Research Article

Blast Induced Crack Propagation and Damage Accumulation in Rock Mass Containing Initial Damage

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Blast induced rock mass damage and crack propagation play important roles in structure safety and stability in mining, quarrying, and civil constructions. This paper focuses on the effect of small blasthole diameter blast on crack propagation and damage accumulation in water-bearing rock mass containing initial damage composed of inherent geological discontinuities and previous multiblast induced damage. To elucidate this effect, theoretical analysis of calculation method for several important blast influencing factors is firstly presented. Secondly, definition of a practical damage variable using ratio of longitudinal wave velocity in rock mass before blast occurrence to that after blast occurrence and derivation of a damage accumulation calculation equation accounting for initial damage and blasting effect are described. Lastly, a detailed description of the conducted in situ blast tests and plan layout of the sonic wave monitoring holes is reported. The results indicate that blast activates and then extends the initial cracks in rock mass, leading to accumulation of rock mass damage. The rock mass damage accumulation can be conveniently quantified using the proposed damage variable. When the damage variable reaches its threshold of 0.19, occurrence of damage in the surrounding rock mass is indicated. It is also found that the blast induced rock mass damage extent and the blast induced vibration velocities decrease nonlinearly with increasing the distance between blast source and monitoring position.

1. Introduction

The application of the drill and blast method (DBM) remains a preferred rock mass excavation method worldwide in engineering practice such as underground caverning, mining, quarrying, tunneling, and dam construction. Despite the advantages of DBM including low construction cost, short construction period, wide acceptability, and broad applicability [1], DBM holds inevitably the disadvantages of deteriorating the surrounding rock mass and creating adverse environmental effects such as ground vibration, fly rock, and back break [2–4]. Among these disadvantages, the deterioration of the surrounding rock mass frequently referred to as blast induced rock mass damage and the blast induced ground vibration are two major concerns for engineers and researchers [5–7].

Blast induced rock mass damage, defined as the propagation of inherent geological discontinuities and/or the formation of new cracks along weak planes in rock mass [8, 9], is affected by both the controllable and the uncontrollable parameters [10, 11]. The controllable parameters including mainly burden, spacing, stemming, subdrilling, and delay time can be optimized during blast design aiming at minimizing blast induced adverse effect, while the uncontrollable parameters are associated with the complex geotechnical characteristics of rock mass [12]. Due to the complexities of the geotechnical characteristics of rock mass and the difficulties in theoretical modelling of the interaction between blast and rock mass response, the extent of blast induced rock mass damage is generally estimated using simplistic models in which some important influencing factors such as the presence of water, explosive confinement, and blast sequence

and geometry are extruded [13]. In these models, the blast induced rock mass damage is predicted and evaluated using the parameter, peak vibration velocity (PVV), or blasting vibration velocity (BVV) as a measure of ground vibration intensity and structure damage extent induced by blasting.

To calculate and predict PVV or BVV, a variety of models have been proposed in the literature based on empirical and/or semiempirical formulae using field monitoring data [14, 15], artificial intelligence algorithms [16–18], and numerical analysis [19, 20]. In addition, a sophisticated blast modelling research tool, namely, the Hybrid Stress Blasting Model (HSBM) has become the latest development within the blasting modelling field and has been validated to be capable of predicting the extent and shape of blast induced damage zone and accounting for the effect of point of initiation and free face boundary conditions [21]. These models used for predicting blast induced ground vibrations; however, according to Lu et al. [22], they are not applicable to prediction of PPV of the vibrations induced by transient release of in situ stress (TRIS). And, on the basis of analysis of the factors influencing the TRIS induced PPV, they developed a new model to accurately forecast the TRIS induced PPV.

