \sigma_{ci} \left( m_b \frac{\sigma_3}{\sigma_{ci}} + s \right)^\alpha
\]

(28)

where \( \sigma_1 \) and \( \sigma_3 \) are the major and minor effective principal stresses; \( \sigma_{ci} \) is the uni-axial compressive strength of the intact rock; and \( m_b, s \) and \( \alpha \) are material constants that can be related to the Geological Strength Index (GSI) and rock damage, which can be acquired by (Hoek et al. 2002; Erik 2012):

\[
\begin{align*}
    m_b &= \exp \left( \frac{GSI - 100}{28 - 14D} \right) m_i \\
    s &= \exp \left( \frac{GSI - 100}{9 - 3D} \right) \\
    \alpha &= 0.5 + \frac{1}{6} \left[ \exp \left( -\frac{GSI}{15} \right) - \exp \left( -\frac{20}{3} \right) \right]
\end{align*}
\]

(29)
where \( m_i \) represents the rock type and hardness, which ranges from 1 to 40; GSI is the geological strength index, used for depicting the structural features of the discontinuities in the rock, generally varying from 5 (for a highly fractured and damaged rock) to 100 (for intact rock); and D reflects the impact of the rock subjected to stress disturbance, ranging from 0 (for an undisturbed, in situ rock) to 1 (for a disturbed rock). In addition, the unconfined compressive strength is given by \( \sigma_c = \sigma_{ci} \frac{S^x}{m_b} \), and the tensile strength is given by \( \sigma_t = -s \sigma_{ci} / m_b \) (see Figure 5).

To verify the accuracy of numerical simulation, this study refers to the data in Yuan’s literature as the calibration object. Yuan et al. (2019c) have combined the physical modelling experiments and discrete element method to study the effect of water-decoupling charge structure on blasting stress distribution and the morphology of blast-induced fractures. The model with the water-decoupling coefficients of 3.29 is selected from Yuan’s literature to calibrate the proposed method in this study. The size of the model is 0.8 × 0.8 m (Width × Height), and the hole diameter at the centre of the model is 0.048.

Figure 6 shows the Failure area surrounding the blasthole caused by the PFC3D and FLAC3D codes. In this paper, the finite difference method is used for simulation. Although the size of the mesh will affect the accuracy of the numerical simulation, the overall law is not affected. As shown in Figure 6, the plastic zones range obtained by the method in this paper is very close to the fracture distribution range obtained by the discrete element method, and the relative error is about 3%, which indicates that the presented method could properly simulate the effect of water-decoupling charge blasting on reservoir permeability enhancement.

3.2. Definition of damage variable

To study the law of rock damage evolution, we must first define the damage variable. The appropriate damage variable is the basis and premise of subsequent analysis and research. Rock damage refers to the process in which new microcracks are produced or the original microcracks are activated under the action of an external load, and
the old and new cracks and pores gradually converge to produce rock failure through crack growth, stress concentration, and strength decline. In the numerical software developed by Itasca Consulting Group, Inc., the failure plasticity index based on the increase in regional stress-plastic flow provides possible yield criteria in various forms (Kaveh Ahangaran et al. 2022). In the process of blasting, unrecoverable plastic deformation occurs in the rock, the rock is damaged, and cracks occur. At this time, all the external blasting load work is converted into plastic work. Based on the energy dissipation method, the rock blasting damage variable is defined as the integrity index, $\eta$, related to energy dissipation, which can be expressed as (Wei et al. 2022):

$$
\eta = 1 - \frac{W_p}{W}
$$  \hspace{1cm} (30)
where $W_p$ is the energy dissipation of plastic deformation and $W$ is the total energy input of the blasting load. When all the input energy is converted into plastic work, $W_p = W$ and $\eta = 0$ (rock damage, resulting in cracks). When all the input energy is converted into elastic work, $W_p = 0$ and $\eta = 1$ (the rock is not damaged, and there is no crack), so $0 \leq \eta \leq 1$. Based on Eq. (30), the calculation results of the integrity index of each element of the reservoir are shown in Figure 8 and Figures 11–13.

