CONTINUUM DAMAGE MODELING OF ROCKS UNDER BLAST LOADING

by

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Abstract

Rock fragmentation by blasting has been in practice in civil and mining industries for centuries. The controlled use of explosives is considered the dominant approach for the purpose of breaking the rock material in any hard rock mining. The fracturing process of rock material, when subjected to blast loading, is a complex phenomenon and requires substantial study. Obtaining desired results in any rock blasting project demands for extensive understanding in two separate engineering fields. The first field of study should focus on the mechanics of dynamic fracturing of brittle rock material in response to blast loading. Given the inherent nature of the blast phenomenon, care should be given in the rock mechanics studies, and the analysis should be handled considering the dynamic behavior of the material. The second field comprises the study of the behavior of stress waves in brittle materials, the controlling parameters of wave attenuation, and the effect of stress wave interactions on dynamic fracturing of rock mass. In this thesis, the strain rate dependency of dynamic tensile strength of Laurentian granite is investigated by the aid of different experimental methods i.e. Hopkinson bar experiments and Split Hopkinson Pressure Bar (SHPB) experiments. The obtained results were combined and implemented as Dynamic Increase Factor (DIF) in the RHT material model using LS-DYNA numerical code. Using the modified RHT material model, two distinct rock blasting problems are studied numerically. First, the effect of stress wave interaction on the resulted rock damage and fragmentation is investigated using a range of initiation delay times. Different scenarios of wave superposition are examined and an optimum delay window is introduced based on the elastic stress wave theory. Second, the effectiveness of current destress blasting practices in burst prone deep underground excavations is studied. The capability of the conventional destressing patterns in alleviating the burst potential is explored by damage investigation and stress studies ahead of a tunnel face before and after destressing. A new destressing pattern is introduced which is successfully capable of transforming the stress states ahead of a tunnel face, reducing the rockburst potential.
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Chapter 1
General Introduction

1.1 Problem statement

Rock breakage by blasting has been in practice in civil and mining industry for centuries. The first application of explosives in mining dates back to early 17th century when black powder was used as an alternative to mechanical tools (Buffington, 2000) to fragment rock. Since then, the controlled use of explosives is considered the dominant approach for the purpose of fracturing the rock material in any hard rock mining. Even though the emergence of Tunnel Boring Machines promoted the mechanical rock breakage in tunnel construction, there still exists a great deal of underground spaces in civil engineering where blasting is the only way of breaking the rock economically.

Mining is considered as the frontier industry in responding to the ever-growing demand of humanity to the raw materials. To keep up with such demand, and considering the limited amount of mineral resources on the earth’s crust, mines go deeper and exploitation of low grade resources become economically justifiable. Meanwhile, new technologies introduced in mining assist the industry to fulfill its objectives much safer and in the cheapest way possible. Intelligent mining systems with autonomous haulage and loading units operating in either underground or surface mines to improve productivity, reliability, and safety of the mining operations are now introduced while automation in many more areas is driving research and development efforts worldwide. In mineral processing, leading companies are investing in new technologies focused on sensor based extraction in order to improve the ore recovery.

The commercial explosives provider companies have also incorporated the latest technology in their products in an attempt to achieve safer and more economic blasting practices. Electronic detonators are introduced to the mining industry since late 1990s, in which the detonating cap is initiated by the aid of electronic chipsets providing less scatter in the initiation timing, which allows better control of the ground vibration and rock fragmentation. In recent years, wireless initiating systems are also introduced
to the mining industry. By eliminating the physical connectors, wireless systems allow for automation of the blasting process to enhance operator safety, improve the ore recovery, and reduce the drilling costs in underground mining methods.

While the new technologies are being introduced and are available for the mine blasting sector, it seems the blasting science is not evolving and keeping up well with the cutting-edge technologies. As a result, mines are not able to benefit from these technologies on their entirety. For instance, the electronic detonators have programmability feature to increments of sub-milliseconds; however there is no agreement on the optimum initiation delay timing which delivers the required fragmentation distribution. These precise detonators provide the ability to benefit from the wave superposition in favor of rock damage and fragmentation, however, the behavior of the stress waves emanating from different locations within the rock mass and the response of the rock material to such dynamic loading is not accurately understood and incorporated in the blast designs. This is originated from the fact that there exists a great deal of empirical designing procedures, which almost dominated the engineering field of rock fracturing by blasting. Empirical methods in blast design practices are usually obtained as a result of statistical analysis performed on data from a number of field tests, however, given the mechanical complexity of the behavior of the rock mass, the dynamic fracturing response of it should be studied in great detail and with a scientific approach. Given the destructive nature of the process of rock blasting, it appears that once the obtained blast results i.e. vibration, fragment size distribution, etc. remain within the desired values range, there is not enough keenness to try to optimize the obtained results. This becomes very important considering the downstream effects of blast result. In the following, two different examples will be provided in which the potential for improvement of the obtained results will be discussed.

The first example deals with the obtained fragment size distributions. In a general sense, once the fragmentation results of $X_{50}$, $X_{80}$, and $X_{max}$ remain in the desired crusher feed size requirement, the blast is considered a success. However, there might be a way to optimize the blasting practice with an effort to introduce further damage to the rock fragments. Introducing micro fractures generally leads to
reduction in the elasticity modulus of the medium, which eventually makes it easier to break. Noting that any rock fragment is formed as a result of coalescence of major cracks, there is a countless number of microcracks which may have been created during the dynamic loading of blasting but couldn’t get enough time to coalesce and create major cracks. These microstructures eventually affect the integrity of the intact rock and reduce its strength. Therefore, further breakage of such deteriorated medium in crushing or milling processes would consume less energy in comparison to an undisturbed rock medium. Considering the energy intensity of the milling process in the downstream of any given mining activity, it appears that modifying blasting practices in an attempt to not only reach desired fragment sizes but also to generate individual fragments with a certain level of micro fractures would provide value-added to the entire mining system by reducing the operating costs.

Preconditioning of the rock mass for the purpose of destressing deep underground spaces in burst prone rock conditions is one of the unique applications of blasting in mining. Like the previous example, empirical designs are dominantly applied, where destress blasting is suggested to alleviate the strain burst potential in underground mining stopes. The detailed mechanism of destressing by blasting has not been studied comprehensively. Therefore, the destressing rounds are carried out using a conventional pattern in which the effect of stress waves on creating blast induced damage zones and micro fractures are not studied well. Given the geometry of destressing patterns, combined with the effect of stress states around the underground openings, the effectiveness of current destressing practices is not verified. It is suspected that these conventional patterns may even result in creation of isolated damage zones, which eventually lead to stress concentration in the rest of rock mass, increasing the probability of strain bursts. Therefore, introducing a definitive design procedure, which emphasizes the effectiveness of implementing destressing, and recommending a case specific blast design geometry is imperative.

Obtaining specific results in any of the examples provided earlier, requires extensive understanding in two separate engineering fields, the field of fracturing and damage development and the effect of wave interactions in the rock mass. It is evident that the theory of blasting needs improvements, at least in the
previously mentioned areas. However, theoretical improvements need to be combined with measurements in the field. In a destructive environment measurements tend to be expensive and, as a result, few of them are obtained. In an environment of variable geology, complex geometry and non-ideal performance of explosives, the complexity of the task can be overwhelming. Hence simplifications need to be made in order to answer the critical questions of what is the effect of strain rate on damage and how damaged rock interacts with stress waves in practical blast design.

1.2 Approach/Methodology

1.2.1 Loading rate dependency in the mechanical properties of rock material

There is a broad spectrum of applications of hard rock blasting in civil, mining, and oil industry as stated previously. From blast pattern design for production purposes in open pit and underground mining, to wall control, smooth blasting, and destress blasting in deep underground spaces, to oil well stimulation, all are applications of the use of explosives which change the properties of a given rock mass either by producing visible fractures or by generating micro fractures. The instantaneous discharge of the energy of the explosive buried in the rock medium, results in a stress wave propagating radially around the blasthole. The study of the response of the rock mass due to the wave action is essential and critical in the behavior of rock material under high loading rate/strain rate condition. The majority of empirical designs consider the mechanical properties of the rock mass, which are achieved under quasi-static loading conditions. Such conditions have stress-strain relationships between $10^{-5}$ to $10^{-1}$ s$^{-1}$ (Liu et al. 2018). However, brittle materials show increased strength when subjected to loading conditions with higher strain rates (Liu et al. 2018). Increasing the loading rate reduces the chance of the micro fractures and the flaws within the rock material to grow and as a result the strength parameters improve. In order to quantify the degree of such improvement in rock strength parameters, the Dynamic Increase Factor (DIF) is introduced which is the ratio of the dynamic to static strength and is reported as a function of strain rate (s$^{-1}$). Figure 1-1 illustrates the DIF for compressive and tensile strength of different rock types.
Several experimental methods are used to achieve the strain rate dependency of the strength of brittle rock material. The Hopkinson Bar tests and the Split Hopkinson Pressure Bar (SHPB) tests are two of the mostly performed experiments to achieve DIF. Such experiments categorize in the first field of
investigation as stated in the last paragraph of section 1.1, providing insight for the engineers for a better understanding of the dynamic response of rock material when subjected to blast loading.

1.2.2 Experimental study of rock fragmentation by blasting

Optimization of rock blasting projects for the purpose of better fragmentation is a complex exercise. As the rock is typically heterogeneous and anisotropic, and data before, during and after blasting are collected but scarce, true optimization may be extremely difficult to achieve. Hence, one can discuss parameters that affect fragmentation in an effort to improve it. Changing the distribution of the explosive material within the area of interest i.e. rock burden, by the aid of adjusting the blasthole diameter, or modifying the blast pattern are methods to change fragmentation results. One of the most important parameters in fragmentation control is initiation timing and sequencing. The energy of explosive, once fired, is discharged in two distinct processes. Immediately after the initiation, a stress wave is created which emanates radially around the blasthole; the second process is the expansion of the detonation products applying pressure on the blasthole wall. The initiation and propagation of the blast induced cracks are majorly controlled by the effect stress waves (Ledoux, 2015). Expansion of the products of detonation, often called gas pressurization, is mostly responsible for the burden detachment and the heave of the rock mass, forming the rock pile (Brinkmann, 1990). It is evident that the stress waves emanating from different blastholes would collide or interact at some point within the rock mass. The question to answer is: How and at what point of time do stress waves interact? Wave interactions and constructive wave superposition create time-dependent zones of increased stress within the rock mass that may favor the fragmentation.

The study of the effect of timing on the rock damage and fragmentation seized more attention with the emergence of electronic detonators. In the literature, there is a great number of experimental works, which emphasized on the study of the effect of initiation delay timing on blast fragmentation (Stagg and Rholl, 1987; Katsabanis et al., 2006; Vanbrant and Espinosa, 2006; Johansson and Ouchterlony, 2013; Katsabanis et al., 2014). Figure 1-2 illustrates the results of particle sizes as a function of the initiation
delay timing in small-scale blocks of granodiorite (Katsabanis et al., 2006). The obtained results suggest that there is an optimum delay period in which the achieved fragment sizes of \( X_{50} \), \( X_{80} \) are minimum. However there are very few points in the graph, which, other than the coarse fragmentation at simultaneous initiation, may show no other trend with respect to delay (Blair, 2009). Furthermore, stress wave propagation and analysis was not monitored, nor was it possible, in these experiments.

The situation is quite typical where there are not many experimental measurements and the understanding of the phenomena is poor. A possible solution for such cases in rock mechanics is numerical modelling, where the model becomes a simplification of reality (Starfield and Cundall, 1988). Hence, the interaction of stress waves, in a simplified version of the experiment, can be studied using numerical modelling in order to explain the results of Figure 1-2. Simplified approaches to examine the effect of delay time have been presented in the past (Rossmanith, 2002; Vanbrant and Espinosa, 2006), while more experimental work was done by Johansson and Ouchterlony (2013) and Katsabanis et al. (2014). Explanation of the effect of delay on fragmentation is still unclear and worthy of investigation.

![Figure 1-2](image.png)

Figure 1-2- Effect of initiation delay timing on 50% and 80% passing sizes after (Katsabanis et al., 2006)
1.2.3 Numerical approaches in rock blasting problems

Computer aided models are suitable tools to investigate the stress wave behavior and the response of the rock material. Considering the limitation in data interpretation and the analysis of the fragmentation experiments, numerical models have the capability to provide a better insight towards the problem of blasting and fragmentation by illustrating the physical occurrences during the blasting process. Numerical tools are generally divided into two separate categories. Finite Element Methods (FEM) and Discrete Element Methods (DEM). FEM models treat the problem as a continuum, which consists of finite number of elements attached together, that cannot detach from each other. In the literature, FEM methods have been widely applied in the analysis of blasting problems. Liu (1996) investigated the effect of air decking and decoupling in blasting using a continuum solution. By demonstrating the physical processes, Liu’s work presented the potential benefits that can be obtained by applying air decking. Based on his work, a minimum beneficial length of air deck can be determined by the equilibrium of energy loss from primary loading to stemming and the energy which was gained from secondary loading. The study of the effect of initiation timing has also been conducted numerically using a continuum approach. Liu studied the fundamental mechanisms which are involved in rock fragmentation using initiation delay times and concluded that in hole wave collision only makes localized damage zones and does not benefit rock fragmentation.

DEM models are capable to simulate the behavior of rock mass as a conglomeration of distinct blocks, which are bounded together and have the capability to detach from one another. Therefore, in a DEM solution, the model is capable of illustrating the formation of the cracks within the medium. Recently Bonded Particle Methods (BPM) are also introduced in which the bonds between particles of the medium have the capability of modeling tension, bending, twisting and shearing between the particles where the rupture in the medium is modeled explicitly as broken bonds. Yi et al. (2018) investigated the effect of timing on fragmentation using a coupled FEM-BPM method and concluded that the simultaneous initiation results coarsest fragmentation. In comparison to FEM models, DEM models require more time and computational capacity. On the other hand, the development and the propagation of a single crack
is not of major concern in blasting engineering, the key indicator is how the blast induced crack system affects the overall mechanical properties of a given continuum.

1.2.3.1 Continuum damage mechanics
There is a distinct branch in fracture mechanics, which emphasizes in analyzing the material deterioration under mechanical loads and is addressed as continuum damage mechanics. In general, fracture mechanics deals with the load bearing capacity and the behavior of solids containing major macroscopic cracks that are embedded in a defect free continuum. However, continuum damage mechanics deals with the load bearing capacity of a damaged medium in which the mechanical properties are weakened in the presence of microcracks and flaws. The constitutive equations which represent the continuum damage mechanics, incorporate one or more scalar or tensorial parameters which represent the degree of degradation of the stiffness and the strength of the material as a result of microscopic crack growth (Krajcinovic and Lemaitre, 1987). As stated earlier, in blasting engineering, the principal objective is to understand the effect of a crack system, which is created as a result of blast loading, rather than the study of the development and the propagation of a single crack. Therefore, continuum damage mechanics appears to be the most representative approach in modeling blasting phenomenon. In general, isotropic damage is defined as a scalar parameter $0 \leq D \leq 1$; in which, $D=0$ corresponds to an undisturbed virgin material with no damage and $D=1$ represents a completely deteriorated material which is unable of carrying any load.

1.2.3.2 Constitutive models based on continuum damage mechanics
Continuum damage mechanics in fracture modelling of brittle material under dynamic loading has been successfully applied during the past 50 years. One of the early works is the formulation of continuum damage mechanics for the study of spalling due to a dynamic load, presented by Davison and Stevens (1973). Their damage parameter was formulated as a vector field describing the direction and magnitude of a disturbed penny-shaped crack. The damage evolution was controlled by crack growth laws determined by the variation of stress state at the point of fracture. One of the most successful damage
models was the one introduced by Grady and Kipp (1980). Their model, which was developed for the purpose of blast fragmentation assessment in oil shale, incorporated an isotropic damage and represented with a scalar parameter. The scalar variable of damage was defined in terms of the volume of idealized penny-shaped flaws in the material. Since their proposed model was simple, and provided an explicit procedure to determine the fragment sizes generated by coalescence of crack during the dynamic fragmentation process, it was extensively applied by other researchers (Boade et al., 1985; Digby et al., 1985; Kuszmaul, 1987; Thorne et al., 1990). Liu (1996) proposed a damage model in which a statistical approach was applied in describing the effect of micro fracture system. In his model, damage was defined as the probability of fracture at a given crack density. Based on his damage model, he incorporated an algorithm in order to predict the fragment sizes.

1.2.4 RHT damage model in LS-DYNA

Introduced by Riedel (2004), RHT is a material model incorporating the features that are capable of modelling dynamic behavior of concrete when subjected to high strain rate loading condition. The model is capable of considering the pressure hardening, strain hardening, strain rate hardening, and strain softening (damage effects). Damage is a scalar variable in this model described as

\[ D = \sum \frac{\Delta \varepsilon^P}{\varepsilon_f} \]  

where \( \Delta \varepsilon^P \) is the accumulated plastic strain and \( \varepsilon_f \) is the failure strain. Tawadrous (2010) studied the physical properties of Laurentian granite and studied the mechanical behavior of this rock type under static and dynamic loading condition. These properties were then used to calibrate the RHT material model for Laurentian granite and investigated its potential in modelling blast induced damage in brittle materials. Since the RHT constitutive model is available in numerical codes and given its capacity in simulating the dynamic response of rock material based on the previous calibration works, RHT was used as the material model in the numerical studies. In this thesis, more experimental work has been done to further investigate the strain rate dependency of Laurentian granite to achieve the Dynamic
Increase Factor which was used in the RHT model. For the purpose of modeling of blasting and fragmentation problems, LS-DYNA-a nonlinear transient finite element code-is used. The code is successfully capable of modeling solid material behavior by the aid of Lagrangian element formulation. It also features space-fixed Eulerian element formulation for modeling fluid material behavior with large deformations, which is the case when modeling explosive materials. The code incorporates robust coupling mechanisms between Lagrangian and Eulerian element formulation when necessary.

