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A finite difference modelling of crack initiation in rock blasting

Meysam Lak[1], *Mohammad Fatehi Marji[2] and Alireza Yarahmadi Bafghi[3]

[1), 2), 3)] *Faculty of Mining and Metallurgical engineering, Yazd University, Yazd, Iran*
[2)] *mfatehi@yazd.ac.ir*

ABSTRACT

The rock blasting process in a borehole applies very high pressure on its surrounding rock media in a very short time (milliseconds) and usually established to break the rocks into smaller pieces. This process occurs in two stages, crack initiation and propagation. In this study, crack initiation caused by blasting in rock is generally considered and modeled numerically using Finite Difference Method (FDM). A time dependent (dynamic) finite difference algorithm is used to simulate the explosion process. The crack initiation due to explosion was modeled around the one, two and three blast holes in the rock media. The blast induced crack patterns in the crack initiation models have been discussed at the different times. Also, the stress waves are simultaneously propagated in the rock media that are investigated in here.

1. INTRODUCTION

Fracture mechanics has been proposed as a possible tool for solving a range of rock engineering problems, such as explosive fracturing, rock cutting, hydrofracturing, rock stability, etc. (Hosseini Nasab and Fatehi 2007). Furthermore, Linear Elastic Fracture Mechanics (LEFM) principles have been widely used in rock fracture mechanics (Rossmanith 1983; Whittaker et al. 1992; Aliabadi 1998). Mechanical behaviors of rocks affected by high explosion loads are difficult and costly to be studied exclusively by instrumentation and experimental works. In addition, the explosion induced fractures in rock propagate very quickly. Therefore, rock dynamic fracture mechanisms can be studied by the sophisticated numerical methods (Dehghan Banadaki 2010; Dehghan Banadaki and Mohanty 2012).

Already, many researchers were focused on crack initiation and propagation in rock due to blasting. Initiation and propagation mechanisms of radial crack around a borehole in blasting processes is a complicated and interesting phenomenon that have been studied in (Ingraffea 1983; Mortazavi and Katsabanis 2001; Cho et al. 2004;

[1)] PhD Student
[2)] Associate Professor
[3)] Associate Professor

Wang 2018). The crack propagation from the two and more radial cracks tips around a blast hole was investigated using displacement discontinuity method in a quasi-static manner (Fatehi 1997; Hosseini Nasab and Fatehi 2007; Fatehi et al. 2010). Some researchers have been focused on the effects of free face, in-situ stresses and load density on the rock fracturing process (Ma and An 2008; Wang and Konietzky 2009; Jian-sheng and Fan-fei 2009; Bendezu et al. 2017; Li et al. 2017). Fracture propagation and fragmentation of rock were numerically studied using a gas-solid model (Yang et al. 2017). Also, the fragmentation zone caused by blasting in rock was predicted in (Youngjong et al. 2017). In (An et al. 2017) a hybrid finite-discrete element modeling was conducted to simulate the fracture extension and fragmentation due to rock blasting. Moreover, in some dynamic studies effects of wave propagation on the rock mass properties has been evaluated (Lak 2014; Lak et al. 2017; Niu et al. 2018).

In this study, crack initiation induced by blasting in rock is generally considered and modeled numerically using FDM. Therefore, three different models were considered: 1) the model with a blast hole in its center, 2) the model with two blast hole that have a typical spacing of 3 m, and 3) the model with three blast hole that are embedded by a triangular pattern. The explosive was exploded in these models and patterns of the induced cracks around the blast holes were investigated and discussed.

2. DYNAMIC NUMERICAL MODELING

In order to simulate the crack process due to rock blasting, in this study one, two and three blast holes models were numerically modeled using FLAC software. Fast Lagrangian Analysis of Continua (FLAC), has a time-marching explicit finite difference algorithm for solving dynamic problems such as rock blasting (Itasca 2016). The geometry of considered models are illustrated in Fig. 1. In present study, the blast holes have 3 m spacing from each other and the radius of the blast holes is 0.04 m.