### 3.3. Effect of different pressure rise times on the damage range

Previous studies have shown that a short pressure rise time tends to produce a large crushed zone and a large number of short cracks; a long pressure rise time tends to produce a smaller crushed zone and longer radial cracks. Considering the effect of blasting permeability enhancement and damage control, the crushed zone around the blasthole should be minimized, and the length of the radial crack should be increased (Cho and Kaneko 2004; Ma and An 2008; Yilmaz and Unlu 2013; Zhu et al. 2013). To verify the accuracy of the defined integrity index, first, the laws of crack propagation and permeability under the same peak pressure and different pressure rise times are studied in this paper. Taking ANFO explosive $K=3$ as an example, the peak pressure, $P_x = 4.16 \times 10^8$ Pa, with $t_r = 189 \mu s$ can be obtained from Tables 7 and 8. Then, we set three groups of pressure rise times, $t_r = 10 \mu s$, $t_r = 50 \mu s$, and $t_r = 100 \mu s$, as shown in Table 9. Eqs. (19)–(21) were used to calculate pressure-time history curves of the same peak pressure at different pressure rise times, as shown in Figure 7.

Figure 8 compares the blasting damage nephogram of rock under different pressure rise times. Blue represents the rock damage area, and red represents the rock integrity area. When the rise time is very short ($t_r = 10 \mu s$), the damage range of rock is also very small, and there are no long cracks (Figure 8a). With the increase in pressure rise time, the damage range increases gradually, and more long cracks are produced. But this law does not always explicitly hold. When the pressure of the shock wave is less than the dynamic compressive strength of rock, the damage range
is reduced. This is also consistent with the research of Yilmaz and Unlu (2013), Ma and An (2008), Cho and Kaneko (2004), Zhu et al. (2013), and other scholars and verifies the accuracy of defining damage variables in Section 3.2.

The damage range of rock is related to the amount of energy dissipated (Ma and An 2008). To analyze the relationship between the damage range and energy

Figure 8. Blasting damage diagram of the same peak value with different rise times.

Figure 9. Peak attenuation curve of the explosion stress wave with different rise times.
dissipated, the peak value of the explosion stress wave at 1 m, 2 m, 3 m, and 4 m from
the centre of the blasthole is monitored, and the expression of the peak value of the
explosion stress wave at different distances is obtained by fitting with the exponential
function \[ P = P_d \frac{\alpha}{r^\alpha} \] where \( \alpha \) is the attenuation index (Eq. (31)). The larger the

Figure 10. Attenuation index broken line with different pressure rise times.

Figure 11. Blasting damage diagram of the TNT explosive.
attenuation index, the faster the energy is attenuated. According to this relationship, the exponential relationship curve of the peak attenuation of the explosion stress wave with the propagation distance (Figure 9) and the broken line diagram of the attenuation index, \( a \), with the pressure rise time, \( t_r \), are obtained (Figure 10).

\[
P = \frac{P_d \bar{r}^{-1.33}}{C_0} t_r = 10 \mu s \\
P = \frac{P_d \bar{r}^{-1.23}}{C_0} t_r = 50 \mu s \\
P = \frac{P_d \bar{r}^{-1.16}}{C_0} t_r = 100 \mu s \\
P = \frac{P_d \bar{r}^{-1.06}}{C_0} t_r = 189 \mu s
\]

where \( P \) is the peak value of the explosion stress wave at different distances; \( P_d \) is the peak value of the detonation wave stress wave of the ANFO explosive, \( P_d = 1.96 \times 10^9 \) Pa; \( \bar{r} \) is the relative position from the centre of the blasthole.