1.3 Key research objectives

The purpose of this thesis was to explain some modern blasting issues, such as the effect of delay time on fragmentation, the effect of in situ stresses on damage evolution and finally to examine blast preconditioning of the rock mass for the purposes of distressing. Since detailed experimental measurements are unavailable, it was decided to use numerical modelling as a laboratory and examine the complex relationship between stress waves and stress wave interaction and damage. The code used for this purpose is the LS-DYNA code (*LS-DYNA Keyword User’s Manual, 2017*) and the damage model was the RHT model which was calibrated for one of the granites of previous experiments (Laurentian granite). The strain rate dependency of the tensile strength of Laurentian granite was studied by the aid of Hopkinson bar experiments and Split Hopkinson Pressure Bar (SHPB) experiments. The results were used to achieve the dynamic increase factor which was then incorporated in the RHT material model in LS-DYNA. With the aid of the calibrated RHT material model, the effect of initiation delay timing on blast induced rock damage and fragmentation is studied with an emphasis on the interaction and superposition of stress waves. The effect of in situ stresses on the shape and the extent of the blast damage zone was also studied numerically. Current conventional tunnel face destress blasting procedures were investigated to assess the effectiveness of current destressing patterns on alleviating the strain burst potential in deep underground burst prone environments. A new destressing pattern was also introduced which shows promise in mitigating the high stress concentration zones ahead of a tunnel development face. The key research objectives are outlined in the following:
• Investigation of the Hopkinson’s mechanism on evaluating the effect of loading rate dependency of the dynamic tensile strength of brittle rocks. Application of variety of experimental methods including Hopkinson’s bar experiments and Split Hopkinson Pressure Bar (SHPB) experiments to obtain the rate dependency behavior of the tensile strength.

• Numerical verification and calibration of the strain rate hardening constitutive models and investigation of the effect of the loading rate function on the resulted damage.

• Investigation of the blast induced stress pulse shape and duration on the resulted rock damage and fragmentation.

• Implementation of the calibrated numerical models to investigate the effect of initiation delay timing on the blast induced rock damage and fragment size distribution.

• Application of the study of the elastic stress waves interaction and superposition for optimization of damage and fragment size distribution in blasting practices.

• Study of the effect of in situ stresses in blast induced rock fracturing in underground blasting activities.

• Investigation of face distressing and preconditioning in deep hard rock mining and tunneling applications. Recommendation of a blasting procedure in rock burst prone underground environment in order to fulfill preconditioning requirements.

The thesis components forming the general framework of this research are illustrated in Figure 1-3.
1.4 Thesis Structure

This document is structured in the form of a manuscript-base thesis, in which all the subsequent chapters are, or will be submitted to scientific journal publications. The thesis consists of five chapters, the main objectives of which are outlined previously and are accomplished through three distinct contributions/chapters. The components of each chapter are as follows:

Chapter 1- General Introduction: provides an overview of the current technological advances in mining and blasting engineering illustrating the targeted areas of interest where more scientific insight towards blasting engineering is still required in order to thoroughly benefit from the available technological products. The key research objectives/contributions are outlined and the experimental and numerical approaches to accomplish the desired objectives are explained.
Chapter 2- *Loading rate dependency in dynamic tensile strength of brittle rocks: Experimental approach and Numerical modeling*: This chapter presents the practical approaches and experimental methods for obtaining the dynamic behavior of brittle rocks. Two distinct experimental approaches are applied to achieve the Dynamic Increase Factor (DIF) of the tensile strength of Laurentian granite. Different setups are experimented to achieve results from Hopkinson’s bar test. A wide range of strain rates $5 < \text{s}^{-1} < 50$ are achieved using a gas gun driven SHPB experiment illustrating the dynamic behavior of the tensile strength of Laurentian granite. The achieved DIF is applied in the RHT material model in LS-DYNA in order to investigate the damage evolution and stress wave behavior during testing.

Chapter 3- *The effect of stress wave interaction and delay timing on blast induced rock damage and fragmentation*: Using the modified RHT material model, this chapter deals with the numerical study of blast induced rock damage and fragmentation. Damage evolution as a function of initiation delay timing is investigated and a parametric study regarding the size of the burden and spacing in a given bench blasting problem is conducted. The effect of stress pulse shape and duration on damage evolution as a function of delay timing is also studied. Using image analysis techniques, the fragment size distributions as a function of initiation delay timing are studied. The damage results are compared with experimental results and an optimum delay window based on elastic stress wave theory is introduced.

Chapter 4- *Tunnel face preconditioning using destress blasting in deep underground excavations*: This chapter deals with the effectiveness of one of the unique applications of using explosives in order to make changes in the mechanical properties of a given rock mass. The conventional tunnel face destressing patterns are modelled numerically and the stress states ahead of a given tunnel face is studied before and after destress blasting. The shape and the extent of blast induced damage zones in the presence of a wide range of in situ stress states are investigated. A new destressing pattern is introduced with the capability of transforming the stress states ahead of a circular tunnel face in order to alleviate the strain burst potential in deep underground mines.
Chapter 5- Conclusions: The summary of the contributions and the concluding remarks are provided in this chapter. Recommendations for the improvements of the experimental and numerical results are addressed pointing out future work and research.
Chapter 2

Loading rate dependency in dynamic tensile strength of brittle rocks:
Experimental approach and Numerical modeling

2.1 Introduction

Rock blasting is an inseparable part of mining and many other rock engineering activities. For decades, regardless of the selected mining method, rock breakage from its in situ state has been achieved using explosives. In many major hard rock projects, blasting is still a practical alternative in comparison with the application of mechanical extraction methods. This illustrates the importance of understanding the blasting phenomenon and the response of the surrounding rock medium. Optimizing the blasting process can be carried out for a variety of objectives, including better fragmentation in mining activities, blast damage control in tunneling applications or preconditioning of the rock mass in deep mining projects. In any blasting optimization problem, it is important to understand how the stress waves and expanding gases generated by explosive materials interact with the rock mass. For this purpose, it is crucial to realize the chemical and mechanical properties of the explosives and the detonation products and the rock mechanics properties of the host rock. The mechanical properties of the rock material can be tested and studied in the laboratory in order to get the strength of the rock under different loading conditions. However, one should keep in mind that blasting is a dynamic phenomenon and therefore the response of the rock material to such dynamic loading conditions is not similar to the response in static laboratory tests.

Rock breakage using explosives occurs due to two major phenomena, the propagation of the stress wave from the initiation point into the rock medium, and the expansion of the detonation products (Hustrulid, 1999). It is the effect of the stress wave that generates radial cracks around the blasthole and also parallel cracks to the free face, which subsequently turn the in situ rock to a cracked medium (Olsson et al., 2002). The expanding gasses contribute to the propagation of previously formed cracks and move the
damaged rock to form a rock pile (Whittaker et al., 1992). The Hopkinson’s mechanism describes the creation of the tensile stress wave at the free face of the blast and the subsequent spalling of the material. The stress wave has a considerable loading rate and the response of the surrounding rock medium should be analyzed accordingly.

In this study, the strain rate dependency of the tensile strength of Laurentian granite is investigated experimentally and numerically. Hopkinson’s bar experiments were conducted in different configurations using rod shaped rock samples. Split Hopkinson Pressure Bar (SHPB) tests were carried out on waffle shaped Laurentian granite samples. The dynamic tensile strength of the samples were obtained at different strain rates and the results of both experiments were combined to achieve the rate dependency function. This function was then applied in the Riedel-Hiermaier-Thoma (RHT) constitutive model. The experiments were then modeled numerically using LS-DYNA, a nonlinear transient dynamic finite element analysis code and the results were compared against those obtained from the experiments.

2.2 Methods of testing dynamic tensile strength

Implementing the Hopkinson’s mechanism is a well-known method to measure the dynamic tensile strength of rocks. Kubota et al. (2008) estimated the dynamic tensile strength of sandstone using rock specimens of 60 mm diameter and 300 mm length. The specimen was subjected to dynamic loading at one end using an explosive material along with a PMMA pipe filled with water as an attenuator. By changing the length of the attenuator they were able to introduce different strain rates into the sample. The reflected wave on the other side of the sample caused tensile cracks and their location could be used to calculate the tensile strength. For Kimachi sandstone the dependency of dynamic tensile strength to the strain rate was estimated as

\[
\sigma_d = 4.78 \dot{\varepsilon}^{0.33} \tag{2-1}
\]

where \(\sigma_d\) was the tensile strength of the specimen and \(\dot{\varepsilon}\) was the strain rate.

Ho et al. (2003) compared the static and dynamic tensile strength of Inada granite and Tage tuff and investigated the dynamic tensile strength under different strain rates. Their dynamic tension tests, which
were achieved based on the Hopkinson’s effect, illustrated the strain rate dependency of the tensile strength of both rock samples. They studied the evolution of microcracks in different loading rates and observed that at high loading rates the number of microcracks increased in comparison to lower loading rate conditions. However, the crack arrests caused by the stress released from neighboring microcracks altered the creation of macro fractures leading to increased stress in the rock without formation of tensile cracks (Ho et al., 2003). In the literature, the difference between the dynamic and the static tensile strength has been reported in many places. Bacon (1962) used a pendulum to send a sharp pulse into the rock sample and reported the dynamic tensile strength of up to 4 times the static one. Rinehart (1965) stated the difference can be explained considering the fact that the microcracks and flaws in the rocks would have less opportunity to engage in fracturing process under dynamic loading, and by increasing the loading rate this opportunity even gets smaller, which results in higher strength. In his study, the difference between static and dynamic tensile strength was in the order of 6-10 times.

Dynamic Brazilian disc test is another method for determining the tensile strength of rocks indirectly. The test is accomplished using the Split Hopkinson Pressure Bar (SHPB). The General apparatus has three main parts: a striker bar, an incident bar, and a transmitted bar where the disc shaped rock sample is placed between the incident and the transmitted bars. The striker bar can be driven using a gas gun or a pendulum hammer where the latter has the capability to introduce lower strain rates than a gas gun driven SHPB. Zhu et al. (2015) used a pendulum hammer driven SHPB and tested samples under intermediate strain rate of 5.2 to 12.9 s\(^{-1}\). In a blasting process, the rock is subjected to strain rates ranging from \(10^4\) to \(10^3\) s\(^{-1}\) (Grady and Kipp, 1980; Chong et al., 1980). Depending on the structure of the machine and the loading method, the SHPB experiment is capable of exerting strain rates in the range of 10 to \(10^4\) s\(^{-1}\) (Ross et al., 1989; Wang et al., 2006). Dai et al. (2010) used a copper disc along with a rubber disc as a pulse shaper attached to the incident bar, in order to be able to send a ramped pulse rather than a conventional rectangular pulse. Xia et al. (2017) stated that, such pulse shaping leads to much better force balance in the sample. Xia et al. (2008) performed a series of tests on Laurentian
granite in order to investigate the dynamic tensile strength. In their experiments, they were able to reach the loading rate of 3000 GPa/s which showed the strength increase of up to 4 times in comparison to the static tensile strength of Laurentian granite.

2.3 Experiments

2.3.1 Mechanical properties of Laurentian granite

In this study, Laurentian granite was selected for experimental work. Laurentian granite is a fine-grained, homogeneous and isotropic granite (Dai and Xia, 2009). Its grain size varies from 0.2 to 2 mm and makes it a suitable rock sample for split Hopkinson pressure bar tests considering the grain size to sample size criterion suggested by Zhou et al (2012). The mechanical properties of Laurentian granite were measured and are given in Table 2-1.

<table>
<thead>
<tr>
<th>Mechanical properties of Laurentian granite</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density (g/cm³)</td>
</tr>
<tr>
<td>Uniaxial Compressive strength (MPa)</td>
</tr>
<tr>
<td>Static Tensile strength (MPa)</td>
</tr>
<tr>
<td>Young’s Modulus (GPa)</td>
</tr>
<tr>
<td>Poisson’s Ratio</td>
</tr>
</tbody>
</table>

2.3.2 Hopkinson’s bar tests

For the purpose of the Hopkinson’s bar test, samples were shot using two different sets of explosives in order to study the response of the rock to different loading configurations. In the first set, the samples were shot using a detonator and a 2.2 gr primer (A3 RDX), together with a PMMA unit as a shock attenuator. The results achieved from each setup will be discussed later.

2.3.2.1 A3 RDX primer as shock wave generator (Set I)

The first set of Hopkinson’s bar tests had a detonator connected to a RDX primer as the shock wave generator component. The explosives were then attached to a PMMA unit in an attempt to attenuate the generated shock wave to a desired value, before entering the rock medium. Figure 2-1 shows the sample
configuration. The rock samples were drilled from a block of Laurentian granite and had a diameter of 26 mm. Due to limitations in the coring machine, the maximum achievable sample length was about 30 cm. The rock samples were then machined on both ends in order to have a smooth and perpendicular surface to the axis. The objective was to use a length of the attenuator such that the compressional stress in the rock specimen would generate a single tensile crack close to the free end, upon its reflection. The formation of a single crack greatly simplifies the analysis for the determination of the critical stress and strain rate in the sample. Four different length of PMMA units were tested: 12, 17, 27 and 37 mm with a diameter of 19 mm (3/4”). It was determined that a 27 mm long PMMA unit was able to reduce the magnitude of the stress wave in order to achieve a single tensile crack. Using shorter length of PMMA led to creation of multiple tensile cracks. In Figure 2-2, the sample at the top is shot using a 12 mm long PMMA attenuator while the sample at the bottom is shot with a 27 mm long PMMA. The magnitude of the compressional stress wave can be compared by considering the damage zone of both samples in the vicinity of the detonation point. The upper sample has significant compressional damage, while in the sample at the bottom, a piece of PMMA is still attached to the end of the rock sample, making the rock look visually intact.

![Figure 2-1-Schematics of Hopkinson’s bar sample using RDX A3 primer and PMMA as wave attenuator](image)

Figure 2-1-Schematics of Hopkinson’s bar sample using RDX A3 primer and PMMA as wave attenuator
Once the appropriate length of the PMMA unit was selected, at the second step of the experiment, the objective was to study the incident wave in the rock sample and its attenuation relationship along the sample. Capturing the incident wave form in the sample was very important for the future analysis of the experimental results. The wave length in the experiment is however significant and most strain records in the relatively short length of the rock specimen would fail to separate the incident from the reflected pulse. To overcome the issue, it was decided to use an extension of the rock sample glued to the free end. By having such configuration, it would be possible to separate the incident and the reflected pulses in a much better fashion and yet study the attenuation properties of the incident wave in a longer distance from the detonation point. By having the extended part, the overall sample length was about 600 mm. Once the samples were prepared, eight strain gauges, placed 40 mm apart, were installed along each sample. The installed gauges were 5 mm long, had a resistance of 120 ohms, and a gauge factor of 2.11. The sample configuration is provided in Figure 2-3. The gauges were then connected to the DatatrapTM data acquisition system.
In order to minimize contact of detonation products with the gauges and connecting wires, and improve the quality of the signals recorded by the gauges, the entire rock samples were covered by foam, which has a very low acoustic impedance and does not result in any wave reflections. Once connected to the data acquisition system, the sample was suspended in the air in order to prevent any wave reflections from the ground. The explosives were then detonated and the stress signals were captured from all the gauges. An example is shown in Figure 2-4, where the incident and reflected waves are shown. The separation of the pulses is clear in this Figure.

Figure 2-4-Strain recordings of Hopkinson’s bar experiment, complete separation of the incident and the reflected pulses
Since the explosives used are similar in both gauged and ungauged samples, one can assume that the incident pulse which was obtained from the gauge readings in longer samples is similar to the one which resulted in the single tensile crack in the short ungauged samples. Therefore, the tensile stress at the crack position can be considered the tensile strength of the rock sample under the specified strain rate.

2.3.2.2 Detonator as shock wave generator (Set II)

The second set of Hopkinson’s bar tests were conducted using a single detonator. Unlike the previous set, there was no attenuator used in this setup, as the generated shock wave was weakened by eliminating the use of the RDX primer. In order to minimize the effect of wave reflections from the surface of the specimen close to the detonation point, one end of each sample was covered by a grout cylinder having an external diameter of 100 mm and a length of 50 mm. The effect of axial wave reflections and the resulted damage on the pulse shape will be discussed later. Figure 2-5 shows the Hopkinson’s bar gauged samples with the extended grout section.

![Figure 2-5](image)

Figure 2-5-Configuration of the gauged samples of Hopkinson’s bar experiment where the first 50 mm of the sample is covered with grout to minimize the effect of wave reflections in the vicinity of detonation point

The gauges were then connected to the data acquisition system. Each sample was covered by foam, as described in the previous section, and hanged vertically with a detonator connected to the end of it. Once the detonator was fired, the incident and the reflected pulses were recorded from all the gauges. The experiment setup is shown in Figure 2-6.
Figure 2-6-Configuration of the Hopkinson’s bar experiment

Once the stress wave properties were captured, the next step was to shoot the samples without gauges. The explosive used was again a single detonator without RDX primer and no PMMA. Like in the
previous section, the objective was to create single tensile crack in the Laurentian granite. Samples with different length were shot with a detonator to achieve the optimum sample length that led to creation of a single tensile crack.

The wave attenuation properties of both methods are provided in Figure 2-7. As illustrated, the recordings show greater scatter in the case of the experiments with RDX and PMMA units. Using a detonator reduces the number of variables that generate scatter. The scattered recordings can be a result of scatter in primer strength, the length of the PMMA attenuator, the smoothness of the contact surfaces, or the quality of coupling between the rock, the PMMA, and the primer.

![Figure 2-7-Compressional stress wave attenuation along the Hopkinson’s bar experiment](image)

2.3.2.3 Modification of the compressional pulse
Creating a clean pulse with a steady strain rate and a single peak was found to be a challenge in conducting the Hopkinson’s bar experiment. Right after the stress wave enters the rock medium, wave reflections from the surface of the sample in the vicinity of the detonation point form a damage zone, and eventually contaminate the gauge readings. To overcome this issue, the first 50 mm of the rock samples were covered with grout. It was found that when using grout at the loading end of the sample, the pulses were cleaner as the grout extension eliminates the damage buildup due to the side reflections. As illustrated in Figure 2-7, this lead to less scattered pulse readings, especially at distances closer to the detonation point. Figure 2-8 shows the pulses with or without the grout extension in the vicinity of the detonation point. It must be noted that the donor charge is different between the two configurations and may have played a role in the pulse shape. The scatter of the observations in both amplitude and pulse shape received from Set I resulted in abandoning this set and concentrating the analysis on the results of Set II.

