Blast Simulation in Underground Mines Using the Combined Finite-Discrete Element Method

RIYU WEI[a], SHENG ZHANG[a,b], WARNA KARUNASENA[c], NAGARATNAM SIVAKUGAN[d], HONGWU ZHANG[b]

[a]Advanced Computational Modelling Centre, University of Queensland, St Lucia, Qld 4072, AUSTRALIA

[b]Department of Engineering Mechanics, Dalian University of Technology, Dalian 116024, CHINA

[c]Faculty of Engineering and Surveying, University of Southern Queensland, Toowoomba, Qld 4350, AUSTRALIA

[d]School of Engineering, James Cook University, Townsville, Qld 4811, AUSTRALI

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Abstract : The purpose of this paper is to investigate the application of a combined finite-discrete element program, such as ELFEN, for the simulation and prediction of backfill stope failure risks due to blast loading effects in underground mines. A real size of a backfilled stope blasted in the nearby rock of the paste fill is modelled. The dynamic fracture propagation and fragmentation in rock and the behaviour of paste fill by using the proposed model are provided and discussed. The results show the proposed model can be applied to provide an effective means to simulate substantial insight into the dynamic fracture processes during the blasting and estimate failure risks of the paste fill by using the peak particle velocity as a measure of blast damage.

Key-Words: finite-discrete element method; paste backfill; blast simulation; rock damage;underground mine

1 Introduction

Paste fill, which is a kind of backfill material in an underground mine, consists of tailing of minimum 15% of fill size less than 20 μm, water and small percentage of binder. Recently paste fill has gained rapid acceptance as an alternative backfill material to the conventional cemented hydraulic fills (Belem & Benzaazoua [1], Rankine & Sivakugan [2]). Paste fill has substantial benefits to mining operations including an effective means of tailings disposal, improvement of local and regional rock stability, greater ore recovery and greatly reduced environmental impacts. Once the ore from a stope is removed, the stope is filled with paste fill. As the mining sequence progresses, stopes adjacent to the fill are mined, and the fill is subjected to blast loads and subsequently exposed. The fill must remain stable at this time to provide support to the area and to avoid reprocessing the material. The purpose of this study is to utilize the combined finite-discrete element program ELFEN for simulating the dynamic fracture propagation and fragmentation and predicting the stability of paste fill in backfilled stopes due to mine blast.

The combined finite-discrete element method is a recently developed numerical method aiming at modeling failing, fracturing and fragmenting solids. The discrete element method (DEM) developed originally by Cundall and Strack [3] is that the domain of interest is treated as an assemblage of rigid or deformable blocks/particles/bodies and the contacts among them can be identified and continuously updated during the entire deformation/motion process. The effect of the blast pressure waveform on the fracture process was studied using the DEM (Donze et al [4]). To simulate the complex behavior of blasting which causes extensive fracture and fragmentation processes, Munjiza et al. [5, 6] developed an interesting detonation gas model based on the combined FEM/DEM.

Up to now, some commercial DEM software, such as ELFEN (Rockfield [7]) and UDEC (Itasca [8]) etc., have appeared. These software already show significant potential in the investigation of discontinuum modelling. ELFEN, a combined finite/discrete element code, enables to model intact behaviour, interactions along existing discontinuities, and it has been successfully applied to the simulation of intense fracturing associated with surface mine blasting, mineral grinding, rock slope (Eberhardt et al. [9]) and underground rock caving (Munjiza et al. [5]). Taking the advantages of ELFEN, a numerical model is developed for the simulation of a backfill stope failure due to blast loading effects in underground mines. The remainder of the paper is structured as follows: Section 2 describes the mining method and the development of the numerical model. Section 3 analyses the dynamic fracture processes and discusses the numerical results. The conclusions are drawn in Sections 4.

2 Mining and Model Descriptions

There are several mining methods. These can be classified as open surface mining and underground mining. Most of the mines are using underground mining methods over open pit surface mining. Underground mining can be divided into room and pillar method and cut and fill method.

2.1 Cut and fill mining

In this study, the cut-and-fill mining method is used for underground mining. Cut-and-fill mining is applied for mining of steeply dipping ore-bodies in strata with good to moderate stability and a comparatively high grade mineralization. It allows selective mining to recover high grade sections separately and leave low grade rock behind in stopes.