Figures 9 and 10 clearly describe the relationship between rise time and energy decay. When the rise time \( t_r = 10 \) \( \mu s \), the attenuation index is the largest. With the increase in rise time, the attenuation index decreases gradually, when \( t_r = 189 \) \( \mu s \), the attenuation index reaches a minimum. This explains the variation law of the rock blasting damage range in Figure 8. When the pressure rise time is very small, the shock wave acts on the rock to form the damage area, but this energy is quickly consumed and cannot continue to generate cracks. With the gradual increase in pressure
rise time, the rock has a higher utilization rate of the explosion energy increasing probability to produce cracks, which increases the damage range.

### 3.4. Influence of decoupling coefficient on the damage range

Through an outdoor blasting test, Wang and Li (2010) found that there is an optimal decoupling coefficient in decoupling charge blasting and determined that the optimal decoupling coefficient is between 2 and 3 under specified test conditions. Yuan et al. (2019c) also verified this point through experiments and numerical simulations. However, it has not been studied whether the optimal decoupling coefficient varies under the action of explosives with different detonation velocities. In this study, a

![Figure 13. Blasting damage diagram of the ANFO explosive.](image)

#### Table 9. Different pressure rise time parameters.

| $P_x$ (Pa) | $\alpha$ | $\beta$ | $\gamma$ | $t_r$ (µs) |
|------------|---------|---------|---------|-----------|
| $4.16 \times 10^8$ | $8.11 \times 10^4$ | $1.22 \times 10^5$ | $6.75$ | $10$ |
| $4.16 \times 10^8$ | $1.62 \times 10^4$ | $2.43 \times 10^4$ | $6.75$ | $50$ |
| $4.16 \times 10^8$ | $8.11 \times 10^3$ | $1.22 \times 10^4$ | $6.75$ | $100$ |
| $4.16 \times 10^8$ | $4.30 \times 10^3$ | $6.45 \times 10^3$ | $6.75$ | $189$ |

#### Table 10. Attenuation index of different decoupling coefficients with different explosives.

| Explosive type | $P_d$ (Pa) | $K = 1$ | $K = 2$ | $K = 3$ | $K = 4$ |
|----------------|---------|--------|--------|--------|--------|
| TNT explosives | $6.50 \times 10^9$ | $1.32$ | $1.24$ | $1.19$ | $1.06$ |
| Emulsion explosives | $3.20 \times 10^9$ | $1.29$ | $1.06$ | $1.15$ | $1.22$ |
| ANFO explosives | $1.96 \times 10^9$ | $1.16$ | $1.07$ | $1.05$ | $1.10$ |
TNT explosive, an emulsion explosive, and an ANFO explosive were used for the numerical simulation of blasting to study the variation rule of the damage range. Figures 11–13 show the blasting damage diagram under the corresponding explosives.

Figure 11 shows the blasting damage diagram of the TNT explosive. When $K = 1$, the damage range of the model is very small, and the damage is concentrated around the blasthole. With the increase in decoupling coefficient $K$, the damage range increases, and it is no longer concentrated around the blasthole but extends outward radially. When $K = 4$, the damage range and length reach their maximum values. Figure 12 shows the blasting damage diagram of the emulsion explosive. When $K = 1$, the damage range is also very small and concentrated around the blasthole. When $K = 2$, the damage range and length reach the maximum. Then, with the increase in decoupling coefficient $K$, the damage range decreases gradually. Figure 13 shows the blasting damage
diagram of the ANFO explosive. When $K = 1$, the law is the same as in Figures 11 and 12. However, when $K = 2$ and $K = 3$, there is little difference in the resulting damage range. When $K = 4$, the damage range is less than that of $K = 2$ and $K = 3$.

The damage range and length of the TNT explosive increase with increasing decoupling coefficient, and there is no optimal decoupling coefficient in the range of $K = 1 \sim 4$. The damage range and length of the emulsion explosive are the largest when $K = 2$. According to the later attenuation index analysis, the damage range of the emulsion explosive is the largest when $K = 3$. By analyzing the reasons, although the peak pressure of the explosion shock wave decreases with increasing decoupling coefficient, due to the high detonation velocity of the TNT explosive, its detonation pressure is the largest under the same charge density. When $K = 4$, the peak pressure of the TNT explosion shock wave is still greater than the dynamic compressive

Figure 16. Peak attenuation curve of the emulsion explosion stress wave.