![The generated pulse in the sample without grout extension](image1)

*Figure 2-8-Effect of grout extension in the generated stress pulse*
2.3.3 Split Hopkinson Pressure Bar (SHPB) experiments

The SHPB system used in this study had a gas gun that accelerates the striker bar. The striker velocity was measured using piezoelectric film sensors located in the path of the striker bar. Two strain gauges were mounted in the middle of the incident and the transmitted bars and were connected to the data acquisition system. The installed gauges were 10 mm long, had a resistance of 120 ohms, and a gauge factor of 2.11. The bars used in this experiment are stainless steel with a diameter of 19 mm. The sample was sandwiched between the incident and the transmitted bars. In order to absorb the energy of the traveling stress wave at the end of the transmitted bar, ballistic gel was cased in a 100 mm diameter pipe and fixed to the chassis of the machine with 50 mm distance from the end of the transmitted bar. A schematic of the SHPB is shown in Figure 2-9.

Figure 2-9- Apparatus of the split Hopkinson pressure bar system

In order to keep the consistency of the study, the samples used in this experiment were cored from the same rock block where the Hopkinson’s bar samples were taken. The sample diameter as suggested by Zhou et al. (2012) was 26 mm.

The sample thickness of 16 mm was selected to be in the range of diameter to thickness ratio range of 0.5 to 1, recommended by Zhou et al. (2012). The sample thickness must also be smaller than the diameter of the steel bars, in order to provide a full contact between the bars and the sandwiched sample. The flat surfaces of the specimens were smooth, free of any irregularities and perpendicular to the axis. The tests were carried out starting from low pressure loading of the gas gun. The loading pressures ranged from 0.2 to 2 MPa (30 to 300 psi), providing a striker velocity of 3 to 30 m/s.
Once the strain signals captured, the incident, the reflected, and the transmitted strain values were available from the data acquisition system. The dynamic forces on either side of the specimen are as follows

\[ P_1(t) = AE(\varepsilon_i(t) + \varepsilon_r(t)) \]  \hspace{1cm} \text{(2-2)}
\[ P_2(t) = AE(\varepsilon_t(t)) \]  \hspace{1cm} \text{(2-3)}

where \( P_i(t) \) is the force history interacting on the incident bar-sample interface and \( P_2(t) \) is the force history interacting on the sample-transmitted bar interface. \( \varepsilon_i(t), \varepsilon_r(t), \varepsilon_t(t) \) are the incident, the reflected, and the transmitted strain histories, respectively. Therefore, the tensile stress history of the sample can be calculated as

\[ \sigma_t(t) = \frac{P_1(t)+P_2(t)}{\pi DT} \]  \hspace{1cm} \text{(2-4)}

where \( \sigma_t(t) \) is the tensile stress history of the specimen and \( D \) and \( T \) are the diameter and the thickness of the sample respectively. Once \( \sigma_t(t) \) is obtained, the peak of the graph can be considered as the tensile strength of the sample. The loading rate can also be achieved by measuring the slope of the linear zone of the \( \sigma_t(t) \) graph. By having the elastic modulus of the sample, this loading rate can be reported as the elastic strain rate in the sample.

2.3.4 Experimental results

2.3.4.1 Results of Hopkinson’s bar experiments

The shape and the specification of the incident pulse from all the gauges on the samples of Set II were collected and studied. Once the crack locations of the ungauged samples were measured, the data from the gauge that was mounted at the nearest location to the position of the crack was retrieved. Figure 2-10 shows the results of the pulse readings in the vicinity of the crack location when the explosive used was a single detonator.
Figure 2-10-Configuration of the compressional stress waves at the location of the tensile crack

Figure 2-11 shows the crack locations on the blank (ungauged) samples, the cracks are created in 25 mm to 35 mm from the free end of the samples.

The selected pulses represented strain rates in the range of 29.2 to 32.8 s\(^{-1}\) showing very low scatter as they were all calculated at the same location on the rock. The pulse was reflected at the free end of the bar and the incident and the reflected pulses were superimposed to calculate the maximum stress level at the crack position, assuming that the crack occurs when the peak of the tensile stress wave reaches the crack position (Kubota et al., 2008). In order to know the exact reflection time, the P wave velocity of the rock was used. Figure 2-12 shows the configuration of the pulses in the crack position of the rock sample.
Figure 2-11—Location of single tensile cracks near the end of the rock samples created as a result of stress wave reflection at the free surface

Figure 2-12—Configuration of the stress waves at the location of tensile crack
Once the strain values achieved, the tensile stress occurring on the rock in the crack position can be calculated using the elastic modulus of the rock medium. The average tensile strength of the Laurentian granite was reported to be 20.5 MPa under the strain rate of 31 s⁻¹.

2.3.4.2 Results of SHPB experiments

The strain recordings from the gauges mounted on both the incident bar and the transmitted bar were analyzed. The stress wave histories are provided in Figure 2-13 and Figure 2-14. The gauge on the incident bar has recorded the compressional incident strain $\varepsilon_i(t)$ and the reflected tensile strain $\varepsilon_r(t)$. The time difference between $\varepsilon_r(t)$ and $\varepsilon_i(t)$ in Figure 2-13 is the time in which the incident wave passes the gauge, travels to the incident bar-sample interface, reflects partially as a tensile wave, and then travels backward until it passes the gauge again. The second gauge readings shown in Figure 2-14 records the compressional wave that is transmitted from the sample into the bar $\varepsilon_t(t)$. In order to analyze the rock strength, all three of these waves should synchronize to time zero, the details of the process of wave superposition is provided in section 2.3.4.3. To extract the driving load on the incident bar-sample interface, $\varepsilon_r(t) + \varepsilon_i(t)$ should also be drawn.
Figure 2-13- Incident and reflected waves captured from the gauge mounted on the incident bar at striker velocity of 8 m/s

Figure 2-14- Transmitted wave captured from the gauge mounted on the transmit bar at striker velocity of 8 m/s
Figure 2-15 shows the results of the wave superposition. Once the superposition was done for each experiment, the stress history of the sample can be calculated using Equation 2-4 and is provided in Figure 2-16. The linear loading part of the curve was selected to calculate the dynamic loading rate. This loading rate was then converted to strain rate using the elastic modulus of the rock sample. The peak point in this curve was selected as the dynamic tensile strength of the specimen.

Figure 2-15-Superposition of the waves in SHPB system
Results of both Split Hopkinson pressure bar tests and Hopkinson’s bar tests are provided in Figure 2-17. The tensile strength of Laurentian granite under a wide range of loading rates is illustrated in this figure. The results obtained from Hopkinson’s bar tests are consistent with the results of SHPB tests at the strain rates near 30 s\(^{-1}\). The results obtained from strain rates under 10 s\(^{-1}\) are very close to the static Brazilian disc test results. The strain rates in the vicinity of 50 s\(^{-1}\) show an increase of the dynamic tensile strength of more than three times its static value.
In order to analyze the results of SHPB experiments and calculate the applied forces on the incident bar-sample interface, the incident $\varepsilon_i(t)$ and the reflected $\varepsilon_r(t)$ waves must be added together. To achieve this, it is necessary to translate both waves to a common time zero. The challenge was to obtain the wave arrival times. The Cumulative sum (CUSUM) method was first used to detect the change of slope in the strain – time histories in order to synchronize the rise time of both the incident and the reflected waves to time zero. Also, the result of $\varepsilon_i(t) + \varepsilon_r(t)$ must be a pure compressional wave. This was considered as an additional constraint to the analysis. In Figure 2-18, the pulse shown in dashed line was the result of the superposition of the incident and the reflected waves. The pulse shown in solid line was created using the CUSUM method without considering the pulse shape constraint, resulting in a tensile component in
the beginning, not consistent with the fact that the sample was under pure compression during the experiment.

![Graph](image)

**Figure 2-18- Effect of applying pulse shaping constraint on superposition of the incident and the reflected waves**

### 2.4 Numerical modeling

The simplicity of the experiments and the understanding of wave propagation in bars make them ideal to examine the results of numerical models used in blasting. In this study, LS-DYNA, developed by Livermore Software Technology Corporation (LSTC) was used as the numerical modeling code. The code is capable of nonlinear transient dynamic finite element analysis which makes it a suitable tool for modeling large deformations and dynamic events like shock wave propagation.

#### 2.4.1 RHT material model

Simulation of brittle material behavior under dynamic loading conditions has been of interest for researchers for the last few decades. Holmquist and Johnson (1993) have presented a constitutive model,
which was capable of simulating concrete and concrete-like materials’ behavior subject to large strains and high strain rates. Malvar et al. (1997) proposed a plasticity model for concrete material with the implementation of a third, independent yield surface. The RHT material model in LS-DYNA presented by Riedel and Borrvall (2011) is a comprehensive constitutive model capable of simulating the behavior of brittle material under dynamic loading conditions. The model is capable of considering the pressure hardening, strain hardening, strain rate hardening, and strain softening (damage effects) (Tu and Lu, 2010). The RHT constitutive model consists of three stress limit surfaces, which are illustrated in the meridian plane in Figure 2-19.

In order to apply the strain rate dependency of the tensile strength in the RHT model, a strain rate hardening law is implemented in the constitutive model. The Dynamic Increase Factor (DIF) represents the increase in the tensile strength as a function of the applied strain rate and the ratio is expressed as

$$DIF(\dot{\varepsilon}) = \left(\frac{\dot{\varepsilon}_p}{\dot{\varepsilon}_0}\right)^\beta$$

2-5

where $\beta$ is a material constant and $\dot{\varepsilon}_0$ is the reference strain rate and can be achieved using experimental data. $\dot{\varepsilon}_p$ is the measured strain rate. Eventually $DIF$ is a multiplier to the static strength in order to account for the increase of the tensile strength under dynamic loading configuration.

![Figure 2-19-Stress limits in RHT constitutive model (Leppanen, 2006)](image-url)
The strain rate dependency of the tensile strength of Laurentian granite was obtained as a combined result of Hopkinson’s bar experiments and SHPB experiments and was provided in Figure 2-17. Using the experimental results, $\beta$ and $\dot{\varepsilon}_0$ material constants can be derived as illustrated in Figure 2-20.

![Graph showing Dynamic Increase Factor (DIF) vs. Strain Rate (s^-1)]

$$DIF = \left( \frac{\dot{\varepsilon}_p}{36.65} \right)^{1.8247}$$

**Figure 2-20- Rate dependency of tensile strength of Laurentian granite and the implemented DIF in RHT constitutive model**

The measured material properties associated with Laurentian granite are provided in Table 2-2. The RHT strength parameters are provided in Table 2-3 (Yi et al., 2017).

**Table 2-2-Mechanical properties of the rock material used in RHT constitutive model**

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass Density</td>
<td>2.65 g/cm$^3$</td>
</tr>
<tr>
<td>Elastic modulus</td>
<td>65 GPa</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.27</td>
</tr>
<tr>
<td>Shear Modulus</td>
<td>25.6 GPa</td>
</tr>
<tr>
<td>Bulk Modulus</td>
<td>47.1 GPa</td>
</tr>
<tr>
<td>Compressive strength</td>
<td>210 MPa</td>
</tr>
<tr>
<td>Tensile strength</td>
<td>16 MPa</td>
</tr>
<tr>
<td>Ref. tensile strain rate</td>
<td>3.65e^-3 s^-1</td>
</tr>
<tr>
<td>Tensile strain rate dep. exponent $\beta_t$</td>
<td>1.82</td>
</tr>
</tbody>
</table>
Table 2-3-Parameters of RHT constitutive model

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Failure Surface constant A</td>
<td>2.6</td>
</tr>
<tr>
<td>Failure Surface Exponent N</td>
<td>0.8</td>
</tr>
<tr>
<td>Lode Angle Q0</td>
<td>0.567</td>
</tr>
<tr>
<td>Lode Angle B</td>
<td>0.0105</td>
</tr>
<tr>
<td>Initial porosity α</td>
<td>1</td>
</tr>
<tr>
<td>Compressive yield surface parameter</td>
<td>0.53</td>
</tr>
<tr>
<td>Tensile yield surface parameter</td>
<td>0.7</td>
</tr>
<tr>
<td>Damage parameter D1</td>
<td>0.04</td>
</tr>
<tr>
<td>Damage parameter D2</td>
<td>1</td>
</tr>
<tr>
<td>Residual surface parameter AF</td>
<td>0.873</td>
</tr>
<tr>
<td>Residual surface parameter NF</td>
<td>0.56</td>
</tr>
<tr>
<td>Pore crush B0</td>
<td>1.22</td>
</tr>
<tr>
<td>Pore crush B1</td>
<td>1.22</td>
</tr>
<tr>
<td>Crush pressure</td>
<td>133 MPa</td>
</tr>
<tr>
<td>Compaction pressure</td>
<td>6 GPa</td>
</tr>
<tr>
<td>Porosity Exponent</td>
<td>3</td>
</tr>
<tr>
<td>Ref. compressive stain rate</td>
<td>3e⁻⁸ s⁻¹</td>
</tr>
<tr>
<td>Failure tensile strain rate</td>
<td>3e²² s⁻¹</td>
</tr>
<tr>
<td>Failure compressive strain rate</td>
<td>3e²² s⁻¹</td>
</tr>
<tr>
<td>Compressive strain rate dependence exponent βc</td>
<td>0.032</td>
</tr>
<tr>
<td>Minimum damaged residual strain</td>
<td>0.01</td>
</tr>
</tbody>
</table>

2.4.2 Modeling SHPB test in LS-DYNA

The SHPB experiment was modeled in 3D configuration using solid elements. The rock sample was modeled using RHT material with a diameter of 26 mm and thickness of 16 mm and it was sandwiched between two metal bars modeled using the elastic material model. The properties of steel used in the model are provided in Table 2-4. The model configuration of the SHPB is provided in Figure 2-21.

Table 2-4- Elastic material properties for the steel bars in SHPB model

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density</td>
<td>8 g/cm³</td>
</tr>
<tr>
<td>Elastic modulus</td>
<td>200 GPa</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.3</td>
</tr>
<tr>
<td>P-wave velocity</td>
<td>5000 m/s</td>
</tr>
</tbody>
</table>
The SHPB experiment was simulated using impact velocities of 10, 20 and 30 m/s. The mass velocity histories of the striker bar are shown in Figure 2-22.
Once the striker bar that is traveling with a velocity of $V_1$ hits the inert incident bar, the resulting particle velocities $V_p$ in the either of the bars is as follows

$$V_1 - V_p = V_p - V_2$$ \hspace{1cm} (2-6)

In which $V_2$ is the initial velocity of the incident bar which is equal to zero. Therefore

$$V_p = \frac{V_1}{2}$$ \hspace{1cm} (2-7)

The generated compressive pulse traveling within the incident bar has a wavelength of twice the length of the striker bar. In the case of this study, the striker bar was 300 mm long, generating a wavelength of 600 mm. The duration of the pulse can be obtained by taking into account the P-wave velocity of the material in which the wave is traveling in

**Figure 2-22** The striker velocities and the post impact particle velocities in SHPB simulations
\[ L = \frac{2L_s}{C_p} \]  

where \( L_s \) is the length of the striker bar and \( C_p \) is the P-wave velocity. The compressive strain pulse is

\[ \varepsilon = \frac{u}{L} = \frac{u}{C_p t} = \frac{V_p}{C_p} \]

where \( \varepsilon \) is the elastic strain, \( u \) is the displacement and \( t \) is the time duration in which displacement of \( u \) occurs. The compressive elastic strain values can be converted into stress values by applying the elastic modulus of the steel material. The stress readings in the middle of the incident bar are provided in Figure 2-23.

![Graph](image)

**Figure 2-23**: The incident and the reflected waves resulted from a range of striker bar impact velocities as demonstrated in Figure 2-22.
The stress histories of the rock samples were calculated by the superposition of the incident, the reflected, and the transmitted pulses and the results are illustrated in Figure 2-24. The tensile strength of the sample was selected as the peak value of stress histories. The results of strain rate dependency of the rock samples from the numerical simulations are provided in Figure 2-25 in which the numerical results are compared against the experimental results.

![Stress history of the rock sample in the numerical simulations](image)

**Figure 2-24-Stress history of the rock sample in the numerical simulations**
Figure 2-25- Strain rate dependency results of numerical simulations vs. experiments

In the RHT material model, damage is a scalar parameter varying between 0 and 1 which demonstrates the material deterioration and the reduction in the strength of the material. The damage parameter is defined as

\[ D = \sum \frac{\Delta \varepsilon^p}{\varepsilon_f} \]

where \( \Delta \varepsilon^p \) is considered as the accumulated plastic strain and \( \varepsilon_f \) is the failure strain. Damage analysis of the RHT rock sample in SHPB simulations are provided in Figure 2-26. In this figure, the RHT damage inside the sample is synchronized with the tension level along the tensile stress history recorded for an element in the middle of the sample. As illustrated in this figure, the elements in the middle of the sample show no accumulated damage until the stress reaches almost 90% of the ultimate strength (Figure 2-26-c). At this point damage development is demonstrated in the elements in the middle of the rock sample.
where damage values of about 0.1 are recorded and rapidly this value, is also generated in the neighboring elements. Eventually, when the stress reaches the peak value (Figure 2-26-d), damaged elements form a zone across the sample, leading to the creation of major cracks in the direction of impact. Figure 2-26-f shows the final state of the damaged rock, where accumulated damage in the elements reached the value of about 0.6. Yi et al. considered this damage level as a macro crack, which is also consistent with the results of SHPB experiments conducted in this study.
2.4.3 Modeling of Hopkinson’s bar test in LS-DYNA

The Hopkinson’s bar experiment was modeled as a 2D axisymmetric simulation, using a Lagrangian grid. The Laurentian granite was modeled using the RHT constitutive model with shell elements (Tawadrous, 2010a). An attenuator was also applied in the model like the one used in the experiments. A polynomial Equation of State (EOS) (LS-DYNA Keyword User’s Manual, 2017) based on published
LASL shock hugoniot data (1980) was applied in order to account for the behavior of the PMMA attenuator in response to the shock loading due to detonation of the explosive. The pressure can be expressed as

$$P = C_0 + C_1 \mu + C_2 \mu^2 + C_3 \mu^3 + (C_4 + C_5 \mu + C_6 \mu^2)E$$

in which

$$\mu = \frac{\rho}{\rho_0} - 1$$

where $\rho$ is the density of the material and $\rho_0$ is the reference density, $P$ and $E$ are pressure and the initial internal energy respectively. Figure 2-27 shows the pressure as a function of $\mu$ in the PMMA material.