The procedure of cut and fill mining can be explained in Fig. 1. Spiral ramps are made in footwall. It provides the access drive to undercut. Cut-and-fill mining excavates the ore in horizontal slices, starting from a bottom undercut, advancing upward. After the ore was drilled and blasted, the muck is loaded and removed from the stope. When the full stope area has been mined out, voids are backfilled with sand tailings or waste rock. The fill serves to support both stope walls and working platform for equipment, when mining the next slice. The fill often consisting of de-slimed sand tailings from the dressing plant of mine, at times complemented by waste rock produced by development excavations, dumped in empty stopes. Paste fills or sand fills are used to fill the voids.

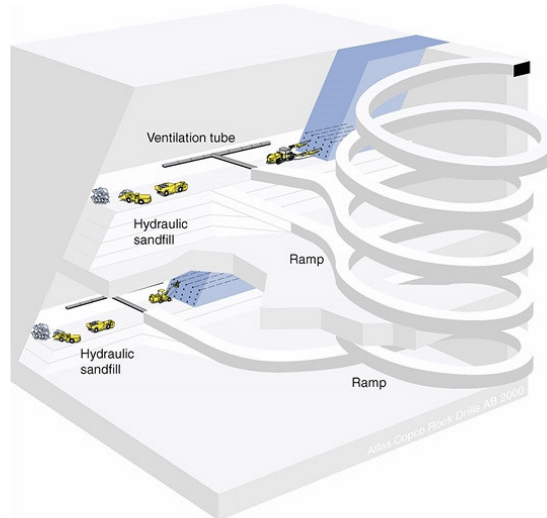

Figure 1 Schematic diagram of under ground mine (cut and fill method)

The elevation view of stope shows the access drives and the blasting pattern in Fig. 2. The blasting advances from bottom to top. The access drives comes from neighboring stope. There is more than one access drive for one stope. The blasting is done in different batches. The bottom level of stope is blasted first and the crushed material is removed. Then the blasting is done for above layer. The filling is done after blasting whole stope and removing all the material.

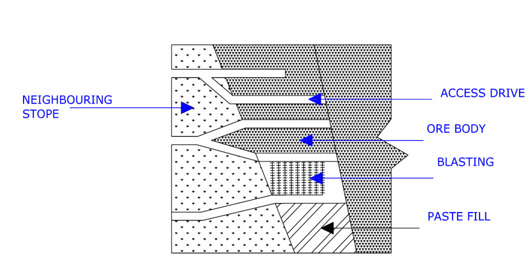

Figure 2 The procedure of drilling and blasting

The plan view of stope arrangement is given in Fig.3. In this case the blasting is done in a secondary stope, which is critical for design.

It has access through the rock (neighboring stope) as shown in Fig. 2. The ore is blasted and removed to create the voids. The rock and neighboring stopes support these voids.

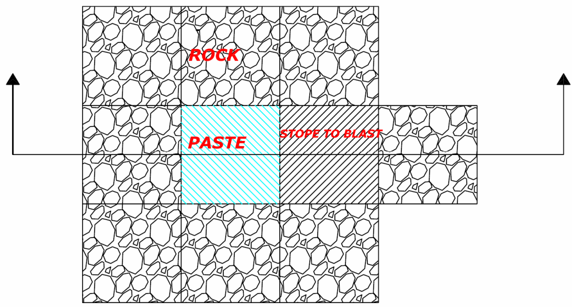

Figure 3 Plan view of stope arrangement

2.2 Model description

Model is based on the scenario of a single column of emulsion explosive in a rock mass, as shown in Fig.4. The rock to be blasted is 35 m in height and 5 m in thickness. The blast hole is 89 mm in diameter. The blast hole drilled from bottom to top, is 25 m deep in this model, and is assumed to consist of 20 m length of explosive material and 5 m of stemming material. The stemming material is used to ensure that the gases provide a load to the walls of the blast hole rather than escaping from the blast hole, and does not provide any strength to the system. Therefore, the stemming material is not modelled. For the purpose of this modelling, it is assumed that there is no paste fill in the vicinity of the explosive column.