Figure 17. Attenuation index broken line of different decoupling coefficients in emulsion explosive blasting.
strength of rock, so it can cause a large range of damage. However, the detonation pressure of the emulsion explosive and ANFO explosive is much lower than that of the TNT explosive. In the range of $K = 1 \sim 4$, there exists an optimal decoupling coefficient such that the peak pressure of the explosion shock wave is equal to the dynamic compressive strength of rock, and the damage range is the largest. If the decoupling coefficient continues to increase, the peak pressure of the explosion shock wave will be lower than the dynamic compressive strength of the rock, and the damage range will be reduced.

Similarly, to analyze the relationship between the damage range and energy dissipation, the exponential function $P = P_d e^{-r^2}$ is used to fit the expression of the peak value of the explosion shock wave of the three explosives at different distances from the centre of the blasthole, and the attenuation index is listed in Table 10. According to the exponential function, the exponential relationship curves of the peak value of the explosion stress wave of the three explosives with the propagation distance (Figures 14, 16, and 18) and the broken line diagrams of the attenuation index with the decoupling coefficient $K$ (Figures 15, 17, and 19) are obtained.

Through the energy attenuation analysis, when the TNT explosive is selected for blasting, the attenuation index decreases with increasing decoupling coefficient, indicating that the energy attenuation is slower and slower, and the utilization rate of rock energy is improved. When $K = 4$, the attenuation index is the smallest, and the damage range is the largest, which corresponds to the damage range in Figure 11. When an emulsion explosive is used for blasting, when $K = 2$, the attenuation indices are $1.07$ and $1.05$, respectively, and there is little difference between the results, which is consistent with the law in Figure 13.

The slower the energy attenuation is, the longer the action time of the shock wave in rock will be, which is conducive to the formation of the rock damage area and the development of cracks and promotes the effect of blasting permeability enhancement. From

Figure 18. Peak attenuation curve of the ANFO explosion stress wave.
the above analysis, it can be seen that the damage range of rock is not only related to the
decoupling coefficient and decoupling medium but also depends on the type of explo-
sive because the detonation velocity and density determine the detonation pressure.

3.5. Effect of blasting on reservoir permeability

To estimate the effect of blasting on improving reservoir permeability, the
Carman–Kozeny formula is used to relate permeability and volumetric strain (Bear
1988; Taron et al. 2009; Cappa and Rutqvist 2011):

\[
\phi = 1 - (1 - \phi_0)e^{-\frac{\nu}{\nu_0}}
\]  

(32)
where $n$ is the permeability increment caused by blasting; $k_i$ and $\phi_i$ are the initial permeability and initial porosity of the reservoir before blasting; $k$ and $\phi$ are the permeability and porosity of the reservoir after blasting; $\varepsilon_v$ is the volumetric strain.

To describe the influence of blasting on reservoir permeability, the average permeability increment $\bar{\xi}$ of the reservoir after blasting can be studied. After blasting, the permeability increment $\xi_i$ and volume $V_i$ of each element are extracted and defined as (Wei et al. 2022):

$$
\tilde{\xi} = \frac{k}{k_i} = \left(1 - \phi_i\right)^2 \left(\frac{\phi}{\phi_i}\right)^3
$$

(33)
Figures 20 and 21 show the changes in the average permeability increment after blasting according to Eq. (34) based on the pressure rise time and the decoupling coefficient. Combined with Figures 8 and 20, it can be seen that under the same peak pressure, the damage range and permeability increment of the reservoir gradually increase with increasing pressure rise time. Figure 21 shows that the permeability increment varies with the decoupling coefficient for different explosives. However, the change in permeability is consistent with the change law of the blasting damage range in Section 3.4; that is, under the action of the TNT explosive, the permeability
of the reservoir gradually increases with the increase in the decoupling coefficient, but under the action of the emulsion explosive and ANFO explosive, the reservoir permeability is the largest when $K = 2$ and $K = 3$, respectively, and the overall permeability is in the order of $\zeta_{TNT} > \zeta_{Emulsion} > \zeta_{ANFO}$.