The PMMA material was also modeled with shell elements using a Lagrangian grid.

![Figure 2-27-Polynomial EOS for PMMA attenuator](image)

The explosive used in the model was the RDX A3 primer with properties given in Table 2-5.
Table 2-5-Material properties of RDX A3 primer

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Density</td>
<td>1.65 g/cm³</td>
</tr>
<tr>
<td>Velocity of Detonation</td>
<td>8.3 m/ms</td>
</tr>
<tr>
<td>CJ Pressure</td>
<td>30 GPa</td>
</tr>
</tbody>
</table>

In addition to the material properties mentioned in Table 2-5, an equation of state is also applied in order to account for the expansion of the detonation products. The JWL equation of state was used in which the pressure is defined as

\[ P = A \left( 1 - \frac{\omega}{R_1 V} \right) e^{-R_1 V} + B \left( 1 - \frac{\omega}{R_2 V} \right) e^{-R_2 V} + \frac{\omega E}{V'} \]  

2-13

where \( P \) is pressure, \( A, B, R_1, R_2 \) and \( \omega \) are material constants, \( V \) is the Volume, \( E \) and \( V' \) are the detonation energy per unit volume and the initial relative volume respectively. The input parameters for JWL equation of state for RDX A3 primer are provided in Table 2-6.

Table 2-6-JWL properties of RDX A3 primer (B. Dobartz and Crawford, 1985)

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>611 GPa</td>
</tr>
<tr>
<td>B</td>
<td>10.65 GPa</td>
</tr>
<tr>
<td>R_1</td>
<td>4.4</td>
</tr>
<tr>
<td>R_2</td>
<td>1.2</td>
</tr>
<tr>
<td>( \omega )</td>
<td>0.32</td>
</tr>
<tr>
<td>E</td>
<td>8.9 kJ/cm³</td>
</tr>
</tbody>
</table>

The explosive material was modeled using an Eulerian grid to allow the detonation products flow and expand within the grid. Once the explosive material is detonated, it will act as a fluid, applying forces on the nearby structure which is modeled using Lagrangian elements. In order to apply the proper coupling mechanism between the Eulerian and Lagrangian elements, LS-DYNA provided a Fluid-Structure Interaction (FSI) command, in which the Lagrangian part of the model is considered as the slave part and the Eulerian elements as the master (Yi, 2013). The configuration of the axisymmetric model of the Hopkinson’s bar test is illustrated in Figure 2-28.
Figure 2-28-Hopkinson’s bar model configuration in LS-DYNA

The location of the tensile crack as calculated from the model is provided in Figure 2-29 where it is compared against a representative experimental result. As mentioned previously, the elements in which the tensile crack has occurred have a damage value above 0.6, similar to the case of the SHPB simulations, therefore, the numerical results were in agreement with the experimental observations.

Figure 2-29--The location of the tensile crack from the Hopkinson’s bar experiment versus the damage location in the RHT simulation
2.5 Conclusion

The loading rate dependency of the tensile strength of Laurentian granite was analyzed by using Hopkinson’s bar and the SHPB experiments. In the case of the Hopkinson bar experiments, strain pulses were recorded at multiple locations of a long bar of granite, impacted by an explosive source. The recorded strain pulses were affected by wave reflections close to the impact zone of the specimen. Eliminating those reflections resulted in clean strain histories that could be interpreted to obtain the tensile strength of the material. Shorter cylindrical samples were impacted by the same explosive source to generate one tensile fracture at the end of the specimen. The tensile strength of the sample and the corresponding strain rate were obtained using the pulses recorded in the longer sample, reflecting the pulse close to the location of the crack at the free end of the sample and superposing the incident and reflected pulses at the location of the crack.

SHPB experiments were conducted using striker velocities from 3 to 30 m/s. Application of lower striker velocities, where the strain rate was less than 7 s\(^{-1}\), resulted in dynamic tensile strength values closer to the static values obtained by Brazilian disc tests. At higher impact velocities, where the strain rate was 55 s\(^{-1}\), the dynamic tensile strength values were more than three times greater than the static values.

The experiments provided the strain rate dependency of the tensile strength of the Laurentian granite which was applied in the RHT constitutive model. The experiments were also modeled using LS-DYNA code. The results of the numerical models were compared against those obtained from the experiments.

The evolution of damage parameter in the RHT material model in the rock samples of the SHPB simulation and creation of tensile crack in the Hopkinson’s bar simulation was analyzed and discussed.
Chapter 3
The effect of stress wave interaction and delay timing on blast induced rock damage and fragmentation

3.1 Introduction
Rock breakage and fragmentation using explosive materials have been practiced in the mining industry for decades. The immediate discharge of a considerable amount of chemical energy of the explosive buried within the rock mass is enough to overcome the strength of the surrounding rock material and break it, Persson et al. (1993). The rock breakage and its detachment occur because of two distinct mechanisms. Right after the detonation, a compressional stress wave hits the blasthole wall and expands radially within the rock mass, creating radial cracks around the blasthole. Once the wave arrives at a free surface, it reflects and travels back into the rock medium in the form of a tensile wave causing spalling and creating cracks parallel to the free face. The second mechanism is the expansion of the products of detonation, applying pressure into the blasthole wall. Although penetration of the gases into the previously formed cracks may contribute to rock fragmentation, the majority of rock damage during the blasting relies on the effect of the stress waves. Ledoux (2015) studied the role of the stress waves and the gases in the development of fragmentation. He applied different strategies to isolate the effect of gases and the stress waves and realized the effect of gases on the rock damage was statistically insignificant or visually indistinguishable. Hence, the energy of the expanding gases may be assumed to contribute to the movement of the rock burden and heave.

For researchers and engineers, the optimization of rock blasting projects for the purpose of better fragmentation has been studied for a long time. One of the most important variables in these studies is the effect of timing in fragmentation. Considering the dynamic nature of blast phenomena, it is obvious that the stress waves emanating from different blastholes would interact at some point within the rock mass. The question to examine is what this interaction means for fragmentation and damage in the rock
mass. With the emerge of electronic detonators in the late 1980s, the study of the effect of timing on rock damage and fragmentation seized more attention by researchers and engineers alike. Unlike the pyrotechnic and electric detonators, electronic detonators are programmable to 1 ms delay increments or less, with virtually no scatter, providing a precise initiation timing for each blasthole. Rossmanith (2002) studied the effect of precise initiation blasting by the use of Lagrange diagrams in 1D. In his research, he studied the mechanisms of wave interactions for two adjacent blastholes. His findings were in favor of applying short delay times between the initiation of two blastholes such that the stress waves interact in a region between the blastholes to achieve a uniform fragmentation. Inspired by the work of Rossmanith (2002), Vanbrant and Espinosa (2006) conducted a series of experiments using short delays aiming to optimize fragmentation by overlapping waves and enhancing the tensile tail of the stress waves. In their field tests, they claimed that their $x_{50}$ parameter was improved by 45.6% compared to the case where long delay times between blastholes were applied.

The investigation of stress wave interactions between two blastholes and the use of short delays to improve fragmentation has been studied by Johansson and Ouchterlony (2013). Using model scale bench blasting experiments, they used inter-hole delayed initiations. The delay times ranged from a state in which the P-wave from the neighboring blasthole has not arrived at the second blasthole to a state in which the S-wave from the neighboring blasthole has well passed the second hole. Their studies showed that there is not enough experimental evidence to suggest an optimum delay time in which the fragmentation is minimum. Furthermore, Yi et al. (2016) and Yi et al.(2017) studied the interaction of stress waves between two adjacent blastholes theoretically and numerically. In their studies, they indicated that in order to have two stress waves interacting between two blastholes, the delay time, which is a function of wave duration, the spacing between blastholes, and the P-wave velocity, should be very short. However, they stated that even if stress wave superposition occurs, the increase in the tensile stress happens in a very small zone around the collision point. As a result, they stated that the improvement of fragmentation by stress wave superposition is impossible. Schill and Sjoberg (2012) studied the effect
of precise initiation on blast induced rock damage and fragmentation in a two-hole 3D numerical model in LS-DYNA. They used the RHT damage model and stated that even though there is an effect of interacting stress waves on the resulted fragmentation, this effect is local and only around the plane of interaction, stating that precise initiation and delay timing does not lead to a dramatic fragmentation improvement as proposed by Rossmanith. Stagg and Rholl (1987) studied the effect of accurate delays on fragmentation and stated that the optimum fragmentation can be achieved when the second blasthole is initiated at a time in which the failure process of the previous blasthole has reached its final state. Katsabanis et al. (2006) conducted a series of small scale tests on granodiorite and concluded that coarse fragmentation results from simultaneous initiation of charges. They stated that fragment size decreases by increasing the delay time until they reach a constant size for a delay of 11ms/m of burden. Furthermore, Katsabanis et al. (2014) also conducted small scale experiments using grout samples and concluded that very short delays result in coarse fragmentation. They also stated that there is a wide range of delays where fragmentation is fairly constant. Blair (2009) questioned the application of delay times and accurate initiation timing for the control of fragmentation and concluded that there are other parameters that control the fragmentation and blast performance.

The main objective of the optimization of rock fragmentation is to try to achieve minimum energy consumption in the downstream of the mining process when the rock pile is fed into the crusher or when grinding takes place. Therefore, not only the fragment size distribution should be studied and optimized, but also the quality and the integrity of the fragments is of interest. Noting that any rock fragment is formed as a result of coalescence of major cracks, there is a countless number of microcracks which may be created during the dynamic loading of blasting but cannot get enough time to coalesce and create major cracks. These microstructures eventually affect the integrity of the intact rock and reduce its strength. Therefore, further breakage of such deteriorated medium in crushing or milling processes would consume less energy in comparison to an undisturbed rock medium. Katsabanis and Gkikizas (2016) studied the effect of initiation timing on the overall burden damage using numerical modeling.
In their work, they studied the fraction of damaged elements as a function of delay timing. However, they only studied major cracks (failed elements). Their results showed scatter in short delay times, while in longer delay times there wasn’t a significant dependency of damage to initiation delay timing.

In this study, the effect of initiation timing of the blastholes on the damage distribution is analyzed numerically. The work is divided into two major parts. First, in small scale modeling, the blast induced burden damage was investigated as a function of the initiation delay timing. In the second part, large scale bench blasting models were used and both damage and fragment size distributions were studied.

For this purpose, LS-DYNA, a nonlinear transient dynamic finite element code, was used. The RHT model, calibrated for the effect of strain rate on strength was used to calculate damage development. An axisymmetric model was first developed in each part and the response of the borehole wall in contact with the explosive charge was studied during blasting. The pressure time history of the wall of the borehole was used as a boundary condition for the borehole wall in the plane strain model, to analyze the effect of the burden size or the effect of timing once the optimum burden size was selected.

### 3.2 Material and model

In order to model the rock material, the RHT material model was used. RHT model in LS-DYNA, developed by Riedel and Borrvall (2011), is an advanced plasticity model to analyze the behavior of brittle structures subjected to impulsive and dynamic loadings. RHT uses a normalized form of effective plastic strain as a damage function, which is a scalar parameter, increasing monotonically whenever the state of the stress in the material is on the yield surface. The damage is defined as explained in Equation 2-10. The rock material properties representing the mechanical properties of Laurentian Granite are provided in Table 2-1. Laurentian granite is a fine grained granite which its properties has been studied well by Kim (2010) and experimental work has been done by the authors (Hashemi and Katsabanis, 2018) to capture its dynamic strength behavior.

The modeling procedure in LS-DYNA is divided into two separate steps. In the first step, an axisymmetric model consisting of a single blasthole was modeled. The objective in this step was to
capture the stress wave history on the blasthole wall resulting from the detonation of an explosive. In the second step, a pulse obtained at a consistent location at the wall of the borehole, was used as a boundary condition in plane strain models of several boreholes firing simultaneously or sequentially. Following such a procedure reduces the calculation times for the plane strain models since the complication of explosive modeling was handled in the axisymmetric model. The modeling procedures of each steps are as follows.

3.2.1 Axisymmetric model

3.2.1.1 Small scale

Using the RHT material model, a cylindrical rock specimen with a radius of 250 mm and length of 500 mm was modeled having a 6 mm diameter blasthole in the center of the cylinder. A column of explosive, simulating a 24gr/m PETN detonating cord was modeled using an Eulerian grid. To calculate the expansion of the detonation products the JWL equation of state was used. The input parameters for JWL equation of state as well as the material properties for PETN are provided in Table 3-1.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>573 GPa</td>
</tr>
<tr>
<td>B</td>
<td>20.16 GPa</td>
</tr>
<tr>
<td>$R_1$</td>
<td>6</td>
</tr>
<tr>
<td>$R_2$</td>
<td>1.8</td>
</tr>
<tr>
<td>$\omega$</td>
<td>0.28</td>
</tr>
<tr>
<td>$E_0$ (initial energy)</td>
<td>7.19 kJ/cm$^3$</td>
</tr>
<tr>
<td>Density</td>
<td>1.26 gr/cm$^3$</td>
</tr>
<tr>
<td>Detonation Velocity</td>
<td>6540 m/s</td>
</tr>
<tr>
<td>Detonation Pressure (CJ Pressure)</td>
<td>14 GPa</td>
</tr>
</tbody>
</table>

Water was used as a coupling medium between the explosive and the rock that was also modeled using Eulerian grid. In comparison to air decoupling, water is commonly used in small scale experiments, Katsabanis et al. (2014), as a decoupling medium to avoid substantial loss of energy, affecting fragmentation and requiring a significant increase of explosive consumption. To describe the response of water to explosive loading, an equation of state is required to account for the compression of the water.
inside the blasthole. The Gruneisen equation of state with C and S values of 1.48 km/s and 1.64 were applied respectively, where C is the intercept of the \( U_s(U_p) \) curve in velocity units, and S is the unitless coefficient of the slope of the \( U_s(U_p) \) curve. The data were obtained from the experimental results of the Los Alamos Science Laboratory (1980).

Since the RHT material was modeled using Lagrangian element formulation, a coupling mechanism should be applied between Eulerian and Lagrangian media. This is fulfilled using the Fluid-Structure Interaction (FSI) command in LS-DYNA. FSI considers the Eulerian elements as master parts and the Lagrangian elements as the slave parts. The contact properties were obtained from the LS-DYNA manual in which it is recommended that Lagrangian elements be larger than the Eulerian elements in order to have precise contact and prevent any possibility of leakage (LS-DYNA Theory Manual, 2018).

The model was built using quadrilateral elements with an element size of 1 mm for Lagrangian and 0.2 mm for the Eulerian elements. Transmit boundaries were selected for all surfaces of the cylindrical block in order to avoid the effect of wave reflections in the boundaries. Considering the detonation velocity of the explosive, the model was run for 80 \( \mu \)s to let the entire explosive column detonate. Strain gauges were placed in the middle section of the wall of the blasthole to capture the pulse, see Figure 3-1.
Figure 3-1-Details of the axisymmetric model: the explosive and water are modeled in Eulerian element formulation and the rock in Lagrangian formulation. The Eulerian elements are coupled and contained in a virtual container filled with void material which its elements overlap with the Lagrangian elements. The strain gauge is located on the very first Lagrangian element on the blasthole wall.

Figure 3-2 illustrates the contours of radial stresses around the blasthole wall as a result of the explosive detonation. The figure also shows the reflection of the shock wave at the contact zone of the water and the rock. The reflected wave hits the blasthole wall multiple times until it attenuates. The effect of wave reflection on the enhancement of the damage zone will be discussed later. The stress pulse captured in the midpoint of the blasthole wall is illustrated in Figure 3-3. The reverberations in the pulse are due to the reflections described earlier.
Figure 3-2-Stress contours around the blasthole wall, 40 µs after detonation. The shock wave within the Eulerian medium reflects upon contact on the blasthole wall and hits the wall multiple times during simulation causing blasthole wall reverberations. The cyclic loading effect attenuates and becomes insignificant after multiple contacts with the Lagrangian rock elements.
Figure 3-3-Stress pulse acting on the blasthole wall of the small scale model as a result of the explosive detonation. The effect of the cyclic loading and the blasthole wall reverberation is shown where after four cycles it attenuates and the rest of the pulse is eliminated and considered zero.

3.2.1.2 Large scale

Following a similar procedure in the case of small scale model, the large scale axisymmetric model was built. The cylindrical rock specimen was 8.0 m long with the diameter of 6.0 m, the blasthole was placed in the center of the cylinder and had a diameter of 160 mm. The rock medium was modeled using the RHT material with 10 mm quadrilateral elements using Lagrangian element configuration. The explosive material was modeled in Eulerian configuration with the element size of 5 mm. The FSI coupling mechanism was also applied to account for the energy transmit between the Eulerian and the Lagrangian media. Unlike the small scale model, the explosive column was in full contact with the rock medium and therefore there was no coupling medium needed as in the case of small scale model. The explosive material was ANFO with JWL parameters provided in Table 3-2. In order to account for the
non-ideality of ANFO reaction within the blasthole, the JWL parameters were adjusted accordingly using the Cheetah thermochemical code (Fried, 1994). Part of the ANFO was not allowed to detonate in the C-J state, so as to match the detonation velocity recorded in such a diameter and rock confinement, but was allowed to react in the expansion zone.