Apart from PPV or BVV as an indicator of the extent of blast induced rock mass damage, rock mass crack initiation and propagation induced by blast also indicate the adverse effect of blasting on the surrounding rock mass. During blasting, the explosive chemical reaction in blasthole changes the explosive from a condemned material to a gaseous product of high pressure and high temperature. In the meantime, stress wave (or shock wave) and explosion gas pressure are produced and loaded on the surrounding rock mass. The stress wave propagates at a higher velocity for a shorter duration in comparison to the explosion gas pressure [23, 24]. The explosion gas pressure can cause further extension and propagation of the cracks around the blasthole that are created by the stress wave. Therefore, both explosive gas pressure and stress wave should be featured in the study of rock mass fracture and fragmentation. In the literature, considerable effort at investigating blast induced rock mass crack initiation and propagation has been made. For example, Zhu et al. [25] numerically studied the role of stress wave loading on crack initiation and propagation during the initial stage of detonation in a borehole by using the AUTODYN 2D code. Based on a coupled numerical method using both LS-DYNA and UDEC, Wang and Konietzky [26] modelled the dynamic fracturing process of jointed rock masses due to blast wave loading. Banadaki and Mohanty [27] conducted laboratory blast experiments and numerical simulation using ANSYS AUTODYN to study blast stress wave induced fracture patterns in granitic rock.

Despite the wealth of the studies on blast induced rock mass damage and blast induced rock mass crack initiation and propagation, it is far from complete for study of blast induced crack propagation and damage accumulation in rock mass containing initial damage composed of inherent geological discontinuities and previous multiblast induced damage [28]. The incomplete knowledge of the effect of small blasthole diameter blasting on rock mass containing initial damage hinders the estimate of long-term stability

of the structure and the surrounding rock mass in that case, indicating the significance of investigating this field of research. Therefore, the objective of this study is to investigate small blasthole diameter blast induced crack propagation and damage accumulation in water-bearing rock mass containing initial damage. The investigation consists of a theoretical analysis of coupled and decoupled blasting shock parameters, a quantitative discussion of blast induced damage and damage accumulation calculation methods, and a well-designed implementation of field blast tests.

2. Blasting Impact Parameters

Although the existence of water medium in water-bearing rock mass can exert an influence over the blasting effect, the stress wave aroused in water in blasthole satisfies the fundamental equations for wave. These equations are expressed as

$$\rho_0 (D - u_0) = \rho (D - u) \quad (1)$$

$$P - P_0 = \rho_0 (D - u_0) (u - u_0) \quad (2)$$

$$E - E_0 = \frac{P + P_0}{2} \left(\frac{1}{\rho_0} - \frac{1}{\rho} \right) \quad (3)$$

where P_0 , E_0 , ρ_0 , and u_0 are, respectively, the pressure, energy, density, and particle movement speed for the water medium before arousing of stress wave, P , E , ρ , and u are, respectively, the pressure, energy, density, and particle movement speed for the water medium after arousing of stress wave, and D is wave velocity propagating in the water medium.

On the interface between explosive and water, the following equation is satisfied:

$$P_w = P_H \quad (4)$$

$$u_0 = u_w$$

where P_w is shock wave's original pressure, u_w is water medium's original movement speed, P_H is detonation pressure, and u_H is detonation products' particle movement speed. Therefore, the following equation can be derived:

$$P_w = P_H = \frac{\rho_e D_e^2}{4} \quad (5)$$

$$u_w = u_H = \frac{D_e}{4}$$

where ρ_e is explosive density and D_e is detonation velocity.

At the moment blasting begins, assuming water pressure $P_0 = 0$ and water particle movement speed $u_0 = 0$ leads to

$$D_e = \frac{\rho_e D_w}{\rho_0} \quad (6)$$

where ρ_0 is original water density and D_w is original shock wave velocity.

An isentropic process can be approximately assumed during shock wave propagation, indicating that water medium's state equation can be expressed as

$$P = A \left[\left(\frac{\rho}{\rho_0} \right)^\beta - 1 \right] \quad (7)$$

The density of water medium subject to compression of shock wave during water-coupling blasting becomes

$$\rho_w = \rho_0 \left(\frac{P_w}{A} + 1 \right)^{1/\beta} \quad (8)$$

Energy attenuation and reduction in peak pressure can occur for shock wave when propagating along blasthole radial direction and compressing water medium. For column charge blasting, the reduction in shock wave's peak pressure can be expressed as a function of distance:

$$P_w = \frac{B}{R^a} \quad (9)$$

Propagating of shock wave to blasthole wall produces a blasthole wall pressure of

$$P_1 = \frac{B}{K_d^a} \left(\pi \rho_e \frac{Q_{vs}}{Q_{VT}} \right)^{\alpha/2} \quad (10)$$

