EXPERIMENTAL AND NUMERICAL INVESTIGATION OF HYDRAULIC 
STIMULATION AS A RISK MITIGATION TECHNIQUE FOR FAULT SLIP 
ROCKBURST HAZARDS IN DEEP UNDERGROUND MINING

by

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B.Sc.Eng., Queen’s University, 2013

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF 
THE REQUIREMENTS FOR THE DEGREE OF 

MASTER OF APPLIED SCIENCE 

in 

THE FACULTY OF GRADUATE AND POSTDOCTORAL STUDIES 

(Geological Engineering)

THE UNIVERSITY OF BRITISH COLUMBIA 

(Vancouver)

April 2018

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Abstract

Fault slip and associated rockbursting present a significant risk to the safety of personnel and infrastructure in deep underground mines. Sudden displacement can occur on pre-existing fault planes when mining alters the stress regime present at depth, with the resulting energy release causing potentially extensive damage to mine workings. Recently, hydraulic stimulation, defined here as the injection of fluid in proximity to a fault, has been explored as a method of reducing fault slip risk. This has the potential to release the built-up stress and strain energy driving slip events before mining is advanced and workers are exposed; however, field trials to date have yielded varying degrees of success. In response, a two-part investigation has been conducted to examine the effectiveness of hydraulic stimulation in preventing fault slip for a range of geometric characteristics. In the first investigation, a laboratory testing procedure was developed in which pressurized water was injected through cylindrical granite specimens containing different offset saw cuts subject to triaxial loading. A state-of-the-art servo-controlled system was used to measure displacements, stress drops and moment magnitudes. Results indicate that heterogeneous fault surfaces produce slip events with greater moment magnitudes than smooth surfaces, and tend to respond less effectively to injection. Three-dimensional (3D) post-test specimen imaging demonstrates that variations in moment magnitude and stress release are linked to the breakdown of asperities and accompanying cohesive strength loss that occurs during shearing. This explains why stimulation is less effective on highly irregular fault surfaces, such as those containing asperities or rock bridges, since fluid pressure acts to reduce frictional strength only. In the second investigation, a numerical model of a hypothetical mining sequence was created to determine the effect of hydraulic stimulation on the frequency and severity of mining-induced slip. Though injection was effective at mitigating slip in planar fault models,
increasing fault strength was observed to correlate with larger, more damaging slip events. Additionally, when fault heterogeneity was explicitly incorporated, stimulation treatments were less effective as slip and fluid propagation were impeded by intact rock segments, illustrating the importance of understanding fault geometry before the successful implementation of field-scale treatments.
Lay Summary

Rockbursts present a significant hazard in underground mining. These catastrophic, earthquake-like events are generally triggered by the stress changes that accompany mining at great depths. They are particularly dangerous because they cannot readily be predicted or prevented with the current techniques available to engineers. A new method of preventing such events is investigated, whereby high-pressure water is pumped underground, allowing it to flow into hazardous areas and pre-trigger earthquakes so that they are less likely to occur when personnel or mine equipment are present. A laboratory study and computer simulations are conducted to test the effectiveness of the technique and factors to which it might be sensitive. The technique was shown to be promising, but its effectiveness was highly dependent on the characteristics of the geological structures in the area where the treatment was applied. Further field-scale studies are recommended before this technique can be used in a working mine.
Preface

This thesis is an original work by the author, Erika Schmidt. The work has been prepared as a manuscript-style thesis consisting of a laboratory testing component and numerical modelling component.

The laboratory testing component of the thesis is presented in Chapter 2. The triaxial testing device hardware was designed and manufactured by GCTS Testing Systems (GCTS), and initial setup was conducted by various members of the UBC rock mechanics research group, with assistance from GCTS. The experimental setup and testing procedure were designed by Erika Schmidt, with input from Dr. Erik Eberhardt. All tests were conducted by Erika Schmidt, with the exception of CTT002 which was conducted by Afshin Amini and Masoud Rahjoo, and the uniaxial compressive strength tests which were conducted jointly by Erika Schmidt, Afshin Amini and Masoud Rahjoo. Erika Schmidt was responsible for processing, analyzing, and interpreting the data, as well as authoring the manuscript for this chapter, which was reviewed by Dr. Erik Eberhardt.

The numerical modelling component of the thesis is presented in Chapter 3. Erika Schmidt built the numerical models, designed and coded the processing algorithm, analyzed and interpreted the results, and authored the manuscript for this chapter. The manuscript was reviewed by Dr. Erik Eberhardt.
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Acknowledgements

I wish to express thanks to my advisor, Dr. Erik Eberhardt, for providing me the opportunity to undertake this research. Without his guidance and support through the duration of my studies, the completion of this work would not have been possible. Additional thanks go to my committee members, Dr. Scott McDougall and Dr. Michael Bostock, for their helpful feedback.

I am incredibly grateful for the support, both academic and personal, from the fellow students in my research group. Acknowledgement goes to A. Amini, J. Aaron, M. Rahjoo and S. Yan for their help in the laboratory. I also wish to thank the faculty and staff of the Earth, Ocean and Atmospheric Sciences department for their support and assistance through the duration of this research project.

Financial support was provided by the National Sciences and Engineering Research Council of Canada (NSERC) as well as the Rio Tinto Centre for Underground Mine Construction (RTC-UMC), which facilitated the completion of this research.

Finally, this work would not have been possible without the unconditional love and support from my family, who both encouraged me to pursue graduate studies in engineering, and inspired a lifelong passion for science and learning, for which I am sincerely grateful.
Chapter 1 – Introduction

1.1 Problem statement

As near-surface mineral resources are progressively mined and depleted, underground mines are extending to greater depths than previously experienced. Consequently, because the in situ stresses encountered at such depths are significantly higher than those found in conventional mining environments, various forms of stress-driven rockmass failure are increasingly being encountered in underground mining operations (e.g., Counter, 2014), and pose a growing overall risk to safety in the mining industry. Rockbursts are a form of high stress failure that occur when mining-induced stresses exceed the strength of a rock mass at depth, resulting in the violent, unpredictable expulsion of rock into the active mining area. Most authors (e.g., Ortlepp, 1992; Wong, 1992) subdivide rockbursts into two categories: strain bursts and fault slip bursts. Strain bursts occur when localized concentrations of high stress develop adjacent to a mine opening resulting in the brittle failure of the intact rock. These events normally occur shortly after the excavation of stopes and access tunnels, and usually involve ejected volumes of less than 100 t of material (Blake & Hedley, 2003). Though they present a considerable hazard, strain bursts tend to be the easier of the two rockburst types to predict and manage (Castro et al., 2009), as their occurrence is usually confined to specific areas that can be delineated using standard tools, like distinct element modelling, and is less dependent on the presence of structural features, such as joints and faults, whose exact geometries are notoriously difficult to identify.

Conversely, fault slip bursts occur when the advancement of a mine excavation alters the stresses acting on a geological structure, triggering sudden shear movement (i.e., slip) along the discontinuity. The resulting energy release associated with these bursts occurs much like a
natural earthquake and can result in larger and less predictable failures found to produce seismic events with documented local magnitudes of up to $M_L = 5.5$ (Grobbelaar et al., 2017) based on the results of a comprehensive literature review of fault slip case histories (Figure 1.1). Faults slip bursts are not restricted to the proximity of the advancing excavation, but rather they have the potential to occur hundreds of metres away from active underground workings (McKinnon, 2006). As shown in Figure 1.1, fault slip-induced rockbursts have become a significant concern in major deep mining regions like the Witwatersrand in South Africa (McGarr et al., 1989), the Coeur D’Alene district of Idaho (White & Whyatt, 1999), and the Sudbury and Abitibi Greenstone Belt regions of eastern Canada (Blake & Hedley, 2003; Yao et al., 2009) as they impact production and present a significant safety risk to mine personnel and equipment.

![Figure 1.1: Results of a comprehensive literature search showing the magnitude, depth, and location of documented mining-induced fault slip case histories.](image)

1.1.1 Fault slip mechanics

Unlike natural earthquakes that occur on faults with lengths extending for hundreds of metres to kilometres (Zoback & Gorelick, 2012), the geological structures along which mine-scale fault
slip bursts occur are typically smaller and less developed, consisting of shorter fault splays or a series of discrete discontinuities separated by intact rock bridges (Gay & Ortlepp, 1979; Ryder, 1987). Together these structures form a preferentially oriented fabric along which fault slip events can rupture (Snelling et al., 2013), an example of which is presented in Figure 1.2. The heterogeneous nature of the discontinuity network implies that brittle failure mechanics during slip will be complex and highly influenced by geometry, where both shearing of discontinuities and breakage of intact segments contribute to overall behaviour. An understanding of the effect of fault geometry on slip is therefore a key component of mitigating the hazards associated with such events.

Figure 1.2: Cross section showing the anatomy of a hypothetical mine-scale fault. Solid lines represent pre-existing discontinuities and dashed lines represent preferentially-oriented intact pieces on which fault slip may nucleate. The principal stresses $\sigma_1$ and $\sigma_3$ act to generate normal stress, $\sigma_n$, and shear stress, $\tau$, on the fault, which may drive the structure to slip.
Mining-induced fault slip is triggered by stresses being redistributed in response to the excavation of rock during the advance of stopes and/or other mine openings (e.g., shafts, drifts, raises, etc.). On adversely-oriented faults, this can cause both an increase in shear driving stress, $\tau$, and a reduction in the normal stress clamping the structure, $\sigma_n$, (Blake & Hedley, 2003), as shown in Figure 1.2. If this trend continues to the point where the induced shear stress exceeds the fault’s shear strength, the structure will slip. This behaviour can be represented via the Coulomb slip criterion:

$$\tau \geq (\sigma_n - p_f) \tan \phi + c$$

where $\phi$ is the friction angle and $c$ is the cohesive strength of the fault surface. The $(\sigma_n - p_f)$ term represents the effective normal stress, where $p_f$ is the fluid pressure acting on the fault. The stored elastic strain energy released during slip, expressed in terms of the seismic magnitude, is a function of the area of slip and shear displacement that accompanies the event.

### 1.1.2 Hydraulic stimulation as a risk mitigation technique

Several different methods for mitigating fault slip rockburst hazards have been investigated and implemented in active mines. These include reducing the size of the excavations, avoiding high stress concentrations through optimized mine sequencing (Sjöberg et al., 2012), use of microseismic monitoring to provide early warning (Malek et al., 2008), and de-stress blasting to release built-up strain energy within the rock mass (Yao et al., 2009). Another method of mitigating fault slip hazards, originally proposed by Board et al. (1992) but which has received renewed attention in recent years due to advances in pumping technologies, uses the injection of pressurized fluid into or in proximity to the fault to pre-trigger slip and energy release before mining is conducted, thereby reducing the risk of events occurring unexpectedly while active
mining is taking place and workers and equipment are present. This treatment, referred to herein as hydraulic stimulation, works on the following premise. Ahead of an active mining face, a critically stressed fault may be present where the shear stress acting to promote slip is slightly less than the shear strength of the fault, which is in part derived from the normal stress clamping the structure (Figure 1.2). After mining passes through or near the fault, continued mining and subsequent increases in $\tau$ and/or reductions in $\sigma_n$ may tip the balance, triggering a fault slip event. To trigger slip before mining advances, hydraulic stimulation can be used to increase the fluid pressure, $p_f$, acting within the fault, reducing the effective normal stress that resists shearing, and promoting slip to occur at a lower driving shear stress. This effect, often referred to as hydraulic shearing or hydroshearing, was originally observed as an unintended consequence of subsurface wastewater injection (Healy et al., 1968) and has also been associated with hydraulic fracturing treatments used to increase hydrocarbon reservoir permeability (National Research Council, 2013). However, provided it can be controlled, fluid-triggered slip may present an opportunity in the mining industry as it has the potential to release built-up shear stress that may be responsible for unanticipated fault slip events (Kaiser et al., 2013).

Several field trials have been conducted to test this concept with varying degrees of success. Both Ohtake (1974) and Guglielmi et al. (2015) conducted injection through a single borehole intersecting a known fault zone and triggered a series of successful small shear displacements accompanied by both seismic and aseismic slip. Another experiment, documented by Board et al. (1992), attempted to pre-trigger slip in an active gold mine via a series of hydraulic stimulation field trials conducted on a known seismically-active fault. However, in this case, only a series of small slip events ($M < 0$) were observed in the trials; no larger events were recorded, suggesting
that the injection treatment would likely not have a meaningful impact on reducing the frequency and severity of future slip events. Hence, although hydraulic stimulation has the potential to activate fault movement, its effectiveness over different structural and geological conditions remains unclear. A better understanding of the sensitivity of slip to fault geometry under both mining-induced stress changes and injection treatments is needed before hydraulic stimulation can be reliably implemented as a risk mitigation technique.

1.2 Thesis objectives

This thesis aims to investigate the role that fault geometry plays in the effectiveness of hydraulic stimulation treatments used to mitigate fault slip risk. Two related studies have been conducted to examine this topic: a small-scale laboratory investigation and a series of numerical modelling simulations.

The objectives of the laboratory investigation are as follows:

1. To develop an experimental method of simulating excavation-induced fault slip on small-scale laboratory specimens. This will allow fundamental slip behaviours to be examined in a controlled environment prior to full scale implementation. The experimental methodology must allow a variety of fault geometries to be tested, and for injection treatment to be performed on the faulted specimens.

2. To measure and interpret the stress and strain response associated with the faulted specimens when external stresses are applied, and when injection treatments are performed.

3. To determine the effect that injection treatments have on releasing the built-up stresses driving fault slip events.
The numerical analysis performed for this thesis builds on the results of the laboratory experiment by simulating the full scale effects of fault slip and injection treatments. The objectives of this second study are as follows:

1. To model the field-scale fault slip response to injection using a two dimensional (2D) distinct-element approach. In this model, the fault will be located adjacent to a hypothetical mining sequence, which acts to trigger fault slip events as it is excavated.
2. To investigate whether injection treatment can effectively reduce the magnitude and frequency of slip events observed throughout the mining sequence.
3. To alter the fault geometry by adding bridges of intact material, and determine the effect this has on the effectiveness of the injection treatment.

1.3 Thesis structure

This thesis has been prepared in a manuscript-based format consisting of an introduction, two research chapters and a conclusion. Chapter 1 introduces the problem and objectives being explored as part of this thesis, including a presentation of relevant literature. Chapters 2 and 3 discuss methodology, testing, and interpretation of results related to the laboratory study and numerical analysis portions of the thesis, respectively. Lastly, Chapter 4 includes a summary of the key conclusions drawn from this study and suggestions for future work.
Chapter 2 - Laboratory Study

2.1 Introduction

2.1.1 Background

Triaxial compression testing has been used in a number of previous experiments to understand fault slip behaviour, including frictional strength characteristics, failure geometry, and energy release that occurs on the fault during slip (e.g., Moore et al., 1990; Blanpied et al., 1995; Savage et al., 1996; Goebel et al., 2012; Mcclaskey & Lockner, 2016). Triaxial compression testing is particularly useful because it allows for control of the principal stress magnitudes to mimic field conditions and mining depth; as such it has been chosen as the testing method used in the current series of experiments. Specimens containing a pre-cut surface are subject to loading and fluid injection, and precise measurements of stress and deformation can be used to derive slip characteristics.

The work presented in this paper builds on results of similar testing programs reported in the literature, which typically measure slip on a planar saw cut in response to applied load. The testing procedure developed involves a unique series of hydraulic stimulation triaxial compression tests that allowed for the investigation of the effects of two key variables on slip behaviour: i) fault surface geometry, determined by altering the characteristics of the slip surface cut into the specimen, and ii) fluid injection, determined by controlling the water pressure acting inside the specimen. To the author’s knowledge, no previously conducted laboratory testing has investigated these variables in such a manner.
2.2 Methodology

2.2.1 Test apparatus

Laboratory tests were conducted using a state-of-the-art servo-controlled triaxial compression test system manufactured by GCTS, integrated with a modified 2500 kN TerraTek load frame (Figure 2.1a). The apparatus houses a cylindrical rock specimen containing an inclined discontinuity (Figure 2.1c), to which various loads are applied. Axial load is applied via a hydraulic actuator and controls the major principal stress, $\sigma_1$, acting on the specimen. Lateral confining pressure is applied via pressurized oil contained within a sealed cell surrounding the specimen, and corresponds to the intermediate and minor principal stresses, $\sigma_2$, and $\sigma_3$. A jacket made of heat-shrink plastic surrounds the specimen to prevent oil from penetrating into the rock. Note that because the confining pressure is isotropic and axisymmetric, $\sigma_2$ is equal to $\sigma_3$. To ensure the seal between the specimen and confining fluid remains intact through the duration of the test, a layer of reinforced rubber coated with epoxy was placed between the specimen and injection platens, and a set of tensioned wire seals were applied to the outside of the specimen overtop of the plastic jacket. A small amount of duct tape was placed over the saw cut to hold it in place during assembly. Finally, water pressure is applied via an injection hole drilled axially through the centre of the specimen intersecting and passing through the discontinuity (Figure 2.1c). The system can generate confining fluid and injected water pressures of up to 200 MPa.
Linear variable differential transducers (LVDTs) are placed at key locations on the apparatus to measure deformation and enable servo control of the various stresses applied to the specimen. Two axial LVDTs and one circumferential LVDT are mounted on the test specimen to measure axial and lateral strains, respectively (Figure 2.1b). An additional LVDT is mounted to the top of the hydraulic actuator to measure the displacement of the loading piston. A schematic diagram of the full testing apparatus is shown in Figure 2.2.
Figure 2.2: Schematic diagram of triaxial testing apparatus. The pre-cut specimen is sealed inside a pressurized cell and is loaded axially until slip occurs. Displacement measurements are recorded via LVDTs mounted in locations shown.

2.2.2 Testing procedure

The aim of the triaxial loading procedure is to realistically simulate the loading conditions and rock mass response in a typical mining environment—specifically how the excavation of stopes gradually results in the transfer of stress to the pillars and excavation boundaries where fault structures may be present. To do this, the laboratory testing procedure was developed to simulate the stress, deformation and fluid pressure characteristics experienced by rock masses subjected to mining-induced loading. The rock specimen is first loaded hydrostatically ($\sigma_1 = \sigma_{2,3}$) by increasing the confining pressure to 25 MPa. The axial load is then gradually increased by lowering the axial actuator at a sufficiently slow rate such that dynamic stress effects can be considered negligible. A rate of 0.2 mm/min was determined to be sufficient for this. Note that
displacement was controlled by the LVDT mounted to the top of the actuator, rather than the specimen LVDTs located inside the cell. This allows for a constant far-field displacement rate to be applied external to the discontinuity structure, simulating how broad mine-scale deformations resulting from stope excavation might act to alter stresses surrounding a faulted rock mass. In this setup, the LVDTs fixed to the specimen are used to measure, rather than control, the deformation response to loading (and later fluid injection). The discontinuity can slip freely in response to deformation of the surrounding system. The boundary of the system in this case is the top of the loading column, which is sufficiently far from the saw cut so as to avoid influencing slip behaviour. Note that because the loading column in the triaxial apparatus includes the steel actuator and layers of rubber and epoxy sealant in addition to the rock specimen, the lab stiffness response may not match that of a field-scale rock mass, but this is acceptable for the current test as it aims to simulate general fundamental behaviour, rather than behaviour unique to a specific set of properties.