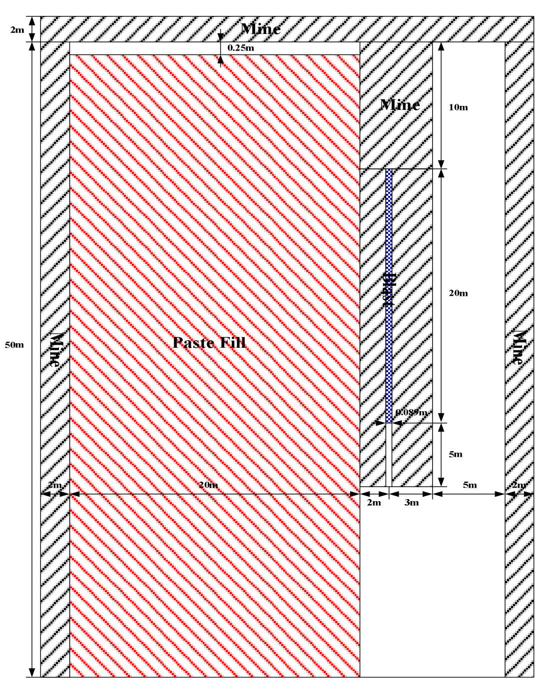

Figure 4 Geometry of the analysis model for blast simulation

The model is developed using the finite-discrete element program ELFEN. As it is the simulation of blast in underground mines, the non-reflecting boundary is assumed in the surrounding boundary of the model. Furthermore, the bottom edge is fixed, the top edge is fixed in the vertical direction, the left and right edges are fixed in the horizontal direction, and other inside boundaries are free. The model is meshed with triangular elements by Delaunay triangulation method. The material properties used in this simulation are listed in Table 1.

Table 1 Material properties

	Paste Fill	Mine Rock
Young's Modulus (GPa)	5.5	95
Poisson's Ratio	0.2	0.2
Density (kg/m^3)	2100	2700
Cohesion (MPa)	0.179	0.2
Damping	0.0638	0.03
Tensile Strength (MPa)	6	5
Fracture Energy (J/mm)	0.06	0.06

A time-varying pressure pulse function is used to model the blast loading for the numerical model, which is applied on the walls of an "equivalent cavity" as defined by (Sharpe [10]).

$$P(t) = t^n p_0 e^{-xt} \quad (1)$$

where t is the duration of blast loading, and $P(t)$ is the pressure at time t. p_0 denotes the initial peak pressure value. p_0, n and x are constants, which determine the shape of the pressure curve. p_0 is chosen so that a maximum pressure of 2.255×10^8 Pa is achieved. In the paper, the values $n=1$ and $x=2000$ are used.

3 Results and Discussion

The dynamic fracture process starts from a continuum representation of the solid material by finite elements. Fracturing in the model is controlled according to a fracture criterion specified through a constitutive model, i.e. Rankine crack model. The processes of the fractures and fragmentations from $t=10\mu s$ up to $t=350\mu s$ are shown in Fig. 5, in which the black lines indicate cracks, and the red and blue zones are paste fill and mine rock, respectively.