Therefore, some parameters corresponding to blasthole wall, according to Kachanov [29], can be calculated as

$$\begin{aligned} \rho_1 &= \rho_0 \left(\frac{P_1}{A} + 1 \right)^{1/\beta} \\ u_1 &= \sqrt{\frac{P_1}{\rho_0} \left[1 - \left(\frac{A}{P_1 + A} \right)^{1/\beta} \right]} \\ D_1 &= \sqrt{\frac{P_1}{\rho_0} \left[1 - \left(\frac{A}{P_1 + A} \right)^{1/\beta} \right]^{-1}} \end{aligned} \quad (11)$$

Reflected wave and transmitted wave are produced from blasthole wall when radially propagated shock wave reaches blasthole wall. To theoretically solve this problem, elastic theory and wave theory can be referred to. The shock wave pressure on blasthole wall under the normal incidence circumstance is calculated as

$$P = \frac{2\rho_m C_p}{\rho_m C_p + \rho_0 D_1} P_1 \quad (12)$$

where $\rho_m C_p$ is rock mass's wave impedance and $\rho_0 D_1$ is water medium's wave impedance when shock wave velocity equals D_1 .

However, coupling charge blasting is hardly encountered in underground rock mass excavation. Generally, there is no choice but treating air or water as the axial cushion and water contained in rock mass as axially decoupled charge structure when implementing analysis. In the experimental study by the authors, in situ blasting tests are performed treating water as the cushion, taking into consideration that rainwater has flowed into blasthole exposed to nature. Initial damage, such as crack, is believed to have been existing in the in situ surrounding rock mass relative to the blasthole due to influence of previous multiblasting tests in the blasthole. In that crack, water is filled; therefore, in study the fact that

blasting exerts influence over the crack water in the in situ surrounding rock mass to the blasthole can be considered as axially water-decoupled.

For convenience in terms of comparison and analysis, the blasthole located out of the rock mass crack water is selected as the study object for decoupled blasting with air as its cushion. For decoupled charge blasting, the produced stress wave and detonation gas will compress the decoupled medium. This compression leads to an increase in the decoupled medium's pressure. Upon reaching peak pressure, the detonation energy will be transmitted to rock mass through the decoupled medium. A 'squeezing' effect on the surrounding rock mass relative to the blasthole can be produced by the energy that is transmitted from blasting stress wave or by detonation products through rock mass crack water [30]. This 'squeezing' effect is in part related to the contraction of brittle material subject to impact load [31], but their energy attenuations are different in energy reflection and energy transmission.

When air is the cushion at the bottom of blasthole, the density of detonation gas that fills the whole blasthole due to expansion can be expressed as follows [32]:

$$\rho'_0 = \frac{L_c}{L_c + L_a} \rho_0 = \frac{\rho_0}{K_L} \quad (13)$$

Meanwhile, detonation gas's pressure is

$$P_0 = \frac{1}{8} \rho_c D^2 \left(\frac{L_c}{L_c + L_a} \right)^6 \approx \frac{1}{8} K_L^{-6} \rho_c D^2 \quad (14)$$

And the sound velocity of detonation gas is

$$C'_0 = \sqrt{\frac{K P_0}{\rho'_0}} = \frac{3\sqrt{2K_L}}{8} K_L^{-3} D \quad (15)$$

When water in blasthole is the cushion at the bottom of blasthole, the density of water subject to squeeze given by blasting stress wave and detonation gas can be expressed as

$$\rho'_w = \frac{L_a}{L_a - h} \rho_w \quad (16)$$

In the meantime, the pressure in blasthole is

$$P_b = \frac{\rho_c D^2}{8} \left(\frac{L_c}{L_c + h} \right)^3 \quad (17)$$

The dynamic failure process of water-bearing rock mass subject to blasting shock wave is a process during which damage accumulates till rupture failure of rock mass occurs. A good deal of defects, such as microfissure and microcracks, exist in rock mass as a brittle material. The existence of water in rock mass expedites stress wave propagation. Rock mass's macro-mechanical properties can be weakened due to microcrack initiation, propagation, and even penetration in rock mass, under the effect of water-decoupled blasting. Definition and selection of damage variable are implemented by the authors with the help of the understandings discussed

above. In an in situ blasting test coupling charge with air being the cushion and coupling charge with water being the cushion are performed simultaneously. In this context, for simplicity, analysis of decoupling charge with air or water being the cushion is merely conducted in this experimental study. With water being the cushion and rock mass crack as well as blast-hole being filled with water, the equations presented above can be used to calculate the parameters corresponding to water-coupling and water-decoupling charges. The calculated parameters enable analysis of the experimental results.