Specimens are subjected to two different testing procedures summarized in Figure 2.3 and detailed as follows. The first procedure, referred to herein as the ‘loading’ procedure, simulates the response of a fault under an induced load where no injection treatment is performed. In this procedure the confining pressure is held constant at 25 MPa and the water pressure is held constant at 0 MPa. The axial load is incrementally increased as described above until a maximum shear displacement of 4 mm is measured on the discontinuity. This is meant to simulate the gradual increase of shear driving stresses that can develop on faults oriented adversely to the direction of differential stress concentration surrounding mine excavations, and the resulting induced slip experienced on the fault. The second procedure, referred to as the ‘injection’
procedure, simulates the effect of hydraulic stimulation. In this scenario the test is started using the same confining pressure, water pressure and loading conditions as before. However, during incremental loading, once the slip threshold is approached, the axial displacement is paused by locking the position of the actuator. The slip threshold was determined from the ‘loading’ procedure tests and confirmed during the ‘injection’ tests by monitoring the LVDTs. In this case the slip threshold corresponded to an axial stress of 54 MPa axial stress, for which slip had yet to occur but a considerable amount of shear stress was observed to have built up along the discontinuity. The water pressure acting on the discontinuity is then increased at a rate of 3 MPa per minute to a maximum pressure of 24 MPa. This scenario simulates injection treatment being performed after mining has altered the in situ stress regime and the fault is in a state of marginal stability. Limiting the water pressure to 24 MPa is necessary to ensure the seal around the specimen is not broken and that the water does not leak into the confining fluid, which is held constant at a pressure of 25 MPa.
2.2.3 Specimen preparation procedure

Tests are performed on 54 mm-diameter rock specimens cored from intact blocks of Hardy Island granite. This material was chosen as it acts as an analog for the hard, brittle rocks present in the mining regions where fault slip-induced rockbursts are normally a concern (Blake & Hedley, 2003). In addition, Hardy Island Granite is massive, homogenous and of excellent rock mass quality with no visible fractures, foliation or anisotropy that could affect deformation characteristics. Samples were prepared according to ASTM D4543. Physical and geotechnical properties of Hardy Island granite are presented in Table 2.1, based on a series of standard uniaxial and triaxial compression tests (ASTM D7012 - Methods B and D) performed on representative intact samples.
Table 2.1: Properties of Hardy Island granite. Results are based on uniaxial and triaxial compression testing conducted on representative intact samples. Full test details can be found in Appendix B.

<table>
<thead>
<tr>
<th>Material description</th>
<th>Grey-coloured granite containing visible quartz, feldspar and biotite grains approx. 2-4 mm in diameter.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Unit weight (kN/m³)</td>
<td># of tests</td>
</tr>
<tr>
<td>Uniaxial Compressive Strength (MPa)</td>
<td># of tests</td>
</tr>
<tr>
<td>Young’s Modulus (GPa)</td>
<td>No confinement</td>
</tr>
<tr>
<td>Poisson’s Ratio</td>
<td>No confinement</td>
</tr>
</tbody>
</table>

The test specimens were then modified by drilling a 15-mm diameter axial fluid injection hole.

An angled saw cut was then made at 35 degrees to the long axis of the core, forming a preferentially-oriented discontinuity surface along which the specimen will slip. The final specimen height-to-diameter ratio is approximately 2.3:1 (however this ratio varies slightly from specimen to specimen due to the machining process).

From this initial core preparation, four different discontinuity geometries were created, referred to herein as the polished cut, straight cut, small asperity and large asperity. The polished cut and straight cut retain the planar surface produced previously by the saw cut process. The polished cut differs in that its surface roughness was further reduced using a flat lap polishing wheel with 950-grit abrasive, followed by titanium dioxide polishing powder. Rough edges and ridges
related to the cutting process for the straight cut surface were removed via hand sanding with 220-grit sandpaper. The small and large asperity surface geometries were prepared with single distinct offset ridges of 1.5 mm and 8 mm, respectively, across the middle of the cut face (Figure 2.1c). These surfaces were prepared by first manually filing the sample face and then finishing using a Dremel rotary tool. The finished ridges produce an interlocking stepped asperity in the middle of the specimen. A 6,200 point/cm² 3D laser scanner was used to verify the offset distances and also determine the surface variation present on each discontinuity, defined here as the out-of-plane deviation of the prepared surface geometry from the ideal surface geometry. Asperity cuts were observed to have more surface variation due to the less precise methods of preparation. Conversely the polished cut has a surface variation so small it is below the accuracy of the scanner, which is approximately 0.13 mm. Dimensions, representative photographs, and representative surface scans for each of the four discontinuity types are shown in Table 2.2. The full specimen dataset can be found in Appendix A, and full details of the specimen preparation and scanning procedure can be found in Appendix C.
Table 2.2: Surface geometry, representative pre-test photographs and representative pre-test 3D scans of four tested discontinuity types. Full dataset can be found in Appendix A.

<table>
<thead>
<tr>
<th>Discontinuity type</th>
<th>Asperity offset (mm)</th>
<th>Surface variation (+/- mm)</th>
<th>Representative specimen photograph</th>
<th>Representative pre-test surface scan</th>
</tr>
</thead>
<tbody>
<tr>
<td>Polished cut</td>
<td>0</td>
<td>&lt; 0.13</td>
<td><img src="image" alt="Polished Cut" /></td>
<td><img src="image" alt="Polished Cut Scan" /></td>
</tr>
<tr>
<td>Straight cut</td>
<td>0</td>
<td>0.15</td>
<td><img src="image" alt="Straight Cut" /></td>
<td><img src="image" alt="Straight Cut Scan" /></td>
</tr>
<tr>
<td>Small asperity</td>
<td>1.5</td>
<td>0.2</td>
<td><img src="image" alt="Small Asperity" /></td>
<td><img src="image" alt="Small Asperity Scan" /></td>
</tr>
<tr>
<td>Large asperity</td>
<td>8</td>
<td>1</td>
<td><img src="image" alt="Large Asperity" /></td>
<td><img src="image" alt="Large Asperity Scan" /></td>
</tr>
</tbody>
</table>
When designing this experiment, sources of error were considered and attempts were made to minimize their effect whenever possible. Though it was impossible to create asperity offsets that were identical between specimens of the same testing group, measuring and quantifying offsets using the 3D scanner helped to minimize surface variations. Limiting the shear displacement during the test to a maximum of 4 mm helped ensure a good seal is maintained to prevent confining fluid from entering the saw cut, and also minimized the effect the plastic jacket might have on slip behaviour by preventing it from undergoing large strains. During testing, fluid pressures, volumes, stresses and displacement rates were carefully monitored to ensure no leakage was occurring, and that loading behaved as programmed. Note that because there is a small delay between sensor feedback and the servo response, variation is still observed in some of the outputs, notably axial stress. However, the scale of these variations is small compared to the overall stress changes observed during slip events, therefore it was deemed acceptable.

2.3 Results
A total of eight fault slip tests were conducted to analyze responses of the four discontinuity types and two test procedures under consideration. Table 2.3 summarizes the test parameters, peak shear stresses, and total shear displacements observed on the discontinuity in each case. Note that the total shear displacement in the loading tests is fixed as tests are programmed to run until 4 mm of shear displacement accrues. Likewise, the peak shear stress in the injection tests is fixed because tests are programmed to increase axial displacement to achieve a threshold axial stress value prior to injection.
Table 2.3: Summary of conducted tests, including measurements of peak stress and total displacement on the specimen discontinuities.

<table>
<thead>
<tr>
<th>Test ID</th>
<th>Test program</th>
<th>Discontinuity type</th>
<th>Asperity offset (mm)</th>
<th>Confining pressure (MPa)</th>
<th>Peak shear stress on discontinuity (MPa)</th>
<th>Total shear displacement on discontinuity (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>HF027</td>
<td>Loading</td>
<td>Polished saw cut</td>
<td>0</td>
<td>25</td>
<td>28.2</td>
<td>4</td>
</tr>
<tr>
<td>HF032</td>
<td>Loading</td>
<td>Straight saw cut</td>
<td>0</td>
<td>25</td>
<td>33.1</td>
<td>4</td>
</tr>
<tr>
<td>HF024</td>
<td>Loading</td>
<td>Small asperity</td>
<td>1.5</td>
<td>25</td>
<td>34.3</td>
<td>4</td>
</tr>
<tr>
<td>HF030</td>
<td>Loading</td>
<td>Large asperity</td>
<td>8</td>
<td>25</td>
<td>59.8</td>
<td>4</td>
</tr>
<tr>
<td>HF028</td>
<td>Injection</td>
<td>Polished saw cut</td>
<td>0</td>
<td>25</td>
<td>13.9</td>
<td>0.36</td>
</tr>
<tr>
<td>HF023</td>
<td>Injection</td>
<td>Straight saw cut</td>
<td>0</td>
<td>25</td>
<td>13.7</td>
<td>0.31</td>
</tr>
<tr>
<td>HF025</td>
<td>Injection</td>
<td>Small asperity</td>
<td>1.5</td>
<td>25</td>
<td>13.7</td>
<td>0.27</td>
</tr>
<tr>
<td>HF031</td>
<td>Injection</td>
<td>Large asperity</td>
<td>8</td>
<td>25</td>
<td>13.8</td>
<td>0.18</td>
</tr>
</tbody>
</table>

2.3.1 The evolution of displacement and stress on the discontinuity surface during slip

Fault shear displacement is calculated using the average of the two axial displacement readings from the specimen LVDTs. The total measured axial displacement, $D_{\text{axial (total)}}$, is made up of two components. A portion, $D_{\text{axial (slip)}}$, is due to slip across the fault, which acts in a compressive, or positive, direction, and another portion, $D_{\text{axial (elastic)}}$, is due to the elastic expansion of the specimen when the axial stress is relieved during slip, which acts in a negative direction. Based on this, the slip component of axial displacement can be obtained as follows:

$$|D_{\text{axial (slip)}}| = |D_{\text{axial (total)}}| + |D_{\text{axial (elastic)}}|$$  \hspace{1cm} (2.1)

Elastic displacement is calculated based on the stiffness (i.e., Young’s modulus) of the specimen and the measured axial stress drop. Once the axial slip displacement is determined, the shear displacement component is calculated using the known angle of the discontinuity.
Figure 2.4 shows the fault shear stress versus shear displacement for the four loading tests, while Figure 2.5 shows the corresponding shear displacement with respect to time. Note that shear stress is calculated based on the measured axial and confining stresses applied to the specimen and the known angle of the surface. Stick-slip behaviour can be observed in each test, characterized by discrete periods of displacement accompanied by a sudden drop in shear stress. The stress drops range between 1 and 30 MPa for the different tests. In Figure 2.4 this episodic behaviour manifests as a characteristic saw-tooth pattern caused by the cycle of stress buildup, slip, and stress relief. As shown in Figure 2.5, slips occur nearly instantaneously over a period of 0.1 s or less. Each slip was accompanied by an audible release of strain energy during testing. Note that stick-slip behaviour should be distinguished from the smaller scale (~3 MPa) stress fluctuations that are not accompanied by significant shear displacement. These are an induced effect caused by the servo controller cycling to maintain control over the position of the actuator and are not considered to be experimentally significant.
The difference in slip signature observed on the planar cuts versus the surfaces containing asperities should be noted. Slip on the planar cuts is much more regular: each slip cycle has a
similar duration and stress drop magnitude. Conversely, the surfaces containing asperities present a much more irregular response characterized by episodes of slip separated by long periods of aseismic creep. This observation points to the effect that surface heterogeneity may be having on discontinuity behaviour. On heterogeneous surfaces, some areas of the discontinuity will fail more easily than others that remain locked under load. As weaker segments gradually fail, driving stresses will concentrate on locked portions of the fault until their strength is exceeded and a larger period of slip and stress drop occurs. This is similar to the effect observed in laboratory tests presented in Goebel et al. (2012) that investigate fault surface heterogeneity. The ultimate result is an evolution of behaviour from the lowest-strength polished surface, which behaves in a similar manner to conventional slip tests (e.g., Moore et al., 1990; Blanpied et al., 1995) to the asperity cuts, whose behaviour begins to resemble responses seen in brittle failure tests on intact rock (e.g., Martin, 1993) which show a clear drop from peak to residual strength.

For comparison, Figure 2.6 and Figure 2.7 show, respectively, the shear stress versus displacement and shear displacement versus time responses of the injection tests, during the phase in which axial loading is locked and the discontinuity is allowed to slip as the injection pressure is increased. In this case the characteristic saw-tooth pattern is not observed because the axial load on the sample is not being increased, but rather is held constant. Nevertheless, stick-slip behaviour is still present, identified in Figure 2.6 by the sudden periods of displacement and stress drop. Compared to the loading tests, the difference in slip signature between the different saw cut geometries is less pronounced in the injection tests, however in Figure 2.7 it can be observed that the asperity cuts experience less total shear displacement, likely because the locked asperities provide greater resistance to shearing than the planar surfaces.
Figure 2.6: Shear stress vs. shear displacement on the discontinuity measured during the injection phase of the four injection tests conducted.

Figure 2.7: Shear displacement on specimen discontinuity vs. time during injection phase of injection tests.
2.3.2 Determining fault slip magnitudes and corresponding stress drops

Seismic moment, $M_0$, and moment magnitude, $M_w$, were calculated for each recorded period of slip via the equations presented in Hanks & Kanamori (1979) and Kanamori (1994):

$$M_0 = \mu A\bar{D}$$

(2.2)

$$M_w = \frac{2}{3} \log M_0 - 6$$

(2.3)

where $\mu$ is the shear modulus of the rock mass surrounding the fault (derived from the elastic properties presented in Table 2.1), $A$ is the area of the slipping surface, and $\bar{D}$ is the average shear displacement experienced on the fault during slip. The units for $M_0$ are Newton-metres.

Note that this experiment is limited to detecting events that rupture the full discontinuity surface causing a measurable axial displacement. To account for this, and to separate such events from background noise, a cut-off seismic moment of 400 Nm was used, below which no events were further analyzed.

The range of magnitudes observed in the loading and injection tests are presented in Table 2.4, along with the measured shear stress drop that occurs on the fault during each slip event. The only test for which no seismic events were recorded, and hence no event magnitudes or stress drops could be calculated, was the injection test conducted on the large asperity. The strength and dilation required to overcome the height of the asperity in this case was too high and water pressure could not be raised high enough to act against the shear resistance of the discontinuity and trigger slip.
Table 2.4: Range of moment magnitudes and shear stress drops corresponding to seismic events detected for each fault slip test. Note that slip could not be initiated in test HF031 and therefore no seismic events were detected.

<table>
<thead>
<tr>
<th>Test ID</th>
<th>Test program</th>
<th>Discontinuity type</th>
<th>Range of moment magnitudes, $M_w$</th>
<th>Range of shear stress drops, $\Delta \sigma$ (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>HF027</td>
<td>Loading</td>
<td>Polished saw cut</td>
<td>-3.9 – -3.3</td>
<td>1.9 – 12.4</td>
</tr>
<tr>
<td>HF032</td>
<td>Loading</td>
<td>Straight saw cut</td>
<td>-4.2 – -3.2</td>
<td>4.9 – 21.8</td>
</tr>
<tr>
<td>HF024</td>
<td>Loading</td>
<td>Small asperity</td>
<td>-4.3 – -3.3</td>
<td>0.5 – 10.8</td>
</tr>
<tr>
<td>HF030</td>
<td>Loading</td>
<td>Large asperity</td>
<td>-3.3 – -3.1</td>
<td>9.7 – 26.0</td>
</tr>
<tr>
<td>HF028</td>
<td>Injection</td>
<td>Polished saw cut</td>
<td>-4.3 – -3.7</td>
<td>0.09 – 3.1</td>
</tr>
<tr>
<td>HF023</td>
<td>Injection</td>
<td>Straight saw cut</td>
<td>-4.2 – -3.4</td>
<td>0.04 – 5.9</td>
</tr>
<tr>
<td>HF025</td>
<td>Injection</td>
<td>Small asperity</td>
<td>-4.3 – -4.0</td>
<td>0.04 – 0.7</td>
</tr>
<tr>
<td>HF031</td>
<td>Injection</td>
<td>Large asperity</td>
<td>n/a</td>
<td>n/a</td>
</tr>
</tbody>
</table>

Naturally, the event magnitudes presented in Table 2.4 are much lower than those observed in deep mining environments, where documented fault slip magnitudes of $M = 2$-5 are typical (Blake & Hedley, 2003). This difference can be attributed to the size of the slip surface and magnitude of shear displacement under consideration, which scale with magnitude via established relationships (Kanamori, 1994). Using these relations as a validation tool, the moment magnitudes obtained from the current experiment are found to be within the expected range. Shear stress drop, on the other hand, is generally not dependent on the scale of the fault and usually ranges between 0.1-10 MPa for natural earthquakes (Zoback & Gorelick, 2012). However, stress drops occurring over localized asperities or stronger portions of fault surfaces can be much higher than this, in some cases up to 70 MPa (McGarr et al., 1979). Hence, the range of stress drops observed is within the expected range, and the larger drops observed in some cases are the result of higher strength portions of the discontinuity surface failing. For more information on earthquake scaling relationships and the validation procedure used in this experiment, refer to Appendix D.
2.4 Discussion

2.4.1 The contribution of geometry to fault strength

For built-up strain energy to be released seismically, a change in fault strength must occur during slip (Ryder, 1987). Consider that if fault strength remained constant, sliding would be stable since each incremental increase in loading stress would be met with a corresponding displacement acting to unload the fault. Instead, in seismic systems there exists a drop from the peak fault strength that acts just prior to slip, to the sliding strength that acts during slip. This drop is accompanied by a sudden release of shear stress and a corresponding release of strain energy that continues until the excess shear stress is completely dissipated (McGarr, 1999). A portion of this strength drop is generally considered to be caused by dynamic velocity and displacement weakening effects as described by rate- and state-dependent friction models (e.g., Lorig & Hobbs, 1990; Dieterich & Kilgore, 1994). However on the less-developed, discontinuous faults present in mining environments, a major component of strength drop is likely caused by physical changes in fault surface properties during slip, including the breakage of asperities or changes in surface roughness due to shearing (Sainoki & Mitri, 2014). A modified version of the Coulomb slip criterion can be used to illustrate this concept:

\[ \Delta \tau = \sigma_n' (\tan \phi_{peak} - \tan \phi_{sliding}) + (c_{peak} - c_{sliding}) \]  (2.4)

where \( \Delta \tau \) is the drop from peak to sliding shear strength, \( \sigma_n' \) is the effective normal stress acting on the surface, \( \phi_{peak} \) and \( \phi_{sliding} \) represent the peak and sliding friction angle of the surface, and \( c_{peak} \) and \( c_{sliding} \) are the peak and sliding cohesive strength of the surface. Note that the frictional and cohesive elements of strength are separated as they typically arise from different underlying phenomena. Friction is developed through interlocking micro-scale surface irregularities that dilate to override each other during shearing and then reseat themselves once
shear motion has stopped (Goebel et al., 2012). Because of this, frictional strength is necessarily dependent on the normal stress acting on the surface. Conversely, cohesive strength is typically associated with intact rock and is permanently altered during shearing due to the breakage of intact segments (Diederichs, 2007). By definition it is independent of the surface normal stress.

In the current experiment, the differences in geometry between tested specimens are responsible for differences in the frictional and cohesive properties of the discontinuity surfaces. The specimens containing planar cuts exhibit predominantly frictional strength behaviour. This is confirmed by 3D scans of the polished and straight cut surfaces presented in Table 2.5, for which the degree of surface variation in the pre- and post-test scans appears the same, suggesting that little, if any permanent surface damage occurred as a result of slip, and deformation was mostly due to the discontinuity surfaces overriding each other. In contrast, the surfaces containing small and large asperities are observed to possess cohesive strength in addition to frictional strength due to the presence of the interlocking asperity ridge in the middle of each specimen. Since these asperities are too large to allow dilation and overriding to occur during slip (relative to the confining stresses applied), shearing of intact material becomes the primary form of deformation, resulting in a permanent change to the geometry of the surfaces. In the pre- and post-test 3D scans of these asperities, noticeable changes to the surface can be observed, either as chipping of the asperity with fragment pieces 2-10 mm in size, or as complete shearing of the edge of the previously-intact asperity. In tests where visible asperity breakage was present, gouge could be observed on the surface after testing as a result of the permanent surface geometry changes.
Table 2.5: Pre- and post-test surface scans of loading test discontinuity surfaces. Asperities display significantly more surface damage after testing than planar cuts.