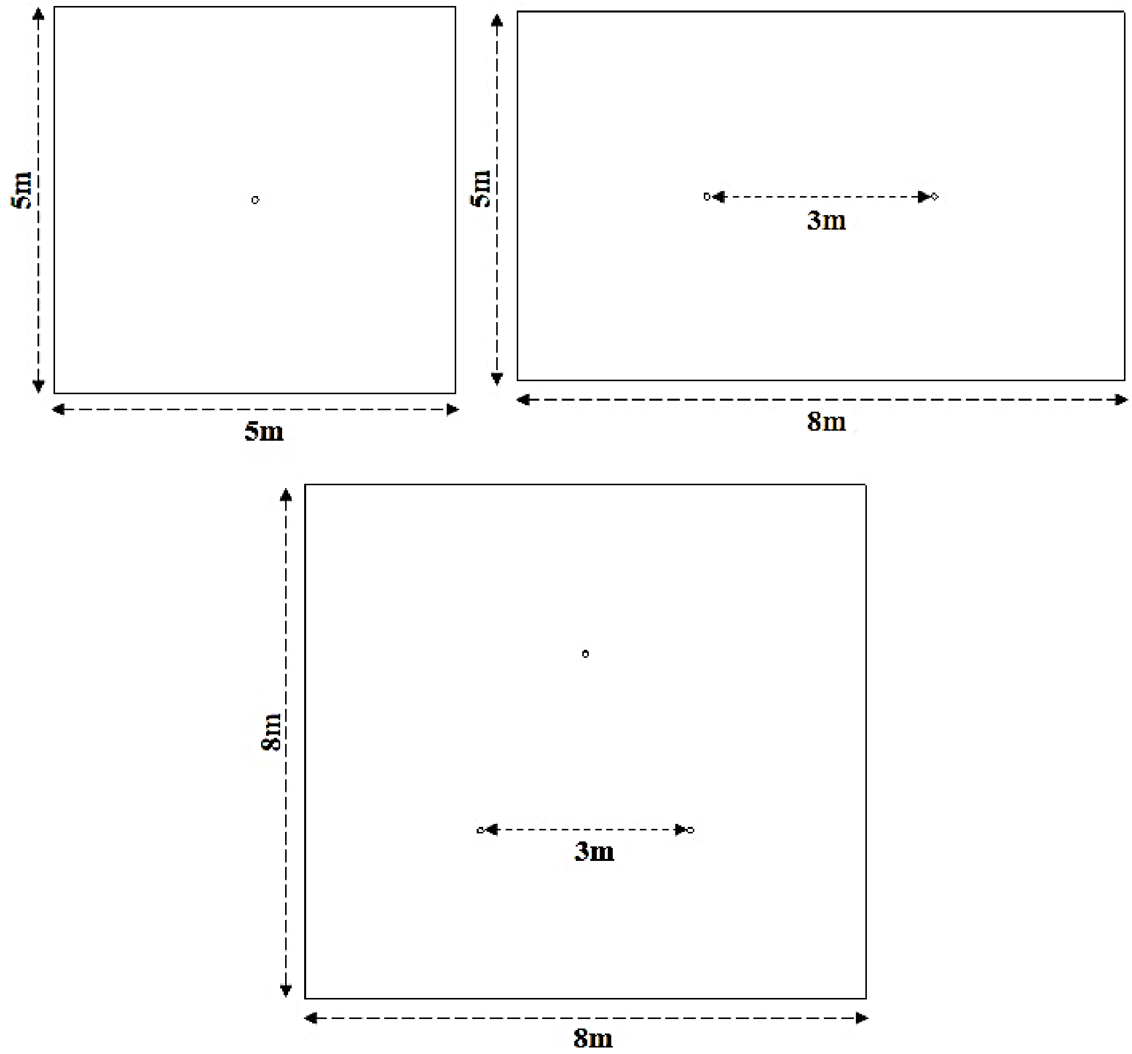

Fig. 1 Geometry of the three different models

In this study, the Mohr-Coulomb constitutive model was assumed as mechanical behavior of the rock and the geomechanical properties of the simulated rock models are tabulated in Tab. 1.