3. Calculation of Blasting Damage Accumulation

3.1. Determination of Damage Threshold. A damage variable is a representation of deterioration degree for material or structure. To define a damage variable, both micro- and macroparameters can be used. The microparameters include crack number, crack length, crack area, and crack volume, etc. And elasticity modulus, yield stress, tensile strength, and density are some of the macroparameters [33–35]. Lemaitre [36], for example, pointed out based on effective stress that for isotropic distribution of microcracks the damage to material was shown in the reduction in elasticity modulus. This point of view gives a damage variable, D , being expressed as

$$D = 1 - \frac{E_1}{E_0} \quad (18)$$

where E_0 and E_1 are, respectively, elasticity moduli of rock mass before and after occurrence of damage.

Assume that rock mass damage is isotropic, rock mass damage can also be quantified using the change in ultrasonic velocity in rock mass. Calculation of longitudinal wave velocity is performed using

$$\begin{aligned} C_0 &= \sqrt{\frac{E_0(1-\mu)}{\rho_0(1+\mu)(1-2\mu)}} \\ C_1 &= \sqrt{\frac{E_1(1-\mu)}{\rho_1(1+\mu)(1-2\mu)}} \end{aligned} \quad (19)$$

where C_0 and C_1 are, respectively, longitudinal wave velocities before and after damage occurrence, ρ_0 and ρ_1 are, respectively, rock mass densities before and after damage occurrence, and μ = Poisson's ratio of rock mass.

In general, detection and representation of rock mass damage are implemented using the difference between longitudinal wave velocity in rock mass before damage occurrence and longitudinal wave velocity in rock mass after damage occurrence. Therefore, the damage variable in (18) can be rewritten as

$$D = 1 - \frac{E_1}{E_0} = 1 - \frac{\rho_1}{\rho_0} \left(\frac{C_1}{C_0} \right)^2 \quad (20)$$

Note that the difference between ρ_0 and ρ_1 is negligibly small, the damage variable, D , in (20) can be expressed as

$$D = 1 - \left(\frac{C_1}{C_0} \right)^2 \quad (21)$$

According to the Chinese construction specification [37], the following equation can be used to determine blast induced rock mass damage degree:

$$\eta = 1 - \frac{C_1}{C_0} \quad (22)$$

where η is rate of change in sonic wave velocities in rock mass before and after damage occurrence. For $\eta > 10\%$, rock mass damage occurs. Consequently, by manipulating (21) and (22), the expression of the damage variable, D , in (21) becomes

$$D = 1 - (1 - \eta)^2 \quad (23)$$

Regarding $\eta = 10\%$ as the threshold for occurrence of rock mass damage leads to the determination of rock mass damage criterion. Substitute 10% for η in (23), then the damage variable D is equated to 0.19. Correspondingly, the critical rock mass damage variable is $D_{cr} = 0.19$. The elastic wave theory reveals that longitudinal wave velocity in rock mass is capable of reflecting some of the mechanical characteristics of rock mass. Hence, it is a convenient and efficient method of determining rock mass damage variable in engineering practice by using the change in longitudinal wave velocity in rock mass before and after damage occurrence.

3.2. Damage Variable and Damage Accumulation Calculation. Original defects of various types exist in rock mass subject to blasting effect. These defects can be regarded as rock mass's initial damage, D_0 . Under effect of blasting stress wave the rock mass crack can be activated. The activated rock mass crack propagates continuously, producing new damage. Therefore, rock mass damage after blasting occurrence, D , should include the initial damage, D_0 , and the new damage induced by blasting effect, ΔD , which gives

$$D = D_0 + \Delta D \quad (24)$$

The initial damage, D_0 , can be calculated using the effective area, A_0 . Assume that in a rock mass of dimension $2L \times h$ there exists a single original crack of length $2a$ and of statistical distribution. Then the initial damage, D_0 , is calculated by

$$D_0 = \frac{A_0}{A} = \frac{2ah}{2Lh} = \frac{a}{L} \quad (25)$$

As is shown in Figure 1, the crack length increment induced by the last blasting after multiblasting of n times is assumed to be $2r_n$. The rock mass damage variable after multiblasting of n times, D_n , according to Ma [38], is calculated as

$$\begin{aligned} D_n &= \frac{2ah + 2r_1h + \cdots + 2r_{n-1}h + 2r_nh}{2Lh} \\ &= \frac{a + \sum_{n=1}^n r_n}{L} \end{aligned} \quad (26)$$