<table>
<thead>
<tr>
<th>Test ID</th>
<th>Test program</th>
<th>Discontinuity type</th>
<th>Pre-test surface scan</th>
<th>Post-test surface scan</th>
</tr>
</thead>
<tbody>
<tr>
<td>HF027</td>
<td>Loading</td>
<td>Polished saw cut</td>
<td><img src="image1" alt="Pre-test image" /></td>
<td><img src="image2" alt="Post-test image" /></td>
</tr>
<tr>
<td>HF032</td>
<td>Loading</td>
<td>Straight saw cut</td>
<td><img src="image3" alt="Pre-test image" /></td>
<td><img src="image4" alt="Post-test image" /></td>
</tr>
<tr>
<td>HF024</td>
<td>Loading</td>
<td>Small asperity</td>
<td><img src="image5" alt="Pre-test image" /></td>
<td><img src="image6" alt="Post-test image" /></td>
</tr>
<tr>
<td>HF030</td>
<td>Loading</td>
<td>Large asperity</td>
<td><img src="image7" alt="Pre-test image" /></td>
<td><img src="image8" alt="Post-test image" /></td>
</tr>
</tbody>
</table>
2.4.2 The relationship between fault geometry, peak stress and slip magnitude

Changes in fault surface geometry appear to affect both the stress drop and moment magnitude observed in the loading tests. In Figure 2.8, which plots moment magnitude vs. shear stress drop for loading-induced slip events, data points tend to group based on saw cut geometry, with the planar cuts generally producing lower magnitudes and stress drops, and the large asperity cut being associated with the highest magnitudes and stress drops. Note that an exception to this observation exists for slip events on the small asperity surface; potential causes for this will be discussed later. An additional observation from Figure 2.8 is the log-linear correlation between moment magnitude and stress drop. This correlation can be traced back to fundamental slip scaling relationships; combining elastic stiffness relationships with Equations 2.2 and 2.3 presented earlier, the shear stress drop, $\Delta \sigma$, and by extension, the drop from peak to sliding strength can be approximately related to moment magnitude as follows:

$$M_w \approx \frac{2}{3} \log \Delta \sigma + \log A - 6$$

(2.5)
Following Equation 2.5, surfaces with a higher $\Delta \sigma$, namely those with greater heterogeneity, produce higher-magnitude events. On the large asperity cut this stress drop was observed to result from the breakage of the asperity, causing a permanent reduction in surface cohesion. Since the planar cuts do not possess the same cohesive strength element, they do not experience the same drop in stress, and hence the magnitudes produced during sliding are lower. The seismicity observed on the planar cuts may instead be governed by the velocity weakening character exemplified in rate- and state-dependent friction models and is a result of microscale changes in pore pressure, temperature, and the population of surface contacts that change as the surfaces override each other (Brace & Martin, 1968; Dieterich, 1978).

The exception to this magnitude correlation is in the loading test performed on the small (1.5 mm) asperity cut. Though the largest episode of slip ($M_w = -3.31$) in this test appears to align...
roughly with the trend of the other tests, there is a population of slips ranging from $M_w = -4.25$ to $-3.67$ that are much smaller than predicted. The reason for this is somewhat unclear but likely twofold. In part, this is probably due to the fact that the 1.5 mm asperity was small enough to allow some dilation and overriding during slip. Because of this, the asperity was not completely sheared off during the experiment (as confirmed by post-test specimen observations), meaning there was still an element of cohesion acting after the fault began to slip. This would logically lead to a higher sliding strength, and a lower resulting strength drop (from peak to residual) and moment magnitude. The second influencing factor could be the greater surface irregularity caused by the presence of the asperity. Such heterogeneity could cause stress to concentrate in certain locations, triggering slips that result in smaller overall displacements and magnitudes than on the planar discontinuities, where release is triggered across the entire surface at once. This behaviour may be suppressed on the 8 mm asperity cut because the asperity itself is large and strong enough to withstand significant deformation for the majority of the test, until stresses become high enough to cause the asperity itself to fail. Essentially the small asperity presents a ‘goldilocks-like’ slip response, whereby it is large enough that the asperity can interlock in places and produce small gradual slips as the two heterogeneous surfaces override each other, but not so large that shearing of the asperity results in a large loss of cohesive strength and correspondingly-sized stress drop and seismic event. This trend can also be observed in Figure 2.4, in which the small asperity produces the smallest periods of stress drop and slip displacement compared to the other sample geometries, despite possessing a strength and surface roughness that is in the middle of the range of the samples tested.
2.4.3 The effect of hydraulic stimulation treatment on slip behaviour

In the hydraulic stimulation tests, injection causes slip to occur at much lower shear stresses than in the loading tests. Slip is not observed to occur in the loading tests until shear stresses of 13.5 to 55.8 MPa are reached. Conversely, discontinuity shear stresses in the injection tests are limited to a maximum of 13.9 MPa as a result of the test design. Despite this, numerous episodes of slip are triggered during the injection tests. Interestingly, there is a notable difference between the moment magnitudes associated with slip events triggered by hydraulic stimulation compared to loading. This is exemplified in Figure 2.9, which presents cumulative frequency-magnitude distributions for events from all test cases. Despite the same set of surface geometries being tested in both the loading and injection programs, injection-triggered slips are associated with consistently lower moment magnitude distributions. Injection tests were also associated with longer periods of aseismic creep occurring between episodes of seismic slip. Guglielmi et al. (2015) noted a similar observation of primarily aseismic slip behaviour occurring within the fluid injection front on a hydraulically-stimulated field-scale fault, and suggested a number of explanations. These include the possibility of fault dilation during shear causing a local drop in water pressure on saturated discontinuities and resultant fault strengthening, or that the high water pressures required to overcome surface strength were bringing the fault into a state of conditional stability (a regime in which low effective normal stresses prevent the occurrence of slip, as explained in detail in Scholz, 1998). Though the latter effect may be present in the current experiment, especially during later stages of the injection tests in which water pressures approach the pressure of the confining fluid, the former does not likely have a significant impact. If it did, the same reduction in magnitude and stress drop present in the injection tests would be observed in the loading tests, since the discontinuity was saturated with water in both
experiments. From the results presented in Figure 2.9 this does not appear to be the case, since the mean moment magnitude of the injection tests is considerably smaller than that of the loading tests. It is also possible that the reduction in event magnitude and frequency is caused by the lower axial stresses present in the injection tests resulting in less total energy being available to drive slip; however, because it is assumed that slip magnitude is proportional to the change in stress, rather than total stress available (Equation 2.5), the exact mechanism by which this would affect results remains unclear.

![Figure 2.9: Cumulative frequency-magnitude plot showing moment magnitude distribution of events from loading and injection tests. Injection produces lower-magnitude slip distributions than loading.](image)

Another possible explanation is the fact that surface heterogeneity may result in fluid pressure acting non-uniformly across the surface. Ishibashi et al. (2017) determined that on fractures created in granite lab specimens that are sized similarly to the current experiment, significant flow channelization can be observed when fluid pressure differentials are applied to the surfaces,
creating preferential flow paths that cover only 40-60% of the total fault surface area. This would lead to slip being triggered preferentially in some small areas, rather than on the whole surface at once. Since in this case slip and stress release is occurring over a much smaller area than the full surface, this effect results in smaller magnitudes and average stress drops being observed. A similar effect was noted in field-scale injection trials by Goldbach (2009).

The former effect applies well to the planar discontinuities in which surface overriding is the controlling slip mechanism, however it does not fully explain the slip response seen on surfaces with a cohesive strength component (i.e., the asperity slip tests). Attention should again be drawn to differences in behaviour between frictional and cohesive elements under shear in this case. Whereas the reduction in normal stress accompanying injection reduces frictional strength by enabling overriding, it does not have the same effect on cohesive strength. Because it is intact cohesive portions of rock that are shearing in the asperity slip tests, increasing fault dilation through injection will not serve to enable significant strength reduction, and the technique is less successful at triggering slip. This is particularly obvious in the case of the large 8 mm asperity for which slip could not be trigged by injection, but also applies to the behaviour of the 1.5 mm asperity, where the largest slip event ($M_w = -3.31$) observed in the loading case was believed to coincide with partial asperity shearing. This was not present in the injection case, where slips were limited to a maximum magnitude of $M_w = -4.0$. Supporting this hypothesis is the observation that post-test asperities are significantly less damaged after the injection tests compared to the loading tests, as shown in Figure 2.10.
Figure 2.10: Post-test specimen photographs comparing asperity surface damage in the a) loading test and b) injection test on the large 8 mm asperity specimen. Note that greater damage can be observed in loading test specimen.

2.4.4 Implications for field-scale hydraulic stimulation treatment

From the results presented above, injection has been shown to successfully initiate slip on pre-existing lab-scale faults of various geometries. In addition, shear stress drops of up to 5.9 MPa are observed to accompany periods of injection-induced slip. This suggests that hydraulic stimulation treatment has potential as a method of pre-triggering slip prior to mining an area while stresses are still below their critical level, in an effort to reduce the stresses driving future fault slip events and stored strain energy releases. Results also suggest that the seismic magnitude of slips triggered by injection is significantly lower than that of slips triggered by stress redistribution alone. If this proves to be a universal effect, there would be less of a risk of high magnitude, damaging events being triggered as a direct result of injection treatments, and stored strain energy would ultimately be released in a more aseismic manner. We also note that fault geometry has a significant impact on the effectiveness of injection treatment to relieve built-up stress. Stronger faults, particularly those containing intact segments that contribute a
cohesive strength component, may not respond to injection treatment as well as persistent discontinuity structures subject to loads approaching the strength of the structure, since the effect of fluid pressure on overcoming cohesive strength is shown to be limited. Additionally, even if strength is predominantly controlled by frictional components, the fluid pressures that can realistically be achieved by current pumping technology may not be high enough to initiate slip and stress relief, particularly if fault surface heterogeneity limits the areas through which the fluid can flow.

In order to validate the responses observed in this study, a key next step would be to determine if field-scale slip responses resemble the behaviour observed in the lab. Geometries tested in the current experiment are much simpler and more regular than those seen in mine-scale faults, in which differing lithologies, rock fabric and intact rock bridges will be present. It is possible that large-scale structural features will affect how stresses are released and redistributed after injection has been conducted. In some cases, this could act to either prevent the release of stress over a wide area, rendering treatment ineffective at preventing future slips, or may merely result in the transfer of stress into areas located farther away from the target injection zone increasing the risk of unintended seismic activity occurring in areas that would otherwise not be prone to slip. Such far field effects cannot be determined from lab testing alone, and will necessitate the use of numerical models that incorporate Discrete Fracture Networks (DFNs), as well as field-scale experiments in realistic rock mass conditions.
2.5 Conclusions

An experimental study involving the development of a unique testing procedure was conducted to explore the influence of discontinuity surface geometry on fault slip behaviour induced by both loading (shear stress increase) and fluid injection (effective normal stress decrease). Triaxial compression tests were performed on granite specimens containing pre-existing discontinuities of various geometries, and resulting stress changes and specimen deformations were monitored. Loading and injection tests produced slip events with moment magnitudes of $M_0 = -4.3$ to $-3.8$ and shear stress drops of 0.004 to 26 MPa, consistent with ranges expected based on the scale of the experiment. Differences in fault geometry are observed to cause considerable variation in fault strength, stress drop, event magnitude, and sliding characteristics. A modified version of the Coulomb slip criterion is used to explain this difference in behaviour, in which variations in frictional vs. cohesive surface strength affect the geomechanical response of the discontinuity to loading and injection.

Fluid injection is observed to trigger slip events with considerably lower magnitudes than events triggered by loading. If this proves to be a universal effect, it suggests that hydraulic stimulation could be an effective method of mitigating fault slip rockburst hazards, by safely pre-triggering stress relief in mining zones susceptible to fault slip before mine personnel enter the area. However, fault geometry appears to have a significant effect on the effectiveness of such a treatment, in that fault zones containing large asperities or portions of intact rock that result in fault strengths greater than what pumping pressures can trigger may hinder the ability of fluid injection to cause slip and release built up strain energy.
Chapter 3 – Numerical Simulations

3.1 Introduction

While laboratory testing methods can be used to simulate the fault surface mechanics governing mining induced fault slip behaviour, the limited size of laboratory experiments prevents them from being effective at simulating the larger-scale effects of fault slip in a mining environment, where stress and displacement changes resulting from slip may persist for hundreds of metres across complex structural regimes in the subsurface. As such, a numerical modelling study was conducted to complement the experimental findings presented in Chapter 2. This involved a series of numerical models designed to simulate the field-scale effects of a hypothetical hydraulic stimulation scenario performed on faults of various geometries through the life of a mining sequence.

3.1.1 Background

Numerical models have become an indispensable tool in the design of solutions to complex mine-scale stability problems. This is particularly true for areas prone to fault slip, where numerical modelling has been used in mine sequence and risk analysis exercises for numerous operating projects (e.g., Malek et al., 2008; Bewick et al., 2009; Yao et al., 2009). However, though numerical models are effective at simulating behaviour for simplified subsurface geometries, existing modelling techniques are limited in their ability to account for the structural heterogeneity present in true rock masses. The reason for this is twofold: (1) it is impossible using current site investigation technologies to identify all structures in a mining environment that may contribute to rock mass behaviour, and (2) current computational power limits the size of model elements to metres or tens of metres, thus limiting the size of structures that can be
accurately represented in a given simulation. Because of this, for a geomechanical model to be useful, it must be able to simplify complex geometries in a way that still allows for accurate results to be produced; otherwise, results will not be reliable. This is particularly problematic when modelling fault slip problems because small changes in geometry will have large effects on the spatial and temporal nature of slip and seismicity (McKinnon, 2006), thus drastically changing how mine designers might approach the mitigation of such a risk. The trial conducted by Board et al. (1992) presents a prime example of this problem in the context of hydraulic stimulation—a numerical model was run to simulate a known fault structure that was to be subjected to hydraulic stimulation treatment in an effort to pre-trigger slip prior to mining. Though the model showed that a large slip response could be expected, the field trial only yielded micro-scale movements (with magnitudes $M < 0$), due to the complex nature of surrounding joint structures that were not explicitly incorporated into the model.

A number of different approaches have been taken when simulating fault slip behaviour and the effects of fluid injection, but core to the accuracy of all modelling results is the choice of fault properties, including, primarily, fault geometry and strength. Most authors (e.g., Ryder, 1987; McGarr, 1999) agree that a drop in shear strength must accompany the initiation of slip in order for strain energy to be released seismically. Otherwise, sliding would be gradual, continuous and aseismic, as no excess shear stress would be permitted to build up and drive episodic slip behaviour. Various mechanisms have been proposed as the cause of this strength drop. For persistent regional-scale fault structures associated with natural earthquakes, strength drop is dependent on changes in velocity and the state of the fault surface during slip (Scholz, 1998). Such effects, termed rate- and state-dependent slip behaviour, are normally attributed to
microscale changes in pore pressure, temperature, and the population of surface contacts that override each other during slip (Brace & Martin, 1968; Dieterich, 1978). However, along smaller, less persistent faults on which excavation-induced slip can occur, this drop in strength is more strongly associated with the breakage of intact portions of rock and other permanent changes in the geometry of the fault that occur as slip. Indeed, Cundall & Lemos (1990) note that rock mass responses modelled using rate-and state-weakening behaviour did not resemble true rockburst responses as closely as displacement weakening responses designed to simulate intact rock breakage.

Various methods have been proposed to appropriately account for the permanent strength degradation that accompanies fault slip. Cundall & Lemos (1990) introduced the continuously-yielding strength criterion, in which the friction angle of the fault surface transitions from a peak to residual value as the displacement occurs on the fault. They use this criterion to successfully simulate dynamic seismic events triggered by mine excavation. Sainoki & Mitri (2014) propose a similar method, whereby joint roughness is incorporated as a component of the friction angle via the Barton-Bandis strength criterion (Barton, 1973). The friction angle is then reduced to a residual value at the onset of slip on the structure. In perhaps a simpler method, appropriate when the values defining the fault roughness or strength reduction profile may not be known, both Zangeneh et al. (2013) and Preisig et al. (2015) incorporate the Coulomb slip criterion into their numerical models of injection-induced fault slip, but assign reduced residual friction angle and residual cohesion values that apply after the fault has slipped. The reduction in friction angle is analogous to the smoothing of undulations and asperities as the fault surfaces override each other, and the reduction in cohesion simulates the breakage of intact portions of rock that occur
along the fault path during slip. When integrated into a flow-coupled distinct element model, this choice of strength criterion was successful in simulating injection-triggered slip in both cases.

Ultimately the role of numerical modelling in the study presented here is to simulate whether hydraulic stimulation can reduce the frequency and severity of mining-induced fault slip events under a variety of fault strength and geometry conditions, which are represented using a reliable set of inputs. To this end, faults in the current study have been modelled in two different ways. In the first set of analyses, faults are represented as persistent, planar, homogeneous structures where the strength is modelled implicitly via the residual Coulomb slip criterion. In the second set of analyses, the faults are represented as heterogeneous structures consisting of planar joints separated by intact rock bridges of various sizes. In each case, the effectiveness of an applied hydraulic stimulation treatment is evaluated.

3.2 Methodology

3.2.1 Software

Modelling was conducted using the two-dimensional (2D) distinct-element modelling software UDEC (Itasca Consulting Group, 2010), a commercially available code that computes the displacement response of a jointed material subject to load. UDEC is particularly well-suited to modelling the behaviour of rock, where stress and displacement are predominantly governed by the presence of a network of discrete fractures. Rock masses are represented as a series of deformable blocks separated by through-going discontinuities. Though the material within individual blocks behaves as an elastic continuum, distinct element models are unique from other codes in that full dislocation between blocks is permitted through block contact relationships that
allow for both the opening/closing of discontinuities, and for discontinuity shear. An additional capability of UDEC is that it can simulate fluid flow through joints via a coupled hydro-mechanical algorithm. A transient compressible fluid model is used, which relies on the following relationship to simulate fluid flow between adjacent domains (defined by element boundaries in the model):

\[ q = -\frac{a^3 \Delta p}{12\mu l} \]  

(3.1)

where \( q \) is the flowrate, \( a \) is the aperture of the joint, \( \mu \) is the dynamic viscosity of the fluid, \( \Delta p \) is the pressure difference between adjacent domains, and \( l \) is the length of the domain. The ability of UDEC to simulate the mechanical response to fluid flow makes it an ideal tool for modelling the effects of hydraulic injection into a fractured rockmass, for which it has been used successfully in a number of documented cases (e.g., Choi, 2012; Zangeneh et al., 2013; Preisig et al., 2015).

### 3.2.2 Model setup

The UDEC model simulates a hypothetical underground mining operation located adjacent to a fault on which slip has the potential to occur. First, a control case is conducted where the full mine sequence is excavated without performing hydraulic stimulation, and the fault displacement and resulting slip magnitudes are recorded. Results are then compared to the same sequence of excavation after a series of hydraulic stimulation pre-treatments are conducted. This sequence is then repeated for six different scenarios where the fault strength and geometry are varied.
The geometry of the model is presented in Figure 3.1. The full excavation is 75 m wide by 105 m high and is located a minimum of 5 m away from a reverse fault dipping 45 degrees. The orebody is mined via a series of 15 by 15 m stopes using a method that resembles longitudinal open stoping: stopes are excavated right to left and top to bottom in the order shown. After excavation, stopes are backfilled prior to excavation of the next stope in sequence.

Figure 3.1: Cross section of UDEC model showing geometry. Block boundaries are shown in pink, and numbered stopes indicate the order of excavation.

Note that since this is a 2D analysis, stopes extend infinitely out of plane. This is a reasonable approximation for a tabular orebody in which the excavation is significantly longer than it is
wide. However, it is unrealistic to assume that each stope will be excavated in a single stage. More likely, stopes will be excavated through a series of blasting rounds conducted out of plane. To simulate this effect, the excavation of each stope is modelled via a series of five internal pressure reductions. After each pressure reduction, the stope is allowed to come to equilibrium.