Tab. 1 Geomechanical properties of the rock models

Density (kg/m^3)	Yung modulus (GPa)	Poisson ration	Cohesion (MPa)	Friction angle (°)	Tensile strength (MPa)
2616	67.3	0.2	22.9	47.4	8.9

In the numerical simulation, a set of formulation was used, to calculate the detonation pressure pulse. Based on (Lopez Jimeno et al. 1995), the detonation pressure of an explosive can be estimated from the following equation,

$$PD = 432\times10^{-6}\times\rho_e\times\frac{VD^2}{1+\left(0.8\rho_e\right)} \tag{1}$$

where *PD* is pressure of detonation in MPa, ρ_e is explosive density in g/cm^3 and *VD* is velocity of detonation in m/s (Lopez Jimeno et al. 1995). In practical purposes the pressure of explosion (*PE*) is estimated as (Hustrulid and Johnson 2008),

$$PE = \frac{1}{2}PD \tag{2}$$

Additionally, the time dependent borehole wall pressure (*P(t)*) is significant as a result of explosion. Many of the past studies indicated functions to obtain *P(t)* such as (Dowding and Aimone 1985; Cho and Kaneko 2004; Chun-rui et al. 2009). The transient spherical cavity pressure is represented by the following expression (Duvall 1953; Brady and Brown 2005),

$$P = P_0\left(e^{-\alpha t}-e^{-\beta t}\right) \tag{3}$$

where P_0 is the peak wall pressure and α and β are positive frequency-dependent decay constants. The transient borehole pressure for long column of explosive is similarly derived as,

$$P = P_0\xi\left(e^{-\alpha t}-e^{-\beta t}\right) \tag{4}$$

where ξ is a variable representing the rising and decaying phases of the pressure pulse, and t_0 is the time required to achieve the peak pressure. They are obtained by (Cho and Kaneko 2004; Bendezu et al. 2017),

$$\xi = 1/\left(e^{-\alpha t_0}-e^{-\beta t_0}\right) \tag{5}$$

$$t_0 = \left(1/\left(\alpha-\beta\right)\right).\log\left(\beta/\alpha\right) \tag{6}$$

The decay constants α and β in the above equations can be achieved by,

$$\alpha = \omega/4\sqrt{2} \tag{7}$$

$$\beta = \omega/2\sqrt{2} \tag{8}$$

$$\omega = \frac{2\sqrt{2}C_p}{3a} \tag{9}$$

$$C_p = \sqrt{\frac{K+\left(\frac{4G}{3}\right)}{\rho_r}} \tag{10}$$

where C_p is P-wave velocity in the rock media, a is borehole radius, K is bulk modulus of the rock media, G is shear modulus of the rock media and ρ_r is rock density (Duvall 1953; Dowding and Aimone 1985).

In the present study, the pressure pulse induced by detonation of a type of gelatinous explosive with a density of 1.5 gr/cm^3 and detonation velocity of 6000 m/s. Accordingly, the input pressure pulse in this study has been calculated as that be shown in Fig. 2.