**Table 3-2-Parameters of JWL equation of state for ANFO**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>7143.31 GPa</td>
</tr>
<tr>
<td>B</td>
<td>2.309 GPa</td>
</tr>
<tr>
<td>R₁</td>
<td>12.527</td>
</tr>
<tr>
<td>R₂</td>
<td>0.918</td>
</tr>
<tr>
<td>ω</td>
<td>0.39</td>
</tr>
<tr>
<td>E₀ (initial energy)</td>
<td>3 kJ/cm³</td>
</tr>
<tr>
<td>Density</td>
<td>0.85 gr/cm³</td>
</tr>
<tr>
<td>Detonation Velocity</td>
<td>3930 m/s</td>
</tr>
<tr>
<td>Detonation Pressure (CJ Pressure)</td>
<td>3.5 GPa</td>
</tr>
</tbody>
</table>

Considering the height of the cylinder and the measured detonation velocity of ANFO, the model was run for 2 ms and the pressure pulse was captured at the middle of the blasthole. The contours of pressure and the resulting damage around the blasthole wall are provided in Figure 3-4. The space between the rock and the explosive column, illustrated in Figure 3-4 (c), accounts for the blasthole wall expansion. Once the detonation head interacts with the blasthole wall, the applied pressure compresses the surrounding rock material, enlarging the blasthole wall diameter. The detonation products expand and fill this space. The extent of the dummy vacuum media should be large enough to cover the resulted expansion to avoid any instability while running the model. The stress pulse captured in the middle of the blasthole wall is provided in Figure 3-5.
Figure 3-4- Contours of a) Damage, b) Radial Stress around the blasthole wall, The wall expansion is evident in figure (c) in which the explosive material fill the resulting space
3.2.2 Plane strain models

The small scale plane strain models were constructed using quadrilateral Lagrangian elements of 1x1 mm. (10x10 mm in the case of large scale models) However, in order for better discretization of the circular shape of the blasthole, the element sizes around the blasthole were about half of this size. Transmit boundary condition was applied to all boundaries except for the free surface boundary. This was achieved using the NON_REFLECTING_2D boundary condition in LS-DYNA. The pulse configuration, which was achieved as a result of the axisymmetric modeling in both small scale and large scale, was applied on the inner boundaries of the small scale and large scale blastholes, respectively, to simulate the explosive loading. This was accomplished by defining a circular SEGMENT_SET around the blasthole and prescribing a predefined load on the segment. Figure 3-6 illustrates the details of the plane strain model in LS-DYNA.
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Figure 3-6-Details of the plane strain model in LS-DYNA. Non Reflecting Boundary was applied on the nodes on three sides of the model to avoid the traveling stress wave reflection upon arrival to the boundaries of the model and contaminate the results. Impedance matching function is computed internally in LS-DYNA for all non-reflecting boundary segments. The recorded stress pulse is applied as a segment load acting in normal direction of the inner side of the blasthole wall segments

3.3 Damage analysis of small scale models

3.3.1 Burden length analysis of a single blasthole

To analyze the effect of the delayed initiation timing on burden damage, the first step was to decide on the size of the burden. For this purpose, four models with single holes were modeled with burden lengths of 80, 100, 120, and 140 mm. The models were all run for 100 µs, which was enough to let the stress wave travel the entire burden, reflect at the free surface, and travel all the way back along the burden. Therefore, the simulations were able to capture the damage evolution as a result of both compressive and the tensile stress waves. Figure 3-7 illustrates the extent of the damage zone of different burden lengths, in which, the red failed element represent the major cracks. As shown in this figure, the damage intensity in front of the blasthole (zone of influence) decreases by increasing the burden length. The damage distribution in the cases of the 80 and 100 mm burden is more favorable, since the tensile cracks,
formed after the wave reflection, are distributed well across the zone of influence, while in the cases of the 120 and 140 mm burden, the cracks are not developed in certain areas in front of the blasthole, leaving undamaged areas. The burden damage values were achieved by getting the damage values of all elements of the burden and reporting their average. The average burden damage, which was achieved by averaging the damage values of each element, was 0.29, 0.24, 0.18, and 0.15 for the cases of the 80, 100, 120, and 140 mm burden respectively. The study of the extent of the major radial cracks shows that in the cases of the burden of 120 mm and 140 mm there is a possibility of no burden detachment, since the radial crack tips were far from the tensile damage zone near the free surface. As a result, the burden length was selected to be 100 mm for delayed initiation timing simulations.

3.3.2 Burden damage analysis of multiple blastholes

3.3.2.1 Delay analysis as a function of blasthole spacing

To analyze the effect of the initiation timing on burden damage, four blastholes were modeled in plane strain configuration having a burden length of 100 mm. The spacing between neighboring holes is an important factor, since it directly alters the way stress waves interact. To evaluate this, three models were built with spacing to burden ratios of 1, 1.5 and 1.75. The models in each case were run with initiation delay timing between the holes ranging from 0 to 100 μs. To be consistent, all simulations were run 200 μs after the initiation of the last hole. This was decided since no damage evolution was observed 200 μs after detonation. The results of the damage intensity as a function of initiation timing are illustrated in Figure 3-8. As shown in the figure, the average burden damage values decrease by increasing the S/B ratio. This is expected since the larger S/B ratio leads to a lower explosives concentration in the blast. The figure also suggests that, for a given S/B ratio, the lowest amount of burden damage occurs when there is no delay time between initiations of the blastholes. In all of the cases, there is a damage enhancement region as a function of delay times after which damage remains steady. Figure 3-8 also shows that the minimum delay time for damage to become maximum is longer in the case of longer spacing distances. This is important, as it means that the optimum delay is a function
of spacing and not just the burden as it is commonly assumed. The stress wave configuration in which the minimum timing requirement is met, will be discussed in the following section.

Figure 3-7- Extent of the damage zone around a single blasthole with different burden length
3.3.2.2 Delay analysis as a function of stress pulse duration

As mentioned previously in the axisymmetric model configuration section, the use of water as a coupling medium makes the stress pulse reflect in the water-rock contact zone. This leads to multiple loading cycles of the blasthole wall until the wave attenuates. To study the effect of multiple wave reflections on the resulting damage intensity, it was decided to eliminate the pulse tail in different lengths in order to eliminate reflections. The study was conducted with three different pulse lengths as shown in Figure 3-9. Pulse I was the original configuration used in the previous section, where there are four loading cycles in this pulse having a pulse duration of 27 µs. Pulse II was created by including the first two major loading cycles leading to a pulse duration of 10 µs. Pulse III was created by considering only the first impact on the blasthole wall having a duration of 5 µs. The analysis of the damage as a function of the initiation delay timing was conducted using each of these pulses as initial loading boundary conditions on the blasthole walls in a pattern of 100 mm×100 mm. The results are provided in Figure 3-10. As illustrated, there is a very small reduction in the burden damage when the pulse duration was shortened and limited to 10 µs, showing the negligible effect of the cyclic loading after the second impact.
However, the elimination of the second impact decreases the damage intensity by about 20% showing the significant contribution of the cyclic loading on the resulting damage.

**Figure 3-9- Three pulses with multiple numbers of loading cycles as a result of wave reflection in the water-rock contact zone**

Observations from the graphs provided in Figure 3-10 show that the point in time where the curve slope changes, occurs at about 25 μs delay timing in all three configurations of the stress pulses. In other words, even though the change in the stress pulse duration alters the resulting damage level, the minimum delay timing requirement in each case to reach to a steady damage configuration is the same in all three cases. The consistency in the minimum delay time suggests that there is a consistent interaction between pulses that determines damage evolution while, as expected, the reverberations in the pulse play an important role.
3.3.2.3 Delay analysis as a function of stress pulse shape

Following the previous section, the study of the damage analysis as a function of the delay times illustrated that there is a specific minimum delay requirement to reach the maximum damage level. It seems once this minimum delay time was achieved, the overall burden damage is independent of the delay times between the blastholes. Since the pulse shape in the previous section was not altered, it was decided to run a series of simulations using different pulses having distinct loading rates and durations to further investigate the effect the stress pulse shape in the resulting damage level and the minimum delay requirements. Three stress pulses were built, having different loading rates and durations as illustrated in Figure 3-11. The shortest pulse was obtained by considering only the first loading cycle, leading to a pulse duration of 5 µs, the medium and long pulses were obtained by applying a time scale factor of 4 and 7 to the short pulse, respectively. Each pulse was used separately in a series of simulations to investigate the dependency of overall burden damage on the delay timing. The results are provided in Figure 3-12.

![Figure 3-10: Effect of pulse duration on the burden damage as a function of initiation delay timing](image)
Figure 3-11- Stress pulses with different loading rates and durations

In general, the comparison of the overall damage intensity reveals the effect of the loading rate and the pulse duration. The short stress pulse had a higher loading rate than the longer pulse, however, since all three of the stress pulse configurations reached the same maximum value of about 2 GPa, in the case of the shorter pulse the rock was placed under loading condition for a very short period of time. Since the material strength had a loading rate dependency and strain rate hardening behavior, the overall damage intensity is much lower in the case of the short pulse. This also can be addressed by comparing the total energy input of the three pulses; however, even though the higher energy magnitude imparted to the material leads to more damage, the loading rate has altered the minimum delay requirement for maximum damage state. As illustrated in the figure, the minimum delay requirement in the case of the short pulse was 25 μs, while this was achieved at about 40 and 50 μs in the cases of the medium and the long pulses respectively.
3.4 Damage analysis of large scale models

3.4.1 Burden length analysis of a single blasthole

In order to analyze the effect of the burden length in a single blast, four plane strain models were built with burden sizes of 2 m, 2.5 m, 3 m, and 4 m. Damage in each model is illustrated in Figure 3-13 where they all show the damage levels 2.0 ms after the detonation. The damage intensity in the zone of influence decreases by increasing the burden length. As shown, the radial cracks emanating from the blasthole reach the tensile damage area near the free surface in the cases of a, b, and c, while there is a visually considerable undisturbed area between the tensile cracks and the tip of the radial cracks in case d. The models with multiple blastholes were built considering the burden size of 3 m.
Figure 3-13- Results of blast induced damage in four different burden configurations: a) 2m, b) 2.5m, c) 3m, d) 4m

3.4.2 Damage analysis as a function of delay time between blastholes

In order to analyze the effect of delayed initiation on the resulting damage, 4 blastholes with the burden and spacing of 3 m was modeled. The initiation timings ranged from 0.0 ms up to 10.0 ms. The burden damage results are illustrated in Figure 3-14. The holes were fired sequentially from left to right, based on the indicated initiation delay time. In each case, for consistency reasons, the model run time was 2.0 ms after the initiation of the last blasthole, e.g. for a total of 5.0 ms in the case of 1.0 ms delay intervals and for a total of 20.0 ms in the case of 6.0 ms delay intervals.
Figure 3-14-Burden Damage levels 2.0 ms after initiation of the last blasthole. The charges are initiated sequentially from left to the right with delay intervals of 0.6, 1.0, 3.0, and 6.0 ms. The extent of the tensile cracks created as a result of the reflected wave of the first blasthole is visible up to the front of the fourth blasthole in all cases. The effect of preconditioning is evident by comparing the resulted damage in front of Hole#1 and Hole#4 in each case.

Figure 3-14 illustrates that the tensile cracks generated as a result of the detonation of the first blasthole extend up to the front of the third hole. When the second blasthole fires, the compressional stress wave mostly reflects upon arrival on these tensile cracks which were formed as a result of the firing of the first blasthole, extending them further in a direction parallel to the free surface. This effect, as shown in Figure 3-14, becomes less significant when moving to the next blastholes. This is most likely due to the
first blasthole firing in an intact environment, and each blasthole, once fired, preconditions the surrounding medium for the next hole. The preconditioning effect reaches a steady state, which in this case appears to happen even after the fourth blasthole is fired, since the cracks parallel to the free surface are still visible in front of it.

In order to be able to capture more realistic results of the effect of delay timing on the resulted damage of the large scale models, it was decided to make models with 8 blasthole in single row with the same blasting pattern, in which it was expected to observe a visually similar damage distribution in each scenario in front of the blasthole#5 to #8. The element size was 10 mm, however near the blastholes the element size was as small as 4 mm allowing for better discretization. This lead to construction of fairly large models having more than 1.6M Lagrangian elements. The model was built by assembling three parts having identical material id and element formulation. The model configuration is provided in Figure 3-15. The applied initiation delay times as well as the model run times are provided in Table 3-3.

In each case the model was terminated 2.0 ms after initiation of the last blasthole.

![Free Surface and Non Reflecting Boundary](image)

**Figure 3-15-** Configuration of large scale plane strain model in LS-DYNA. Application of the non-reflecting boundary is important in order to limit the size of the model and reduce the computational time

**Table 3-3-** Plane strain model run times based on the applied initiation delay intervals

| Delay Time (ms) | 0.0 | 0.5 | 1.0 | 1.5 | 2.0 | 2.5 | 3.0 | 3.5 | 4.0 | 5.0 | 6.0 | 7.0 | 8.0 | 9.0 | 10.0 |
|----------------|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|-----|
| Run Time (ms)  | 2.0 | 5.5 | 9.0 | 12.5| 16.0| 19.5| 23.0| 26.5| 30.0| 37.0| 44.0| 51.0| 58.0| 65.0| 72.0|
The resulted average burden damage was recorded in each model as in the case of the small scale models and is illustrated in Figure 3-16.

The graph in Figure 3-16 depicts almost the same behavior as observed in the small scale models. The simultaneous initiation results in the least burden damage, while allowing more initiation delay timing results in larger damage values, while it appears that after a delay of 2.0 ms the burden damage is almost steady. However, unlike the case of the small scale models, the average burden damage decreases after the delay time of 5.0 ms. This could be explained considering the significantly higher concentration of explosives in the small scale models. The combination of lower explosives consumption and longer distances between blastholes results in the development of large fragments from earlier detonating charges that do not allow any further damage accumulation in their masses from later detonating charges. Hence, there appears to be an optimum delay time. To investigate this, it was decided to model damage in a tighter pattern. The test pattern was reduced to 2.0 m × 2.0 m and the results are provided in Figure 3-17. As illustrated in this figure, once the minimum delay requirement reaches, the damage remains steady and independent of the applied initiation delay timing.
3.4.2.1 Wave superposition and enhancement of the tensile tail of the stress wave

The graph in Figure 3-16 demonstrated that the resulted damage at 0.5 ms delay time is slightly larger than the damage resulting at a delay of 1.0 ms. Considering the P wave velocity of the rock medium and the pattern size of 3m×3m, at the time of 0.5 ms, the compressional wave from the first blasthole is close to the neighboring hole. It was decided to examine the idea of wave superposition and the overlap of the tensile tail, which was mentioned by Johansson and Ouchterlony (2013). In their work, the ideal timing was

\[ T_{\text{ideal}} = T_d + T_1 - T_0 \]  \hspace{1cm} (3-1)

where \( T_d \) is the wave travel time along one spacing, \( T_1 \) is the duration of the first half wave at a distance equal to the spacing and \( T_0 \) is the original duration of the first half wave. Having a dynamic crack growth and interaction with the stress waves in the plastic model, the exact wave shape and duration was not...
possible to be captured; Hence, four additional delay intervals were modeled between 0.5 ms and 1.0 ms in an attempt to achieve the $T_{\text{ideal}}$.

Figure 3-18 - Average burden damage as a function of initiation delay timing (pattern size: 3.0×3.0)

a) The effect of wave superposition and enhancement of the tensile tail, the tightness of such optimum window is evident in the overall results shown in figure b

Figure 3-18 (a) illustrates the results of wave superposition and the $T_{\text{ideal}}$. It is obvious that the precise superposition of the waves provides the highest damage results at 0.7 ms, however, considering the very short delay window this wave superposition provides, given the scatter in detonation time, and the heterogeneity of the host rock, the authors believe that it is not practical to prescribe such delay timing. The overall damage dependency graph in Figure 3-18 (b) shows that there is a much longer optimum delay window from 2.5 ms to 5.0 ms. The achieved damage levels in this window are almost the same as the damage at the $T_{\text{ideal}}$.

3.5 Fragmentation analysis of large scale models

As stated previously, in order to assess blasting performance, two major indicators should be studied: The quality and the strength of the rock fragments, and the fragment size distribution. Damage analysis was carried out to fulfill the first indicator; image analysis is carried out to accomplish the second indicator. For this purpose, WipFrag, an image analysis software commonly used in fragmentation analysis, was used to achieve the fragment size distribution. It was decided to analyze the area of the
blast between the boreholes and the free face. The damage pattern of a typical case is shown in Figure 3-19.

**Figure 3-19** - Burden damage results in large scale models with pattern size of 3.0m×3.0m-stabilization of preconditioning is evident after the Hole#4. (Charges were fired sequentially from left to the right with 3.5 ms delay intervals)

In order to prepare images for image analysis software, it was decided to blank out the damaged zones in the model that would indicate development of very small fragments (fines) in the blast. However, it should be decided which damage level in the elements corresponds to the creation of a macro crack. Five elements with different damage levels were selected and analyzed and the results are illustrated in Figure 3-20. As illustrated, the element that has gone under damage level of 1.0 (Figure 3-20 c), does not withstand any further loading as the effective stress histories indicate zero stress for that specific element once it reaches the damage level of 1.0 (Figure 3-20 a). Furthermore, the study of the accumulated plastic strain (Figure 3-20 b) also depicts that in the case of the same element, further loading results in element distortion and further plastic strain, while in the cases of elements with the damage levels of <1 the accumulated plastic strain values remain constant. This is in agreement with the definition of damage in RHT material model as described previously. The elements with the damage level of 1.0 were therefore blanked out from the model representing the major cracks.
Figure 3.20-Element deletion process: a) Effective stress (v-m) values for the selected elements, the values for element #985536 at t=5.75 ms has dropped to zero where the same element at t=5.75 ms has reached the damage level of 1.0 in figure c and demonstrates sudden increase in accumulated plastic strain in figure b. The rest of the elements under study sustain different levels of damage as a result of the accumulated plastic strain. The selected elements are illustrated in figure d. figure e was achieved by deletion of the elements with damage level of 1.0, resembling a network of macrocracks in the rock material.