Model boundaries must be set sufficiently far from the excavation area to ensure that the boundary conditions do not affect the model response. A model size of 425 m wide by 345 m high was deemed sufficient for this. Because displacement boundaries will arrest slip at the points where the fault intersects the edge of the model, stress boundary conditions are used instead, and only the corners of the model are pinned. When creating fractures in the model, care was taken to select appropriately-sized blocks. Because modelling each rock fracture discretely would lead to unreasonably large runtimes, a representative fracture network is needed to simulate the bulk effect of the rock mass. To create this network, a balance needed to be struck between making blocks small enough that they could simulate field-scale behaviour realistically, but not so small that the model would fail to compute in a reasonable amount of time. Ultimately a square gradational block size was chosen with smaller blocks adjacent to the fault and excavation areas, and larger blocks in the surrounding regions. Block sizes for each region are presented in Table 3.1. Note that a test simulation was conducted using 2.5 m blocks in the fault zone instead of 5 m blocks, however results closely resemble those produced using the larger blocks. Hence, for computational efficiency, 5 m blocks were deemed sufficient for simulations going forward. Results of the test simulations are included in Appendix E.
Table 3.1: Edge length of discrete blocks found within each zone of the UDEC mine model

<table>
<thead>
<tr>
<th>Region</th>
<th>Block edge length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fault zone</td>
<td>5</td>
</tr>
<tr>
<td>Excavation zone</td>
<td>7.5</td>
</tr>
<tr>
<td>Surrounding rock mass</td>
<td>15</td>
</tr>
<tr>
<td>Far-Field rock mass</td>
<td>40</td>
</tr>
</tbody>
</table>

Hydraulic stimulation is conducted as a series of treatments occurring after the excavation of each level, starting with the third level from the top as no slip events were observed to occur prior to this point. It is hypothesized that each round of hydraulic stimulation should create a stress shadow around the fault adjacent to the injection area (similar to the effect observed by Preisig et al. (2015)), thereby allowing excavation of the following level to occur without triggering significant seismic activity. Injection aligns with the midpoint of each level at a point 12.5 m above the fault as shown in Figure 3.1. Injection is conducted at a rate of 6.7 L/s, which aligns with pumping capacities deemed to be achievable in underground operations (Kaiser et al., 2013). However, it is worth noting that due to the 2D nature of the simulation, the injection source “point” represented in cross-section actually behaves as a linear injection front extending out of the 2D plane of the model. Injection rates, which are modelled per metre out of plane, must therefore be factored to better approximate the behaviour of a true injection point source in three-dimensional (3D) space. Values assumed for this factor vary as it is dependent on specific model geometry; for the model presented in Preisig et al. (2015) a reduction factor of 1/70 was assumed. Initial model runs from the current analysis suggest that the portion of the fault surface affected by the fluid propagation front is approximately 90 m in diameter. Hence, a reduction factor of 1/90 was applied to the modelled flow rate to produce a more realistic injection volume over this area. The duration of injection was 20 minutes for each pre-treatment phase, which
corresponds with typical injection durations used for hydraulic fracture treatments in cave mining (Kaiser et al., 2013). Since it is assumed that the mine is fully dewatered, the injected fluid is left to drain after the pre-treatment round is completed, prior to the excavation of the next level of stopes in sequence. Finally, in order to track the behaviour of the fault through the duration of the simulation, a series of monitoring points are placed every 7 m along the fault, aligning with the corners of the 5 m blocks modeled in the fault zone.

### 3.2.3 Input parameters

Rock mass parameters reflect those of a typical good quality granite or granodiorite host rock, and are presented in Table 3.2. Note that elastic properties are assumed, such that any inelastic deformation is localized to the fault contacts in the model. Block contacts, other than those that comprise the fault, are modelled as construction joints with high strength and stiffness values that resist shear movement. This prevents the model from producing secondary fragmentation as a result of fault slip and excavation, which is outside of the current scope of work. Fault strength properties are modelled using the Coulomb slip criterion (Equation 1.1); the difference between peak and residual friction angles in the base case model is designed to implicitly represent the strength loss that occurs as the irregularities on the fault surface are sheared during slip. Note that fault strength values in the base case model are chosen to represent equivalent average strengths for an embryonic fault splay, which will possess significant roughness and lack any considerable gouge coating; as such, fault strength values are higher than those typically assumed for fully developed natural fault zones where surface roughness is largely smoothed down to produce a gouge layer. Fluid properties assume water is used as the injection fluid, and joint and fault apertures are based on values obtained from laboratory fracture flow tests.
(Ishibashi et al., 2017). Backfill properties are designed to represent cemented rock fill material. Field stresses are modelled based on typical depths and stress ratios encountered in deep underground mines, in which the major and intermediate principal stresses are often horizontal, and the minor principal stress is vertical and corresponds to the weight of overburden (McKinnon, 2006).

### Table 3.2: Input properties from the base case UDEC mine model

| Rock mass properties |  |  |  |
|----------------------|------------------|------------------|
| Rock mass Young’s modulus, $E_{\text{rock}}$ | 50 GPa | Rock mass density, $\rho_{\text{rock}}$ | 2700 kg/m$^3$ |
| Rock mass Poisson’s ratio, $\nu_{\text{rock}}$ | 0.26 |  |  |

<table>
<thead>
<tr>
<th>Backfill properties</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Backfill Young’s modulus, $E_{\text{backfill}}$</td>
<td>1.1 GPa</td>
<td>Backfill density, $\rho_{\text{backfill}}$</td>
</tr>
<tr>
<td>Backfill Poisson’s ratio, $\nu_{\text{backfill}}$</td>
<td>0.06</td>
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<table>
<thead>
<tr>
<th>Fault properties</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Fault peak friction, $\phi_p$</td>
<td>40°</td>
<td>Fault normal stiffness</td>
</tr>
<tr>
<td>Fault residual friction, $\phi_r$</td>
<td>30°</td>
<td>Fault shear stiffness</td>
</tr>
<tr>
<td>Fault cohesion, $c$</td>
<td>0 MPa</td>
<td>Fault dilation</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Fluid flow properties</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Fluid density, $\rho_{\text{fluid}}$</td>
<td>1000 kg/m$^3$</td>
<td>Joint and fault residual aperture</td>
</tr>
<tr>
<td>Fluid viscosity</td>
<td>$10^{-3}$ Pa.s</td>
<td>Joint and fault aperture at zero normal stress</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Field stress properties</th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Vertical stress, $\sigma_v$</td>
<td>$\rho_{\text{rock}} \times g \times \text{depth}$</td>
<td>Depth to top of excavation</td>
</tr>
<tr>
<td>Horizontal in-plane stress, $\sigma_{h1}$</td>
<td>2 $\sigma_v$</td>
<td>Gravitational acceleration, $g$</td>
</tr>
<tr>
<td>Horizontal out-of-plane stress, $\sigma_{h2}$</td>
<td>1.5 $\sigma_v$</td>
<td></td>
</tr>
</tbody>
</table>

#### 3.2.3.1 Fault strength and geometry variations

A sensitivity analysis was performed by altering the base case parameters in Table 3.2 to explore the effect of changes in fault strength and geometry on results. Including the base case, six scenarios were run with properties presented in Figure 3.2. The first three scenarios, termed the “low strength”, “medium strength” and “high strength” fault models, represent the fault as a homogeneous structure that is persistent across the length of the model. Fault strength is altered
by changing the peak friction angle that applies to the fault surface. This has the effect of changing the magnitude of the drop from peak (static) to sliding (dynamic) strength, applicable to before and after slip occurs. This method of modelling simplifies the true fault geometry significantly, in that it assumes that surface roughness and other geometric irregularities can be represented implicitly by assigning appropriate peak and residual friction angles to the model surface. Because of their simplicity, homogeneous fault models are typically favoured over more complex methods when modelling fault behaviour. However, as demonstrated by Hammah et al. (2008) and Havaej et al. (2012), including geometric irregularities like surface waviness or rock bridging explicitly can produce significantly different behaviour from what homogenous models suggest. To test this effect in the current work, the remaining three scenarios, termed the “small rock bridge”, “medium rock bridge” and “large rock bridge” models, represent rock bridges explicitly by modelling the fault as discrete joints separated by intact pieces of rock. A constant friction angle is assigned to jointed segments, whereas intact segments are assigned peak friction angle and cohesion values, which revert to residual values when slip occurs and the intact segment is broken. Three different rock bridge sizes are tested, as shown in Figure 3.2. Note that on both the rock bridge block contacts in the heterogeneous model and the fault contacts in the homogeneous model, fluid flow is only permitted once the segment has broken or slipped. This is meant to simulate how intact rock bridges act as geometric barriers to flow until they are broken, however it should be noted that the 2D nature of the model may present a limitation in this respect, in that a true 3D rock mass may contain interconnected joints out of plane that permit flow around such barriers, depending on their size and relative persistence.
Figure 3.2: Schematic diagrams of the six UDEC models being tested, showing fault strength properties and geometry for each case.
3.2.4 Model execution

To appropriately simulate resultant stresses and displacements, static equilibrium must be achieved after each period of excavation, stress reduction, backfilling, and injection. UDEC uses a built-in “solve” algorithm to do this, which incrementally cycles the model until either the average unbalanced force is below a tolerance defined by the “solve ratio”, or when a limiting number of cycles is reached. To ensure that model is not prevented from reaching equilibrium prematurely by the cycle limit, the value of this parameter was increased from the default by a factor of 100. Additionally, to test whether the unbalanced force tolerance was appropriate, two different solve ratios were tested during initial model validation runs. Since both solve ratios yielded nearly identical results, the larger of the two tolerances was chosen for all future models for computational efficiency. Details of the UDEC model validation runs can be found in Appendix E, and an example of the code used to build and execute the models is included in Appendix F.

3.2.5 Data analysis algorithm

After execution, the UDEC models produced a series of raw stress and displacement time histories, which were then processed using a unique algorithm to derive metrics including stress change and the magnitude of seismicity. Inherent to this algorithm is the ability to extract discrete seismic events from the model, which, to the author’s knowledge, has not been attempted in UDEC mining simulation models created by others. The first processing step must be to identify periods of slip as discrete events. To do this, we assume that each period of seismicity corresponds to a single period of stress redistribution during one excavation cycle.
Though retrogressive failure has the potential to cause aftershocks that occur some duration of time after the causative stress redistribution (Snelling et al., 2013), case studies indicate that there is still a strong temporal correlation between fault slip events and excavation activity (Hart et al., 1988; Williams et al., 1992); hence, this assumption is considered appropriate. Because each stope is excavated in a series of 5 internal pressure reductions and there are 25 stopes in total in the model, there exists 125 discrete stress redistribution opportunities for slip to occur in the model.

Stress and displacement history data are available for each period of stress reduction at each history point. The analysis algorithm imports these files into MATLAB and automatically aggregates and processes data using code presented in Appendix F. First, for each excavation step, the algorithm determines the: i) net shear displacement, ii) drop from peak to final shear stress, and iii) time that slip began at each history point. This information is indexed in a single file for each event based on the spatial location of each history point, and is then used to compute data for all events triggered from the full excavation sequence. For each excavation stage, the algorithm tests whether fault slip has occurred. This is determined based on whether the shear displacement at any point on the fault exceeds an assumed tolerance of 0.5 mm. Then, for each slip event the following is calculated and indexed in a single file by event number:

1. The length of the slipping segment;
2. The average shear displacement occurring over the entire slip segment;
3. The average stress drop occurring over the entire slip segment;
4. The time at which slip was initiated; and
5. The location where slip was initiated.

These data can then be used to calculate the size of each seismic event, as measured via the seismic moment, $M_0$, and moment magnitude, $M_w$, parameters. These are calculated using the following relations (Hanks & Kanamori, 1979; Kanamori, 1994):

$$M_0 = \mu A \bar{D}$$

(3.2)

$$M_w = \frac{2}{3} \log M_0 - 6$$

(3.3)

where $\mu$ is the shear modulus of the rock mass surrounding the fault, $A$ is the area of the slipping surface, and $\bar{D}$ is the average shear displacement experienced on the fault during slip. The units of $M_0$ are Newton-metres. The shear modulus is derived from the Young’s modulus and Poisson’s ratio values presented in Table 3.2. To calculate area, it is assumed that the fault slips over a square surface with a length out of plane that is equal to the length in plane. Though the dimensions of the slip area on a given fault are variable and dependent on fault geometry and scale, this aspect ratio is considered generally appropriate for faults with widths less than that of the seismogenic crustal zone (Hanks & Bakun, 2002).

### 3.3 Results

#### 3.3.1 Base case

The purpose of the base case model is to determine the difference in fault slip behaviour observed when the mine sequence is excavated without hydraulic stimulation, compared to the same excavation sequence when hydraulic stimulation treatment is performed. Note that the base
case refers to the scenario where the fault is represented as a homogeneous structure with medium strength properties (Figure 3.2).

### 3.3.1.1 Model response to excavation

Figure 3.3 shows the differential stress profile that results from excavation of the full mine sequence without the use of hydraulic stimulation treatment, where differential stress is defined as the difference between the major and minor principal stresses acting at a given point in the subsurface. Differential stress can be observed to concentrate in the walls surrounding the excavation, and as the sequential excavation progresses, this stress is gradually transferred to the rock mass surrounding the fault. In adversely-oriented areas, particularly the area directly below the excavation, this results in a reduction in the normal stress clamping the fault, coupled with an increase in the shear stress driving movement on the fault. Once this stress combination exceeds the fault strength, as defined by the Coulomb slip criterion presented in Equation 1.1, the fault will begin to slip in the direction parallel to the greatest driving stress. This effect can be observed in Figure 3.4, which presents the total shear displacement accrued on the fault from the various periods of slip triggered during sequential excavation. Displacement is observed directly below the mining region, and is prevented from progressing outside of this area due to the additional clamping stress present distal to the excavation.
Figure 3.3: Differential stress profile generated in the area surrounding the mine and fault following full excavation of the mining sequence when no injection treatments are performed a priori.
Figure 3.4: Total shear displacement accrued on the fault following full excavation of the mining sequence when no injection treatments are performed a priori.

It is important to recognize that the total fault displacement represented in Figure 3.4 does not occur as a single slip event, but rather as a series of discrete slip events in response to each stage of mine excavation. Using the algorithm presented in Section 3.2.5, the magnitude and distribution of these events on the fault can be determined and is shown in Figure 3.5. In total, 56 slip events were observed, with a maximum moment magnitude of 2.7. This aligns well with the range of documented seismic magnitudes from mining-induced fault slip events (Blake & Hedley, 2003). Displacement history records indicate that the first period of slip coincides with the removal of stope 11 at the beginning of the third level of excavation. Note that the slip
initiation time is naturally specific to the geometry, regional geology, strength conditions and stress profile under consideration in the current model and will vary widely for different mine models and excavation sequences.

![Graph showing distribution of fault slip event hypocentres](image)

**Figure 3.5**: Distribution of fault slip event hypocentres observed during full excavation of the mining sequence when no injection treatments are performed a priori. The colour and size of each point correspond to the moment magnitude of the event.

The clustering of seismic events in Figure 3.5 shows quite clearly that the event hypocentres align with both the region of normal stress reduction (i.e., unclamping) and the region of active shear displacement on the fault. The largest events are observed to occur in the centre of the slipping region, with progressively smaller seismic events located nearer to the edges. This could
perhaps indicate how slip propagates in stages as the active portion of the fault grows; large events correspond to the initial reduction in strength triggered as the fault first begins to slip, and smaller events occur progressively as the shear zone gradually expands towards the outer extent of the mining region and abutment stresses interacting with the fault.

3.3.1.2 Model response to injection

To determine the effect of hydraulic stimulation on model behaviour, the excavation sequence presented in Section 3.3.1.1 is repeated with fluid injection conducted prior to each level of stoping, beginning with the third level as this was the first instance that fault slip was observed in the previous model run. Figure 3.6 shows the representative fluid pressure profile that occurs just prior to shut in, or the point at which injection stops and the excess fluid pressure is allowed to drain (note that the second period of injection is shown in this figure). The injection point is offset slightly from the fault, and the injected fluid permeates through the fracture network in the subsurface until it reaches the fault and surrounding area. The injected fluid travels farthest in the direction perpendicular to the minor principal stress, $\sigma_3$, as there is less clamping stress to overcome on fractures in this orientation, therefore fracture apertures will be wider. This is similar to the behaviour observed by Preisig et al. (2015), and aligns with the crack opening theory presented by Zoback et al. (1977). In this case, because the fault is assigned the same fluid flow parameters as the surrounding fracture network, the fault is not observed to act as a preferential flow path. In reality, differences in permeability resulting from shear-induced dilation may have an effect on the fluid flow pattern (Ishibashi et al., 2017), however this effect is not simulated in the current model.
Figure 3.6: Representative fluid pressure profile present after the second period of injection treatment, prior to the model being allowed to drain. Fluid mainly permeates along block contact boundaries in a direction perpendicular to $\sigma_3$.

In Figure 3.6, the excess fluid pressure generated by injection is, predictably, highest near the injection point, and dissipates as the distance from the injection point increases. It is interesting to note that the region of increased fluid pressure does not extend the full length of the fault, but rather it permeates along the fault for approximately 25 m. Given that it is this fluid pressure increase that is responsible for triggering stress release prior to mining, Figure 3.6 would suggest that this particular round of injection treatment will be effective for slipping regions within the central areas of the mine boundaries. This suspicion is confirmed by Figure 3.7, which presents the change in differential stress in the rock mass from the point in time just prior to injection, to
that immediately following shut in, after the excess fluid pressure has drained. As shown, the region through which the injection fluid front permeated experienced a differential stress drop ranging from 0-48 MPa. This effectively results in a stress shadow forming along the fault adjacent to the point of injection, with the slip-released stress being shed to surrounding areas further along the fault, which in turn experience a differential stress increase.

Figure 3.7: Change in rock mass differential stress in the base case fault model from the point in time prior to the second injection period, to immediately after the model is permitted to drain. Stress is reduced in the area adjacent to the injection point and shed to surrounding regions on the fault.
How does the stress response to injection impact the distribution of slip events triggered by mining? Figure 3.8 shows the cumulative number of slip events occurring at a given extraction ratio, defined as the ratio of mined out areas to the total area to be mined. As shown, the cumulative event count is significantly reduced when hydraulic stimulation treatment is used; 41 total slip events occur when stimulation is performed compared to 56 events when no treatment is performed. This indicates that injection has a demonstrable effect on reducing the occurrence of damaging periods of slip. However, though the total number of events is observed to decrease, slip is shown to occur earlier in the mining sequence when injection treatment is used (just after the first round of injection is conducted) which corresponds to a lower extraction ratio (41% in the hydraulic stimulation case, compared to 43% in the case where no treatment was performed). This is linked to the fact that rather than eliminating the accumulated stress, the injection treatment caused a transfer of stress to surrounding outer areas, thus increasing the driving stress on these peripheral areas of the fault. In turn, this could potentially cause segments of the fault to slip earlier than they would in the absence of the injection pre-conditioning treatment being performed. It should also be noted that the specific injection treatment used in this simulation was not successful at eliminating fault slip completely. It appears that treatments would need to target several areas of the fault instead of one to achieve this result, due to the stress concentrations that were observed to remain after the current injection sequence was performed.
Figure 3.8: Cumulative number of slip events triggered at a given extraction ratio in the mining sequence simulated for the base case mine geometry, with and without the use of hydraulic stimulation treatment.

Cumulative frequency plots are commonly used to analyze seismic event distributions, and have been applied to the mining induced slip events generated from the base case mining scenario (Figure 3.9). Several interesting observations can be gleaned from this plot. In addition to the total number of slip events being reduced following the hydraulic stimulation treatment (the vertical intercept in Figure 3.9a), the maximum moment magnitude of events associated with the mine sequence in which no treatment was performed (the horizontal intercept in Figure 3.9a) is notably higher than the hydraulically stimulated case ($M_w = 2.73$ when no stimulation is performed compared to $M_w = 2.46$ with stimulation treatment). This indicates that the hydraulic stimulation is capable of reducing both the magnitude and total number of slip events triggered.
during excavation. This likely occurs via a form of pre-triggering slip, as the injection is responsible for breaking the initial bonds preventing fault movement, including asperities and rock bridges. The permanent reduction in fault strength associated with this breakage is likely associated with large strain releases, and consequently the large shear stress drops and moment magnitudes observed in the untreated mining scenario are not present when hydraulic simulation is performed.