Consequently, for rock mass containing initial damage, the rock mass damage variable in the far-field of blasting

FIGURE 1: Schematic illustration of rock mass damage evolution induced by multiblasting.

FIGURE 2: Schematic illustration of rock mass damage induced by single-hole blasting.

source after multiblasting of n times, D_n , can be expressed as

$$D_n = D_{n-1} + D_0 \frac{r_n}{a} \quad (27)$$

And the general form of D_n in (27) can be written as

$$D_n = \frac{a + \sum_{n=1}^n r_n}{L} = D_0 + \frac{\sum_{n=1}^n r_n}{L} = \frac{D_0}{a} \left(a + \sum_{n=1}^n r_n \right) \quad (28)$$

The existence of initial damage in rock mass, such as joint fissure, causes the blasting stress wave to decay in rock mass. The decay degree is varying depending on the joint fissure distribution and form. Figure 2 depicts the rock mass damage effect induced by single-blasthole blasting.

The decay of the original axial stress peak for air-coupling charge blasting satisfies the following equation:

$$P_c = \frac{K \rho_0 D_c^2}{8} \left(\frac{r_i}{r} \right)^6 \left(\frac{r}{R} \right)^{\alpha/(1-D_0)} \quad (29)$$

where K is amplification coefficient for detonation gas pressure on blasthole wall, ranging generally from 8 to 11, ρ_0 is explosive density, D_c is detonation velocity, r_i is explosive diameter, r is blasthole diameter, R is distance to blasting

source, and α is decay coefficient. The decay coefficient, α , can be expressed as a function of dynamic Poisson's ratio, μ_d , which is

$$\alpha = 2 - \frac{\mu_d}{1 - \mu_d} \quad (30)$$

$$\mu_d = 0.8\mu$$

where μ is rock mass's static Poisson's ratio.

Water-coupling blasting is formed when a great deal of water fills rock mass crack and blasthole. For this case, the shock pressure on blasthole wall can be calculated using (12).

The particle vibration velocity, v , which can be used to calculate the stress peak, is generally obtained by in situ monitoring and by resorting to the Sadov's formula:

$$v = K \left(\frac{\sqrt[3]{Q}}{R} \right)^\alpha \quad (31)$$

where K is site coefficient related to rock mass property, blasting parameter, and blasting method, Q is maximum charge in blasting segment, R is distance to blasting source, and α is decay coefficient.

The dynamic stress peak at this point produced by blasting is

$$P_c = \rho_r C_r v \quad (32)$$

The total crack length in a rock mass of reference dimension $2L \times h$ subject to multiblasting can be calculated using the empirical formula proposed by Meng et al. [39] of

$$l_{\max} = 2r_{\max} = 50.26a_t \left(\frac{\rho_r C_r v}{1 - D_0} \right)^2 \left(\frac{1}{\sigma_d} \right)^{2.23} \quad (33)$$

where ρ_r is rock mass density, C_r is longitudinal wave velocity, v is particle vibration velocity, and σ_d is dynamic tensile strength.

Combination of (29), (31), (32), and (33) enables the calculation of rock mass damage accumulation. In recent decades, great progress has been made in ultrasonic testing technology and relevant testing equipment [40, 41]. Therefore, it is feasible to determine rock mass damage using sonic wave test in engineering practice.

4. Analysis of Blast Induced Crack Propagation in Rock Mass Containing Initial Damage

Characteristics of blasthole wall crack propagation before and after blasting occurrence are shown in Figure 3. From Figure 3 it can indicated that I-type crack as illustrated in Figure 4 dominates the blasthole wall propagation induced by blasting. Reflection and diffraction occur for stress wave induced by detonation gas when it propagates to the original crack. The reflection occurs on the crack surface while the diffraction occurs at the original crack tip, intensifying the near-crack-tip stress field [42]. In Figure 4, the stress field

FIGURE 3: Photograph showing crack propagation in blasthole before and after multiblast.