Since the elements were mostly square in shape with the size of 10 mm or less, the amount of particles with sizes <10 mm were considered fines in the fragmentation analysis. Table 3-4 shows the amount of
fines at the delay times used in the calculations, which were achieved by recording the model area before and after the element deletion process.

**Table 3-4** - The percentage of particles with sizes <10mm (fines) in the fragmentation analysis

<table>
<thead>
<tr>
<th>Delay (ms)</th>
<th>Original area (m²)</th>
<th>Deleted area (m²)</th>
<th>Fines (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>4.66</td>
<td>5.69</td>
<td>15.5%</td>
</tr>
<tr>
<td>0.5</td>
<td>6.64</td>
<td>14.1%</td>
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<tr>
<td>0.7</td>
<td>6.47</td>
<td>16.0%</td>
<td></td>
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<tr>
<td>1</td>
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<td></td>
</tr>
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<td>1.5</td>
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<td></td>
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<td>2</td>
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<td>18.0%</td>
<td></td>
</tr>
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<td>18.1%</td>
<td></td>
</tr>
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<td>3</td>
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<td>18.7%</td>
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<tr>
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<td>17.9%</td>
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</table>

The images were then analyzed in WipFrag and the measured fragment size percentiles were used to fit a particle size distribution function. Proposed by Ouchterlony (2005), the Swebrec function was applied to fit the measured data. Swebrec is considered the best fitting distribution function in all passing sizes (Ouchterlony, 2009; Sanchidrián et al., 2012). The basic Swebrec function with three parameters is as follows

\[ P(x) = \frac{1}{1 + \left[ \ln(x_{\text{max}}/x) / \ln(x_{\text{max}}/x_{50}) \right]^b} \]  

where \( P(x) \) is the fraction of the fragments passing a sieve size of \( x \), and \( x_{50} \) and \( x_{\text{max}} \) being the 50% and the maximum passing size, and \( b \) is the undulation exponent. The fits were carried out by minimizing the minimum least square error. Table 3-5 shows the fitting parameters for each delay time simulation. Figure 3-21 illustrates the overall fragmentation analysis steps.
Table 3-5- Parameters of Swebrec function from curve fitting

<table>
<thead>
<tr>
<th>Delay Time (ms)</th>
<th>X_{50} (mm)</th>
<th>X_{max} (mm)</th>
<th>b</th>
<th>R²</th>
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<tbody>
<tr>
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<td>98.2%</td>
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<td>1.33</td>
<td>98.6%</td>
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<td>900</td>
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</tr>
<tr>
<td>7</td>
<td>306</td>
<td>1680</td>
<td>1.60</td>
<td>98.2%</td>
</tr>
<tr>
<td>8</td>
<td>328</td>
<td>1540</td>
<td>1.61</td>
<td>98.2%</td>
</tr>
<tr>
<td>9</td>
<td>343</td>
<td>1380</td>
<td>1.34</td>
<td>99.3%</td>
</tr>
<tr>
<td>10</td>
<td>312</td>
<td>1450</td>
<td>1.43</td>
<td>98.8%</td>
</tr>
</tbody>
</table>

The results of x_{50} and x_{80} passing sizes are provided in Figure 3-22. The obtained fragmentation results showed that the simultaneous initiation of the charges leads to the creation of the coarsest fragments. As shown in the figure, the case of overlapping the tensile tail improved the fragment sizes for both x_{50} and x_{80} passing sizes. The results also show that the x_{50} values, once the optimum delay window achieved, are independent of the further delay times. However, the study of the x_{80} values shows that applying delays beyond the optimum delay window leads to formation of larger fragment sizes. This could be related to the observed damage reduction in Figure 3-18, which appears by applying excessive delay times beyond the optimum window. As illustrated in Figure 3-23, applying longer delay times forms larger fragment sizes away from the blastholes and near the free surface suggesting that any wave interaction opportunity in this zone might have been missed by applying excessive delay times.
Figure 3-21 - The steps of fragmentation analysis: a) Blast induced damaged burden, b) Deletion of the elements with damage level of 1.0 resembling the macrocracks in the model, c) Results of image processing and the fragment sizes, d) Application of the Swebrec fitting function to achieve the fragment size distribution
Figure 3-22- Resulted $x_{50}$ and $x_{80}$ fragment sizes as a function of delayed initiation, the effect of overlapping the tensile tails is evident in improving the fragment sizes for both $x_{50}$ and $x_{80}$ at 0.7 ms delay time. The optimum delay window of $2.0 \text{ ms} < T_{\text{delay}} < 6.0 \text{ ms}$.
Figure 3-23- The effect of applying longer delay times on the creation of coarse fragment sizes near free surface, from the top the applied delay times are: 2.5 ms, 5.0 ms, 6.0 ms, and 10.0 ms.
3.6 Stress wave interaction in large scale models

The study of the wave behavior and interaction in a plastic model, in which the traveling stress wave actively interacts with the existing cracking system, creating more damage is not an easy task. In most cases the different components of the wave are not even traceable which makes it difficult to interpret the cause of the resulting damage. Meanwhile, understanding the overlapping effects of the waves from neighboring blastholes is still essential for optimization of blasting practices. It was decided to reduce the complexity of the numerical models by eliminating the damage model using the elastic properties only. The objective was to investigate the phenomenon, which leads to development of the optimum window of delay intervals depicted in Figure 3-18 and Figure 3-22. The study consisted of two steps; first, only one blasthole was fired and the single wave behavior was studied, then in the next step, the second blasthole was fired at different delay times to study a variety of wave interaction scenarios. With a P-wave velocity of about 5.5 km/s, not only the initial induced compressional wave, but also its tensile reflection has already passed the neighboring blasthole way before the time in which the optimum delay window has been recorded as indicated in Figure 3-24. This brings the idea that no wave interaction has to be in effect in the study! However, using dynamic photoelasticity and holography, Rossmanith and Fourney (1983) applied high speed photography to capture the elastic wave propagation and reflection components. They illustrated that the reflection of a longitudinal wave on a free boundary creates a shear wave traveling behind the reflected longitudinal wave with a velocity equal to the S-Wave velocity. Chi et. al (2019), also investigated fracture process of granite blocks under dynamic loading of blasting and showed the creation of a reflected shear wave travelling behind the reflected P-wave. As depicted in Figure 3-24-b, the shear wave front is traveling with a velocity of about 55% of the P-wave velocity behind the reflected longitudinal wave.
Figure 3-24- Elastic wave configuration at 1.35 ms after detonation in pattern size of 3.0m×3.0m
- Contours of a) Pressure (mean stress) b) Maximum principal stress. Creation of the shear wave as a result of the P wave reflection on free boundary. The resulted shear wave head travels with the velocity of S wave behind the reflected P wave.

Hence, it was decided to study the overlapping of the reflected shear wave with the stress wave emanating from the second blasthole. Multiple trials of the overlapping procedure revealed that the x
stress component (stress in the direction of spacing) of the reflected shear wave cancels out within the tensile tail region of the second blasthole during the overlapping. As illustrated in Figure 3-25, the superposition of the reflected shear wave of the first blasthole with the incident wave of the second blasthole leads to weakening of both pulses. The study showed that the shorter delay times lead to longer and greater overlapping, which eventually cause greater weakening of the pulses.

![Figure 3-25](image_url)

**Figure 3-25- Superposition of the reflected shear wave of the Hole#1 and the incident wave of the Hole#2.**
Since the reflected shear wave and the stress wave from the second blasthole are traveling in opposite directions within the burden area, it appears that in order to avoid such destructive overlapping, the reflected shear wave should pass the second blasthole before it fires. The timing study reveals that the passing time of the reflected shear wave from the second blasthole is synchronized with the start of the optimum delay time window. In order to develop a simple formula to calculate the time of arrival of the reflected shear wave to the second blasthole and determine the minimum delay time for fragmentation optimization, if the front of the reflected shear wave is assumed to be a straight line, from Figure 3-26

![Figure 3-26](image)

**Figure 3-26** - Schematic of the wave configuration where the reflected shear wave from Hole#1 arrives at Hole#2

\[
c d = v_x(t - t_0)
\]

\[
c e = C_s(t - t_0)
\]

\[
a a_{1}' = C_p \times t_{a_{1}'}
\]

\[
\sqrt{B^2 + S^2} = C_p \times t_{a_{1}'}
\]

\[
t_{a_{1}'} = \frac{\sqrt{1 + m^2} \times B}{C_p}
\]

\[
c a_{1}' = v_x(t_{a_{1}'} - t_0)
\]
\[ v_x = \frac{m C_p}{\sqrt{1+m^2}-1} \]  

where \( v_x \) is the compressional wave front velocity at the free surface in the direction of spacing between the holes, \( t_0 \) is the arrival time of the compressional wave at the free surface, \( C_s \) and \( C_p \) are the S and P wave velocities respectively, and \( m \) is the S/B ratio. In such configuration, \( t \) is the arrival time of the reflected shear wave at Hole#2. From triangles in Figure 3-26

\[
\frac{cd}{S} = \frac{ce}{ce-B} \tag{3-10}
\]

\[
\frac{v_x(t-t_0)}{S} = \frac{C_s(t-t_0)}{C_s(t-t_0)-B} \tag{3-11}
\]

\[
t = \frac{B}{C_p} (\sqrt{1+m^2} + \frac{C_p}{C_s}) \tag{3-12}
\]

In order to achieve the minimum required delay timing, the reflected shear wave should pass before initiation of the second hole, therefore

\[
T_{min} = \frac{B}{C_p} (\sqrt{1+m^2} + \frac{C_p}{C_s}) + \tau_s \tag{3-13}
\]

where \( T_{min} \) is the minimum delay requirement to achieve the optimum delay timing window and \( \tau_s \) is the duration of the reflected shear wave. Figure 3-27 shows the position of the reflected shear wave 2.0 ms after the initiation of the first blasthole.
3.7 Discussion

The numerical modelling study of the burden damage intensity as a function of the initiation delay timing of multiple blastholes reveals that there is a certain minimum delay requirement in order to reach the maximum damage level. The elastic wave study of the large scale models suggested that the location of the reflected shear wave of the first blasthole, upon detonation of the second blasthole, plays a significant role in the wave history in the burden, possibly influencing damage and fragmentation. The study indicated that there is a lower bound for the selection of short delay times between blastholes in a row. Selection of the delay times shorter than the minimum delay requirement would result in lower damage levels in the burden, although higher damage levels were observed when the tensile tails of the neighboring P waves are overlapped. However, such a practice requires extensive precision to reach the optimum. While the lower bound for the delay requirement was estimated by the study of the elastic stress wave interaction, the selection of a delay timing range for the optimum delay between blastholes also requires an upper bound to be introduced. The overall burden damage and \( x_{80} \) fragment size distributions in the case of large scale models reveal the existence of such an upper bound, beyond which burden suffers less damage and fragmentation yields larger \( x_{80} \) sizes.

Figure 3-27- Position of the reflected shear wave 2.0 ms after the detonation of the first blasthole
The results of small scale models are in agreement with the experimental results obtained by Katsabanis et. al. (2014). In their fragmentation analysis, they concluded that very short delays result in coarse fragmentation suggesting lower burden damage values as described in this study. They also showed that there exists a wide range of delay times in which the fragmentation is fairly constant. This statement is also in agreement with the numerical results achieved in this study where damage values remain constant once the minimum delay requirement is achieved. The numerical results provided significant insight to the observed experimental results, explaining the controlling mechanisms, which leads to an optimum delay timing. These findings could not be explained by the experiments given the lack of detail and the scatter in the experimental results.

The present study deals with the effects of stress waves only. The medium is treated as a continuum and gas penetration in the rock mass is ignored. Gas penetration into the major cracks is the ultimate driving force for burden detachment, creating discontinuities in the mass of the blast that inhibit stress wave interaction (Brinkmann, 1990). The development of the discontinuities was captured here, however the displacement and opening of them was not captured and may have some additional effect on fragmentation. The 2D study of the blast induced damage is also a simplification made in this study. A more sophisticated 3D model which is capable of capturing the effect of stress waves and gases in a discrete medium representing the inherent discontinuities of any given rock mass, might be beneficial for better observation of stress waves interaction within the medium.

3.8 Conclusion

The effect of the delayed initiation timing on the resulting burden damage was analyzed numerically on small scale and large scale model configurations using LS-DYNA. A cylindrical model having a central blasthole was modeled in each case in a 2D axisymmetric setting and the response of the blasthole wall in the form of a compressional stress pulse was captured. The resulted stress pulse was produced by detonating water-coupled PETN (24 g/m) detonating cord in the case of the small scale models and ANFO in the case of the large scale models and was applied to the inner boundaries of the blastholes.
modeled in a 2D plane strain setting. The rock material in all cases was modeled using the RHT plasticity model, which was calibrated for the strain rate dependency of granite.

It was observed that the burden damage increases as a result of the delay time in the initiation of the neighboring blastholes. In the case of the small scale models, this burden damage increase stops at a specific time beyond which the additional delay timing has almost no effect on burden damage. In the case of the large scale models, once the damage levels reach to a maximum, there is an optimum window beyond which the damage levels decrease.

Considering the effective stress, the accumulated plastic strain and the damage values of all the elements in the numerical model, an element deletion criterion was carried out to create discontinuities in the finite element model, representing the blast induced macroracks. In the case of the large scale models with pattern size of 3 m×3 m the burden images having macroracks were plotted and used for image analysis which was carried out using the WipFrag image analysis software in order to obtain the fragment size. The Swebrec function was used to fit the measured percentiles and obtain the fragment size distributions. The results of $x_{50}$ and $x_{80}$ fragment sizes illustrated a delay timing dependency behavior similar to the case of damage analysis. While the $x_{50}$ values remain almost constant after a minimum delay requirement, the $x_{80}$ values increase with the delay times greater than 5.0 ms. The increase in the $x_{80}$ sizes are considered the reason for damage decrease at the same delay times.

The effect of wave superposition for the purpose of the enhancing the tensile tails of the P waves of two neighboring blastholes showed that overlapping of the tensile tails improves the fragmentation. However, considering the very short timing intervals, such a practice in mining operations is not suggested. The effect of the pulse duration on the resulting burden damage and eventually on the minimum delay requirement was investigated. In an elastic model, the development and the travel time of the reflected S wave and its effect on the burden damage was studied. It was concluded that in order to reach the optimum fragmentation window, the reflected shear wave from an earlier detonating hole has to completely pass the later detonating blasthole before it is fired, to avoid the destructive
overlapping of the shear wave of the first blasthole with the incident wave of the second blasthole. Also, it was suggested that the delay time between boreholes should allow full development of damage zone around a previously detonating blasthole.
Chapter 4
Tunnel face preconditioning using destress blasting in deep underground excavations

4.1 Introduction
Underground mining and deep tunneling activities have always been subjected to numerous risks, which, if not handled on a timely manner, would compromise the safety of the personnel and impose economic threats due to production disruption and consequent reconditioning expenses (Donoghue, 2004; Bahn, 2013; Cao et al., 2015). These risks include methane gas explosion, unforeseen water flush into the underground spaces, fire, ground fall, and rockbursts (Dou et al., 2012). Rockbursts, characterized as an abrupt failure of rock (Simser, 2019), occur due to seismic activities associated with major discontinuities, such as faults or dykes, or because of stress concentration in pillars or in the tunnel faces and walls (Blake and Hedley, 2003; Andrieux et al., 2013). Hedley (1992) describes rockburst as an instantaneous and violent failure of rock in which the rock fragments are ejected into the underground openings. Kaiser and Cai (2012) classified the rockburst into three types: strainburst, pillar burst, and fault-slip burst. In deep underground excavations, rockbursts are associated with sudden release of the strain energy stored in brittle rock (Diederichs, 2007). Rockbursts are either “mining-induced” in which there is no significant increase in dynamic stresses due to a remote seismic event, or “dynamically-induced” in which an energy transfer of a remote seismic event causes a dynamic stress increase, leading to damage in the rock (Kaiser and Cai, 2012). The mechanism of rockburst phenomenon is explained by means of the stored post peak strain energy in the brittle material (Brummer, 1998). The excessive accumulated strain energy has a potential to transform into a form of kinetic energy leading to acceleration and abrupt movement of rock fragments. This kind of potential energy is greater for homogeneous materials i.e. competent rock material with lesser amounts of natural flaws and micro fractures with fewer mineralogical varieties (Brummer, 1998).
Appropriate opening space dimensions, proper mining sequences and rates, and careful filling practices are considered “strategic” methods in alleviating the occurrence of rockbursts (O’Donnell, 1999). However, even with precise and prudent mining activities, the burst potential could still be considered a safety hazard. Therefore, methods to mitigate the risks posed by rockbursts include the installation of enhanced supports systems, and the use of destressing and preconditioning as “tactical” methods to control damage (Morrison and Macdonald, 1990). In the literature, several research papers discuss the different aspects of designing a support system in underground burst prone openings, emphasizing on the complexity of ground support procedures for seismically active underground excavations (Kaiser and Cai, 2012; Malan and Napier, 2018; Li et al., 2019; Morissette and Hadjigeorgiou, 2019). Unlike conventional support systems, where the main objective is the controlling and maintaining the loose rocks from falling into the openings due to gravity, the support systems in burst prone underground spaces should withstand dynamic loading and buckling of the rock mass in response to violent and sudden rock failures (Kaiser and Cai, 2012). Tactical methods to control rockbursts by preconditioning the rockmass, include hydraulic fracturing and distress blasting to reduce excessive stresses and mitigate the burst potential (Jun et al., 2012; Zhu et al., 2017; Kang et al., 2018).