Figure 3.9: Cumulative frequency distributions for a) moment magnitude and b) shear stress drop of seismic events triggered during excavation of the base case mine geometry with and without hydraulic stimulation treatment.
It is worth noting that most natural fault zones display a linear relationship through higher magnitude portions of the cumulative frequency magnitude curve when viewed in logarithmic space. The negative slope of the linear portion of this curve, termed the “b-value”, is often used as a measure of the relative number of low vs. high magnitude events, and is a common metric considered when analyzing the seismic hazard potential of natural fault zones (El-Isa & Eaton, 2014). The global average b-value calculated for natural fault zones is 1.02, though b-values of between 0.3-2.5 have been reported (El-Isa & Eaton, 2014). Similarly, b-values associated with mining-induced slip events have a reported range of 0.6-1.4 (Urbancic et al., 1992). B-values derived from the current simulation are 1.7 for the non-treated case, and 1.2 when hydraulic stimulation is used during excavation. The drop in b-value when hydraulic stimulation is performed indicates that a greater proportion of high vs. low magnitude events are observed in the case where hydraulic stimulation treatment is performed compared to when no treatment is used. This suggests that hydraulic stimulation may be more effective at preventing lower magnitude events from occurring, compared to higher magnitude events. Such behaviour may be linked to the extent that fluid is able to permeate along the fault zone; as shown in Figure 3.6, since injected fluid in the current simulation only travels partway along the fault, it may trigger the release of driving stresses in this limited region, but will not be effective at mitigating slip events that rupture the entire length of the fault, which would naturally produce the largest slip events (see Equation 3.2).

It is also worth noting that in the lower end of the magnitude spectrum, frequency magnitude relationships tend to deviate from the linear trend discussed previously. In natural systems, this is
due to measurement limitations, in that small events will be outside of the detection limit of many recording instruments (Christensen et al., 2002). In the current numerical simulation a similar effect is observed due to the discretization resolution of the model; since the fault contact is divided into 7 m discrete segments for calculation purposes, detection of slip is only possible in intervals of this length. Slips occurring for smaller lengths than this will not be recorded.

In Figure 3.9b, injection treatment is observed to result in an overall reduction of both the total number of seismic events, and the maximum stress drop triggered by mining activities. Stress drop is considered to scale with moment magnitude, such that higher magnitude events will generally be associated with greater drops in stress during slip (Zoback & Gorelick, 2012). The fact that both methods of measuring slip event size—magnitude and stress drop—yield similar responses to injection treatment increases confidence in the findings.

3.3.2 The effect of fault geometry on slip event size and distribution

Recall that two methods of simulating fault strength and geometry were tested in this study (Figure 3.2): in three of the cases, the fault was modelled as a persistent structure with equivalent homogeneous strength properties (as previously presented), and in the remaining three cases, the fault was modelled as a series of joints separated by intact rock bridges where the overall strength of the fault was altered by changing the rock bridge lengths. The results of these two sets of models will be discussed separately.
3.3.2.1 Homogeneous fault properties

When the entire fault is assigned a homogeneous set of properties, the temporal distribution of slip events varies depending on the strength of the fault. This variation is present in Figure 3.10, where the total event count can be observed to increase as fault strength decreases. Slip onset is also shown to occur at lower extraction ratios when fault strengths are lower. Given that the excavation sequence and geometry are identical in each of the three homogeneous fault cases, increasing the strength of the fault can be interpreted to delay slip until later in the mining sequence, when the concentration of shear driving stress on the fault is higher and the clamping stress is lower. In the case of the high strength fault, since the majority of the mine sequence has been excavated by the time the first period of slip occurs, fewer total slip events are observed. However, it should be noted that because greater excess shear stress concentrations are permitted to build up on the fault prior to slip, both the maximum moment magnitude and shear stress drop observed on the high strength fault are increased, as shown in Figure 3.11. Consequently, more stress and strain energy is released when slip ultimately does occur, and high magnitude seismicity results.
Figure 3.10: Cumulative number of slip events triggered at a given extraction ratio in the mining sequence simulated for homogeneous fault strength models, with and without the use of hydraulic stimulation treatment.
For all three fault strengths presented in Figure 3.11, hydraulic stimulation results in a decrease in the maximum magnitude and stress drop associated with slip events. On the low and medium strength fault, stimulation also reduces the total number of slip events observed, however, on the high strength fault, stimulation increases the total events observed. This effect is related to the timing of slip onset; when hydraulic stimulation treatment is conducted, slip commences during the excavation stage immediately following the first period of injection in all cases. Though the increased fluid pressure associated with stimulation creates a stress shadow in the region immediately surrounding the injected fluid front (Figure 3.7), the stress concentrations outside of

![Cumulative frequency distributions](image)

**Figure 3.11:** Cumulative frequency distributions for a) moment magnitude and b) shear stress drop of seismic events triggered during excavation of homogeneous fault strength models, with and without hydraulic stimulation treatment.
this region act to trigger slip in the far-field regions located farther away from the active injection area. In the example of the high strength fault that was not treated by hydraulic stimulation, the structure was able to resist movement until the later stages of excavation. When stimulation is performed, slip occurs much earlier since stress concentrations on specific areas of the fault are much higher. Hence, the total number of slip events is notably increased.

### 3.3.2.2 Heterogeneous fault properties

When the fault is modelled with heterogeneous properties, the intact rock bridges contribute far greater to the strength that resists shear movement than the pre-sheared joints, due to the high cohesive strength assigned to these elements. Hence, by altering the length of the rock bridges, the overall strength of the fault structure changes. Figure 3.12 shows that this change in strength produces notably different distributions of events. Similar to the trend observed in Figure 3.10, as the size of the rock bridges increases, resulting in an overall strength increase along the fault, the total number of slip events decreases. As observed in Figure 3.13a, the decrease in total events that accompanies the greater rock bridge size is associated with a higher maximum event magnitude. This once again suggests that the stronger portions of the fault are resisting movement until later in the excavation sequence when more driving stress and less resisting stress has developed on the fault, leading to a larger total release of strain energy and event magnitude. Figure 3.13b follows the same general trend for the small and medium-sized rock bridges, wherein the stronger structures are associated with higher shear stress drops. However the large rock bridge model, which does not experience a large maximum stress drop, follows a different pattern. It is possible that the absence of this stress drop is due to the large rock bridges
being so strong that they resist breakage, and therefore the large magnitude slip events associated with complete shearing of the fault regions adjacent to the excavation no longer occurs. Since the intact portions of rock bridge arrest the propagation of larger slips, the very large stress drops observed in the non-hydraulically stimulated case are prevented from occurring.

Figure 3.12: Cumulative number of slip events triggered at a given extraction ratio in the mining sequence simulated for heterogeneous fault strength models, with and without the use of hydraulic stimulation treatment.
Interestingly, Figure 3.12 and Figure 3.13 show that, whereas stimulation treatment performed on the fault with homogeneous properties resulted in a significant reduction in both the total number of slip events and the maximum moment magnitude (Figure 3.10 and Figure 3.11), in all three of the heterogeneous fault models we see a much less significant response, where there is little to no difference in the event count and magnitudes in the hydraulic stimulation vs. non-treatment cases (Table 3.3). In this case, the hydraulic stimulation treatment appears to be much less effective at mitigating slip events. This appears to be due to the intact segments arresting both fluid propagation and slip, because their strength is so high that it prevents breakage and
shear movement (recall that fluid can only flow when the intact block contacts in the model are broken). In particular, this appears to be an effect of modelling the rock bridge segments using cohesive strength. As per the Coulomb slip equation (Equation 1.1), injection reduces the effective normal stress clamping the fault, which has the effect of decreasing the frictional strength of the structure. However, cohesive strength is not affected by fluid pressure, therefore on segments where cohesive strength contributes significantly to the overall strength, injection is less effective at triggering stress and strain energy release. This effect can be observed in Figure 3.14, which shows the differential stress change on the fault containing medium-sized rock bridges from the point just before the second round of injection to afterwards. Compared to Figure 3.7 which shows the same results for the medium strength homogeneous model, the magnitude of differential stress release is significantly smaller in both magnitude and extent. The presence of the rock bridges appears to be concentrating the stress change in discrete areas surrounding the open joints, but not the rock bridges, as slip is prevented in these areas. We do not observe this effect in the homogeneous fault strength model since strength is averaged across the fault, causing the maximum strength that injection must overcome to initiate slip to be lower. This finding is both interesting and significant, in that it means that geometry can have a large effect on fault model behaviour, and oversimplification of strengths and geometry may be responsible for some homogeneous fault models being inaccurate.
Table 3.3: Summary of slip magnitudes, stress drops and event counts for all fault models, with and without hydraulic stimulation treatment.

<table>
<thead>
<tr>
<th>Model</th>
<th>Hydraulic stimulation treatment</th>
<th>Total slip event count</th>
<th>Maximum moment magnitude, $M_w$</th>
<th>Maximum shear stress drop, $\Delta\sigma$ (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Homogeneous fault, low strength</td>
<td>No treatment</td>
<td>59</td>
<td>2.73</td>
<td>11.1</td>
</tr>
<tr>
<td></td>
<td>Treatment</td>
<td>48</td>
<td>2.46</td>
<td>8.2</td>
</tr>
<tr>
<td>Homogeneous fault, medium strength</td>
<td>No treatment</td>
<td>56</td>
<td>2.73</td>
<td>16.5</td>
</tr>
<tr>
<td></td>
<td>Treatment</td>
<td>41</td>
<td>2.46</td>
<td>9.3</td>
</tr>
<tr>
<td>Homogeneous fault, high strength</td>
<td>No treatment</td>
<td>39</td>
<td>3.04</td>
<td>31.2</td>
</tr>
<tr>
<td></td>
<td>Treatment</td>
<td>43</td>
<td>2.46</td>
<td>10.8</td>
</tr>
<tr>
<td>Heterogeneous fault, small rock bridges</td>
<td>No treatment</td>
<td>57</td>
<td>2.61</td>
<td>11.1</td>
</tr>
<tr>
<td></td>
<td>Treatment</td>
<td>57</td>
<td>2.59</td>
<td>9.9</td>
</tr>
<tr>
<td>Heterogeneous fault, medium rock bridges</td>
<td>No treatment</td>
<td>35</td>
<td>2.85</td>
<td>46.0</td>
</tr>
<tr>
<td></td>
<td>Treatment</td>
<td>34</td>
<td>2.86</td>
<td>40.5</td>
</tr>
<tr>
<td>Heterogeneous fault, large rock bridges</td>
<td>No treatment</td>
<td>26</td>
<td>2.86</td>
<td>33.7</td>
</tr>
<tr>
<td></td>
<td>Treatment</td>
<td>23</td>
<td>2.81</td>
<td>41.2</td>
</tr>
</tbody>
</table>
3.4 Discussion

The models presented above illustrate how hydraulic stimulation results in a transfer of stress away from the area of injection to regions further afield. This suggests that the potential for slip in the injection area can be significantly reduced via this technique, in theory supporting the use of hydraulic stimulation as a fault slip mitigation measure in deep mines. However, there are a
number of associated findings from the modelling results that are critical to the effectiveness of such a treatment, which should be considered before any large-scale treatment is undertaken. First, given that the stress released from injection areas is shed to surrounding regions, treatment could unintentionally act to bring previously stable fault sections into stress states where slip can be triggered, resulting in additional unintended seismicity. The hazard of this seismicity will need to be weighed against the mitigating effects of treatment; for example, if areas can be correctly identified as posing high fault slip risk, they can be selectively targeted. This may allow for prevention of the largest, most damaging events (such as those that result in stope collapse) in exchange for a series of less damaging events that are pushed to the periphery of the mining region away from personnel and active operations. It is also worth noting that the injection sequence used in the current models is relatively imprecise, in that it targets entire mining levels measuring 75 m across, rather than specific regions adjacent to each excavation round. This is primarily due to modelling limitations, in that a more complex injection sequence would require 3D modelling and invariably lead to significantly longer model runtimes. However, through the sequence of an entire life of mine, which may last for decades, it is much more feasible to envision a complex pattern of injection occurring ahead of each mine advance, in an effort to ensure that excessive stress concentrations remain further afield from active mining areas. The issue with this level of selective planning is that it relies on a detailed knowledge of the location and properties of slip-prone structures in the rock mass. Currently the degree of uncertainty in this respect is very high in most existing mining projects; in many cases, historical slip events have occurred on structures that were not identified until after catastrophic slip events occurred (e.g., Williams et al., 1992).
The results of the current simulation also speak to the need to include fault geometry details explicitly in numerical models, given that geometry changes had a notable effect on the pattern of fluid propagation, stress release, and the ultimate evaluation of hydraulic stimulation effectiveness. Though averaging properties such as fault strength and cohesion implicitly across the modelled fault may give a first approximation of behaviour, there exist some key processes that cannot be simulated in this way, including the behaviour of overlapping asperities and intact rock segments that are stronger than surrounding regions of the fault zone, which may become locked and impede shear movement. Fault slip behaviour is shown to be heavily dependent on the geometry of the fault structure in question, which speaks to the need for better methods of mapping and detecting structural features in mining areas, and also the need for Discrete Fracture Network (DFN) tools that can simulate a range of possible subsurface geometries and calculate a more realistic range of behaviours.

Finally, before field scale hydraulic stimulation can be effectively implemented, the rate and duration of injection must be optimized. Injection parameters used in the current simulation were based on previous field experience in cave mine operations, where the injected fluid is used to induce hydraulic fracturing to improve fragmentation of ore zones. However, because hydraulic stimulation aims to trigger movement on pre-existing fractures, rather than the creation of new fractures as in hydraulic fracturing, it is possible that the chosen rates are not optimal for this particular application. Preliminary studies (Kaiser et al., 2013; Preisig et al., 2015) have indicated that typically, optimal rates to induce hydraulic shearing are lower and injection
durations are longer than those that trigger hydraulic fracturing. Specific injection parameters will likely depend on the effective permeability of the subsurface generated through the network of fractures present in the rock mass. This has been known to vary by orders of magnitude in some cases (Ishibashi et al., 2017), therefore an investigation of groundwater flow properties is equally important to an understanding of fault strength and geometry when selecting potential sites where hydraulic fracturing treatment may be successful.

3.5 Conclusions

A numerical investigation was undertaken to simulate a hypothetical mine excavation sequence in an area susceptible to fault slip, wherein a hydraulic stimulation treatment was implemented in an attempt to reduce the frequency and severity of fault slip events triggered by the mining sequence. To do this, a unique algorithm was developed to extract information about the slip events observed through the life of the mining sequence, including the event count, moment magnitude, stress drop, hypocentre location, and other metrics. Hydraulic stimulation treatment decreased both the total number of events (56 to 41) and the maximum magnitude of the events (from 2.73 to 2.46) triggered during mining. Observations of the differential stress changes within the rock mass during treatment indicate that the injected fluid acts to pre-trigger shear movement and strain energy release on the fault, resulting in a transfer of driving shear stress from the area adjacent to the injection point, to regions farther away from the area of interest. This acts to inhibit fault slip in the area surrounding the injection point when nearby stopes are ultimately mined, however it also increases the risk of slip on far-field areas of the fault where slip may not have otherwise occurred.
The geometry and strength of the fault, and the method by which these properties are modelled are observed to affect the model’s response to excavation and hydraulic stimulation; when the fault is represented as a planar structure with homogeneous strength, higher-strength faults produce fewer, larger fault slip events on the structure, whereas weaker faults experience smaller, more numerous slips when subject to the same excavation sequence. A similar response is observed when geometry elements such as rock bridges and asperities on the fault surface are modelled explicitly, however hydraulic injection treatments were much less effective at reducing fault slip hazard in this case. Rock bridges possess higher cohesive strength, and also impede the flow of injected fluid, such that stimulation treatments are not able to exceed the fault’s strength and trigger strain energy release as effectively. This is an important finding because it suggests that field-scale fault geometry will have a major impact on the effectiveness of the hydraulic stimulation treatment, and that numerical modelling strategies that oversimplify geometry may not be accurate when compared against field-scale behaviour. Ultimately this speaks to the need for a better understanding of the subsurface structural geology and fault geometry if hydraulic stimulation treatments are to be effective as a fault slip mitigation technique in deep underground mines.
Chapter 4 – Conclusions and Future Work

4.1 Summary of key findings and conclusions from the experimental investigation

Research carried out for the laboratory study involved the creation of a unique sample preparation and testing procedure designed to simulate the behaviour of mine-scale faults of various geometries undergoing slip triggered by excavation and fluid injection. By preparing samples with through-going saw cuts angled appropriately to the axes of loading and a fluid injection hole intersecting each cut, fault slip could be achieved and monitored via a load cell and strain transducers mounted to the exterior of the sample. After analyzing the results of the experiment, the following key observations were made:

- Loading and injection tests produced characteristic stick-slip behaviour, in which slip events with moment magnitudes of $M_0 = -4.3$ to $-3.8$ and shear stress drops of 0.004 to 26 MPa were observed. Using seismic scaling relationships, these values are within expected ranges given the size of the slip surface under consideration, helping to validate the experimental method developed.

- Both loading and injection-induced slip was accompanied by a reduction in shear stress within the sample. This confirms both that shear movement can generate a release of stress and strain energy driving fault slip events, and that injection can act as a pre-triggering mechanism for this movement and accompanying energy release.

- Heterogeneous fault surface geometries, consisting of small and large-offset saw cuts in the presented experiments, and homogeneous surfaces, consisting of smooth saw cuts, tended to produce differences in the moment magnitudes and shear stress releases.
associated with slip events. 3D imaging of post-test specimens demonstrated that this variation is linked to the breakdown of surface asperities and accompanying loss of cohesive strength that occurs during the shearing of heterogeneous fault surfaces, which is largely absent on smooth faults where surfaces are observed to override each other during shearing such that little asperity breakage occurs. A modified version of the Coulomb slip criterion is used to illustrate this concept and describe the link between fault geometry, strength loss, and stress release.

- Fluid injection is observed to trigger slip events with considerably lower magnitudes than events triggered by loading. If this proves to be a universal effect, it suggests that hydraulic stimulation could be effective at safely pre-triggering stress relief prior to mining an area.

- However, injection treatment was observed to be less effective on heterogeneous fault geometries, in which a maximum stress release of 0.7 MPa was observed, compared to smooth fault surfaces, where shear stress releases of up to 5.9 MPa could be achieved. Because injection acts to trigger slip by reducing the frictional strength component present on the fault surface, the heterogeneous fault geometries that possess greater cohesive strength relative to frictional strength are predicted to resist the slip and stress release associated with injection treatment. The field-scale application of this technology will therefore be highly dependent on subsurface fault geometry, and will need to be evaluated on a case-by-case basis prior to successful implementation of hydraulic stimulation as a risk mitigation technique.
Based on these observations, the following conclusions can be drawn relative to the thesis objectives:

1. An experimental method was successfully developed capable of simulating fault slip on laboratory-scale specimens. Results demonstrated that fundamental slip behaviours could be investigated using this technique in a controlled environment before implementing at the mine scale. The experimental methodology allows for a variety of fault geometries to be tested with respect to geometrical irregularities relative to the assumption of a perfectly planar surface.

2. The stress and strain responses associated with the fault slip experiments were successfully measured, allowing for the comparative analysis of the results without and with the use of fluid injection treatments.