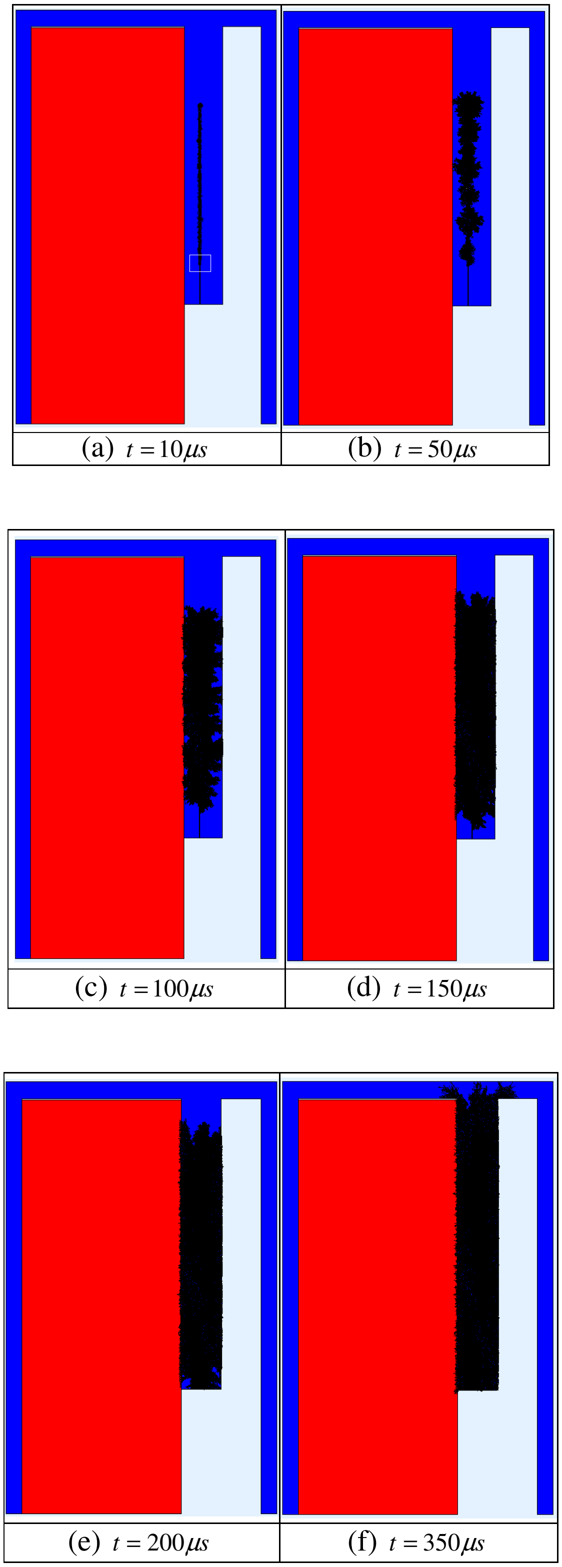

Figure 5 Fracture patterns at different times

Due to the short detonation period that ends at about $t = 5\mu s$, the compressive and tensile stress fields should radiate from the blast hole. Fig. 5(a) shows initiation and propagation of a large number of radial cracks starting around the blast hole, i.e. the vicinity of the compressive failure zone. Then the tensile stress field causes the radial cracks to extend, which is then followed by stress releases around the extending cracks. The differences in the crack growth velocities increase as time increases. This is because the cracking is affected by different spatial microscopic strengths for an applied stress level. As the fracture processes continue extending, a large number of separate interacting fragments, i.e., discrete elements are created,. This will result in further fracturing of both the remaining continuum and previously created discrete elements.

The peak velocities of individual material points along different height levels in paste fill are shown in Figs. 6-11. It can be observed that the peak velocities decrease quickly as the distance from blast hole increases. The peak velocities in Figs. 6 and 11 are lower than in Figs. 7-10. This is because the horizontal surfaces at various heights are vertical (what does this mean?) to the blast hole and the individual material points in Figs. 7-10 are closer to the blast hole. Figs. 12-14 show velocities against times at material points I, II and III, as specified in Figs. 6-8. It is noteworthy that the velocities of these points are still subject to violent ocillation after the peak phase as the time increases. An explanation for this is that the processes of the fractures and fragmentation result not only from detonation of explosive but also from considerable fragments interacting each other.

