INVESTIGATION OF THE PRE TO POST PEAK

STRENGTH STATE AND BEHAVIOUR OF CONFINED ROCK MASSES

USING MINE INDUCED MICROSEISMICITY

by

Adam Lee Coulson

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Department of Civil Engineering

University of Toronto

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ABSTRACT

As hard rock mining progresses into higher stress mining conditions through either late stage extraction or mining at depth, the rock mass is driven not just to the peak strength, but often well into the post-peak until complete ‘failure’ occurs and easier mining conditions become evident. Limited research has been accomplished in identifying the transition of the rock mass and its behaviour into the post-peak and this research investigates this behaviour in detail. As the rock mass progressively fails, fractures are initiated through intact rock and extension and shear failure of these and pre-existing features occurs. Associated with this failure are microseismic events, which can be used to give an indication of the strength state of the rock mass. Based on an analogy to laboratory testing of intact rock and measurement of acoustic emissions, the microseismicity can be used to identify, fracture initiation, coalescence of fractures (yield), localization (peak-strength), accumulation of damage (post-peak) and ultimate failure (residual strength) leading to aseismic behaviour. The case studies presented in this thesis provide an opportunity to examine and analyse rock mass failure into the post-peak, through the regional and confined failures at the Williams and the Golden Giant mines, both in the Hemlo camp in Northern Ontario, Canada. At the Williams mine, the progressive failure of a sill pillar region into the post-peak was analysed; relating the seismic event density, combined with a spatial and temporal examination of the principal components analysis (PCA), to characterize the extent, trend and state of the yielding zone, which formed a macrofracture shear structure. Combined with observations of conventional displacement instrumentation, indicates regional dilation or shear of the rock mass occurs at or prior to the point of ‘disassociation’ (breakdown of stable PCA trends) when approaching the residual strength. At the Golden Giant mine, the complete process from initiation to aseismic behaviour is monitored in a highly stressed and confined pendent pillar. The PCA technique and focal mechanism
studies are used to define significant stages of the failure process, in which a similar macrofracture structure was formed. Temporal observations of key source parameters show significant changes prior to and at the point of coalescence and localization. In both studies linear and non-linear modelling was performed to determine the stress path and investigate constitutive failure behaviour.
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DEDICATIONS

This thesis is dedicated to my wife Helen Moroz, son Zachary and my mother Pat Coulson and in memory of Thomas Coulson (1932-1997), Tekla Moroz (1927-2005) and Joseph Moroz (1924-1993).
# TABLE OF CONTENTS

<table>
<thead>
<tr>
<th>PAGE</th>
</tr>
</thead>
<tbody>
<tr>
<td>ABSTRACT ................................................................. ii</td>
</tr>
<tr>
<td>ACKNOWLEDGEMENTS .................................................... iv</td>
</tr>
<tr>
<td>DEDICATIONS ............................................................... v</td>
</tr>
<tr>
<td>TABLE OF CONTENTS ...................................................... vi</td>
</tr>
<tr>
<td>LIST OF FIGURES .......................................................... xi</td>
</tr>
<tr>
<td>LIST OF TABLES ............................................................ xxxii</td>
</tr>
<tr>
<td>NOMENCLATURE .............................................................. xxxiii</td>
</tr>
</tbody>
</table>

## 1 INTRODUCTION ................................................................ 1

1.1 Overview ......................................................................... 1

1.2 Laboratory Scale Strength Testing and Acoustic Emissions (AE) ........................................... 2

1.3 Field Scale Response to Failure and Microseismicity (MS) ..................................................... 7

1.3.1 Brittle and Strain Weakening Behaviour of Rock Masses ................................................. 9

1.3.2 Ductile or Perfectly Plastic Behaviour of Rock Masses ..................................................... 14

1.3.3 General Observations of the Current Knowledge of Rock Mass Behaviour .... 14

1.3.4 Mine Induced Microseismic Analysis .............................................................................. 15

1.4 Thesis Research and Objectives ............................................................................. 17

1.5 Structure of Thesis .............................................................................. 18

## 2 MICROSEISMIC ANALYSIS THEORY ................................ 20

2.1 Introduction ......................................................................... 20

2.2 Microseismic Analysis Based on Location and Source Size .............................................. 21

2.2.1 Seismic Event Density and Clustering Density .............................................................. 22

2.2.2 Principal Component Analysis Technique ...................................................................... 27

2.2.2.1 Principal Component Analysis Parameter Evaluation .............................................. 31

2.3 Overview of Mine Seismology Seismic Source Parameters ............................................... 32

2.3.1 Seismic Source Parameters ......................................................................................... 33

2.3.2 Seismic Source Mechanisms ....................................................................................... 36

2.3.2.1 Overview ................................................................................................................. 36

2.3.2.2 Source Mechanisms from First Motion Studies ..................................................... 38
3 INVESTIGATION OF THE PRINCIPAL COMPONENT ANALYSIS (PCA) TECHNIQUE TO DETERMINE FAILURE STATES - A CASE STUDY WILLIAMS MINE, CANADA ................. 42

3.1 Introduction .............................................................................................................. 42

3.2 William Mine Sill Pillar Study .................................................................................. 43
   3.2.1 Overview of Hemlo Mining Camp ................................................................. 43
   3.2.2 Williams Mining History Overview ............................................................... 43

3.3 Geological Overview .............................................................................................. 46

3.4 Geomechanical Properties ....................................................................................... 48
   3.4.1 Joint Mapping ................................................................................................... 48
   3.4.2 Material Properties ............................................................................................ 50
   3.4.3 Rock Mass Classification and In situ stress regime ........................................... 52

3.5 3D Mine Model and Linear Elastic Modelling ........................................................... 53

3.6 Williams Microseismic System ................................................................................. 53
   3.6.1 Location Error and Array Errorspace ............................................................ 54

3.7 Temporal and Spatial Analysis of Regional Sill Failure ............................................. 57
   3.7.1 Regional Seismicity and Spatial Changes in Microseismic Intensity .................. 57
   3.7.2 Cluster Identification ......................................................................................... 67
   3.7.3 Yearly Variation in PCA Trends Across the Sill ................................................. 70
   3.7.4 Detailed PCA Variations Along the 9390 Level Footwall Development ............ 79
      3.7.4.1 General Observations of Temporal PCA analysis for Slice 2 ................. 79
      3.7.4.2 Analysis of Conventional SMART Cable Instrumentation ...................... 88
      3.7.4.3 Correlation of Rock Mass Deformation to Changes in the Behaviour of the Microseismicity Above the 9390 Level .................................................... 96
      3.7.4.4 Linear Elastic Stress Path and Shear Stress Analysis ............................... 109
      3.7.4.5 Non-linear Modelling of 9390 Level Footwall Zone (stope 26) .............. 134

3.8 Conclusions ........................................................................................................... 142

4 APPLICATION OF THE PCA TECHNIQUE TO DETERMINE FAILURE STATES OF A CONFINED ROCK MASS - A CASE STUDY AT THE GOLDEN GIANT MINE, CANADA. .... 148

4.1 Introduction ............................................................................................................. 148

4.2 Golden Giant Mine ................................................................................................ 150
   4.2.1 Shaft Pillar Mining History Overview ............................................................ 151
   4.2.2 Geomechanical Properties and Mine Model ................................................ 153
   4.2.3 Microseismic System and Array Consistency ................................................ 156
      4.2.3.1 Location Error and Array Errorspace ....................................................... 160
      4.2.3.2 Comparison of Online (Automatic Picks) versus Offline (Manual Picks) on Location and Source Parameter Calculations ........................................... 161
4.2.3.3 Comparison of The Effect of Individual Triaxial Accelerometers on the Averaged Data ........................................................................................................................................... 163

4.3 Temporal and Spatial Analysis of Regional Failures ...................................................... 166

4.3.1 Regional Microseismicity and Cluster Identification .......................................................... 166

4.3.2 Temporal Analysis of the East Slot (EOS) Pendant Pillar (4600 L) ............................. 171

4.3.2.1 Yearly PCA Analysis of EOS 4600L ........................................................................ 171

4.3.2.2 Detailed Temporal PCA Analysis 2002 to 2005 EOS 4600L .................................. 174

4.3.2.3 Linear Elastic Modelling and Stress Path ................................................................. 184

4.3.2.4 Observations of Temporal Changes in Source Parameters ........................................ 192

4.3.2.5 Seismic Scaling Relationships and b-value ................................................................. 208

4.4 Summary and Discussion ............................................................................................... 215

4.4.1 Limitations of the PCA Technique ................................................................................. 215

4.4.2 Limitations of Source Parameters ................................................................................. 217

4.4.3 Linear Elastic Stress Modelling ..................................................................................... 217

4.4.4 Large Magnitude Events ............................................................................................. 219

4.5 Conclusions ..................................................................................................................... 220

5 FOCAL MECHANISM STUDY OF MINE INDUCED MICROSEISMIC EVENTS DURING VARIOUS STAGES OF FAILURE OF A CONFINED ROCK MASS IN RELATION TO THE PRE TO POST-PEAK STRENGTH CONDITIONS ................................................................................................. 225

5.1 Introduction ......................................................................................................................... 225

5.2 Calibration of Microseismic Array for First Motion Studies ............................................. 230

5.3 Focal Mechanism Analysis ................................................................................................. 232

5.3.1 Method and Data Selection ............................................................................................. 232

5.3.2 Overall Faulting Mechanism, Comparison to Modelled Stress Orientation ... 238

5.3.3 Overall Faulting Mechanisms, Comparison to Geology and PCA ............................. 241

5.3.4 Temporal Changes of Faulting Mechanisms .................................................................. 245

5.3.5 Summary of Behaviour and Comparison to Other Sites ............................................. 249

5.4 Stress Analysis ................................................................................................................... 252

5.4.1 Ubiquitous Joint Analysis ............................................................................................... 252

5.4.1.1 Rationalization of the Macrofracture Orientation Based on Stress .......................... 259

5.4.2 Non-Linear Modelling and Estimated Post Peak Parameters ...................................... 261

5.5 Fracture Network Analysis ................................................................................................. 270

5.6 Conclusions ....................................................................................................................... 284
LIST OF FIGURES

Figure 1.1  The three parameters determined from laboratory triaxial compression tests: crack initiation (σ_{ci}); crack damage (σ_{cd}), and peak strength (σ_{f}). ΔV/V, volumetric strain (after Martin 1997) and superimposed typical cumulative AE response (after Eberhardt 1998)........................................4

Figure 1.2  Triaxial testing of a sample of Westerly Granite (σ_{3} = 50 MPa), monitored with acoustic emissions (AE) showing the development of a failure plane through localization of cracks (AE). At point (a) interaction, (b) coalescence and localization of cracks to a macrofracture structure, (c) to (f), strong strain-softening behaviour occurs, (g) approaching the residual strength (AE) no longer as strongly localized (after Lockner et al., 1992; Martin, 1997). ................................................................................................................................................5

Figure 1.3  Triaxial testing of Westerly Granite using AE feedback control (i) Deviatoric stress, average velocity and cumulative AE with time. AE locations are shown looking along the strike of the first fracture (ii), and into the plane of the eventual fracture. Periods A-F are marked on (i). (Thompson et. al., 2006)...............................................................................................................6

Figure 1.4  A typical stress-strain curve from a discrete element simulation involving a bonded particle model under triaxial loading with a σ_{3} of 20 MPa. Accumulation of new cracks (top) and cumulative crack growth (right axis). Points of crack initiation (σ_{ci}), interaction (σ_{cd}), coalescence, localization (σ_{f}) and eventual aseismicity at the residual strength (σ_{cr}) indicated (after Diederichs 2002)...............................................................................................................................................8

Figure 1.5  Characteristic of behaviour (phenomenology) of rocks in (a) brittle and (b) ductile states (after Hajjabolmajid, 2001)................................................................................................................................................10

Figure 1.6  Schematic representation of the behaviour of the rock mass as a function of confinement compared to an intact sample (note: intact (E = 55 GPa) and rock mass modulus (E_{rm} = 17.8 GPa), are based on the material properties found at the Hemlo Camp, the proportional rock mass stiffness based on the GSI ~ 60)................................................................................................................................................11

Figure 1.7  Comparison of laboratory crack initiation, to rock mass initiation at the URL (Martin, 1997). (b) Representation of the bi-linear failure envelope based on damage initiation thresholds and spalling limit from field observations (after Diederichs, 2000; Kaiser et. al., 2000), and (c) Schematic of bi-linear failure curve with damage limits from Brunswick mine, “damage and spalling failure map” (after Diederichs et al., 2002, Diederichs et. al., 2007). Note Δσ_{crit}/Δσ_{3} = slope, (A), of the linear relationship, σ_{1} = Aσ_{3} + Bσ_{c} where A= 1 to 1.5 and B=0.3 to 0.5 Δσ_{crit}= σ_{1} / Bσ_{c} [i.e. normalization of the linear relationship w.r.t. the rock mass strength]........12

Figure 2.1  Microseismicity recorded in the Williams mine sill pillar region analysed in this paper from September 1999 to February 2005. (a) and (c) The sill region analysed here is bounded from stope 28 to stope 16, and analysis slices 1 to 6 are 40 m wide spanning two stopes (Note position of section 9430 E). Approximately 33,000 events where recorded and located during this time frame........................................................................................................................................24

Figure 2.2  (a) Distribution of Madariaga source radius determined in the Golden Giant Shaft Pillar region (b) Representation of worst case event location in a 125 m³ voxel. (c) Calculation of clustering index (CII) for a 5 x 5 x 5 m cube (125 m³) with 5 events (blue line CII for centre event and red line CII for corner events), and 4 events only (green line CII for corner events). (d) Contour of event density for 125 m³ voxels with the lower contour limit set to 5 events and (e) 5 event per voxel iso-surface of all events recorded in the sill region. ........................................................................26
Figure 2.3  Principal Components Analysis Technique. (a) determination of optimum spatial window size, $D$, based on the distribution of the cluster of events hypocentre inter-distances in Euclidian space, using the a Normal Distribution Cumulative Density Function. (b) Determination of the mean hypocentral location of the events that fall within the Spatial Window, situated on the event of interest, and (c) calculation of the spread matrix describing the variance of the hypocentres location to the mean hypocentral location, and (d) application of the principal components analysis through eigenvector and eigenvalue decomposition to produce and ellipsoid with strike and dip determined for the overall trend of the events surrounding the event of interest. (e) Application of the temporal sliding window, in the example shown, $N'$ is 50. .................................................................................................................28

Figure 2.4  Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_o$), corner frequency, ($f_c$) and energy flux, ($J_c$), fitted with a Brune $f^{-2}$ spectral decay.........34

Figure 2.5  Focal Mechanisms based on first motion studies. (a) Example of double-couple "pure shear" event with P-wave radiation patterns (after Urbancic and Young, 1992), (b) Non double-couple mechanisms (after Hasegawa et. al., 1989) and (c) Double-couple mechanisms with principal stress orientations (after Ramsay and Huber, 1987). ................................................39

Figure 3.1  Longitudinal 3D view of the Hemlo mines, and solid model of mining geometry. (A) Williams Mine Sill Pillar Region, (B) Golden Giant Mine Shaft Pillar Region. Shades, represent yearly mining up until September 2005. Inset geographical location of Hemlo Camp in Ontario, Canada. 44

Figure 3.2  Longitudinal Section of the Williams mine, February 2005. ..................................................................................................45

Figure 3.3  Typical geology plan : Williams mine 9390 L in the sill pillar region stopes 23 to 15.......47

Figure 3.4  Stereonet (Dips©) of main joint sets found at the Williams mine, and typical thought the Hemlo Camp (after Bronkhorst et al., 1993; Kazakidis, 1990)........................................................................49

Figure 3.5  Errorspace analysis of Williams seismic system array geometry (a) Array Sept 1999 to Sept 2000 and (b) corresponding Errorspace (m) and (c) Array Sept 2000 to Feb 2005 and corresponding Errorspace (m). (e) online location errors for unfiltered events in the 9390L FW cluster, slice 2, from 1999 to 2005. ..................................................................................................56

Figure 3.6  Microseismicity recorded in the Williams mine sill pillar region analysed in this case study from September 1999 to February 2005. (a), (b) and (c). Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25]. The sill region analysed here is bounded from stope 28 to stope 16, and analysis slices 1 to 6 are 40 m wide spanning two stopes (Note position of section 9430 E). (d) Contour of event density for 125 m³ voxels with the lower contour limit set to 5 events Approximately 33,000 events where recorded and located during this time frame. Also, shown is longitudinal grid plane at section 9860N (b), used for historical seismic density plotting in section 3.7.1..............................................................58

Figure 3.7  Williams mine sill pillar seismic activity and sill pillar stope blasting. (a) Histogram of daily activity from Sept. 1999 to Feb. 2005, (b) Cumulative events with corresponding large macro events. Also in (a) the time periods when full waveform data was available ..................59

Figure 3.8  View looking west of the 26/25 x-cut pillar burst on the 9390 level. This is thought to be the direct cause of the 2.7 Mn (Nuttli) event, and also caused moderate damage to the #25 x-cut and minor damage to the #26 x-cut............................................................................61

Figure 3.9  Yearly seismic event densities (5 x 5 x 5 m voxel), contoured on section 9860N cutting through the centre of the football main cluster on the 9390 L (see Figure 3.6b), (a) Events from Sept 1999 to March 2000 (6,651 events), (b) April 2000 to Dec 2000 (6,662 events), (c) Jan 2001 to Dec 2001 (5,301 events), (d) Jan 2002 to Dec 2002 (3,876 events), (e) Jan 2003 to Dec 2003 (8,721 events), (f) Jan 2004 to Dec 2004 (1,830 events). Also, note sill region analysed here spans from stope 27 to 16..............................................................63
Figure 3.10 Expansion of yield surface in the Williams mine sill pillar region, based on the clustering density of 5 events per 125 m³ voxel. Plots are based on cumulative seismicity from the beginning of monitoring in September, 1999 to (a) June 2000, (b) Dec 2000, (c) Dec 2001, (d) Dec 2002, (e) Dec 2003, and (f) Feb 2005. 65

Figure 3.11 Plot of voxel event count above the 9390 L haulage drift, for transverse section 9432.5E, show increase in seismic intensity from Sept 1999 to Feb 2005. Inset shows the pixelation plot of the event density for transverse section 9432.5E and the location of the voxels plotted in the graph, Also shown as the white line in the inset is the general location of the conventional instrumentation SMART cables, used to monitor displacements and cable loads. 66

Figure 3.12 Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 2 Events (2633). Note mining geometry is fixed at Dec 1999. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25] 68

Figure 3.13 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 2 Events (2038). Note mining geometry is fixed at Dec 1999. 68

Figure 3.14 Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 2 Events (1775). Note mining geometry is fixed at Dec 1999. 69

Figure 3.15 Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 2 Events (3719). Note mining geometry is fixed at Dec 1999. 69

Figure 3.16 Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 2 Events (823). Note mining geometry is fixed at Dec 1999. 70

Figure 3.17 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 1. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 and (f) distribution of clustering events in slice 1 from Sept 1999 to Dec 2004. Note predominant pervasive trend is outlined in bold. 71

Figure 3.18 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 2. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 and (f) distribution of clustering events in slice 2 from Sept 1999 to Dec 2004. Note predominant pervasive trend is outlined in bold. 72

Figure 3.19 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 3. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 and (f) distribution of clustering events in slice 3 from Sept 1999 to Dec 2004. Note predominant pervasive trend is outlined in bold. 73

Figure 3.20 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 4. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 (no events) and (f) distribution of clustering events in slice 4 from Sept 1999 to Dec 2003. Note predominant pervasive trend is outlined in bold. 74

Figure 3.21 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 5 and 6. (a) Sept 1999 to Dec 2000 slice 5, (b) 2001 slice 5, (c) distribution of clustering events in slice 5 from Sept 1999 to Dec 2001,(d) Sept 1999 to Dec 2000 slice 5, (e) 2001 (no events) and (f) distribution of clustering events in slice 6 from Sept 1999 to Dec 2003. 75

Figure 3.22 Stereonets of principal components analysis derived planes of individual temporal windows (50 events, except for the starting window of each year), for clustering events above the 9390 level, in slice 2. Temporal window number, date of last event in window and major plane indicated. 80
Figure 3.23  Temporal window variation in PCA derived planes of mean strike (red stars), dip (purple squares) plotted against the left axis and ellipticity (black triangles) plotted against the right axis, for clustering events above the 9390 L FW haulage, Slice 2, versus time in (a) and versus temporal window number in (b). Large magnitude events are also indicated in both plots. Numbered points in (a) indicate time frames of minor temporal plane orientation changes identified from the CMPCA discussed in Section 3.7.4.3.4 for comparison.

Figure 3.24  (a) Event rate (moving average number of events every 7 days), for clustering events in slice 2, above the 9390 L FW haulage, (b) Plot of percentage of events that had PCA planes with ellipticity >2.5, per 50 event temporal window for the same cluster, and also indicated large magnitude events and mining of stopes, (the first diamond approximates the date of the void blast and last diamond the date of the mass blast for each stope). Numbered points relate to a significant drop in the number of planes with ellipticity < 2.5, and the corresponding PCA temporal window number (e.g. Win#8) is indicated for reference to Figure 3.22 a to d.

Figure 3.25  (a) Plan view of 9390 L sill pillar region, showing boundaries of seismic analysis slices and location of conventional instrumentation (SMART cables). (b) Longitudinal view of conventional instrumentation and contours of depth of displacement that exceed 1 mm dilation, for three points in time, 12-2000, 12-2002 and 07-2003. Note instruments 17, 18, 19 and 20 showed negligible movement during the monitoring period. Also shown on (b) depth of caving on 9415L as of 12-1999 following 2.6 Mn event (Yi, 1999).

Figure 3.26  Instrument 26-1 (a & c) and 26-2 (b & d) SMART cable response for nodal displacement in mm relative the toe of the instrument versus time (top), and plot of displacement in mm versus depth along the cable, with datum at the back of the excavation. Note in figure (f) the deep dilation, 9 m up from the back at the end of the monitoring period, while bulking is occurring over the first 3.5 m.

Figure 3.27  (a) Temporal window variation in PCA derived planes of mean strike, dip and ellipticity, for clustering events above the 9390 L FW haulage, Slice 3, and large magnitude events indicated versus time. (b) Event rate (moving average number of events every 7 days).

Figure 3.28  (a) Temporal window variation in PCA derived planes of mean strike, dip and ellipticity, for clustering events above the 9390 L FW haulage, Slice 4, and large magnitude events indicated versus time. (b) Event rate (moving average number of events every 7 days).

Figure 3.29  (a) Temporal window variation in PCA derived planes of mean strike, dip and ellipticity, for clustering events above the 9390 L FW haulage, Slice 1, and large magnitude events indicated versus time. (b) Event rate (moving average number of events every 7 days).

Figure 3.30  Temporal variation of PCA planes determined for the clustering events above the 9390 level haulage in slice 2, (a) using 25 point running average of the PCA strike and comparing the analysis using a 50 event temporal window with a 50 event shift (N'=50), versus a 50 event continuously shifting temporal window centred on ±25 events and a shift of 1 event (N'=Cont ±25), and performing 25 point running average on the ellipticity calculated for both analyses. (b) 25 point running average of the PCA dip for (N'=50) and (N'=Cont. ±25). Note that all planes where filtered out if the F-parameter was less than 0.5 (50%). Also, noted are the large magnitude events, the points of disassociation in each slice and the points of sudden and deep dilation in the SMART cables.

Figure 3.31  (a) Longitudinal view of Williams and Golden Giant Mine used for three dimensional boundary element stress analysis. The colour represent yearly mining, determined from 1995 until September 2005. (b) Detailed view of analysis grids, encompassing the footwall development (Mining step 12-2000 shown).

Figure 3.32  Contours of Maximum principal stress (\(\sigma_1\)), plotted on the mid level grid (9407 elv.) between the 9390 and 9415 levels, for yearly mining steps 12-1999 to 12-2004. Left inset, contours of \(\sigma_1\) on a transverse grid (Section 9400E) through the centre of the footwall at stope 26, and right inset yearly mining geometry.
Figure 3.33  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 1 and 2 on section 9400E (26 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$, $\sigma_2$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds. ...........................................................................................................114

Figure 3.34  (a) Average linear elastic stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 2 on section 9435E (24 stope) Inset shows location stress averaging polygons. (b) Average linear elastic stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds. ...........................................................................................................115

Figure 3.35  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 3 on section 9475E (22 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds. ...........................................................................................................116

Figure 3.36  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 4 on section 9515E (20 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds. ...........................................................................................................117

Figure 3.37  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 5 on section 9555E (18 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds. ...........................................................................................................118

Figure 3.38  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 6 on section 9600E (16 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds. ...........................................................................................................119

Figure 3.39  Comparison of (a) iso-surface of clustering density (5 events per 125 m³ cell) at December 2001, versus (b) to (d) the linear elastic iso-surface of the brittle Hoek-Brown factor of safety $<1.0$ for $m=0$ and $s=0.09$ ($\sigma_1 - \sigma_3 = 0.3\sigma_c$) for year 12-1997, 12-1999 and 12-2001. Note perspective view is South, looking at the Sill region from the HW, and section 9400 E is identified for next Figure. ...........................................................................................................127

Figure 3.40  Progressive evolution from 12-1999 to 12-2004, of failure zone (FOS < 1.0) for transverse section 9400E (Slice 2 – 9390-26 stope), using the brittle Hoek-Brown failure parameters of $m=0$ and $s=0.09$ ($\sigma_1 - \sigma_3 = 0.3\sigma_c$) ...........................................................................................................128

Figure 3.41  Progressive evolution from 12-1999 to 12-2004, of failure zone (FOS < 1.0) for transverse section 9400E (Slice 2 – 9390-26 stope), using the Hoek-Brown rock mass strength parameters of $m=2.397$ and $s=0.0117$ ...........................................................................................................129

Figure 3.42  Linear elastic ubiquitous joint analysis based on the stress state at 06-2001 above the 9390 level haulage section 9400E (Stope 26) view looking West. Plots of joint or plane factor of safety (FOS) for the Mohr-Coulomb joint model of $\phi = 30^\circ$ and $C = 0$ MPa. (a) FOS for plane oriented horizontally (Strike R/ Dip [090,00]), (b) sub horizontal, simulating the C-set [090,10], (c) sub vertical, simulating the PCA dominant orientation [090,40], also indicated are the ubiquitous joint planes oriented at $10^\circ$ and $40^\circ$ used to determine the shear and normal stress path histories and (d) sub vertical, simulating the A-set orientation [270,70] ..................................................131

Figure 3.43  Linear Elastic Ubiquitous Joint analysis based on the stress state at 06-2001 above the 9390 level haulage section 9400E (Stope 26) plotting (a) variation of mean factor of safety (FOS) versus joint orientation, measured from the horizontal counter clockwise (CCW), for three material models of i. $\phi_i = 30^\circ$ and $C = 10$ MPa, ii. $\phi_i = 30^\circ$ and $C = 0$ MPa and iii. $\phi_i = 0^\circ$ and $C = 10$ MPa. Note also indicated is the predominant range of the PCA trend for Slice 2. (b) Shows the shear and normal stress path histories on two joint orientations (see previous figure) from 12-1993 to 12-2004. ...........................................................................................................132
Figure 3.44  (a) Detailed view and boundary conditions, of the pseudo hybrid non-linear model of 9390L haulage at #26 stope, transverse section 9400E, showing also the location of the query line. (b) Comparison of modelled relative vertical displacements along query line with measured instrument response for SMART 26-1 (12-1999 to 09-2002), and (c) comparison of modelled relative vertical displacements along query line with measured instrument response for SMART 26-2 (07-2001 to 05-2003), the model is multistaged, using H-B brittle-plastic (Table 3.7) and dilation parameter = 0.8. Note dashed line shows displacements of the last reading and occurrence of deep dilation.................................................................................................................. 135

Figure 3.45  Contour plots of the maximum principal stress, $\sigma_1$, at mining step 12-2002, showing the comparative development of element shear (failed elements) for four different non-linear modelling parameters, (a) perfectly plastic post peak behaviour based on H-B peak rock mass parameters, Dil=0, (b) H-B brittle-plastic post peak behaviour using the best fit post peak parameters (Table 3.7), Dil=0.8, (c) same model but with Dil=0.6, and (d) Equivalent Mohr-Coulomb, peak and post peak parameters, Dil=11.1° equivalent to 0.8 for the H-B. (e) Comparison of seismic development, around footwall drift in front of #26 stope, seismic and residual strength lines. .................................................................................................................. 139

Figure 3.46  Contour plots of the minimum principal stress, $\sigma_3$, at mining step 12-2002, for the four different non-linear modelling parameters (see previous Figure and Table 3.7). Note the development of high confinements (> 35 MPa) at the core of the pillars between the footwall haulages, in the region where seismicity occurs................................................................................................................................. 141

Figure 3.47  (a) Stereonet (Dips©) of main joint sets found at the Golden Giant mine in the shaft region and east of the shaft, and typical thought the Hemlo Camp (after Leduc, 1991). (b) Mapping from four sub-vertical borehole (length 20-30m) camera surveys, of the east and west walls of the shaft. Note that the B-set here is representative more of extensile fracturing of the shaft walls and the distinct absence of C-sets (Coulson et al., 1995). ............................................................... 155

Figure 3.48  Errorspace analysis of Golden Giant dense Shaft Array (a) Array Feb 2003, uniaxial (circle) and triaxial (triangle) sensors (b) Corresponding errorspace (m) based on the nearest 17 sensors to the analysis volumes centroid (note: average number of sensors triggered 17) (c) Distribution of online location errors for events in the East of slot cluster (EOS), from April 2002 to Sept 2005. .................................................................................................................................................. 157

Figure 3.49  Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_{\omega}$), corner frequency, ($f_c$) and energy flux, ($J_0$), fitted with a Brune $f^{-2}$ spectral decay...... 158

Figure 3.50  (a) East of Slot (EOS) cluster of events (1561) spanning from May 2002 to December 2003, used for analysis of effects of triaxial array consistency on source parameter calculation, and effects of low frequency cut-off. (b) Subset of events (250) spanning from May 2002 to May 2004 which where manually picked and used for analysis of automatic versus manual calculations and fault plane solutions (Chapter 5). ......................................................................................... 162

Figure 3.51  Comparison of Automatic picking versus manual picking of p-wave arrivals for a representative 250 event sub-set of the data in the EOS cluster, spanning from May 2002 to May 2004. (a) Northing, (b) Easting, and (c) Depth (axis range for all is 30 m). (d) Moment Magnitude, M (axis range –1.8 to –0.4). ................................................................................................................................. 163
Figure 4.8  Comparison of Seismic Moment, \( (M_o) \) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials (a, b, c and d), and (e) comparison of temporal variation of average seismic moment for each 3 triaxial analysis, using a 50 event moving average. ..........165

Figure 4.9  (a) Microseismicity recorded in the Golden Giant Shaft Pillar region for one year of data during 2003. Analysis area EOS indicated. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25] (b) Detailed long section around the destress slot of advancing seismicity with time a mining. Events coloured by date, see legend (after Mercer, 2003). .......................................................................................................................167

Figure 4.10  Expansion of yield surface in the Golden Giant shaft pillar region, based on the clustering density of 5 events per 125 m3 voxel. Plots are based on cumulative seismicity from the beginning June, 1994 to (a) Feb 2002, (b) Aug 2002, (c) Dec 2002, (d) Feb 2003, (e) Aug 2003, and (f) Feb 2005. ..................................................................................................................................................................................168

Figure 4.11  Location of yearly microseismicity in the Q1/Q2 cluster and stereographic distribution of PCA derived planes for (a) 2002 post localization (b) 2003 – probable disassociation. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25]..................................................................................................................................................................................170

Figure 4.12a,b  Location of yearly microseismicity in the EOS cluster and stereographic distribution of PCA derived planes for (a) 2002 – pre-coalescence, (b) 2003 – coalescence/ localization, and disassociation. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25]..................................................................................................................................................................................172

Figure 4.13a  Stereonets of PCA derived planes of individual 50 event temporal windows, for clustering events above the 4600L, EOS cluster. Temporal window number, date of last event in window and major plane indicated. Dashed border represents low number of events with ellipticity > 2.5. ..................................................................................................................................................................................175

Figure 4.14  (a) Temporal variation in mean PCA derived planes for EOS cluster from May 2002 – July 2005, showing variation in mean strike and dip, and ellipticity per temporal window. Also indicated are stopes mined (e.g. S5 mined between 4620 and 4600 levels is named 460-S5). (b) Event rate in EOS cluster, (c) Temporal variation plotted against PCA temporal window number. ..................................................................................................................................................................179

Figure 4.15  Temporal variation of PCA planes determined for the events in the EOS cluster, (a) using 25 point moving average of the PCA strike and comparing the analysis using a 50 event temporal window with a 50 event shift (N'=50), versus a 50 event continuously shifting temporal window centred on ± 25 events and a shift of 1 event (N'=Cont.+25), and moving average ellipticity, (b) moving average of the PCA dip for (N'=50) and (N'=Cont.+25), and standard error of strike, dip and ellipticity ((N'=Cont.+25) (c) moving average of the number of planes dropped due to ellipticity < 2.5.............. ..........................................................182

Figure 4.16  Linear elastic principal stresses determined from Examine3D, for the Golden Giant shaft pillar region. (a) and (b), stress magnitudes of the maximum (\( \sigma_1 \)) and minimum (\( \sigma_3 \)) principal stresses respectively for Dec 2001 prior to mining of the destress slot, and (c) and (d) the same but for the Dec 2003, after completion of the main stopes of the destress slot. (note the first ‘top’ view is of a plane cutting the 4600 L, the ‘front’ view of a plane cutting through the shaft, and ‘right’ view of a plane cutting through the EOS region, while the ‘perspective view is looking at the shaft pillar region from the hanging wall.)..............................................................................................................185

Figure 4.17  (a) Average linear elastic stress path from 1993-2005, from the defined polygon on section 10480E (inset) covering the region of the clustering of events between the 4600 and 4620 levels east of the destress slot. The peak rock mass failure curves are displayed for reference. (b) Average linear elastic principal stress magnitudes versus time. Also plotted for comparison is the average stress path determined at the Williams Mine for the 9390L FW cluster Slice 2, Region 2, on Sec 9435E (Coulson, 2008a). .........................................................................................................................................................186

Figure 4.18  Comparison of contours of factor of safety (FOS) on section 10480E through the core of the EOS cluster, for the Hoek-Brown Brittle parameters (a) & (b) initiation damage limit \( 0.3\sigma_c \) compared to (c) & (d) initiation damage limit \( 0.37\sigma_c \). (e) & (f) Hoek-Brown rock mass peak strength parameters. ..............................................................................................................................................................................................................189
Figure 4.19  Temporal variation of source parameters (EOS – Cluster) using a 50 event moving average for, (a) Moment magnitude, (b) Seismic moment, (c) Seismic Energy. .................................193

Figure 4.20  Temporal variation of source parameters (EOS – Cluster) using a 50 event moving average for, (a) Apparent Stress, (b) Static Stress Drop (Madariaga Model), (c) Dynamic Stress Drop (Madariaga Model). .............................................................................................................194

Figure 4.21  (a) Temporal variation of Source Complexity ($\Delta\sigma_d/\Delta\sigma$), for the EOS Cluster using a 50 event moving average. Estimation of fracture geometry from a cross section on the 10480E plane through the cluster of events based on location, source radius and PCA calculated orientation of events for (b) the period prior to interaction, 04/18/2002 to 01/18/2003 and (c) the period during coalescence and localization (high ellipticities), 03/05/2003 to 04/30/2003 (from Chapter 5).  ............................................................................................................................195

Figure 4.22  Temporal variation of source parameters (EOS – Cluster) using a 50 event moving average for, (a) $E_s/E_p$ ratio, (b) Source radius (Madariaga), (c) Apparent radius. .........................196

Figure 4.23  Comparison of distribution of Seismic Moments, for pre-interaction, coalescence/localization and post localization (disassociation), indicating a shift in the population mean. (a) Seismic Moment and (b) Log(Seismic Moment). .................................................................201

Figure 4.24  Identification of a shift in the population mean of Apparent Stress at Brunswick Mine (after Simser and Falmagne, 2004). Note Upper right view is a plot of the events apparent stress and location in the South Bulk Ore zone before the mass blast, while the upper left denotes the position of the mass blast and a sample of the event recorded in the period immediately after. The distribution curve for the period following is based on a greater sample than displayed. ........................................................................................................................................202

Figure 4.25  Comparison of the cyclic behaviour of the average Seismic Moment and Source Radius (EOS–Cluster), for (a) the period covering Interaction and Coalescence\ Localization and (b) Post Localization and into Disassociation.................................................................................................................................205

Figure 4.26  Temporal variation of source parameters for only events West of the raise (Sec 10490E) in the EOS – Cluster for, (a) Moment magnitude, (b) Seismic Energy, and (c) Source Radius. Note the trend follows the pattern as for the entire cluster, however, the peak Magnitudes and Energies, prior to Yield are reduced ( Total # of event is 1782 versus 2761 for the entire data set). ........................................................................................................................................207

Figure 4.27  Scaling relationships. (a) and (b) Seismic Moment, $M_o$, versus Source Radius, $r_o$, with lines of constant Static Stress Drop. (c) and (d) Seismic Energy, $E_o$, versus Seismic Moment, $M_o$, with lines of constant Apparent Stress and trend line.................................................................209

Figure 4.28  (a) Log-linear Frequency magnitude distribution, and magnitude range over which b-values (slope of the line) were calculated for all data in the EOS cluster (b) Frequency – magnitude for data for Pre-Interaction, Localization and Post-Localization (see Table 4.3) (c) Temporal variation of b-value for a 50 event moving window (backward looking) for cumulative and non-cumulative b-values, (d) Event rate for the EOS cluster.................................................................................................................................213

Figure 5.1  All recorded events during 2003 in the shaft pillar area, showing the location of the destress slot and shaft and the east of slot (EOS) cluster of events analysed in this chapter. ......227

Figure 5.2  Yearly temporal analysis of PCA for EOS cluster (a) Source locations and plot of PCA poles in 2002 – Pre-Interaction, (b) Locations and PCA poles 2003 – Interaction, Coalescence, Localization and Disassociation, (c) Locations and PCA poles 2004 – Post Disassociation and (d) Location and PCA poles 2005 – Post Disassociation. .................................................................227

Figure 5.3  (a) Temporal variation in mean PCA derived planes for EOS cluster from May 2002 – July 2005, showing variation in mean strike and dip, and ellipticity per temporal window plotted against time of the last event in the window. Indicated for points [1] to [5] are estimation of the strength states of the rock mass during failure. Also indicated are the time periods A to E in which samples of events where analysed using first motions. (b) Event rate in EOS cluster and stopes mined (e.g. S5 mined between 4620 and 4600 levels is named 460-S5).................228

Figure 5.4  Array Feb 2003, uniaxial (circle) and triaxial (triangle) sensors (a) View North, (b) View East and (c) Rotated Plan View looking down the orebody. .................................................................231
Figure 5.5  Example of selection criteria for fault plane analysis (a) Online location of events in the EOS cluster from Jan 1, 2003 to April 30, 2003 (1050 events) covering periods B and C (b) Same period but for events triggering a minimum of 20 sensors (342 events total) and selected events at the core of the cluster (100 events) with selected events only in the view west (c) Relocated events in cluster after re-picking of P- and S-waves. Note colour legend is based on the number of sensors hit.................................234

Figure 5.6  Comparison of temporal variation of seismic moment for the (a) the entire EOS cluster of events for 2003 and (b) the sub-set of events used for the first motion study at the core of the cluster and using a filtering criteria of greater than 20 sensors triggered....................237

Figure 5.7  Typical fault plane solutions determined for events in Periods B and C (February 22 2003 to May 30 2003) associated with the stress state of interaction and coalescence/localization. Note Blue triangle are compressional (+ve) and Yellow triangles are tensile (-ve) first motions, while squares are uncertain polarities.................................239

Figure 5.8  Comparison of (a) linear elastic induced stress tensor orientation based on event locations of the fault plane sub-sets for February 2003 to May 2003 (Periods B and C) to the stress axis of all 243 focal mechanism for (b) the P-axis, the compression axis equivalent to \( \sigma_1 \), (c) the B-axis, the null axis equivalent to \( \sigma_2 \), (c) and the T-axis, the tension axis, equivalent to \( \sigma_3 \). ...............................................................................................................................240

Figure 5.9  Comparison of poles plotted on lower hemisphere stereographic projections determined from (a) Joint mapping, (b) PCA derived planes for 2003 only for the EOS cluster, (c) All probably Fault planes (243) based on stress and clustering for April 2002 to April 2004 (Periods A to E) of sampled sub-sets and (d) All other nodal (Auxiliary) planes. .............................242

Figure 5.10  (a) Joint Mapping with grouping of all determined fault plane solutions for April 2002 to April 2004 (Periods A to E) of sampled sub-sets for (b) all probable reverse faults, (c) all probable normal faults and (d) all probable strike-slip faults, and (e) all reverse fault nodal planes, (f) all normal nodal planes and (g) all strike-slip nodal planes with auxiliary planes noted. .........................................................................................................................243

Figure 5.11  Temporal breakdown of all probable fault planes for the various periods (a) Pre-interaction, April 19 2002 to January 31 2003 - Period A, (b) Interaction, February 12 2003 to March 4 2003 – Period B, (c) Coalescence and Localization, March 6 2003 to April 30 2003 – Period C, (d) Point of disassociation and post disassociation, June 15 2003 to December 30 2003 – Period D, and (e) Post disassociation, January 1 2004 to April 16 2004 – Period E........246

Figure 5.12  Visual summary of the proportion of changing focal mechanisms determined for the various temporal periods A to E. ...............................................................................................................................247

Figure 5.13  Lockerby Sill Pillar Study (after Sampson-Forsythe, 1994), determined P-, B- and T-axis for 194 reverse faulting mechanisms in the Sill, with model induced linear elastic stress tensor orientations indicated. Reverse Fault plane solutions with probable fault planes oriented in a similar fashion to Golden Giant. ..............................................................................................................251

Figure 5.14  Linear elastic ubiquitous joint analysis based on the stress state at 02-2003 above the 4600 level David Bell Stopes on a section (1048OE) at the core of the EOS cluster, view looking West. Plots of joint or plane factor of safety (FOS) for the Mohr-Coulomb joint model of \( \Phi = 30^\circ \) and \( C = 10 \) MPa. (a) FOS for plane oriented horizontally (Strike R/ Dip [090,00]), (b) FOS for plane oriented sub horizontally, simulating the C-set orientation [090,25], (c) FOS for plane oriented sub vertically, simulating the dominant PCA or Fault Plane orientation [090,45], also indicated are the ubiquitous joint planes oriented at 25\(^\circ\) and 45\(^\circ\) used to determine the shear and normal stress path histories and (d) FOS for plane oriented sub horizontally, simulating the A-set orientation [270,60].................................................................................................................254
Figure 5.15  Linear Elastic Ubiquitous Joint analysis based on the stress state at 02-2003 for the EOS region plotting (a) variation of mean factor of safety (FOS) versus joint orientation, measured from the horizontal counter clockwise (CCW), for three material models of i. $\phi = 30^\circ$ and $C = 10$ MPa, ii. $\phi = 30^\circ$ and $C = 0$ MPa and iii. $\phi = 0^\circ$ and $C = 10$ MPa. Note also indicated is the predominant range of the PCA and Fault plane trend (b) Shows the shear and normal stress path histories on two joint orientations (see previous figure) from 12-1989 to 10-2003. .........................................................................................................................255

Figure 5.16  Non-Linear (Parameter Set 1 – Table 5.6) ubiquitous joint analysis based on the stress state at 12-1995 on a section (10480E) at the core of the EOS cluster, viewing West. Plots of joint or plane factor of safety (FOS) for the Mohr-Coulomb joint model of $\phi = 30^\circ$ and $C = 10$ MPa. (a) FOS for plane oriented horizontally (Strike R/ Dip [090,00]), (b) FOS for plane oriented sub horizontally, simulating the C-set orientation [090,25], (c) FOS for plane oriented sub vertically, simulating the dominant PCA or Fault Plane orientation [090,45], also indicated are the ubiquitous joint planes oriented at 25° and 45° used to determine the shear and normal stress path histories and (d) FOS for plane oriented sub horizontally, simulating the A-set orientation [270,60] (e) Minimum FOS contour for orientation [090,50], for the purely cohesive material model.................................................................................................................257

Figure 5.17  Non-Linear (Parameter Set 1 – Table 5.6) Ubiquitous Joint analysis based on the stress state at 12-1995 for the EOS region plotting (a) variation of mean factor of safety (FOS) versus joint orientation, for the three material models of i., ii. and iii. Note also indicated is the predominant range of the PCA and Fault plane trend (b) Shows the shear and normal stress path histories on two joint orientations (see previous figure) from 12-1989 to 10-2003. ...............258

Figure 5.18  (a) Geometry of Phase² model for section 10480E with boundary tractions applied. (b) Non-linear model using parameter set 1 (base case Brittle-Plastic analysis) applied over the entire domain (here the domain was not perfectly plastic), showing development of similar shear at the core of the EOS cluster, at mining step 12-1995. .........................................................................................264

Figure 5.19  Comparison of Peak failure envelopes and Post-Peak residual envelopes used in the parametric analysis using Phase². Based on Table 5.6. Also, shown is a typical element failure path for brittle-plastic post peak behaviour. .................................................................267

Figure 5.20  Contour plots of non-linear maximum principal stress, $\sigma_1$, with element failure (Shear= x, Tension= O), at mining step 06-1996, the stage of the shear development in the core of the EOS region. Brittle-Plastic models (a) parameters set [1] (base case), (b) parameter set [2], (c) parameter set [3], (d) parameter set [4] and Perfectly Plastic models (e) parameter set [5] and (f) parameter set [6] .................................................................................................................268

Figure 5.21  (a) Single isolated crack geometry, based on a Griffiths elliptical crack with tensile extension wings (after Hoek and Bieniawski, 1965) compared to the sliding crack model (after Kemeny and Cook, 1991). (b) side view of proposed fracture geometry, (c) plan view and (d) 3D view. 271

Figure 5.22  (a) All fault planes (b) PCA derived planes during localization Period C. .................273

Figure 5.23  Visualization of fracture network based on fault plane solutions for (a) Sections cut through all fault planes (b) Sections cut through fault planes recorded during localization (Period C) .................................................................................................................275

Figure 5.24a & b  Visualization of fracture network based on PCA derived planes for (a) Sections cut through PCA derived planes for Period A – pre-interaction and (b) Period B – Interaction to Localization. .................................................................................................................276