3. The interpretation of the test results provided insights into the effects that the injection treatments have on releasing built-up stresses responsible for fault slip hazards. It was clearly demonstrated that treatments were capable of triggering slip and the associated release of stress and strain energy, however the effectiveness of treatment was highly dependent on the geometry and cohesive strength of the fault surface, where cohesive elements, such as asperities, were observed to impede injection-triggered slip. Such knowledge must ultimately be used to inform the careful application of injection treatments on the field scale if such applications are to be successful.
4.2 Summary of key findings from the numerical modelling investigation

The numerical investigation involved the development of a numerical procedure capable of simulating a mining sequence for which the redistribution of stresses related to excavation results in the critical loading of a nearby fault structure, triggering a fault slip rockburst event and associated seismicity. Such events are a major concern to the safety and integrity of deep mining operations. Various fault geometries were simulated and subjected to a hydraulic stimulation treatment, and a unique algorithm was developed to analyze the results of each treatment and determine its effectiveness at mitigating fault slip hazards through the life of the mining operation. The following key observations were made:

- Fault slip was observed to occur in each of the six fault models, beginning at the start of the third level of excavation (equivalent to an extraction ratio of 40%). When hydraulic stimulation was performed, a reduction in both the total number of slip events (56 to 41) and the maximum magnitude of the events (from $M_0 = 2.73$ to 2.46) was observed. This suggests that injection treatment has the potential to reduce the risks associated with mining in slip-prone areas.

- Observations of the differential stress changes within the rock mass during hydraulic stimulation treatment indicate that the injection-related changes to the effective stress field results in a transfer of driving shear stress away from the areas adjacent to the injection point and towards regions located further away from the area of interest. In this way, injection effectively creates a stress shadow in which strain energy is released through shear displacements occurring in areas of high fluid pressure. However, the near-
field relief of stress results in the transfer of stresses to the adjacent far-field areas, which may have the unintended consequence of triggering slip in regions that otherwise may not have presented a fault slip hazard.

- Fault strength was observed to affect the modelled response to excavation and injection. Higher strength faults, and those that contained larger intact rock bridge segments experienced larger, less frequent slip events; lower strength faults and those with smaller intact segments experienced more numerous slip events with lower moment magnitudes.

- The method by which fault structure was incorporated into the model was observed to have a significant effect on the effectiveness of the injection treatment indicated by the simulation. In heterogeneous fault models, where structural elements such as asperities and rock bridges are modelled implicitly by altering the friction and cohesion values for different segments along the fault surface, the model is observed to respond well to treatment, in that a significant reduction in the frequency and magnitude of slip events is observed. However, when rock bridges are modelled explicitly as segments of intact rock in between a disconnected joint network, the stress and strain energy release triggered by the stimulation treatment is significantly impeded. Since rock bridges possess a higher cohesive strength and may also act as a geometric barrier restricting the flow of injected fluid through the fault, stimulation is not able to overcome the high-strength portions of the fault structure and injection treatment is ineffective. This speaks to the need for a detailed understanding of the subsurface fault geometry in slip-susceptible areas, and the incorporation of this geometry explicitly into numerical models, such that field conditions are simulated accurately.
Based on these observations, the following conclusions can be drawn relative to the thesis objectives:

1. A field-scale fault slip and hydraulic injection scenario was successfully modelled using a 2D distinct-element approach. The developed procedure allows for the triggering of fault slip in response to the sequential excavation of a series of mine stopes.
2. The numerical procedure developed allows for different fault geometries and degrees of strength heterogeneity (i.e., rock bridges) to be tested with respect to their effect on the effectiveness of a planned injection treatment.
3. The model results suggest that hydraulic injection treatments have the potential to effectively reduce the magnitude and frequency of fault slip events observed throughout the mining sequence. However, given the sensitivity of treatment to both overall fault strength and the presence and distribution of rock bridges on the fault structure, the structural geology and other fault properties of potential field sites must be carefully evaluated to ensure that treatment will ultimately be effective.

4.3 Suggestions for future work

The primary contribution of the current work is a proof of concept of hydraulic stimulation as a fault slip risk mitigation technique, showing that this technology has the potential to be effective under a variety of subsurface conditions. However, the work presented here also highlights a number of key issues that must be considered prior to the effective implementation of hydraulic
stimulation in an operating mine environment. To this end, the following future work is recommended:

- Since the effectiveness of hydraulic stimulation treatments is shown to be highly dependent on fault geometry, laboratory tests should be performed on a wider range of potential fault geometries and loading conditions than those presented here. For example, testing conducted on incomplete saw cuts containing intact rock bridges could be used to identify whether fracture propagation through these segments significantly affects slip behaviour to the point where it differs from trends observed in the current investigation. An exploration of the effect of confinement on asperity shearing vs. overriding behaviour would also produce useful insights.

- The use of acoustic emission (AE) monitoring in fault slip laboratory experiments should be considered. AE monitoring is more sensitive than strain monitoring when used to detect micro-scale seismicity occurring over portions of the slip surface. In addition, when properly calibrated, AE monitoring can be used to delineate the location of micro-scale slips as a result of loading and injection on fault surfaces.

- To expand on the results of the simple geometry variations tested in the heterogeneous UDEC fault models, it is recommended that a series of Discrete Fracture Network (DFN) models be developed to test a wider range of potential fault geometries that could be encountered in deep mining areas, similar to the models presented in Rogers et al. (2010). With a sufficiently large dataset, statistical analysis of trends in results may significantly supplement the initial observations made in the current study.
Fluid propagation modelling should be performed to determine optimal injection rates and volumes used to trigger hydraulic shearing via stimulation treatment. 3D fluid propagation modelling could also be used to test the ability of intact rock bridges to impede fluid flow along a fault, which has been assumed in the current analysis.

Finally, provided that conclusive results can be gleaned from future laboratory and numerical investigations, hydraulic stimulation field trials should be considered for appropriately identified areas in which favourable fault properties, stress conditions, and mine geometry are present to support the effective application of this technology.
References


Appendices

Appendix A – Supplemental laboratory test results

The following section comprises data collected in support of the laboratory test results presented in Chapter 2. A total of 8 fault slip tests were conducted, and the following information is included from each test:

- Pre-test specimen photos;
- Pre-test specimen description;
- Pre-test specimen dimensions (length, diameter, hole diameter, saw cut offset) and mass;
- Specimen preparation descriptions;
- Test descriptions;
- Post-test specimen photos;
- Post-test specimen descriptions;
- Plots of fault shear displacement vs. time; and
- Pre- and post-test 3D fault surface scans.
<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree straight, polished saw cut (measured from axis of the cylinder).</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.7087</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>127.25</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>53.7</td>
</tr>
<tr>
<td><strong>Hole Diameter [mm]</strong></td>
<td>15.3</td>
</tr>
<tr>
<td>------------------------</td>
<td>------</td>
</tr>
<tr>
<td><strong>Saw Cut Offset [mm]</strong></td>
<td>None</td>
</tr>
<tr>
<td><strong>Specimen Prep</strong></td>
<td>Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw. The halves were smoothed by hand using 150-grit sandpaper, and then polished on a flat lap wheel using 600- to 950-grit polishing powder, followed by titanium dioxide finisher.</td>
</tr>
<tr>
<td><strong>Test Description</strong></td>
<td>25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min to a total shear displacement of 4 mm on the saw cut.</td>
</tr>
<tr>
<td><strong>Post-Test Photos</strong></td>
<td><img src="image" alt="Post-Test Photo" /></td>
</tr>
<tr>
<td>Post-Test Specimen Description</td>
<td>Visible shear displacement observed between two specimen halves. Fault surfaces appear similar to pre-test condition and free of gouge. Specimen halves remained intact besides a few very small (1-2 mm) chips on the edges of the saw cut.</td>
</tr>
<tr>
<td>--------------------------------</td>
<td>------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>Results Plots</td>
<td><img src="image" alt="Fault Shear Displacement vs. Time - Loading Phase" /></td>
</tr>
</tbody>
</table>

![Image of granite specimens](image)
Pre-Test Fault Surface 3D Scan

Pre-Test Surface Roughness - Polished Saw Cut

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Polished Saw Cut
## Table A.2  
**HF028 – Polished Saw Cut Injection Test**

### Pre-test Photos

Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree straight, polished saw cut (measured from axis of the cylinder).

<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree straight, polished saw cut (measured from axis of the cylinder).</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.6819</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>123.2</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>53.6</td>
</tr>
<tr>
<td>Hole Diameter [mm]</td>
<td>15.0</td>
</tr>
<tr>
<td>Saw Cut Offset [mm]</td>
<td>None</td>
</tr>
<tr>
<td>Specimen Prep</td>
<td>Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw. The halves were smoothed by hand using 150-grit sandpaper, and then polished on a flat lap wheel using 600- to 950-grit polishing powder, followed by titanium dioxide finisher.</td>
</tr>
</tbody>
</table>
25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min until 54 MPa axial stress was reached. Water pressure was then increased at an injection rate of 3 MPa/minute to a maximum pressure of 24 MPa.

| Test Description | 25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min until 54 MPa axial stress was reached. Water pressure was then increased at an injection rate of 3 MPa/minute to a maximum pressure of 24 MPa. |

| Post-Test Photos | ![Post-Test Photos](image_url) |
| Post-Test Specimen Description | Visible shear displacement observed between two specimen halves. Fault surfaces appear similar to pre-test condition and free of gouge. Specimen halves remained intact besides a few very small (1-2 mm) chips on the edges of the saw cut. |

| Results Plots | ![Fault Shear Displacement vs. Time - Loading and Injection Phases](image) |
Pre-Test Surface Roughness - Polished Saw Cut

X Coordinate (mm)

Y Coordinate (mm)

Surface Roughness (mm)
<p>| Pre-Test Specimen Description | Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree straight saw cut (measured from axis of the cylinder). |
| Mass [kg] | 0.6819 |
| Axial Length [mm] | 122.7 |
| Diameter [mm] | 53.6 |
| Hole Diameter [mm] | 15.0 |</p>
<table>
<thead>
<tr>
<th>Saw Cut Offset [mm]</th>
<th>None</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Specimen Preparation</strong></td>
<td>Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw. The halves were then smoothed by hand using 150-grit sandpaper and finished with 220-grit sandpaper. Specimen has a small chip top left and bottom right of one half of specimen during sawing, ~18mm long and 1-2 mm wide.</td>
</tr>
<tr>
<td><strong>Test Description</strong></td>
<td>25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min to a total shear displacement of 4 mm on the saw cut.</td>
</tr>
<tr>
<td><strong>Post-Test Photos</strong></td>
<td><img src="image1.jpg" alt="Image" /> <img src="image2.jpg" alt="Image" /></td>
</tr>
<tr>
<td><strong>Post-Test Specimen Description</strong></td>
<td>Visible shear displacement observed between two specimen halves. Fault surfaces appear similar to pre-test condition and free of gouge. Very faint scratches are present in some areas of the fault surface and are aligned with the direction of shear. Specimen halves remained intact besides a few very small (1-2 mm) chips on the edges of the saw cut and end surfaces.</td>
</tr>
<tr>
<td><strong>Results Plot</strong></td>
<td><img src="image" alt="Fault Shear Displacement vs. Time - Loading Phase" /></td>
</tr>
</tbody>
</table>
Pre-Test Fault Surface 3D Scan

Pre-Test Surface Roughness - Straight Saw Cut

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Straight Saw Cut

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree straight saw cut (measured from axis of the cylinder).</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.6955</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>128.6</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>54.22</td>
</tr>
<tr>
<td>Specimen Preparation</td>
<td>Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw. The halves were then smoothed by hand using 150-grit sandpaper and finished with 220-grit sandpaper.</td>
</tr>
<tr>
<td>Test Description</td>
<td>25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min until 54 MPa axial stress was reached. Water pressure was then increased at an injection rate of 3 MPa/minute to a maximum pressure of 24 MPa.</td>
</tr>
<tr>
<td><strong>Post-Test Specimen Description</strong></td>
<td>Visible shear displacement observed between two specimen halves. Fault surfaces appear similar to pre-test condition and free of gouge. Specimen halves remained intact besides a few 2-10 mm chips on the edges of the saw cut and end surfaces.</td>
</tr>
<tr>
<td>----------------------------------</td>
<td>--------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td><strong>Results Plot</strong></td>
<td><img src="image" alt="Fault Shear Displacement vs. Time - Loading and Injection Phases" /></td>
</tr>
<tr>
<td><strong>Pre-Test Fault Surface 3D Scan</strong></td>
<td>Scan unavailable. For a representative scan, refer to pre-test scan from HF032.</td>
</tr>
</tbody>
</table>
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Straight Saw Cut

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Table A.5  HF024 – Small Asperity Loading Test

<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree saw cut (measured from axis of the cylinder) with 1.5 mm asperity centred horizontally on the cut surface.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.6865</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>123.6</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>53.5</td>
</tr>
<tr>
<td><strong>Hole Diameter [mm]</strong></td>
<td>14.6</td>
</tr>
<tr>
<td>------------------------</td>
<td>------</td>
</tr>
<tr>
<td><strong>Saw Cut Offset [mm]</strong></td>
<td>1.5</td>
</tr>
<tr>
<td><strong>Specimen Preparation</strong></td>
<td>Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw, and then a 1.5 mm offset was filed down from the top half of each cut surface by hand using a diamond file. The cut surfaces were then smoothed by hand using 150-grit sandpaper, followed by 220-grit sandpaper. Some chipping around edge of pieces occurred during specimen prep. Small (~0.1 mm) surface irregularities present due to filing process.</td>
</tr>
<tr>
<td><strong>Test Description</strong></td>
<td>25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min to a total shear displacement of 4 mm on the saw cut.</td>
</tr>
<tr>
<td><strong>Post-Test Photos</strong></td>
<td><img src="image1" alt="Post-Test Photos" /> <img src="image2" alt="Post-Test Photos" /></td>
</tr>
<tr>
<td>Post-Test Specimen Description</td>
<td>Visible shear displacement and dilation observed between two specimen halves. Edges of the asperity have been mostly sheared off leaving a surface that is rougher than the original saw cut. Fault gouge surrounds the sheared-off area. An approx. 25 mm piece was chipped off the left side of one of the specimen halves, and several smaller pieces were chipped off the centre duct hole.</td>
</tr>
<tr>
<td>---</td>
<td>---</td>
</tr>
</tbody>
</table>

Results Plots

![Fault Shear Displacement vs. Time - Loading Phase](image_url)

**Fault Shear Displacement vs. Time - Loading Phase**

- **Shear Displacement (mm)**: 0 to 4.5
- **Time (s)**: 400 to 2400
Pre-Test Fault Surface 3D Scan

Pre-Test Surface Roughness - Small Asperity

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Small Asperity

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
### Pre-test Photos

![Granite specimen](image1)

![Granite specimen](image2)

<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree saw cut (measured from axis of the cylinder) with 1.5 mm asperity centred horizontally on the cut surface.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.7045</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>125.9</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>53.8</td>
</tr>
<tr>
<td>Hole Diameter [mm]</td>
<td>14.6</td>
</tr>
<tr>
<td>Specimen Preparation</td>
<td>Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw, and then a 1.5 mm offset was filed down from the top half of each cut surface by hand using a diamond file. The cut surfaces were then smoothed by hand using 150-grit sandpaper, followed by 220-grit sandpaper. Some chipping around edge of pieces occurred during specimen prep. Small (~0.1 mm) surface irregularities present due to filing process.</td>
</tr>
<tr>
<td>----------------------</td>
<td>--------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>Test Description</td>
<td>25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min until 54 MPa axial stress was reached. Water pressure was then increased at an injection rate of 3 MPa/minute to a maximum pressure of 24 MPa.</td>
</tr>
<tr>
<td>Post-Test Photos</td>
<td><img src="image1.png" alt="Post-Test Photos" /> <img src="image2.png" alt="Post-Test Photos" /></td>
</tr>
<tr>
<td>Post-Test Specimen Description</td>
<td>Visible shear displacement and dilation observed between two specimen halves. Edges of the asperity have been chipped but not fully sheared off. Fault gouge surrounds the sheared-off area. An approx. 50 mm piece was chipped off the top of one of the specimen halves.</td>
</tr>
<tr>
<td>-----------------------------</td>
<td>--------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>Results Plots</td>
<td><img src="image-url" alt="Fault Shear Displacement vs. Time - Loading and Injection Phases" /></td>
</tr>
</tbody>
</table>
Pre-Test Fault Surface 3D Scan

Pre-Test Surface Roughness - Small Asperity

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Small Asperity

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree saw cut (measured from axis of the cylinder) with 8 mm asperity centred horizontally on the cut surface.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.6480</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>117.5</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>54.0</td>
</tr>
<tr>
<td><strong>Hole Diameter [mm]</strong></td>
<td>14.9</td>
</tr>
<tr>
<td>------------------------</td>
<td>------</td>
</tr>
<tr>
<td><strong>Saw Cut Offset [mm]</strong></td>
<td>8</td>
</tr>
</tbody>
</table>

**Specimen Preparation**
Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw, and then an 8 mm offset was ground down from the top half of each cut surface using a diamond-tipped rotary tool. The cut surfaces were then smoothed by hand using 150-grit sandpaper, followed by 220-grit sandpaper. Some chipping around the edges occurred during specimen prep. ~1 mm surface irregularities present due to grinding process.

**Test Description**
25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min to a total shear displacement of 4 mm on the saw cut.

**Post-Test Photos**

- ![Image 1](image1.png)
- ![Image 2](image2.png)
| Post-Test Specimen Description | Significant breakage around both the asperity and top edges of each saw cut. The asperity has been sheared off along a series of newly-formed surfaces surrounding the centre of the original cut. Significant gouge and broken material is present at the previous site of the asperity and the underlying shear surface is rougher than the original saw cut. |
| Results Plots | ![Fault Shear Displacement vs. Time](image) |
Pre-Test Fault
Surface 3D Scan

Pre-Test Surface Roughness - Large Asperity

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Large Asperity
### Table A.8  
**HF031 – Large Asperity Injection Test**

**Pre-test Photos**

<table>
<thead>
<tr>
<th>Pre-Test Specimen Description</th>
<th>Granite specimen containing quartz, feldspar and biotite grains approx. 2-4mm in diameter. Contains 35 degree saw cut (measured from axis of the cylinder) with 8 mm asperity centred horizontally on the cut surface.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass [kg]</td>
<td>0.6114</td>
</tr>
<tr>
<td>Axial Length [mm]</td>
<td>113.8</td>
</tr>
<tr>
<td>Diameter [mm]</td>
<td>52.8</td>
</tr>
<tr>
<td><strong>Hole Diameter [mm]</strong></td>
<td>15.0</td>
</tr>
<tr>
<td>------------------------</td>
<td>------</td>
</tr>
<tr>
<td><strong>Saw Cut Offset [mm]</strong></td>
<td>8</td>
</tr>
</tbody>
</table>

**Specimen Preparation**
Specimen drilled from an intact granite block using a 54 mm bit. Hole drilled axially through specimen using a 15 mm drill bit. Specimen ends were then ground down until flat. Specimen was cut in half at a 35 degree angle using a rock saw, and then an 8 mm offset was ground down from the top half of each cut surface using a diamond-tipped rotary tool. The cut surfaces were then smoothed by hand using 150-grit sandpaper, followed by 220-grit sandpaper. Some chipping around the edges occurred during specimen prep. ~1 mm surface irregularities present due to grinding process.

**Test Description**
25 MPa confining pressure and 2 MPa deviator stress were applied to the specimen under drained conditions (water pressure held constant at 0 MPa). The specimen was then compressed by lowering the axial actuator at a displacement rate of 0.2 mm/min until 54 MPa axial stress was reached. Water pressure was then increased at an injection rate of 3 MPa/minute to a maximum pressure of 24 MPa.