FIGURE 4: Schematic illustration of I-type crack in blasthole induced by blasting.

at point A of distance r to crack tip O and of angle θ from direction OX is calculated by

$$\begin{aligned}\sigma_x &= \frac{K_I(t)}{\sqrt{2\pi r}} \left(1 - \sin \frac{\theta}{2} \sin \frac{3\theta}{2} \right) \cos \frac{\theta}{2} \\ \sigma_y &= \frac{K_I(t)}{\sqrt{2\pi r}} \left(1 + \sin \frac{\theta}{2} \sin \frac{3\theta}{2} \right) \cos \frac{\theta}{2} \\ \tau_{xy} &= \frac{K_I(t)}{\sqrt{2\pi r}} \sin \frac{\theta}{2} \cos \frac{3\theta}{2} \cos \frac{\theta}{2}\end{aligned}\quad (34)$$

where $K_I(t)$ is dynamic stress intensity factor, depending on blasting stress wave shape.

For I-type crack, unstable propagation occurs when the strain energy release rate reaches its threshold, which is

$$\frac{dW}{da} - \frac{dE}{da} > G_0 \quad (35)$$

Because I-type crack dominates the blasthole wall original crack, the stress intensity factor for the blasthole wall original crack can be approximately calculated by

$$K_I = 2.24P\sqrt{\pi(a+r)} \quad (36)$$

where P is detonation gas or water pressure, a is original crack length, and r is blasthole diameter.

As the criterion for dynamic crack propagation is complex, approximate treatment is implemented, which gives that the original crack propagation is induced by the blasting stress wave. Therefore, a quasi-static criterion for blast induced crack propagation is derived and is expressed as

$$K_I \geq K_{IC} \quad (37)$$

The minimum detonation pressure corresponding to initial propagation of the original crack, according to manipulation of (27) and (28), can be calculated by

$$P_C = \frac{K_{IC}}{2.24\sqrt{\pi(a+r)}} \quad (38)$$

By using the equations presented above, the minimum explosive equivalent required for propagation of original crack in in situ blasting tests can be derived, which lays the foundations for further calculation of the blasting energy and the damage diameter, etc.

5. In Situ Blast Test

The in situ blasting tests in this study are conducted in an outdoor blasting test site. The blasthole and the sonic wave monitoring holes (SWMHs) are drilled into marble rock mass. The blasting pattern used is characterized by single-blasthole, small explosive equivalent, and multiblasting. The use of the blasting pattern is to study the blast induced rock mass damage accumulation characteristics of different distances to a particular geological section. To monitor the blasting sonic wave velocity, the intelligent sonic tester Geode

FIGURE 5: Layout of blasthole and sonic wave monitoring holes.

FIGURE 6: Three-dimensional illustration of locations of blasting source, blasthole, and sonic wave monitoring holes.

is used. And an ultraportable microseismograph is utilized to monitor the blasting vibration. A high-precision, high-resolution digital camera is used to photograph the blasthole crack propagation before and after blasting occurrence.

The plan layout of blasthole and SWMHs is shown in Figure 5. In this figure, the quadrate blue dot represents the blasthole of diameter 75 mm, and the ten circular red dots represent the SWMH of depth 3.3 to 6.0 m and of diameter 60 mm. For convenience in terms of comparison of testing results, in each group the two SWMHs are positioned symmetrically with an identical distance to the blasting source. And the depths of these SWMHs are varied. Therefore, an all-around, three-dimensional sound signal for the fractured rock mass of different distance to blasting source is derived, enabling a comprehensive analysis of blast induced rock mass damage. Figure 6 shows three-dimensional illustration of locations of blasting source, blasthole, and sonic wave monitoring holes. A comparison of rock mass damage before and after blasting occurrence is made in Figure 3 derived from photograph using the high-precision, high-resolution digital camera. The monitoring items include particle vibration velocity, sonic wave velocity, and rock mass crack propagation, etc.

FIGURE 7: Particle vibration velocity-time curve for sonic wave monitoring hole No. 9.