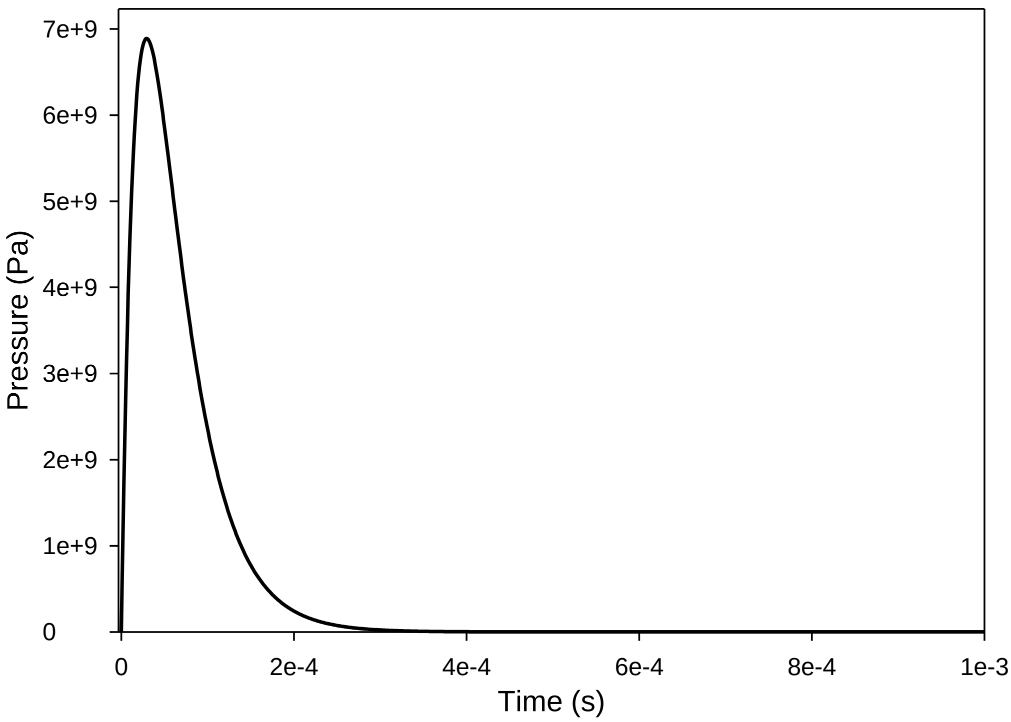

Fig. 2 Input pressure pulse

The pressure pulse, shown in Fig. 2, was applied to the blast holes and influenced on the rock media around the blast holes. Results of this phenomenon are presented and discussed in the next section.

3. RESULTS AND DISCUSSION

After the blasting, shock wave propagated in the rock media and the induced stresses caused cracks initiated around the blast hole. Fig. 3 shows the induced crack patterns around one, two and three blast holes models (shown in Fig. 1) due to blasting procedure.