**Post-Test Photos**

![Post-Test Photos](image-url)
<table>
<thead>
<tr>
<th><strong>Post-Test Specimen Description</strong></th>
<th>Large (1-5 cm) pieces chipped off the top of both specimen halves. Some very small (~1 mm) chips off the edge of the asperity but otherwise both asperity halves remained intact. Little to no gouge present on fault surfaces.</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Results Plots</strong></td>
<td><img src="image" alt="Fault Shear Displacement vs. Time - Loading and Injection Phases" /></td>
</tr>
</tbody>
</table>
Pre-Test Fault Surface 3D Scan

Pre-Test Surface Roughness - Large Asperity

Y Coordinate (mm)

X Coordinate (mm)

Surface Roughness (mm)
Post-Test Fault Surface 3D Scan

Post-Test Surface Roughness - Large Asperity

Y Coordinate (mm)

Surface Roughness (mm)

X Coordinate (mm)
Appendix B – Summary of intact rock property tests on Hardy Island granite

Rock property tests were performed on intact cylinders of Hardy Island Granite. Testing consisted of 13 specimen unit weight measurements, 4 uniaxial compressive strength tests, and 2 triaxial compression tests performed at confining pressures of 20 MPa and 25 MPa, respectively. Test results are summarized in Table B.1 and Table B.2, and Figure B.1 through Figure B.3. Stress-strain plots from individual tests are presented in Figure B.4 through Figure B.7. Note that Young’s modulus and Poisson’s ratio values were calculated using stress-strain tangent lines at 50% of the peak axial stress.

Table B.1: Summary of unit weight measurements performed on Hardy Island Granite.

<table>
<thead>
<tr>
<th>Sample ID</th>
<th>Unit weight (kN/m3)</th>
</tr>
</thead>
<tbody>
<tr>
<td>G001</td>
<td>26.8</td>
</tr>
<tr>
<td>G002</td>
<td>26.6</td>
</tr>
<tr>
<td>G003</td>
<td>26.6</td>
</tr>
<tr>
<td>G004</td>
<td>26.2</td>
</tr>
<tr>
<td>G005</td>
<td>26.4</td>
</tr>
<tr>
<td>G006</td>
<td>26.4</td>
</tr>
<tr>
<td>G006B</td>
<td>26.6</td>
</tr>
<tr>
<td>G007</td>
<td>26.4</td>
</tr>
<tr>
<td>G008</td>
<td>26.5</td>
</tr>
<tr>
<td>G009</td>
<td>26.5</td>
</tr>
<tr>
<td>G010</td>
<td>26.4</td>
</tr>
<tr>
<td>G037</td>
<td>26.2</td>
</tr>
<tr>
<td>G043</td>
<td>26.1</td>
</tr>
<tr>
<td><strong>Mean</strong></td>
<td><strong>26.4</strong></td>
</tr>
<tr>
<td><strong>Standard deviation</strong></td>
<td><strong>0.2</strong></td>
</tr>
</tbody>
</table>
Table B.2: Summary of uniaxial compressive strength measurements performed on Hardy Island Granite.

<table>
<thead>
<tr>
<th>Test ID</th>
<th>Uniaxial compressive strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS004</td>
<td>200</td>
</tr>
<tr>
<td>UCS006</td>
<td>184</td>
</tr>
<tr>
<td>UCS008</td>
<td>199</td>
</tr>
<tr>
<td>UCS009</td>
<td>194</td>
</tr>
<tr>
<td>Mean</td>
<td>194</td>
</tr>
<tr>
<td>Standard deviation</td>
<td>6</td>
</tr>
</tbody>
</table>

Figure B.1: Failure envelope derived from uniaxial and triaxial tests performed on Hardy Island Granite.
Figure B.2: Distribution of Young’s modulus vs. confining stress from tests performed on Hardy Island Granite. Points represent the tangent modulus at 50% of the peak axial stress.

Figure B.3: Distribution of Poisson’s ratio vs. confining stress from tests performed on Hardy Island Granite. Poisson’s ratio calculated using the tangent modulus at 50% of the peak axial stress.
Figure B.4: Stress-strain plots from uniaxial compression test UCS008.

Figure B.5: Stress-strain plots from uniaxial compression test UCS009.
Figure B.6: Stress-strain plots from triaxial compression test CTT002.

Figure B.7: Stress-strain plots from triaxial compression test CTT007.
Appendix C – Lab specimen preparation and 3D scanning procedure

Specimens are cored from intact blocks of Hardy Island granite (refer to Appendix B for properties). Coring is performed using a 54 mm-diameter diamond-tipped drill bit. The rough ends of the cores are removed using a rock saw and the surfaces are ground down until they are flat and perpendicular to the core walls. A 15 mm-diameter duct hole is then drilled axially through the centre of the specimen, also using a diamond-tipped core bit. This is the hole through which water pressure will be controlled during the triaxial test. To produce the specimen discontinuities, a circular rock saw is used to cut through the specimens on a plane angled 35 degrees from the axis of the core. This angle will naturally have an effect on the slip behaviour in the test as it will alter the amount of normal stress/shear stress acting on the discontinuity, however it was decided to hold the angle constant for all tests so this variable could be controlled. Besides some small 1-5 mm chips off the edges of the cut, this method was found to produce a smooth surface with few irregularities. Any irregularities left behind are smoothed away with a diamond file, producing a planar discontinuity with less than 0.3 mm of surface variation.

Preparation of the straight cut surface is performed by hand sanding the discontinuity using 150-grit sandpaper, followed by 220-grit sandpaper. To form the polished cut, the discontinuity surface is smoothed on a flat lap polishing wheel using progressively finer grits of wet aluminum oxide powder. After the surface is smoothed with 950-grit powder, it is polished using titanium oxide finisher. To produce the asperity cuts, the top half of each side of the discontinuity is filed down using a lubricated diamond grit tool, leaving behind a linear edge that interlocks with the
The opposite surface to form an offset. The small asperity cuts are filed down by hand to an offset of 1.5 mm. The large asperity cuts are ground down using a rotary tool to a final offset of 8 mm.

Three-dimensional (3D) images of all finished surfaces are obtained using a desktop laser scanner manufactured by NextEngine. The scanner produces a point cloud of raw data which must then be oriented in order to calculate surface variation perpendicular to the saw cut. This calculation is performed in MATLAB using the code presented in Figure C.1. Note that the user must define the offset distance and angles between the reference plane parallel to the scanner and the plane of the saw cut; these are determined manually by visualizing the raw data in 3D Builder prior to running the code.

```matlab
filename = 'EXAMPLE.xyz';
%User-defined file containing raw XYZ point cloud data from scan
xz_angle = 0.6;
%User-defined angle between XZ axis of the saw cut and reference plane parallel to the scanner (in degrees)
yz_angle = -13;
%User-defined angle between XZ axis of the saw cut and reference plane parallel to the scanner (in degrees)
m = dlmread(filename);
zplane = zeros(length(m),2);
for i = 1:length(m)
    zplane(i,1) = -tan(degtorad(xz_angle))*m(i,1) -tan(degtorad(yz_angle))*m(i,2);
zplane(i,2) = (m(i,3) - zplane(i,1))*cos(degtorad(xz_angle))*cos(degtorad(yz_angle));
zplane(i,3) = zplane(i,2)- 33.55;
end
%Calculates surface variation as difference between point cloud and reference plane parallel to saw cut
scatter(m(:,2),zplane(:,2),1,'fill')
for i = 1:length(m)
    if zplane(i,2) > 34.55 || zplane(i,2) < 32.55 || m(i,2) > -26
        m(i,1) = NaN;
m(i,2) = NaN;
zplane(i,3) = NaN;%%
    end
end
%Removes extraneous data points
scatter(m(:,1),m(:,2),8,zplane(:,3),'fill');
```

**Figure C.1:** MATLAB code used to convert raw scan data into surface variation plots.
Appendix D – Slip scaling relationships in laboratory tests

Magnitudes observed in laboratory fault slip experiments are naturally much smaller than those corresponding to natural or mining-induced fault slip events. However, laboratory results should be scalable. That is to say, the fundamental relationships between magnitude, slip area, shear displacement and stress drop defined in Equations 2.2, 2.3 and 2.5 should be maintained. Figure D.1 (originally presented in Zoback & Gorelick, 2012) was created to visually represent these relationships. The range of magnitudes from loading and injection tests was plotted against slip area for lab tests from the current experiment. These events range in size from $M_w = -4.2$ to $M_w = -3.1$, and plot on the lower left side of the diagram, whereas mining-induced fault slip events, which have typical magnitudes of $M < 2$ to $M = 5.5$ (McGarr, 1992; Grobbelaar et al., 2017), would plot on the upper right end of the diagram and beyond. As shown, the shear displacement values observed in lab tests align with the range expected from the plot (between 0.001 and 0.1 mm of slip on the discontinuity).
It is also useful to compare experimental stress drop values to those calculated theoretically. The shear stress drop can be approximately related to seismic moment as follows:

$$M_0 = \Delta\sigma A^{3/2}$$ (A.1)

Because stress drop is obtained uniquely via axial load cell and confining pressure readings, and slip area remains essentially constant throughout the test, $\Delta\sigma$ can be compared against seismic moment as a means of validating that this relationship is observed. The relationship between seismic moment and stress drop in the events from the current lab experiment is presented in
Figure D.2. As expected, the relationship follows a linear trend with larger stress drops corresponding to larger seismic moments.

Figure D.2: Seismic moment vs. shear stress drop on the discontinuity for slip events triggered during loading and injection tests.
Appendix E – UDEC model validation runs

A set of validation runs were conducted in UDEC to determine the appropriateness of modelling parameters included in the main body of the thesis. The need for this test arises from the fact that numerical models inherently simplify the behaviour of mechanical systems, and must therefore be calibrated to ensure their outputs reliably simulate real-world behaviour. Some parameters can be calibrated more easily than others; for example, material properties can be compared against laboratory and field-scale measurements to ensure validity. However, other parameters do not have physical analogs in full scale systems, and must therefore be estimated using other methods. Specifically, validation models were run were used to evaluate the choice of the following parameters:

1. Discrete element block size; and
2. Solve ratio used to bring the model to equilibrium.

Three simulations were run to test the validity of the above parameters; the parameter values assigned in each simulation are presented in Table E.1. Parameters in the control case were assigned the same values used in the series of models presented in the main body of the thesis. The control case parameter values are considered to be reliable if varying the parameters in question does not significantly impact the model outputs.

Table E.1: Summary of parameter values tested in UDEC model validation runs.

<table>
<thead>
<tr>
<th>Model</th>
<th>Block edge length (m)</th>
<th>Solve ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Control Case</td>
<td>5</td>
<td>$1 \times 10^{-5}$</td>
</tr>
<tr>
<td>Low Solve Ratio</td>
<td>5</td>
<td>$5 \times 10^{-6}$</td>
</tr>
<tr>
<td>Small Block Size</td>
<td>2.5</td>
<td>$1 \times 10^{-5}$</td>
</tr>
</tbody>
</table>
Results from the three simulations are presented in Figure E.1. As shown, all three test cases produce notably similar frequency-magnitude distributions. The maximum moment magnitude and number of events recorded in the control case and the low solve ratio model are identical; 56 events with a maximum moment magnitude of 2.73 are observed in each case. Similarly, the small block size model produced a total event count and maximum moment magnitude that are within 2% of the results obtained from the control case.

![Cumulative frequency plot of moment magnitude distributions in UDEC model validations runs.](image)

The total shear displacement accrued on the fault in each of the models is also comparable. 9.3 cm of movement was recorded in both the control case and the low solve ratio model, whereas
9.7 cm of movement was observed in the small block size model, which is within 5% of the displacement observed in the control case.

Ultimately, the results of the validation runs indicate that the choice of a more precise element network and unbalanced force tolerance produce results that are not significantly different from the control case, which has a significantly faster runtime than either of the more precise validation runs. Hence, the use of the control case parameters is considered appropriate for the purpose of this study.
Appendix F – UDEC numerical modelling code

The code used to build and execute the base case hydraulically-stimulated UDEC model is presented below. Note that in addition to running the simulation, this code instructs UDEC to produce a series of output files for each of the history points included in the model, so that stress and displacement results can be exported and processed outside of UDEC.

```plaintext
config fluid

;CREATE BLOCK AND FAULT
round 0.1
edge 2
block -75,-65 -75,280 350,280 350,-65 ; 425 M X 345 M MODEL
crack -75,285 285,-75 ;45 DEG ANGLE FAULT 5 M FROM EXCAVATION BOUND

;CREATE EXCAVATION BOUNDS (UPPER 3 LEVELS)
crack 100 160 100 115
crack 100 115 175 115
crack 175 115 175 160
crack 100 160 175 160
crack 100 145 175 145
crack 100 130 175 130
crack 115 115 115 160
crack 130 115 130 160
crack 145 115 145 160
crack 160 115 160 160

;CREATE EXCAVATION BOUNDS (LOWER 4 LEVELS)
crack 175 115 175 55
crack 175 55 160 55
crack 160 55 160 115
crack 175 70 145 70
crack 145 70 145 115
crack 175 85 130 85
crack 130 85 130 115
crack 175 100 115 100
crack 115 100 115 115

;CREATE MESH BOUNDS

;DIVIDE AND MESH THE MODEL
table 1 269.9,-65.1 -75.1,280.1 280.1,-65.1
gendiv edge 5 range inside table 1
gendiv angle 0 spacing 5 origin 270,-65 range inside table 1
gendiv angle 90 spacing 5 origin 270,-65 range inside table 1
gendiv edge 5 range inside table 1

table 2 220.1,-65.1 -75.1,230.1
jset angle 0 spacing 40 origin -75,-65 range inside table 2
jset angle 90 spacing 40 origin -75,-65 range inside table 2
gendiv edge 40 range inside table 2
```

144
145

table 3: 329.9, -65.1 204.9, 59.9 204.9, 189.9 74.9, 189.9 -15.1, 280.1 350.1, 280.1 350.1, -65.1
jset angle 0 spacing 40 origin 350, -65 range inside table 3
jset angle 90 spacing 40 origin 350, -65 range inside table 3
gen edge 40 range inside table 3

table 4: 219.9, -65.1 -75.1, 230.1 205.1, 60.1 205.1, 190.1, 75.1, 190.1 -75.1, 280.1 350.1, 280.1 350.1, -65.1
jset angle 0 spacing 40 origin 350, -65 range inside table 4
jset angle 90 spacing 40 origin 350, -65 range inside table 4
gen edge 15 range inside table 4

table 5: 330.1, -65.1 205.1, 60.1 205.1, 190.1, 75.1, 190.1 -75.1, 280.1 350.1, 280.1 350.1, -65.1
jset angle 0 spacing 40 origin 270, -65 range inside table 5
jset angle 90 spacing 40 origin 270, -65 range inside table 5
gen edge 7.5 range inside table 5

table 6: 190.1, 24.9 190.1, 175.1 39.9, 175.1
jset angle 0 spacing 7.5 origin 280, -65 range inside table 6
jset angle 90 spacing 7.5 origin 280, -65 range inside table 6
gen edge 15 range inside table 6

generate edge 7.5 range inside table 6

generate edge 15 range inside table 5

save 'control_geometry_injmesh.sav'

;ASSIGN BLOCK FLOW PROPERTIES

group joint 'Joint:BlockContact'
joint model area jks = 6E11 jkn = 6E11 jfric = 89 jcohesion = 10E10 jtension = 10E10 jperm = 83 & ares = 2E-6 azero = 2E-5 range group 'Joint:BlockContact'

;NOTE: stiffness set 10x higher than fault values to prevent slip on block contacts
;NOTE: jperm = 83 chosen based on viscosity of water
;NOTE: flow law exponent/multiplier left as default
;NOTE: new contacts default to these parameters

;ASSIGN FAULT PROPERTIES

joint model residual jks = 6e10 jkn = 6e10 jfric = 40 jrfric = 30 jcohesion = 0 jrescoh = 0 & jtension = 0 jperm = 83 ares = 2e-6 azero = 2e-5 & range region (274.9944, -65.0099) (-70.0049, 280.0016) (-69.9979, 280.0049) (275.0139, -64.9992)

ini sat 0 ;ASSUMING MINE IS DENUATED

;LOCK IN FAR FIELD STRESSES

insitu stress -113.8e6, 0, -56.9e6 ygrad 0.054e6, 0, 0.027e6 szz -85.35e6 zgrad 0, 0.0405e6

;NOTE: stresses vary as a function of depth, assuming dens=2700kg/m^3, sxx=2syy, szz=1.5syy

;APPLY FAR FIELD STRESS TO BOUNDARIES

boundary stress -113.8e6, 0, 0 ygrad 0.054e6, 0, 0 range 349.9, 350.1 -65.1, 280.1
boundary stress -113.8e6, 0, 0 ygrad 0.054e6, 0, 0 range -75.1, -74.9 -65.1, 280.1
boundary stress 0, 0, -58.655e6 range -75.1, 350.1 -65.1, -64.9
boundary stress 0, 0, -49.34e6 range 279.9, 281.279.9, 280.1

;FIX CORNERS OF THE MODEL

boundary xvel 0 range -75.1, -74.9 -65.1, -64.9
boundary xvel 0 range 349.9, 350.1 -65.1, -64.9
boundary xvel 0 range 349.9, 350.1 279.9, 280.1
boundary xvel 0 range -75.1, -74.9 279.9, 281.279.9, 280.1
boundary yvel 0 range -76, 279.9, 279.281

boundary yvel 0 range 349.9, 351 -66, -64
boundary yvel 0 range 349.9, 351 279.9, 281
boundary yvel 0 range -76, 279.281
boundary pp 0.0
fluid density = 1000
set gravity 0,-10