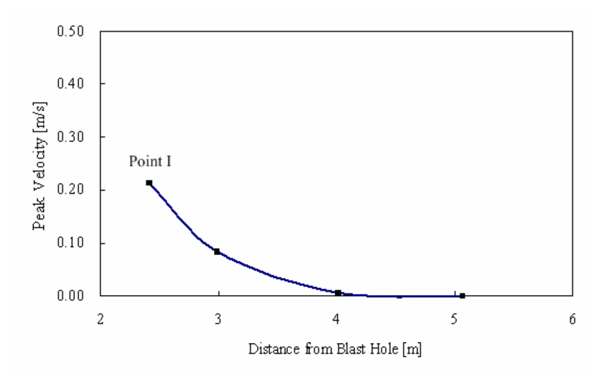

Figure 6 Peak velocity at height of 45 m in paste fill

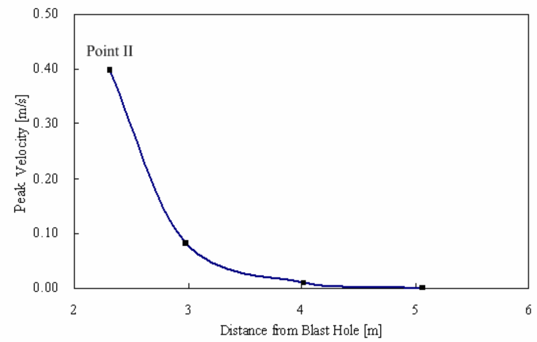

Figure 7 Peak velocity at height of 40 m in paste fill

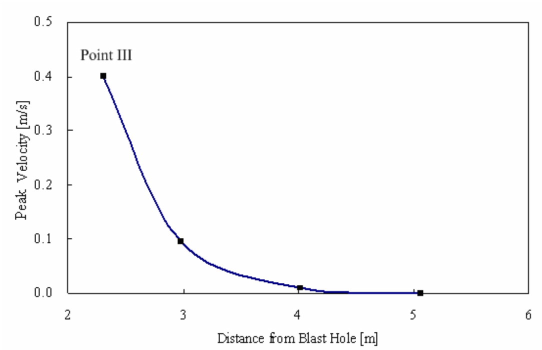

Figure 8 Peak velocity at height of 35 m in paste fill

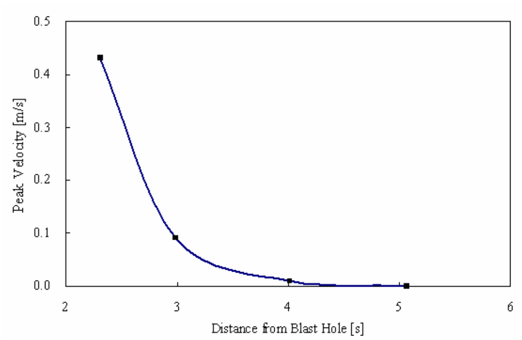

Figure 9 Peak velocity at height of 30 m in paste fill

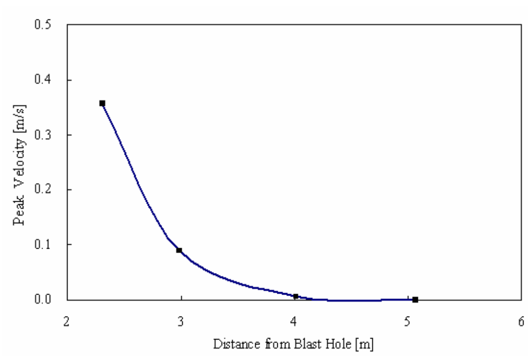

Figure 10 Peak velocity at height of 25 m in paste fill

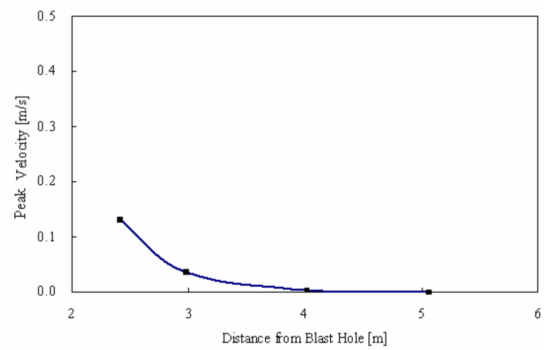

Figure 11 Peak velocity at height of 20 m in paste fill

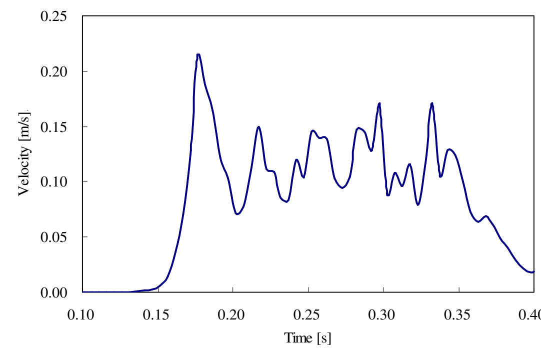

Figure 12 Velocity at Point I (See Fig. 6)

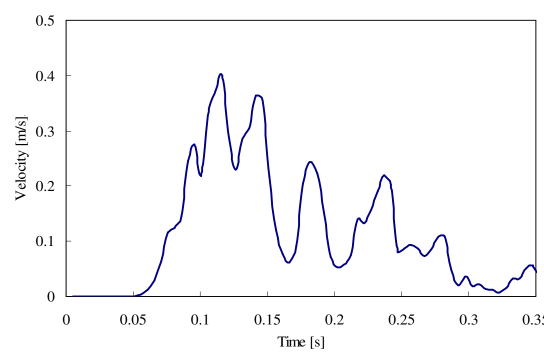

Figure 13 Velocity at Point II (See Fig.7)

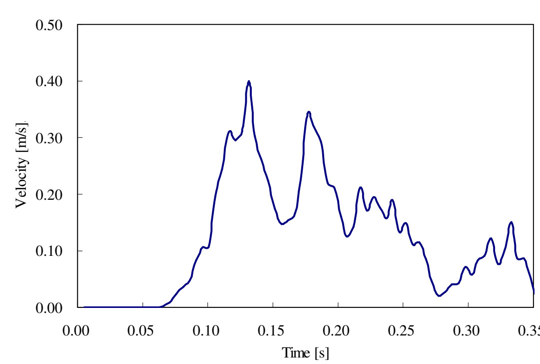

Figure 14 Velocity at Point III (See Fig. 8)

4 Conclusion

A real-size model has been developed using a combined finite-discrete element program ELFEN to predict and simulate backfill stope failure risks due to blast loading effects in underground mines. The dynamic fracture processes are shown starting from detonation of explosive to fracturing to separate interacting fragments. The extensions of fracture processes related to blast-induced hole breakdown start from compressive stress fields to tensile stress fields, and mainly result in the motion and interaction of separate fragments. The velocities for individual material points in paste fill have been determined to investigate the local and regional stability. The combined finite-discrete element method has demonstrated that it is possible to have significant potential to predict failure risks of backfilled stopes and provide assistance in mining design.

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