Figure 5.25  (a) Mapping of a normal fault in South Africa (After Gay and Ortlepp, 1979, (b) Location of the normal shear in relation to mining front (c) Rotated image of PCA derived planes on Section 10480 E, image rotated clockwise 90° about the B-axis ($\sigma_2$ –axis) to align stresses with the Gay and Ortlepp’s fault. .................................279
Figure 5.26  (a) Diagram of the Riedel experiment, (R) Riedel Shears, (R') conjugates Riedel shears (after Tchalenko, 1970). (b) Discrete Riedel Shear fractures observed in clay experiments (i.e. $\Phi$ is internal angle of friction) and Skempton's principal displacement fracture connecting Riedel shears (after Cho et. al., 2008) (c) Tectonic shear zone in diorite, UCS=212MPa, showing a natural shear zone (after Cho et. al., 2008), and (d) Dasht-e Bayaz earthquake fault (after Tchalenko, 1970). ......................................................................................280

Figure 5.27  Development of a shear fracture for a 1.85 MPa normal stress test on PFC synthetic rock (after Cho et. al., 2008)...........................................................................................................281

Figure 5.28  (a) Fracture pattern developed at each stage of stress-strain curve on overconsolidated clay (after Tchalenko, 1970 modified by Cho et. al., 2008). At peak shear strength (Stage A), the first Riedel shear appear at 12° to the horizontal. (Stage B) Riedel shear are extended and a few new shears are generated at about 8° to horizontal. (Stage C) New shears named "P shears" appear at an inclination of –10°. (Stage D) Principal displacement surface is formed. (b) PCA derived planes indicating direction of shear development at localization. (c) Fault plane solution derived planes indicating direction of fractures at localization...................................................................................................................282

Figure A1.1  Definition of the clustering index, (a) definition of the clustering index function (CIf), and (b) and classification of cracks according to their interaction based on their inter-distance ($d_i$) (after Falmagne 2002; Reyes-Montes 2004), and (c) use and definition of the clustering index at the URL, comparison of clustering events ($C_i \geq 0.5$) to surveyed notch outlines using the apparent radius model (after Falmagne, 2002)........................................................................322

Figure A1.2  Microseismicity recorded in the Williams mine sill pillar region analysed in this study from September 1999 to February 2005. The sill region analysed here is bounded from stope 28 to stope 16, and analysis slices 1 to 6 are 40 m wide spanning two stopes. Approximately 33,000 events where recorded and located during this time frame. Also, shown is longitudinal grid plane at section 9860N (b), used for seismic density plotting in section 4.1. .........................323

Figure A1.3  WM Sept 1999 - Feb 2005 – Seismic Density Section 9433.75E, Voxel Size = 15.6 m$^3$ (2.5x2.5x2.5 m) (a) location of grid plane,(b) pixelation plot of events density and (c) contoured events on the grid plane................................................................................................324

Figure A1.4  WM Sept 1999 - Feb 2005 – Seismic Density Section 9432.5E, Voxel Size = 125 m$^3$ (5x5x5 m) (a) location of grid plane,(b) pixelation plot of events density and (c) contoured events on the grid plane. .................................................................324

Figure A1.5  WM Sept 1999 - Feb 2005 – Seismic Density Section 9430.0E, Voxel Size = 1000 m$^3$ (10x10x10 m) (a) location of grid plane,(b) pixelation plot of events density and (c) contoured events on the grid plane. ...........................................................................................................325

Figure A1.6  WM Sept 1999 - Feb 2005 – Seismic Density Section 9437.5E, Voxel Size = 15625 m$^3$ (25x25x25 m) (a) location of grid plane,(b) pixelation plot of events density and (c) contoured events on the grid plane......................................................................................325

Figure A1.7  Comparison of source radius based on (a) Madariaga and (c) Apparent radius to (b)&(d) Tensile Radius. Data from adjacent Golden Giant Mine (Coulson, 2008b). .........................326

Figure A1.8  (a) Representation of worst case event location in a 125 m$^3$ voxel. (b) Calculation of clustering index (CIf) for a 5 x 5 x 5 m cube (125 m$^3$) with 5 events (blue line CIf for centre event and red line CIf for corner events), and 4 events only (green line CIf for corner events). (c) Contour of event density for 125 m$^3$ voxels with the lower contour limit set to 5 events and (d) 5 event per voxel iso-surface of all events recorded in the sill region. .........................................................327

Figure A1.9  WM Sept 1999 - Feb 2005 – Seismic Density Section 9432.5E, Voxel Size = 1 m$^3$ (1x1x1 m) (a) location of grid plane,(b) pixelation plot of events density and (c) contoured events on the grid plane. .................................................................328
Figure A2.10 Principal Components Analysis Technique. (a) determination of optimum spatial window size, D, based on the distribution of the cluster of events hypocentre inter-distances in Euclidian space, using the a Normal Distribution Cumulative Density Function. (b) Determination of the mean hypocentral location of the events that fall within the Spatial Window, situated on the event of interest, and (c) calculation of the spread matrix describing the variance of the hypocentres location to the mean hypocentral location, and (d) application of the principal components analysis through eigenvector and eigenvalue decomposition to produce and ellipsoid with strike and dip determined for the overall trend of the events surrounding the event of interest. (e) Application of the temporal sliding window, in the example shown, N’ is 50. ...............................................................................................................335

Figure A2.11 Cluster of events above the 9390 L main footwall haulage drive, for PCA parameter analysis ’Slice 2’, for all 2003 events (Coulson, 2008a)................................................................................................................................................336

Figure A2.12 Effect of varying the size of the Spatial Window D, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the determined PCA planes and their orientation and distribution when plotted on a lower hemisphere stereographic projection, for D values of 5, 10, 15, 20, 23.4 and 50 m. .................................................................................................................................337

Figure A2.13 (a) Effect of varying the size of the Spatial Window D, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the temporal variation in strike of the determined PCA planes for each event. Note the temporal window size is fixed at 50 events, and the strike is plotted using smoothing, based on a 25 event moving window. (b), 25 point moving Standard Deviation during the major change in the predominant orientation of the PCA planes for varying Spatial Dimensions. (c) Relationship of overall (averaged) standard deviation, standard errors, ellipticity (x10) and F-parameter for the entire time period versus Spatial Window Dimension, D. At a D value of >20 m the variance starts to plateaux, and the ellipticity is at a minima...................................................................................................................338

Figure A2.14 Effect of varying the size of the Temporal Window, N’, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the determined PCA planes and their orientation and distribution when plotted on a lower hemisphere stereographic projection, for N’ values of 10, 25, 50, 100, 250 and 500 events.............................................................................................................................................339

Figure A2.15 Effect of varying the size of the Temporal Window, N’, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the temporal variation in strike of the determined PCA planes for each event. Note the Spatial Window size is fixed at 20 m, and the strike is plotted using smoothing, based on a 25 event moving window. (b), 25 point moving Standard Deviation during the major change in the predominant orientation of the PCA planes for varying Temporal Window sizes. (c) Relationship of overall (averaged) standard deviation, standard errors, Ellipticity and F-parameter for the entire time period versus Temporal Window Size, N’. At a N’ value of >50 events the variance and average ellipticity starts to plateaux........340

Figure A2.16 Effect of varying the temporal window shift (temporal window size fixed at 50 events) (a) to (d) on the stability of the PCA derived planes and (e) the temporal change in orientation of the strike of the planes with time using a 25 point moving average to smooth the data. ........341

Figure A3.17 Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level (Ω), corner frequency, (fc) and energy flux, (Jc), fitted with a Brune f^2 spectral decay.......349

Figure B1.1 Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from September 29, 1999 to March 31, 2000. A total of 6,651 events. Section 9860 N used for seismic density contours shown in (b). ..............................................................................................................................................352

xxii
Figure B1.2  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from April 1, 2000 to December 31, 2000. A total of 6,662 events. Section 9860 N used for seismic density contours shown in (b). .........................................................352

Figure B1.3  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2001 to December 31, 2001. A total of 5,301 events. Section 9860 N used for seismic density contours shown in (b). .........................................................353

Figure B1.4  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2002 to December 31, 2002. A total of 3,876 events. Section 9860 N used for seismic density contours shown in (b). .........................................................353

Figure B1.5  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2003 to December 31, 2003. A total of 8,721 events. Section 9860 N used for seismic density contours shown in (b). .........................................................354

Figure B1.6  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2004 to December 31, 2004. A total of 1,830 events. Section 9860 N used for seismic density contours shown in (b). .........................................................354

Figure B1.7  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 1 Events (1587). Note mining geometry is fixed at Dec 1999. ..........................................................356

Figure B1.8  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 2 Events (2633). Note mining geometry is fixed at Dec 1999. ..........................................................356

Figure B1.9  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 3 Events (2757). Note mining geometry is fixed at Dec 1999. ..........................................................357

Figure B1.10 Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 4 Events (2485). Note mining geometry is fixed at Dec 1999. ..........................................................357

Figure B1.11 Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 5 Events (2815). Note mining geometry is fixed at Dec 1999. ..........................................................358

Figure B1.12 Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 6 Events (899). Note mining geometry is fixed at Dec 1999. ..........................................................358

Figure B1.13 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 1 Events (954). Note mining geometry is fixed at Dec 1999. ..........................................................359

Figure B1.14 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 2 Events (2038). Note mining geometry is fixed at Dec 1999. ..........................................................359

Figure B1.15 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 3 Events (1254). Note mining geometry is fixed at Dec 1999. ..........................................................360

Figure B1.16 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 4 Events (554). Note mining geometry is fixed at Dec 1999. ..........................................................360

Figure B1.17 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 5 Events (383). Note mining geometry is fixed at Dec 1999. ..........................................................361

Figure B1.18 Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 6 Events (87). Note mining geometry is fixed at Dec 1999. ..........................................................361

Figure B1.19 Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 1 Events (586). Note mining geometry is fixed at Dec 1999. ..........................................................362

Figure B1.20 Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 2 Events (1775). Note mining geometry is fixed at Dec 1999. ..........................................................362

Figure B1.21 Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 3 Events (990). Note mining geometry is fixed at Dec 1999. ..........................................................363

Figure B1.22 Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 4 Events (266). Note mining geometry is fixed at Dec 1999. ..........................................................363
Figure B1.23  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 5 Events (130) . Note mining geometry is fixed at Dec 1999.................................................................364
Figure B1.24  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 6 Events (110) . Note mining geometry is fixed at Dec 1999.................................................................364
Figure B1.25  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 1 Events (733) . Note mining geometry is fixed at Dec 1999.................................................................365
Figure B1.26  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 2 Events (3719) . Note mining geometry is fixed at Dec 1999.................................................................365
Figure B1.27  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 3 Events (3356) . Note mining geometry is fixed at Dec 1999.................................................................366
Figure B1.28  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 4 Events (577) . Note mining geometry is fixed at Dec 1999.................................................................366
Figure B1.29  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 5 Events (189) . Note mining geometry is fixed at Dec 1999.................................................................367
Figure B1.30  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 6 Events (117) . Note mining geometry is fixed at Dec 1999.................................................................367
Figure B1.31  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 1 Events (227) . Note mining geometry is fixed at Dec 1999.................................................................368
Figure B1.32  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 2 Events (823) . Note mining geometry is fixed at Dec 1999.................................................................368
Figure B1.33  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 3 Events (553). Note mining geometry is fixed at Dec 1999.................................................................369
Figure B1.34  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 4 Events (124). Note mining geometry is fixed at Dec 1999.................................................................369
Figure B1.35  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 5 Events (61) . Note mining geometry is fixed at Dec 1999.................................................................370
Figure B1.36  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 6 Events (80) . Note mining geometry is fixed at Dec 1999.................................................................370
Figure B2.1  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes............................................372
Figure B2.2  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes............................................372
Figure B2.3  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2002 to Dec 2002. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes............................................373
Figure B2.4  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2003 to Dec 2003. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes............................................373
Figure B2.5  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes............................................374
Figure B2.6  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes............................................375
Figure B2.7 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2 Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes. 375

Figure B2.8 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Jan 2002 to Dec 2002. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes. 376

Figure B2.9 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Jan 2003 to Dec 2003. (a) Long section, (b) View west and (c) Plan view of slice 2 events. (d) PCA derived planes. 376

Figure B2.10 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes. 377

Figure B2.11 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes. 378

Figure B2.12 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes. 378

Figure B2.13 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2002 to Dec 2002. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes. 379

Figure B2.14 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2003 to Dec 2003. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes. 379

Figure B2.15 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes. 380

Figure B2.16 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes. 381

Figure B2.17 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes. 381

Figure B2.18 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2002 to Dec 2002. (a) Long section, (b) View west and (c) Plan view of slice 4 events. (d) PCA derived planes. 382

Figure B2.19 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2003 to Dec 2003. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes. 382

Figure B2.20 NO EVENTS - Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes. 383

Figure B2.21 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 5, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 5 events. (d) PCA derived planes. 384

Figure B2.22 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 5 Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 5 events. (d) PCA derived planes. 384

xxv
Figure B2.23  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 6, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 6 events. (d) PCA derived planes. .................................385

Figure B2.24  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 0; 12/26/1999, (b) Temporal Window = 1; 02/05/2000 (c) Temporal Window = 2; 04/25/2000 (d) Temporal Window = 3; 06/09/2000. Note date is based on last event in temporal window. ..................................................................................................387

Figure B2.25  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 4; 07/30/2000, (b) Temporal Window = 5; 08/08/2000 (c) Temporal Window = 6; 09/06/2000 (d) Temporal Window = 7; 12/17/2000. Note date is based on last event in temporal window. ..................................................................................................388

Figure B2.26  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 8; 03/09/2001, (b) Temporal Window = 9; 04/28/2001 (c) Temporal Window = 10; 06/12/2001 (d) Temporal Window = 11; 06/28/2001. Note date is based on last event in temporal window. ..................................................................................................389

Figure B2.27  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 12; 06/29/2001 Note jump in instrument 24_1 after 3.1 Mn event and slight change in seismicity, (b) Temporal Window = 13; 06/29/2001 (c) Temporal Window = 14; 07/01/2001 (d) Temporal Window = 15; 07/06/2001. Note date is based on last event in temporal window. ..................................................................................................390

Figure B2.28  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 16; 07/15/2001, (b) Temporal Window = 17; 08/06/2001 (c) Temporal Window = 18; 09/03/2001 (d) Temporal Window = 19; 10/03/2001. Note date is based on last event in temporal window. ..................................................................................................391

Figure B2.29  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 20; 11/30/2001 Note jump in instrument 24_1 at 4.5 m & change in seismicity, (b) Temporal Window = 21 12/30/2001 (c) Temporal Window = 22; 01/15/2002 (d) Temporal Window = 23; 01/23/2002. Note date is based on last event in temporal window. ..................................................................................................392

Figure B2.30  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 24; 02/18/2002, (b) Temporal Window = 25; 04/09/2002 (c) Temporal Window = 26; 05/04/2002 (d) Temporal Window = 27; 05/11/2002. Note date is based on last event in temporal window. ..................................................................................................393

Figure B2.31  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 28; 06/26/2002, (b) Temporal Window = 29; 08/24/2002 (c) Temporal Window = 30; 10/01/2002 (d) Temporal Window = 31; 11/21/2002. Note date is based on last event in temporal window. ..................................................................................................394

Figure B2.32  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 32; 12/26/2002, (b) Temporal Window = 33; 12/31/2002 Note jump in instrument 26_1 of 55 mm at 4.5m depth also instrument 24_1 and 24_2 fail at depth on the other side, and cause a drop in the number of planes with ellipticity for the next two windows (c) Temporal Window = 34; 01/12/2003 (d) Temporal Window = 35; 02/06/2003. Note the events plotted in these last two figures are all the events in the temporal window Ellip> 1.0. 395
Figure B2.33 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 36; 03/12/2003, (b) Temporal Window = 37; 04/09/2003 (c) Temporal Window = 38; 04/30/2003 (d) Temporal Window = 39; 05/11/2003, ........................................396

Figure B2.34 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 40; 05/14/2003, (b) Temporal Window = 41; 05/27/2003 (c) Temporal Window = 42; 05/30/2003 (d) Temporal Window = 43; 05/30/2003. Note date is based on last event in temporal window. .......................................................................................397

Figure B2.35 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 44; 05/31/2003, (b) Temporal Window = 45; 06/02/2003 (c) Temporal Window = 46; 06/10/2003 (d) Temporal Window = 47; 06/20/2003. Note date is based on last event in temporal window. .......................................................................................398

Figure B2.36 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 48; 07/07/2003, Note jump in instrument 26_2 of 16 mm at 9 m depth and point of disassociation for this cluster (b) Temporal Window = 49; 07/25/2003 (c) Temporal Window = 50; 08/28/2003 (d) Temporal Window = 51; 08/28/2003. Note date is based on last event in temporal window. .......................................................................................399

Figure B2.37 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 52; 08/28/2003, (b) Temporal Window = 53; 08/29/2003, Note drop in number of planes with ellipticity for this and the next window, same time as haulage is paste filled from stope 26 to 23(c) Temporal Window = 54; 08/29/2003 (d) Temporal Window = 55; 08/30/2003. Note date is based on last event in temporal window..................................................400

Figure B2.38 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 56; 09/03/2003, (b) Temporal Window = 57; 09/16/2003, Note on the Sept 13 the 3.5 Mn event occurred and cause a permanent change in the seismicity as the back above the 9390 became aseismic, this could have been the failure of the back onto paste? (c) Temporal Window = 58; 10/02/2003 (d) Temporal Window = 59; 11/020/2003. .............401

Figure B2.39 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 60; 12/31/2003, (b) Temporal Window = 61; 03/05/2004 (c) Temporal Window = 62; 04/15/2004 (d) Temporal Window = 63; 07/18/2004. Note date is based on last event in temporal window. .......................................................................................402

Figure B2.40 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 64; 01/31/2005, END OF ANALYSIS..........................................................403

Figure C1 a. Plan view of 9390 L sill pillar region, showing boundaries of seismic analysis slices and location of conventional instrumentation (SMART cables). b. Longitudinal view of conventional instrumentation and contours of depth of displacement that exceed 1 mm dilation, for three points in time, 12-2000, 12-2002 and 07-2003. Note instruments 17, 18, 19 and 20 showed negligible movement during the monitoring period. Also shown on (b.) depth of caving on 9415L as of 12-1999 following 2.6 Mn event (Yi, 1999) ................................................405

Figure C2 Instrument 27 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note : labelled dates are in mm-dd-yy format.....406

Figure C3 Instrument 26-1 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note : labelled dates are in mm-dd-yy format.....407
Figure C4 Instrument 26-2 SMART cable response (a) nodal displacement in mm relative to the Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....408

Figure C5 Instrument 25 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....409

Figure C6 Instrument 24-1 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....410

Figure C7 Instrument 24-2 SMART cable response (a) nodal displacement in mm relative to the Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....411

Figure C8 Instrument 23 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....412

Figure C9 Instrument 22 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....413

Figure C10 Instrument 21-1 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....414

Figure C11 Instrument 21-2 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....415

Figure C12 Instrument 20 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....416

Figure C13 Instrument 19 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....417

Figure C14 Instrument 18 SMART MPBX response (a) nodal displacement in mm relative to Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....418

Figure C15 Instrument 17 SMART MPBX response (a) nodal displacement in mm relative to Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....419

Figure C16 Instrument 16 SMART MPBX response (a) nodal displacement in mm relative to Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format....420

Figure D1.1 Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_0$), corner frequency, ($f_c$) and energy flux, ($J_0$), fitted with a Brune f $^{-2}$ spectral decay. .......424

Figure D1.2 Comparison of Automatic picking versus manual picking of p-wave arrivals for a representative 250 event sub-set of the data in the EOS cluster, spanning from May 2002 to May 2004. (a) Northing, (b) Easting, and (c) Depth (axis range for all is 30 m). (d) Online location error based on the simplex algorithm and source to sensor distance assuming a linear path. ...........................................................................................................................................425

xxviii
Figure D1.3  Comparison Automatic picking versus manual picking of p- and s- arrivals on source parameter calculations for the same previous data sub-set. (a) Moment magnitude ($R^2 = 0.82$), (b) Seismic moment ($R^2 = 0.87$), (c) Seismic energy ($R^2 = 0.94$), (d) $E_s/E_p$ ratio ($R^2 = 0.75$), (e) Source Radius ($R^2 = 0.85$) and (f) Apparent stress ($R^2 = 0.97$).

Figure D2.4  Comparison of source parameters automatically calculated with Triaxial sensors only versus Triaxial plus Uniaxial (note the uniaxial dominate the average), for (a) Moment Magnitude, $M_o$, (b) Radiated Seismic Energy, $E_o$, (c) $E_s/E_p$ ratio, and (d) Source Radius, $r_o$. (e) Temporal change of average seismic moment, $M_o$, calculated with uniaxial and triaxial sensors (compare to Figure D2.8a).

Figure D2.5  Comparison of Seismic Moment, ($M_o$) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials. Note for the data set not all triaxials are always used.

Figure D2.6  Comparison of Radiated Seismic Energy, ($E_o$) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials. Note for the data set not all triaxials are always used, this results in no change for events that did not already include the sensor excluded (i.e. 1:1 relationship results).

Figure D2.7  Comparison of Source Radius, ($r_o$) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials. Note for the data set not all triaxials are always used.

Figure D2.8  Comparison of temporal changes of the average source parameters for each 3 triaxial analysis dropping one sensor, using a 50 event moving average. (a) Seismic Moment, $M_o$, (b) Radiated Seismic Energy, $E_o$, and (c) Source Radius, $r_o$.

Figure E1  Golden Giant mine shaft pillar microseismicity, from June 22, 1994 to Dec 25, 1994. A total of 114 events. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25].

Figure E2  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1995 to Dec 31, 1995. A total of 661 events.

Figure E3  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1996 to July 07, 1996 (Note database missing events until the end of 1996). A total of 232 events.

Figure E4  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1997 to Nov 20, 1997 (Note: system had triggering issues –random triggers for Nov to Dec 1997). The development of the new East Return Air Raise is captured and location within 8 m. A total of 927 events.

Figure E5  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1998 to Dec 31, 1998. A total of 1819 events.

Figure E6  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1999 to Dec 31, 1999. A total of 1415 events.

Figure E7  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2000 to Dec 31, 2000. A total of 1175 events.

Figure E8  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2001 to Dec 31, 2001. A total of 1141 events.

Figure E9  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2002 to Dec 31, 2002. A total of 7358 events. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25].

Figure E10  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2003 to Dec 31, 2003. A total of 13,034 events.

Figure E11  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2004 to Dec 31, 2004. A total of 4527 events.

Figure E12  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2005 to Sept 31, 2005. A total of 5720 events.
Figure E13  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2005 to May 31, 2005. This is a partial breakdown of the previous figure to highlight the aseismicity directly below the 4620 L which started after the point of disassociation and May was the last period that events occurred in the EOS cluster around 460-S2 stope. A total of 3682 events. ..................................441

Figure E14  Golden Giant mine shaft pillar microseismicity, from June 1, 2005 to Sept 23, 2005. Note before the 460-S2 stope is mined this region becomes completely aseismic. A total of 2039 events....................................................................................................................................441

Figure F1  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = -8; 04/01/2002, (b) Temporal Window = -7; 04/21/2002 (c) Temporal Window = -6; 05/05/2002 (INITIATION) (d) Temporal Window = -5; 05/23/2002. Note date is based on last event in temporal window. ........443

Figure F2  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = -4; 07/05/2002, (b) Temporal Window = -3; 09/14/2002 (c) Temporal Window = -2; 11/30/2002 (d) Temporal Window = -1; 12/27/2002. Note date is based on last event in temporal window. ..................444

Figure F3  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 1; 02/12/2003, (b) Temporal Window = 2; 02/15/2003 (c) Temporal Window = 3; 02/17/2003 (INTERACTION) (d) Temporal Window = 4; 02/17/2003. Note date is based on last event in temporal window. .........445

Figure F4  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 5; 02/18/2003, (b) Temporal Window = 6; 02/20/2003 (c) Temporal Window = 7; 02/22/2003 (d) Temporal Window = 8; 02/23/2002. Note date is based on last event in temporal window............446

Figure F5  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 9; 02/24/2003, (b) Temporal Window = 10; 02/27/2003 (c) Temporal Window = 11; 02/28/2003 (d) Temporal Window = 12; 03/02/2003. Note date is based on last event in temporal window..........................447

Figure F6  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 13; 03/05/2003 (COALESCENCE AND LOCALIZATION), (b) Temporal Window = 14; 03/08/2003 (c) Temporal Window = 15; 03/13/2003 (d) Temporal Window = 16; 03/18/2003. Note date is based on last event in temporal window..........................448

Figure F7  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 17; 03/25/2003, (b) Temporal Window = 18; 04/01/2003 (c) Temporal Window = 19; 04/07/2003 (d) Temporal Window = 20; 04/16/2003. Note date is based on last event in temporal window..........................449

Figure F8  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 21; 04/28/2003, (b) Temporal Window = 22; 05/16/2003 (c) Temporal Window = 23; 05/26/2003 (d) Temporal Window = 24; 06/03/2003. Note date is based on last event in temporal window..........................450

Figure F9  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 25; 06/14/2003, (b) Temporal Window = 26; 06/30/2003 (DISASSOCIATION) (c) Temporal Window = 27; 07/17/2003 (d) Temporal Window = 28; 08/24/2003. Note date is based on last event in temporal window. ..................................................451

Figure F10  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 29; 09/29/2003, (b) Temporal Window = 30; 10/18/2003 (c) Temporal Window = 31; 11/17/2003 (d) Temporal Window = 32; 12/30/2003. Note date is based on last event in temporal window..............452

xxx
Figure F11  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 33; 02/13/2004, (b) Temporal Window = 34; 03/29/2004 (c) Temporal Window = 35; 04/24/2004 (d) Temporal Window = 36; 06/09/2004. Note date is based on last event in temporal window

Figure F12  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 37; 08/31/2004, (b) Temporal Window = 38; 11/17/2004 (c) Temporal Window = 39; 11/30/2004 (d) Temporal Window = 40; 12/07/2004. Note date is based on last event in temporal window

Figure F13  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 41; 12/12/2004, (b) Temporal Window = 42; 12/30/2004 (c) Temporal Window = 43; 01/31/2005 (d) Temporal Window = 44; 02/12/2005. Note date is based on last event in temporal window

Figure F14  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 45; 03/19/2005, (b) Temporal Window = 46; 04/05/2005 (c) Temporal Window = 47; 04/18/2005 (d) Temporal Window = 48; 05/04/2005. Note date is based on last event in temporal window

Figure F15  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 49; 05/23/2005, (b) Temporal Window = 50; 09/04/2005. Note date is based on last event in temporal window
LIST OF TABLES

<p>| Table 3.1  | Summary of Main Joint Set and Properties (after Kazakidis, 1990 and Coulson, 2004) | 49      |
| Table 3.2  | Summary of Mechanical Testing Performed at the Williams and Golden Giant | 51      |
| Table 3.3  | Summary of Rock Mass Classification At Williams Mine (Based on mapping at Williams, Golden Giant and David Bell Mines) | 51      |
| Table 3.4  | Large Magnitude Events (&gt; 0 Mn) Recorded at Williams Mine in the Sill, Block 4 Central (B4-C) and other parts of the Mine (Period of Monitoring 1999 to Feb. 2005). Events that occurred in the analysis region of the sill, are in bold type. | 60      |
| Table 3.5  | Summary of PCA major planes for Williams sill based on a yearly analysis of clustering events for footwall clusters above 9390L | 76      |
| Table 3.6  | Summary of SMART Cable and SMART MPBX displacements over the monitoring period for the 9390L haulage between stopes 16 and 27 | 91      |
| Table 3.7  | Failure Criterion and Parameters Used in Modelling. (Rock mass parameters based on a GSI = 60) | 120     |
| Table 3.8  | Comparison of damage levels using the Hoek-Brown Brittle parameters based on linear elastic modelling between for Slices 1 to 4 (\sigma_c = 175) MPa, (m = 0) (note: (\sigma_1=\sigma_3+(m\sigma_3+\sigma_c)^2)^{0.5})) | 123     |
| Table 4.1  | Main Joint Sets and Properties (after Kazakidis, 1990 and Coulson, 2004) | 155     |
| Table 4.2  | Comparison of determined damage levels using the Hoek-Brown Brittle parameters based on linear elastic modelling between Golden Giant mine and Williams mine Slice 2, Sec 9435E (Chapter 3) (\sigma_c = 175) MPa, (m = 0) (note: (\sigma_1=\sigma_3+(m\sigma_3+\sigma_c)^2)^{0.5})) | 188     |
| Table 4.3  | Statistical summaries of source parameters for the EOS cluster | 197     |
| Table 5.1  | Summary of event filtering for determination of good quality fault plane solutions at the core of the EOS cluster | 235     |
| Table 5.2  | Comparison of EOS Cluster Population Statistics to Extracted Sub-set for Fault Plane Solutions | 236     |
| Table 5.3  | Comparison of Linear Elastic Induced Principal Stress Orientations with Overall Stress Axis Orientations Determined from Fault Plane Solutions | 240     |
| Table 5.4  | Comparisons of Overall Fault Plane Orientations with PCA and Geology (Strike/R/Dip) | 242     |
| Table 5.5  | Summary of focal mechanism type determined based on the most probable fault plane solutions for the various temporal periods | 248     |
| Table 5.6  | Summary of Non-linear Parameters Analysed | 265     |</p>
<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>Modified Tresca gradient coefficient</td>
</tr>
<tr>
<td>AE</td>
<td>Acoustic Emissions</td>
</tr>
<tr>
<td>$A_0$</td>
<td>Signal amplitude at the source</td>
</tr>
<tr>
<td>$\alpha$</td>
<td>P-wave velocity</td>
</tr>
<tr>
<td>a</td>
<td>H-B parabolic index</td>
</tr>
<tr>
<td>$a_{\text{max}}$</td>
<td>Maximum acceleration rms trace</td>
</tr>
<tr>
<td>$\beta$</td>
<td>S-wave velocity</td>
</tr>
<tr>
<td>$\beta$</td>
<td>Angle from crack plane</td>
</tr>
<tr>
<td>B</td>
<td>Null axis (FPS)</td>
</tr>
<tr>
<td>B</td>
<td>Modified Tresca intercept coefficient</td>
</tr>
<tr>
<td>C</td>
<td>Cohesion (intact)</td>
</tr>
<tr>
<td>$C_m$</td>
<td>Cohesion peak strength (rock mass)</td>
</tr>
<tr>
<td>$C_r$</td>
<td>Cohesion residual strength (rock mass)</td>
</tr>
<tr>
<td>CDF</td>
<td>Cumulative distribution function</td>
</tr>
<tr>
<td>CMPCA</td>
<td>Continuous central moving principal component analysis</td>
</tr>
<tr>
<td>CLVD</td>
<td>Compensated linear vector dipole (complex shear)</td>
</tr>
<tr>
<td>Clf</td>
<td>Cluster index function</td>
</tr>
<tr>
<td>Cli</td>
<td>Cluster index</td>
</tr>
<tr>
<td>c</td>
<td>Cohesion</td>
</tr>
<tr>
<td>$c$</td>
<td>Wave velocity</td>
</tr>
<tr>
<td>c</td>
<td>Extension crack length</td>
</tr>
<tr>
<td>$c_0$</td>
<td>Crack diameter</td>
</tr>
<tr>
<td>$c_p$</td>
<td>P-wave velocity</td>
</tr>
<tr>
<td>$c_s$</td>
<td>S-wave velocity</td>
</tr>
<tr>
<td>$\Delta\sigma$</td>
<td>Static stress drop</td>
</tr>
<tr>
<td>$\Delta\sigma_d$</td>
<td>Dynamic stress drop</td>
</tr>
<tr>
<td>D</td>
<td>Diameter of spatial window</td>
</tr>
<tr>
<td>D</td>
<td>H-B disturbance factor</td>
</tr>
<tr>
<td>$D$</td>
<td>Maximum seismic displacement</td>
</tr>
<tr>
<td>DEM</td>
<td>Discrete element method</td>
</tr>
<tr>
<td>dil</td>
<td>Unidirectional dilation (bulking) or displacement measure from SMART instruments</td>
</tr>
<tr>
<td>Dil</td>
<td>Dilation parameter (Phase2) Volumetric dilation</td>
</tr>
</tbody>
</table>

---

xxxiii
<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>$d_{ij}$</td>
<td>Crack or event separation, inter-event distance</td>
</tr>
<tr>
<td>$E$</td>
<td>Young’s modulus</td>
</tr>
<tr>
<td>$E_i$</td>
<td>Young’s modulus (intact)</td>
</tr>
<tr>
<td>$E_{rm}$</td>
<td>Young’s modulus (rock mass)</td>
</tr>
<tr>
<td>$E_{li}$</td>
<td>Ellipticity</td>
</tr>
<tr>
<td>EMR</td>
<td>Electromagnetic radiation</td>
</tr>
<tr>
<td>EOS</td>
<td>East of Slot</td>
</tr>
<tr>
<td>$E_o$</td>
<td>Total radiated seismic energy</td>
</tr>
<tr>
<td>$E_s$</td>
<td>S-wave energy</td>
</tr>
<tr>
<td>$E_p$</td>
<td>P-wave energy</td>
</tr>
<tr>
<td>$F$</td>
<td>F-parameter (PCA)</td>
</tr>
<tr>
<td>$F_c$</td>
<td>Seismic radiation pattern coefficient</td>
</tr>
<tr>
<td>FEM</td>
<td>Finite element method</td>
</tr>
<tr>
<td>FPS</td>
<td>Fault plane solutions</td>
</tr>
<tr>
<td>FOS</td>
<td>Factor of Safety</td>
</tr>
<tr>
<td>$f_c$</td>
<td>Corner frequency</td>
</tr>
<tr>
<td>GSI</td>
<td>Geological Strength Index</td>
</tr>
<tr>
<td>H-B</td>
<td>Hoek- Brown failure criteria</td>
</tr>
<tr>
<td>Ja</td>
<td>Joint alteration</td>
</tr>
<tr>
<td>$J_c$</td>
<td>Energy flux</td>
</tr>
<tr>
<td>Jn</td>
<td>Joint number</td>
</tr>
<tr>
<td>Jr</td>
<td>Joint roughness</td>
</tr>
<tr>
<td>$k$</td>
<td>Maximum far field horizontal to vertical stress ratio</td>
</tr>
<tr>
<td>K</td>
<td>Number of events inside spatial window</td>
</tr>
<tr>
<td>$K_c$</td>
<td>Madariaga model coefficient</td>
</tr>
<tr>
<td>L1</td>
<td>Absolute deviation estimator of arrival time residuals</td>
</tr>
<tr>
<td>L2</td>
<td>Least squares estimator of arrival time residuals</td>
</tr>
<tr>
<td>LEFM</td>
<td>Linear elastic fracture mechanics</td>
</tr>
<tr>
<td>$\lambda_1, \lambda_2, \lambda_3$</td>
<td>Eigenvalues: maximum, intermediate and minimum</td>
</tr>
<tr>
<td>$\mu$</td>
<td>Mean</td>
</tr>
<tr>
<td>$\mu$</td>
<td>Bulk modulus</td>
</tr>
<tr>
<td>M-C</td>
<td>Mohr-Coulomb failure criteria</td>
</tr>
<tr>
<td>M</td>
<td>Moment magnitude</td>
</tr>
<tr>
<td>$M_n$</td>
<td>Nuttli magnitude recorded by Geological Survey of Canada</td>
</tr>
<tr>
<td>$M_r$</td>
<td>Richter magnitude</td>
</tr>
<tr>
<td>$M_o$</td>
<td>Seismic moment</td>
</tr>
</tbody>
</table>
MPBX  Multiple point borehole extensometer
MTI   Moment tensor inversion
m    H-B coefficient
mb   H-B coefficient peak strength (rock mass)
i    H-B coefficient (intact)
i    H-B coefficient residual strength (rock mass)
N    Total number of events in cluster
N    Normal bond strength (force)
N'   Total number of events in temporal window
NGI  Norwegian geotechnical institute
n    Total number of inter-distances
\hat{n}_1, \hat{n}_2, \hat{n}_3 Eigenvectors: maximum, intermediate and minimum
\nu   Poisson’s ratio
\Omega_o Low frequency spectral level or plateau,
PCA  Principal components analysis
PFC  Particle flow code
P-   Seismic compressional wave
P    Pressure axis (FPS)
\Phi  Friction angle
\Phi_m  Friction angle peak strength (rock mass)
\Phi_r  Friction angle residual strength (rock mass)
Q    Tunneling quality index
Q'   Modified rock mass quality index
Q    Seismic signal quality factor
R    Source to sensor distance
RMR  Rock Mass Rating
RQD  Rock Quality designation
\rho  Density
r_{oi} Crack radius
r_o    Source radius (Madariaga)
r_a    Apparent source radius (Cai)
r'_a   Asperity radius
rms   Root mean squared
\sigma Standard deviation
\sigma_1 Maximum principal stress
$\sigma_2$  Maximum principal stress  
$\sigma_3$  Maximum principal stress  
$\sigma_a$  Apparent stress  
$\sigma_c$  Uniaxial compressive strength (intact)  
$\sigma_{ci}$  Crack initiation stress (intact)  
$\sigma_{cd}$  Critical crack damage stress, crack interaction and coalescence (intact)  
$\sigma_f$  Peak strength stress (intact)  
$\sigma_{cr}$  Residual strength plateaux (intact)  
$\sigma_n$  Normal stress to plane  
$\sigma_{ci}'$  Crack initiation stress (rock mass)  
$\sigma_{cd}'$  Critical crack damage stress, crack interaction and coalescence (rock mass)  
$\sigma'_f$  Peak strength stress (rock mass)  
$\sigma_{cr}'$  Residual strength plateaux (rock mass)  
$S_{ij}$  Covariance matrix tensor, spread matrix (PCA)  
$S$  Shear bond strength (force)  
$S$-  Seismic shear wave  
S.D.  Standard Deviation  
SMART  Displacement instrumented cable bolt  
SRF  Stress reduction factor  
s  H-B coefficient  
$s_i$  H-B coefficient (intact)  
$s_i'$  H-B coefficient (rock mass)  
T  Tension axis (FPS)  
T  Tensile strength  
TSX  Tunnel sealing experiment  
UCS  Uniaxial compressive strength  
URL  Underground Research Laboratory (Pinawa, Manitoba, Canada)  
$\Delta V/V$  Volumetric strain  
$V_a$  Apparent volume  
$v_{max}$  Maximum recorded velocity  
$\overline{X}_m$  Mean epicentral/hypocentral location  
$\psi$  Critical crack orientation
CHAPTER 1

INTRODUCTION

1.1 Overview

A significant amount of research has been published that focuses on the properties and behaviour of small laboratory samples of hard rock up to the peak strength and well into the post-peak failure region. Similarly, much research has focused on the bulk properties of \textit{in situ} rock masses up to the peak strength. In contrast, there has been comparatively little work in the literature that deals with the field-scale properties and behaviour of hard rock after significant damage or failure (i.e., the post-peak behaviour). This is largely due to the expense and technical difficulty in performing large controlled compressive tests at the rock mass scale, (Hoek & Diederichs 2006), and from a Civil engineering standpoint, failure is considered just after the peak strength, when the material no longer fulfills its engineering function (Jaeger and Cook 1979; Castro 1996).

In the hard rock mining industry, at high stresses, developed through either late stage extraction, or mining at depth, the rock mass is often driven not just to the peak strength but often well into the post-peak until ‘complete failure’ occurs. With the incorporation of microseismic systems into many mines around the world, and the advances in providing inexpensive field instrumentation, there is a unique opportunity to study this stress driven rock mass failure into the post-peak, in the ‘field scale laboratory’. Close to excavation openings under low confining stress, local failure of the rock mass can lead to failure of the support and caving of the excavation. In pillar regions under greater confinement, progressive failure results in microseismic ($< 0 \text{Mn}^1$) and seismic ($> 0 \text{Mn}$) events, which can be damaging to nearby underground openings. Complete regional failure is often noted when the rock mass becomes ‘aseismic’, (i.e. with increases in induced elastic stress there is little or no seismic activity recordable at the energy level of the fracturing). Generally, once this state has been reached mining can progress easily. The main issues evolve around determining when the rock mass starts to yield, when is the peak strength reached and whether the rock mass has truly ultimately failed, and what is the post peak behaviour of the rock mass. Identifying this is

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$^1$ Mn = Nuttli Magnitude scale, used by the Geological Survey of Canada
important from an operational standpoint in making valid production decisions regarding sequencing of stope extraction. Our current predictive tools for complex mine models are generally based on linear elastic modelling and calibration to observational data (Diederichs et al., 2002), and are based on historical seismic histories for interpretation of the likelihood of reoccurrence of seismicity. These can over estimate the strength of the rock mass in regions that may have already failed (Coulson et al. (2002). In the research presented in this thesis we are largely interested in determining at what point this regional rock mass failure, (i.e. failure of a confined region), occurs.

In order to use the microseismicity generated during regional rock mass failure, to better understand the behaviour during the failure process, analogies are drawn from the laboratory testing of intact rock specimens in which acoustic emissions (AE) of microcrack development have been recorded. Although there is much controversy regarding the scale effect on the strength of rock, it is the author’s belief, based on this research, that the fundamental mechanics of failure at various scales are similar. It is, however, the magnitude and amplitude of the deformations that are scale dependent.

1.2 Laboratory Scale Strength Testing and Acoustic Emissions (AE)

Much has been learned regarding the brittle failure process of hard rock from laboratory testing of intact samples, (Brace et al., 1966; Wawersik & Brace, 1971; Hoek & Brown, 1980; Bieniawski, 1967; Mogi, 1966; and Martin, 1997), and more recently combined with analysis of acoustic emissions, (Scholz, 1968; Lockner, 1993; Falls, 1993; Eberhardt, 1998; Pettitt, 1998; Thompson et al. 2006). Based on the research of Martin (1997) illustrated in Figure 1.1 and using the work of Lockner (1993), performed on a laboratory triaxial test to illustrate development of microcracks, three key stress related stages in the pre-peak to peak failure process have been observed and are well recognised (Figure 1.2).

Following crack closure, and the true linear elastic deformation phase of the material, generally at a stress level of 0.4±0.1 of \( \sigma_f \), (intact peak strength), crack initiation, (\( \sigma_{ci} \) – Stage 1), has been observed to occur, associated with an increase in the background level of acoustic emission (AE). This is followed by a stage of stable crack propagation, resulting in damage accumulation, up to the critical damage stress, (\( \sigma_{cd} \) – Stage 2), corresponding to the point of volumetric strain reversal and generally found at a stress level of 0.75±0.1 of \( \sigma_f \). After this stage unstable fracture propagation occurs with a significant increase in AE. This has been identified as the point of ‘true yield’, when axial strains become permanent, (Martin, 1997), and was
termed the long–term strength as defined by Bieniawski (1967). At this point the crack density is such that cracks have started to interact causing macroscale dilatancy, but are generally relatively randomly distributed through the core of the sample (Figure 1.2 – point a). On attaining the peak strength, (σ_f –Stage 3), these cracks start to coalesce to a macrofracture structure, (Figure 1.2 – point b), and as volumetric dilation occurs in brittle rocks strong strain softening behaviour is exhibited.

In the post peak this strain softening behaviour is characterised with increased formation of strain localizations to a failure plane or macrofracture structure on which sliding is exhibited, the microcracks recorded by the AE being strongly localized to this failure plane, (Figure 1.2 – points c to f). As the stress state progresses further into the post-peak, and approaching the residual strength, these cracks dissipate from the core region, the AE becoming ‘aseismic’ at the core in an analogy to the rock mass microseismicity (Figure 1.2 point g). Here, in this research, this has been called the point of disassociation, (i.e. disassociation of the AE or microseismic events from the predominant failure plane, resulting changes in the spatial distribution of microseismic events), and is thought to occur as the main failure region undergoes significant dilation, resulting in local strain softening, and cessation of AE in the region and potential activation of conjugate shear planes.

At present, little research at the laboratory scale has been performed with the measurement of acoustic emissions into the post-peak, all the way to the residual strength. Thompson et. al. (2006), performed a number of tests on Westerly granite using a giga RAM recorder, which was able to continuously record AE full wave form data on a 40 GB buffer over a 268 s window, at a sampling frequency of 5 MHz as well as triggered by AE. The advantage of the increased recording rate was to be able to better observe the post-peak AE in which previous triggered systems result in a loss of data due to saturation. The main experiment, performed all the way to the residual, used AE feedback load control to back off the load, (using a servo-controlled testing machine), creating quasi-static conditions. Of interest is the fact that, at the drop in peak strength, localization and coalescence of events occurs to a macrofracture structure, causing strain-softening behaviour, and in this experiment the localization was not confined to a single feature, but in the post peak, a secondary feature formed through the core of the sample resulting in more substantial strain softening behaviour (Figure 1.3). In two other experiments of slow and fast loading, used to observe the macrofracture fracture propagation speed, and more realistic conditions for rock failure in the field, the formation of the macrofracture could be observed. Very shortly into the post peak, however, the system still suffered from saturation of the recorder and location of AE were not possible.
Figure 1.1 The three parameters determined from laboratory triaxial compression tests: crack initiation ($\sigma_{ci}$); crack damage ($\sigma_{cd}$), and peak strength ($\sigma_f$), $\Delta V/V$, volumetric strain (after Martin 1997) and superimposed typical cumulative AE response (after Eberhardt 1998).
Figure 1.2 Triaxial testing of a sample of Westerly Granite ($\sigma_3 = 50$ MPa), monitored with acoustic emissions (AE) showing the development of a failure plane through localization of cracks (AE). At point (a) interaction, (b) coalescence and localization of cracks to a macrofracture structure, (c) to (f), strong strain-softening behaviour occurs, (g) approaching the residual strength (AE) no longer as strongly localized (after Lockner et al., 1992; Martin, 1997).
Figure 1.3 Triaxial testing of Westerly Granite using AE feedback control (i) Deviatoric stress, average velocity and cumulative AE with time. AE locations are shown looking along the strike of the first fracture (ii), and into the plane of the eventual fracture. Periods A-F are marked on (i). (Thompson et. al., 2006)
Diederichs (2000) performed micro mechanical studies using a two dimensional discrete element simulation involving bonded particle models, (Particle Flow Code, PFC$^{2D}$, Potyondy and Cundall, 2004). A typical confined test is shown in Figure 1.4. As can be noted, even though the model cannot simulate unstable crack propagation, the model exhibits the same key stages of failure as identified by others above. Of significance to this study is the generation of cracks in the post-peak, which dissipate or disperse from the main macrofracture on reaching residual strength as deformation is initiated along this structure, the sample thus becoming ‘aseismic’, and showing similar behaviour to Lockner’s (1993) lab experiments (Figure 1.2).

1.3 Field Scale Response to Failure and Microseismicity (MS)

In contrast to laboratory testing of brittle rocks, there has been comparatively less research to identify and understand the behaviour of the rock mass at the peak strength and into the post peak. Most of the research thus far has concentrated on observation of the rock mass in the pre-peak and/or post peak but, in generally low confinement regions being either close to openings, (Martin, 1997; Diederichs, 2000; Hajiabdolmajid, 2001; Diederichs et. al., 2007), or close to a mining induced caving front (Falmagne, 2002). These studies are of particular interest as the authors have made direct comparisons of rock mass failure to laboratory observations. In this thesis the research is directed primarily to regions of the rock mass away from the direct excavation surface, in which, (based on linear elastic modeling), confinement increases significantly beyond the levels determined in these previous studies, and into a realm that is little understood in terms of the peak and post peak behaviour.