;SOLVE EQUILIBRIUM
solve
reset disp jdisp

;SET STRESS-DISPLACEMENT HISTORY
history ststress 274.92929,-64.92926 ;id1
history ststress 269.92929,-59.929268
history ststress 264.92929,-54.92927
history ststress 259.92929,-49.929276
history ststress 254.92929,-44.929276
history ststress 249.92929,-39.929276
history ststress 244.92929,-34.92928
history ststress 239.92929,-29.929281
history ststress 234.92929,-24.929281
history ststress 229.92929,-19.929283
history ststress 224.92929,-14.929284
history ststress 219.92929,-9.929283
history ststress 214.92929,-4.929282
history ststress 209.92929,0.07071475
history ststress 204.92929,5.0707154
history ststress 199.92929,10.070714
history ststress 194.92929,15.070715
history ststress 189.92929,20.070719
history ststress 184.92929,25.070719
history ststress 179.92929,30.070715
history ststress 174.92929,35.070717
history ststress 169.92929,40.070717
history ststress 164.92929,45.070717
history ststress 159.92929,50.070717
history ststress 154.92929,55.070717
history ststress 149.92929,60.070713
history ststress 144.92929,65.07072
history ststress 139.92929,70.07072
history ststress 134.92929,75.07072
history ststress 129.92929,80.07072
history ststress 124.92929,85.07072
history ststress 119.92929,90.07072
history ststress 114.92929,95.07072
history ststress 109.92929,100.07072
history ststress 104.92929,105.07072
history ststress 99.92929,110.07072
history ststress 94.92929,115.07072
history ststress 89.92928,120.07071
history ststress 84.92929,125.07072
history ststress 79.92928,130.07071
history ststress 74.92928,135.07071
history ststress 69.92928,140.07071
history ststress 64.92929,145.07071
history ststress 59.92928,150.07071
history ststress 54.92928,155.07071
history ststress 49.92929,160.07071
history ststress 44.92928,165.07071
history ststress 39.92928,170.07071
history ststress 34.92928,175.07072
history ststress 29.92928,180.07071
history ststress 24.92928,185.07071
history ststress 19.92928,190.07071
history ststress 14.92928,195.07071
history ststress 9.92928,200.07071
history ststress 4.92928,205.07071
history ststress -0.07071401,210.07071
history ststress -5.070715,215.07071
history ststress -10.07071,220.07071
history ststress -15.070714,225.07071
history sstress -20.07071,230.07072
history sstress -25.07071,235.07071
history sstress -30.07071,240.07071
history sstress -35.07071,245.07071
history sstress -40.07071,250.07071
history sstress -45.07071,255.07071
history sstress -50.07071,260.07071
history sstress -55.07071,265.07071
history sstress -60.07071,270.07071
history sstress -65.07071,275.07071
history sstress -69.929306,279.92926 ;id70

history sdisplace 274.9293, -64.92926 ;id71
history sdisplace 269.9293, -59.929268
history sdisplace 264.9293, -54.92927
history sdisplace 259.9293, -49.929276
history sdisplace 254.92929, -44.929276
history sdisplace 249.92929, -39.929276
history sdisplace 244.92929, -34.92928
history sdisplace 239.92929, -29.929281
history sdisplace 234.92929, -24.929281
history sdisplace 229.92929, -19.929283
history sdisplace 224.92929, -14.929284
history sdisplace 219.92929, -9.929283
history sdisplace 214.92929, -4.929282
history sdisplace 209.92929, 0.07071475
history sdisplace 204.92929, 5.0707154
history sdisplace 199.92929, 10.070714
history sdisplace 194.92929, 15.070715
history sdisplace 189.92929, 20.070719
history sdisplace 184.92929, 25.070719
history sdisplace 179.92929, 30.070715
history sdisplace 174.92929, 35.070717
history sdisplace 169.92929, 40.070717
history sdisplace 164.92929, 45.070717
history sdisplace 159.92929, 50.070717
history sdisplace 154.92929, 55.070717
history sdisplace 149.92929, 60.070713
history sdisplace 144.92929, 65.070712
history sdisplace 139.92929, 70.070712
history sdisplace 134.92929, 75.070712
history sdisplace 129.92929, 80.070712
history sdisplace 124.92929, 85.070712
history sdisplace 119.92929, 90.070712
history sdisplace 114.92929, 95.070712
history sdisplace 109.92929, 100.070712
history sdisplace 104.92928, 105.070712
history sdisplace 99.92929, 110.070712
history sdisplace 94.92929, 115.070712
history sdisplace 89.92928, 120.07071
history sdisplace 84.92929, 125.070712
history sdisplace 79.92928, 130.07071
history sdisplace 74.92928, 135.07071
history sdisplace 69.92928, 140.07071
history sdisplace 64.92929, 145.07071
history sdisplace 59.92928, 150.07071
history sdisplace 54.92928, 155.07071
history sdisplace 49.92929, 160.07071
history sdisplace 44.92928, 165.07071
history sdisplace 39.92928, 170.07071
history sdisplace 34.92928, 175.070712
history sdisplace 29.92928, 180.07071
history sdisplace 24.92928, 185.07071
history sdisplace 19.92928, 190.07071
history sdisplace 14.92928, 195.07071
history sdisplace 9.92928, 200.07071
def zonk
;BASED ON ZONK FUNCTION IN UDEC FISH LIBRARY
;MODIFIED FOR 5-STAGE STRESS REDUCTION AND EXPORT OF HISTORY POINTS
;HISTORY FILE LABELED IN FOLLOWING FORMAT: 'control_stopeX_redX_histX.txt'
;stope = stope being excavated, red = reduction stage (1-5), hist = history ID number
;STOPE_NUM MUST BE DEFINED IN FUNCTION CALL

;mark gridpoints that are on interior boundary and set to force boundary
ib=block_head ; start of block list
loop while ib # 0 ; loop through all blocks
    igp=b_gp(ib) ; start of gridpoint list for block ib
    loop while igp # 0 ; loop through all gridpoints
        ibou=gp_bou(igp) ; index of boundary corner associated with gridpoint
        if(ibou) < 0 then ; if address is negative then it is interior
            ibou2=abs(ibou)
            if (imem(ibou2+2)) = 4 then
                imem(ibou2+2)= 1 ; force boundary
                imem(ibou2+3)= 1 ; force boundary
                gp_extra(igp) = 1.0
            endif
        else
            gp_extra(igp) = 0.0
        endif
    igp=gp_next(igp) ; next gridpoint
endloop
ib= b_next(ib) ; next block in list
end

def reduce
ib=block_head ; start of block list
loop while ib # 0 ; loop through all blocks
    igp=b_gp(ib) ; start of gridpoint list for block ib
    loop while igp # 0 ; loop through all gridpoints
        ibou=gp_bou(igp) ; index of boundary corner associated with gridpoint
        if(ibou) < 0 then ; if address is negative then it is interior
            ibou2=abs(ibou)
            if gp_extra(igp) > 0.0 then
                forcex=fmem(ibou2+4) ; get current total x-force
                forcey=fmem(ibou2+5) ; get current total y-force
                fmem(ibou2+4)= forcex * red_factor ; reduce reaction force
                fmem(ibou2+5)= forcey * red_factor ; reduce reaction force
            endif
        endif
    igp=gp_next(igp) ; next gridpoint
endloop
ib= b_next(ib) ; next block in list

save 'control_properties_medmesh_lostiff.sav'
def relax
old_factor = 1.0
new_factor = .5
loop i (1,4) ;LOOPS FIRST 4 STRESS REDUCTIONS, STRESS IS REDUCED BY HALF EACH TIME
  startcyc = step ;DEFINE CYCLE AT WHICH HISTORY SHOULD START RECORDING
  red_factor = new_factor/old_factor
  reduce
    command
      solve cycle 10000000
    endcommand
  loop j (1,140) ;LOOPS THROUGH ALL HISTORY POINTS
    sname = 'Histories\control_stope' + string(stope_num)
    rname = '_red' + string(i)
    hname = '\hist' + string(j) + '.txt'
    histfile = sname + rname + hname
    command
      history write j begin startcyc histfile
    ;RECORDS A UNIQUELY NAMED FILE FOR EACH HISTORY POINT, FOR EACH STRESS REDUCTION
    endcommand
  endloop
  old_factor = new_factor
  new_factor = new_factor/2
endloop
startcyc = step
red_factor = 0 ;FINAL STRESS REDUCTION TO 0
reduce
  command
    solve cycle 10000000
  endcommand
loop j (1,140) ;LOOPS THROUGH ALL HISTORY POINTS
  sname = 'Histories\control_stope' + string(stope_num)
  rname = '_red5'
  hname = '\hist' + string(j) + '.txt'
  histfile = sname + rname + hname
  command
    history write j begin startcyc histfile
  ;RECORDS A UNIQUELY NAMED FILE FOR EACH HISTORY POINT (LAST STRESS REDUCTION)
  endcommand
endloop

;STOPE 1
set stope_num = 1
delete range 160 175 145 160
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope1.sav

;STOPE 2
boundary interior xfree yfree
block fill 165 150
gen edge 15 range 160 175 145 160
group zone 'Rock:Backfill' range 160 175 145 160
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 2
delete range 145 160 145 160
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope2.sav

;STOPE 3
boundary interior xfree yfree
block fill 150 150
gen edge 15 range 145 160 145 160
group zone 'Rock:Backfill' range 145 160 145 160
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 3
delete range 130 145 145 160
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope3.sav

;STOPE 4
boundary interior xfree yfree
block fill 135 150
gen edge 15 range 130 145 145 160
group zone 'Rock:Backfill' range 130 145 145 160
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 4
delete range 115 130 145 160
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope4.sav

;STOPE 5
boundary interior xfree yfree
block fill 120 150
gen edge 15 range 115 130 145 160
group zone 'Rock:Backfill' range 115 130 145 160
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 5
delete range 100 115 145 160
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope5.sav

;STOPE 6
boundary interior xfree yfree
block fill 105 150
gen edge 15 range 100 115 145 160
group zone 'Rock:Backfill' range 100 115 145 160
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 6
delete range 160 175 130 145
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope6.sav
;STOPE 7
boundary interior xfree yfree
block fill 165 135
gen edge 15 range 160 175 130 145
group zone 'Rock:Backfill' range 160 175 130 145
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 7
delete range 145 160 130 145
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope7.sav

;STOPE 8
boundary interior xfree yfree
block fill 150 135
gen edge 15 range 145 160 130 145
group zone 'Rock:Backfill' range 145 160 130 145
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 8
delete range 130 145 130 145
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope8.sav

;STOPE 9
boundary interior xfree yfree
block fill 135 135
gen edge 15 range 130 145 130 145
group zone 'Rock:Backfill' range 130 145 130 145
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 9
delete range 115 130 130 145
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope9.sav

;STOPE 10
boundary interior xfree yfree
block fill 120 135
gen edge 15 range 115 130 130 145
group zone 'Rock:Backfill' range 115 130 130 145
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 10
delete range 100 115 130 145
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope10.sav

;BACKFILL PREVIOUS STOPE
boundary interior xfree yfree
block fill 105 135
gen edge 15 range 100 115 130 145
group zone 'Rock:Backfill' range 100 115 130 145
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve

;ASSIGN FLOW PROPERTIES
set flow compressible
set capratio 3
set fluid_dg_type = 3 ;reduce computation time
fluid bulk 100e6 ;reduced from typical value to speed computation time
history flowtime

;START INJECTION
well flow 7.41e-5 atdomain 137.5 85 ;inject at a rate of 400 L/min, 2D geometry factor of 1/90
cycle ftime 1200 ;inject for 20min ;inject at midpoint of level, 12.5 m above fault
set flow off
pfix ppressure 0
solve

;STOPE 11
call 'Zonk_5steps.fis'
set stope_num = 11
delete range 160 175 115 130
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 12
boundary interior xfree yfree
block fill 165 120
gen edge 15 range 160 175 115 130
group zone 'Rock:Backfill' range 160 175 115 130
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 12
delete range 145 160 115 130
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 13
boundary interior xfree yfree
block fill 150 120
gen edge 15 range 145 160 115 130
group zone 'Rock:Backfill' range 145 160 115 130
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 13
delete range 130 145 115 130
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 14
boundary interior xfree yfree
block fill 135 120
gen edge 15 range 130 145 115 130
group zone 'Rock:Backfill' range 130 145 115 130
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 14
delete range 115 130 115 130
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 15
boundary interior xfree yfree
block fill 120 120
gen edge 15 range 115 130 115 130
group zone 'Rock:Backfill' range 115 130 115 130
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 15
delete range 100 115 115 130
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope15.sav

;BACKFILL PREVIOUS STOPE
boundary interior xfree yfree
block fill 105 120
gen edge 15 range 100 115 115 130
group zone 'Rock:Backfill' range 100 115 115 130
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve

;ASSIGN FLOW PROPERTIES
set flow on

;START LEVEL 3 INJECTION
pfree
well flow -7.41e-5 atdomain 137.5 85 ;cancel flow in other well
well flow 7.41e-5 atdomain 145, 77.5
inject at a rate of 400 L/min, 2D geometry factor of 1/90
cycle ftime 1200 ;inject for 20min ;inject at midpoint of level, 12.5 m above fault
save inject_lvl3_20min_factored.sav

set flow off
pfix ppressure 0
solve

;STOPE 16
call 'Zonk_5steps.fis'
set stope_num = 16
delete range 160 175 100 115
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 17
boundary interior xfree yfree
block fill 165 105
gen edge 15 range 160 175 100 115
group zone 'Rock:Backfill' range 160 175 100 115
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 17
delete range 145 160 100 115
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 18
boundary interior xfree yfree
block fill 150 105
gen edge 15 range 145 160 100 115
group zone 'Rock:Backfill' range 145 160 100 115
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 18
delete range 130 145 100 115
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 19
boundary interior xfree yfree
block fill 135 105
gen edge 15 range 130 145 100 115
group zone 'Rock:Backfill' range 130 145 100 115
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 19
delete range 115 130 100 115
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope19.sav

;BACKFILL PREVIOUS STOPE
boundary interior xfree yfree
block fill 120 105
gen edge 15 range 115 130 100 115
group zone 'Rock:Backfill' range 115 130 100 115
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve

;ASSIGN FLOW PROPERTIES
set flow on

;START LEVEL 4 INJECTION
pfree
well flow 7.41e-5 atdomain 145, 77.5
well flow 7.41e-5 atdomain 152.5, 70 ;inject at a rate of 400 L/min, 2D geometry factor of 1/90
cycle ftime 1200 ;inject for 20min ;inject at midpoint of level, 12.5 m above fault
save inject_lvl4_20min_factored.sav

set flow off
pfix ppressure 0
solve

;STOPE 20
call 'Zonk_5steps.fis'
set stope_num = 20
delete range 160 175 85 100
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 21
boundary interior xfree yfree
block fill 165 90
gen edge 15 range 160 175 85 100
group zone 'Rock:Backfill' range 160 175 85 100
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 21
delete range 145 160 85 100
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 22
boundary interior xfree yfree
block fill 150 90
gen edge 15 range 145 160 85 100
group zone 'Rock:Backfill' range 145 160 85 100
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 22
delete range 130 145 85 100
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope22.sav

;BACKFILL PREVIOUS STOPE
boundary interior xfree yfree
block fill 135 90
gen edge 15 range 130 145 85 100
group zone 'Rock:Backfill' range 130 145 85 100
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve

;ASSIGN FLOW PROPERTIES
set flow on

;START LEVEL 5 INJECTION
pfree
well flow -7.41e-5 atdomain 152.5, 70
well flow 7.41e-5 atdomain 160, 62.5
;inject at a rate of 400 L/min, 2D geometry factor of 1/90
cycle ftime 1200 ;inject for 20min ;inject at midpoint of level, 12.5 m above fault
save inject_lvl5_20min_factored.sav
set flow off
pfix ppressure 0
solve

;STOPE 23
call 'Zonk_5steps.fis'
set stope_num = 23
delete range 160 175 70 85
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax

;STOPE 24
boundary interior xfree yfree
block fill 165 75
gen edge 15 range 160 175 70 85
group zone 'Rock:Backfill' range 160 175 70 85
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve
call 'Zonk_5steps.fis'
set stope_num = 24
delete range 145 160 70 85
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope24.sav

;BACKFILL PREVIOUS STOPE
boundary interior xfree yfree
block fill 150 75
gen edge 15 range 145 160 70 85
group zone 'Rock:Backfill' range 145 160 70 85
zone model elastic density 2e3 bulk 4e8 shear 5e8 range group 'Rock:Backfill'
solve

;ASSIGN FLOW PROPERTIES
set flow on

;START LEVEL 6 INJECTION
pfree
well flow -7.41e-5 atdomain 160, 62.5
well flow 7.41e-5 atdomain 167.5, 55
;inject at a rate of 400 L/min, 2D geometry factor of 1/90
cycle ftime 1200 ;inject for 20min ;inject at midpoint of level, 12.5 m above fault
save inject_lvl6_20min_factored.sav

set flow off
pfix ppressure 0
solve

;STOPE 25
call 'Zonk_5steps.fis'
set stope_num = 25
delete range 160 175 55 70
boundary interior xvel 0 yvel 0
cycle 1
zonk
relax
save control_excavate_stope25.sav
Appendix G – MATLAB code for processing UDEC outputs

The MATLAB code presented below uses the raw output files created by UDEC to collect information regarding stress and displacement changes in the model. It then processes this information to calculate distributions of slip, stress drop and moment magnitude corresponding to each excavation sequence being modelled. Results are saved in a text file that can be read by excel.

```matlab
% MAKEFOLDERS creates a specific folder structure in Windows Explorer so that UDEC can populate with histories
for s = 1:25
    for r = 1:5
        dir = sprintf('control_stope%d_red%d',s,r);
        mkdir(dir);
    end
end

% SUMMARIZEEVENT cycles through all the UDEC history records collected from a single excavation stage and collects/calculates the following information from each record:
% - x location
% - y location
% - peak shear stress
% - final shear stress
% - shear disp at peak stress
% - final shear disp
% - dist along fault (relative to the left end)
% - shear stress drop
% - delta shear disp
% - time of start of slip
% The information is then summarized in a new text file in this order (hence, this file includes location-based information for one event)
% Next, a new file is created to summarize information from all events. This information includes:
% - point where slip starts
% - point where slip ends
% - length of slipping section
% - area of slipping section (length^2)
% - avg disp over slipping section
% - seismic moment
% - moment magnitude
% - average stress drop
% - time when section started slipping
% - location where section started slipping
% NOTE: THERE MUST BE 70 HISTORY POINTS FOR THIS FUNCTION TO WORK
% NOTE: SHEAR MODULUS SET TO 20 GPA FOR MAGNITUDE CALCULATION
% NOTE: THIS CODE APPLIES FOR EXCAVATION SEQUENCES WITH 1 STOPE AND 5 REDUCTIONS/STOPE

datasummary = NaN(125,10);
for s = 1:25 %for 25 stopes
```

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for i = 1:5 %for 5 reduction stages
summary = NaN(70,10);
for n = 1:70 %for all 70 history points
m = n + 70;
stressfilepath = sprintf('control_stope%d_red%d\hist%d.txt',s,i,n);
%cycles through stress history files
dispfilepath = sprintf('control_stope%d_red%d\hist%d.txt',s,i,m);
%cycles through displacement history files
rawstressfile = dlmread(stressfilepath,"",6,0);
%read stress history record in (time, stress) format
rawdispfile = dlmread(dispfilepath,"",6,0);
%read displacement history record in (time, disp) format
xylocation = dlmread(stressfilepath,"",[2 5 2 6]); %read X Y location
x_coord = xylocation(1,1);
y_coord = xylocation(1,2);
x_start = -70; %leftmost x-coordinate on fault (starting point)
y_start = 280; %topmost y-coordinate on fault (starting point)
fault_loc = sqrt((x_coord - x_start)^2 + (y_coord - y_start)^2);
%distance along fault relative to left end
[peak_str,peak_ind] = max(rawstressfile(:,2)); %extracts the peak stress and the row at which it occurs
init_disp = rawdispfile(peak_ind,2); %extracts the displacement at the time of peak stress
finalDisp = rawdispfile(end,2); %extracts the final displacement
delta_disp = abs(finalDisp - init_disp); %calculates the net slip
if deltaDisp > 5e-4 %only calculate if the fault has slipped
postpkstress = rawstressfile(peak_ind:end,2); %extracts column of post peak stress values
final_str = min(postpkstress); %extracts the minimum post peak stress
stress_drop = final_str - finalDisp; %calculates the shear stress drop
start_time = rawstressfile(peak_ind,1); %calculates the time at which slip was initiated
else
peak_str = NaN;
%peak stress, final stress, stress drop and slip start time are NaN if fault has not slipped
final_str = NaN;
stress_drop = NaN;
start_time = NaN;
end
summary(n,:) = [x_coord y_coord peak_str final_str init_disp finalDisp fault_loc stress_drop delta_disp start_time]; %write line in summary file
end
summaryfilepath = sprintf('control_stope%d_red%d\eventsummary.txt',s,i);
dlmwrite(summaryfilepath,summary) %save summary file for entire reduction stage
if any(summary(:,3)) %if any segments of the fault have slipped
for n = 1:70
if isnan(summary(n,3))
else
lg_dist = summary(n-1,7);
startrow = n;
break
end
end
for n = 70:-1:1
if isnan(summary(n,3))
else
sm_dist = summary(n+1,7);
end
end
endrow = n;
break
end

slip_length = lg_dist - sm_dist;
% calculate length of slipping segment (incl. both end points)
slip_area = slip_length^2;  % slip area assuming square slip surface
avg_disp = mean(summary(startrow:endrow+1,9));
% average displacement across slipping portion of fault (including one end point)
mu = 20e10;  % shear modulus of 20 GPa as per UDEC model
moment = mu*slip_area*avg_disp;  % seismic moment
moment_mag = 2/3*log10(moment)-6;  % moment magnitude
avg_strdrop = mean(summary(startrow:endrow,8),'omitnan');
% average stress drop of slipped points (didn't include ends)
[slipstart_time,slipstart_ind] = min(summary(:,10));  % time and index of slip start
for n = slipstart_ind:70
  if summary(n,10) == slipstart_time
    laststart_ind = n;
  end
end
slipstart_loc = (summary(slipstart_ind,7)+summary(laststart_ind,7))/2;
% location of slip start (averaged if multiple areas slip at once)
row = i+5*(s-1);
datasummary(row,:) = [sm_dist lg_dist slip_length slip_area avg_disp moment
  moment_mag avg_strdrop slipstart_time slipstart_loc];
end
dlmwrite('AllEvents.txt', datasummary)