3.6. Effect of blasting on reservoir drainage volume

Referring to the research of Zhu et al. (2013, 2016), this paper conducted a numerical simulation of gas drainage from a reservoir after blasting permeability enhancement. The model uses the result documented after blasting in Section 3.4 and assumes that the initial permeability is $1 \times 10^{-13}$ m$^2$, the permeability of the reservoir after blasting can be calculated from Eq. (33). The initial gas pressure in the sample is 5 MPa, and the set pressure of the blasthole is 0.1 MPa, and the boundary is set as the impermeable boundary. After 100 hours of drainage, the curve of the residual content of reservoir gas under the action of the three explosives is shown in Figures 22–24.

In Figures 22–24, the gas content of the reservoir before blasting decreases to 98.9% after 100 h of drainage. Under the influence of the TNT explosive, at $t = 100$ h, the gas content reduces to 93.1%, 88.2%, 78.5%, and 68.6% for decoupling charges of $K = 1$, $K = 2$, $K = 3$, and $K = 4$, respectively. Under the action of the emulsion explosive and ANFO explosive, the drainage efficiency is the highest when the decoupling coefficient is $K = 2$ and $K = 3$, and after 100 h, the reservoir content is reduced to 89.6% and 94.0%, respectively. Because the damage degree of the reservoir decreases gradually with increasing distance from the blasthole, the permeability near the blasthole is the largest; that is, the farther away from the centre of the blasthole, the smaller the permeability. This is also the reason why the gas content decreases rapidly at the beginning and gradually slows. The results show that increasing the damage range of the reservoir can not only improve its permeability but also can increase the efficiency of gas drainage.

4. Conclusions

Explosive blasting is an important technical means to improve reservoir permeability. It is highly important that we develop a deep understanding of the damage area and permeability change mechanism caused by explosive blasting. Based on the calculation formula of the pressure rise time, this paper simulates the influence of blasting on the permeability enhancement effect of rock reservoirs. Through a comprehensive analysis of the damage range, permeability enhancement coefficient, and reservoir drainage volume before and after blasting, the following conclusions can be drawn:

a. Under the same peak pressure, the attenuation of the explosion stress wave decreases with increasing pressure rise time. The slower the energy attenuation is, the larger the influence range of the stress wave and the larger the damage range.

b. When studying the influence of different decoupling coefficients on the damage range of rock, due to the high detonation velocity and high detonation pressure of the TNT explosive, the peak pressure of the shock wave is always greater than the dynamic compressive strength of rock within the research range of this paper,
and because the energy attenuation decreases with increasing decoupling coefficient, the damage range of rock increases gradually. When $K = 3$ and $K = 2$, the energy attenuation of the emulsion explosive and ANFO explosive is the slowest, and the damage range is the largest.

c. The changes in reservoir permeability and drainage volume are consistent with the damage range. The larger the damage range is, the greater the permeability increment of the reservoir and the higher the drainage efficiency.

Explosive blasting is a very complex process. The effect of reservoir blasting on permeability enhancement not only depends on the properties of the rock but is also closely related to the pressure rise time, decoupling coefficient, and type of explosive. This study will provide some guidance for actual construction.

Competing interests

The authors have declared that we have no financial and personal relationships with other people or organizations that can inappropriately influence our work.

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Data availability statement

Some or all data, models, or code generated or used during the study are available from the corresponding author by request.

Disclosure statement

No potential conflict of interest was reported by the authors.

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