What is known about the behaviour of rock masses is that at the rock mass scale or as the scale of any rock sample increases, the peak strength of the material significantly reduces as a result of the increase of micro and macro discontinuities, (Hoek and Brown, 1980), and the lower strength is based on the weakest-link theory of fracture propagation (Diederichs, 2000). The peak rock mass strength envelope has been suggested by Hoek and Brown (1980), [the Generalized Hoek-Brown (H-B) failure criteria], to be a function of rock mass quality based on RMR (Bieniawski, 1989) or GSI (Hoek et. al., 1995). From non-linear modeling, based on back analysis of near excavation failure, this criterion, combined with appropriate post-peak parameters, has been able to achieve similar displacement behaviour, (Crowder et. al., 2006), or depth of failure behaviour (Hajiabdolmajid et. al., 2002, Diederichs et. al., 2007). However, for linear elastic modeling this failure criterion generally under estimates the extent of failure, and hence damage limits based on the Hoek-Brown brittle failure parameters, (Martin et. al., 1999), have been found to be more appropriate.
Figure 1.4 A typical stress-strain curve from a discrete element simulation involving a bonded particle model under triaxial loading with a $\sigma_3$ of 20 MPa. Accumulation of new cracks (top) and cumulative crack growth (right axis). Points of crack initiation ($\sigma_{ci}$), interaction ($\sigma_{cd}$), coalescence, localization ($\sigma_f$) and eventual aseismicity at the residual strength ($\sigma_{cr}$) indicated (after Diederichs 2002)
The post-peak stress-strain behaviour of intact rocks, (Figure 1.5), and implied for rock masses through intuition and back analysis but not measured, (Hoek and Brown, 1997), can be simplified to follow three distinctive mechanisms: Perfectly Brittle, Strain-Softening or Strength-Weakening (Brittle-Plastic) and Ductile behaviour (Perfectly Plastic). A schematic representation of the behaviour of rock masses, used for discussion is presented in Figure 1.6.

### 1.3.1 Brittle and Strain Weakening Behaviour of Rock Masses

At very low or no confinement purely brittle failure, (complete loss of strength), occurs resulting in spalling and slabbing seen at excavation surfaces. This failure mechanism is dominated by dilatational extension fracturing, similar to axial splitting in a laboratory uniaxial compressive test, and microseismicity events are expected to be predominantly tensile in nature (Cai et al., 1998). Stacey (1981) proposed a simple extensional strain criteria to predict the extent of this fracturing for simple cases. It has been observed from a number of studies, (Martin, 1997; Diederichs, 2000; Falmagne, 2002; Diederichs et al., 2002), that the onset of microseismicity occurs at close to the equivalent laboratory point of crack initiation threshold ($\sigma_{ci}$), (Figure 1.6 and Figure 1.7a). Diederichs (2000) notes that, based on field observations and micro mechanical studies, the crack initiation level is 'universal' at all rock scales and relatively insensitive to confinement.

The next stress level stage in the pre-peak failure process is crack interaction ($\sigma_{cd}$), and point of true yield which was identified at the rock mass scale by Falmagne (2002), through the study of microseismicity using a clustering index, (modified from Lockner et al., 1992), and comparisons to rock mass failure by stress induced overbreak. Falmagne (2002) identified, using linear elastic modelling, that the crack interaction stage, based on a critical clustering seismic density and calibration to caving, occurs roughly at an equivalent stress level to crack initiation, ($\sigma'_{cd}$ [rock mass] $\approx \sigma_{ci}$), (Figure 1.6). Based on Falmagne (2002), and other investigations, (Martin, 1997; Castro et al., 1996, Diederichs, 2000), Diederichs (2000) notes that for low confinement regions and massive rock the yield envelope, defined by $\sigma_{cd}$ at the rock mass scale, collapses to the crack initiation threshold and could be defined by a spall limit (Figure 1.7b). This led to the proposal of the bi-linear failure curve (Diederichs, 2000; Kaiser et al., 2000), when considering in situ rock mass strength at low confinement (Figure 1.7c), and is based on the concept that the cohesive strength component dominates at low strains and low confinement (i.e. $m=0$ [H-B] or $\Phi = 0$ [Mohr-Coulomb]), whereas the frictional strength component dominates at large strain and high confinement (Martin, 1997).
Figure 1.5 Characteristic of behaviour (phenomenology) of rocks in (a) brittle and (b) ductile states (after Hajiabdolmajid, 2001)
Figure 1.6 Schematic representation of the behaviour of the rock mass as a function of confinement compared to an intact sample (note: intact ($E_i = 55$ GPa) and rock mass modulus ($E_{rm} = 17.8$ GPa), are based on the material properties found at the Hemlo Camp, the proportional rock mass stiffness based on the GSI ~ 60)
Figure 1.7 Comparison of laboratory crack initiation, to rock mass initiation at the URL (Martin, 1997). (b) Representation of the bi-linear failure envelope based on damage initiation thresholds and spalling limit from field observations (after Diederichs, 2000; Kaiser et al., 2000), and (c) Schematic of bi-linear failure curve with damage limits from Brunswick mine, “damage and spalling failure map” (after Diederichs et al., 2002, Diederichs et al., 2007). Note $\Delta\sigma_{\text{crit}}/\Delta\sigma_3 = \text{slope, (A), of the linear relationship, } \sigma_1 = A\sigma_3 + B\sigma_c\text{, where } A=1\text{ to }1.5\text{ and } B=0.3\text{ to }0.5\text{. }\Delta\sigma_{\text{crit}} = \sigma_1/ B\sigma_c\text{ [i.e. normalization of the linear relationship w.r.t. the rock mass strength].}
Thus, at low confinement, unstable propagation of extensile fractures results in the peak strength being reached immediately ($\sigma_f' [\text{rock mass}] \approx \sigma_{cd} [\text{rock mass}] \approx \sigma_d$). This theoretical bi-linear damage and spalling failure map (Figure 1.7c), has been used as a basis to calibrate linear elastic modelling to observed damage behaviour based on qualitative intensities of microseismic events and pillar damage with some success (Diederichs et al., 2002).

As confinement increases at the rock mass scale, the length of extension fracturing is inhibited, (Diederichs, 2000; Hajiabdolmajid, 2001), and the post peak behaviour changes to strain softening, (Sture and Ko, 1978; Hajiabdolmajid, 2001; Crowder et al., 2006, Diederichs et al., 2007), with a more gradual reduction to a residual plastic strength plateaux (Figure 1.6). Again microseismicity is expected to start at the crack initiation threshold, however, it is expected to become more shear rich, (Urbancic, 1991; Urbancic and Trifu, 2000), with the tensile component inhibited due to the reduction in extension fracturing and increased confinement. Similar to the laboratory triaxial test, (Figure 1.2), it is postulated, and observed in this research, that this shear behaviour results in coalescence and localization to a macrofracture structure (Figure 1.6). It is also postulated that the rock mass becomes ‘aseismic’ when reaching the residual strength as deformations may become plastic or dilations large enough to prevent recordable microseismicity. The limit for this strain softening or strain weakening behaviour may be up to the limit of Mogi’s (1966) brittle-ductile transition zone, (Figure 1.7b), as has been suggested by Diederichs (2000) and Falmagne (2002), although in the case of the first author this was noted to be applicable only for massive to moderately jointed rock. It is suggested in the research carried out in this thesis that this transition state may not be applicable to define the transitional behaviour from brittle to ductile at the rock mass scale when using linear elastic stress modelling, which does not allow for progressive changes in the geometry and stress shedding as with non-linear stress modelling.

Diederichs (2007) has proposed a peak strength criteria for non-linear modelling of tensile spall for massive rock around drift openings, in this case a spall criteria, based on Griffiths theory, (after Hoek, 1968), with a post peak residual envelope that exhibits brittle behaviour at low confinements and strain-hardening at higher confinements. Based on the observation of ‘confined failure’ at the two case studies examined, strain-hardening at higher confinements may be unrealistic for a more jointed rock mass in which excavations exist. Diederichs (2007) notes, however, that this model is only applicable for low confinement and small strains, and that a third unspecified envelope (ultimate residual) is required for large strains.
A cohesion weakening-friction strengthening model has been proposed by Hajiabdolmajid (2001), for low confinement and is based on the observations of Martin (1993) and Martin and Chandler (1994) for Lac du Bonnet Granite, and Schmertmann and Osterberg (1960) for soils in which, unlike in classical Mohr-Coulomb theory, there is non-simultaneous mobilization of cohesion and friction. That is, peak mobilization of friction can only occur once significant amounts of cohesion have been lost.

For brittle behaviour or strength loss, this dilatational extension fracturing is accompanied by a volume increase. Diederichs (2000) notes that dilatant crack accumulation in conditions of global extensile strain, result in the delayed mobilization of friction, (due to open fractures), and leads to the brittle behaviour, (i.e. the failure is based primarily on cohesion loss, as the only way to rebuild stress is to first close the fractures and then mobilize the friction). As the confinement increases the amount of dilation will hence decrease, as extension fracturing is inhibited, and thus reduce the brittle behaviour.

1.3.2 Ductile or Perfectly Plastic Behaviour of Rock Masses

Under very high confinement, ductile behaviour may occur, (Mogi, 1965), with no strength loss following the peak but perfectly-plastic or strain-hardening behaviour, such as in the formation of plastic shear zones, (Watterson, 1999), or pseudo plasticity created under high confinement at the core of a pillar. At these very high confinement conditions the creation of small extension fractures are suppressed, thus in the lab leading to very few acoustic emissions, or at the rock mass scale, little microseismicity. At these conditions plastic flow may not be localized, and there is negligible volumetric strain and no dilatant behaviour (Hadjiabdolmajid, 2001). Lajtai et. al. (1994) identified the formation of a macroshear structure in potash salt rock, formed by en echelon overlapping ‘tensile’ crack arrays when under increased confinement. The significance of this is that the material exhibited overall plastic behaviour, but still with the formation of tensile extension cracks, and indicates that behaviour is not always as expected based on these simple assumptions.

1.3.3 General Observations of the Current Knowledge of Rock Mass Behaviour

It is important to note that these are the expected behaviours that may be observed, but at what state of confinement they will occur in the rock mass for a given rock mass quality is still under debate. It was proposed by Hoek (2004), on the discussion of determining the post-peak behaviour of rock masses based on linear elastic stress modelling as a first approximation, that spalling behaviour can be expected at a $\sigma_1/\sigma_3 > 10$, brittle-shear behaviour (strain-softening) 10
$\frac{\sigma_1}{\sigma_3} > 3.4$ to 2, and ductile (strain hardening) behaviour, $\frac{\sigma_1}{\sigma_3} < 3.4$ to 2. These rough limits are based on combined observation in the rock mass and from laboratory testing. The value of $\frac{\sigma_1}{\sigma_3} = 3.4$, comes from the classic brittle-ductile transition zone determined by Mogi (1966) for intact rocks. The range was been proposed by Hoek (2004), based on triaxial testing observations of the post-peak behaviour of broken rocks, namely Panguna andesite (Hoek and Brown, 1980). The only other major series of triaxial testing performed on broken rock, (not coal), is that of Hobbs (1970), however, only the peak strength characteristics were observed. Other researchers have looked at true triaxial testing using a polyaxial testing frame, (King et al., 1995; Pettit, 1998), to induce fractures of a specific orientation, however, the stresses employed were generally not sufficiently large to cause significant damage in hard crystalline rocks (Pettitt, 1998). A new polyaxial testing frame has been developed at the University of Toronto, (Young, 2007), with greatly increased load capability in all axes and will hopefully be able to answer better the effects of discontinuities and the transition from brittle-ductile behaviour of disturbed rock samples.

1.3.4 Mine Induced Microseismic Analysis

Many underground mines have incorporated microseismic systems into their operations, relying mainly on the location of microseismicity and an estimation of the magnitude of events to aid in day to day decision making regarding regions of high stress that are seismogenic in nature, (e.g. Simser and Falmagne, 2004; Beck and Brady, 2002; Hudyma et al., 2002; Mendecki, 1993). Presently our ability to model and make proactive decisions regarding these seismically active regions, (portions of the rock mass that are failing either due to high stress or relaxation), is hampered by our lack of understanding of the behaviour of the failure process and our limited ability to determine the correct peak and post peak parameters to use in modelling. Although, as mentioned previously, some success through back analysis for near field failure close to drift openings has been made (Crowder et al. 2006; Hajiabdolmajid, et al., 2002; Diederichs et al. 2007). In order to improve our understanding of the relationship of fracture generation measured microseismically during the failure process of brittle rock masses under high stress, it is important to understand better the fundamental behaviour of the mechanisms involved.

Research of the peak and post-peak behaviour of rock masses specifically using microseismicity, (moment magnitudes < 0), has concentrated as previously discussed generally in regions under low confinement and in close vicinity to excavations. Significant knowledge has been gained through the controlled experiments of the Underground Research Laboratory, (URL), toward understanding the fracture generation process, (Martin, 1997; Gibowicz et al.,
1991; Feigner and Young, 1992,1993; Hajiabdolmajid et al., 2002; Cai et al., 1998; Young and Collins, 2001; Reyes-Montes, 2004). These observations are, however, based on a single strong brittle rock with limited geological structure, and relatively small volumes of failure close to the openings. In the mining environment, encompassing larger volumes of varying rock types with geological structure, research has been carried out attempting to define the pre-peak to peak strength behaviour of the rock mass in relation to caving and the mine induced generation of microseismic activity, using the spatial coalescence of events through the clustering index (Falmagne, 2002). In more general mine wide studies, not specifically relating microseismicity to the strength state of the rock mass, there has been investigation into the source parameters of microseismically induced events, (Urbancic, 1991; Bird, 1993; Mercer, 1999), and also the study of source mechanisms by either first motion studies or moment tensor inversion of mining induced events, (Gibowicz, 1989; Urbancic, 1991; McCreary et al., 1993; Sampson-Forsythe, 1994; Urbancic et al., 1993; Trifu, 2001 and Trifu and Shumila, 2002). There have also been many studies on macro or large mine induced seismic events, (moment magnitude > 0), mainly in the South African goldfields, [e.g. McGarr (1991); Brummer and Rorke (1990), Spottiswoode (2001), van Aswegen and Butler (1993), and Mendecki (1996)], however, it should be recognized that, although these studies are relevant, the characteristics of large mining induced events may not be the same as the microseismicity studied in the previous research.

Although much has been learned from these studies, generally, they are based on limited data, or may only capture a portion of the failure process. The reason for limited data, especially for source mechanism studies, is that the process requires the determination of first motions, which can be very slow and labour intensive. Additionally, because the failure process is not well understood, grouping of a range of different complex failure behaviours may occur, making interpretation difficult.

In order to examine the post-peak mechanisms of a confined rock mass, it is first necessary to determine at which stage of failure the rock mass is in. The goal of this research is to see if the stages of failure can be identified from the microseismic analysis with field displacement monitoring information if present, to better understand the post-peak behaviour. It is important to know what is happening and when it is happening before developing a revised constitutive model to improve forward modelling of rock mass failure. Kaiser (2007) notes that even after the formation of the International Society of Rock Mechanics in 1962, developed partially because of a lack of understanding of the strength of rock masses, the community today is still struggling with understanding the behaviour of the rock mass in relation to strength and
deformation characteristics, and urges that researchers need to concentrate on the fundamentals of behaviour.

1.4 Thesis Research and Objectives

In this research we are largely interested in determining at what point regional rock mass failure, (i.e. failure of a confined region distant from the immediate excavation surface), occurs and whether mine induced microseismicity can be used to define the stages of the failure process. This thesis is based on two case studies, in which failure of confined regions of the rock mass were determined to go through the complete stages of failure from microseismic initiation to aseismic behaviour. These case studies are based at two different mines the Williams mine and the Golden Giant mine, which are situated in the same rock mass and use similar microseismic systems.

The overall behaviour observed at Williams and Golden Giant mines was perceived to be brittle strain softening due to the removal of stress effects. This was inferred based on changes in the mining environment. At the Williams mine the failure of the rock mass predominantly occurred in between the lateral pillars formed between the haulage drifts in the footwall of the orebody. Prior to significant and intense microseismic activity, the sill pillar region was highly stressed where the crushing of blast holes, dog earing and failure of in-stope raise bores, and the triggering of seismic flurries through blasthole drilling activity, would make mining difficult. These effects led to reduced mining activity in the sill stopes, however, due to increased regional stresses from mining outside of the sill pillar region, microseismic activity in the footwall of the sill continued over a 3 year period. It was found that as the microseismicity abated through aseismic activity starting from the east end of the sill, mining could advance behind this ‘failed’ front without any stress effects that had been seen historically. This was perceived as failure in the footwall, causing stress shadowing of the mining region. This failure due to the observed lack of stress induced damage and lack of squeezing of blastholes, has been inferred to be strain softening or strength weakening, causing the induced stresses to fall below the initiation stress. Similar behaviour was observed at the Golden Giant mine, in which significant stress induced damage occurred in and around the region analysed during development of a destress slot for the shaft. Here following a more rapid development of microseismic activity over the time frame of months rather than years, it was found that after the mining region had become aseismic, the stopes could be easily drilled and mined without stress induced damage or squeezing of blastholes. Again it has been inferred that this was due
to strain softening or strength weakening of the rock mass reducing stresses to a level below the seismic initiation stress and approaching the inferred residual strength. The main interpretations in this thesis are based upon this brittle strain softening behaviour that eventually ensued and was evident from the change in the mining environment.

The key objective of this research is to determine if it is possible to identify different stages of failure from pre-peak to the post peak, using microseismicity, and to better understand the behaviour of the rock mass as it transitions through failure. The objective of the microseismic analysis strategy is to determine if run of mine data without the need for complex post processing and filtering can be used to gain insight into the various stages of failure.

1.5 Structure of Thesis

The investigation of the behaviour of the failing rock mass for the two case studies is based primarily on spatial and temporal analysis of microseismicity developed and recorded during the complete failure process. The primary basis for the analysis is founded on application of the principal component analysis (PCA) technique applied to determine changes in the spatial trends of the clustering seismic events, combined with the development of a simplified clustering density analysis to identify the point of potential rock mass yield. This method was primarily chosen, based fundamentally on event locations, as at the Williams mine the quality and continuity of the full wave form seismic data available made performing more detailed analysis of seismic source parameters and source mechanisms difficult. At the Golden Giant mine the data quality was significantly improved and more in depth analysis of the seismic source parameters and mechanisms could be achieved.

In Chapter 2 the theory behind the development of the clustering seismic density and evaluation of sensitivity to voxel size and source radius is evaluated. The theory behind the principal component analysis (PCA) technique is presented and the results of an evaluation of the best controlling parameters to perform temporal evaluations, a compromise between sensitivity and stability, is summarised. Additionally, the theory behind seismic source parameter calculation used at the Golden Giant mine and theory of source mechanism determination using first motion studies is described.

In Chapter 3 a region of the Williams Mine Sill Pillar is examined through spatial and temporal analysis of microseismicity, investigating the use of the principal components analysis, (PCA),
technique to characterise the behaviour and orientation of clustering mine induced seismicity combined with analysis of displacement monitoring instrumentation. Linear elastic modeling is used to understand the loading path, and calibration of damage limits, and non-linear modelling is performed to understand the required post peak parameters that will produce similar model conditions to those observed in the field. The emphasis, as previously stated, is to determine if it is possible to identify different stages of failure from pre-peak to the post peak, using microseismicity, and to better understand the behaviour of the rock mass as it transitions through failure. The concept of the Point of Disassociation, as observed from this study is presented.

In Chapter 4 a similar style failure to those analysed at the Williams mine is identified in a region of a pendent pillar adjacent to a destress slot at the Golden Giant mine. The PCA technique is applied to determine the failure state of the rock mass, and based on this the temporal variation in key source parameters is observed. Linear elastic modelling is again used to determine the loading path, and calibration of damage limits made with comparison to the Williams mine case study.

In Chapter 5 the failure identified at the Golden Giant mine of a macrofracture shear structure, is analysed in greater detail using a first motion study to observe changes during the failure process of focal mechanisms, and relate the individual fracture orientations to those of the group behaviour determined from the PCA technique. Linear elastic and non-linear stress modelling are used to rationalize the orientation of the fractures, and post peak constitutive behaviour to produce similar shears. A simplified physical fracture model is developed and used it investigate the potential fracture network that evolves in the macrofracture shear structure.

At the end of each one of the case study chapters detailed conclusions and recommendations are made, and the main overriding conclusions, contributions and recommendations presented at the end of the thesis in Chapter 6.
CHAPTER 2

MICROSEISMIC ANALYSIS THEORY

2.1 Introduction

Standard field monitoring instrumentation such as extensometers and stress cells, or visual observations from borehole cameras, geotechnical drill core logging and site mapping give the engineer direct information about local behaviour or location specific behaviour of the rock mass. In contrast, microseismic systems that have become more prevalent at many underground mine sites around the world, have the ability to indirectly monitor large volumes of the rock mass and give information about the visually unseen portions of the rock mass that may not be easily accessible. These microseismic systems have become an integral part of the engineers toolbox, for giving indications of active areas where the rock mass has started to ‘yield’ or areas which may be prone to ‘rockbursts’.

Although much work has been performed on the analysis of full waveform data from mine sites and derived source parameters (Kazakidis, 1990; Urbancic, 1991; Gibowicz et. al., 1991; McGarr, 1993; Mendeki, 1994; Bird, 1993; Alcott et. al. 1998; Mercer, 1999, Beck, 2002; Young et. al., 2004; Trifu and Shumila, 2002; Simser et. al. 2002; Urbancic and Trifu, 2000), these studies require a reasonable array of triaxial sensors to accurately characterise the source in terms of magnitude, energy and mechanism and even with a good array, trends in data may still not be clear or illuminating (Mercer, 1999).

At the Williams Mine, only limited information could be gained from the triaxial sensors, although a complete record of source locations existed, there was unfortunately incomplete full waveform records for all events (discussed in Chapter 3). In order to study temporal changes or seismic indicators to determine the state of the rock mass, for the Williams Mine Case study (Chapter 3) the research focuses on the relationship of the location of the events to one another (Section 2.2). Located sources are probably the most reliable parameter determined from microseismic systems, the automatic source location algorithm employed at both sites is discussed Chapters 3 and 4, and is based on the simplex algorithm (ESG, 2006). In a mining situation when workers hear a sizable microseismic event, the first question is where did the
event locate, then next question invariably is how big was it (i.e. what is the magnitude of the event). In Section 2.3 the background theory behind more in depth analysis of microseismicity, through source parameter determination and source mechanisms is discussed. These were analysed for the Golden Giant Case study (Chapters 4 and 5), which had a well-constrained array with continuous full waveform data available.

2.2 Microseismic Analysis Based on Location and Source Size

There are a number of techniques that have been applied to analysing microseismic event clusters based primarily on their location. These techniques are generally used to determine the magnitude of clustering or define planar trends of seismicity that may relate to geological structure, based on the assumption that seismicity occurs on pre-existing features (Urbancic et al. 1993). These analysis techniques work on the assumption that the events are related to one another through the rock mass joint or fracture fabric, and induced stress.

Falmagne (2002) developed a modified clustering index function, \(C_{If}\), based on work by Lockner et al. (1992), to try to quantify the point of interaction and coalescence \(\sigma_{cd}\) of events, thus identifying the point of ‘true yield’ of the rock mass in analogy to failure of laboratory samples. The clustering index function is based on the inter event distance and degree of interaction dependent on the source size, (volume of inelastic deformation), and is used to determine the cluster index, \(C_{Ii}\), this being the effect of all historical neighbouring events, requiring a cumulative history (Appendix A1). Falmagne (2002), went on to use the clustering index to spatially identify regions where the rock mass was yielding, correlating this to the amount of overbreak, (caved rock), that occurred at the Lac Short mine.

Reyes-Montes (2004) used a technique of relative location based on the method of Gibowicz and Kijko (1994), to improve the location accuracy of events before analysing the cluster index. The method uses a master event and then relocates all the other events in the cluster based on the velocity structure of the master. This can remove location inaccuracies due to an incorrect velocity model or the presence of excavated voids (Reyes-Montes, 2004) and can improve the definition of linear planar structures that fit geological and source mechanism observations (Saccorotti et al., 2002). Reyes-Montes (2004) applied this method on the microseismic data generated around the tunnel sealing experiment (TSX) at the Underground Research Laboratory (URL) to improve resolution of the seismic cloud and then used the clustering index to filter out non-interacting events.
The clustering index is used here to assess the potential for events to be interacting based on the seismic event density and estimated source radius. This is used to estimate the ‘clustering density’, which is an estimate of the point at which interaction might be occurring and thus yield of the rock mass (Section 2.2.1). Although the use of relative location may have greatly aided in improving the overall relative location accuracy, because of the complexity and time involved in processing the data, this avenue was not followed.

Reyes-Montes (2004), in the same study at the URL, went on to apply the three-point method, (after Fehler et. al., 1987), to identify internal structures based on clustering events only and found that these fit with those physically observed from analysis of the fracture patterns around the tunnel. This three-point method has also shown some promise in identifying potentially activated structures underlying the distribution of events in a block caving environment when compared to PFC modelled fractures (Reyes-Montes et. al., 2007). One draw back to the three-point method, as applied by Reyes-Montes (2004), is that it must be normalized against a synthetic dataset of randomly distributed events to remove bias by the shape or volume of the cluster analysed. Also, events must be filtered out that are either too close together, (i.e. less than the source location error), or separated too far apart, (not potentially interacting based on their clustering index), and is intended to function on clustering events only. Thus, this requires a trial and error approach to obtain dominant structures, and the filtering can significantly reduce the dataset to be analyzed.

Another method that will be discussed in greater detail in Section 2.2.2, is the Principal Component Analysis (PCA) method, which is a statistical technique applied to determine the spatial trends in a cluster of seismic events. This technique has been used previously to relate mine induced seismicity to active structures that correlate to geology, (Coulson, 1996; Kasier et. al., 2005) and to fault plane solutions, derived from first motion studies and stress inversion (Urbancic et. al., 1993). One advantage of the PCA technique over the three-point method is that it describes the three-dimensional fit of seismic events to potential planar features, whereas the later method assumes a flat planar trend a priori. The PCA method has been found to be very useful in describing the behaviour of the microseismicity and the rock mass, and forms the basis for major observations in this research.

### 2.2.1 Seismic Event Density and Clustering Density

During this study a great amount of microseismicity was recorded at both the Williams mine (Chapter 3), and Golden Giant Mines (Chapter 4). A typical plot of microseismicity recorded in
the Williams mine sill pillar region from 1999 to 2005 can be seen in Figure 2.1 (discussed in detail in Chapter 3). In order to get a more defined picture of where and to what intensity the microseismicity was occurring, a cumulative seismic event density algorithm was developed to visualize the most intense regions. This was achieved by counting the number of events that occur within a specific voxel (cubic) volume, evenly distributed over the domain. Although the idea of seismic event density is not new (Maxwell, 1993), it's main use has been to filter out events based on density criteria (Kaiser et al., 2005). Practical application of this form of visualization is limited, although, Beck and Brady (2002), used a cell evaluation method to estimate the probability of microseismic occurrence, and event and source parameter contouring has been a staple visualisation tool of the Integrated Seismic Systems (ISS) International’s JDi software (Mayer and Mendecki, 1995, Mendecki, 1994,1996). Additionally, seismic event density analysis can also be accomplished with a number of other software packages, such as Map3D© (Mine Modelling Ltd., 2006), Hyperion Seismic Software Suite (ESG, 2006) and Examine3D© (Rocscience Inc., 2007).

A study of the effect of voxel size, which can have a large impact on the resolution of the seismic intensity, was performed (Appendix A1.2). It was found that the best resolution appears to come from a voxel dimension of 5 x 5 x 5 m (125 m³). From a practical stand point a voxel size of 5 x 5 x 5 m, is often what is used as a Selective Mining Unit (SMU) for block modelling, when Kriging of grade data is performed for open stope mining, and physically can be visualized as the dimensions of a common excavated drift in section. This voxel size is also quite appropriate in this case, if the source radii are considered.

As discussed earlier, at the Williams mine there were little to no confident source parameters that could be determined, however, at the adjacent Golden Giant mine (Chapter 4) online event source radii are determined using the Madariaga model (ESG, 2006). The source radii determined for a year of data at Golden Giant Mine around the shaft pillar area (~19,000 events) were found to be normally distributed, with mean source radius 1.85 m and ranged from 0.8 to 2.5 m with a standard deviation of 0.3 m (Figure 2.2a). To all intensive purposes, this site is of a similar rock mass quality, using the same type of seismic system, and under similar stress conditions to the events occurring at the Williams mine. It was found at the Golden Giant mine that the events exhibit non-similar scaling behaviour, resulting in a limited source radius (Chapter 4). The same behaviour would be exhibited at the Williams mine, given the same rock mass conditions.
Figure 2.1  Microseismicity recorded in the Williams mine sill pillar region analysed in this paper from September 1999 to February 2005. (a) and (c) The sill region analysed here is bounded from stope 28 to stope 16, and analysis slices 1 to 6 are 40 m wide spanning two stopes (Note position of section 9430 E). Approximately 33,000 events where recorded and located during this time frame.
This source size does not seem unreasonable from observations made at the Hemlo camp, in which stress driven fractures observed in the back of development drifts and stopes, after failure and caving, can be in the order of 2 to 5 m radii, (although it is important to note that these stress induced fractures could be due to relaxation and tensile extension fracturing). Additionally, through investigation of mining induced seismicity in Polish mines and Canada, Gibowicz (Gibowicz et al., 1991), has found that Madariaga's model provides reasonable agreement with independent visual observations underground.

A rough approximation to determine if events are coalescing and onset of yield occurring would be if at least 5 events of the mean Madariaga source size, (1.85 m), are contained within the 125 m³ voxel (Figure 2.2b). Evaluation of the clustering index (Cli) for a worse possible case event distribution in a voxel, (i.e. four events at the corners and one at the centre) was calculated for varying source dimensions (Appendix A1.3). It was determined that potential interaction/coalescence of events (Cli >0.5 after Falmagne, 2002) will be achieved over the range of source radii (± 1 standard deviation), identified here, for a minimum of 5 events (Figure 2.2c). Thus, at this density these events will probably be interacting, if not coalescing, and have a Cluster Index, Cli >0.5. The seismic event density for the events recorded in the Williams sill, on analysis section 9430 E, (see Figure 2.1a & c for location) has been plotted in Figure 2.2d, using 5 events/voxel as a lower bound contour limit. In three dimensions a potential rock mass yield region can be identified contained within the iso-surface contour of 5 events per voxel (Figure 2.2e), this being identified here as the clustering density and potential point of true yield. It should be pointed out that this is a rough estimate, as the actual source rupture envelope in the Madariaga model is considered to be a 2D circular surface, however, by applying this limiting condition a reasonable approximation to the potential 'yield' front is achieved.

Discussion of various source models and size estimates can be found in Appendix A1.3 and Equations specified in A3.3. It was determined from seismic density analysis and determination of the clustering density that the Madariaga model produced a more appropriate source size than other models, in particular the Tensile source model proposed by Cai et. al. (1998). This model may be appropriate for low confinement conditions close to openings, but when applied here where the confinement is relatively high, it produces source size estimates that are too small to achieve regional interaction/coalescence (Appendix A1.3), and thus results in only isolated potential yielded regions that do not fit with the observations.
Figure 2.2  (a) Distribution of Madariaga source radius determined in the Golden Giant Shaft Pillar region (b) Representation of worst case event location in a 125 m$^3$ voxel. (c) Calculation of clustering index (CIi) for a 5 x 5 x 5 m cube (125 m$^3$) with 5 events (blue line CIi for centre event and red line CIi for corner events), and 4 events only (green line CIi for corner events). (d) Contour of event density for 125 m$^3$ voxels with the lower contour limit set to 5 events and (e) 5 event per voxel iso-surface of all events recorded in the sill region.
2.2.2 Principal Component Analysis Technique

The principal component analysis (PCA), also known as the method of principal parameters (Michelini & Bolt 1986), is a statistical technique for the determination of spatial trends in a cluster of seismic events. This method was first used to identify active fault structures in earthquake seismology (Michelini & Bolt 1986; Posodas et. al., 1993; Saccorotti et. al., 2002). It has been used by Urbancic et. al. (1993), Coulson (1996) and Kaiser et. al. (2005) to relate mine induced seismicity to active structures, that correlate to geology, and in the case of the first author to fault plane solutions derived from first motion studies and stress inversion of fault planes. The main assumption of the technique is that events are inter-related to one another through the stress regime and geology, or in this case, as determined from the cluster analysis (Chapter 3), of the fracture generation zone. Since trends are based on inter-event or inter-fracture communication, it is necessary to treat the data in a time sequential manner.

The technique first requires the identification of a cluster of events. In this research clusters are identified by a combination of visual extent and the seismic event density to determine regions where coalescence may be taking place. In the case studies presented latter, the seismic activity was present over years of mining. For this reason temporal analysis of the clusters could be investigated more thoroughly than in previous studies.

First each identified cluster, (based on a year of events), was analyzed to determine the distribution of event inter-distances in Euclidean space (Eqn. 2.1), which was generally found to conform to a Normal Distribution with a slight positive skew (Figure 2.3a).

\[
\text{Event Inter-distance, } d_{ij} = \sqrt{(x_i - x_j)^2 + (y_i - y_j)^2 + (z_i - z_j)^2}.
\]  

[2.1]

where:  \( x,y,z \) = event Cartesian coordinates  
\( i,j = 1,\ldots,N \)

and

Total number of inter-distances,  \( n = \sum_{i=1}^{N} (N - i) \)  

[2.2]

where:  \( N = \) total number of events in cluster

From this a characteristic dimension, called the diameter of the spatial window, \( D \), is determined (Figure 2.3a and b). From synthetic data and previous studies, (Posodas et. al., 1993; Coulson, 1996; Urbancic et. al., 1993), it has been found that a ‘\( D \)’ value that captures 75% of the inter-distances, produces reliable, and stable results [Section 2.2.2.1].
Figure 2.3 Principal Components Analysis Technique. (a) determination of optimum spatial window size, D, based on the distribution of the cluster of events hypocentre inter-distances in Euclidian space, using the a Normal Distribution Cumulative Density Function. (b) Determination of the mean hypocentral location of the events that fall within the Spatial Window, situated on the event of interest, and (c) calculation of the spread matrix describing the variance of the hypocentres location to the mean hypocentral location, and (d) application of the principal components analysis through eigenvector and eigenvalue decomposition to produce and ellipsoid with strike and dip determined for the overall trend of the events surrounding the event of interest. (e) Application of the temporal sliding window, in the example shown, N' is 50.
As there were a large number of clusters of events to be analysed over a number of years, the algorithm was automated to determine the corresponding $D$, from the normal probability density function (Eqn. 2.3) and then calculating the cumulative distribution function (CDF) in Equation 2.4 plotted in Figure 2.3a:

$$P(x) = f(x, \mu, \sigma) = \frac{1}{\sigma \sqrt{2\pi}} \exp\left(-\frac{(x - \mu)^2}{2\sigma^2}\right)$$  \[2.3\]

where: $x = d_{ij}$, event inter-distance, $\mu, \sigma = \text{mean and variance of inter-distances}$

$$\text{CDF} = \sum_{i=1}^{n} P(x_i)$$  \[2.4\]

The basic process of the PCA involves positioning a spherical volume of dimension $D$ (the spatial window), on an event in the cluster (Figure 2.3b) and then determining the mean hypocentral location of all the events inside this volume:

$$\overline{x}_m = \left(\frac{1}{K}\right) \sum_{j=1}^{K} x_{jm}$$  \[2.5\]

where: $x_{jm} = \text{event coordinate}$ $j = 1, \ldots, K$ $m = 1, \ldots, 3$

$K = \text{number of events inside the spatial window}$

$j = \text{event number}$

$m = \text{Cartesian axes}$

Next the spread matrix is determined, this being the variance in the Cartesian coordinates to the mean hypocentral location, the multivariate covariance matrix:

$$S_{im} = \frac{1}{K} \sum_{j=1}^{K} (x_{ji} x_{jm}) - \overline{x}_i \overline{x}_m$$  \[2.6\]

where: $x_{ji}, x_{jm} = \text{event coordinates}$ $j = 1, \ldots, K$ $i,m = 1, \ldots, 3$

$\overline{x}_i, \overline{x}_m = \text{sample mean}$

Decomposition of this diagonalized $3 \times 3$ matrix, results in the eigenvalues ($\lambda_1 > \lambda_2 > \lambda_3$) and eigenvectors ($\hat{n}_1$, $\hat{n}_2$, $\hat{n}_3$), [i.e. the principal components of the spread matrix], which can be interpreted geometrically as describing ellipsoids if a dominant trend exists (Figure 2.3c and d). Planar trends are identified for ellipsoids that have a major to minor eigenvalue ratio $> 2.5$ (after
Posadas et al., 1993). Here this ratio is called the ellipsoids ellipticity (Eqn. 2.7) and the larger the value the flatter the plane or the greater spread of the trend.

\[ \text{Ellipticity (Eli)} = \frac{\lambda_1}{\lambda_3} \]  

[2.7]

As we are interested in identification of mainly planar trends, event planes with ellipticity < 2.5 are dropped from further analysis. Thus, an orientation of the strike and dip of the ellipsoid or PCA plane can be identified. This process is repeated for all of the events in the cluster, and the mean orientation of the event cloud can be obtained by plotting the events on a lower hemisphere stereographic projection.

So that only events that are ‘related’ to one another in time are examined, it was found through a sensitivity analysis, (section 2.2.2.1), that a minimum 50 events (N') at a time should be analyzed. In this case a fixed width moving ‘temporal window’, was used such that the PCA technique was applied to the first 50 events, then the next 50 etc., until all the events in the cluster have been sampled and analyzed, (Figure 2.3e). The procedure explained here was developed into an algorithm based on (Urbancic et al., 1993 and Coulson, 1996), and can automatically process many clusters of events at a time, once the events coordinates have been extracted to separate database files.

Additionally, in order to remove outliers, that had very high ellipticity a modified version of the F-parameter (Posadas et al., 1993) was used:

\[ F \text{- parameter} = \frac{K}{N'} \]  

[2.8]

where: \( K = \) number of events in the spatial window  
\( N' = \) number of events in the temporal window (50)

This differs from Posadas et al. (1993) in that the number of events in the temporal window (N') is used instead of the total number of events in the cluster (N), and relates the proportion of the events captured inside the spatial window to the number of events analysed at a given time. Thus, when performing moving averaging of PCA plane ellipticity, or taking the average for sampled 50 event windows, events are dropped if the F-parameter is less then 0.5, [i.e. less than 50% of events in the temporal window are used to determine the PCA plane]. This normally occurs for isolated events on the edge of the cluster and can result in exceedingly high ellipticities.
2.2.2.1 Principal Component Analysis Parameter Evaluation

In order to effectively use this method a number of key parameters were investigated, these are: the size of the spatial window, D, the sample size or temporal window size, N’, and the effect of the sliding window, the later two used to identify temporal variation in the observed trends. Changes in these parameters act on the smoothing of observed trends and temporal variations. An analysis of the effects of changing these parameters on a dataset that showed strongly defined PCA derived trends and temporal variation was performed to obtain the optimum compromise between smoothing and sensitivity of the technique (Appendix A2).

It was found from analysis of varying D values, (Figure 2.3a), that a value based on 75% of the inter-distance CDF, achieves sensitivity with reduced scatter in results, and was chosen as the point of inflection of the average standard errors, and minima of the ellipticity (Appendix A2.1). This value is almost equivalent to the mean inter distance plus one standard deviation, which was found to produce similar results, although the former determination of D has been used throughout this research.

A 50 event temporal window appears to give the most stable results with minimal scatter but still being sensitive enough to observe temporal variations in the spatial distribution of events or PCA derived planes (Appendix A2.2). Evaluation of the overall errors and the average ellipticity, similar to Posadas et. al. (1993), indicated that a for a sample size of 50 events this is the point at which the errors and the ellipticity all start to plateau. Additionally, this sample size was also chosen as being convenient to assess the individual temporal windows (~50 poles) on lower hemisphere stereographic projections using Dips© (Rocscience, 2005). With much less than this number of PCA derived planes, identification of stable population groups can become difficult. It was found that the absolute minimum number of poles or temporal window size to obtain similar results was 25, and if the sample size is increased to 100 events, then too much smoothing can occur with loss of temporal variation.

No significant effect on the observed distribution of PCA derived planes was found with changing the sliding window from a full shift of 50 events (Figure 2.3e) to continuous analysis (Appendix A2.3). However, it was found that using a continuous central moving principal component analysis (CMPCA), [i.e. perform PCA on an event with the sample of 25 events prior and 24 events in the future, and after one iteration, shift the window to the next event], investigated in Appendix A2.3, results in improved determination of the time of temporal changes in the PCA derived planes. However, in order to evaluate a continuous analysis, it is
more efficient to use a direct moving average of the determined strike and dip over
contouring grouped planes every shift on lower hemisphere stereographic projections. This
however, can lead to inaccurate identification of the average trend, if more than one trend
exists within the averaging window, or there is a wide scatter of poles. The preferred method of
identification of the orientation of the mean trend of this three-dimensional data is through using
stereographic projection and a Fisher distribution, which has benefit when observing sparse
data sets, (Rocscience, 2005). The 50 event sample, shift 50 events, is the standard analysis
technique used in this research to obtain the correct orientation, while the CMPCA is used to
determine more accurately the timeline of any changes.

2.3 Overview of Mine Seismology Seismic Source Parameters

In the past 20 years microseismic monitoring which first started in the 1930’s (Green, 1998;
Hedley, 1992) has progressed with state of the art software and hardware to achieve
continuous real time location, and automatic processing of seismic source parameters such as
magnitude, energy and source radius based on acquisition of full waveform seismic event data
(Urbancic & Trifu, 2000; Young, et. al., 2002,2004; Mendecki, 1993,1996). Much has been
learned from fundamental studies of source parameters and source mechanisms of
microseismic events (Magnitude < 0 Mn) at underground mines in Canada (Urbancic, 1991;
Feigner & Young, 1992; Gibowicz et. al., 1991, Bird, 1993, Cai et. al., 1998; Mercer, 1999;
Collins et. al. 2002; Young et. al., 2002; Trifu, 2001, Simser and Falmagne, 2004), as well as
the study of mine induced seismic events (Magnitude > 0 Mn) in South Africa (McGarr, 1993;
Spottiswoode; 2001; Mendecki, 1993,1996; van Aswegen & Butler, 1993). The later two
authors have concentrated on what is termed ‘Quantitative Seismology’, using changes in two
independent source parameters (e.g. energy and moment), to characterise the state of the rock
mass. Recently there has been a move to probabilistic methods and seismic hazard or
rockburst hazard assessment using seismic data (i.e. Alcott et. al., 1998; Beck and Brady,
2002; Hudyma et. al., 2002; Coulson et. al., 2002, Kaiser et. al., 2005). These later two uses of
seismicity and source parameters, although useful from an operator stand point if a rational
criteria can be matched that fits the observations, are generally, used to broadly classify zones
based on different states of stress (i.e. black boxing the behaviour), and by doing this there is
loss in the ability to identify the specific processes of failure and behaviour that are occurring.
Ortlepp (2005), notes that “although much has been gained in processing of microseismicity,
researchers are still searching for the ‘holy grail’ of adequate understanding of the rockburst
mechanism”, and of the mechanisms of microseismicity itself. The analysis of source
parameters carried out at the Golden Giant Mine (Chapter 4) was found to shed some light on the behaviour of the failing rock mass and relationship of seismicity, and gives insight into the behaviour of the fracturing process with notable changes in the key source parameters during yield.

2.3.1 Seismic Source Parameters

A more detailed discussion of the seismic source parameters used in this research can be found in Gibowicz et al. (1991), Urbancic (1991), Bird (1993), and Mendecki (1996), and will only be summarised here with key equations summarised in Appendix A3. The primary software used to acquire the full waveform data and calculate the source parameters is based on the methodology laid out in Urbancic (1991), and incorporated into the ESG Hyperion Software (ESG, 2006) used to process the seismic records in this research.

The key source parameters based on the double-couple source model (see Section 2.3.2) that are presented in this research are: measures of the source strength; the seismic moment, $M_o$, (Aki, 1968), the linearized scale of logarithm of $M_o$, the moment magnitude, $M$ (Hanks and Kanamori, 1979), and the total radiated seismic energy, $E_o$ (Snoke, 1987) the sum of the contributions of the energy of the P- and the S-wave components (Appendix A3.2). A typical full waveform seismogram recorded in the cluster of events analysed for the Golden Giant case study is shown in Figure 2.4. The seismic moment and energy are source model independent descriptions of the strength of the event at its source, and are used in the calculation of other source parameters and are probably the best scalar descriptions of the rupture process.

The ratio of component radiated energies, $E_s/E_p$ ratio, has been used by authors (Gibowicz et al., 1991; Urbancic, 1991; Cai et al., 1998) to classify pure shear events ($E_s/E_p > 10$ to 20) from events that have enriched P- or depleted S-wave energies ($E_s/E_p < 10$) and may indicate failure mechanisms with non-shear volumetric (dilatational or tensile) components, such as combined shear-tensile failure, and is investigated in Chapter 4. The description of the size of the source used here, the source radius, $r_o$, (which is model dependent) is based on the quasi-dynamic Madariaga (1976) circular rupture model (Appendix A3.3), which has been found to give a reasonable approximation to the rupture or fracture size found in underground mines (Gibowicz et al., 1991), and also at this site as discussed in the previous section.
a. Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_0$), corner frequency, ($f_c$) and energy flux, ($J_c$), fitted with a Brune $f^{-2}$ spectral decay.

$\Omega_0 = 1.87 \times 10^{-11}$

$J_c = 1.17 \times 10^{-10}$

$\Omega_0$ (low frequency plateau)

$J_c = 1.17 \times 10^{-10}$

$\Omega_0$ (low frequency plateau)

$J_c = 1.17 \times 10^{-10}$

$\Omega_0$ (low frequency plateau)

$J_c = 1.17 \times 10^{-10}$

$\Omega_0$ (low frequency plateau)

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$\Omega_0$ (low frequency plateau)

$J_c = 1.17 \times 10^{-10}$

$\Omega_0$ (low frequency plateau)

$J_c = 1.17 \times 10^{-10}$

$\Omega_0$ (low frequency plateau)
These key source parameters are found from the three fundamental spectral parameters observed in the frequency domain (Figure 2.4c): the \textit{low frequency spectral level or plateau}, $\Omega_0$ (the basis for the seismic moment which is proportional to this and an average of the P- and S-wave contributions), the \textit{corner frequency}, $f_c$, the intercept of the lower frequency level and the slope of the high frequency spectral decay assuming a $f^{-2}$ slope (the basis for the source radius which is inversely proportional and again an average of the P- and S-wave contributions), and the \textit{energy flux}, $J_e$, the integral of the squared ground velocity (the basis for radiated energy which is proportional to this, but the sum of the P- and S-wave contributions).