Discussions of single-blasthole blast induced sonic wave velocity, particle vibration velocity, and rock mass damage accumulation for different distances to the blasting source are made in this study. The basic parameters in this experimental study are rock mass density $\rho_r = 2800 \text{ kg/m}^3$, original rock mass damage variable $D_0 = 0.56$, original rock mass crack length = 1.2 m, rock mass dynamic tensile strength $\sigma_d = 18 \text{ MPa}$, blasthole diameter = 75 mm, explosive diameter of $r_1 = 55 \text{ mm}$ and $r_2 = 35 \text{ mm}$, explosive density $\rho_0 = 1100 \text{ kg/m}^3$, detonation velocity = 4000 m/s, and rock mass Poisson's ratio $\mu = 0.25$.

In situ blasting tests of different explosive equivalents and of different blasthole depths are conducted in this study. SWMH No. 9 as shown in Figure 5 is selected as the study object. For this SWMH, the particle vibration horizontal velocity versus time curve is presented in Figure 7. The particle vibration velocities for different distances to blasting source are paralleled between theoretical and experimental results in Figure 8. And Figure 9 shows the comparison of sonic wave velocity versus distance to blasting source between theoretical and experimental results. In Figure 10, the variations in blast induced rock mass damage variable, D , with an increase in the distance to blasting source are compared between theoretical and experimental results. From Figure 8 it can be indicated that the particle vibration velocity decreases nonlinearly with increasing the distance to blasting source. Occurrence of rock mass damage is identified when the monitored D_n becomes greater than D_{cr} equaling 0.19.

The deep blasthole camera detection before and after blasting occurrence indicates that the blast induced rock mass crack propagation pattern is rather complex. Nevertheless, the rock mass crack length increment induced by the first blasting, Δr , dominates, which means that observable damage to rock mass occurs after the first blasting. When $D_n > D_{cr} = 0.19$ is satisfied after multiblasting of certain explosive

FIGURE 8: Particle vibration velocities for various distances between blasting source and sonic wave monitoring hole.

FIGURE 10: Experimental and theoretical curves showing accumulated rock mass damage.

FIGURE 9: Experimental and theoretical sonic wave velocities versus distance between blasting source and sonic wave monitoring hole.

equivalent, then the blasting test for this explosive equivalent comes to an end because rock mass damage has occurred.

It is indicated from Figures 8 and 9 that the decay of the particle vibration velocity is related to the distance to blasting source. The rate of the decay is greater for smaller distance to blasting source. The reason this phenomenon occurs is that wave propagation condition is poorer for smaller distance to blasting source due to a larger extent of blast induced rock mass damage. For two SWMHs adjacent to each other, the difference between their particle vibration velocities decreases with an increase in the distance to blasting source.

Figure 10 shows that the blast induced rock mass damage variable, D , decreases with an increase in the distance to blasting source, indicating a damage of smaller degree that blasting causes to rock mass in far-field of blasting source. This rock mass damage of smaller degree can be further reduced when joint fissure is encountered because the joint fissure in rock mass has a strong inhibition on blasting stress wave.

6. Conclusions

The conclusions drawn from this study are summarized as follows.

- (1) For small blasthole blasting in rock mass containing initial damage, the method of calculating relevant parameters is analyzed. The analysis is conducted under four different circumstances which are water being the cushion at the bottom of blasthole, air being the cushion at the bottom of blasthole, coupled charge, and decoupled charge. To conveniently characterize rock mass damage, a damage variable is defined using the rate of change in the longitudinal wave velocities in rock mass before and after blasting. And the equation for calculating blast induced rock mass damage accumulation is derived.
- (2) In situ small blasthole blasting tests are conducted. The blasthole wall crack propagation subject to blasting is experimentally observed and theoretically calculated. Analysis of the derived results indicates that blasting activates the original crack in rock mass, and the effect of the first blasting dominates. Besides, occurrence of rock mass damage adjacent to the blasthole can be identified when rock mass damage variable satisfies $D_n > D_{cr} = 0.19$.

- (3) Experimental results indicate that the distance to blasting source controls the extent of blast induced rock mass damage. With an increase in the distance to blasting source, a nonlinear decrease in rock mass damage increment or accumulated rock mass damage is observed, and the rate of the decrease drops gradually. This phenomenon is also applicable for the particle vibration velocity.

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 there are no conflicts of interest regarding the publication of this paper.

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