The use of explosives for the purpose of destressing was first applied in the South African gold mines in 1950’s (Roux, Leeman and Denkhaus, 1957; Hill and Plewman, 1958). Their destressing practices was conducted for the purpose of preconditioning of the mining face at deep levels at East Rand Properties Mines (ERPM) in which 3m long blastholes were drilled into the stope face where the normal production holes were 1m long. Since there was a considerable reduction in face bursting and subsequent accidents, statistical reports considered this method as a successful way of mitigating the effect of stress concentration and the resulted excessive strain energy stored in the rock mass near underground openings (Roux, Leeman and Denkhaus, 1957). In order to alleviate the burst prone rock masses, it is important to introduce more inhomogeneity leading to development of microfractures and thus lowering the stiffness and dissipating the excessive stored energy (O’Donnell, 1999). Therefore the objective of
destress blasting is the promotion of new microfractures and fracture surfaces and the increase of shear deformation of existing fracture surfaces. These alterations in the behavior of the rock mass would eventually change the brittle deformation mode of the system (Blake, Board and Brummer, 1998). Tang (2000) developed a new geomechanical model in an attempt to assess the effectiveness of destress blasting practices numerically. He studied the effect of in situ stresses and their orientation on the resulted damage of various destress blasting patterns.

O’Donnell (1999) has studied the application of destress blasting techniques in deep underground mines and discussed the different practices that has been carried out in order to conduct preconditioning and destressing of rock mass. Figure 4-1 illustrates the destress blasting hole pattern which was carried out in Creighton mine in Sudbury, Ontario. The pattern consisted of two horizontally drilled face holes each 7.3m long, loaded with 1.5 meters of ANFO filled at the bottom of the hole. The four corner holes were drilled up and out, and down and out at 30° with respect to the direction of the drift advancement. The corner and wall holes were 3.6 m long and the bottom 0.6 m of each hole was filled with ANFO. As shown in the figure, the pattern contains four additional wall holes, drilled at 45° to the direction of the face advancement and with a few degrees above horizontal, to drain water.

![Pattern of the conventional destress blasting rounds at the depth of 2200 in Creighton mine, Sudbury, Canada (O’Donnell, 1999)](image-url)
Despite the considerable consensus to the qualitative explanations towards the effectiveness of destressing by blasting, there is a general lack of detailed explanation of the actual mechanism. Figure 4-2 illustrates the conceptual effect of the additional fractured zone, caused by blasting, ahead of the drift development face, which not only results in a reduction of the peak stresses, but also pushes the highly stressed zone further away from the tunnel face. Roux et. al (1957) argued that by such alteration in the stress states, the occurrence of strainburst is alleviated. In the literature however, there is scarce evidence, that the current destress blasting patterns would lead to formation of a homogenously fractured zone, as expected and illustrated in Figure 4-2. Considering the large spacing between the blastholes of the conventional patterns, together with the effect of direction of the principal stresses on the extent and orientation of radial cracks, these patterns may only lead to formation of isolated blast induced fractured zones, or “damage packets”. With no interaction between the damage zones of neighboring blastholes, it is doubtful that current destressing practices transfer the highly stressed area further away ahead of the tunnel face at great depth.

Figure 4-2- Conceptual effect of destress blasting mechanism on transferring the highly stressed zone ahead of a tunnel development face by introduction of an extended fractured zone (Roux et. al 1957).
Recently, Drover et. al (2018) have investigated the face destressing blast design for deep hard rock tunneling. They studied the effect of in situ stresses on the blast induced fractures by the aid of Hybrid stress blasting numerical models. As a result, they have presented an alternative design concept for destress blasting in deep underground hard rock mining practices, which takes into account the effect of in situ stresses on the directionality and length of introduced fractures.

This chapter is structured as follows. First, the effect of in situ stresses in blast induced rock fracturing is analyzed by the aid of continuum damage modeling using LS-DYNA. For this purpose, numerical models were constructed in which stress initialization was applied to account for the existing in situ state of stresses. The orientation and the extent of fractures in response to blast loading of single holes were then analyzed. Models with different states of in situ stresses were constructed to obtain a detailed explanation of the effect of stresses on blast induced damage. Then, the existing conventional destressing patterns were investigated in an effort to analyze their resulted damage zones where in situ stresses were applied as boundary conditions. Eventually, a new conceptual destress blasting pattern was introduced in order to further investigate the capability of the introduced damage zones to transfer the highly stressed zone ahead of the tunnel face further away.

4.2 Effect of in situ stresses on blast induced damage zone

In any blasting practice, the resulted rock damage, in the form of cracks and microcracks, is a function of rock mass quality, and explosive type and strength; in underground blasting projects however, the desired damage is also a function of the magnitude and the orientation of the existing in situ stresses (Fleetwood, 2011). The in situ stress field forms an anisotropic stress concentration around the blasthole as illustrated in Figure 4-3. In the presence of such stress confinement, the majority of the blast induced radial cracks tend to propagate in the direction of least confinement, which, as shown in Figure 4-3, is at the direction of maximum principal stress. This phenomenon was first addressed by Obert and Duvall (1967). The alignment of the blast induced radial cracks to the direction of maximum principal stresses was also validated experimentally by Jung et. al (2000). In order to analyze the effect of in situ stresses
on the shape and the extent of the blast-induced damage, a series of numerical models were built with different in situ stresses, where the direction of the maximum principal stress was horizontal with a horizontal to vertical stress ratio (K ratio) of 1.5.

Figure 4-3- Contours of major principal stresses around a blasthole with diameter of 50 mm. The far field major principal stress magnitude of 60 MPa in horizontal direction with k=1.5.

The models were built in 3D configuration in LS-DYNA. RHT material was used to model the rock medium which its properties are provided in Table 2-2 and Table 2-3. The explosive used was ANFO which its JWL parameter and physical properties are provided in Table 3-2. A 2 m long blasthole with a diameter of 50 mm was modeled inside a block of 6m×6m×4m (Figure 4-4-left). The element sizes varied from 5 mm in the vicinity of the blasthole to 150 mm at the boundaries. Stress initialization was applied as boundary condition using Load_Segment code in LS-DYNA, in which a load curve was applied on the selected boundaries. The boundary condition on the segments of the opposite sides were single point roller boundaries as illustrated in the Figure 4-4-Right. All sides of the rock block were assigned non-reflecting boundary conditions, to avoid any damage created because of wave reflection.
on the boundaries. Since LS-DYNA is a transient code, such stress initialization results in a compressional stress wave traveling inside the medium. It is crucial to allow enough time for stresses to stabilize inside the rock medium in order to account for the applied in situ stresses (Figure 4-5).

**Figure 4-4-Left) 3D model configuration in LS-DYNA, a 2m long blasthole with diameter of 50 mm filled with ANFO inside a rock block of 6m×6m×4m. Right) Applied boundary condition: Load Segment on Up and Right boundaries, Rollers on Down and Bottom boundaries.**

Considering the size of the model and the elastic wave velocity of the material, the stresses were stabilized at 10 ms after the initializations of the boundary loadings. Hence, the explosives were detonated at the time of 10 ms. The resulted damage zones for different stress states are provided in Figure 4-6. As illustrated, the radial cracks extend up to 1.0 m in any direction around the blasthole where there is no boundary condition applied as initial stresses, resulting a 2 m wide damage zone (Figure 4-6-a). In the presence of initial stresses, the evolution of the damage zone is anisotropic and in the direction of maximum principal stresses. Comparison of the length of the damage zones obtained by applying different magnitudes of stress initializations reveals that there is a direct relationship between the magnitude of the in situ stresses and the extent of the blast induced damage zones (Figure 4-6-b, c).
Figure 4-5-Stabilization of the stresses; a-1 to a-4: contours of maximum principal stresses at 1, 2, 3, and 6 ms after stress initialization; b-1 to b-4: contours of minimum principal stresses at 1, 2, 3, and 6 ms after stress initialization; c: time dependency of the stabilization of the in situ stresses in LS-DYNA
Figure 4-6- Dependency of the blast induced damage zone on the presence of in situ stresses; Up: No stress initialization applied, isotropic damage evolution. Middle: Maximum principal stress of 20 MPa in horizontal direction with $k=1.5$ applied prior to detonation, major damage evolution in the direction of Maximum principal stress. Down: Maximum principal stress of 60 MPa in horizontal direction with $k=1.5$ applied prior to detonation.
The calculated extent of the damage zones under the stress state assumed is shown in Table 4-1. Figure 4-7 illustrates the dependency of the extent of blast induced damage zone to the in situ stresses. In this figure, the normalized damage zone (ratio of extent of damage zone to the borehole diameter) is reported as a function of stress concentration factor in the direction of maximum principal stresses \((3\sigma_3-\sigma_1)\) normalized by the uniaxial strength of the rock material. Note that under stress concentration values greater than 0.4, the damage extent is a constant value representing the radius of the blast induced crushed annulus around the blasthole wall. Considering the tremendous amount of pressure applied on the blasthole wall upon the detonation of the explosive, creation of such crushed zone is inevitable despite the magnitude of the applied initial in situ stresses. Therefore, the damage extent around a blasthole subjected to in situ stresses can be obtained using

\[
 r = \begin{cases} 
 -83.7 \left( \frac{\sigma_{\text{max}}}{\sigma_c} \right) + 40.7 \times R & \text{if} \quad \frac{\sigma_{\text{max}}}{\sigma_c} \leq 0.4 \\
 4.5 \times R & \text{if} \quad \frac{\sigma_{\text{max}}}{\sigma_c} > 0.4 
\end{cases}
\]

where \(r\) is the radius of damage extent in the direction of maximum principal stresses, \(R\) is the blasthole radius, \(\sigma_c\) is the uniaxial compressive strength of the rock material, and \(\sigma_{\text{max}}\) is the stress intensity factor on blasthole wall in the direction of maximum principal stress. The critical value of 0.4 for \(\sigma_{\text{max}}/\sigma_c\) also suggests that there is a limitation on the application of ANFO considering the energy input of the explosive versus the in situ stress magnitudes. In other words, in order to obtain a desired damage extent under higher in situ stress magnitudes, a stronger explosive type is needed.

| Model # | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 |
|---------|---|---|---|---|---|---|---|---|---|----|----|----|----|----|----|
| \(\sigma_1\) (MPa) | 0 | 3 | 9 | 15 | 21 | 30 | 42 | 50 | 60 | 69 | 75 | 90 | 111 | 160 | 180 | 210 |
| \(\sigma_3\) (MPa) | 0 | 2 | 6 | 10 | 14 | 20 | 28 | 33 | 40 | 46 | 50 | 60 | 74 | 107 | 120 | 140 |
| Damage Extent (m) | 2.0 | 1.9 | 1.75 | 1.55 | 1.45 | 1.15 | 1.0 | 0.85 | 0.65 | 0.6 | 0.22 | 0.22 | 0.23 | 0.23 | 0.22 | 0.20 |
Figure 4-7- Normalized blast induced damage extent as a function of stress intensity factor on the blasthole wall in the direction of the maximum principal stresses.

4.3 Damage zones and the stress study of the current destress blasting pattern

A numerical model containing two face holes with a spacing of 2.0 m and four corner holes at 30° to the direction of the face advancement (Figure 4-8) was built with hole diameter of 50mm loaded with ANFO to analyze the effectiveness of the conventional destressing patterns. The study took place in two different steps. First, the blast induced damage zones were studied in a model with no in situ stresses. At the second step, the model was analyzed under an in situ stress condition, in which the extent of the
blast induced damage zones around the holes were studied, and the stress state ahead of the tunnel face before and after destress blasting was investigated.

![Diagram of destress blasting pattern](image)

**Figure 4-8** Design of a typical destress blasting pattern in a 5.0m wide tunnel where two 5.0m long blastholes were drilled into the tunnel face with spacing of 2.0m, corner holes were drilled at 30° to the direction of the tunnel face advancement, the 2.0m end of the blastholes were filled with ANFO. All blastholes initiated simultaneously at the bottom.

### 4.3.1 Destressing damage zones with no in situ stresses

The numerical model was built in LS-DYNA in a 3D configuration, where the element sizes ranged from 5 mm near the blastholes to 400 mm near the boundaries. The model was 20m×20m×7m in which the explosive charges were located 3.0m away from the development face inside the rock material (Figure 4-9). To eliminate the effect of any stress wave reflections on the boundaries of the model, the entire model boundaries were attributed as non-reflecting boundaries.

The resulted damage zones are illustrated in Figure 4-10. As discussed in section 4.2, the blast induced radial cracks in the case of zero in situ stresses are propagated almost equally in any direction around the blasthole and the damage zone is extended about 1.0 m around the holes. The Damage results shown in Figure 4-10, suggest that a 2.0 m long spacing between the blastholes drilled in the face is a reasonable value since the cracks from neighboring holes appear to coalesce and create a unified damage zone that covers the zone or rock between the holes, ahead of the tunnel face. Despite the acceptable spacing of
the face holes, the damage zones of the corner-drilled holes are very far from the area of interest ahead of and around the tunnel face. Such damage packets created in the rock mass might impose excessive stress concentration, in the case of high in-situ stresses and increase the probability of a rockburst.

Figure 4-9- Model configuration in LS-DYNA representing the conventional destressing pattern with two blastholes drilled into the middle of tunnel face and four at the corners.

Figure 4-10-Result of blast induced damage zones in a conventional destressing pattern in a tunnel with the width of 5.0 m.
4.3.2 Destressing damage zones in the presence of in situ stresses

To analyze the effect of the current destressing pattern on the redistribution of the stress states ahead of the tunnel face, a model was built based on the geometry of the models in section 4.3.1. The stress initialization procedure was carried out by applying load segments on either sides of the model as described in Figure 4-4. Considering the large size of the model, stress stabilization required more time in comparison to Figure 4-5-c. Figure 4-11 shows the timing requirement for the stress stabilization. As illustrated in this figure, it takes about 40 ms for the stresses to reach and stabilize at the desired values of 60 and 40 MPa for the maximum and minimum principal stresses respectively. In the figure, $\sigma_1$ and $\sigma_3$ are in the X and Y directions respectively. These stress values were selected to represent the stress state within the rock material at the depth of 1500 m with a k ratio of 1.5.

![Stress Stabilization Process](image)

**Figure 4-11-** Stress stabilization process in destress blasting models in LS-DYNA, once the stresses are fixed at the desired values, the explosives were initiated at 40 ms where the resulted blast induced stress pulse is captured on the stress profiles.

The total model run time was set to 80 ms, in order to allow enough time for the stress redistributions to stabilize after blast loading. The blast induced damage results are provided in Figure 4-12. As expected, in the presence of the in situ stresses the damage zones are mostly aligned in the direction of the maximum principal stresses. Regarding the two face-drilled holes, there seems to be no interaction
between the damage zones. Furthermore, the corner-drilled holes resulted in damage packets totally disconnected and isolated from their neighboring holes.

**Figure 4-12** - Result of blast induced damage zones in a conventional destressing pattern in a tunnel with the width of 5.0m. Maximum principal stress state of 60 MPa in horizontal direction with k ratio of 1.5 representing a tunnel depth of 1500m.

### 4.3.3 Post blast stress states ahead of the tunnel face

Even though the ultimate objective of destress blasting in the literature is considered to push the highly stressed zone further away, ahead of the excavation advancement face (Toper et al., 1997)(Figure 4-2), there are few efforts that illustrate, either numerically or experimentally, the stress states ahead of the tunnel development face after destress blasting. Post blast stress results obtained from the 3D numerical models in LS-DYNA, described in the present work, demonstrate that the current destressing pattern may not act as efficient as expected, to fulfill the objective of mitigating the burst potential ahead of the tunnel face. The stress contours of Figure 4-13, captured at 40 ms after detonation, illustrate that localized destressing zones are created around each blasthole, which are not successful in transferring the high stresses ahead of the tunnel face further away. Despite a very limited area in the center of the tunnel face, the stress values remain unchanged in the rest of the area of interest (Figure 4-14).
Figure 4-13-Post blast contours of principal stresses captured at 40 ms after explosive detonation. a) Illustration of the cross-sections in which the stress states are captured. b) Contours of maximum principal stresses at predefined cross-sections XY, YZ, and XZ. c) Contours of minimum principal stresses at predefined cross-sections XY, YZ, and XZ.
Figure 4-14- unsuccessful destressing of the tunnel face; left) selected elements ahead of tunnel face for post blast stress analysis, right) stress values of the selected elements, notice the stress state before and after the destressing took place: Blast induced stress pulse is captured at 40 ms with a duration of about 7 ms.

4.4 Numerical analysis of a conceptual destress blasting pattern

The results obtained from section 4.3 illustrated the limitation of the conventional destressing practices on efficient mitigation of strain burst potential in the tunnel development face. Drover et. al (2018) pointed out the limited fracturing and lack of interaction between the micro-fractured damage zones of these widely spaced destress blasting patterns and questioned the capability of such destressing designs on transferring the high stress states further away ahead of tunnel faces in burst prone underground environments. Inspired by the work of Drover et al (2018), a conceptual destress blast pattern was designed in order to numerically investigate the feasibility of redistribution of the stress states ahead of a tunnel face by introducing a network of blast-induced fractures and damage zones.

4.4.1 Model Configuration

In order to better interpret the stresses around the tunnel face, a 3D numerical model of a circular tunnel with a diameter of 5.0 m was built in LS-DYNA. The tunnel excavation advancement was assumed to be 3.0 m on each round using explosives. 6.0 m long destressing holes were drilled from the tunnel face...
and their bottom 2.0 m were filled with ANFO (Figure 4-15 a). Far field stress states were applied on the model boundary as described in Figure 4-4 and Figure 4-5. The entire model boundaries were non-reflecting to avoid any undesired wave reflections. The model had dimensions of 20 m × 20 m × 20 m and the maximum and minimum principal stresses in the X and Y directions had magnitudes of 60 MPa, and 40 MPa respectively.