For the Golden Giant Study due to the large amounts of data recorded these spectral parameters are automatically calculated using a time-domain methodology adapted from Andrews (1986) and outlined by Urbancic \textit{et al.}, (1996) (see Appendix A3.1), and provides an objective and stable methodology for determining source parameters with less than 4% difference from frequency domain calculated parameters, (i.e. removes practitioner bias and allows for fast real-time evaluation of source parameters).

Other source parameters used in this study are the measure of seismic stress release such as the \textit{static stress drop}, $\Delta\sigma$, (after Brune, 1970) an estimate of the shear stress release, equal to the ratio of $7/16 M_o/r_o^3$, and also the \textit{dynamic stress drop}, $\Delta\sigma_d$, (after McGarr, 1991) which is proportional to the maximum recorded acceleration and sensitive to the most energetic sub-events, (Trifu \textit{et al.}, 1995). These are both model dependent parameters, here based on the Madariaga source radius and model coefficients respectively for consistency (Appendix A3.4).

A measure of the \textit{source geometric complexity}, i.e. whether the source is a simple homogenous event, (e.g. simple shear or an isolated fracture), or more complex inhomogeneous event, (e.g. interacting shears/fractures with barriers or asperities), can be found from the ratio of the dynamic to static stress drops. Gibowicz and Kijko (1994) have suggested that for a simple homogenous event, with a constant rupture velocity throughout the rupture area, that $\Delta\sigma_d$ and $\Delta\sigma$ should be similar, resulting in ratio’s close to 1.0, while complex sources, have a ratio, $\Delta\sigma_d/\Delta\sigma$, greater than 1.0.

A third commonly used stress estimate is the \textit{apparent stress}, $\sigma_a$, (Wyss and Brune, 1968), which is the ratio of $E_o/M_o$ multiplied by the shear modulus\textsuperscript{1}, and some researchers have suggested, similarly with the other stress estimates, to be proportional to the magnitude of the induced stress (Mendecki, 1993; Simser and Falmagne, 2004). High stress regions tend to

\textsuperscript{1} Here from the wave equation Shear Modulus, $\mu = \rho \beta^2 = 2700 \text{ kg/m}^3 \times 3500^2 \text{ m/s} = 33.1 \text{ GPa}$
release more seismic energy and tend not to allow as much deformation due to high clamping, (high apparent stress), while low apparent stress events may be due to lower stresses, or areas that have shed stress due to rock mass fracturing, (Simser and Falmagne, 2004). Mendecki (1993), notes that the apparent stress is model independent. For mine seismicity it has been observed that, as the P-wave energies cannot be neglected, as in earthquake seismology and the study of pure shear events, the apparent stress, which in the later case would be expected to be of similar magnitude to the static stress drop, is generally much lower and can be considered an independent parameter (Gibowicz et. al., 1991). Thus, Mendecki (1993) proposed the apparent volume, $V_a$, the ratio of $M_o / 2 \sigma_a$, (as generally in mining $\Delta \sigma \geq 2 \sigma_a$), as a scalar estimate of the volume of coseismic inelastic deformation. If a spherical volume is considered this gives rise to the apparent radius, $r_a$, (Cai et. al., 1998) defined as $3 \sqrt[3]{(3V_a / 4\pi)}$, which is expected to be larger than, but proportional to, the source radius. The apparent radius is used here as a comparison to temporal changes in the source radius, being independent of the corner frequency.

### 2.3.2 Seismic Source Mechanisms

#### 2.3.2.1 Overview

In earthquake seismology the most common force system used to describe the rupture process is the based on the point double-couple (Figure 2.5a) representing slip on a fault and its corresponding radiation pattern (Urbancic and Young, 1992). Two predominant methods of determining various source mechanisms are: through Fault Plane Solutions (FPS) based on first motion studies only described by Urbancic (1991) and through Moment Tensor Inversion (MTI) described by Trifu and Shumila, (2002a). Investigation of seismic mechanisms have been made at various mines by using FPS (Gibowicz, 1989; Urbancic, 1991; McCreary et. al., 1993; Connors et. al., 1993; Urbancic et. al., 1993; Sampson-Forsythe, 1994) to rationalize the fault planes generally to geological structure, and using MTI by Feigner and Young (1992), Mendecki (1996), Young and Collins (2001), Trifu and Shumila (2002a) for example to rationalize the fault planes to geological or fracture structures, but to also investigate the proportions of non-double couple pure shear components in various locations. Generally, it has been found that close to openings, the mechanisms can have a large tensile component related to tensile fracturing (Feigner and Young 1992; Young and Collins, 2001), while those further away related potentially to geological structure can be predominantly shear in nature (Trifu and Shumila, 2002a).
In the case of FPS, the assumption is that the source is pure shear in nature with no volume change, and requires the P-wave first motions, the source location and the azimuth or plunge of the assumed straight ray path from the source to the receiver. In the case of moment tensor inversion for the general solution, the source does not have to be considered purely shear but is characterised by the second order seismic moment tensor (3 x 3 matrix) of the 9 elemental force couples (Aki and Richards, 1980), which are related to the measured displacements at the sensors by the elastodynamic properties of the medium (the ‘Greens Function’). Decomposition of the moment tensor to it’s eigenvalue and eigenvectors can result in three potential models based on the trace of the eigenvalues describing the volume change, of the isotropic components (explosive or implosive), double-couple components (pure shear) and deviatoric components (compensated linear vector dipole, CLVD [complex shear]). The various proportions of these components can give insight into the source mechanism, and also obtain fault plane solutions describing the rupture orientation. Evaluation of moment tensor inversion requires all of the same components as for FPS, with the addition of the recorded amplitudes of the system, the elastic properties of the medium and the assumption of a homogenous, isotropic and infinite medium. In order to solve for the 6 independent components in the general moment tensor solution a minimum of 6 independent displacements recorded by the sensors are required. Theoretically a minimum of 2 triaxial sensors could be used to solve the solution but from a practical standpoint in order to obtain stable solutions from triaxial sensors a minimum of 4 to 6 sensors are required (Trifu pers. comm., 2008). Also, even at this number of sensors it is recommended to compliment the solution with the incorporation of uniaxial sensors (Trifu and Shumila, 2002b). At the Golden Giant mine 4 triaxials sensors where available, however, due to issues in obtaining valid sensor orientations, (Chapter 5), it was not possible to obtain stable general moment tensor solutions even with the incorporation of uniaxial sensors. Hence the primary method used to identify potential source mechanisms was based solely on first motion studies. It was found by Urbancic (1991) that even if the sources are depleted in shear wave energy (Es/Ep ratios < 10), and having a large dilatational component, reasonable fault plane solutions can be obtained and suggest that the mechanism is a combined shear-tensile rupture.

The use of first motions on AE has also been reviewed by Lockner (1993) who identified from a number of studies that the resultant AE produced from uniaxial and triaxial tests, indicates possible double-couple quadrant polarities but predominantly mixed polarities with no double couple solution suggesting complex shearing. In more recent studies using MTI, Pettitt (1998) confirmed that from polyaxial testing these complex shear type failures predominate, with also compressional failures postulated as asperity crushing. Thompson et. al. (2005) who performed
experiments on a saw cut sample of westerly granite tested triaxially, found that prior to slip in the pre-peak the moment tensors of AE were generally compressive source mechanisms indicating grain crushing while in the post peak or post slip the mechanism conformed more to a double-couple mechanism with fault plane solutions mimicking the orientation of the saw cut. This indicates that MTI's have the potential to more readily track changes in the rupture process than can be inferred with FPS based on first motion studies alone, and for future studies like this one would be preferred.

2.3.2.2 Source Mechanisms from First Motion Studies

As previously mentioned determination of focal mechanisms by first motions is made on the basis of a double-couple force system characterised by a quadrilateral seismic radiation pattern, (Figure 2.5a), and is based on the idealized Mohr stress circle in that shear, (orientation of maximum shear stress), occurs at 45° to the applied couple. The intersection of the two planes (the nodal planes) is the null axis (B-axis out of plane); the pressure axis (P) bisects the nodal planes in the direction of maximum compressional motion (i.e. 45° to the nodal planes); and the tension axis (T) bisects the nodal planes in the direction of the maximum dilatational motion (i.e. 45° to the nodal planes). The quality of the fault plane solutions is directly related to the source sensor positioning, the number of polarity readings and the fit to a double-couple solution. Urbancic (1991) has suggested that a minimum of 15 polarities with 'good' focal sphere coverage are required to obtain adequate solutions, based on synthetic data testing.

By using the distribution of the polarities in azimuth and take-off angle around the event location, information is obtained on the wave radiation from the source assuming far field conditions. The measured P-wave polarities recorded at the sensors are projected onto the unit focal sphere, and translated to an equal area lower-hemisphere stereonet to give the fault plane solution (Urbancic, 1991). If a double-couple mechanism, (representing shear-slip), on a plane is present then, based on the relative distribution of polarities, the faulting type can be inferred.

Figure 2.5b and c shows typical faulting mechanisms and their resultant fault plane solutions for double-couple mechanisms. For non double-couple mechanism if the resultant polarities are predominantly -ve this infers an implosion (compressive failure) and has been suggested by Hasegawa et. al. (1989) as possibly being represented as a pillar burst, or wall burst (Figure 2.5b).
Figure 2.5  Focal Mechanisms based on first motion studies. (a) Example of double-couple "pure shear" event with P-wave radiation patterns (after Urbancic and Young, 1992). (b) Non double-couple mechanisms (after Hasegawa et al., 1989) and (c) Double-couple mechanisms with principal stress orientations (after Ramsay and Huber, 1987).
If the resultant polarities are all +ve this infers an explosion (tensile failure), and can be inferred as a cavity collapse or tensile fault rupture Hasegawa et al. (1989), or tensile fractures close to excavation surfaces or in the core of pillars (McCreary et al., 1993; Young and Collins, 2001; Trifu and Shumila, 2002a). Additionally, explosions created by blasting will also form this mechanism and is the method used to confirm the orientation of sensors and calibrate the polarities prior to carrying out first motion studies of microseismic events (Chapter 5). The additional non-double couple mechanism, which is not illustrated, is produced from a mixture of polarities with no clear solution and could be classified as a complex shear style failure. However, in practice it is found that the algorithm used will generally always find a solution, the quality being measured by the fit of the polarities to that solution.

Shear type failures in which double-couple solutions can be determined, are classified as normal, reverse (thrust), strike-slip and combined strike-slip dip-slip faults, as illustrated in Figure 2.5c. In this illustration the relative orientation of the principal stresses to induce these failures has been included. It should be pointed out that the type of double-couple shear fault is really based on the point of observation in relation to the stress field, and by geological convention normal faults have the major principal stress oriented vertically (P-axis is vertical and T-axis sub-horizontal), while reverse faults have the major principal stress oriented sub-horizontally (P-axis is sub-horizontal and T-axis is vertical). For a pure strike-slip event both the P- and T- axes are horizontal, and for combined strike- and dip-slip events the P- and T- axes are rotated from the horizontal and vertical.

As can be noted for these fault planes, two possible planes, (nodal planes), are defined based on the double-couple, which are the fault plane and the auxiliary plane. Determination of the nodal planes is made, based on the maximum-likelihood algorithm as described by Brillinger et al. (1980) and Udias et al., (1982) and discussed in more detail in Urbancic (1991). In this research the commercial package, Visual Fault plane solutions (VFps - ESG, 2006) is used based on this theory to determine fault plane solutions and is part of the ESG Hyperion Software (ESG, 2006). Identification of which of the nodal planes is the actual fault plane must be made with additional information. This assessment can be made through identification of clusters of nodal poles with a knowledge of the stress regime, based either on a fundamental mechanical understanding of the likely shear plane based on the loading conditions, or using stress inversion techniques (Gephart and Forsyth, 1984; cited Urbancic et al., 1993), and/or comparison to geological structural information. Urbancic et al., (1993), identified correlation of fault plane solutions to geological joint structure, stress inversion and the dominant PCA trend at two mines sites. However, at both sites although there was a strong correlation between fault
plane solutions and the PCA derived planes, correlation with the predominant joint structure was weaker. It is suggested based on the research presented in this thesis that the more likely correlation is not to geological structure but to intact fracturing of the rock mass. Analysis of fault plane solutions with the PCA technique is used as confirmation and investigation into the rupture orientations (Chapter 5).
CHAPTER 3

INVESTIGATION OF THE PRINCIPAL COMPONENT ANALYSIS (PCA) TECHNIQUE TO DETERMINE FAILURE STATES - A CASE STUDY WILLIAMS MINE, CANADA.

3.1 Introduction

The case study mine presented in this chapter provides a unique opportunity to examine and analyse rock mass failure into the post-peak, through the regional and confined footwall failure of a highly stressed sill pillar at the Williams Mine. In the chapter the author examines the use of the spatial and temporal distribution of seismicity over a 5 year period of progressive failure into the post-peak region, from microseismic initiation to aseismic behaviour, relating the seismic event density, combined with the temporal examination of the principal components analysis (PCA) outlined in Chapter 2, to characterize the extent, trend and state of the yielding zone, which has been interpreted as the formation of a macrofracture shear structure. This data is used in conjunction with conventional displacement instrumentation, SMART cables and SMART MPBX, (Bawden et. al., 2000a), which were installed during the failure of this sill, below where a significant amount of the microseismicity developed. This displacement data is used to aid in the identification of failure and indicates regional dilation of the rock mass at complete failure when approaching the residual strength. Three-dimensional linear elastic modeling is used to understand the loading path and calibration of damage limits for comparison to other research, and two-dimensional non-linear modelling is performed to understand the required post peak parameters that will produce similar model conditions to those observed in the field. The emphasis is to determine if it is possible to identify different stages of failure from pre-peak to the post peak, using microseismicity, and to better understand the behaviour of the rock mass as it transitions through failure.
3.2 William Mine Sill Pillar Study

3.2.1 Overview of Hemlo Mining Camp

The Hemlo camp is located in Northern Ontario, on the North shore of Lake Superior between the towns of Marathon and Manitouwadge and is composed of three mines; Williams Mine, Golden Giant Mine and David Bell Mine (Figure 3.1). Each operation is mining the same gold rich tabular steeply dipping orebody, using sub-level open stope mining for the thicker ore (7 to 40 m) and alimak mining, at Williams and David Bell, for the thinner ore (< 7 m). Historical backfilling of stopes used cemented rock fill for primary stopes and sand-fill for secondary stopes, but each operation has progressed to cemented paste backfill. The three operations have varying production rates of 6,000, 2,500 and 1,500 tonnes per day respectively employing different mining sequences. This site was chosen for this study, as two of the mines, Williams and Golden Giant mines (Figure 3.1), were mature mines with relatively high extraction ratios and as a result relatively high mining induced stress that had caused a number of ground control issues, resulting from stress driven failure (Nickson et al. 1998; Leblanc and Murdoc, 2000; Bawden & Jones, 2002). It was anticipated that these failures would become more pronounced as mining continued. Also, both mines had microseismic systems with a number of years of data collection at the commencement of the study (2002), and also had extensive displacement monitoring of important infrastructure. This case study concentrates on the extraction of a portion of the main Williams mine sill pillar located at a depth between 872 (9450L) to 952 m (9370L) below surface, and between stopes 28 and 16, approximately 240 m along strike (Figure 3.1 Area A and Figure 3.2).

3.2.2 Williams Mining History Overview

A detailed description of the original mine design is documented by Bronkhurst et al. (1993). The main design concept at Williams focused on employing primary–secondary sub-level open stoping (25 m sub spacing, 20 m stope strike length), initiated on two mining horizons. The upper block 3 is sequenced to develop an overall pyramid shape, thus shedding stress off to the east and west abutments (Figure 3.2). The second lower horizon (block 4) was mined by primary-secondary stoping, originally to be mined as a full pyramid however, due to ground control problems, this was not possible. The intension was to leave a 50 m high sill pillar between the two horizons, and then mine in a pillarless retreat fashion from east to west, to reduce the risk of high stress anticipated from modelling (Bronkhurst et al., 1993). In order to
achieve this change in sequence a 40 m wide under-cut was required two levels below block 3. These large back dimensions (ore thickness was over 40 m in the far east end of the sill), in high stress ground were probably one of the main reasons for the onset of ground stability problems in the sill. Initial problems occurred in 1996, with stress driven caving occurring on the far east end of the sill pillar in stopes 2 to 4 (Figure 3.2). This was followed by structurally related caving at the western end of the central sill region in stopes 28 and 30 (Figure 3.2a), as discussed by Leblanc & Murdoch (2000). The sill region was known to be microseismically active in regions of high stress at this time. In March 1999 the first significant large seismic event (3.0 Mn) occurred, causing significant damage to the footwall development (Bawden et. al., 2000), especially on the 9415 level (Figure 3.2a) and led to the installation of a microseismic system.

Figure 3.1 Longitudinal 3D view of the Hemlo mines, and solid model of mining geometry. (A) Williams Mine Sill Pillar Region, (B) Golden Giant Mine Shaft Pillar Region. Shades, represent yearly mining up until September 2005. Inset geographical location of Hemlo Camp in Ontario, Canada.
The 9415 footwall (FW) drift in this region was subsequently abandoned and a bulkhead emplaced, with the intention of accessing the ore zone from the hangingwall. Rockburst support (shotcrete and 10 m debonded Garford cables) was installed on the levels below 9415L in the sill region, and was fully instrumented with SMART MPBX and SMART cables (DaGraaf et al., 1999; Bawden et al., 2000). Other indications that the sill was highly stressed where the crushing of blast holes, dog earing and failure of in-stope raise bores, and the triggering of seismic flurries through blasthole drilling activity, making mining difficult. For the next 3 years, mining in the sill was greatly reduced, but the sill pillar still progressed to failure due to regional increases in induced stress, and 10 large > 0 Mn events were recorded, most thought to be a result of a foliation shear identified to the south of the FW haulage (Bawden and Jones, 2005). The 9390 L drift, although remaining open during this time, showed significant deformation on support (>50 mm). A section of this level, between stopes 26 and 23, was abandoned and tightly paste backfilled in late August 2003. This followed a pillar strain burst (2.7 Mn) on the haulage x-cut pillar between stopes 26 and 25 in May 2003.
Rehabilitation attempts found the back of the drift to be highly fractured to at least a depth of 7 m making support of this drift impractical (Bawden, 2003). The last major seismic event in the sill region, occurred in September 2003 (3.5 Mn), following the mining of the east end of the central sill pillar. Mining of the sill pillar stopes commenced in 2004, with no incidence, as the area became aseismic from east to west throughout most of the sill (Bawden and Jones, 2005).

The ease in mining without stress related problems is significant as rock mass failure, based on the location of microseismicity, was predominantly between the footwall haulage drives of the sill region (Section 3.7.1). This indicates that there was a significant decrease in mining induced stress to a level below the fracture initiation stress, causing stress shadowing of the main ore zone, which allowed for mining of the stopes from late 2003 to 2005, without the previous stress related problems. Based on this, it has been inferred that significant strain-softening (brittle behaviour) of the footwall region occurred to cause stress shadowing of the ore zone.

The area of interest for this study covers this central sill pillar region below block 3, from installation of the microseismic system in 1999 to the last period of on-site data collection in February 2005. During the course of the study a total of 5 site visits were made to the Hemlo Camp to collect data, from surface databases, discuss the observations of the mining personnel and observe the progress of the underground operation to relate modelling results, and seismic observations with stress induced failure.

3.3 Geological Overview

The Hemlo orebody lies on the south side of the east-west striking Heron Bay belt of metamorphosed Precambrian rocks, and found at the contact between the overlaying metasedimentary rocks of the Cedar Creek formation and underlying metavolcanics (LeBlanc and Murdock, 2000) of the Moose Lake Porphyry Complex, and Rule Lake Member. Johnston (1996) gives a full description of the geology of the Hemlo deposit, and concludes that the gold enrichment in the ore zone is formed as a result of an intrusive complex characterised by multiple, nested epizonal porphyritic stocks and multiple felsic and mafic dykes, with several episodes of hydrothermal alteration and mineralisation associated with the intrusive complex, compounded by later stage deformation and metamorphism. Johnston (1996) found no evidence for the ore body being associated with a major fault structure as had previously been suggested.
The main geologic components, which make up the deposit (Figure 3.3 – showing geological mapping of the 9390 level representative of the region of interest in this study) tend to be similar in structural nature and geomechanical properties, characterised mainly by foliation. The immediate footwall schists (3 units\(^1\)), transition into the ore zone (5 units), these being an altered but mineralised extension of the 3 units, with disseminated gold enrichment over the zone (David Bell, 1999), with the hangingwall formed by metasediments (7 units). This foliation is also characterised by the felsic porphyry intrusives stock and mafic sills, which run parallel to the ore horizon with a thickness of a 0.1-0.5 m and spaced 0.2-2 m apart, found throughout all zones. For the most part the mineralisation is contained within one main tabular economic lens, although smaller sub-parallel lenses do occur, and at the Williams mine the main zone splits into two economically separate lenses West of the central sill pillar. The main ore zone varies in thickness from 3 to over 40 m, the thinner regions being at the periphery of the ore body, the average thickness at Williams is in the order of 25 m. The dip of the foliation and ore body varies from 60° at the David Bell and Golden Giant mines east of the Golden Giant shaft, to 70° which predominates throughout Williams Mine.

**Figure 3.3** Typical geology plan: Williams mine 9390 L in the sill pillar region stopes 23 to 15.

\(^1\) Each mine's geology department used slightly different unit numbering systems and descriptions of the same geological units. The unit numbering system described here is appropriate for the Golden Giant and David Bell mines of which the author is most familiar (the only major difference is that Williams mine calls the hangingwall metasediments unit 4 instead of 7, and the feldsparitic and sericitic ore unit 6 instead of 5) Note also that the baritic ore 5d (5-20% barite), is not the same as the barite zone, 5i, which has a high level of barite> 20% and can be considerably weaker.
The other main structural elements, are the late stage intrusive dykes and shears. Two main types of dyke exist, the large Diabase dykes and smaller Lamprophyre dykes (Figure 3.3). The former can be 5 to 50 m thick and persistent over thousands of metres, being vertical in nature and crossing the lithology at high angles. Three of these Diabase dykes intercept the orebody at the Williams mine, the main largest of these is the ‘Lac dyke’ that is found on the Eastern boundary between the Golden Giant and Williams mines, the other two are smaller, the central dyke subdivides stopes 16 and 17 and the western dyke runs through stope 53 (Figure 3.2). The lamprophyre dykes also cross cut the foliation at high angles, but are of only 0.2 to 1 m thickness and only persist over 10’s of metres. These tend to be sparsely populated, and are mainly noticed at the Williams mine. When they occur in small swarms however, can cause structural instability (Leblanc and Murdoch, 2000). Although there are no major faults within the entire Hemlo camp, small shears associated and parallel to the dyke contacts occur, as well as foliation shears which, as the name suggests, run parallel to the foliation (Figure 3.3). These later shears can be filled with sericitized or gouge material a few centimetres thick and can persist over 10 to 30 m (Kazakidis, 1990). Again both are sparse, but where present can present structural issues.

3.4 Geomechanical Properties

3.4.1 Joint Mapping

The four prominent joint sets that are pervasive through each domain (hanging wall (HW), ore zone (OZ) and footwall (FW)) are displayed in Figure 3.4 and summarised in Table 3.1. This dataset is based on mapping at the Williams mine (Bronkhurst et al., 1993) and mapping performed at Golden Giant, predominantly west of the shaft (Kazakidis, 1990), and confirmation mapping during site visits. The prominent joint set as previously mentioned is the foliation set, joint set A, dipping north 70°, and striking east-west. The other major joint set is the steeply dipping (~90°) B-set striking North-South. A sub-horizontal joint set C is present, however it tends to be more sparsely distributed than the other two sets even when vertical mapping exists. The C-set is used in stability analysis as a worst case, however, in the region of the sill pillar this set does not predominate. It is also important to note that the B- and C-set tend to have a lower persistence that the A-set, and traces are often seen to terminate at a dominant A-set structure. All of these sets, regardless of domain, tend to have similar joint properties, being smooth to slightly rough and planar, \( J_r = 1.0 \) (Barton et al., 1974), with none to minor joint alteration, \( J_a = 0.75 \) to 1.5 (Barton et al., 1974). The exception is for the C-set which can
often have a calcite coating $Ja = 1.5$ to $2.0$, and the A-set at the immediate HW which can sometimes contain mica schist, $Ja = 3.0$ to $4.0$. The other set that exists is the D-set, (and conjugate E-set). This set is even more sporadic than the C-set, and can be considered to be random, and is generally not found within the ore zone. Often when it is identified, the joint surface sometimes show evidence of slickensides, suggesting that it may be a minor shear. At Williams this set has mostly been identified in the upper east footwall of block 3, however, in the area of the sill pillar region, no major occurrence has been noted.

<table>
<thead>
<tr>
<th>Joint Set (Mean Orientations)</th>
<th>Strike (rhr)</th>
<th>Dip</th>
<th>$Jr$</th>
<th>$Ja$</th>
<th>Spacing (m)</th>
<th>Persistence (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A – Foliation Joints</td>
<td>281</td>
<td>70</td>
<td>1.0</td>
<td>1-1.5</td>
<td>0.125 – 1</td>
<td>3-10</td>
</tr>
<tr>
<td>B – Vertical Joints</td>
<td>004</td>
<td>86</td>
<td>1.0</td>
<td>0.75-1</td>
<td>0.5 – 2</td>
<td>1-5</td>
</tr>
<tr>
<td>C – Sub-Horizontal Joints</td>
<td>103</td>
<td>11</td>
<td>1.0</td>
<td>1.0-2.0</td>
<td>0.5 – 3</td>
<td>1-5</td>
</tr>
<tr>
<td>D- Sub-vertical “shear”/ joint</td>
<td>001</td>
<td>47</td>
<td>0.5-1</td>
<td>0.75-1</td>
<td>2- 10</td>
<td>3-10</td>
</tr>
<tr>
<td>E- Sub-vertical “shear”/ joint</td>
<td>181</td>
<td>45</td>
<td>---</td>
<td>---</td>
<td>2-10</td>
<td>---</td>
</tr>
<tr>
<td>F- Vertical Joint Set</td>
<td>045</td>
<td>89</td>
<td>---</td>
<td>---</td>
<td>0.5-1</td>
<td>---</td>
</tr>
</tbody>
</table>

Table 3.1 Summary of Main Joint Set and Properties (after Kazakidis, 1990 and Coulson, 2004)

1 Persistence is estimated from observations of back and hangingwall failures at Williams, Golden Giant and David Bell Mines by the Author.
3.4.2 Material Properties

A number of intact laboratory testing campaigns, on core samples, for the various units have been performed at the Hemlo Camp and are summarised in Table 3.2. For the majority of the rock units, the intact rock fabric can be described as a fine to medium grained, crystalline rock with evidence of schistocity. The microscale effect of this schistosity is evident from the point load tests, in which comparison of diametral to axial tests, indicate on average a 50% increase in strength when samples are tested perpendicular to the direction of foliation, and indicate transverse anisotropy characteristic of metamorphic rocks (Nasseri et al., 2003). On the large scale the rock mass can be considered to be transversely isotropic in properties, and this is also noticed through seismic velocity surveys (Kazakidis, 1990). As can be seen from Table 3.2, the uniaxial compressive strength (UCS), varies between 100 to over 200 MPa, and the mean strength is around 175 MPa for the footwall and ore zone and approximately 160 MPa for the hanging wall units. The main variance for the values of UCS are predominantly a result of the foliation that is pervasive through all domains, however, for the most part because the rock mass is preferentially loaded through mining induced stress perpendicular to foliation [attaining close to the peak strength of the intact rock based loading orientation in relation to anisotropy, (Hoek and Brown, 1980)], the axial strength is more representative. A mean UCS value of 175 MPa has been used in both cases studies to allow for direct comparison, and although analyses could be performed using the spread of strengths this complicates comparisons. The notable lower strength unit within the ore zone, is generally found in the barite rich unit (Barite > 25%), and greater instability and stress fracturing has been observed by mine personnel when this unit is pervasive. Generally this unit is not dominant, so has been separated out from the overall average (additionally, although anecdotal evidence supports the lower strength, this is based on limited testing).

For the most part the material properties are relatively constant across domains. Limited triaxial testing has been performed at the Hemlo camp, however, based on a series of tests performed on core (mineralogy unknown) from the Golden Giant mine (Queens University, 1994), the value of m\(_i\) (Hoek and Brown, 1980) can be determined to lie between 13 and 17. Crowder et al. (2006) used a value of \(m_i = 10\) (based on suggested value for schists; Hoek et al., 1995), and although this might be conservative, from back analysis appears to be reasonable.
### Table 3.2 Summary of Mechanical Testing Performed at the Williams and Golden Giant Zone Unit Point (1) Load Diametral Point (1) Load Axial UCS (2,3) E (2,3) ν (2,3) T (3,4) Mean (MPa) S.D. Mean (MPa) S.D. Mean (MPa) S.D. Mean (GPa) S.D. Mean (SxD10^-3) Mean (MPa) S.D.

<table>
<thead>
<tr>
<th>Zone</th>
<th>Unit</th>
<th>Load</th>
<th>Diametral Point (1)</th>
<th>Load Axial</th>
<th>UCS (2,3)</th>
<th>E (2,3)</th>
<th>ν (2,3)</th>
<th>T (3,4)</th>
</tr>
</thead>
<tbody>
<tr>
<td>HW</td>
<td>7b</td>
<td>92</td>
<td>(5) 29 144 (2) 9</td>
<td>175 (5) 64</td>
<td>53.9</td>
<td>7.4</td>
<td>0.271</td>
<td>76</td>
</tr>
<tr>
<td></td>
<td>7d</td>
<td>110</td>
<td>(7) 43 153 (7) 29</td>
<td>207 (3) 54</td>
<td>67.6</td>
<td>22</td>
<td>0.329</td>
<td>48</td>
</tr>
<tr>
<td></td>
<td>7f</td>
<td>75</td>
<td>(2) 9 143 (2) 23</td>
<td>125 (5) 26</td>
<td>46.4</td>
<td>11</td>
<td>0.240</td>
<td>38</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Overall</td>
<td></td>
<td>163 54 54.2</td>
<td>14.5</td>
<td>0.272 64</td>
<td>14.0</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>163 54 54.2</td>
<td>14.5</td>
<td>0.272 64</td>
<td>14.0</td>
</tr>
<tr>
<td>OZ</td>
<td>5a/b</td>
<td>97</td>
<td>(2) 18 136.3 (2) 9</td>
<td>171 (4) 25</td>
<td>57.7 7.8 0.255 95</td>
<td>14.1 2.6</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>5b</td>
<td>79</td>
<td>(6) 31 225 (7) 36</td>
<td>194 (3) 46</td>
<td>53.4 10.4 0.336 30</td>
<td>17.1 4.4</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>5d</td>
<td>92</td>
<td>(10) 28 122 (10) 36</td>
<td>164 (4) 36</td>
<td>60.3 10.5 0.258 34</td>
<td>10.5 2.1</td>
<td></td>
<td></td>
</tr>
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</tr>
<tr>
<td></td>
<td>5i</td>
<td></td>
<td>Overall</td>
<td></td>
<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Overall no 5i (Barite rich Zone)</td>
<td>174.6 34.3 57.5</td>
<td>9.6 0.278 68</td>
<td>13.6 4.2</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
<td></td>
</tr>
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<td></td>
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<td></td>
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<td></td>
<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
<td></td>
</tr>
<tr>
<td>FW</td>
<td>3a/b</td>
<td>87</td>
<td>(7) 37 208 (3) 8</td>
<td>177 (3) 19.4</td>
<td>55.3 5.9 0.292 23</td>
<td>19.3 3.3</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
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<td></td>
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<td></td>
<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
<td></td>
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<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
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<td></td>
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<td></td>
<td></td>
<td></td>
<td>163 41.3 58.2</td>
<td>9.2 0.277 69</td>
<td>13.4 4.5</td>
<td></td>
</tr>
</tbody>
</table>

Note:  
1. Point Load Testing Golden Giant Mine (Noranda Technology Centre, 1988)  
2. Strength and Elastic Constants for Golden Giant Mine (University of Toronto, 1988)  
3. Strength and Elastic Constants for Williams Mine (Golder Assoc., 1986)  
4. Tensile Testing (Noranda Technology Centre, 1988)  
(Subscript represent the number of samples tested)

### Table 3.3 Summary of Rock Mass Classification At Williams Mine (Based on mapping at Williams, Golden Giant and David Bell Mines)

<table>
<thead>
<tr>
<th>Domain</th>
<th>UCS (MPa)</th>
<th>Joint</th>
<th>RQD</th>
<th>Jn</th>
<th>Jr</th>
<th>Ja</th>
<th>Q'</th>
<th>σ’ (MPa)</th>
<th>SRF</th>
<th>Q</th>
<th>A1</th>
<th>A2</th>
<th>A3</th>
<th>A4</th>
<th>A5</th>
<th>B</th>
<th>RMR89</th>
<th>GSI</th>
</tr>
</thead>
<tbody>
<tr>
<td>HW</td>
<td>Mean</td>
<td>163</td>
<td>A,B,C</td>
<td>80</td>
<td>9</td>
<td>1</td>
<td>1</td>
<td>8.9</td>
<td>40 5</td>
<td>1.8</td>
<td>12</td>
<td>15</td>
<td>10</td>
<td>20</td>
<td>15</td>
<td>0</td>
<td>72</td>
<td>67</td>
</tr>
<tr>
<td></td>
<td>Max</td>
<td>200</td>
<td>A,B,C,D®</td>
<td>90</td>
<td>6</td>
<td>1</td>
<td>1</td>
<td>15.0</td>
<td>30 1</td>
<td>15.0</td>
<td>12</td>
<td>17</td>
<td>10</td>
<td>24</td>
<td>15</td>
<td>0</td>
<td>78</td>
<td>73</td>
</tr>
<tr>
<td></td>
<td>Min</td>
<td>100</td>
<td>A,B,C,D®</td>
<td>50</td>
<td>12</td>
<td>1</td>
<td>3</td>
<td>1.4</td>
<td>60 15</td>
<td>0.1</td>
<td>7</td>
<td>13</td>
<td>9</td>
<td>17</td>
<td>15</td>
<td>0</td>
<td>61</td>
<td>56</td>
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<tr>
<td>OZ</td>
<td>Mean</td>
<td>175</td>
<td>A,B,C</td>
<td>88</td>
<td>9</td>
<td>1</td>
<td>1</td>
<td>9.8</td>
<td>50 7.5</td>
<td>1.3</td>
<td>12</td>
<td>16</td>
<td>10</td>
<td>21</td>
<td>15</td>
<td>0</td>
<td>74</td>
<td>69</td>
</tr>
<tr>
<td></td>
<td>Max</td>
<td>200</td>
<td>A,B,C</td>
<td>95</td>
<td>6</td>
<td>1</td>
<td>1</td>
<td>15.8</td>
<td>30 1</td>
<td>15.8</td>
<td>12</td>
<td>17</td>
<td>10</td>
<td>24</td>
<td>15</td>
<td>0</td>
<td>78</td>
<td>73</td>
</tr>
<tr>
<td></td>
<td>Min</td>
<td>100</td>
<td>A,B,C</td>
<td>75</td>
<td>9</td>
<td>1</td>
<td>2</td>
<td>4.2</td>
<td>70 15</td>
<td>0.3</td>
<td>7</td>
<td>13</td>
<td>10</td>
<td>18</td>
<td>15</td>
<td>0</td>
<td>63</td>
<td>58</td>
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<tr>
<td>FW</td>
<td>Mean</td>
<td>175</td>
<td>A,B,C</td>
<td>82</td>
<td>9</td>
<td>1</td>
<td>1</td>
<td>9.1</td>
<td>40 4</td>
<td>2.3</td>
<td>12</td>
<td>15</td>
<td>10</td>
<td>20</td>
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<td>72</td>
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<td></td>
<td>Max</td>
<td>200</td>
<td>A,B,C</td>
<td>90</td>
<td>6</td>
<td>1</td>
<td>1</td>
<td>15.0</td>
<td>30 1</td>
<td>15.0</td>
<td>12</td>
<td>17</td>
<td>10</td>
<td>24</td>
<td>15</td>
<td>0</td>
<td>78</td>
<td>73</td>
</tr>
<tr>
<td></td>
<td>Min</td>
<td>100</td>
<td>A,B,C</td>
<td>50</td>
<td>12</td>
<td>1</td>
<td>2</td>
<td>2.1</td>
<td>70 15</td>
<td>0.1</td>
<td>7</td>
<td>13</td>
<td>9</td>
<td>17</td>
<td>15</td>
<td>0</td>
<td>61</td>
<td>56</td>
</tr>
</tbody>
</table>

Note: Max and Min are not extreme values but represent a reasonable range from observed mapping.  
Q’ – Modified rock mass quality index (after Mathews et al., 1981; Potvin, 1988)  
Q – Norwegian Geotechnical Institutes Q-rating ((Barton, 1974); SRF after (Hutchinson and Diederichs, 1996); (Jw=1)  
RMR89 – Rock Mass Rating (Bieniawski, 1989)  
GSI – Geological Strength Index (Hoek et al., 1995) (GSI = RMR89 – 5 for (RMR > 25))
3.4.3 Rock Mass Classification and In situ stress regime

From underground geotechnical mapping during this study at Williams and from other studies at Golden Giant and David Bell mines (Kazakidis, 1990; Nickson et. al., 1994; Coulson, 2004), the overall range of rock mass classification by domain has been summarised in Table 3.3. These values represent the general range that is observed at all three operations for the main ore zone region.

The modified rock mass quality, $Q'$ (Potvin, 1988) ranges from $\sim 2$ to 15, however, in the region of the sill pillar a range of 2 to 10 is more appropriate. The Geological Strength Index (GSI) (Hoek et. al., 1995), is based on the Rock Mass Rating (RMR$_{89}$) (Bieniawski, 1989), ranges from 56 to 73. Again for the sill pillar region a range of 56 to 69 is representative. As has already been pointed out there is relatively little difference in the bulk rock mass quality between each domain, the rock being described as a fair to good quality rock mass. Although local variations do exist these are not representative of the overall rock mass quality. The range of the NGI Q rating (Barton et. al., 1974) is from 0.1 to 15, the lower values reflect the influence of induced stress on the rock mass.

A number of in situ stress measurements have been carried out at Williams and the Golden Giant mines by Golder Associates using CSIR and USBM overcoring techniques (Kazakidis, 1990). Additional stress measurements were carried out at the David Bell mine by Queen’s University using the 3D CSIR (Leeman Cell) overcoring method in 1992 (David Bell, 1999). The in situ stress regime that has been used for modelling in this study and others (Nickson et al., 1998) can be summerised as:

$$\sigma_1 = 0.0437 \text{ MPa/m, } \sigma_2 = 0.0299 \text{ MPa/m, } \sigma_3 = 0.0214 \text{ MPa/m}$$

where $\sigma_1$ trends 358°, plunges 10°, $\sigma_2$ trends 093°, plunges 28°, and $\sigma_3$ trends 250°, plunges 60° with k=2.0. The far field principal stress is oriented roughly perpendicular to the orebody strike, the minimum stress being close to vertical. The majority of numerical modeling performed at all three mines over time have been based on this stress regime. The mine surface elevation at Williams is generally taken as 10321.5 m.
### 3.5 3D Mine Model and Linear Elastic Modelling

A 3D geometrical model of the three mines in the Hemlo camp was constructed in AutoCad© (2002) (Figure 3.1). The model was inherited from previous studies at the Hemlo camp (Nickson et. al. 1998) however, extensive updating was required for the areas of interest and current time frame of mining.

The model has historical stope extraction dates assigned to each 3D stope mesh on a yearly basis from 1992 to 1995 and monthly after this. This model was used as the basis for analysis of the location of seismic event clusters and seismic densities in relation to mining and was also used as the basic building block for importing the geometry into two linear elastic 3D boundary element modeling packages, Examine³D (Rocscience, 2007) and Map3D (Mine Modelling Ltd., 2006). The former package was used as a stepping stone for hybrid non-linear modeling using Phase², the procedure for which is outlined in Crowder et. al. (2005, 2006). The later package was used for detailed linear elastic modeling around excavation geometry and post processing of stress histories, using a stress averaging routine (Rizkalla, 2001; Falmagne 2002). The modelling packages were compared to ascertain the required level of discretisation in these large models to achieve a minimum difference of 2-5% in induced elastic principal stress for the same mining geometry. The two modeling packages use different boundary element techniques; the former uses the direct method of computation (Shah, 1993), resulting in determination of nodal displacements, hence is faster to recalculate stresses at different regions once run, however, takes longer to initially run at given level of discretisation than the later package which uses the indirect fictitious stress method (Mine Modelling Ltd, 2006) and is computationally more efficient, however, must be rerun from the start if a different observation region is required.

### 3.6 Williams Microseismic System

In September 1999, following the large 3.3 Mn event and failure on the 9415 level, a small 8 channel portable microseismic system by Engineering Seismology Group (ESG) Canada Inc., was installed around the sill pillar region, utilizing uniaxial accelerometers (flat response, ±3 dB, between 50 Hz and 5 kHz and sensitivity of 30 V/g). On September 22, 2000, the system was upgraded to an 80 channel ESG Hyperion system (ESG, 2006), consisting of 48 uniaxial and 4 triaxial accelerometers (flat response, ±3 dB, between 1 Hz and 5 kHz at 0.3 V/g).
Unfortunately, only one of these triaxials was within the sill pillar region, and did not perform optimally, resulting in no online source parameters.

The system is fully automated using an automated p-picking routine, and employing the Simplex algorithm using the L2 norm to locate events (Urbancic, 1999; ESG 2006) for the first pass, then p-wave picking again, over a smaller window rejecting sensors with large residuals and refining the picks with tighter signal/noise threshold, which are then relocated again using the Simplex algorithm to determine a final location. A 205 ms window of the full waveform data, sampled at 20 kHz, is captured and stored to database files with arrival times, pick information and sensor file, and the locations plus all other relevant data stored to another database. Although no velocity survey has been performed at Williams, a survey was performed in the past at Golden Giant Mine (Kazakidis, 1990), and found the mean p-wave velocity to be between 6063 m/s to 6087 m/s depending on the survey method. From this survey it was also found that the velocity can vary by up to 5% greater when parallel versus perpendicular to the foliation, indicating transverse anisotropy as discussed earlier. However, for a reasonable array using the Simplex algorithm with the L2 norm, and the mean velocity, it was found that the locations could be within 4 m of test blasts (Mercer pers. com., 2006) and thus employing a complex velocity model, is probably not warranted. At Williams a mean p-wave velocity of 6000 m/s is used with no velocity model, and verification to simple test blasts were performed at the implementation of the system (Jones pers. com., 2003).

A complete database of the all located events and their location information as well as identified blasts was obtained spanning a time period from September 1999 to February 2005, however, due to site technical difficulties some portions of the full waveform data were lost. These losses tended to occur at the most critical time frames when there was intense seismic activity (Figure 3.7a), and losses were due in part to operator error and not software. Thus, the objective was to see what could be learned from the source locations alone as this was the most consistent dataset.

### 3.6.1 Location Error and Array Errorspace

As no formal documentation existed on location accuracy at Williams, it was important to investigate the location error based on the array configuration using an error space analysis (Urbancic, 1991; Coulson, 1996). An algorithm was developed to determine changes in the sensor array configuration from the sensor files from 1999 to 2005. It was found that the array was very stable with no major changes within the sill region.
An error space analysis\(^1\) (Coulson, 1996) was conducted on the array for the sill region using the 8 sensors in the array for the portable system and indicated theoretical overall location errors in the order of 10 -15 m (Figure 3.5 a and b), with online location errors \(\sim\)7 m (ESG, 2006). The array covers the footwall region of the sill pillar reasonably, but errors rapidly increase into the hangingwall, a distance of one array radius outside of the array.

Analysis of the expanded mine wide array, for 12 sensors triggered, (determined as the average number of sensors used for sill events), improves to 8 to 10 m (Figure 3.5 c and d), with mean online errors from located events of 7 to 8 m. The array is predominantly planar in nature, and gives good location accuracy of 2 – 4 m in Easting and Depth, but slightly poorer accuracy of 6 – 8 m in Northing. However, the microseismicity that is analysed in the footwall of the sill region is in a relatively homogenous error space, such that all events are subject to the same location errors. Analysis of the online location errors for unfiltered events from a footwall cluster between 9390 and 9415 levels (see Section 3.7.2), spanning both arrays is shown in Figure 3.5e. As can be noted, these events have relatively tight error (Average Error \(\approx\) 5 m), with minimal spread, as they all fall within the sensor array. As will be discussed in the following section the main clusters that were analysed using the PCA technique were between the 9390 and 9415 levels, the average maximum dimensions of these cluster being 45 m. Hence, the relative errors are around 11%. Based on work performed by Posodas \textit{et. al.,} (1993) on perturbing synthetic data from a single plane, the determined orientation of the structure would not be significantly effected (< \(\pm\)10° in dip and < \(\pm\)1° in azimuth) for relative errors around 10%. On this basis, given that the average PCA trends are derived from the statistical contouring of poles on a lower hemisphere stereographic projection, the location errors are of an acceptable magnitude for the dimension of the cluster analysed.

\(^1\) The error space is calculated from the standard errors of the covariance matrix of the time-distance equation \((\mathbf{A}^\text{T}\mathbf{A})^{-1}\) from the sensors to chosen grid locations and by applying time reading error based on 0.75 ms picking error such that the reading error is \(= 0.00075(N_{\text{st}}/(N_{\text{st}}-1))^{0.5}\), where \(N_{\text{st}}\) is the number of stations used (Trifu, 1983; Coulson, 1996).
Figure 3.5 Errorspace analysis of Williams seismic system array geometry (a) Array Sept 1999 to Sept 2000 and (b) corresponding Errorspace (m) and (c) Array Sept 2000 to Feb 2005 and corresponding Errorspace (m). (e) online location errors for unfiltered events in the 9390L FW cluster, slice 2, from 1999 to 2005.

Online Source Location Error based on Simplex Algorithm for 9390 L Footwall
Events in Slice 2, From Sept 1999 to Feb 2005

Mean Error = 5.28 m
Std. Dev. = 3.03
3.7 Temporal and Spatial Analysis of Regional Sill Failure

3.7.1 Regional Seismicity and Spatial Changes in Microseismic Intensity

During the monitoring period approximately 33,000 events were recorded and located within the sill pillar region (Figure 3.6a to c). A histogram of the daily event activity (Figure 3.7a and b), indicates that there were approximately 6 main episodes of intense microseismic activity (> 200 events per day), termed flurries of events. Associated with these flurries are 8 macroseismic events (> 0 Mn), recorded by the Geological Survey of Canada's regional seismic system (GSC, 2005), that were thought to have occurred in the sill pillar region (Figure 3.7). These events, as well as the other macro events recorded at Williams, have been documented in Table 3.4. The exact timing of the large events in relation to the flurries of activity could not be determined prior to 2002, as no record of the time of the events could be found, however, it was found from the last two major events in 2003 (2.7 and 3.5 Mn) that these occurred approximately 4 hours after the main flurries of activity started.