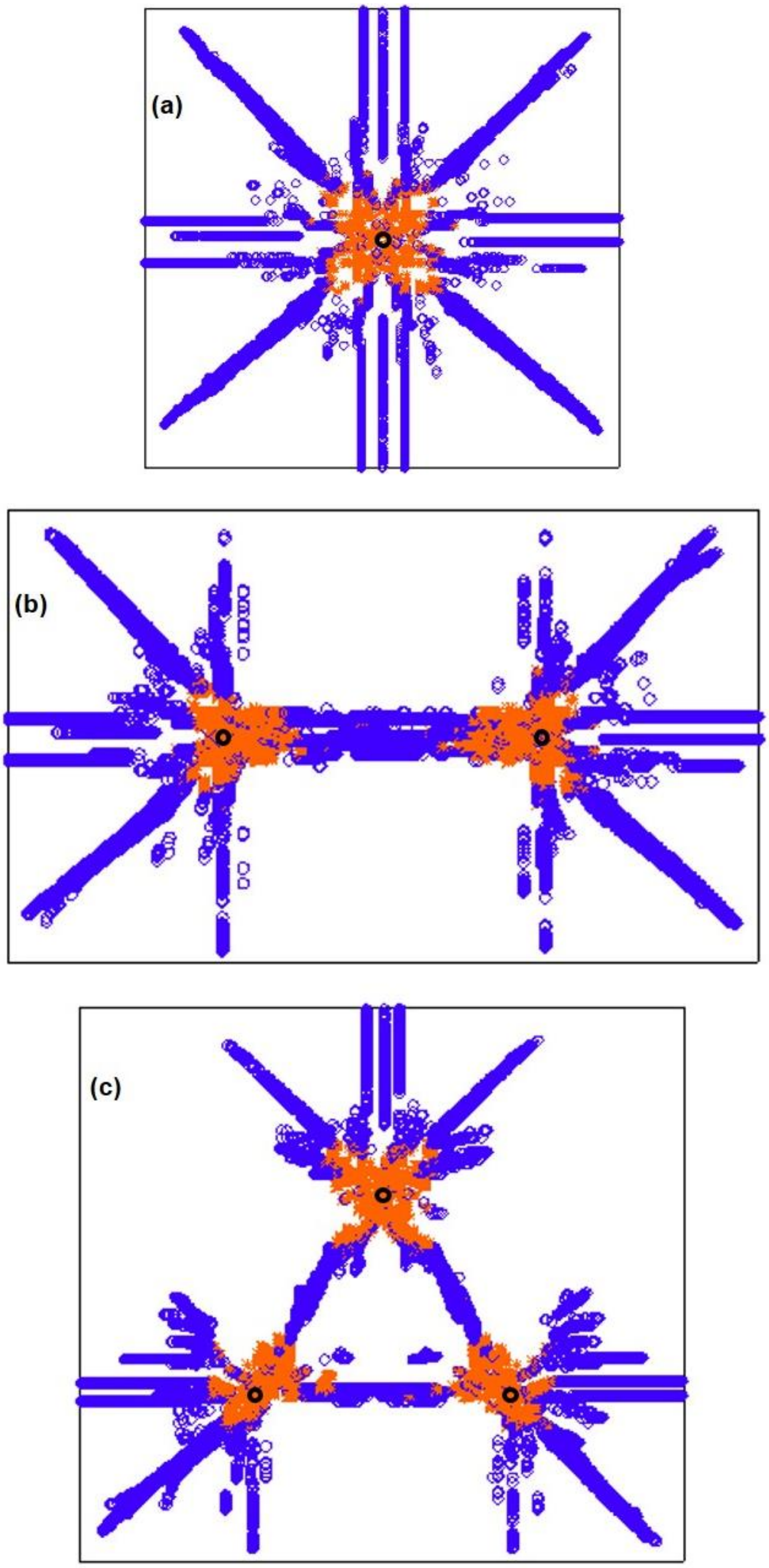

Fig. 3 Blast induced cracks patterns around, a) one, b) two, and c) three blast holes

In order to better represent the major cracks around the blast holes and discussion, the main paths of the cracking process were identified and illustrated in Fig. 4.