![Figure 4-15- Configuration of a novel tunnel face destressing model in a circular tunnel in LS-DYNA. a) Direction of advancement of the tunnel face with a 3.0 meter long excavation round and 2.0 meter long destressing charges located ahead of the excavation round for the purpose of preconditioning the tunnel face for the next round. b) tunnel cross-section illustrating the location of the destressing charges at the face.](image)

The destressing pattern consisted of three rows of horizontally drilled blastholes with a diameter of 50 mm (Figure 4-15 b). The extent of the anticipated damage zone from a single blasthole under the mentioned stress states was calculated using Equation 4-1. The spacing between the charges was chosen to be 0.8 m based on the procedure explained on Section 4.2 and the results shown in Figure 4-7 in order to assure coalescence of the radial cracks in neighboring holes. Five blastholes were modeled on the horizontal centerline of the tunnel face in which the distance from the last blasthole to the tunnel wall was less than a meter. Two additional rows of blastholes were placed at a distances of 1.6 m from the center line at the top and bottom sections of the tunnel. Each row had three blastholes. As a result, a total
of eleven blastholes were used in each destressing round. Since the previous analysis illustrated a
negligible change of the stress states of the elements away from the blast induced damage zones before
and after destress blasting (Figure 4-14), the new pattern was designed to introduce a distributed damage
zone in the stress concentration area ahead of the tunnel face.

4.4.2 Blast induced damage zones and the stress analysis ahead of the tunnel face

The numerical model was run for 80 ms in LS-DYNA. The first 40 ms allowed the medium to reach a
steady stress state due to the introduction of in situ stress. At 40 ms the destressing charges were
detonated and the calculations considered 5 ms of additional time, to allow for damage development due
to the propagation of the stress waves induced by the blast. Then, 3.0 meters of the tunnel, corresponding
to the advance of the production blast, were excavated numerically, deleting the elements of the model
corresponding to the current excavation round. This was accomplished using the Element_Death_Solid
keyword in the LS-DYNA code. Deleting these elements leads to a new tunnel face for which the
distressing blast was carried out. The destress blast and tunnel advancement procedure are illustrated in
detail in Figure 4-16.

Once the tunnel excavation took place, the model ran for 35 ms to let the stress redistributions occur so
that the model reaches a post-blast steady stress state. Then, the maximum and minimum principal
stresses ahead of the distressed tunnel face were compared against the case in which no distress blasting
was used. Figure 4-17 and Figure 4-18 illustrate the maximum and minimum principal stress states
before and after destressing ahead of the tunnel face.
Figure 4-16- Modeling of destress blasting procedure in LS-DYNA; a) Charge detonation of the 2m long destressing holes at 40 ms when the in situ stresses were stabilized within the model, b) destressing damage zones 2ms after detonation c) 5ms after detonation when the damage evolution was completed within the destressing zone. Excavation of 3.0m of the tunnel takes place by removing the elements; new tunnel face in which the destressing has been executed is ready for the next round.
Figure 4-17-Stress contours ahead of the tunnel face; Left I-V) Contours of maximum principal stresses at 0.0, 0.5, 1.0, 2.0, and 4.0m ahead of a tunnel face before (I-Va) and after (I-Vb) destressing. Right I-V) Contours of minimum principal stresses at 0.0, 0.5, 1.0, 2.0, and 4.0m ahead of a tunnel face before (I-Va) and after (I-Vb) destressing.
Stress plots in different cross-sections clearly demonstrate the stress states around a circular underground opening before destressing. These pre-destressing plots show stress magnitudes and distributions ahead of the tunnel face, illustrating the peak of the maximum and minimum principal stresses at distances 0.5 m to 1.0 m ahead of the tunnel face (Figure 4-17, II-a). Advantages of the new destress blasting pattern design are evident in these plots. The controlled rock disturbance ahead of the tunnel face, achieved as a result of blast-induced damage, is capable of redistributing the stress states. The profiles of maximum and minimum principal stresses are demonstrated in Figure 4-19. As illustrated, the overall stresses in
the zone of destressing have been decreased by about 30% and 20% for the cases of maximum and minimum principal stresses respectively. The results also show that the peak stresses are transformed from near the tunnel face to 3.0 m ahead of the tunnel face. This can be seen in Figure 4-17-V, where the magnitude of principal stresses after destressing is greater than the case of pre-destressing at the distance of 4.0m ahead of tunnel face.

Figure 4-19- Profiles of maximum and minimum principal stresses ahead of the tunnel development face before and after destressing
4.5 Conclusion

Destress blasting has been in practice for a long time in deep underground hard rock mining operations, where there is a rock burst potential. The purpose of distressing blasts in tunnel development is to create a controlled fractured zone ahead of the face, where there is a high stress concentration, as a result of in-situ stresses in the rock mass. Conventionally, this kind of distressing activity includes two horizontally drilled long face holes and four inclined blastholes at each corner of the development drifts and tunnels. In this study, the shape and the extent of the blast induced damage zones in the presence of a wide range of in situ stresses were investigated numerically by the aid of LS-DYNA code. The 3D damage results of single blastholes illustrated that in the presence of in situ stresses the radial cracks tend to mostly propagate in the direction of maximum principal stresses. The results also revealed the linear dependency of the blast-induced damage extent to the stress concentration factor on the wall of the blasthole in the direction of the maximum principal stress. The results of the calculations suggested that these patterns, when applied in deep underground operations, have very limited capability on redistributing and transferring the stress states, ahead of tunnel development faces. Moreover, the wide spacing between the blastholes of these distressing patterns may result in the creation of isolated damage packets, which may worsen the burst potential by over-stressing the rock between these isolated damage zones.

Considering the expected shape and the extent of the blast induced damage zone under a specified in situ stress condition, a conceptual distressing pattern was designed and its effects on damage and mitigation of high stresses were established numerically. The calculated damage ahead of the tunnel face formed controlled fracture planes, which reduced the magnitudes of maximum and minimum principal stresses ahead of the tunnel face, and pushed the peak stresses further away from the development face. It is worth noting that the results obtained from this work are limited to the properties of the explosives and the rock materials applied. The model treats the rock medium as a continuum without considering the effect of existing weakness planes and fractures and damage is the result of stress waves. In a deep underground mine, this is a reasonable assumption supported by experiments in the laboratory (Ledoux,
2015). As far as the explosive is concerned, explosives other than ANFO will generate higher pressures and will most likely result in somewhat larger damage zones. The effectiveness of the proposed design is not expected to change significantly by the choice of explosive in the destress blast.
Chapter 5

General Conclusion

5.1 Thesis summary and conclusions

Chapter 2 The dynamic fracturing process was studied under blast induced loading condition and the obtained tensile strength was compared against the results of the quasi-static Brazilian disc test. The dynamic response curve was achieved by the aid of two experimental methods: Hopkinson bar experiment and the Split Hopkinson Pressure Bar Experiment (SHPB). The Hopkinson bar experiment was conducted using rod shape samples of Laurentian granite with diameter of 26 mm. Two different sets of samples were prepared, the first set of samples were about 300 mm long and were used to capture the tensile cracks. Different combinations of explosives along with PMMA wave attenuators were used to achieve a single tensile crack in the sample. Strain gauges were used on the second set of samples in order to capture the blast induced stress wave properties traveling along the sample. In order to successfully separate the compressional induced stress wave from the reflected tensile wave, the gauged samples’ length was longer than the original ungauged samples. With this technique, the reflected tensile wave was attenuated enough to avoid tensile cracks along the gauged samples. By having the location of the tensile cracks from the first set of samples and knowing the stress wave configuration at the location of the tension crack, the dynamic tensile strength was obtained. The technique requires clean pulses without noise produced by the electromagnetic interference of the detonating charge or by unwanted reflections of the pulses during the experiments. One of the challenges in obtaining the results was to create a clean pulse with a steady strain rate. This was crucial in order to conduct a precise wave superposition and timing study. Immediately after detonation, stress wave reflections from the sides and the back of the sample (in the vicinity of the detonation point) result in a complicated pulse that travels the length of the pulse. To allow the formation of a simpler pulse, consisting of the stress wave front, followed by the side release waves, the first 50 mm of the samples were covered with grout. The grout
extension controlled the compressional damage in the vicinity of the detonation point, leading to much cleaner pulse readings.

The SHPB experiments were also conducted to achieve the strain rate dependency of the tensile strength of Laurentian granite. Samples with thickness and diameter of 16 mm and 26 mm prepared and a wide range of striker bar velocities (3-30 m/s) were applied by adjusting the gas gun pressure (0.2 – 2 MPa). Samples were sandwiched between the incident bar and the transmitted bar and the stress pulses were recorded in the middle of the incident bar and the transmitted bar using 120 ohm strain gauges. The incident and the reflected pulses were obtained from the strain gauge mounted on the incident bar, and the transmitted pulse was obtained from the gauge mounted on the transmitted bar. The recorded pulses were captured and analyzed using the MREL DatatrapII data acquisition system. Stress pulse superposition was conducted to obtain the stress histories applied on either sides of the sandwiched rock sample. Consequently, the stress history of the sample was calculated, and from it the peak stress was reported as the tensile strength of the rock sample while the strain rate was obtained from the loading slope of the stress history curve. The strain rate, which is a function of the striker bar velocity, ranged from 5 to 50 s$^{-1}$. The tensile strength values where close to the static tensile strength values at the lower bond of the applied strain rate. At strain rates about 50 s$^{-1}$ the tensile strength of the samples were more than three times greater than the results obtained from Brazilian disc tests.

The combined results of these two experimental methods were used to achieve the Dynamic Increase Factor (DIF) in the RHT constitutive model. Once the DIF was implemented in the material model, the experiments were modeled using LS-DYNA in order to verify the results of the model. The SHPB experiments with striker bar velocities of 10, 20, and 30 m/s were simulated and compared against the results from the experiments.

**Chapter 3** In the study of rock damage and fragmentation by blasting, one of the most important variables is initiation timing. This is crucial, since stress waves are responsible for the majority of rock fracturing that occurs during blasting. Therefore, different values of initiation times result in different
combinations of wave interactions within the rock mass, leading to a variety of damage and fragmentation values. Experimental methods are mostly conducted in order to modify the fragmentation results by changing the timing variable. However, in most cases experiments are not instrumented to capture the physical phenomena during blast loading or pulse recordings are obtained in few selected points and are difficult to analyze. In this thesis, a 2D numerical model was built in an attempt to comprehensively study the damage evolution during blasting and the effect of stress waves in the resulted damage. An axisymmetric model was first built, representing a small scale blasthole filled with detonating cord and decoupled using water. The stress pulse in the middle of the blasthole was captured and used as a boundary condition in plane strain models, where a series of blastholes were initiated with different delay times. Damage as a function of initiation delay timing was studied. A minimum delay timing requirement was observed, demonstrating the fact that the applied delay times should not be selected shorter that the required time for damage evolution around a single blasthole. A parametric study was conducted to investigate the effect of blasthole spacing. Models with S/B ratios of 1, 1.5, and 1.75 were built in plane strain configuration. The initiation delay timings were varied from 0 to 100 µs. It was concluded that, even though the burden damage decreases by increasing the S/B ratio, however, the minimum delay timing requirement increases by increasing spacing contrary to the commonly used approach that scales delay time with burden (Otterness et al., 1991; Cunningham, 2005). The sensitivity of the introduced minimum delay timing requirement to the shape and the duration of the stress pulse was also investigated. It was concluded that, since the damage evolution is a function of rise time of the stress pulse, the minimum delay requirement is shorter in the case of the pulses with shorter rise times. Application of water as a coupling medium results in multiple wave reflections inside the blasthole leading to a cyclic loading system. It was concluded that the wave reverberation leads to more damage in the overall results, however, the minimum delay requirement is not a function of this cyclic loading system, since the rise time was not changed.
For the purpose of fragmentation analysis, large scale models with burden and spacing of 3m were built in plane strain configuration. Eight blastholes with diameter of 160 mm and filled with ANFO were modeled in a single row. The analysis conducted by varying the delay times between the holes from 0 to 10 ms. Once the blast induced fractures were obtained, the fragment size distributions were achieved using the WipFrag image analysis tool. The Swebrec fragmentation distribution was used to obtain the 50% and 80% passing sizes. It was concluded that, the delay times in which the tensile tails of stress waves from neighboring holes overlap, lead to finer fragment sizes. However, this results in a very small delay time window which is not practical, given the variability in rock mass parameters. A wider delay window, past the previous optimum, was introduced, which results in an acceptable improvement of fragmentation. The controlling mechanism of this optimum delay window was investigated using the elastic stress wave theory and it was concluded that the minimum delay time is associated with the passage of the reflected shear wave from the neighboring hole. Given the destructive superposition of the reflected shear wave with the incident wave of the neighboring blasthole, it was recommended that the delay times not be selected shorter than the time at which the reflected shear wave completely passes the neighboring blasthole. It was also concluded that application of longer delay times lead to the creation of coarser fragments near the free surface, suggesting that any wave interaction opportunity in this zone might be missed by applying excessive delay times.

Chapter 4 Destress blasting is considered as an effective method in deep underground hard rock excavations where there is a potential for strain bursts. While the conventional destress blast patterns are widely in practice for more than half a century, there is a lack of quantitative explanation towards the effectiveness of these patterns in mitigating the burst potential by introducing fracture surfaces and transforming the stress states ahead of a tunnel development face. Considering the fact that the blast induced damage is a function of the stress states around the blasthole upon detonation, a numerical study was conducted to study the shape and the directionality of the blast induced damage zones in the presence of in situ stresses. In a 3D model, different combinations of maximum and minimum stress values were
applied with a constant horizontal to vertical ratio of $k=1.5$ and the damage extent of a single blasthole as a function of the applied stress state was investigated. It was observed that in the presence of in situ stresses, the radial cracks tend to mostly propagate in the direction of maximum principal stresses. The numerical results showed that the damage extent decreases with the increase of the in situ stresses. The conventional destress blasting pattern was modeled in a 3D configuration and the damage results were studied in the presence of in situ stresses. It was concluded that, given the wide spacing between the blastholes, these patterns are not capable of creating damage zones ahead of the tunnel face in order to transform the stress field. Moreover, it was concluded that the isolated damage packets which were obtained as a result of these conventional patterns might introduce stress concentrations in zones of undisturbed rock between the damage packets. A new destress blast pattern was introduced to be applied in a circular tunnel with a diameter of 5 meters. The pattern consisted of 6 meter long horizontal blastholes in three rows drilled into the tunnel face in which the 2 m bottom end of the holes were charged using ANFO. Numerical stress studies before and after destressing, illustrated the effectiveness of the new pattern in transforming the stress states ahead of the tunnel face pushing the peak stresses 3 meter away from the tunnel development face. The suggested pattern created controlled fracture planes that reduced the stress magnitudes which is necessary for alleviating the rockburst potential.

5.2 Recommendations and future work

- The experimental study of the dynamic tensile strength was conducted using Laurentian granite samples. Laurentian is a fine grained granite and grain size may play a role in the dynamic increase factor. It is recommended to conduct these experiment on coarser grained rock samples in order to investigate the effect of grain size in the resulted strain rate dependency curve and obtain dynamic increase factor on a range of rocks.

- Since in this thesis only the dynamic tensile strength was studied, it is recommended that the rate dependency analysis also be conducted in obtaining the dynamic compressive strength of rock samples. The extent of the crushed annulus around the blasthole is majorly controlled by
compressive strength of the rock and adjusting the DIF for compressive strength provides more realistic damage results in the crushed zone.

- Even though the RHT constitutive model applied in this thesis is one of the most comprehensive material models available for modeling the dynamic behavior of brittle materials, it is recommended that other constitutive models like the LS-DYNA Concrete Model 159 (CSCM) or K&C model be used and the resulted damage compared with the results obtained from this work.

- In the fragmentation studies, the diameter of the blastholes was a fixed value. It is recommended that the effect of delay timing be studied in combination with different charge distribution settings achieved by changing the blasthole diameter. Changing the blasthole diameter alters the damage distribution around the blasthole, while in practice upscaling is conducted assuming a linear dependency, it is recommended that such upscaling practices investigated numerically.

- It would also be beneficiary to investigate the sensitivity of the damage and fragmentation results to the energy partition of the explosive used in the numerical models.

- The numerical models in the analysis of effect of delay timing in blast induced damage and fragmentation were built in 2D plane strain configuration. It is recommended to conduct the analysis in 3D configuration and study the fragment size distributions in cross-sections between the blastholes in order to investigate damage and fragmentation along a given bench height, which could not be captured in a 2D model.

- Application of delay timing leads to preconditioning of the rock material for the next blasthole. This idea was studied in chapter 3 where eight blastholes in a single bench blasting row was modeled, it is recommended that the effect of preconditioning between blast rows to be studied by modeling multiple bench blasting rows.

- The numerical models in this study are only able to capture the effect of stress waves on the resulted damage. It is recommended to model the rock blasting phenomenon using other
numerical modeling techniques like the Smooth Particle Hydrodynamics (SPH) method in order to capture the effect of gas expansion in addition to the stress waves on the resulted damage. The results obtained from different numerical codes promote comprehensive understanding of the blasting and fragmentation phenomenon, which eventually leads to better designing procedures.

- Collection of more data would be beneficiary in obtaining the damage dependency to stress intensity factor equation. This would be achieved by modeling blastholes with different hole diameter and/or using rock types with different strength values.
- Conducting experimental studies to capture damage zone dependency to the in situ stresses is also of interest to verify the results obtained numerically.
- The charge priming location, initiation delay times, and use of decked charges are the parameters, which control the distribution of the resulted damage in the area of study ahead of tunnel development face, where destress blasting is practiced. The sensitivity of the resulted damage to each of these parameters should be investigated.
- Field trials of the destress blasting pattern design introduced in this thesis are recommended in order to verify the numerical results and quantify the effectiveness of this pattern.

5.3 List of contributions

• The effect of stress wave interaction and delay timing on blast induced rock damage and fragmentation. *Published in: Journal of Rock Mechanics and Rock Engineering*
  
  [https://doi.org/10.1007/s00603-019-02043-9](https://doi.org/10.1007/s00603-019-02043-9)

• Tunnel face preconditioning using destress blasting in deep underground excavations. *Journal of Tunneling and Underground Space Technology (Under Review).*
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