Thus, these events can be thought of as regional adjustment or reaction to degradation of the rock mass by the seismic flurries and not the seismic flurries being a result of the large events. The 2.7 Mn event was determined to be a pillar burst in the haulage drift and cross cut (x-cut or x/c) pillar nose between x-cuts for stopes 26 and 25 (Figure 3.8), and was precipitous in the eventual abandonment of the 9390 level haulage between stopes 26 and 23. This flurry of activity was also a direct result of development and mining of the #20 stope between 9450 and 9390 levels (Figure 3.7). The last major event to occur in the sill was a 3.5 Mn event (the largest recorded at Williams), which was thought to be associated with a major foliation shear identified 20 m south of the haulage drift (Bawden and Jones, 2005), but, which caused negligible damage to infrastructure. There was no real forewarning of this large event apart from the start of the microseismic activity, and no mining activity occurred prior to the event. The most damaging event was the first 3.0 Mn event, which caused back failure of the 9415 haulage level from stopes 26 to 18, and moderate damage on 9450 and 9390 levels in March, 1999. This damage was probably the result of dynamic shake down, as once the drift support had been upgraded, less significant damage was recorded for the subsequent large events (Leblanc and Murdock, 2000), however, displacement instrumentation installed in the back of drifts did show local dilation in the immediate back (2-3 m up) of 20-30 mm, at the same time as some of these large events (Bawden et al., 2000; Bawden and Jones 2002) (Table 3.4).
Figure 3.6 Microseismicity recorded in the Williams mine sill pillar region analysed in this case study from September 1999 to February 2005. (a), (b) and (c). Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25]. The sill region analysed here is bounded from stope 28 to stope 16, and analysis slices 1 to 6 are 40 m wide spanning two stopes (Note position of section 9430 E). (d) Contour of event density for 125 m³ voxels with the lower contour limit set to 5 events. Approximately 33,000 events were recorded and located during this time frame. Also, shown is longitudinal grid plane at section 9860 N (b), used for historical seismic density plotting in section 3.7.1.
Figure 3.7 Williams mine sill pillar seismic activity and sill pillar stope blasting. (a) Histogram of daily activity from Sept. 1999 to Feb. 2005, (b) Cumulative events with corresponding large macro events. Also in (a) the time periods when full waveform data was available.
Table 3.4 Large Magnitude Events (> 0 Mn) Recorded at Williams Mine in the Sill, Block 4 Central (B4-C) and other parts of the Mine (Period of Monitoring 1999 to Feb. 2005). Events that occurred in the analysis region of the sill, are in bold type.

<table>
<thead>
<tr>
<th>Date</th>
<th>GSC Mag. (Mn Nuttli)</th>
<th>Region¹ (Slice) with large event rate</th>
<th>In Sill</th>
<th>Comments and Approx. Location</th>
</tr>
</thead>
<tbody>
<tr>
<td>03/29/1999 18:10:00</td>
<td>3.0</td>
<td>S5</td>
<td>Y</td>
<td>(B4-C)² GSC recorded, caused extensive damage 9415L 26x/c to 18x/c and some damage on 9390L and 9450L.</td>
</tr>
<tr>
<td>12/17/1999 18:42:55</td>
<td>2.6</td>
<td>S4</td>
<td>Y</td>
<td>(B4-C) 17 days after 390-16³ stope blasted, caused increase in dilation in back of 9450L FW drive 24 to 19x/c.</td>
</tr>
<tr>
<td>01/09/2000 05:26:47</td>
<td>1.6</td>
<td>S4</td>
<td>Y</td>
<td>(B4-C) No significant damage 9450L to 9370L.</td>
</tr>
<tr>
<td>01/09/2000 05:26:47</td>
<td>1.7</td>
<td>S4</td>
<td>Y</td>
<td>(B4-C) No significant damage 9450L to 9370L.</td>
</tr>
<tr>
<td>07/28/2000 14:11:09</td>
<td>2.8</td>
<td>S1</td>
<td>Y</td>
<td>(B4-C) Around Stope 415-21, SMART showed jump in dilation of 1 - 5 mm at 18x/c and 17x/c located 1 to 2 m in the back on 9415L.</td>
</tr>
<tr>
<td>04/22/2001 09:25:02</td>
<td>2.3</td>
<td>S3</td>
<td>Y</td>
<td>(B4-C) #3 OP (9390L, 390-21) Ore pass pulled, minor damage to 9390L west of 23x/c, SMART showed minor increase 2 – 3 mm at a depth of 1 to 2 m in 390-24 x/c.</td>
</tr>
<tr>
<td>06/28/2001 20:26:26</td>
<td>3.1</td>
<td>S1, S2, S3</td>
<td>Y</td>
<td>(B4-C) 390-21 to 26, 40m south of FW minor damage and SMART showed dilation of 20-30 mm, 2-3 m in the back 390-24 x/c.</td>
</tr>
<tr>
<td>04/03/2002 07:24:37</td>
<td>0.5⁴</td>
<td>Y</td>
<td></td>
<td>(B4-C) uMag (1.4) 370-23 stope, no significant damage.</td>
</tr>
<tr>
<td>04/08/2002 09:34:28</td>
<td>0.5⁴</td>
<td>Y</td>
<td></td>
<td>(B4-C) uMag (3.0) #3 OP, minor spall 9370L FW drive South wall across from 370-23 stope.</td>
</tr>
<tr>
<td>05/18/2002 10:00:00</td>
<td>0.5⁴</td>
<td>N</td>
<td></td>
<td>(B4-W) Seismic Flurry back failure in 415-38.</td>
</tr>
<tr>
<td>06/21/2002 11:47:00</td>
<td>0.5⁴</td>
<td>N</td>
<td></td>
<td>(B5-C) uMag (1.32) E-conveyon 9135 L.</td>
</tr>
<tr>
<td>10/30/2002 10:00:00</td>
<td>0.5⁴</td>
<td>N</td>
<td></td>
<td>(B4-W) Seismic Flurry 9475 L &amp; 9450 L, E. of 38x/c.</td>
</tr>
<tr>
<td>03/16/2003 03:12:40</td>
<td>2.6</td>
<td>N</td>
<td></td>
<td>(B5-W) 215-35 (time GSC N.B. seismic system down).</td>
</tr>
<tr>
<td>03/16/2003 03:13:48</td>
<td>1.9</td>
<td>N</td>
<td></td>
<td>(B5-W) 215-35 (time GSC N.B. seismic system down).</td>
</tr>
<tr>
<td>03/16/2003 05:37:58</td>
<td>2.1</td>
<td>N</td>
<td></td>
<td>(B5-W) 215-35 (time GSC N.B. seismic system down).</td>
</tr>
<tr>
<td>05/30/2003 08:26:55</td>
<td>2.7</td>
<td>S1, S2, S3</td>
<td>Y</td>
<td>(B4-C) 390-26/25 pillar burst, abandon drift 26 to 23 Occurred after flurry of activity started at 04:26:54, SMART showed no major jump in dilation.</td>
</tr>
<tr>
<td>09/13/2003 18:25:56</td>
<td>3.5</td>
<td>S1, S2, S3</td>
<td>Y</td>
<td>(B4-C) 450-24x/c Macro event, minor damage to 26/27x/c pillar, 27x/c and 21x/c on 9390L (drill on 9450 jumped 1 ft) Occurred after flurry of activity started at 14:26:04, SMART showed no major jump in dilation.</td>
</tr>
<tr>
<td>10/16/2003 13:08:19</td>
<td>2.6</td>
<td>N</td>
<td></td>
<td>(B5-W) 190-38 stope rock burst in stope followed by 25,000 t FOG of Back.</td>
</tr>
<tr>
<td>01/13/2005 11:12:21</td>
<td>2.5</td>
<td>N</td>
<td></td>
<td>(B5-C) 9147L Dyke Burst.</td>
</tr>
</tbody>
</table>

Notes:
¹ S1, S2, S3, S4, S5, S6 = refer to the six 40 m wide slice taken through the sill for seismic analysis (See Figure 3.6a)
² (B4-C) = block 4 central, (B4-W) = block 4 west, and (B5-W) = block 5 west (refer to Figure 3.1).
³ Stope naming convention refers to the undercut of the stope, such that 390-16 is stope 16 mined between 9415 and 9390 levels.
⁴ Magnitude estimated based on mine wide reported event/s but not recorded by Geological Survey of Canada (GSC).
Figure 3.8 View looking west of the 26/25 x-cut pillar burst on the 9390 level. This is thought to be the direct cause of the 2.7 Mn (Nyttli) event, and also caused moderate damage to the #25 x-cut and minor damage to the #26 x-cut.
At present without any other information, it is assumed that these large events, which are most likely pure-shear events\(^2\) occurred on a large structure, which is required for production of large energies, and could be associated to the footwall foliation shear previously discussed. These events are believed to be a by-product of stress redistribution, caused by degradation and probable strain-softening behaviour associated with the intense microseismic activity. It is commonly understood that in mine induced seismicity there are two types of events, the smaller events associated directly to specific mining zones in relation to stress, and the larger events often distant and associated to geological structure triggered apparently randomly and associated more with the overall mine extraction (Gibowicz, 1989).

Another, possible explanation for the large events could be that the rupture area required, was developed through a network of generated fractures under confinement in the clusters of seismicity. However, it would be expected that these large shear-slip events would have caused significant displacement in these regions; this is not supported by the conventional instrumentation placed above the back of the development and close to the core of the seismic clusters, nor is it supported by observations of damage in the haulages (Bawden and Jones, 2002). However, the possibility still exists.

Analysis of the distribution of microseismicity recorded in the sill on a yearly basis, indicates that there was a shift in intensity from east to west following the intense period of activity in July 2000, the mining front following behind this migration (Bawden and Jones, 2005). The seismic density was analysed for each year to show the migration in the intensity of the microseismic activity (Figure 3.9). The event density has been contoured on a section (9860N, Figure 3.6b) running through the most intense activity in the footwall. The actual plots of seismicity can be found in Appendix B1 (Figure B1.1 to B1.6). As has previously been noted, the majority of the events locate in the footwall region between the footwall haulages (Figure 3.6d and Appendix B1, Figures B1.1 to B1.6). At the 9370 level, the seismicity is associated in the footwall, between the 9370 and 9390 levels, but clustering also occurs in the immediate footwall of the ore zone, at the leading edge of the stope backs (Figure 3.6d and Appendix B1). As can be seen in Figure 13, the main cluster of seismicity that is constant, throughout all time frames after July 2000, is the footwall cluster between 9390 and 9415 levels (Figure 3.6d, Figure 3.9), and especially between stopes 24 to 26.

\(^2\) Es/Ep ratio for the 3.5 Mn event, (the only large event with full wave forms in the database) was determined to be around 113 from calculation based on uniaxial sensors, with a Madariaga source radius of 48 m. Based on a single functioning triaxial sensor, the Es/Ep ratio of 23 was calculated with a source radius of 54 m. Pure shear is recognised if the Es/Ep ratios are > 20.
Figure 3.9 Yearly seismic event densities (5 x 5 x 5 m voxel), contoured on section 9860N cutting through the centre of the footwall main cluster on the 9390 L (see Figure 3.6b). (a) Events from Sept 1999 to March 2000 (6,651 events), (b) April 2000 to Dec 2000 (6,662 events), (c) Jan 2001 to Dec 2001 (5,301 events), (d) Jan 2002 to Dec 2002 (3,876 events), (e) Jan 2003 to Dec 2003 (8,721 events), (f) Jan 2004 to Dec 2004 (1,830 events). Also, note sill region analysed here spans from stope 27 to 16.
This is the core region of the sill that sees the stresses gradually increase, and for which it is felt the most complete data set exists following rock mass failure through the various stages until ‘aseismic’ behaviour occurs in late 2004, early 2005.

Above the 9415 level the seismicity is most intense in late 1999, early 2000, again with the main clusters being located in the footwall between the 9415 and 9450 levels, and is mainly associated with the drift back failure post March 1999, when the first large 3.0 Mn event occurred. The activity above the drift predominantly subsides, until late 2003, when some seismicity in the far west end around stopes 25 and 26 occurs, which is thought to result from a combination of increased induced stress through mining, and stress redistribution through failure of the footwall region between the 9390 and 9415 levels. The fact that the seismic activity is predominantly located in the footwall of the orebody, is different from other mining operations, primarily base metal mines such as the Brunswick Mine (Coulson, et. al., 2002, Diederichs et. al., 2002, Simser et. al., 2002) or Lac Short Mine (Falmagne, 2002) where the microseismicity tends to occur in the stronger brittle massive sulphides, rather than the weaker and softer country rock. However, given the fact that there is very little difference in the rock mass properties between the different domains at the Williams mine the seismic activity is occurring in the highest stress regions around the footwall development as confirmed by the three dimensional linear elastic stress modelling to be discussed later. It is important to note that the ore development was kept to a minimum amount during this period with only some cross cut development as in Figure 3.3.

Analysis over time from 1999 to 2005 of the iso-surface of the clustering density, (minimum of 5 events per 125 m³ voxel), based on the cumulative event density, shows the development of the yield front (Figure 3.10). [Note: it is expected that the rock mass is yielding or close to the peak strength inside this volume; this does not mean the rock mass has completely failed, but rather is continuing to undergo degradation]. This yielding region of the rock mass encompasses the footwall development by December 2001, and basically stabilizes, with an increase in intensity or rock mass degradation with time. A transverse section through the most intense region around stope 25 (Figure 3.11 and Figure 3.6d), indicates that some voxels at the centre of the cluster, had in excess of 45 events occur within the 5 x 5 x 5 m (125 m³) volume by December 2003, and it would be expected the rock mass has undergone significant degradation. At the end of the monitoring period, in February 2005, the majority of the sill pillar had become predominantly ‘aseismic’.
Figure 3.10  Expansion of yield surface in the Williams mine sill pillar region, based on the clustering density of 5 events per 125 m$^3$ voxel. Plots are based on cumulative seismicity from the beginning of monitoring in September, 1999 to (a) June 2000, (b) Dec 2000, (c) Dec 2001, (d) Dec 2002, (e) Dec 2003, and (f) Feb 2005.
Evolution of Event Density Over Time For a Line of Voxels on Transverse Section 9432.5E at Elevation 9402.5 Just above the back of 9390 Level Haulage

Figure 3.11 Plot of voxel event count above the 9390 L haulage drift, for transverse section 9432.5E, show increase in seismic intensity from Sept 1999 to Feb 2005. Inset shows the pixelation plot of the event density for transverse section 9432.5E and the location of the voxels plotted in the graph. Also shown as the white line in the inset is the general location of the conventional instrumentation SMART cables, used to monitor displacements and cable loads.

It can be seen from Figure 3.10f that, although the yield front encompasses most of the footwall of the sill there are notable regions where the recorded seismic density does not correspond to yield (i.e. did not reach the critical density of 5 events per voxel), such as the majority of the east end (stopes 19 to 17 on the 9390 level), the far west end (stope 27), and the region below the 9450 level at the top of the sill. This is probably as result of beginning monitoring after significant seismicity had already occurred, and identifies one of the major problems with using this cumulative approach to determine which regions have yielded. If a full history does not exist then some assumptions have to be made about the state of the rock mass. Additionally, this approach is based on recorded microseismicity, for a given array and frequency bandwidth of monitoring. Smaller events that may be occurring close to openings could be of a higher frequency, and lower energy (Reyes-Montes, 2004; Collins et al., 2002), outside the range of this system. There is also, the potential for fractures to propagate in a stable ‘aseismic’ nature (Pettitt, 1998). However, that being said, the most likely explanation here for the potential yield volume not encompassing the entire sill, is the lack of historical seismic data. Assuming that and based on the migration in seismicity from east to west, the best location for a more
complete data set exists in the west portion of the sill pillar around stope 26 to 24 and above the 9390 level.

3.7.2 Cluster Identification

The sill region was broken down into six 40 m wide (two stope widths) slices (Figure 3.6a) and events were visually analysed along with the event densities on a yearly basis, to identify clusters. The yearly events for slice 2 can be seen in Figure 3.12 to Figure 3.16, and all the events per slice and year are plotted in Appendix B1, Figures B1.7 to B1.36. The seismicity shows very strong directional clustering above and below the footwall haulage drifts, the clustering around the haulages being maintained from slice 1 (stope 26) to slice 4 (stope 19 and 18), where the strong clustering breaks down. It is possible to see that there is clustering of microseismicity around the No. 3 ore pass, visible in Figure 16b and more strongly Figure 3.14b. It is not known whether this was failure of the ore pass or ore pass noise, however, all the events associated with the ore pass were subsequently filtered out during further analysis.

Subsets of clusters related to each slice, haulage and stope, were extracted to be analysed separately using the PCA technique to quantify the direction of these trends and to see if there was a temporal change during the failure process. The clusters were extracted on a yearly basis, using the clustering event density as a guide to the limit of the clusters extent, although events outside of the 5 event clustering density iso-surface were included if they were visually linked to the same cluster. Other methods for defining clusters have been performed by using a Ward variance method, (Kaiser et. al., 2005), or an error ellipsoid uncertainty collapsing method, (Jones and Stuart, 1997) for interconnecting structures. However, for the most part this was not felt warranted in this study as the cluster appeared visually distinct. The rough outlining of these clusters for slice 2 can be seen in Figure 3.12b to Figure 3.16b. Exception to the hard slice dividing volume was made for clusters that visually overlapped the individual volumes. This was only done for events in slice 4. In all some 95 clusters where identified (approximately 20 per year) and analysed using the PCA technique. This case study presented in this Chapter, will however concentrate on the events in the footwall between the 9390 and 9415 levels, as this provides a more complete picture of the eventual footwall failure. Additionally, based on the relatively consistently low location errors for events in the footwall zone (Figure 3.5e), the PCA technique was run on all of the events in the determined clusters regardless of location error. The technique is a statistical method, and Coulson (1996) found that the technique was robust enough to determine the same trend on unfiltered events versus those that had been filtered for the best locations with the smallest location error.
Figure 3.12 Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 2 Events (2633). Note mining geometry is fixed at Dec 1999. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25].

Figure 3.13 Williams Mine Microseismicity Jan 2001 - Dec 2001 – Slice 2 Events (2038). Note mining geometry is fixed at Dec 1999.
Figure 3.14  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 2 Events (1775). Note mining geometry is fixed at Dec 1999.

Figure 3.15  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 2 Events (3719). Note mining geometry is fixed at Dec 1999.
3.7.3 Yearly Variation in PCA Trends Across the Sill

Analysis of the yearly trends per slice can be seen in Figure 3.17 to Figure 3.21, for slices 1 to 6 and overall observations in Table 5. The complete yearly analysis with yearly event location for clusters between the 9390 and the 9415 levels for each slice can be seen in Appendix B2 (Figures B2.1 to B2.23).

It is possible to observe for slices 1 to 4 that there is a clearly defined trend (shown with a bold window, in Figure 3.17 to Figure 3.21, and defined in bold in Table 3.5) in the determined PCA planes. This dominant trend is relatively stable for the clusters in slice 2 and 3 from 1999 to 2003 (Figure 3.18 and Figure 3.19). The average orientation of this trend in terms of strike and dip is around [090, 40], and there appears to be a tendency to steepen to around [090, 54] in 2002 and then drop back to [090,40] in 2003 (Figure 3.18 and Figure 3.19; Table 3.5).
Figure 3.17 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 1. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 and (f) distribution of clustering events in slice 1 from Sept 1999 to Dec 2004. Note predominant pervasive trend is outlined in bold.
Figure 3.18 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 2. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 and (f) distribution of clustering events in slice 2 from Sept 1999 to Dec 2004. Note predominant pervasive trend is outlined in bold.
Figure 3.19 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 3. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 and (f) distribution of clustering events in slice 3 from Sept 1999 to Dec 2004. Note predominant pervasive trend is outlined in bold.
Figure 3.20  Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 4. (a) Sept 1999 to Dec 2000, (b) 2001, (c) 2002, (d) 2003, (e) 2004 (no events) and (f) distribution of clustering events in slice 4 from Sept 1999 to Dec 2003. Note predominant pervasive trend is outlined in bold.
Figure 3.21 Stereonets of principal components analysis (PCA) derived planes for yearly analysis of clustering events above the 9390 level footwall haulage drive in slice 5 and 6. (a) Sept 1999 to Dec 2000 slice 5, (b) 2001 slice 5, (c) distribution of clustering events in slice 5 from Sept 1999 to Dec 2001. (d) Sept 1999 to Dec 2000 slice 5, (e) 2001 (no events) and (f) distribution of clustering events in slice 6 from Sept 1999 to Dec 2003.
Table 3.5  Summary of PCA major planes for Williams sill based on a yearly analysis of clustering events for footwall clusters above 9390L.

<table>
<thead>
<tr>
<th>Slice</th>
<th>Year 2000</th>
<th>Year 2001</th>
<th>Year 2002</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td># Events</td>
<td>D Planes</td>
<td>State</td>
</tr>
<tr>
<td>1</td>
<td>290</td>
<td>17</td>
<td>Localization Peak to Post Peak</td>
</tr>
<tr>
<td>2</td>
<td>395</td>
<td>22</td>
<td>Interaction to Localization Peak</td>
</tr>
<tr>
<td>3</td>
<td>438</td>
<td>27</td>
<td>Localization Peak</td>
</tr>
<tr>
<td>4</td>
<td>630</td>
<td>28</td>
<td>Post-Peak</td>
</tr>
<tr>
<td>5</td>
<td>319</td>
<td>25</td>
<td>Post-Peak to Residual</td>
</tr>
<tr>
<td>6</td>
<td>22</td>
<td>18</td>
<td>Aseismic Residual Failed</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Slice</th>
<th>Year 2003</th>
<th>Year 2004</th>
<th>Total* Events</th>
<th>Avg D</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td># Events</td>
<td>D Planes</td>
<td>State</td>
<td># Events</td>
</tr>
<tr>
<td>1</td>
<td>219</td>
<td>17</td>
<td>Post Peak to Residual</td>
<td>71</td>
</tr>
<tr>
<td>2</td>
<td>1136</td>
<td>20</td>
<td>Disassociation Post-Peak to Residual</td>
<td>204</td>
</tr>
<tr>
<td>3</td>
<td>882</td>
<td>24</td>
<td>Disassociation Post-Peak to Residual</td>
<td>104</td>
</tr>
<tr>
<td>4</td>
<td>166</td>
<td>20</td>
<td>Post-Peak to Residual</td>
<td>0</td>
</tr>
<tr>
<td>5</td>
<td>---</td>
<td>---</td>
<td>Shadowed</td>
<td>417</td>
</tr>
<tr>
<td>6</td>
<td>---</td>
<td>---</td>
<td>---</td>
<td>22</td>
</tr>
</tbody>
</table>

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The number of events and total events in this table is the number of events that had ellipticity > 2.5. Approximately 10% of the events when analysed had ellipticities < 2.5 depending on the circumstance.
During 2003 and into 2004 there is a breakdown in the stability of the trend, with other minor trends becoming apparent, in these clusters. This change in orientation from the dominant trend has been termed disassociation, in that the relationship of the events to one another have disassociated from the predominant stable trend. This temporal change in the behaviour of the PCA trends coincides with the point at which the core region of the footwall of the sill started to become aseismic in patches, and was considered to be ‘failed’ or ‘fully yielded’ (Bawden and Jones, 2005). The other significant occurrence is that in 2003 there were two large seismic events located in the sill region, the 2.7 Mn pillar burst at stopes 26 to 25 in slice 2, and the 3.5 Mn located in the approximate region of the footwall in slices 2 and 3. After this point of disassociation, in late 2003 and into 2004 mining commenced at an accelerated pace with mining from stopes 20 (390-20, 415-20), stope 22 (390-20, 415-20), stope 19 (370-19, 390-19, 415-19), stope 24 (370-24, 390-24, 415-24) and stope 21 (370-21, 390-21), without incidence.

As previously stated mining of these stopes was relatively easy with no stress affects reported and no loss of raise bores or drill holes, although once the stopes were opened up, gradual sloughing from the end walls would still occur suggesting that stress or deformation were reduced below the fracture initiation stress but not totally alleviated or that the stress fracturing followed by relaxation allowed for unravelling (Jones pers. comm., 2003; Bawden and Jones, 2005). Prior to this the raise bores that were generally placed well in advance would suffer excessive breakout (‘dog earing’) and sloughing, generally with loss of drill holes. Additionally, any mining activity would often result in a flurry of seismic activity in the sill. The change in conditions suggests that the ore zone had become stress shadowed by failure of the footwall region. This drop in stress level can only occur if strain-softening (strength-weakening) of the failed rock mass occurs.

In slices 1 and 4 on either side of the most intense seismically active region, the dominant PCA trends of [090,30] and [090,50] respectively (Figure 3.17 and Figure 3.21; Table 3.5), breakdown much earlier than in the central core, and disassociation appears to occur in 2002 for slice 1 and 2001 for slice 4. It is important to point out that in both slice 1 and 4 the regions are not fully populated by microseismic events and it appears that portions of these regions have already become aseismic. This is especially understandable in slice 1 which is on the western abutment of this sill region where, probably during the caving failure of the 28 stope in

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3 Stope naming terminology: 390-20 refers to the undercut horizon (9390L) of the 20 stopes, 415-24 refers to the undercut horizon (9415L) of the 24 stopes.
February 1997 on structurally controlled lamprophyre dykes, (Leblanc and Murdock, 2000), additional seismicity occurred, but was not recorded as no seismic system existed. These incomplete data sets make these two regions a little more difficult to interpret.

At the far eastern end of this sill region in slices 5 and 6 (Figure 3.21), there are no stable trends as identified in the other slices, and it is presumed that this region was predominantly well into post peak and close to being completely failed by the time the microseismic system was installed. This resulted in aseismic behaviour occurring after 2001 in slice 5, and in 2000 in slice 6, which was later shadowed by mining in 2002.

Although there is no guarantee of a complete seismic history in any region of this sill, the most likely regions that have more complete records from interaction, to localization (yield), to the peak and into the post peak, are clusters in slices 2 and 3. These clusters represent the most complete data set and have the greatest number of events per volume recorded (Table 3.5). These, along with slices 1 and 4, will be analysed in greater detail in the next section.

Additionally, it can be noted that the characteristic spatial window dimension D tends to range between 22 to 24 m for most of the clusters in slices 2 to 4 (Table 3.5). Considering that the volumes analysed are equivalent at 40 m along strike, by 35 m high and 35 m deep, this dimension closely represents the distance between the sub level haulage drives of the 9390 and 9415 levels (23 m, inclined distance floor to floor) and is the geometrically controlling factor in the minimum dimension of the clusters (i.e. the events are contained between the drifts).

Based on this coarse yearly analysis of the seismic data in the sill pillar region, and assuming that disassociation represents a change in the inter event relationship or inter fracture communication, resulting from regions of the rock mass becoming aseismic (failed), the general trend is for the footwall of the sill to fail from east to west. Slices 5 and 6 had probably already failed in 2000, and with slice 4 failing in 2001, and with part of slice 1 failing in 2002, then followed by slices 3 and 2 sometime in 2003.

The next section will take a more detailed temporal analysis of clusters above the 9390 level in slices 1, 2, 3 and 4, combining analysis of the displacement instrumentation installed on the 9390 level, with numerical modelling, to understand the stress history, and possible post peak behaviour.
3.7.4 Detailed PCA Variations Along the 9390 Level Footwall Development

In order to determine the exact time of the temporal change in the dominant PCA trends, each cluster per analysis slice was broken down into the 50 event temporal windows and analysed individually on lower hemisphere stereographic projections using Dips©. Changes in the trends, number of events with ellipticity greater than 2.5 and average ellipticity, relative to any physical influence, either from mining activity, large seismic events or from displacements (dilation of the rock mass), recorded in the SMART instruments installed in the back of the 9390 level haulage drive were noted. The main focus of this detailed analysis is presented for the cluster of events in slice 2 above the back and to the south of the 9390 level footwall haulage, which was also investigated through the PCA sensitivity analysis (Appendix A2). The analysis is summarised for the other regions.

3.7.4.1 General Observations of Temporal PCA analysis for Slice 2

The PCA derived planes for the individual 50 event temporal windows are plotted in Figure 3.22 a to d, for the events in slice 2, above the 9390 level haulage. It should be pointed out that the analysis is a breakdown of the yearly analysis, and that as such the total number of events for a year of data is not always divisible by 50. This sometimes results in a temporal window, always at the start of the year, being run sometimes on less than 50 events. If the number of events in the first temporal window of the year is less than 25 then the window is discarded, or if just under taken with less significance (as in Figure 3.22a, temporal window 8). This is based on the analysis of varying the temporal window size discussed in Chapter 2 (Section 2.2.2.1). Combining all of the planes from the individual temporal windows for each year, results in the distribution of poles shown in the previous section (Figure 3.18). The date displayed on the stereonets in Figure 3.22a to d represents the date of the last event in that 50 event temporal window. The location of the events per temporal window, and orientation of determined PCA planes for the events in this cluster can be seen in Appendix B2 (Figure B2.24 to B2.40).

As can be seen from the distribution of poles of the PCA derived planes (Figure 3.22a), the trend stays predominantly stable with minor fluctuations, the predominant averaged trend of the poles for each temporal window conforming to a plane striking approximately East (060° -110°) and dipping to the south between 40° to 60° (average strike [Right-Hand-Rule] and dip [090°, 45°]). This trend is maintained, except for a sudden drop in the number of PCA derived planes with ellipticity > 2.5 on four occasions (Figure 3.24b) which will be discussed later, until a significant change in the trend at temporal window 48 (07-07-2003) to a secondary average trend striking West and dipping to the North 30° (average strike and dip [298°, 31°]).
Figure 3.22a Stereonets of principal components analysis derived planes of individual temporal windows (50 events, except for the starting window of each year), for clustering events above the 9390 level, in slice 2. Temporal window number, date of last event in window and major plane indicated.
Figure 3.22b Stereonets of PCA derived planes of individual temporal windows (50 events, except for the starting window of each year), for clustering events above the 9390 level, in slice 2. Temporal window number, date of last event in window and major plane indicated. Dashed border represents low number of events with ellipticity > 2.5.
Figure 3.22c Stereonets of PCA derived planes of individual temporal windows (50 events), for clustering events above the 9390 level, in slice 2. Temporal window number, date of last event in window and major plane indicated. Dark border indicates point of disassociation and major change in failure direction.
Figure 3.22d Stereonets of PCA derived planes of individual temporal windows (50 events), for clustering events above the 9390 level, in slice 2. Temporal window number, date of last event in window and major plane indicated. Dark border indicates point of disassociation and major change in failure direction.
After this, although the trend does revert back to the previous predominant trend on occasion, it becomes unstable with a rotation in strike to 120° to 130° (temporal windows 51 to 54, 08-28-2003) which is related to a flurry of activity preceding the large magnitude 3.5 Mn event (09-13-2003) after which there is a sustained change in the trend to strike 280° and dipping at around 60° to the North [280°, 60°]. These significant changes in the orientation have been indicated with bold borders for the stereonets in question in Figure 3.22c and d. The point at which there is an identifiable change in the predominant trend, with instability of this for the following temporal windows, has been called the **point of disassociation**, when the trend is no longer associated with the main failure mechanism. The change in trend could possibly be interpreted as activation of the conjugate structure, allowing for increased degrees of freedom of block movement. It is postulated that at this point the rock mass has become highly fractured, and has started to dilate, thus causing patches of the rock mass to become aseismic. This is not to say the fracturing process in these regions has completely ceased, but the size and energy radiated by much smaller fractures is not enough to trigger the system. At this stage, or close to this, the rock mass has undergone significant strain softening and that the rock mass is approaching its residual strength, as shortly after this in all of the clusters analysed the entire region of the rock mass becomes entirely aseismic.

The mean stereonet plane orientation based on the Fisher distribution of poles for maximum pole concentration in each temporal window was plotted versus the time of the last event in the temporal window and plotted against the temporal window number to summarise the changes in orientation on one plot, including the average ellipticity\(^4\) for a given temporal window (Figure 3.23 a and b). Early on in the monitoring period in July 2000 there is a marked increase in the average ellipticity from 10 to > 15, indicating a relatively planar or strongly defined trend. This coincides with development of the onset of the clustering density (Figure 3.10b), indicating the potential for coalescence of events or potential fractures, and represents localization of damage and strains to a potential macrofracture structure orientated [090°, 45°] in terms of strike and dip. This localization and coalescence occurs around the same time as the 2.8 Mn event (Table 3.4, Figure 3.23), and a minor increase in the event rate for this cluster was noted (Figure 3.24a). This occurred following the mining of the distant 390-15 stope, which may have been the trigger to cause an excitation of seismicity in the sill (Figure 3.7a) and resulted in a shift in the concentration of seismicity from east to west (Figure 3.9b).

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\(^4\) Average Ellipticity is determined for a given set of events within the temporal window, but averaged only on events that had F-parameter > 0.5 (50% events used to determine plane and ellipticity) to remove outliers with high ellipticity based on only a few of the events captured within the spatial window, these generally found at the periphery of the cluster of events.
Figure 3.23 Temporal window variation in PCA derived planes of mean strike (red stars), dip (purple squares) plotted against the left axis and ellipticity (black triangles) plotted against the right axis, for clustering events above the 9390 L FW haulage, Slice 2, versus time in (a) and versus temporal window number in (b). Large magnitude events are also indicated in both plots. Numbered points in (a) indicate time frames of minor temporal plane orientation changes identified from the CMPCA discussed in Section 3.7.4.3.4 for comparison.
Figure 3.24  (a) Event rate (moving average number of events every 7 days), for clustering events in slice 2, above the 9390 L FW haulage, (b) Plot of percentage of events that had PCA planes with ellipticity >2.5, per 50 event temporal window for the same cluster, and also indicated large magnitude events and mining of stopes, (the first diamond approximates the date of the void blast and last diamond the date of the mass blast for each stope). Numbered points relate to a significant drop in the number of planes with ellipticity < 2.5, and the corresponding PCA temporal window number (e.g. Win#8) is indicated for reference to Figure 3.22 a to d.
The significance of this suggests that the sill region was in a critical state and relatively sensitive to minor changes in stress and, combined with the concept of coalescence and localization through the increase in ellipticity and onset of the clustering density, would suggest that this region of the sill in slice 2, is close to or at the peak strength and has started to yield. The ellipticity drops to 10 in early 2001, and then again drops to a relatively consistent level between 5 and 7 for the rest of the time until the region becomes entirely aseismic in late 2004.

It is important to note that the seismic system was expanded on Sept 22, 2000 to include additional sensors of the mine wide array. The drop in average ellipticity based on these 50 event temporal windows, appears to occur at the same time, although the PCA trend is maintained with absolutely no change. The major change in the system other than increased location accuracy outside of this sub array, was that the trigger sensitivity was increased from a threshold of 6 to 8 sensors, which could impact the number of smaller events recorded, potentially affecting the ellipticity. In slice 3, to be discussed later, the high value of ellipticity is maintained for one temporal window after the system changed (i.e. only events located with the expanded system). Additionally, a continuous shifting PCA analysis was run on the data [Section 3.7.4.3.4], and it was found that the ellipticity had actually started to drop in this region prior to system expansion. This suggests that this is probably a coincidence, but cannot be ruled out as a reason for the reduced ellipticity to a lower background level. In a similar study at Golden Giant mine (Chapter 4) the increased ellipticity was maintained up until the point of disassociation, however, the failure at Golden Giant, from coalescence to disassociation happened over a few months instead of years.

As was noted previously, the PCA trend determined from the maximum Fisher concentration on the stereonets remains relatively stable from September 1999 to June 2003, with some steepening in early 2002, until the trend changes significantly and flips over to the lower quadrant on the stereonet at the point of disassociation (Figure 3.23a or b, Figure 3.22c [win=48]) and Appendix B2). The most significant observation at this time was the sudden dilation of 16 mm in SMART cable (26-2) at a depth of 9 m in the back of the 9390 level just west of the cluster of events and just prior to the temporal change in the trend. This deep dilation or shearing would be expected to have caused significant strain softening in the region of the instrument, possibly resulting in aseismic behaviour affecting the trend. After this point the trend is unstable with a rotation in strike to between 120° to 150° (more easily seen in Figure 3.23b, related to temporal windows 51 to 54 Figure 3.22c), until the 3.5 Mn event significantly alters the trend to around [280°,60°] for the next 5 temporal windows (Figure 3.22d,
Figure 3.23a) after becoming wholly aseismic above the back of the 9390 level haulage (Appendix B2, Figure B38c).

Observation of the conventional instrumentation in the form of SMART cables installed at the cross cut intersections with the footwall haulage drive gives a strong indication that these temporal changes in the trend of the seismicity are influenced through dilations within the rock mass causing regions or patches of the rock mass to become aseismic and are discussed in the following sections.

### 3.7.4.2 Analysis of Conventional SMART Cable Instrumentation

As part of the rehabilitation program proposed following the Major March 29, 1999, 3.0 Mn event, SMART cables and SMART MPBX (DaGraaf et. al., 1999; Bawden et. al., 2000), were installed with the upgraded cable design for the 9450, 9390 and parts of the 9370 and 9415 levels. The location of the instruments on the 9390 level are displayed on plan and long section in Figure 3.25, and can be seen, marked on the view West of Figure 3.17f to Figure 3.21f, and in Appendix B2 (Figure B2.1 to B2.23). These instruments were not all installed at the same time and some cover varying time periods, however, most were not read after July 2003 or when the instruments had shown large deformations between anchors of over 50 mm, close to the limit of the potentiometers (60 mm). The instruments were generally installed with another regular bulge or modified geometry cable, and in varying orientations, generally with the head at the toe of the hole, so that they could be plated at the collar, and form an active part of the support. The SMART cable geometry was specifically design to match the cables in the ring, and monitor the performance of the cables and deformation of the rock mass. Here cables in the rings were plain cables without bulges for the first 5 m (predecessors to fully debonded cables), however the nodes of the SMART cables were still cemented in the plain cable geometry section, generally on a 1 m spacing for the first 5 m and 2 m spacing with bulges beyond this. Each instrument can monitor the displacement of up to 6 nodes, recording deformations using a linear potentiometer in the head, with potentiometers having a travel of 2.5 inches (63.5 mm) (DaGraaf et. al., 1999). For standard double Garford cables, 50 mm of deformation equates to a strain of 5% for 1 m bulge spacing and would take the SMART cables over the ultimate capacity, likewise for double plain cables of the SMARTS, 50 mm between the 1 m nodes would mean failure.
Figure 3.25 (a) Plan view of 9390 L sill pillar region, showing boundaries of seismic analysis slices and location of conventional instrumentation (SMART cables). (b) Longitudinal view of conventional instrumentation and contours of depth of displacement that exceed 1 mm dilation, for three points in time, 12-2000, 12-2002 and 07-2003. Note instruments 17, 18, 19 and 20 showed negligible movement during the monitoring period. Also shown on (b) depth of caving on 9415L as of 12-1999 following 2.6 Mn event (Yi, 1999).
Figure 3.26  Instrument 26-1 (a & c) and 26-2 (b & d) SMART cable response for nodal displacement in mm relative the toe of the instrument versus time (top), and plot of displacement in mm versus depth along the cable, with datum at the back of the excavation. Note in figure (f) the deep dilation, 9 m up from the back at the end of the monitoring period, while bulking is occurring over the first 3.5 m.
Table 3.6 Summary of SMART Cable and SMART MPBX displacements over the monitoring period for the 9390L haulage between stopes 16 and 27.

<table>
<thead>
<tr>
<th>Location</th>
<th>Slice</th>
<th>Serial # (Length)</th>
<th>Depth of Displacement (Dilation, dil.) &gt; 1 mm</th>
<th>Max Depth of Large dil. (m)</th>
<th>Value of dil. (mm)</th>
<th>Date Occurred (mm-dd-yyyy)</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>27</td>
<td>1</td>
<td>S0699-12 (8 m)</td>
<td>6 7.5 7.5 7.5 7.75</td>
<td>7.75</td>
<td>2.5</td>
<td>05-12-2003</td>
<td>Small but gradual increase over length of instrument. Installed Partially failed ground.</td>
</tr>
<tr>
<td>26_1</td>
<td>1</td>
<td>S0699-07 (8 m)</td>
<td>3.5 4.5 6 --- ---</td>
<td>4.5</td>
<td>55</td>
<td>12-10-2002</td>
<td>Jump in dil. 55 mm at a depth of 4.5 m in Dec, ’02. Prior to drop in # of PCA events slice 2.</td>
</tr>
<tr>
<td>26_2</td>
<td>1/2</td>
<td>S0201-08 (10 m)</td>
<td>--- 1.5 3 3.5 9</td>
<td>9</td>
<td>16</td>
<td>06-20-2003</td>
<td>Jump in dil. 16 mm at depth, prior to <strong>Disassociation in slice 2</strong>.</td>
</tr>
<tr>
<td>25</td>
<td>2</td>
<td>S0699-17 (8 m)</td>
<td>--- --- 6 6.5 7.5</td>
<td>6</td>
<td>12</td>
<td>05-12-2003 (last reading)</td>
<td>Jump in dil. 38 mm at depth, same as drop in # of PCA planes slice 2 and prior to <strong>Disassociation slice 2</strong>.</td>
</tr>
<tr>
<td>24_1</td>
<td>2</td>
<td>S0699-04 (8 m)</td>
<td>2.5 4.5 6 7.5 ---</td>
<td>7.5</td>
<td>38</td>
<td>01-08-2003</td>
<td>Jump in dil. 51 mm at depth, same as drop in # of PCA planes slice 2 and prior to <strong>Disassociation slice 3</strong>.</td>
</tr>
<tr>
<td>24_2</td>
<td>2/3</td>
<td>S0901-02 (10 m)</td>
<td>--- 3.5 3.75 9 ---</td>
<td>7</td>
<td>9</td>
<td>01-14-2003</td>
<td>Jump in dilation at depth, same as drop in # of PCA planes slice 2 and prior to <strong>Disassociation slice 3</strong>.</td>
</tr>
<tr>
<td>23</td>
<td>3</td>
<td>S0699-10 (8 m)</td>
<td>2.5 3.75 4.5 4.6 4.75</td>
<td>4.75</td>
<td>8</td>
<td>07-08-2003 (last reading)</td>
<td>Gradual increase of moderate dil. to a depth of 4 – 5 m. Readings stopped early.</td>
</tr>
<tr>
<td>22</td>
<td>3</td>
<td>S0699-05 (8 m)</td>
<td>2.5 2.5 2.75 7.5 7.5</td>
<td>7.5</td>
<td>1</td>
<td>03-11-2003 (last reading on 07-08-2003)</td>
<td>Jump in dil. 5-15 mm on 03-11-2003 at a depth 3.5 m, same as <strong>Disassociation slice 3</strong>.</td>
</tr>
<tr>
<td>21_1</td>
<td>4</td>
<td>S0699-08 (8 m)</td>
<td>3.5 7.75 --- --- ---</td>
<td>7.75</td>
<td>51</td>
<td>03-19-2001</td>
<td>Jump in dil. 51 mm at depth, same as <strong>Disassociation slice 4</strong>.</td>
</tr>
<tr>
<td>21_2</td>
<td>4</td>
<td>S0201-05 (10 m)</td>
<td>--- 8.5 8.75 --- ---</td>
<td>8.75</td>
<td>20</td>
<td>09-25-2002</td>
<td>Installed in yielded ground close to residual, dil. at depth, failed after 415-17 mined.</td>
</tr>
<tr>
<td>20</td>
<td>4</td>
<td>S0699-06 (7 m)</td>
<td>5.5 5.5 5.5 5.5 6.5</td>
<td>6.5</td>
<td>22</td>
<td>07-22-2003</td>
<td>Very little change appears to be dilating/shearing 5 mm at depth 5.5 m. Jump in dil. after 390-20 mined.</td>
</tr>
<tr>
<td>19</td>
<td>5</td>
<td>S0699-02 (7 m)</td>
<td>7 7 7 7 7 7 7</td>
<td>7</td>
<td>0.5</td>
<td>09-27-1999</td>
<td>No Change with time. Installed in failed rockmass.</td>
</tr>
<tr>
<td>18</td>
<td>5</td>
<td>M0400-02 (10 m)</td>
<td>9 9 SD SD SD</td>
<td>9</td>
<td>1</td>
<td>08-23-2000</td>
<td>No Change with time. Installed in failed rockmass.</td>
</tr>
<tr>
<td>17</td>
<td>6</td>
<td>M0400-09 (10 m)</td>
<td>9 9 SD SD SD</td>
<td>9</td>
<td>0.25</td>
<td>08-23-2000</td>
<td>No Change with time. Installed in failed rockmass.</td>
</tr>
<tr>
<td>16</td>
<td>6</td>
<td>M1198-07 (10 m)</td>
<td>9 SD SD SD SD</td>
<td>9</td>
<td>1</td>
<td>12-01-1998</td>
<td>No Change with time. Installed in failed rockmass.</td>
</tr>
</tbody>
</table>

Notes: SD = Shadowed by Mining; Shaded cells = large deep dilation of rock mass or failed rock mass. ‘---’ = Instrument not installed (no data) or if after deformations no longer read.
Also based on static testing of cables (Hyett et al., 1995), typical yield of a double Garford cable with a 1 m bulge spacing occurs at about 1% strain (10 mm) or around 42.5 tonnes, with the peak capacity reached at a strain of 2 to 2.5% (20 to 25 mm) or 48 tonnes with the ultimate capacity at 3% strain (30 mm) or 52 tonnes. Here because the first 5 m are plain cable and would generally strip rather than rupture they could tolerate up to 150 mm of stretch over this distance before the ultimate capacity is reached, and were successful in retaining and holding together the rock mass above the back of the drift.

All the instruments during this monitoring period where manually read. Unfortunately readings where not always made on a regular basis. Thus, obtaining the exact date of sudden displacements, as did occur on a number of occasions, was not always possible. However, most instruments were generally read after a significant occurrence, such as stope blasting or a large magnitude event, and so give a reasonable estimate of the time of movement to within a couple of days. Also, it should be noted that these instruments where installed with the integrity of the intersections in mind, and as such do not traverse the most dense core of seismicity which was generally slightly to the south of the haulage drift (see Figure 3.6d and Figure 3.17f to Figure 3.21f). It was, however, still possible to infer and draw some conclusions between the instrument responses and effect on the distribution of the seismicity.

It is important to recognise that the displacements observed from the instruments are inferred to be unidirectional dilation of the rock mass when under lower confinement (5 – 20 MPa) within the immediate region of the haulage back (3 - 4 m). This unidirectional dilation towards the excavation surface is termed as bulking by Kaiser et al. (1996). When deep dilations are observed at a depth of 7 to 9 m, this dilation is probably not related to bulking (unidirectional) but could be either shear displacement or volumetric dilation related to mobilization of the rock mass at the point of disassociation through the formation of conjugate structures to the main fracture propagation direction. Dilations that have been measure from the instrumentation are denoted in figures as abbreviated to (dil) and should not be confused with the dilation parameter (Dil) used in Phase2 non-linear modelling to approximate the displacements. This dilation parameter applies a volumetric increase (in all directions) with a subsequent increase in the confinement (Section 3.7.4.5).