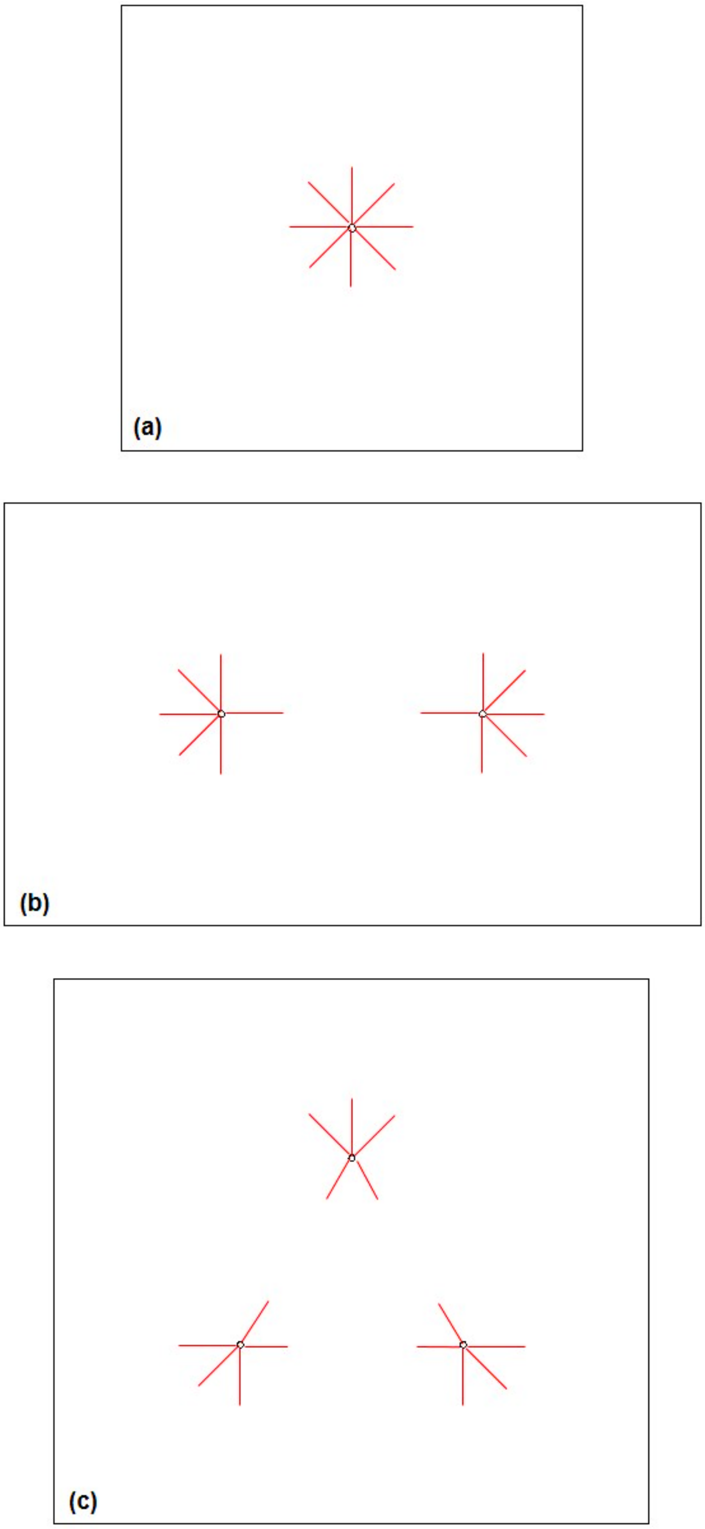

Fig. 4 Schematic illustration of the cracks patterns around, a) one, b) two, and c) three blast holes

As can be observed in Fig. 4-a, there are 8 cracks initiated by the explosion around a blast hole (N=8). Fig. 4-b shows that 6 cracks were initiated around each borehole in the model with two blast holes. On the other hand, when two blast holes are simultaneously exploded, only the 6 initiated cracks can be seen around the each one (N=6). Also based on Fig. 4-c, 5 cracks were created around the each blast hole. In this case can be said that the explosion of the three blast holes at the same time has caused to initiate 5 cracks around them (N=5). Moreover, orientation of the induced cracks in the Figs. 4-b and 4-c depends on the blast hole positions related to each other. Generally, it can be understood that some induced cracks around the single blast hole are removed in the models with two and three blast holes.

4. CONCLUSIONS

The crack initiation around one, two and three blast holes was investigated and numerically modeled due to rock blasting. A time-marching finite difference technique was utilized to simulate the shock wave propagation in rock media. As the results show, there are 8 radial cracks initiated by the explosion around a blast hole. Also, 6 radial cracks were initiated in the model with two blast holes. Additionally, 5 radial cracks were created around each blast hole in the model with three blast hole. The orientation of the induced radial cracks in the models with two and three blast holes depends on the blast-hole positions related to each other. It should be noted that in addition to the mentioned radial cracks, especially in the models with two and three blast holes, there are few cracks which cannot extend as large as the others.

The radial crack patterns (number of radial cracks and their orientation) resulted in this study can be used in numerical modeling of the crack propagation around the blast holes as crack tip elements.

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