The depth of displacement, or assumed dilation (bulking), for all of the 15 instruments installed from the 9390-27 to the 9390-16 stopes, was analysed over time to provide a history of the progressive dilation with time over the back of the 9390 level footwall haulage. The nodal displacements versus time, depth and magnitude of displacement for each instrument on the
The depth of dilation greater than 1 mm, is indicated in the longitudinal view (Figure 3.25b), for three time periods, from December 2000, December 2002 and July 2003, after which the majority of the instruments were no longer read due to the haulage drift being partially paste back filled from cross cut 26 to 23. The drift was not tight filled along the entire length, but tight filled at stope 26 with a gradual slump to stope 23. The footwall haulage was effectively abandoned, with further access to stopes being from the hanging wall. These observations are summarised in greater detail in Table 3.6. For all of the instruments reviewed on the 9390 level the depth of displacement is taken at either the mid point between nodes, or greater, depending on the magnitude of the dilation\(^5\). Also, plotted on Figure 3.25b is the depth of measured failure that occurred on the 9415 level following the 3.0 Mn event and the later 2.6 Mn event in December 1999 (Yi, 1999).

Most instruments showed a gradual increase in dilation (bulking) over the first 2 to 4 m, from December 2000 to December 2002, with the maximum amount of dilation being at the collar of the hole or closest to the excavation back. This depth of dilation or bulking (Kaiser et. al., 1996), compares relatively well with the depth of failure or displaced rock observed early on the 9415 level above, (Figure 3.25b), and it was possible to achieve estimation of the non-linear post peak softening behaviour (Crowder et. al., 2006).

The majority of instruments showed a small jump (~ 2 to 5 mm), in the amount of dilation at a depth of 2 to 4 m following the 3.1 Mn event (June 28, 2001), and are probably the result of dynamic shaking created from such an energetic event. Correspondingly, there was an increase in seismicity at the same time as the 3.1 Mn event (Figure 3.24a, Figure 3.27b, Figure 3.28b, and Figure 3.29b) for the microseismic clusters around the 9390 level in slices 1 to 4, indicating the significance of the event in the sill region. However, the main point of interest, which was seen in 5 of the instruments (26-2, 24-1, 24-2, 22 and 21-1), was a sudden and generally large dilation (> 15 mm up to 50 mm) occurring at depth along the instrument (between the second to last and deepest anchor), which was found to occur just prior to disassociation in slices 2, 3 and 4, and has been summarised in Table 3.6, and can be seen for

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\(^5\) As the exact location for displacement in the instrument or nature of the displacement is not known (i.e. the nodal displacement could be the result of a single fracture or many multiple fractures equidistant between nodes), but can be inferred to have occurred somewhere between two nodes of the instrument, the mid point is generally taken as the depth of dilations > 1 mm. In Table 3.6, the depth has been increased from \(\frac{1}{4}\) to \(\frac{1}{3}\) to \(\frac{2}{3}\) in between the nodes over time to indicate that the deformations are increasing, even though all that is known is that one of the nodes is moving more than the other.
instrument 26-2 in Figure 3.26b and d. To show the progressive nature of failure across the sill, depth of dilation versus date have been shaded in Table 3.6, to indicate the potential and likely scenario that the rock mass surrounding the instrument has failed, or was close to failure based on the depth of dilation.

Instruments in cross cut intersections 19, 18, 17 and 16, located in slices 5 and 6, did not show any significant movement during the monitoring period (Appendix C, Figures C13 to C16). Based on the assumption from the yearly PCA analysis that these regions, (slice 5 and 6) had already failed and were probably approaching the residual strength of the rock mass, this aids confirmation that the rock mass may have failed. This also indicates that strain softening occurred with a significant drop in the deviatoric stress to a level not great enough to cause further deformations (i.e. stresses had been shed by failure through previous dilation and strong strain softening behaviour prior to installation). This is interesting as classically it might be considered that, at reaching residual strength, the rock mass would act plastically (i.e. deform with no volume change or increase in sustainable load). However, under this failed stress state the region may have become so dilated (strain-softened), that the subsequent induced stresses were not enough to provide closure, and thus no change is seen in instruments. Additionally, during the installation of the MPBX in crosscut 17, it was noted that: “the drilling conditions were good, but the hole took a lot of grout” (Mine Observer - Jones, 2000), another indicator that there may have been open interconnected fractures. Also during the mining of 415-18, 390-18, 390-17 and 415-17, no significant problems where noticed during drilling or extraction of the stopes indicating a significant drop in the stress level.

The two instruments in the transitional rock mass on either side of the pre-yielded rock mass appear to be instruments in cross-cut 27 (9390-27) and at the other side in cross-cut 20 (9390-20). Both of these instruments showed minor deformation and dilations (1 to 5 mm) at depth, right from the time of installation in early Jan 2000 and Oct 1999 respectively, potentially indicating that these regions on the 9390 level were close to the residual strength at installation but not quite there with some minor dilations still occurring.

Generally, the instruments located between these exhibit similar behaviour as previously discussed, i.e. gradual dilation at a depth of 2-4 m over a long time frame, followed by sudden increases in dilation at depth, occurring just prior to the point of disassociation noted from the PCA analysis (discussed in more detail for slices 1 to 4 in the next section). The two main exceptions to this were instruments in cross cuts 25 and 23. The instrument in the 25 cross cut was installed late, (early 2002), and showed minor dilation occurring at depth from very early
on (Appendix C; Fig C5), which gradually increases, with the exception of a sudden but moderate jump of 5 mm at a depth of 6 m, in May 2002, with no apparent connection to anything, except spalling of shotcrete and small ejected shards (Mine Observer - White, 2002). Additionally there was potential rehabilitation of support in this region at the time (White, 2002). Also, because of access issues following the rock burst (2.7 Mn) on the 26 and 25 cross-cut pillar on May 30, 2003, the last time the instrument was read was prior to this on May 12, 2003. Hence deformations of the rock mass surrounding this instrument potentially could have happened around the time of disassociation in slice 2, or with the pillar burst itself, but was never recorded.

The other instrument that also did not show similar behaviour to its neighbours was in cross cut 23. This instrument showed gradual and continuous, moderate to major dilations only to a depth of 4.5 m by July 8 2003 the date of the last reading, with no major dilation at depth (Appendix C; Figure C8). There are a number of potential explanations for the lack of deep dilation; firstly 20 m to the south of instrument is the old No. 3 ore pass, (Figure 3.25a), which had ballooned in dimension to 15 m EW and 8 m NS before the first major event (3.0 Mn in March 1999), and was subsequently stabilized with cable bolts and backfilled (Yi, 1999), but could act to slightly change the stress regime, and more importantly could change the fracture regime. Another potential reason is that this was a shorter 8 m instrument and any deep dilation that occurred could have occurred above the top of the instrument, or it could have been that the instrument was isolated in a portion of not fully failed ground and failure through deep dilation could have occurred after the last date of reading.

However, from the other instruments a reasonable picture based on the timing of deep dilation above the back of the 9390 level drift can be made (Table 3.6). This suggests that failure had already occurred in slices 5 and 6 by December 2000 and then propagated to slice 4 in March 2001 (based on instruments 21-1 and 21-2). This was followed by failure on the other side of the sill region in slice 1 in December 2002 (Instruments 26-1), and then shortly after this failure in slice 3 (and eastern part of slice 2) in between January 2003 and March 2003 (based on instruments 24-1, 24-2 and 22). The final failure occurred in between cross cuts 26 and 25 of slice 2 in June 2003 (based on Instrument 26-2) after the 2.7 Mn pillarburst. The final large magnitude event, the 3.5 Mn (Sept 2003), was probably associated with a portion of the rock mass that was not fully failed probably around the 25 stope in slice 2, but none of the instruments in this region could be accessed.
An additional point that should be stressed is that the rock mass in the footwall region failed before mining of the stopes in the ore zone, which would have caused jumps in dilation though blast vibration. Deep and sudden dilations recorded by the instruments and changes in the seismicity, were a result of the progressive failure of the footwall, and not related to the local mining of stopes immediately adjacent to the instruments, the sill failure being a result of the regional stress increases caused by mining far outside of the zone.

### 3.7.4.3 Correlation of Rock Mass Deformation to Changes in the Behaviour of the Microseismicity Above the 9390 Level

Detailed temporal breakdowns of the yearly run PCA technique for the individual 50 event temporal windows was performed for the other slices, and have been summarised with the event rate and average ellipticity for a given temporal window in Figure 3.27 to Figure 3.29, similar to the slice 2 cluster analysis. The following discussion will review the detailed analysis of the other slices first and then discuss in more detail analysis at the core of the sill in slice 2 in relation to the deformation identified from the SMART instrumentation.

#### 3.7.4.3.1 Detailed Analysis of Slice 3

In slice 3 just to the east of slice 2 (spanning the footwall haulage in front of stopes 23 and 22), it was again found that there was a strong increase in the ellipticity early on, starting in December 1999 and continuing into January 2001, with average ellipticities greater than 10 (Figure 3.27a). This coincides with the onset of the clustering density in late 2000 and early 2001, indicating that the rock mass probably reached its peak strength, and based on this localization and coalescence of events, had started to yield. The ellipticity again drops down to a background level between 5 and 10 but is less stable than in slice 2. The PCA trend is similar to that seen in slice 2 with an East-striking trend dipping slightly shallower to the south [090°, 33°]. This does suddenly change to strike West but steeply dipping to the North, [270°, 85°], for one temporal window, (covering April 12 to April 29 2001), just prior to the 2.3 Mn event, (April 22, 2001), and at the same time period that disassociation was determined for the cluster in slice 4. The PCA orientation then settles into a relatively stable trend as before, striking East and dipping to the south [090°, 30° to 55°] for the next two years with a steepening of the trend in early 2002 similar to slice 2. The first major change was not taken as the point of disassociation for this cluster, because the trend of the seismicity subsequently stabilized, and this first major trend is felt to be a reaction of the rock mass in relation to possible stress shedding caused by failure of the sill further east in slice 4 and the large magnitude 2.3 Mn event.
In March 2003 there is a significant swing in the strike and dip of the trend to \([130^\circ, 60^\circ]\) and a flurry of activity (Figure 3.27a and b), with an increase in the ellipticity to greater than 15, indicating the formation of a secondary localization confined within the footwall and around the haulage of the 23 stope. This change has been taken as the point of disassociation, with confirmation from the SMART instruments that flank this cluster.

The instruments in cross-cut 24 intersection (24-1 and 24-2) showed sudden and large dilations (38 mm and 15 mm respectively) at depth (7.5 m and 9 m respectively) in January 2003. Additionally, the instrument in cross-cut 22 intersection (22) showed a moderate but sudden dilation of 15 mm at a depth of 3.5 m, just prior to the point of disassociation (Table 3.6 and Appendix C). This dilation would cause significant strain softening at depth, with patches of the rock mass approaching the residual strength and initiating a change in the inter-fracture communication, thus affecting the trend of the seismicity as determined from the PCA. The zone above the 9390 Level haulage becomes aseismic around the 22 stope, at the same time as the dilation noted in the SMART instrument installed in the 22 intersection. Following this change, the PCA trend is less stable and becomes predominantly more horizontal following the 3.5 Mn event, (Sept 13, 2003), with a shift in the highest density of the seismicity further to the south and at the same elevation as the 9390 level haulage. The point of disassociation is subtler in this cluster than in the slice 2, and is based on the minor secondary localization and dilation of the surrounding instruments. This secondary localization is interesting as it shows that the rock mass can have similar behaviour to laboratory triaxial testing, with secondary localization in the post peak as identified by Thompson et al. (20___).

Additionally, following the jump in dilation at cross-cuts 24 and 22, rehabilitation of the back was attempted starting at cross-cut 22 at the end of March 2003. The comments from ground control were:

"spoke to operator - broken but drillable ground for first 20’ (6 m) then hits a totally broken area which jams & ends up snapping the rods. Driller is trying a bit designed for badly broken ground. He is also moving to the east, in hopes of better ground." (Mine Observer - Des Riviers, 27-Mar-2003).

This observation also adds credence to the concept that further into the rock mass, at the approximate depth of deep dilation starting at 6 to 7 m, the rock mass is even more fractured and dilated or sheared, corresponding to the position of the macro fracture or shear structure identified by the microseismicity. Further attempts to drill into the back of the drift resulted in similar conditions and the abandonment of this section of the haulage.
Figure 3.27  (a) Temporal window variation in PCA derived planes of mean strike, dip and ellipticity, for clustering events above the 9390 L FW haulage, Slice 3, and large magnitude events indicated versus time. (b) Event rate (moving average number of events every 7 days).
Figure 3.28 (a) Temporal window variation in PCA derived planes of mean strike, dip and ellipticity, for clustering events above the 9390 L FW haulage, Slice 4, and large magnitude events indicated versus time. (b) Event rate (moving average number of events every 7 days).
Figure 3.29 (a) Temporal window variation in PCA derived planes of mean strike, dip and ellipticity, for clustering events above the 9390 L FW haulage, Slice 1, and large magnitude events indicated versus time. (b) Event rate (moving average number of events every 7 days).
To summarise, the postulated behaviour based on the seismicity and SMART instrumentation, suggests that the rock mass in this region started to yield around January 2000 (based on coalescence and localization from the PCA ellipticity and clustering density). This is postulated as the formation of a macrofracture or shear structure and the rock mass potentially reached the peak strength sometime between January 2000 and January 2001. After this, progressive damage and probably moderate strain-softening (post peak) occurred in the core of the seismic cluster, (i.e. some dilation or shearing but is controlled due to confinement), over the next couple of years. Then in January to March 2003, the fracturing and shearing propagated well above the back of the haulage to cause more substantial dilations in the deep rock mass (identified by the change in the predominant PCA trend and deep dilatations in the SMART cables). This dilation or shearing at depth causing substantial strain softening, resulting in patches of the rock mass becoming aseismic and probably approaching the residual strength. This aseismicity and failure spreading through the cluster, (failing rock mass), resulted in an unstable PCA trend and a change in the group behaviour from the dominant trend of the macrofracture shear structure, until eventual aseismicity and complete failure occurs in 2004.

3.7.4.3.2 Detailed Analysis of Slice 4

In slice 4 further east along the sill, (spanning the footwall haulage in front of stopes 21 to 20), there is only a very minor increase in the ellipticity early on, (Figure 3.28a), indicating that this region was probably already in the post peak and had yielded prior to the installation of the microseismic system. Also, the clustering density is not reached in this region until after the July 2001 flurry related to the 3.1 Mn event (Figure 3.28b, Figure 3.10c). This is most probably due to an incomplete data set and identifies one of the main issues regarding reliance on the seismic clustering density as an indicator alone.

The average trend of the PCA planes is not as stable in the first part of the monitoring period in late 1999 and early 2000 (Figure 3.28a), and although the overall trend is similar to slice 2 and 3 at [090°,50°], there is a great amount of dispersion in the poles of this cluster for the time period associated with the large magnitude 2.6 Mn (12-17-1999), 1.6 and 1.7 Mn (01-09-2000) events (Appendix B2, Figure B2.16). These large events were thought to have occurred within the region of the 21 and 20 stopes close to the footwall haulage on the 9390 level and close to the electrical sub (Bawden, 2000). The influence of these large events and relation to this cluster are emphasised by the increased event rate at the same time.
Following these events the PCA trend does settle down for a short period with an average orientation of strike and dip around [090°, 60°], (Figure 3.28a), however, in March 2001, there is a large change with significant scatter in the determined poles and a loss of poles due to ellipticities determined to be less than 2.5, resulting in a swing in the average trend to [249, 32], after which the average trend becomes unstable, significantly following the 3.1 Mn event (Figure 3.28a). This point of disassociation, corresponds to deep dilation of 51 mm at a depth of 7.5 m, noted in the SMART cable installed in the intersection of the 21 cross cut (Instrument 21-1, 03-19-2001 – Table 3.6, Appendix C). Again, as in the cluster of events in slice 3, the change in the trend is related to deep dilation or shearing of the rock mass related to the potential macrofracture shear structure, causing regions of the rock mass to dilate and strain soften, resulting in patches of aseismic behaviour effecting the trend of the group behaviour of the seismicity and the inter-fracture communication through this region.

After the point of disassociation the seismicity does continue and a significant change in the trend is noticed during or following the 3.1 Mn event (06-28-2001), which was thought to be located in the footwall, somewhere close to the 24 stope. Also, after the first SMART instrument failed in cross-cut 21 a second instrument was installed in May 2001 (21-2, Table 3.6 and Appendix C). This instrument showed an increase in dilation of 2 mm at depth of 8.5 m following the 3.1 Mn event, and then showed some gradual dilation or shearing of 10 to 15 mm over the entire length of the instrument, until the 9415-17 stope was blasted on Sept 25 2002 (415-17 mined Sept to October 2002), after which the readings of the SMART instrument became unstable.

Again, as in clusters in slice 2 and 3, the change in the trend of the PCA derived planes, (the point of disassociation), appears to be related to dilation occurring deep within the rock mass causing strain softening and associated portions of the rock mass to become aseismic, altering the trend of the seismicity or fracture propagation from the potential macofracture shear structure formed earlier on. It is felt that, at this time, the majority of the rock mass in this region has completely failed and was close to the residual strength. The second instrument installed in cross cut 21 shows some residual movement but was probably already installed in predominantly failed ground.

3.7.4.3.3 Detailed Analysis of Slice 1
In slice 1 at the west end of the sill, (spanning the footwall haulage in front of stopes 27 to 26), analyses indicate an increase in the average ellipticity to >10 between January and September 2000, similar to the clusters analysed in slices 2 and 3 but not quite as significant (Figure...
The clustering density is reached in two separate clusters situated above and to the south of the 9390 level footwall haulage in front of the 27 and 26 stopes by December 2000, (Figure 3.10b) indicating that potential interaction and coalescence of fractures is occurring and that the rock mass is in a state of yield. The PCA analysis was run combining both clusters as some seismicity did exist in between. Although chaining, (simultaneous occurrence of events in distinct clusters), may be occurring between the two clusters early on to strengthen the PCA trend based on geometry, (i.e. they appear to be occurring either side of the cross cuts), the number of events and distribution did not warrant separating them out. It is felt that the application of the PCA technique is still valid to identify changes in the trend, as aseismicity in one cluster, (which occurred in the far west cluster) will affect the trend and will identify that some change in the rock mass is occurring. Also, it should be stated again that the full width of this slice is not populated with events, and that some seismicity undoubtedly occurred in the far west end as a result of the caving failure that occurred in the 28 stope between 9370 and 9450 levels in January 1997 before the system was installed. This may also be an explanation of the less strong ellipticity early on, as the rock mass had probably started to yield prior to the installation of the system. This is also backed up by the fact that the SMART instrument installed at cross cut 27 showed minor dilation, (1 to 5 mm, the later at the collar of the hole), over the entire length of the instrument (Table 3.6, Appendix C). This indicates that this instrument was probably installed in a yielded rock mass in the post peak that was close to residual strength.

The average PCA trend during localization and high ellipticity, is around \([090^\circ, 33^\circ]\) and is relatively stable (Figure 3.29a and Appendix B2), conforming to the same trend identified in other clusters but slightly flatter, and assumed to be the predominant direction of the macrofracture structure creating yield of the rock mass. This trend swings and flattens in 2001 but is still relatively stable to an orientation of \([067^\circ, 24^\circ]\), still probably associated with this structure. In early 2002 (January and February), the trend swings even more to \([050^\circ, 48^\circ]\) due to the western part around stope 27 becoming aseismic and the there is a significant change in August 2002 with the trend oriented North South \([357^\circ, 82^\circ]\). This latter change has been taken as the point of disassociation for the western part of this slice, although potentially the more subtle change in early 2002, could also qualify. Here interpretation of the point of disassociation is more difficult.

After August 2002, the PCA trend becomes unstable, with a number of poles being dropped due to low ellipticity, and another significant change in orientation to \([359,55]\) following the sudden dilation of 50 mm in the immediate back (4.5 m depth) of cross cut 26, as identified.
From SMART 26-1 (Table 3.6 and Figure 3.26a & c). Through 2003 and into 2004 the trend is very unstable with a large number of poles being dropped for any given temporal window due to low ellipticities, and indicating a more volumetric event cloud than one that is conforming to the macrofracture structure. In mid 2004 the event rate for the region significantly drops and becomes aseismic. The events persist in the eastern part of this analysis slice after disassociation, primarily due to the fact that the events are associated with the main cluster around the 25 stope in slice 2, and are the western edge of this cluster, that does not show disassociation until June 2003.

This is the only cluster discussed here that does not show disassociation following sudden deep dilation of the rock mass, but is related to the aseismicity that occurs in the western edge of the slice in front of stope 27, (Appendix B2). It is felt that this is primarily due to this region being transitional and in the post peak, but not quite at the residual strength at the time of the system and SMART cable installation.

3.7.4.3.4 Detailed Analysis of Slice 2 and use of the CMPCA

As discussed earlier the temporal analysis of the cluster of events in slice 2 showed relative stability of the PCA trend up until the point of disassociation (Figure 3.23a), apart from four occasions when the number of calculated PCA planes with ellipticities > 2.5 significantly dropped for a couple of temporal windows, as can be seen in Figure 3.24b.

To determine whether the selection of the temporal windows using a 50 event window with a shift of 50 had any effect on the distribution of the planes, as well as the effect of lower number of events in the first temporal window at the start of the years, the PCA analysis was run on the entire cluster of events from 1999 to the middle of 2004 using the average D value, (22 m) for the spatial window. This analysis was performed by first applying the 50 event temporal window with a shift of 50 events, and then comparing this to a continuous central moving principal component analysis, (CMPCA), centring the temporal window on ± 25 events and shifting the temporal window every event (Appendix A2.3). An additional objective was to also see the effect on the ellipticity at the start of the monitoring period when localization occurs. In order to properly evaluate the average ellipticity it was necessary to remove outliers that had high ellipticity based on only a few events in the spatial window. This was performed by filtering out all events with low F-parameters (ratio of K/N') < 0.5 (50%). The results of the analysis using a 25 point moving average to smooth the data can be seen in Figure 3.30. The continuous ±25 continuous shifting analysis compares very closely to the 50 event, shift 50 method.
Figure 3.30 Temporal variation of PCA planes determined for the clustering events above the 9390 level haulage in slice 2, (a) using 25 point running average of the PCA strike and comparing the analysis using a 50 event temporal window with a 50 event shift (N'=50), versus a 50 event continuously shifting temporal window centred on ± 25 events and a shift of 1 event (N'=Cont +25), and performing 25 point running average on the ellipticity calculated for both analyses. (b) 25 point running average of the PCA dip for (N'=50) and (N'=Cont +25). Note that all planes where filtered out if the F-parameter was less than 0.5 (50%). Also, noted are the large magnitude events, the points of disassociation in each slice and the points of sudden and deep dilation in the SMART cables.
When comparing Figure 3.23a to Figure 3.30a, the major temporal changes related to the point of disassociation and ellipticity are the same, however, there are a number of additional spikes (minor temporal changes) that can be seen in Figure 3.30a labelled 1 to 5. The main reason for the difference is that the average strike and dip in Figure 3.23a, is constructed from the maximum average concentration of poles on the lower hemisphere stereographic projection, while the average strike and dip in Figure 3.30 is constructed through the direct averaging of the vector magnitudes of all poles regardless of clustering. This results in the addition of minor trends that are ignored when using the stereographic maximum concentration, and is not suitable for determining the true mean 3D orientation of the planes, however, these minor temporal changes do indicate that something different is affecting the group behaviour of the event cloud. At time frames 1, 3, 4 and 5, there was a significant number of PCA derived planes that were rejected, as a result of determined ellipticities being < 2.5 (Figure 3.24b and Figure 3.30). This results in greater scatter of the determined poles (note: the identified time frames on Figure 3.30 have been labelled with the corresponding temporal window numbers for identification in Figure 3.22a to d). Points 1, 3 and 4 appear to coincide with the points of disassociation in the other regions of the sill, and it has been postulated that as the rock mass dilates in these zones, causing strain softening and stress shedding, there is some regional readjustment occurring, generally in the form of intensification of events in the core of the cluster in this slice, but resulting in a more volumetric trend than a strongly planar one.

Analysis of the initial drop in ellipticity indicates that it started to fall in late August 2000, just prior to the expansion to the mine wide system on September 22, 2000, after which the ellipticity decreased more rapidly. This indicates that the fall in ellipticity is not a result of the system expansion; however, the eventual level that it reached may be lower due to the larger system, and higher trigger level threshold. The key point however, regarding the ellipticity is not the decrease but the significant increase that occurred following a lower background level, initiated prior to the 2.8 Mn event, and the increase in activity resulting in the clustering density being reached. This is the point that localization occurs and represents the point of true yield, the rock mass being at its peak strength. Following localization the rock mass is thought to be in a post peak state based on source parameter analysis of a similar failure at the Golden Giant mine (Chapter 4).

The swing in strike associated with time period at point 2 (Figure 3.30a), is the steepening of the PCA trend at the beginning of 2002. In reality the strike does not rotate gradually to 210° as indicated in the Figure 3.30, but is the result of a small number of PCA planes being orientated
vertically with the strike switching 180°. As no external influence could be identified other than a reduction of mining activity in the region, it is felt that this is just the variation in the propagation of the events and potential fractures, during a relatively less seismically active period.

The most marked drop in the number of planes that had ellipticity < 2.5 (Point 4 Figure 3.30a and Figure 3.24b), was found at the same time as deep dilation in the 24-1 and 24-2 SMART instrument (corresponding to Windows 34 and 35 in Figure 3.22b), that caused an intensification of events to the south of the haulage in front of the 25 stope (Appendix B2, Figures B2.32c & d). These more densely clustered events were spatially distributed more spherically, and resulted in planes being dropped and a relatively scattered trend during this time. The point of disassociation could have been taken at this point, except for the fact that this was short lived and the PCA trend stabilized back to the previous dominant trend, until late June 2003, the point of disassociation chosen for this cluster.

The other significant drop in the number of planes with ellipticity came after disassociation at point 5 (Figure 3.30a and windows 53 and 54 Figure 3.22c & d), when there was a sudden increase in the event rate in August 28, 2003 (Figure 3.24a), again with planes being dropped due to tight clustering. The event rate subsequently dropped to background levels until another smaller increase, just prior to the 3.5 Mn event, on September 13, 2003 which caused the back of the 9390 level haulage in this region to become completely aseismic. It is difficult to identify what the cause of the flurry of activity was, as the only mining activity going on in the region was the drilling of the 9390-22 stope, and sometime in late August this section of the footwall haulage was backfilled with paste fill from stope 26 to 23. Although not tightly filled, it would have been anticipated that the paste filling should have helped to stabilize the region, although not being able to get the fill tight will have resulted in it being less effective. Thus, this last flurry of activity, and large event, may have been the last part of the sill in this region failing completely, with probably significant dilation occurring but unfortunately not being measured as the instruments had already been abandoned.

3.7.4.3.5 Summary of Behaviour Interpreted From Seismicity and Displacement Instrumentation

From the PCA analysis, seismic density and identification of dilation from the SMART instrumentation, it has been postulated that the significant increases in the average ellipticity,
are related to localization corresponding to true yield of the rock mass and probably close to the point of the peak strength being reached. The dominant PCA trend appears to represent the formation of a macrofracture structure, either occurring through the fracturing of intact rock and networking of these fractures or through the scenario of slip on pre-existing joint related to the C-set (Figure 3.4) combined with inter joint bridge fracturing to form a persistent structure. The main factor contrary to this latter scenario is the scarcity of C-set joints in this region, and the fact that the general dip of these structures at Williams is considerably less than the identified PCA trend. The point of disassociation appears to be related to either deep dilation or shear created by propagation of the macrofracture structure to above the back of the drift. This dilation, at the point of disassociation, is interpreted to represent significant strain softening and ‘failure’ of patches of the rock mass such that they approach their residual strength. It is not known, but is anticipated that following localization, during the continued damage accumulation at the core of the seismic cluster, some minor dilation and gradual strain softening occurs, but the significant changes do not come until the point of disassociation when the rock mass looses significant strength. The changes in the PCA trend at the point of disassociation are postulated as being caused by regions of the rock mass becoming dilated and thus aseismic, this could be through the activation of a conjugate shear structure creating increased degrees of freedom of block movement. The strongest evidence for this activation of a conjugate set is from the analysis of slice 2. The PCA technique appears to be able to capture these changes in the rock mass from pre-peak to yield and peak to post peak to approaching the residual strength and, although not perfect, gives a reasonable indication of the state of the rock mass. The spatial trend determined from the PCA also appears relatively sensitive to changes occurring in the rock mass as measured by the conventional SMART cable instrumentation.

This macrofracture shear zone has a defined continuity along strike of the haulage drives, from the 26 stope in slice 1 to the 20 stop in slice 4, where by at either end the seismicity is more dispersed. The orientation of this structure determined from the mean PCA derived planes after localization, appears to indicate that the structure or group behaviour of the events is oriented on average between [090, 33] to [090,57], for slices 2 to 4, depending on the time. The strength in the strike prior to disassociation is undoubtedly partially controlled by the geometry of the problem with the haulages striking East-West. However, it is noted that the strike can vary from sample to sample between 060° and 110°, which may be more a function of the sampling itself than the actual variation.

In order to understand what the stress conditions are that are driving this failure, it is important to determine what the stress state and stress path is, and whether the post peak conditions of
strain softening can be back calculated to show the same displacements and similar behaviour as observed in the field. The post peak behaviour in this confined rock mass is undoubtedly strain softening; as previously stated, on commencement of mining activity in the main sill (stopes of 20 to 27), no problems associated with drill hole loss, squeezing or major seismicity occurred within the region, with mining appearing to be in a low stress environment, caused by the failure and stress shadowing of the footwall region. Additionally, the relationship of the orientation of this structure in terms of mean dip will be investigated further using a ubiquitous joint approach.

### 3.7.4.4 Linear Elastic Stress Path and Shear Stress Analysis

In order to determine the potential stress history, the starting point for this large three dimensional (3D) mine model is to use 3D linear elastic modelling. Although linear elastic models cannot exhibit yield, failure, and stress redistribution, (in the case of strain softening behaviour), it has been shown through a number of studies that applying a semi-empirical approach by definition of damage limits, calibration and prediction of the potential extent of damage can be achieved (Diederichs et. al., 2002; Falmagne, 2002; Martin, et. al., 1999; Nickson et. al.,1998, Castro et. al. 1996). The other objective was to use the resultant linear elastic stresses created from the large global model and apply them to a smaller simpler two dimensional (2D) non-linear finite element model (Phase², Rocscience, 2006), for regions that could be approximated with a two dimensional assumption to investigate the potential post peak parameters (Crowder et. al., 2006).

As previously mentioned, two boundary element models were utilized in this study, Examine³D (Rocscience, 2007) and Map3D (Mine Modelling Ltd., 2006), the former for general stress analysis and a stepping stone to the Phase² modelling, the later for detailed analysis of linear elastic stress histories around openings, utilising a stress averaging routine developed originally at the Noranda Technology Centre (Rizkalla, 2001) and detailed by Falmange (2002) and summarised here. The stress averaging routine is used to determine the stress histories of the principal stress magnitudes, (regardless of tensor orientation), of certain regions representing the cluster of events, and is preferred over direct determination of stresses at the event sites, as it greatly reduces the data. Additionally, the stress averaging routine employed allows for unbiased stress averages, regardless of stress interpolation point density (Falmagne, 2002). Also, use of the two different modelling packages aided in checking the results of both models, such that agreement in principal stresses to < 2% could be achieved through set mesh
discretization and modelling efficiency parameters\(^6\). The stress regime is based on the far
field stresses summarised in Section 3.4.3.

The three dimensional model that was used in this study is shown in Figure 3.1. It was found
that the model could be reduced to only the Williams Mine and Golden Giant Mine, eliminating
the David Bell Model, without compromising the stress magnitude\(^7\) in the vicinity of the sill
(Figure 3.31a). A number of analysis grids that are utilised in the next section are shown in
Figure 3.31b. The transverse sections were placed slightly off centre of the seismic analysis
slices, and were used to determine stress histories in the region of the cluster of events.

### 3.7.4.4.1 Overall Linear Elastic Stress Observations

The calculated maximum principal stress \((\sigma_1)\), has been plotted for the yearly mining steps in
Figure 3.32. As can be seen there is a strong increase in stress in the region of the footwall
advancing from East to West (right to left) over time. Although the highest stress intensity is
found in and around the mined stopes in the eastern half of the sill (around stope 17 and the
dyke), the stress in the footwall haulages \((\sigma_1 = 80 \text{ to } 90 \text{ MPa contour})\) advances faster than in
the ore zone, well ahead of the mining front, and is most evident in the plot of stresses in
December 2000 (12-2000). This modelled stress wave is similar to what has been identified
from the analysis of microseismicity and the SMART instrumentation, and fits with the general
assumption that the sill region analysed here fails from East to West. The confinement or
minimum principal stress, \(\sigma_3\), was found to be relatively high compared to other similar studies,
(Diederichs et. al., 2002; Falmagne, 2002) in the region of the footwall seismic clusters, at
around 20 to 30 MPa, increasing only slightly to the later value with time. The significance of
this will be discussed in the next section. The location of the seismicity in the footwall occurs in
the region of the maximum principal stress between the footwall drives, failure being initiated
here before failure occurring in the ore zone at the same elevation.

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\(^6\) Examine3D was modelled with a moderate mesh discretization for reasonable run times, using linear stress elements, stress
tolerance 0.0001 and required a constant stress multiplier (1.168) to achieve the same induced stress magnitude for this large
model as Map3D using constant stress elements and fine discretization based on modelling parameters of STOL=0.1, AL=4,
AG=4, DOC=2, DOL=2, DON=0.5, DOE=4, DOG=4 and DOR=5.

\(^7\) Elimination of David Bell resulted in a Principal stress change of 0.5%, while elimination of Golden Giant resulted in a stress
change of up to 10% in high stress areas, for the same year of analysis 12-2003.
Figure 3.31  (a) Longitudinal view of Williams and Golden Giant Mine used for three dimensional boundary element stress analysis. The colour represent yearly mining, determined from 1995 until September 2005. (b) Detailed view of analysis grids, encompassing the footwall development (Mining step 12-2000 shown).
Figure 3.32 Contours of Maximum principal stress ($\sigma_1$), plotted on the mid level grid (9407 elv.) between the 9390 and 9415 levels, for yearly mining steps 12-1999 to 12-2004. Left inset, contours of $\sigma_1$ on a transverse grid (Section 9400E) through the centre of the footwall at stope 26, and right inset yearly mining geometry.
There are however limitations in the linear elastic model, as previously stated. Regions that would fail in reality are not modelled, and especially the thin pillars left between the 17 and 16 stopes (the dyke) and the 30 and 28 stopes, would probably have failed. Modelling these failures would increase the stresses very slightly, but the overall stress path would stay the same. Modelling the progressive failure in the footwall by removal or softening of the material could also take the model one stage further, and would be expected to more significantly change the induced stress regime. The intention was however, to see the basic linear elastic stress path, and to what extent this could be used to calibrate damage limits based on this simplified model. Additionally, these linear elastic stresses from the regional model are used for the boundary stresses applied to the simplified non-linear model.

### 3.7.4.4.2 Correlation of Stress with Observed Damage Points from PCA analysis

The average linear elastic stress paths for each transverse section between the 9390 and 9415 levels, representing the clusters of seismicity in the footwall, have been plotted in Figure 3.33 to Figure 3.38, as stress paths in $\sigma_1$ versus $\sigma_3$ space and as principal magnitude histories versus time. Four averaging polygons are used to investigate the stress path at various places within the cluster of seismicity, and these averaging regions are indicated in Figure 3.33, and are kept constant for the other transverse sections. The main polygon of interest that represents the core of the seismic clusters is region 2. At the corresponding mining steps in the model, the point in time of coalescence and point of disassociation from the PCA have been identified on the stress path in each one of the plots (larger squares). As no conclusive results could be achieved for slices 5 and 6 (stress sections 9555E and 9600E, Figure 3.36 and Figure 3.38), the potential behaviour based on the other slices has been estimated. The other stress path of interest is from region 4, which is directly above the back of the 9390 level haulage. These stress paths have been identified at the time the path passes the lowest damage limit corresponding to initiation, and at the time that deep dilation occurred in the back of the drift (larger circles). As reference, the peak rock mass strength envelopes in terms of the Hoek-Brown and equivalent Mohr-Coulomb parameters (Table 3.7) are plotted in Figure 3.33a to Figure 3.38a. These envelopes were determined from the intact rock properties based on an average GSI of 60 for the footwall region using the relationships proposed by Hoek et. al., (2002). These envelopes tend to over predict the strength of the rock mass when combined with linear elastic modelling and under predict the depth of failure around openings (Martin et. al., 1999). This is notable from the stress paths, as none transect the rock mass peak strength envelopes during the loading phase, although failure is evident.
Figure 3.33  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 1 and 2 on section 9400E (26 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$, $\sigma_2$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds.
Figure 3.34  (a) Average linear elastic stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 2 on section 9435E (24 stope region). Inset shows location stress averaging polygons. (b) Average linear elastic stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds.
Average Linear Elastic Principal Stresses for the Core of Cluster of Events
Footwall 9390 Level Cluster Slice 3 - Sec 9475E (390-22 stope region)

Mohr-Coulomb Peak Strength
(Rock mass)

Brittle-Ductile intact
Transition S1/S3=3.4

Hoek-Brown Peak Strength
(Rock mass)

H-B Brittle Limits

Brittle-Ductile disturb.
Transition S1/S3=2

Sigma 3 (MPa)

Sigma 1 (MPa)

Date (mm/yy)

Average Principal Stress versus Time for the Core of Cluster of Events
Footwall 9390 Level Cluster Slice 3 - Sec 9475E (390-22 stope region)

Oct 2003 390-22
May 2003 390-20 & 415-20
Nov 2003 415-22
Apr 2004 390-19, 390-24, 370-24

Average Principal Stress (MPa)

Fig. 3.35 (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 3 on section 9475E (22 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds.
Figure 3.36  (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 4 on section 9515E (20 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds.
Figure 3.37 (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 5 on section 9555E (18 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds.
Figure 3.38 (a) Average stress paths from 1993 to Feb 2005, for different regions within the cluster of events above the 9390 level, Slice 6 on section 9600E (16 stope) Inset shows location stress averaging polygons. (b) Average stress $\sigma_1$ and $\sigma_3$ versus time. Sill stopes mined, indicated with solid diamonds.
Table 3.7  Failure Criterion and Parameters Used in Modelling. (Rock mass parameters based on a GSI = 60)

<table>
<thead>
<tr>
<th>Intact</th>
<th>Rock Mass Peak Strength</th>
<th>Post Peak Strength</th>
</tr>
</thead>
<tbody>
<tr>
<td>Criterion</td>
<td>UCS (MPa)</td>
<td>$m_l$</td>
</tr>
<tr>
<td>Lab Meas. H-B/M-C$^1$</td>
<td>173.2</td>
<td>12.9</td>
</tr>
<tr>
<td>Brittle Post Peak H-B</td>
<td>175</td>
<td>10</td>
</tr>
<tr>
<td>Brittle Post Peak M-C$^3$</td>
<td>162</td>
<td>43</td>
</tr>
<tr>
<td>Plastic Post Peak H-B</td>
<td>175</td>
<td>10</td>
</tr>
<tr>
<td>Plastic Post Peak M-C</td>
<td>162</td>
<td>43</td>
</tr>
<tr>
<td>H-B Brittle limits – A$^5$</td>
<td>175</td>
<td></td>
</tr>
<tr>
<td>H-B Brittle limits – B</td>
<td>175</td>
<td></td>
</tr>
<tr>
<td>H-B Brittle limits – C</td>
<td>175</td>
<td></td>
</tr>
<tr>
<td>H-B Brittle limits – D</td>
<td>175</td>
<td></td>
</tr>
</tbody>
</table>

Notes:
1. Based on limited triaxial testing of Hemlo core from Golden Giant Mine, UCS based on Hoek-Brown (Queen’s, 1994).
2. Rock mass strength (Peak Strength) based on Hoek et. al.(2002) with $D=0$ and $a=0.5$, and simplified intact parameters (Crowder et. al., 2006).
4. Brittle or strain softening post peak parameters found from back analysis of near field displacements above 9390L using 26-1 and 26-2 SMART instruments. These are directly applicable to within one tunnel diameter of drift but may not be exact for deep field displacements due to a change in the dilation parameter. Plastic post peak parameters are included for reference but do not simulate displacements or observed behaviour.
5. Hoek-Brown Brittle parameters used to define damage limits in Linear Elastic Modelling only, and based on Martin et. al., 1999).

$E_i$, Intact Modulus = 55 GPa
$E_m$, Rock mass Modulus = 17.8 GPa (based on Hoek et. al.(2002) with $D=0$)
$\nu$, Poisson’s Ratio = 0.25
The suggested reason for this, in massive to moderately jointed brittle rocks, is that during
the fracturing process there is non-simultaneous mobilization of cohesion and friction, [i.e.
friction cannot be mobilized until the cohesion has been lost (Martin and Chandler, 1994)], and
led to the proposed use of the Hoek-Brown brittle failure parameters, (Martin et. al., 1999), to
determine the onset of fracturing and depth of failure. This is achieved by setting the value of
m=0 in the Hoek-Brown equation, (equivalent to the friction term in the Mohr-Coulomb
equation), and Martin et. al. (1999), has identified, through empirical evidence, that initiation of
stress induced brittle failure close to openings occurs when the maximum tangential stress
exceeds 0.4 ± 0.1 of the uniaxial compressive strength ($\sigma_c$) of the material, which equates to an
s = 0.09 to 0.25, (note the s value is equivalent to the cohesion in the Mohr-Coulomb equation).
Falmagne (2002) and Diederichs et. al. (2002), have noted through similar empirical analyses
that these damage limits can be used to calibrate linear elastic models to observed damage
initiation. The former author found that damage initiation, occurring at the same time as
coalescence and the point of true yield can occur as low as 0.28$\sigma_c$, however, it should be
pointed out that variation in the rock strength, modelling accuracy and stress averaging can
easily effect this calibration. For this reason, a number of damage limits using the Hoek-Brown
brittle parameters (Table 3.7) have been superimposed on Figure 3.33a to Figure 3.38a, to
identify any trends in the data. A set UCS based on the mean (175 MPa) has been used
throughout these analyses for ease of data interpretation, and for comparison to other analyses
by Falmagne (2002) and Diederichs et. al. (2002) who also approximated damage limits based
on mean UCS. It is important to recognise that variations in the strength related to the foliation
do exist, however, as previously stated the direction of loading is oriented at 70° ± 5° such that
the maximum strength of the intact rock could be considered.

The general linear elastic average stress path for the core of the clusters of events (Region 2,
Figure 3.33b), indicates progressive almost monotonic loading in the maximum principal stress,
$\sigma_1$, oriented horizontal and perpendicular to the ore body (increasing from 55 to 100 MPa). The
minimum principal stress, $\sigma_3$, orientated vertically, also gradually increases to a lesser extent,
with this increase being less the further east along the sill (increasing from 20 to 30 MPa).
Additionally, the intermediate principal stress, which is generally disregarded, stays relatively
stable over time with little difference between analysis regions, and is oriented horizontally and
parallel to the orebody (increasing from 30 to 35 MPa). As mining commenced in 2003, in the
central part of the sill, there is a notable drop in principal stresses, as the footwall haulage is
shadowed by mining in the sill, except for the region in front of 9390-26 stope (Figure 3.33b),
which is not shadowed until 2006. The point of disassociation and complete failure of the
confined rock mass, was identified to occur in slices 1 to 5, based on the seismic analysis, prior
to stress shadowing. The only exception to this is the seismicity in slice 6, (stress path for section 9600E, Figure 3.38b). Although no conclusive result from the seismic analysis could be achieved, based on the modelled stress levels, this region may not have achieved complete failure before stress shadowing occurred. The point of initiation is difficult to interpret for all of the sill, as the seismic system was installed for the most part following initiation of seismicity in the region. However, in slice 2 it has been assumed, because there was a predominant shift in the seismic density to this region, that the point of initiation occurred in January 2000. This stress level is almost equivalent to that indicated at coalescence and localization, and based on this and the similar findings of Falmagne (2002) (discussed in the introduction) this point of observed coalescence/ localization is assumed to be the same stress state as the initiation stress for the other slices.

Based on this linear elastic stress path, failure in the footwall of the rock mass is almost analogous to a laboratory triaxial test at constant confinement, with loading being increased gradually and monotonically in the same axis until failure, while maintaining relatively constant confinement. Here the loading direction is horizontal and perpendicular to the strike of the ore body.

From the identified point of coalescence and localization from the principal components analysis and clustering density, yield in the core of the seismic cluster (stress paths for region 2), occurs at the lower damage limit based on the Hoek-Brown brittle parameters at m=0 and s=0.09 or a constant deviatoric stress level of \( \sigma_1-\sigma_3 = 0.3\sigma_c \). This is consistent for seismic slices 1 to 4 (sections 9400E to 9515 on Figure 3.33 to Figure 3.36 and summarised in Table 3.8), and by applying the damage level to stress paths for seismic slices 5 and 6, the dates have been back analysed and are consistent with the observation presented previously. This point of coalescence and localization, (assumed to occur at the same level in stress as initiation, based on slice 2) is also consistent with observation of other researchers (Martin, 1999; Falmagne, 2002 and Diederichs et. al., 2002). The point of disassociation, postulated to be failure of the rock mass to a state close to the residual strength, occurs at a damage level slightly higher than this at m=0 and s=0.14, or a constant deviatoric stress of \( \sigma_1-\sigma_3 = 0.375\sigma_c \), and is again consistent for seismic slice 1 to 4 (Figure 3.33 to Figure 3.36, Table 3.1). The important point here is that the linear elastic stress increase required to take the rock mass to failure is relatively small from the point of coalescence to disassociation, and in terms of the maximum principal elastic stress is an increase of \( \sim 10 \text{ MPa} \), or 12.5 percent, beyond 80 MPa at a confinement of around 25 MPa. The question arises as to whether this failure could have been self propagating with time, i.e. if no more mining occurred, would the failure and seismicity still
have continued? It is this author’s feeling that the region would have stabilized, as this failure appeared to be relatively gradual over time in contrast to the more rapid failure observed at the Golden Giant Mine (Chapter 4).

### Table 3.8 Comparison of damage levels using the Hoek-Brown Brittle parameters based on linear elastic modelling between for Slices 1 to 4 $\sigma_c = 175$ MPa, $m = 0$ (note: $\sigma_1=\sigma_3+(m\sigma_3+s\sigma_c^2)^{0.5}$)

<table>
<thead>
<tr>
<th></th>
<th>Slice 1 and 2 Sec 9400E (26 Stopes)</th>
<th>Slice 2 Sec 9435E (24 Stopes)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Date</td>
<td>$\sigma_1$ (MPa)</td>
<td>$\sigma_3$ (MPa)</td>
</tr>
<tr>
<td>Initiation</td>
<td>Jan '00</td>
<td>73</td>
</tr>
<tr>
<td>Coalescence/Localization</td>
<td>Jul '00</td>
<td>76</td>
</tr>
<tr>
<td>Disassociation</td>
<td>Jun '03</td>
<td>89</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th></th>
<th>Slice 3 Sec 9475 E (22 Stopes)</th>
<th>Slice 4 Sec 9515 E (20 Stopes)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Date</td>
<td>$\sigma_1$ (MPa)</td>
<td>$\sigma_3$ (MPa)</td>
</tr>
<tr>
<td>Initiation</td>
<td>---</td>
<td>---</td>
</tr>
<tr>
<td>Coalescence/Localization</td>
<td>Jan '00</td>
<td>78</td>
</tr>
<tr>
<td>Disassociation</td>
<td>Mar '03</td>
<td>94</td>
</tr>
</tbody>
</table>

For the stress paths developed above the immediate back of the 9390 level footwall haulage (stress paths for region 4), the date of when the path first crossed the initiation/coalescence damage limit of $0.3\sigma_c$ line were determined from modelling. In general the predominant year for seismic slices 1 to 5, or the haulage between stopes 26 and 18, all pass this damage limit in around December 1997. There is evidence, that at around this time, the footwall haulage on 9390 level had started to show signs of stress deterioration. This deterioration was in the form of stress spalling occurring in the upper footwall haunch of the drift which was driven originally with a flat back, and would occasionally ‘rat tail’ up and cause slabs of the back to become unstable, resulting in increased support or up to a 1 m higher profile than designed (Bawden, 1997). The footwall drive design was subsequently changed to a shanty back. Other indications of stress effects in the region were the failures in the 28 and 30 stopes in April 1997 (Leblanc and Murdock, 2000).
The point of deep dilation in the instruments located on the stress sections analysed, occurs at a damage limit around 0.4σ_c (m=0, s=0.16). Prior to this dilation of the rock mass by greater than 1 mm had generally occurred to a depth of 3 to 4 m, however, for this analysis it was felt that trying to identify this point in time was not as significant, as the instruments, generally reached this point by a small jump in displacement following the 3.1Mn event in June 2001. This being more a result of dynamic shaking, than pure stress induced bulking.

The failure of the footwall region resulted in strain softening behaviour. This may be contrary to the fact that the stress paths lie below or close to the ‘intact’ brittle-ductile transition zone suggested by Mogi (1966), σ_1/σ_3 = 3.4, and plotted on Figure 3.33 to Figure 3.38 for reference. This could potentially suggest that the post peak behaviour should be plastic (i.e. ductile behaviour), however this was not the case. Also, at the Golden Giant mine in which a similar study was undertaken (Chapter 4), the confinement at failure was considerably greater, but still occurred on an equivalent brittle damage limit of 0.4σ_c, well below Mogi’s line. The observed behaviour at the Golden Giant mine was again strain softening. Falmagne, (2002), used the Mogi limit as a defining region to differentiate between material behaviour. Between the σ_1/σ_3 > 5 (spall limit determined by Falmagne, 2002), and down to σ_1/σ_3 > 3.4, strain softening behaviour would predominate, suggesting that below σ_1/σ_3 < 3.4 ductile plastic post peak behaviour would occur (see also Chapter 1, Figure 1.7d for comparable limits). For stress paths and failure points that were below the brittle-ductile transition zone, but did cave and produce seismicity, it was suggested by the author that stress averaging and linear elastic modelling of the problem may make these points invalid.

Diederichs (2000) has noted from a micro mechanical study of intact rock that Mogi’s line is potentially a physical threshold, below which no extensile fracturing and dilation can occur and that pure shear behaviour should predominate (Figure 1.7d). However, stress paths and the fracture initiation points that occur below this line are still valid, as the crack initiation threshold is universal for various scales regardless of confinement (Diederichs, 2000). These events are not expected to be pure shear, as based on the complimentary study at the Golden Giant mine (Chapter 4), in which a similar macrofracture shear structure was studied, and found to be at even greater linear elastic modelled confinement than here. The events had distinctively low shear wave to compressional wave energy ratios (E_s/E_p), these being of a range (E_s/E_p < 5) suggesting that dilation and potential extension fracturing is still occurring and that fractures were not pure shear (pure shear failures E_s/E_p > 10), but combined shear-tensile failures, with tension presominating.
It should be additionally recognised that discontinuities or stress driven fractures, will have an influence on the local stress field surrounding the fracture, especially if dilation occurs through small shear behaviour and asperity mismatch, not just dilation due to extension fracturing. This would, in reality, result in local variation in confinement, allowing extension fractures to propagate and interact. Further, Hoek and Brown (1980) identified that this transition zone did not hold for samples that had planes of weakness or had been previously broken, and in the absence of published guidelines suggested a transition zone of $\sigma_1/\sigma_3 = 2$ rather than 3.4. In this case the stress paths lie above this classification line.

This study and the complementary study at Golden Giant mine (Chapter 4), suggests that the rock mass, failed in a strain softening brittle fashion to cause stress shedding and stress shadowing, which is contrary to the theoretical behaviour based on Mogi’s transition zone for crystalline intact rock. Although this behaviour in the form of the formation of a shear band is expected for confined and ductile material the observed behaviour is through strain softening. It is suggested that either this is not a distinct classification and a transitional zone exists or that there are other factors at play that allow for the eventual failure to propagate to openings causing the significant strain-softening behaviour. It should also be noted that this modelling is based on linear elastic modelling which does not allow for local failure and stress shedding which would be expected to cause a reduction in the confinement and an potential increase in the induced principal stress. More research into the influence of discontinuities on the effect the transitional behaviour from brittle to ductile should be performed to confirm Hoek-Brown’s (1980) finding’s where this lower limit is potentially more applicable to the rock mass or disturbed laboratory samples.

### 3.7.4.4.3 Comparison of Clustering Density with Linear Elastic Yield Region based on the Hoek-Brown Brittle Failure Parameters

From the linear elastic modelling the point of coalescence and localization, determined to be the point of true yield of the rock mass, was found to consistently occur at a damage limit corresponding to a constant deviatoric stress term of $\sigma_1-\sigma_3 = 0.3\sigma_c$ or $m=0$ and $s=0.09$ using the Hoek-Brown brittle failure parameters. In order to compare the measured clustering density with this linear elastic damage level, the iso-surfaces for the clustering density of 5 events per 125 m$^3$ voxel for the time frame of 12-2001, were compared to the iso-surface applying these Hoek-Brown brittle failure parameters for factor of safety (FOS) < 1.0 (Figure 3.39) for the time frames of 12-1997, 12-1999 and 12-2001. As expected, the overall FOS < 1.0 envelope
determined for the comparative year of 12-2001, shows the potential for considerably more yield than identified from the seismicity at the clustering density. This is primarily due to the incomplete data set of the seismicity recorded; however, there are some interesting parallels. As can be seen in the initial stages of 12-1997, the brittle yield envelope (FOS < 1.0), first develops above the back of the footwall haulages in the region of the 27 to 23 stopes on the 9390 level, and progressively increases into 12-1999, until by 12-2001 this yield envelope encompasses the entire footwall between the haulages. Thus, leaving a donut like hole of non yielded rock in the region of the ore zone, and agreeing with the preferential failure of the footwall in this region and the location of the resultant seismicity.

This is more easily identified in a transverse section on 9400E going through the 26 stope as observed over different mining steps (Figure 3.40). In this linear elastic model, by again applying the Hoek-Brown brittle failure parameters, representing initiation, coalescence and localization the region in between the 9415 and 9390 levels show that full yield in the core of the pillar between the levels starts around 12-2001, and increases until yield engulfs the entire sill in 12-2004. Additionally, on the level above between the 9450 and 9415 levels at the top of the sill, similar behaviour occurs through the preferential yield occurring in this region starting in 12-1996. This preferential failure of the footwall, would undoubtedly reduce the failure in the ore zone as was experienced, especially if this failure is strain-softening in behaviour. This simple modelling again provides a useful insight into the failure processes that occurred within the sill region analysed.

Application of the Hoek-Brown rock mass strength parameters (Table 3.7) shows that these parameters are conservative for linear elastic modelling, as the failure that is predicated in the footwall when applying the brittle parameters does not materialise with the stronger limits (Figure 3.41). However, the rock mass parameters are applicable to characterise the strength of the rock mass in non-linear modelling, in which the rock mass can progressively fail and shed stress to other regions, and have been found to be applicable with the use of strain-softening parameters to match the deformations measured around drift openings (Crowder et al., 2006), and develop similar shear behaviour as identified by the PCA (section 3.7.4.5).
Figure 3.39 Comparison of (a) iso-surface of clustering density (5 events per 125 m$^3$ cell) at December 2001, versus (b) to (d) the linear elastic iso-surface of the brittle Hoek-Brown factor of safety <1.0 for $m=0$ and $s=0.09$ ($\sigma_1 - \sigma_3 = 0.3\sigma_3$) for year 12-1997, 12-1999 and 12-2001. Note perspective view is South, looking at the Sill region from the HW, and section 9400 E is identified for next Figure.
Figure 3.40 Progressive evolution from 12-1999 to 12-2004, of failure zone (FOS < 1.0) for transverse section 9400E (Slice 2 – 9390-26 stope), using the brittle Hoek-Brown failure parameters of \( m=0 \) and \( s=0.09 \) (\( \sigma_1 - \sigma_3 = 0.3 \sigma_2 \)).
Figure 3.41 Progressive evolution from 12-1999 to 12-2004, of failure zone (FOS < 1.0) for transverse section 9400E (Slice 2 – 9390-26 stope), using the Hoek-Brown rock mass strength parameters of m=2.397 and s=0.0117.

3.7.4.4 Ubiquitous Joint Modelling - Correlation of Critical Failure Direction to PCA Trend

In order to correlate the significance of the trend determined in the principal components analysis, a ubiquitous joint analysis can be performed with the linear elastic model. It is possible to calculate the normal stress and maximum shear stress developed on planes of certain orientation, and by comparison of these to one another identify the most likely to fail (Coulson, 1996; Falmagne, 2002; Diederichs et al., 2002; Connors et al., 1993; Samson-Forsythe, 1994). Here the problem has been simplified to the two dimensional transverse plane bisecting the 9370-26 stope at section 9400E perpendicular to the orebody. The model analysed here is the basis for the two dimensional non-linear finite element model using Phase² (Rocscience, 2006), discussed in the next section, but here the simplified drift model is run over multiple mining steps with application of the mine wide stresses from the three dimensional boundary element model, to provide tractions on the exterior boundary of the drift model.
(Crowder and Bawden, 2005), but run in a linear elastic mode. This ubiquitous joint
analysis could also be performed in either the Examine\textsuperscript{3D} or Map3D models, but it was felt that
the two dimensional model was sufficient for the purpose of this study.

As has been discussed earlier (Chapter 1), from laboratory testing the generation of fractures in
a brittle rock may be more related to the cohesive failure of the material than to the frictional
resistance, such that frictional resistance is not mobilized until cohesion is lost (Martin and
Chandler, 1994; Martin, 1997). For joints it is considered that this in not the case, and for open
joints, the strength is primarily governed by surface roughness and the frictional resistance
developed under normal and shear loading. Three different material models where reviewed
with the ubiquitous joint analysis, using a Mohr-Coulomb model typical values:

\begin{enumerate}
\item Cohesion = 10 MPa, Friction = 30° – equivalent to the rock mass strength and
upper most strength of a joint with strong infill material, assuming a combined
cohesion-friction model,
\item Cohesion = 0 MPa, Friction = 30° – equivalent to a joint or fracture that is pre-
existing and not healed, and
\item Cohesion = 10 MPa, Friction = 0° – Used to determine the lowest strength
possible based on purely cohesional failure, assuming the non-simultaneous
application of friction and cohesion.
\end{enumerate}

In the model, an analysis box was placed at the location of the core of the seismic cluster, and
the average normal, $\sigma_n$, and maximum shear stresses, $\tau$, in the form of joint factor of safety
(joint shear strength/ shear stress), were determined at the grid point locations for various
orientations (Figure 3.42). The three orientations in this two dimensional plane that are
important are the C-set, (inclined south at ~ 0 -15°), dominant PCA trend (inclined south at ~35
- 50°) and A-set (inclined north at ~ 65-75°). As can be seen in the case shown in Figure 3.42,
the minimum factor of safety for a joint (friction = 30°, cohesion= 0 MPa), is obtained at an
orientation of 40°.

In order to see the effect of the variation of the orientation and material strength on the factor of
safety (FOS), the average FOS at the query points (Figure 3.42), has been determined for each
model and orientation and plotted in Figure 3.43a, using the loading conditions found at 06-
2001 (after localization), although any mining step, would produce similar results. Here, it must
be recognised that the direction of the maximum induced principle stress is virtually horizontal
so makes interpretation relatively easy, (see tensor orientation Figure 3.42). From the plot the
minima of the FOS lines gives the orientation of the most likely direction of failure. For the
material model ($\Phi = 30^\circ$, $c=10$ MPa), the minimum is around $40^\circ$ (the conjugate failure orientation $-30^\circ$), while for the joint model ($\Phi = 30^\circ$, $c=0$ MPa), is around $35^\circ$ (the conjugate failure orientation $-25^\circ$), and for the cohesive model ($\Phi = 0^\circ$, $c=10$ MPa), the minimum is around $50^\circ$ (the conjugate failure orientation $-40^\circ$). The later defines the direction of maximum shear based on Morh’s circle and is close to $45^\circ$.

If the trend defined from the PCA is considered to be the overall direction of fracture propagation of intact rock, defining a macrofracture shear structure, it is evident that either of the three models could be applicable. The cohesional strength model however, best fits the overall variation in the orientation of the structure defined by the seismicity.

![Figure 3.42](image.png)

Figure 3.42 Linear elastic ubiquitous joint analysis based on the stress state at 06-2001 above the 9390 level haulage section 9400E (Stope 26) view looking West. Plots of joint or plane factor of safety (FOS) for the Mohr-Coulomb joint model of $\Phi = 30^\circ$ and $C = 0$ MPa. (a) FOS for plane oriented horizontally (Strike R/ Dip [090,00]), (b) sub horizontal, simulating the C-set [090,10], (c) sub vertical, simulating the PCA dominant orientation [090,40], also indicated are the ubiquitous joint planes oriented at $10^\circ$ and $40^\circ$ used to determine the shear and normal stress path histories and (d) sub vertical, simulating the A-set orientation [270,70].
Ubiquitous Joint Analysis FOS versus Orientation
Mohr-Coulomb Shear Analysis 9390L FW Stope 26 (9400E)

Figure 3.43 Linear Elastic Ubiquitous Joint analysis based on the stress state at 06-2001 above the 9390 level haulage section 9400E (Stope 26) plotting (a) variation of mean factor of safety (FOS) versus joint orientation, measured from the horizontal counter clockwise (CCW), for three material models of i. $\Phi = 30^\circ$ and $C = 10$ MPa, ii. $\Phi = 30^\circ$ and $C = 0$ MPa and iii. $\Phi = 0^\circ$ and $C = 10$ MPa. Note also indicated is the predominant range of the PCA trend for Slice 2. (b) Shows the shear and normal stress path histories on two joint orientations (see previous figure) from 12-1993 to 12-2004.
Although not totally conclusive, this could provide confirming evidence that even at these relatively high confinements, the direction and fracture propagation is primarily controlled by cohesional failure of the material, suggesting a non-simultaneous application of friction and cohesion in the failure criteria. The possibility that the sub-horizontal joint set, the C-set, is the primary seed for fracture rupture appears unlikely based on the mechanics present here, as these shear conditions are not optimal to create failure in this direction. Additionally, as previously stated, the non-prevalence of the C-set joints, does not make this a likely candidate in the first place. This is again confirmed by analysis of the, $\sigma_n - \tau$ stress paths of structures oriented at either $10^\circ$ or $40^\circ$ to the south, located at the core of the seismic cluster, (Figure 3.42c), which shows, a move away from shear failure conditions with increased loading for the former orientation, while a move towards shear failure, at a higher shear stress for the later (Figure 3.43b).

This is somewhat different from other studies, (Urbancic et. al., 1993; Bird 1993; Connors, et. al., 1993; Samson-Forsyth, 1994; Mercer, 1999; Falmagne, 2002) in which the authors tried to resolve the seismicity and stress condition to the most likely joint to fail, with the general idea that the seismicity is related to shear or tensile failure of the joint sets. Reyes-Montes (2004), noted that the analysis of the trend of seismicity close to an excavation in a massive, poorly jointed rock mass at the URL, fitted closely to the orientation of the tensile stress fracturing (extension fracturing) surrounding the tunnel. Here at the Williams mine it is believed that these events are related most strongly to fracturing of the intact rock to cause a macrofracture shear structure and not joint slip.

The overall rock mass can be considered to be transversely isotropic as a result of the foliation set (A-set). The influence of this on the failure of this confined footwall region is probably negligible, as the orientation of stress here, ($\sigma_1$ oriented at $70^\circ \pm 5^\circ$ to the foliation), is such that the foliation set is unlikely to fail at the core of the seismic cluster, and based on triaxial testing of a variety of anisotropic intact rocks, (Hoek and Brown, 1980; Sarglou and Tsiambaos, 2008) material with inherent micro anisotropy, would be close to it's peak strength. Also, in this two dimensional analysis, the transverse vertical B-set is not considered. However, based on the orientation of the joint set parallel to $\sigma_1$, and considering that $\sigma_2$, the clamping stress ($\sigma_n$) is constant and higher than $\sigma_3$, then like the $0^\circ$ orientation of the joint FOS analysis (Figure 3.43a), this discontinuity orientation would be highly unlikely to fail.
3.7.4.5 Non-linear Modelling of 9390 Level Footwall Zone (stope 26)

A non-linear analysis is required to investigate the post peak parameters and behaviour of the rock mass during the failure process. Previous studies, (Crowder et al., 2006), concentrated on back analysing the required parameters that would achieve reasonable agreement with measured displacements of failed rock in the near field, of an excavation. Other studies, (Hajiabdolmajid et al., 2002; Diederichs et al., 2007), concentrated on the same near field failure region, but by back analysing the required parameters or model that was required to achieve an equivalent depth of failure from visual observations. Here the author investigates the required post peak parameters that can both achieve near field displacements or similar shear behaviour as observed in the field. As will be identified, to fully evaluate this would require full three dimensional (3D) non-linear modelling, using either continuum models (FLAC\textsuperscript{3D} -Itasca 2008; ANSYS, 2008; ABAQUS – SIMULIA Inc. 2008) or discontinuous/discrete models (3DEC, PFC\textsuperscript{3D} –Itasca, 2008, ELFEN – Rocfield Software ltd., 2008). However, while evaluating this large mine model in such packages is not prohibitive, the time required for adequately meshing of such a model was considered to be outside the scope of this study.

A starting point was to simplify the area of interest to a two dimensional, (2D) section, and use Phase\textsuperscript{2}, (Rocscience, 2006), a two dimensional continuum finite element package which, when combined with application of boundary traction obtained from the full 3D linear elastic boundary element model of the overall mine geometry, (Examine\textsuperscript{3D}), was found to give insight into the potential post peak parameters required to achieve similar behaviour. This analysis is based on the procedure discussed by Crowder and Bawden (2005) and Crowder et al., (2006), and takes a 2D transverse section through the footwall drifts of the 26 stope. This area was primarily analysed as it was the last region to fail in the sill, had two complete displacement readings, (SMART 26-1 and 26-2), and was most applicable to be simplified to a relatively stable 2D model. Following on from Crowder et al., 2006, the model was extended in time from 09-2002 to 09-2004 to be able to push the model to the point of complete failure, and be able to include analysis of an additional instrument, SMART 26-2, placed later in the mine sequence.

The 2D section that was analysed in this study is shown in Figure 3.44a inset. It should be noted that, in order to obtain model stability, only two of the haulages, 9390 and 9370 levels could be included in the model. The addition of the 9415 level, which would make this section more accurate, could not be included due to it’s proximity to the boundary, used for traction application from the linear elastic model. This boundary could not be pushed out any further, due to interaction with the boundary element surface mesh of the mining, for which application of zero tractions, internal to the stopes, causes numerical instability in the non-linear model.
Figure 3.44  
(a) Detailed view and boundary conditions, of the pseudo hybrid non-linear model of 9390L haulage at #26 stope, transverse section 9400E, showing also the location of the query line. (b) Comparison of modelled relative vertical displacements along query line with measured instrument response for SMART 26-1 (12-1999 to 09-2002), and (c) comparison of modelled relative vertical displacements along query line with measured instrument response for SMART 26-2 (07-2001 to 05-2003), the model is multistaged, using H-B brittle-plastic (Table 3.7) and dilation parameter = 0.8. Note dashed line shows displacements of the last reading and occurrence of deep dilation.
This model is therefore an approximation to the relative stress levels and geometry, and cannot be used as an exact replication, but as an approximation.

As previously stated, the generalised Hoek-Brown (H-B) rock mass failure criteria, tends to under predict the depth and amount of failure for linear elastic modelling. However, it was found from back analysis that, when used for non-linear modelling in which stress shedding of failed regions results in elevation of stresses outside of the failed zones, the basic criteria based on the rock mass characterization (GSI), can be used successfully with the application of brittle-plastic post peak parameters, to achieve similar depth of failure and displacements in the near field (1 to 4 m above the drift back). If the peak strength failure criteria is set to the damage limits, determined from the H-B brittle failure parameters, (m=0, s=0.09 to 0.25), it is not possible to achieve model stability, as too much failure occurs and this is only intended for linear elastic modelling. This has also been noted by (Diederichs et. al., 2007).

From a parametric study, the post peak parameters (although not unique) found to match depth of failure and displacements, when applying the H-B failure criteria are summarised in Table 3.7, and the results of comparison to instruments are shown in Figure 3.44b & c. A close fit was achieved between modelled and measured results prior to deep dilation, and in the case of the second instrument (Figure 3.44c), the modeled displacements for this sequence are in excellent agreement. The displacements are not absolute but relative to the toe of the instrument and over the monitoring period. This match can only be achieved by developing the stress history and failure over time using brittle-plastic post peak parameters, [i.e. there is a reduction in strength of the material once the failure criteria has been reached, to the residual strength and then following a non-associated flow rule along the residual envelope]. A perfectly plastic model does not develop the necessary displacement and depth of failure. The residual strength was found through a parametric fit and taken at face value, would suggest that there is a cohesive strength of the residual material at zero confinement. This is an apparent cohesion required to fit to the stress conditions. In setting this residual to go through the origin (sr = 0), generally resulted in numerical instability, or an inability to fit the correct displacements through increases in mr. The detailed findings of a similar parametric study are discussed in Crowder et. al., 2006, but generally, an increase in mr results in a reduction of the depth of failure and displacements, likewise to a less significant extent an increase in sr results in reduced displacements and depth of failure. The additional parameter that greatly controls the amount of displacement and curvature of displacement versus depth is the dilation parameter. In general an increase in the dilation parameter, results in an increase in the confinement on
reaching the residual strength and an increase in displacements, with a slight reduction in the depth of failure.

A point regarding dilation is that in this continuum model, as with all continuum models, real dilation of the failed material, is not actually exhibited, and thus is simulated by applying volume change after failure, in the flow rule, through application of a dilation parameter. For the H-B model this is based on a substituting \( m_{dil} (Dil) \) for \( m_b \) in the non-associated flow rule following Vermeer and de Borst (1984). It has been suggested that \( Dil \) should range from 0 to 0.6 of \( m_b \) (Rocscience, 2006). Here the best fit for displacements, 3.5 m up into the failed back, was found to range from 0.8 to 1.2 (0.33\( m_b \) to 0.5\( m_b \)).

The modelled failure above the immediate back of the drift occurs through a combination of tensile and shear failure (Figure 3.44a). Tensile slabbing occurs close to the excavation back and transitioning to shear failure in the apex of the failed region. A similar response has been modelled by Diederichs et al. (2007), for spalling failure of tunnels, in which application of brittle-plastic parameters resulted in appropriate depths of failure around a tunnel. Also, Hajiabdolmajid et al. (2002), could achieve similar depths of failure using a confinement weakening-friction strengthening model, but in neither cases did the authors attempt to assess the appropriate displacements. Additionally, both of these studies were directed at stress conditions in which the confining stress was significantly lower (5 - 10 MPa), and the depth of failure (\( d_f \)) (i.e. distance from the original surface to the edge of the unsupported failed zone) of only 0.5 m for a 1.75 m radius (a) circular excavation correlating to a \( d_f/a \) ratio of 0.3. While here the confinement in the immediate back (1 to 2 m) is significantly higher at 10 – 20 MPa with the drift failure region having a \( d_f/a \) ratio of 1.4 (\( d_f = 3.5 \) m from instrumentation on 9390 level and assuming similar depth of failure through caving as identified in the 9415 level above and \( a = 2.5 \) m).

Further away from the immediate back, it was found that shear failure predominates and develops with clearly defined trends (Figure 3.45a to d). For the perfectly plastic model, by definition, there is no volume change and thus \( Dil = 0 \), then no shear development away from the opening occurs, primarily due to the lack of stress shedding. The induced stress field does not change and thus the stress contours are similar to the linear elastic analysis. For the condition using H-B parameters that best fits the immediate displacements in the back, (Figure 3.45b), it is can be seen that shear develops away from the drifts, in the lower south and upper north corners, along a trend of 20° to 35°. If the dilation parameter is lowered to \( Dil=0.6 \) or 0.25\( m_b \) (Figure 3.45c), there is a substantial increase in the shear development. Generally,
reducing the dilation parameter reduces the application of additional incremental confinement, and allows for greater shear failure until the model becomes unstable (Dil <= 0.2). Again this shear develops but at a more consistent trend of 35° and is similar to what was seen in the seismicity above the 9415 level drift early on (Figure 3.45e).

One of the issues with applying this constant dilation flow rule is that it does not allow for variation of dilation, dependent on different confinement regions. Cundall et. al., (2003) has proposed a flow rule with dilation dependent on plastic strain and confinement, although it has been suggested to only apply to low to moderate confinements. It is quite evident that in order to model the failure in the core of the pillar between the drifts, for a continuum model the dilation would have to vary and reduce with confinement. However, this is only one part of the problem. The other significant modelling limitation with the dilation parameter (Dil), is that a volumetric dilation is applied. In reality dilation in low confinement regions close to the opening, is unidirectional towards the excavation surface, termed bulking as identified by Kaiser et. al., (2000). Considering the core of the macrofracture structure or pillar between the haulages, if dilation is occurring as identified by the instrumentation, then this could be also towards the openings in the direction of the minimum principal stress. It would have to be considered that these deformations are not just occurring to the drifts immediately below and above, but also into the mined and filled stopes below the 9370 level. In Figure 3.15 similar macrofracture shear structures are visually seen to develop between the 9390 and 9370 levels and between the 9370L and the back of the 24 stope. It is important to recognise the limitations of this continuum modelling, not only are we empirically determining the post peak parameters, but also greater investigation is required to determine the correct application of dilation.

Changing the model from H-B to the equivalent Mohr-Coulomb parameters (Table 3.7, Figure 3.45d), but using the same equivalent dilation parameter of 0.8 (in this case 0.33Φb = 11.1°), which should be compared to Figure 3.45b, again creates significant shear failure to be developed from the drift, with a steepening in the trend to ~40°. The key change here is the increase in the tensile strength to 5 MPa, over 1.5 MPa from the H-B rock mass model. Diederichs (2007) has also noted differences from both, suggesting the stark separation between the tensile yield function and the shear function creates slightly different but equally plausible responses. The other change between the models is a slight reduction in the strength in order to fit the linear relationship to the parabolic function of H-B (Figure 3.45f).
Figure 3.45 Contour plots of the maximum principal stress, $\sigma_1$, at mining step 12-2002, showing the comparative development of element shear (failed elements) for four different non-linear modelling parameters, (a) perfectly plastic post peak behaviour based on H-B peak rock mass parameters, $\text{Dil}=0$, (b) H-B brittle-plastic post peak behaviour using the best fit post peak parameters (Table 3.7), $\text{Dil}=0.8$, (c) same model but with $\text{Dil}=0.6$, and (d) Equivalent Mohr-Coulomb, peak and post peak parameters, $\text{Dil}=11.1^\circ$ equivalent to 0.8 for the H-B. (e) Comparison of seismic development, around footwall drift in front of #26 stope, seismic analysis slice 1, and (f) Stress path above 9390 L drift for equivalent M-C model showing peak and residual strength lines.
It was found that by increasing the tensile strength, a reduction and flattening in the development of shear failure results. Andersson et al., (2007), in their 2D non-linear modelling of a pillar failure, under low confinement using Phase², also noted that the equivalent M-C brittle-plastic model parameters showed improved and a better fit to the observed failure over the H-B modelling parameters. This is largely based on greater control of the tensile cut-off. It appears that the tensile strength which may govern the start of formation of the shear zones, has a more important role than anticipated in continuum modelling.

For all of the models, the majority of shear failure that develops (although not at its peak extent) occurs at the stress condition of 06-2001, and extends slightly for further mining steps, causing a general reduction in the amount of confinement at the core of the pillar and increase in stress. As can be seen in Figure 3.46, at the core of the pillar confinement, \(\sigma_3\), appears to be increased over the surrounding rock mass and is relatively high at greater than 35 MPa. The average stress path at the core of this region is plotted with time for the M-C model, as well as the stress path for the immediate back, (1 to 3 m above), in Figure 3.33. As can be seen the immediate back fails early on and follows the residual strength curve but the core region has significant confinement added as a result of stress shedding and this path is well below the Brittle-Ductile transition line suggesting plastic behaviour should occur. This could be the result of simplifying the model to 2D, but does suggest that this transition line should be investigated further for the rock mass, as this region did fail in a brittle response.

It was not possible to produce failure initiating between the drifts, this being a result of the model becoming unstable through failure propagating to the boundaries (place of applied tractions) for subsequent mining steps into 2004. If the model peak strength is lowered, to achieve failure earlier on, instability again becomes an issue. Although failure of the core region could not be modelled, this modelling does suggest that similar trends in shear failure as identified from the PCA analysis, can be achieved by application of post peak brittle-plastic parameters, resulting in strain-softening behaviour and not plastic ductile behaviour.
Figure 3.46  Contour plots of the minimum principal stress, $\sigma_3$, at mining step 12-2002, for the four different non-linear modelling parameters (see previous Figure and Table 3.7). Note the development of high confinements (> 35 MPa) at the core of the pillars between the footwall haulages, in the region where seismicity occurs.

In order to achieve greater stability of the model at the boundaries of the applied tractions, it would be necessary to move to a full three dimensional non-linear model which was outside the scope of this research. As discussed earlier however, if continuum modelling is performed then in order to properly evaluate the effect of dilation, a variable dilation parameter, based on confinement should be implemented with consideration unidirectional deformations. This emphasises the potential benefit of researching this behaviour using discontinuous/ discrete element models where development of complex constitutive models in not required. These are, to some extent, still in their infancy, and application to such a large model has yet to be performed to the authors knowledge, but it is suggested that the starting place may be to use a hybrid method.
3.8 Conclusions

Analysis of the temporal changes in the spatial trend of the seismicity using the principal components method combined with an estimate of the clustering density, appears to show strong potential for characterizing the state of the regional rock mass as it progresses through stages of failure. It is important to recognize that the type of regional rock mass failure that this method was applied to in this study, is developed under conditions of progressive, almost monotonic loading of principal stresses, (i.e. failure of a confined rock mass, without caving). In stress failures also involving caving, it is expected that tensile spalling failures would be at play, and may complicate the trend of the seismicity.

- From the seismic events density coalescence of events, or fractures, was identified to occur at a density of 5 events per 5 x 5 x 5 m voxel. This event density was determined to be a reasonable estimate of the critical clustering density, and at this event density the clustering index, (Cli), is expected to be > 0.5, indicating that events or fractures are at least interacting, and are probably coalescing. The clustering index, being based on a mean source radius of 1.85 m and standard deviation of 0.3 m, [identified at the neighbouring Golden Giant mine], is postulated as the point of true yield, and when achieved, correlates to observed stable dominant trends determined from the principal component analysis of event clusters. Shortly after this point is reached, almost simultaneously, there was a significant increase in the average ellipticity of the PCA derived planes over a number of temporal windows from a background level of 5 to > 15. This point is postulated as the localization of fractures to form a macrofracture shear structure developed though en echelon fractures of intact rock. Based on an analogy to the triaxial testing of laboratory specimens, the point of formation of this localized macrofracture structure, is postulated to represent the peak or near peak strength of the rock mass. After this point, it is thought, that the rock mass is in the post peak strength state, and continues to progressively fail, gradually strain softening up until the point of disassociation, when more significant strain softening and dilation of the rock mass occurs.

- The PCA technique identified that the dominant trend of the observed group behaviour was stable and oriented with an average strike and dip of [090°, 40°] following coalescence, up until the point of disassociation, when the trend becomes unstable. From a linear elastic ubiquitous joint analysis, this indicates that the macrofracture orientation is developed in the direction of maximum shear, and that distribution in
orientation of the PCA derived planes is best represented based on a cohesional strength material model. Additionally, this ubiquitous joint analysis indicates that the most likely fracture direction to fail under shear is related to the dominant PCA trend and that the sparsely populated sub-horizontal joint set, the C-set, would be unlikely fail compared to this orientation.

- The ‘point of disassociation’, that has been recognised from the change in the relatively stable dominant PCA trend \([090^\circ, 40^\circ]\), was found in the central region of this failure (Slice 2) to switch to a potential conjugate shear structure oriented \([298^\circ, 30^\circ-60^\circ]\) and becomes unstable after this point. It is inferred that the potential formation of this conjugate structure allows for greater degrees of freedom of rock blocks resulting in dilation and partial aseismicity. This point of disassociation was recognised to occur more subtly in the other analysis slices. However, from investigation of the \textit{in situ} displacement monitoring instrumentation, located in the haulage excavation below the formation of this postulated macrofracture shear structure, the point of disassociation was found in three of the analysis slices to occur following significant, sudden dilation of 10 to >50 mm at a measured depth of 7.5 to 9 m. This displacement response indicates that a different dilation regime was initiated over the ‘bulking and spalling zone’ contained within ~ 3 – 4 m of the excavation back. These significant displacements were observed to occur at the up dip edge of the macrofracture structure. It has been postulated that this change in the group behaviour, (the point of disassociation), is related to this significant dilation or shearing of the rock mass, causing patches of the rock mass to become ‘aseismic’, and change the spatial correlation of the events to one another. It is at this point, prior to complete aseismicity, that significant portions of the rock mass are thought to be close to or approaching the residual strength, making an analogy to observations of laboratory triaxial testing. These dilations infer that strain softening behaviour of the rock mass has occurred, and based on observation from subsequent mining, indicates that the stress level has reduced to at least below the initiation stress level. This is inferred as mining behind this failed footwall region progressed easily with no stress effects in drill holes, raises or subsequent development being noted post disassociation. It is postulated that gradual dilation is probably occurring at the core of the microseismic clusters following localization, (gradual strain softening associated to this), but is controlled by confinement in the core, and continues up until the point of disassociation, when the dilation appears to become more significant, as measured at the edge of the macrofracture structure.
• In contrast to this, in regions that were determined to be already ‘failed’ due to no observable stable PCA trend, and reduced levels of microseismicity (analysis slice 5 and 6), the field displacement instrumentation, placed probably subsequently to failure, recorded no deformations in the back of the haulage and appear to indicate that these regions were already strain softened (relaxed) and at their residual strength.

• Analysis of the microseismic clusters using the PCA technique indicates that the group behaviour of events/fractures seems to be sensitive to distant and local changes in the rock mass strength, as it progresses through failure. This was often noted with a drop in the number of PCA derived planes with ellipticity > 2.5, which often occurred following small jumps in dilation in the immediate back of the 9390 level haulage. The dominant PCA trend can be affected also by large magnitude events or failure of other regions identified in this sill pillar study, for short periods following regional rock mass adjustment. These changes should be recognised as not the point of disassociation (residual strength of the rock mass), if the dominant trend re-establishes quickly, (within one 50 event temporal window), following this readjustment phase.

• In most mine modelling in the past it has been assumed that the development generally has little impact on the linear elastic stresses developed in the ore zone, the place of most interest to mining. However, here the lateral development of the footwall haulage drifts has a large impact on the stress concentrations, especially as there is little difference in the material properties of the footwall and the ore zone. The footwall haulages acting as the first major stress riser, where in between these the majority of the microseismicity and postulated shear fracturing is taking place. The geometry here controls the stress development and the location of and lateral trend of the seismicity, helping to fix the strike identified from the principal components analysis.

• Large magnitude events appear to occur during a flurry of activity and seem to be a reaction to the ongoing fracturing. The large events are either caused through a redistribution of stress in the footwall onto a larger structure south of the haulages based on observed damage, but could potentially be the result of shear failure occurring on a system of aligned and networked fractures associated to the core regions of clustering seismicity (regions of high seismic density located just south of the haulages). The flurries of activity do appear to affect the trend of the principle components analysis, and are the result of weakening of the rock mass and the increase in fracture propagation and initiation. The majority of the large events associated with flurries of
activity occurred in the pre-peak to peak strength of the rock mass based on the strength state characterised by the PCA and seismic density methods. The exception to this is the 2.7 Mn pillar burst, which is in a different class compared to the other large events, and the last 3.5 Mn event associated with the last flurry of activity in S2, the last region to dilate and fail. This last event could have potentially been the failure of an orphaned region of non-degrading rock mass.

- The three dimensional linear elastic modelling is useful in determining the stress path associated to the failure of this footwall region, and with calibration of stress levels associated to damage limits, further acts as a useful first approximation to estimate regions of the rock mass that are yielding and failing. Based on the Hoek-Brown brittle failure parameters, initiation, coalescence and localization of these confined regions were identified to occur at equivalent stress levels, at a deviatoric stress term \(\sigma_1 - \sigma_3\) of around \(0.3\sigma_c\) (i.e. \(m=0, s=0.09\)), and the point of disassociation, (failure of patches of the rock mass to close to the residual strength), was determined to occur at a slightly higher damage level of around \(0.375\sigma_c\) (i.e. \(m=0, s=0.14\)).

- This study and the complementary study at Golden Giant mine (Chapter 4), suggests that the rock mass eventually failed in a strain-softening brittle fashion to cause stress shedding and stress shadowing, to a stress below the initiation stress as indicated by the mining without stress induced damage. This is contrary to the Mogi transition state determined for crystalline intact rock, based on levels of linear elastic modelled stresses, which would indicate that ductile behaviour (plastic deformation) should dominate. Hoek and Brown (1980) identified that this transition zone did not hold for samples that had planes of weakness or had been previously broken, and in the absence of published guidelines a transition zone of \(\sigma_1/\sigma_3 = 2\) rather than 3.4 has been suggested more recently by Hoek (2004). Although this behaviour in the form of the formation of a shear band is expected for confined and ductile material the observed behaviour appears to be strain-softening. It is suggested that either this is not a distinct classification and a broader transitional zone exists for the rock mass or that there are other factors at play that allow for the eventual failure to propagate to openings causing the significant strain-softening behaviour. It should also be noted that this modelling is based on linear elastic modelling which does not allow for local failure and stress shedding which would be expected to cause a reduction in the confinement and an potential increase in the induced principal stress. More research into the influence of discontinuities on the effect the transitional behaviour from brittle to ductile should be
performed to confirm Hoek-Brown’s (1980) finding’s where this lower limit is potentially more applicable to the rock mass or disturbed laboratory samples. Use of polyaxial testing with the measurement of acoustic emissions (Young, 2007) should help to advance this research.

- Based on the simplified 2D non-linear finite element continuum modelling, it was determined that the only way to achieve equivalent near field displacements and depth of failure as observed in the conventional instrumentation would be through the application of a brittle-plastic dilatant model, simulating strain softening behaviour. In this modeling it was not possible, partially due to oversimplification and also due to model stability issues, to create full failure in between the haulage drifts. It was found, however, that the dilation parameter, as well as the modelling criterion applied, (Generalized Hoek-Brown [GBH] versus Mohr-Coulomb [MC]), can have a large effect on the extent and orientation of similar shear development to that observed from the microseismic analysis. Decreasing the amount of dilation increases and steepens the shear development when using the GHB brittle-plastic parameters. Additionally, if the criterion is changed to the equivalent MC brittle-plastic parameters, the shear development increases and steepens to 40° similar to that identified in the field. By contrast if perfectly plastic non-dilatant (or dilatant) post peak parameters are applied there is no shear development and no decrease in stress. This modelling suggests that similar trends in shear failure identified from the PCA method could only be achieved by application of a brittle-plastic dilatant post peak parameters.

- One of the issues with applying this constant dilation flow rule is that it does not allow for variation of dilation, dependent on different confinement regions. Cundall et. al., (2003) has proposed a flow rule with dilation dependent on plastic strain and confinement, although it has been suggested to only apply to low to moderate confinements. It is quite evident that in order to model the failure in the core of the pillar between the drifts, for a continuum model the dilation would have to vary and reduce with confinement. However, this is only one part of the problem. The other significant modelling limitation with the dilation parameter (Dil), is that a volumetric dilation is applied. In reality dilation in low confinement regions close to the opening is unidirectional towards the excavation surface, termed bulking as identified by Kaiser et. al., (2000). It is important to recognise the limitations of this continuum modelling, not only are we empirically determining the post peak parameters, but also greater investigation is required to determine the correct application of dilation.
• With regards to the bi-linear failure curve, it was identified that the observed spall limits, determined for low confinement near field displacements, are only applicable to linear elastic modelling calibration or very low confinement, as it was found from non-linear modelling that displacements could only be approximated with a brittle-plastic model, based on the GHB (or equivalent MC) peak rock mass strength failure curve. Application of a reduced peak strength envelope based on the m=0 approach, results in instability of the model even at low stresses, and cannot simulate the post peak correctly, and would require full implementation of the cohesion-weakening friction-strengthening (CWFS) constitutive model (Hajiabdolmajid, 2001) to evaluate properly. These limitations in the spall limits for non-linear modelling have also been recognised by Diederichs et. al., (2008).
CHAPTER 4

APPLICATION OF THE PCA TECHNIQUE TO DETERMINE FAILURE STATES OF A CONFINED ROCK MASS - A CASE STUDY AT THE GOLDEN GIANT MINE, CANADA.

4.1 Introduction

In previous research (Chapter 3) it has been found that at the rock mass scale for an almost monotonically loaded region under moderate confinement, the key stress points identified from laboratory testing ($\sigma_{ci}$, $\sigma_{cd}$, $\sigma_f$, and $\sigma_r$ residual strength) can also be interpreted based on microseismicity and using an analogy to laboratory AE. This was accomplished using a combination of the event clustering density and the principal components analysis to observe temporal changes in the formation of microseismic events at various stages of failure.

In Chapter 3 it was found that with increased confinement further away from the opening, these coalescing events can form a macrofracture structure through localization of failure, similar to observations in the laboratory (Lockner et. al., 1992). In any rock masses this is the point of true yield and is probably close to or at the peak strength. Following this point with increased damage accumulation, significant strain-softening, (inferred through observation of reduced stress damage to openings behind the region of failure), and deep dilation or shearing, (measured through displacements of in situ instrumentation), of the rock mass occurs at the point of disassociation, causing patches of the rock mass to become 'aseismic', and change the spatial correlation of the events to one another. It is at this point prior to complete aseismicity, that significant portions of the rock mass are thought to be approaching the residual strength.

The previous study that followed the complete failure process of the rock mass at the Williams mine (Chapter 3) was performed using microseismic data based primarily on event locations only. In this case study, (Chapters 4 and 5), at the Golden Giant mine, an improved microseismic system, with greater location accuracy and calculation of source parameters, is
used to observe temporal changes in the evolution of the microseismicity recorded during complete failure, (initiation to aseismic behaviour), of a confined region of the rock mass under similar monotonic loading conditions. The emphasis is to determine if it is possible to identify different stages of failure from pre-peak to the post peak, using microseismicity, and to better understand the behaviour of the rock mass as it transitions through failure, using seismic source parameters to give an insight into the fracture mechanisms. The research direction is to be able to produce an improved constitutive model of the failure process and required parameters to model this behaviour using continuum or discontinuous/discrete modelling. However, in order to calibrate models it is first necessary to identify when the rock mass is entering the post peak and improve our understanding of the behaviour.

Here, a specific region of a large pendent pillar has been identified, (Figure 4.1), that can be classified as a stress driven failure of the rock mass while under moderate to high confinement (40 to 50 MPa), which goes through full stages of failure from initiation to aseismic behaviour. Once this state was reached the rock mass was considered to be completely failed, (at it’s residual strength), and it was found that mining could resume without detrimental stress damage to openings, and crushing of boreholes that had previously occurred, indicating that strain softening behaviour to a stress level below the initiation stress had resulted. This was inferred as similar behaviour to the complimentary study at the Williams mine (Chapter 3). In this chapter, three dimensional linear elastic modelling, with determination of damage limits, is performed to determine the stress path history and compared to the Williams case study.

The key objective in this case study is to use the principal components analysis technique, and the determination of the event clustering density, to define the strength state of the rock mass based on the microseismic activity to give a reasonable indication of initiation, interaction, coalescence (yield), localization (peak) and disassociation (post-peak approaching the residual strength) states of the rock mass during failure. From this failure state characterization the temporal variation of key source parameters, such as the magnitude (seismic moment), energy, shear wave to compressional wave energy ratio (Es/Ep), source radius, apparent stress, dynamic and static stress drops, the ratio of these defining the events geometrical source complexity, are performed to improve our understanding and shed light on the behaviour of the fracturing process leading to rock mass degradation. Additionally, in this Chapter changes in the seismic scaling relationship during the failure process are investigated as well as temporal variation in the b-values (Gutenberg and Richter, 1944; Young and Collins, 2001), and discussed later in the Chapter.
Figure 4.1 Longitudinal 3D view of the Hemlo mines, and solid model of mining geometry. (A) Golden Giant Mine Shaft Pillar Region, (B) Williams Mine Sill Pillar Region (Chapter 3). Shades, represent yearly mining up until September 2005. Inset geographical location of Hemlo Camp in Ontario, Canada.

As large amounts of data are recorded in the failure analyzed in this case study, the source locations and source parameters are based on automatic waveform picks and online calculations, with minimal user manipulation, with the objective of being able to determine if temporal changes can be observed from a well-configured microseismic system in real-time on run of mine data. The automatic locations and source parameters are compared to a selected representative dataset, used for focal mechanism studies, (Chapter 5), on which manual picking of p- and s- wave arrivals was performed, to observe the difference between manual and automatic processing.

4.2 Golden Giant Mine

The Hemlo camp is located in Northern Ontario, on the North shore of Lake Superior between the towns of Marathon and Manitouwadge and is composed of three mines; Williams Mine, Golden Giant Mine and David Bell Mine (Figure 4.1). As discussed in Chapter 3, each operation is mining the same gold rich tabular, steeply dipping orebody, using generally sub-level open stope mining for the thicker ore (7 to 40 m). Historical backfilling of stopes used
cemented rock fill for primary stopes and sand-fill for secondary stopes, but each operation has progressed to cemented paste backfill. The Hemlo camp was chosen for this study, as two of the mines, the Williams and Golden Giant mine, were mature mines with relatively high extraction ratios and as a result relatively high mining induced stress that had caused a number of ground control issues, resulting in stress driven failure, (Bawden & Jones 2002, Nickson et al. 1998).

Both mines had microseismic systems with a number of years of data collection at the commencement of the study, (2002), and also had extensive displacement monitoring of important infrastructure. This chapter concentrates on the extraction of a portion of the Golden Giant shaft pillar located at a depth between 631 m (4690L) to 812 m (4500L) below surface, with particular attention to the east side of the shaft destress slot, (Bawden, et al., 2000b), between the 4620 and 4600 levels (Figure 4.2).

4.2.1 Shaft Pillar Mining History Overview

Early on in the life of the Golden Giant mine the shaft pillar region was identified as a critical region for rock mass monitoring, as the shaft was sunk in the footwall to within 9 m of the ore zone, and only 30 m to the south of neighbouring active David Bell mining which commenced in 1990 (Figure 4.2). The first 32 channel ‘Queens’ microseismic system was installed in 1989 to monitor the region, and an early indication of rock mass instability occurred through caving of the back of the Q1/Q2 pillar above, the 4500 level in January 1992, with a brief flurry of seismic activity that rose, above and at the time of the cave, to around the 4533 level (Grant et al., 1993). The decision was taken to maintain the Q1/Q2 pillar until much later in the mine life, with close monitoring, to aid in stabilization of the 4600 level shaft infrastructure. Through regional increases in stress related to mining outside of the shaft pillar area, stress effects were noticed through stress spalling in October 1995 on the east and west shaft walls above the 4600 level, and on local lateral development on the 4600 level itself. The first minor increase in microseismic activity experienced at this time, was associated with the Q1/Q2 pillar overcut, the south east side of the shaft station and the east abutment of the ore zone adjacent to David Bell, all on the highly stressed 4600 level (Nickson et al., 1998, Appendix E). This led to a significant upgrade in the shaft support above and below the 4600 level, with the addition of shaft wall displacement monitoring of the most critical areas.
The region remained relatively quiet until October 1998 when a small strain burst on a potentially brittle structure occurred on the nose of the Q1/Q2 cross cut pillar on the 4600 level. There was no significant increase in seismicity and no forewarning for this event. The decision was taken to continue to upgrade the support on the main haulage and close off and paste backfill the Q1/Q2 overcut that had continued to deteriorate due to progressive stress spalling. The next 3 years were relatively quiet in the region and the shaft walls had suffered little to minimal dilation, (< 5 mm) since 1995 (Bawden, 2001). However, due to production constraints, which were recognized in 1998, it was found to be necessary to mine the ore around the shaft while still maintaining access. This led to an extensive study by the Noranda Technology Centre, Bawden Engineering, Rocscience Inc., Queens University, Knight Piesold, and Golden Giant Engineering staff to investigate a mining plan that would successfully accomplish this (Newmont Canada, 2001).

The distress slot, (Figure 4.2) was designed to be able to achieve extraction of the shaft pillar ore while still being able to fully utilize the shaft and alleviate the anticipated issues of increased stress loading from mining outside of the region (Coulson, 1998; Bawden et. al.,
2000b; Newmont Canada, 2001). The basic concept was to mine three stopes approximately 14 m wide and full ore thickness, (12 m) up to within 9 m of the shaft and then extend the slot over a 4 m thickness into the waste rock away from the shaft at an angle of 66°, using a pyramid sequence and electronic blasting. Backfilling of the stopes was accomplished with a special tailings paste and ‘pearl’, (expanded ¼" polystyrene balls) backfill, to create a low modulus fill for controlled closure in the thin slot. As part of the shaft pillar extraction plan the microseismic system was upgraded to a dense array around the shaft, and the shaft instrumentation was upgraded to perform displacement and stress monitoring at various elevations in the shaft above 4600 level, (Newmont Canada, 2001).

The destress slot was started in April 2002 and successful mining of the slot and shaft pillar were completed in Sept. 2005, after which the mine closed in December 2005. Of interest is that, unlike the adjacent Williams mine, (Chapter 3), even though the stress levels were as high, no large magnitude events, (>0 Mn), occurred during the extraction. Presented here is an analysis of one regional rock mass failure, associated with the east side of the destress slot, above the back of mined stopes, which forms a pendant pillar.

4.2.2 Geomechanical Properties and Mine Model

As discussed in more detail in Chapter 3 and summarized here, the main geological components that make up the deposit tend to be similar in nature and geomechanical properties across the domains of the hanging wall (HW) metasediments, ore zone (OZ) alteration zone and footwall (FW) metavolcanic feldspathic schists, characterised mainly by foliation created through metamorphism. The gold enriched orebody, which follows this foliation dips here at approximately 60° - 65° to the North and in the shaft pillar region varies in thickness from 20 m around the Q1/Q2 to 12 m in the S-stopes of the destress slot. No major faults have been observed in the Hemlo camp. Other major structural elements that do however exist, are the occasional foliation shears 10 to 30 m long and a few centimeters thick, (which are not pervasive in the region of the shaft pillar), and Diabase dykes which are vertical and trend perpendicular to the ore body. Two of these dykes bound limits of the shaft pillar, one on the far western stope boundary of the Q1/Q2 pillar which is approximately 5-10 m thick, and the second is less vertical dipping 75° to the East with a thickness of 1-2 m and intersects the David Bell boundary at the 4620 level (Figure 4.2). These dykes, although brittle in nature and sometimes structurally problematic for stope end walls, do not appear to be major contributors to microseismicity.
The four prominent joint sets that are pervasive through each domain, (HW, OZ and FW) are displayed in Figure 4.3 and summarised in Table 4.1. This data set is based on mapping at the adjacent David Bell mine (Leduc, 1991), and mapping east of the shaft (Kazakidis, 1990). It is interesting to note that west of the Q9 stope the orebody rotates to strike directly East-West and steepens to 70°, the main trend for the Williams mine. The prominent joint set as previously mentioned is the foliation set, joint set A, dipping north 60°, and striking approximately east-west. The other major joint set is the steeply dipping (~90°) B-set striking North-South. A sub-horizontal joint set C is present, however it tends to be more sparsely distributed than the other two sets even when vertical mapping exists, as identified by borehole camera surveys, (Coulson et al., 1995), in the region on either side of the shaft (Figure 4.3b). There are also sporadic occurrences of a sub-vertical D-set joints striking North-South and dipping at 45°. This set may potentially be small shears as often striations are noted on the joint surface, however, this set is not present in any major density in this region.

At the Hemlo camp there is only a minor variation in the bulk rock mass properties and quality between each domain (Chapter 3), and from point load testing and velocity surveys, (Kazakidis, 1990), the material can be classified as transversely isotropic. Loading from mining induced stress is preferentially oriented perpendicular to the foliation, and maximum axial strength is considered more representative (Hoek and Brown, 1980). From intact testing, the uniaxial compressive strength \( \sigma_c \) varies from 130 to 200 MPa, with a mean strength of 175 MPa, Young’s Modulus \( E \) of 55 GPa and Poisson ratio of 0.278 for the footwall and ore zone rocks at Golden Giant mine. For this study the mean GSI, (Hoek & Brown, 1997), of the ore zone, found from field mapping, was determined to be \( \sim 60 \), and the rock mass parameters based on the generalized Hoek-Brown failure criterion, for an \( m_i = 10 \) (schist) are: \( m_b = 2.34 \), \( s = 0.0117 \), \( a = 0.503 \) with a rock mass modulus of 17.8 GPa, (the equivalent rock mass Mohr-Coulomb parameters, are \( \Phi_{mb} = 33.5^\circ \), \( C_m = 14 \) MPa, \( T = 10 \) MPa). From back analysis of near excavation failure at the adjacent Williams mine and here (Crowder et al., 2006; Chapter 3), the best post-peak residual parameters to fit this peak failure envelope and field instrumentation were found to be \( m_r = 0.1 m_b \), \( s_r = s \), dilation = \( \frac{1}{3} \) to \( \frac{1}{2} m_b \) or in terms of the equivalent Mohr-Coulomb criteria, \( \Phi_r = 15.2^\circ \), \( C_r = 7 \) MPa, dilation = \( 16.7^\circ \), which fitted the shear failure (Chapter 3) more accurately indicating brittle strain-softening behaviour.

The full three dimensional mine model (Chapter 3), incorporating all three mines, (Figure 4.1), was again used for visual identification of microseismic events in relation to mining and for linear elastic boundary element stress modelling, (Map3D© and Examine3D), and 2D non-linear hybrid finite element modelling, (Phase2), investigated in Chapter 5.
Figure 4.3  (a) Stereonet (Dips©) of main joint sets found at the Golden Giant mine in the shaft region and east of the shaft, and typical thought the Hemlo Camp (after Leduc, 1991). (b) Mapping from four sub-vertical borehole (length 20-30m) camera surveys, of the east and west walls of the shaft. Note that the B'-set here is representative more of extensile fracturing of the shaft walls and the distinct absence of C-sets (Coulson et al., 1995).

Table 4.1  Main Joint Sets and Properties (after Kazakidis, 1990 and Coulson, 2004)

<table>
<thead>
<tr>
<th>Joint Set (Mean Orientations)</th>
<th>Strike (rhr)</th>
<th>Dip</th>
<th>Jr</th>
<th>Ja</th>
<th>Spacing (m)</th>
<th>Persistence (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A – Foliation Joints</td>
<td>262</td>
<td>60</td>
<td>1.0</td>
<td>1-1.5</td>
<td>0.125 – 1</td>
<td>3-10</td>
</tr>
<tr>
<td>B – Vertical Joints</td>
<td>344</td>
<td>86</td>
<td>1.0</td>
<td>0.75-1</td>
<td>0.5 – 2</td>
<td>1-5</td>
</tr>
<tr>
<td>C – Sub-Horizontal Joints</td>
<td>085</td>
<td>25</td>
<td>1.0</td>
<td>1.0-2.0</td>
<td>0.5 – 3</td>
<td>1-5</td>
</tr>
<tr>
<td>D- Sub-vertical “shear”/ joint</td>
<td>354</td>
<td>51</td>
<td>0.5-1</td>
<td>0.75-1</td>
<td>2-10</td>
<td>3-10</td>
</tr>
<tr>
<td>E- Sub-vertical “shear”/ joint</td>
<td>181</td>
<td>45</td>
<td>---</td>
<td>---</td>
<td>2-10</td>
<td>---</td>
</tr>
<tr>
<td>F- Vertical Joint Set</td>
<td>045</td>
<td>89</td>
<td>---</td>
<td>---</td>
<td>0.5-1</td>
<td>---</td>
</tr>
</tbody>
</table>

1 Persistence is estimated from observations of back and hangingwall failures at Williams, Golden Giant and David Bell Mines by the Author.
The model has historical stope extraction dates assigned to each 3D stope mesh on a yearly basis from 1992 to 1995 and monthly after this. The stress gradient used in modeling is based on a number of overcoring tests at the camp, being: \( \sigma_1 = 0.0437, \sigma_2 = 0.0299, \sigma_3 = 0.0214 \) (MPa/m) the trend and plunge are \([358^\circ, 10^\circ], [93^\circ, 28^\circ], [250^\circ, 60^\circ]\), respectively, with \( k = 2.0 \). [Note: this stress regime has been used for a number of other studies at the camp, (Nickson et. al., 1998; Newmont Canada, 2001; Bawden and Jones, 2002), and has been used here to keep constancy in the modelling results.]

4.2.3 Microseismic System and Array Consistency

As previously stated, in February 2002 just prior to commencement of mining of the destress slot the microseismic system was expanded to an 80 channel Hyperion system (ESG, 2006). The mine wide array, consisting originally of 3 triaxial accelerometers, (flat operating response \( \pm 3 \) dB, between 1 Hz and 5 kHz at 0.5 V/g), oriented using a tilt switch, then fully grouted in place, and 43 uniaxial accelerometers, (flat response, \( \pm 3 \) dB, between 50 Hz and 5 kHz and sensitivity of 30 V/g). The dense shaft array, part of this system, consisted of 25 sensors, including the triaxial accelerometers. An additional triaxial accelerometer, #16 (Figure 4.4a), was incorporated on January 10, 2003, and 4 more uniaxial accelerometers in the shaft region on February 21, 2003, to bring the dense shaft array up to 4 Triaxial and 26 Uniaxial sensors. The general spacing between sensors was in the order of 20 to 50 m in the shaft region, while the triaxials were 60 to 140 m from the centre of the cluster analyzed here, all with a clear line of sight.

An automatic arrival picker, (Appendix D1), and location algorithm is employed for the majority of events using a simplex algorithm employing the L2 norm to locate (ESG, 2006). Although the triaxial accelerometers employed at the site have a flat operating response between 1 Hz and 5 kHz, evaluation of the effect of low frequency cut-offs found that below a frequency of 150 Hz, low frequency noise became a contributing factor to degradation in some signals, creating significant scatter in the evaluation of the spectral parameters from cut-offs below this. A typical full wave form trace can be seen in Figure 4.5. Thus a Butterworth band pass filter was applied between 250 Hz and 5 kHz, and as the majority of corner frequencies were found to lie between 400 and 1000 Hz, this system and filtering should be optimal.
Figure 4.4  Errorspace analysis of Golden Giant dense Shaft Array (a) Array Feb 2003, uniaxial (circle) and triaxial (triangle) sensors (b) Corresponding errorspace (m) based on the nearest 17 sensors to the analysis volumes centroid (note: average number of sensors triggered 17) (c) Distribution of online location errors for events in the East of slot cluster (EOS), from April 2002 to Sept 2005.
Figure 4.5  Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_o$), corner frequency, ($f_c$) and energy flux, ($J_c$), fitted with a Brune $f^{-2}$ spectral decay.
The full waveform data is sampled at a rate of 20 kHz, and a 205 ms window of all triggered sensors is stored to the database with determined arrivals, while locations and source parameters are stored to a separate database. Here, as no in situ attenuation properties had been determined and because the source to triaxial sensor distances were small, in the range of 60 to 140 m, attenuation was not accounted for. From previous studies at this similar frequency range and source to sensor distance, (Gibowicz et al., 1991, Mercer, 1999, Urbancic et al., 1996), it has been found that attenuation corrections are not significant versus standard errors for estimation of the spectral parameters, and a correction based on a Q value of 200 would result in a possible amplitude increase of 10 to 20%, which are not significant (Urbancic et al., 1996; see Appendix A3 for formulation of attenuation).

A complete database of source locations for the mine exists from 1994 until September 2005, and full waveform records were obtained from January 2002 until closure. During the mining of the destress slot some 30,000 microseismic events were recorded in the shaft pillar region, and ~2,800 in the East of Slot cluster of events (EOS) predominantly analyzed in this Chapter. As part of this study an underground site check was made on the location and orientation of all sensors in the shaft region and a number of stations required minor correction, thus for consistency all events in the clusters analyzed were relocated using a minimum of 6 triggered sensors, (the average number of sensors triggered was 17). Automatic source parameters were recalculated based on a minimum of 2 triaxial sensors, (70% of events used 4 triaxials, 25% used 3 and only 5% used 2), using the original arrivals, before detailed analysis was performed. Comparison of the original dataset versus the corrected data indicated that the effect of changes to the sensor files were negligible, (0.5 to 1 m change in location, and minor change to the previously calculated source parameters based on this). It should be noted, however, that a number of source parameters for events were added to the database, which had inadvertently been removed from the database during manual processing (P-arrival re-picking of some stations and relocation\(^1\)) by the mine. This accidental deletion of source parameters was found to be during the critical time frame of June 2002 to mid February 2003, and was probably one of the main reasons why a significant temporal change in the source parameters that has been identified in this research was not previously seen in other studies (Mercer, 2003).

\(^1\) Note during the analysis period, the mine performed manual processing of events close to the shaft region including the cluster analysed here, upwards of 80% in 2002, dropping to 50% from Jan 2003 until May 2003, after which all processing was automatic. However, manual processing by the mine did not systematically re-pick every arrival only arrivals that appeared to be in error (either re-picking or dropping stations, and would often add picks that were not attained by the automatic picker), with the majority of the picks determined by the automatic P-wave arrival picker (Renelli pers. comm., 2002).
4.2.3.1 Location Error and Array Errorspace

The PCA technique, although a statistical method, requires reasonable locations and an understanding of the location accuracy. Although a full tomographic survey was not conducted with the dense array, from previous investigations, (Kazakidis, 1990), the mean P-wave velocity was determined to be between 6063 to 6087 m/s ±250, but varied by up to 5%, based on the direction of observation, (higher velocities parallel to foliation). However, with a reasonable array the effect of this transverse isotropy of the velocity model is negligible. A test blast was conducted for the dense array, and locations found to be within an error of 4 m, (Mercer pers. com., 2005), and from a detailed study of development blasts in the region (Chapter 5) locations were also around this range.

An average velocity of 6096 m/s was used for this study and an analysis of the errorspace\(^2\), (Coulson, 1996), indicates that this is a well conformed array with an average of 4 - 6 m theoretical error in the region of interest (Figure 4.4a &b). The array did change slightly a number of times over the analysis period (12 significant sensor file changes for the shaft pillar region), however, the errorspace as determined in Figure 4.4b was found to be reasonably consistent. This array was a significant improvement over the original array from 1995 to 2001, which had location errors in the order of 8 – 12 m in the region of the shaft, with the higher error range for events recorded from 1995 to 1996. Prior to 1995, no information on the sensor locations existed.

For the main clusters of events reviewed in this study, (EOS cluster, see Section 0), analysis of the online location errors determined from the time residuals and based on automatic picks indicates that the mean location error is 4 m, with a relatively low variance and a standard deviation of 2 m (Figure 4.4c). The average maximum dimension of this cluster was found to be around 48 m, hence a relative error based on this of around 8% can be assumed. As previously discussed in Chapter 3 based on work performed by Posodas et al., (1993) on perturbing synthetic data from a single plane, the determined orientation of a structure would not be significantly effected (< ±10° in dip and < ±1° in azimuth) for relative errors around 10%. On this basis, given that the average PCA trends are derived from the statistical contouring of poles on

\(^2\) The errorspace is calculated from the standard errors of the covariance matrix of the time-distance equation \((A\hat{A})^{-1}\) from the sensors to chosen grid locations and by applying time reading error based on 0.75 ms picking error such that the reading error is = 0.00075\(Nst(Nst-1)^{0.5}\), where Nst is the number of stations used (Trifu, 1983; Coulson, 1996). Here the average number of sensors triggered was found to be 17 during the key analysis period, and the closest 17 sensors to the centroid of the grid box were used for the errorspace analysis.
a lower hemisphere stereographic projection, the location errors are of an acceptable magnitude for the dimension of the cluster analysed. From the author’s experience, this array represents one of the best in a mining setting for location accuracy outside of the studies performed at the URL (Reyes-Montes, 2004).

4.2.3.2 Comparison of Online (Automatic Picks) versus Offline (Manual Picks) on Location and Source Parameter Calculations

From a related study on the focal mechanism of the events in the cluster analyzed here, a representative (Chapter 5) data set of 250 events spanning the time frame from May 2002 until May 2004 (Figure 4.6b), had P- and S- arrivals manually picked and source parameters recalculated to compare to the automatic online results. Figure 4.7a, b, & c, shows the comparison of the effect on location of the events, indicating that in general the majority of the events locate within 3 m of the original locations, this being lower than the location error due to the array. The differences in location are evenly distributed about the 1:1 relationship, indicating no bias in the automatic picking routine and the average absolute differences were found to be 1.6, 2.0 and 1.4 m, for Northing, Easting and Depth respectively. Based on this analysis of the automatic picks versus manual picks, it concluded that the automatic picks for location purposes are within an acceptable level and can be confidently used for further analysis (Appendix D1).

As can be seen in Figure 4.5a, sometimes the theoretical S-wave pick can be slightly late, hence it was important to evaluate the effect of manual versus automatic picks on the source parameters. In Figure 4.7d an example of the comparison between automatic versus manual picking for the moment magnitude (M) can be seen. As can be noted there is relatively little scatter evenly distributed about the 1:1 relationship, with an average slope offset of ±3%, and a regression coefficient of 0.82. In a more detailed comparison of all the source parameters (Appendix D1) this was found to hold for $M_o$, $E_o$, $E_s/E_p$, $r_o$ and $\sigma_a$, with average slope offsets (± %) of 5%, 10%, 13%, 2% and 6% respectively. These results suggest reliable estimates of source parameters can be obtained based on the automatic picking routine.
Figure 4.6  (a) East of Slot (EOS) cluster of events (1561) spanning from May 2002 to December 2003, used for analysis of effects of triaxial array consistency on source parameter calculation, and effects of low frequency cut-off. (b) Subset of events (250) spanning from May 2002 to May 2004 which where manually picked and used for analysis of automatic versus manual calculations and fault plane solutions (Chapter 5).
Figure 4.7 Comparison of Automatic picking versus manual picking of p-wave arrivals for a representative 250 event sub-set of the data in the EOS cluster, spanning from May 2002 to May 2004. (a) Northing, (b) Easting, and (c) Depth (axis range for all is 30 m). (d) Moment Magnitude, M (axis range –1.8 to –0.4).

4.2.3.3 Comparison of The Effect of Individual Triaxial Accelerometers on the Averaged Data

Generally, for basic mine interpretation of source parameters a minimum of 4 triaxial sensors are required to produce stable solutions for a small array (Hudyma and Brummer, 2007). For more in depth studies of focal mechanisms using moment tensor inversion, this number should be increased to 6 to 8 or include uniaxial sensors in the estimation (Trifu and Shumila, 2002). Uniaxial sensors can be included to calculate source parameters for a smaller triaxial array, if good focal sphere coverage can be obtained, as was evident here (Chapter 5). The source parameters were investigated based on the uniaxial sensors, which were found to show similar
temporal changes in these parameters to the triaxial sensors, with however, a substantial offset in the magnitude of the values (Appendix D2). These sensors have a different frequency response, are more sensitive to clipping and were found to be on the whole more influenced by noise than the triaxials creating greater scatter in mainly the corner frequencies and resultant source radii (Appendix D2). Thus, more confidence was found in the triaxial sensor response alone and these are used as the basis for this study. The main reason for averaging source parameters over a number of sensors is to achieve adequate radial coverage, due to non-uniform seismic radiation patterns from the rupture plane, (Madariaga, 1976), and potential site effects (Urbancic et al., 1996). At the Golden Giant mine, although the triaxial sensors are relatively well configured around the zone of interest, in order to make sure that the resultant temporal changes that are observed were not the influence of a single sensor, (especially the addition of triaxial sensor #16 during the monitoring on January 10, 2003), a rigorous analysis of the influence of each sensor was performed (Appendix D2). This was achieved by selectively removing one of the triaxial sensors from the sensor file, and recalculating the source parameters for a temporal sub set of the events from May 2002 to December 2003 (Figure 4.6a), during which time significant changes in the source parameters was identified.

Figure 4.8 shows the results of one of the comparisons for seismic moment. As can be noted a relatively good correlation can be made to the 1:1 relation for the majority of the range, however sensor #13 which is within 100 m of the cluster, shows a tendency to bias larger moments for smaller magnitude events. This may be due to its orientation with respect to the ruptures, being along strike of the foliation, such that higher frequency signals are attenuated less than the other triaxials. This may indicate that individual sensor attenuation correction in this case may have harmonized the source parameters closer to the average and the 1:1 relation. The added complexity however, required to determine a variable attenuation for each sensor was outside the scope of this research, with emphasis placed on observing any relevant relative changes based on the run of mine online data. As can be seen for Figure 4.8e, an example of the moving average of $M_o$, for the cluster of events analysed in this Chapter, for all triaxial excluded cases, exactly the same trends with slightly varying amplitudes were obtained. Similar results were found for the other source parameters (Appendix D2). The main concern that the addition of sensor #16 could have influenced the temporal changes was found not to be the case. The significance of these temporal changes will be discussed in the following section.
Figure 4.8  Comparison of Seismic Moment, ($M_o$) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials (a, b, c and d). and (e) comparison of temporal variation of average seismic moment for each 3 triaxial analysis, using a 50 event moving average.
4.3 Temporal and Spatial Analysis of Regional Failures

4.3.1 Regional Microseismicity and Cluster Identification

The destress slot functioned as designed, by deflecting high stress away from the shaft. As can be seen in Figure 4.9a and b, the majority of the yielding rock mass, as indicated by the microseismicity, occurred in the back and sides of the destress slot, while shedding stress off to the east and west abutments and the Q1/Q2 pillar. This yielding front can also be seen to grow around the destress slot and concentrate in the abutments (especially the east of slot region above the 4600 level), as identified from analysis of iso-surfaces of the clustering density, using a minimum of 5 events per 125 m³ voxel (Figure 4.10). This is a cumulative analysis, and assumes a complete data set, which may not be the case in lower Q1/Q2 pillar or above the back of the Q4/Q5 cave. However, this is based on a relatively complete data set spanning from June 1994 until September 2005. Although the sensitivity and accuracy of the locations recorded from the original 32 channel microseismic system is lower than that of the dense array, it is possible to get a reasonable picture of the history of the region. From the first notable flurry of activity around the shaft in 1995, of 661 events per year, the event rate in the shaft region averaged around 1,300 events per year from 1996 through 2001 (Appendix E). Most of this seismicity appears to be randomly distributed throughout the region with the exception of notable concentrations in the Q1/Q2 pillar. As the last mining in the region was in 1995, this seismicity is primarily attributable to regional increases in stress related to mining outside of the shaft pillar area and indicates the relative sensitivity of the stress state of the region. This rate increased significantly in 2002 with the mining of the destress slot and incorporation of the dense array to over 7,358 events per year, 13,034 in 2003, with 4,527 and 5,729 in 2004 and 2005 respectively (Appendix E).

It is interesting to note that two regions in the shaft pillar were completely aseismic or did not accumulate enough seismicity during the recorded monitoring (1994-2005) to reach the clustering density (Chapter 2 and 3), i.e. the upper section of the Q1/Q2 pillar between the 4600 and 4566 levels (456-Q1/Q2 stope) and the S4/S3 stope boundary between the 4620 and 4600 levels (460-S4 stope) (Figure 4.9a, Figure 4.10e and Appendix E). The former region of the Q1/Q2 may be attributed to a lens of a weaker rock unit known as ‘mafic fragmental ore’, which was found to cave above the 4566 level, over a transverse depth of 5 m all along the hanging wall contact and was first noted in 2002 upon re-entry into the region. On drilling of the 456-Q1/Q2 in late 2004, this lens was found to have caved almost to the level above.
Figure 4.9  (a) Microseismicity recorded in the Golden Giant Shaft Pillar region for one year of data during 2003. Analysis area EOS indicated. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25] (b) Detailed long section around the destress slot of advancing seismicity with time a mining. Events coloured by date, see legend (after Mercer, 2003).
Figure 4.10  Expansion of yield surface in the Golden Giant shaft pillar region, based on the clustering density of 5 events per 125 m³ voxel. Plots are based on cumulative seismicity from the beginning June, 1994 to (a) Feb 2002, (b) Aug 2002, (c) Dec 2002, (d) Feb 2003, (e) Aug 2003, and (f) Feb 2005.
This unit may have also been present in the S4/S3 stope, although this could not be confirmed as geological identification of the ore here was minimal at the time of the extraction. However, the 460-S3 stope, predominantly mined itself through caving, up to within 5 m of the sill, after blasting of the adjacent stope but one level above, the 462-S4 stope in March 2005. The main point is that this very localized more friable rock, could possibly have fractured at a high frequency or the fractures where so small to radiate limited energy outside of the range of the microseismic system, (aseismic) but was most definitely fractured.

During the mining of the destress slot a minimal amount of seismicity was recorded locally around the shaft, indicating that the shaft was positioned well inside the stress shadowed region, (Mercer, 2003), and this was also confirmed from stress monitoring instrumentation in the shaft (Bawden, 2003). Most of the clusters of events associated with the destress slot were temporally transient. The seismicity closely followed the destress slot excavation, progressing outward East and West of the slot some 5 to 15 m. After the level of seismicity had decayed, these yielded zones were mined (Figure 4.9b). These transient clusters were found to be difficult to analyze with the PCA technique to obtain stable trends, as it is postulated that two processes are occurring: transverse fracturing a result of tensile spall, and stress driven shear fracturing. The two most geometrically stable regions in which the seismicity accumulated were the Q1/Q2 central pillar core and the east of slot (EOS) region around the 460-S2 stope (Figure 4.2a). The former was analyzed but is complicated by two internal raises, within the pillar that were found to attract seismicity early on in the life of the mine (Appendix E). The application of the PCA technique to the Q1/Q2 cluster indicated generally a strong vertical North-South trend (Figure 4.11a and b), which may be construed as axial splitting, although is probably more geometrically controlled by the raises and the stope geometry. The best region for analysis was the EOS, 460-S2 stope region between the 4620 and 4600 levels, in which complete records of the failure process from initiation to complete aseismic behaviour were observed before the region was mined. Although a small raise does exist in the footwall, this was found not to be a major contributor to seismicity, and was probably filled with caved rock from above as a result of the brief 1995 flurry. The EOS region above the 4600 level is a analogous to a large ‘triaxial rock mass’ sample being gradually loaded in the horizontal direction, similar to a small scale triaxial laboratory sample in terms of loading conditions, however with a slower loading rate.
Figure 4.11 Location of yearly microseismicity in the Q1/Q2 cluster and stereographic distribution of PCA derived planes for (a) 2002 post localization (b) 2003 – probable disassociation. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25]
4.3.2 Temporal Analysis of the East Slot (EOS) Pendant Pillar (4600 L)

The cluster analyzed here undergoes gradual loading, with principal stresses being applied almost monotonically and is discussed in the following sections. This is a stress driven failure of a confined rock mass with no caving, as the David Bell stopes below the 4600 level were tightly sand-filled. It forms a similar failure type to that analysed in the Williams mine sill pillar footwall (Chapter 3). The events in the cluster were first extracted yearly based on a visual identification of spatially discrete clustering of events and using the eventual zone of the clustering event density as a guide (Figure 4.10). It is important to note that other than defining the appropriate volume to extract the events, no further filtering of the data was performed. The cluster was then analyzed in greater detail using the PCA observations to define the state of the rock mass along with observations from linear elastic modelling, and used as a guide to investigate temporal changes in the source parameters.

4.3.2.1 Yearly PCA Analysis of EOS 4600L

The PCA technique was applied to the identify the dominant trends in the spatial distribution of events, first on a yearly basis, 2002, 2003, 2004, and 2005, with the poles of resultant ellipsoids for each year plotted on lower hemisphere stereographic projections (Figure 4.12a, b, c and d). It was found that during 2002, although seismicity in the zone had initiated, there was no strong correlation of spatial trends. The poles of the ellipsoids, or PCA derived planes, are generally dispersed and a number of planes do not reach the 2.5 ellipticity criteria to be defined as a planar trend. However, in 2003 a dominant strong and stable trend was identified, (Figure 4.12b), having a mean orientation in terms of strike and dip of [075°, 52°]. Under closer examination this trend was found to weaken towards the middle of 2003, and as can be seen there appears to be a great degree of variability in the planes for 2004, and 2005 with a wide distribution of the PCA derived planes and unstable trends. The diameter of the spatial window, D, found from the cumulative density distribution of the inter-distances for each year of data, (2002 to 2005), was found to be, 22, 20, 20, and 19 m respectively. As determined at the Williams Mine (Chapter 3), this distance appears to correlate with the minimum dimension of the cluster and appears to be closely related to the sub-level spacing (20 m) bounding the cluster. One distinctive difference here is that, during all of the years accept for 2002, very few events are dropped due to low ellipticity. Additionally, the F-parameter (k/N’ = number of events in the spatial window/number of events in the temporal window), predominantly stays greater than 50%, and during the stable trend in 2003 is greater than 90%, indicating that events occurring sequentially in time are occurring very close to one another.
Figure 4.12a,b  Location of yearly microseismicity in the EOS cluster and stereographic distribution of PCA derived planes for (a) 2002 – pre-coalescence, (b) 2003 – coalescence/ localization, and disassociation. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 24, orange = > 25]
Figure 4.12c,d  Location of yearly microseismicity in the EOS cluster and stereographic distribution of PCA derived planes for (c) 2004 – post disassociation, (d) 2005 – post disassociation. (Note: in both years there is a distinctly aseismic region below 4620L. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25]
In the Williams study it was identified that events on the outer edges of the cluster often had low F-parameters and resulted in false high ellipticities, which had to be removed from the analysis. Here, because of the greater clustering this was not necessary. This greater clustering is thought to be primarily a function of the increased location accuracy for the dense array at Golden Giant.

### 4.3.2.2 Detailed Temporal PCA Analysis 2002 to 2005 EOS 4600L

In order to observe the temporal change of this dominant trend in greater detail, the individual 50 event temporal windows, used to determine the PCA derived planes, were analyzed on stereonets individually (Figure 4.13, and with locations Appendix F). As can be seen in the progression of pole concentrations, the trends are relatively dispersed and unstable until the beginning of 2003 (temporal window 1, Figure 4.13a), and become strongly defined and more stable with little scatter of poles until the end of June 2003 (temporal window 26). The mean stereonet plane orientation based on the Fisher distribution of the maximum pole concentration for each PCA temporal window is plotted versus time of the last event in the window. This, as with the Williams study, allows changes in orientation with average ellipticity for a given window to be summarized on one plot, (Figure 4.14a), and can be directly compared to the event rate versus time (Figure 4.14b). It can be seen from this data that there is greater stability in the strike of the planes than the dip, this is in part due to geometry of the region, in that the strike of these PCA derived planes follow the strike of the ore body, where as the dip is controlled by the fracture propagations and is more variable.

The regional rock mass failure in this pendant pillar occurs within a few months once initiated, unlike the similar study at the Williams mine, (Chapter 3), that occurred over 3 years. In order to visually see the key identifying failure points the variation is plotted against PCA temporal window number, thus stretching the time scale in the most active period (Figure 4.14c).

The first minor flurry of activity in the region occurs in May 2002, during the mining of the first stope in the destress slot, and this point in time is estimated to be the point of fracture initiation. Starting in late December 2002, the dominant PCA trend starts to stabilize to an overall orientation, on average of [075°, 50°], with little scatter in the poles, and is maintained for the next 6 months (26 temporal windows). Potential fracture interaction (yield strength of the rock mass), has been determined to occur on February 17, 2003, based on the yield envelope of the clustering event density expanding over this region, (Figure 4.10c and d).
Figure 4.13a Stereonets of PCA derived planes of individual 50 event temporal windows, for clustering events above the 4600L, EOS cluster. Temporal window number, date of last event in window and major plane indicated. Dashed border represents low number of events with ellipticity > 2.5.
Figure 4.13b Stereonets of PCA derived planes of individual temporal windows (50 events), for clustering events above the 4600 L, EOS cluster. Temporal window number, date of last event in window and major plane indicated. Dark border indicates point of disassociation and major change in failure direction.
Figure 4.13c Stereonets of PCA derived planes of individual temporal windows (50 events), for clustering events above the 4600 L, EOS cluster. Temporal window number, date of last event in window and major plane indicated. Dark border indicates post disassociation and major change in failure direction.
Figure 4.13d Stereonets of PCA derived planes of individual temporal windows (50 events), for clustering events above the 4600 L, EOS cluster. Temporal window number, date of last event in window and major plane indicated. Dark border indicates post of disassociation and major change in failure direction.
Figure 4.14  (a) Temporal variation in mean PCA derived planes for EOS cluster from May 2002 – July 2005, showing variation in mean strike and dip, and ellipticity per temporal window. Also indicated are stopes mined (e.g. S5 mined between 4620 and 4600 levels is named 460-S5). (b) Event rate in EOS cluster, (c) Temporal variation plotted against PCA temporal window number.
This expansion is a result of the major flurry of event activity during the mining of the adjacent 462-S5 stope which had its final blast on February 22, 2003 (Figure 4.12b and Figure 4.14b). At this point the average ellipticity, describing the strength of the planar trend, starts to increase from a background level of 5 to 10, and two weeks later (March 5 to 17, 2003), significantly increases to above 25, indicating strong coalescence and localization (Figure 4.14a and c) as was also identified for the Williams case study (Chapter 3).

This behaviour is postulated to be the formation of a macrofracture structure, and is not thought to be a single feature, but a yielding process zone, or shear zone, developed through en echelon stress driven fractures, or networking of fractures. From a study of the first motions of a representative sub-set of these events, the average orientation of fault plane solutions for these mainly reverse faulting mechanisms, was found to be oriented [072°, 51°] in the same direction as the dominant PCA trend (Chapter 5). As will be identified from their source parameters, these events are most probably creation of new fractures of the intact rock, and not fault slip on pre-existing structure of the C-set, which has limited prevalence in the area and a significantly flatter dip (Figure 4.3). Additionally, a review of the detailed geology shows no evidence of a structure with this orientation, and this mechanism is the same as identified through similar confined stress driven failure at the Williams mine (Chapter 3). A ubiquitous joint analysis (Chapter 5) indicates that this failure is in the direction of maximum shear, the failure direction corresponding to an orientation that equates to the minimum cohesional strength of the material. This macrofracture structure is thought to be of similar genesis to those on a much smaller scale, found through laboratory triaxial testing, (Lockner et. al., 1992), and on formation, probably represents the peak strength of the rock mass, such that after the formation and with increased damage accumulation, the rock mass transitions into the post peak.

The breakdown of the dominant trend occurs around June 22, 2003, with the trend, switching to an average strike and dip of [260°, 80°]. It switches back for two temporal windows (Figure 4.13b and Figure 4.14a and c), and then, after Sept. 15 2003 becomes unstable. This first change is identified as the point of disassociation, based on destabilization of the dominant trend and potential activation of a conjugate fracture set or shear. From the study at the Williams mine, (Chapter 3), this represents regions of the rock mass where significant dilations identified from deep displacements of SMART cables, cause patches to become aseismic, and it is at this point that it is thought that the rock mass has started to approach it's residual strength. This dilation would cause significant strain-softening behaviour, although it is expected that directly following localization, during further damage accumulation, that gradual
dilation and strain softening occur continuously until a region becomes partially aseismic. The idea that dilation is occurring comes from evidence of correlations in South Africa of physical deformations with cumulative apparent volume, (van Aswegen and Butler, 1993) and also in Canada at Brunswick Mine, where cumulative apparent volume correlated to physical rate of convergence of drift back, (>30 mm), below a seismically active cluster (Simser, 2000). Here, although no direct measurement of displacements could be made, the point of disassociation in June 2003 is coincidental with the identification of major floor heave, (> 0.5 m), in the 4620 level above, and after this point the region directly below this level, (~ 5 m), becomes aseismic (Figure 4.12c and d). As previously noted, the major trend that was seen prior to the point of disassociation becomes unstable from the middle of 2003 and throughout 2004 and 2005. This indicates that although similar fracture interactions are occurring, the rock mass in this region has become highly fractured and may have started to dilate significantly, causing a breakdown in the group behaviour. The major trends switch between an average orientation of [255°, 65°] (which is similar to the A-set orientation but sometimes steeper and may be interpreted as a conjugate structure allowing for greater degrees of freedom), a horizontal trend and the previously stable major trend [075°, 50°]. The main weakness in the interpretation that the second trend identified is a true conjugate structure is that generally the dip tends to be too steep. This could be related to more complex fracture interaction reducing the identification with the PCA of these smaller structures in relation to the overall macrofracture shear zone. A secondary localization can be seen in early 2005, noted by the short increase in the ellipticity (Figure 4.14), during the mining of 460-S4 stope, but is fleeting, occurring on the far abutment and not considered to be significant. The region at the core of the cluster, the highest event density region, is identified as being completely aseismic at the end of May 2005, although it was concluded that the majority of the ‘failure’ (i.e. the rock mass approached or reached it’s residual strength) occurred in 2003, (Bawden, 2003; Mercer, 2003).

In order to determine whether the selection of temporal windows using a 50 event window with a shift of 50, had any effect on the determined distribution of PCA derived planes, the PCA was run on the entire cluster of events from May 2002 until Sept 2005, using a continuous analysis centring the temporal window on ± 25 events and shifting the temporal window every event (central moving principal components analysis, CMPCA). This has been compared to the PCA derived planes determined using the original method performed on the entire cluster, (50 event, shift 50 sample), using a 25 point moving average, that was found to be most appropriate to smooth the data without compromising the temporal changes (Appendix A2).
Figure 4.15  Temporal variation of PCA planes determined for the events in the EOS cluster, (a) using 25 point moving average of the PCA strike and comparing the analysis using a 50 event temporal window with a 50 event shift (N'=50), versus a 50 event continuously shifting temporal window centred on ± 25 events and a shift of 1 event (N'=Cont.+25), and moving average ellipticity, (b) moving average of the PCA dip for (N'=50) and (N'=Cont.+25), and standard error of strike, dip and ellipticity ((N'=Cont.+25) (c) moving average of the number of planes dropped due to ellipticity < 2.5.
The average strike and dip with average ellipticity for both methods is displayed in Figure 4.15a and b, and although this is not an entirely appropriate method for averaging three dimensional data, it is clear that the same trends as determined using stereographic projection, (Figure 4.14), are again identified. The effect of linear averaging of all PCA planes reduces the resultant strike or dip when there is a mixture of orientations, differing from the use of stereonets in which the low concentration of poles are dropped from the major plane averaging window.

Observation of the standard errors for strike and dip for the CMPCA, (Figure 4.15b), shows that prior to the flurry of activity on February 17 2003, errors are larger indicating more disrupted trends. As coalescence and localization occurs there is significant drop in the standard errors indicating group behaviour of events or fractures, positionally related to one another, follow the same orientation, (the dominant PCA trend), and have strongly associated behaviour until the first point of disassociation is noticed, after which the events or fractures do not appear to be as strongly influence or related to one another, causing their overall orientations to vary and for the trend to become unstable. The instability in the trend found after the point of disassociation, is not a function of the fixed temporal window selection, as the same swings occur when the continuous analysis is used, matching the sampled PCA trends relatively closely, and the changes as identified in Figure 4.14 are replicated in Figure 4.15.

The identification of the state of the rock mass strength as identified by the temporal PCA technique, is investigated further with the analysis of the linear elastic stress history and the source parameters in the following sections. The dates determined from the continuous central moving PCA and the event rate summarised on Figure 4.15 and used as identifiers in the following section are:

1. Fracture initiation – 05/05/2002 (First flurry of activity)
2. Fracture interaction – 02/17/2003 (Major flurry of seismicity, critical clustering density reached, ellipticity starts to increase)
3. Fracture localization – 03/05/2003 (Ellipticity significantly increases)
4. First point of disassociation – 06/22/2003 (First breakdown of major PCA trend)
5. Second point of disassociation – 09/15/2003 (PCA trend becomes unstable)
6. Secondary localization – 02/05/2005 (Minor increase in ellipticity)
7. Completely Aseismic (S2) – 05/05/2005 (Approx. date that seismicity in the 460-S2 stope ceases, and is only in the far east abutment)
4.3.2.3 Linear Elastic Modelling and Stress Path

In order to determine the potential stress history, the starting point is to use 3D linear elastic boundary element modelling. The modelling packages employed are Examine3D, (Rocscience, 2007), and Map3d, (Mine Modelling Ltd., 2006); the former for general stress analysis and a stepping stone to Phase², (two dimensional non-linear finite element modelling); the later for detailed analysis of linear elastic stress histories around openings, utilising a stress averaging routine developed originally at the Noranda Technology Centre, (Rizkalla, 2001), and detailed by Falmagne (2002), (summarised in Chapter 3, Section 3.7.4.4). As previously mentioned, (Chapter 3), the use of the two modelling packages also aids in the check and comparison of required discretization and modelling efficiency parameters³ to achieve the same stress analysis as was reviewed again here (< 2% difference). The in situ stress regime is the same as that used to model the Williams mine, (Chapter 3), and summarised in Section 4.2.2.

The calculated maximum ($\sigma_1$) and minimum ($\sigma_3$) principal stresses are plotted in Figure 4.16, for the mining stage just prior to development of the destress slot, (Dec 2001), and for the period after completion of the main stopes in the destress slot (Dec 2003). As can be seen there is a significant decrease in $\sigma_1$ around the shaft of 30 to 40 MPa as a result of the stress shadowing from the slot stopes, with a minor decrease in $\sigma_3$ of around 5-10 MPa. The region of the east abutment, at the centre of the cluster analysed here, (EOS region), was already highly stressed prior to mining of the slot stopes, ($\sigma_1 > 100$ MPa), the microseismicity being associated to this high stressed area in which the confinement is also moderately high at around 40 MPa, and localized between the 4600 and 4620 levels. Following mining of the destress stopes there is a minor increase in the both principal stresses by 5 to 10 MPa, however, the seismicity started before the full slot was mined, and it is felt that this region (EOS) was already on the verge of instability, but that the confinement was inhibiting fracture propagation until a critical density of fractures occurred allowing coalescence and causing the strong localization.

The average stress path history, from 1993 to 2005, for a region at the core of the cluster of events is plotted in Figure 4.17, and indicates a gradual and almost monotonic loading of the principal stresses, similar to a laboratory triaxial test, with the intermediate principal stress ($\sigma_2$), being close in magnitude to $\sigma_3$ (Figure 4.17b).

³ Examine3D was modelled with a moderate mesh discretization for reasonable run times, using linear stress elements, stress tolerance 0.0001 and required a constant stress multiplier (1.168) to achieve the same induced stress magnitude for this large model as Map3D using constant stress elements and fine discretization based on modelling parameters of STOL=0.1, AL=4, AG=4, DOC=2, DOL=2, DON=0.5, DOE=4, DOG=4 and DOR=5.
Figure 4.16 Linear elastic principal stresses determined from Examine3D, for the Golden Giant shaft pillar region. (a) and (b), stress magnitudes of the maximum ($\sigma_1$) and minimum ($\sigma_3$) principal stresses respectively for Dec 2001 prior to mining of the destress slot, and (c) and (d) the same but for the Dec 2003, after completion of the main stopes of the destress slot. (note the first ‘top’ view is of a plane cutting the 4600 L, the ‘front’ view of a plane cutting through the shaft, and ‘right’ view of a plane cutting through the EOS region, while the ‘perspective view is looking at the shaft pillar region from the hanging wall.)
Figure 4.17  (a) Average linear elastic stress path from 1993-2005, from the defined polygon on section 10480E (inset) covering the region of the clustering of events between the 4600 and 4620 levels east of the destress slot. The peak rock mass failure curves are displayed for reference. (b) Average linear elastic principal stress magnitudes versus time. Also plotted for comparison is the average stress path determined at the Williams Mine for the 9390L FW cluster Slice 2, Region 2, on Sec 9435E (Coulson, 2008a).
On Figure 4.17a, damage limits based on the Hoek-Brown brittle failure parameters, (Martin et. al., 1999), have been plotted as a reference. Originally this damage criteria was used empirically to determined the stress levels at which brittle failure from spalling is initiated at the excavation surface, but has been used by other authors to indicate damage levels when analysing linear elastic stresses, (Falmagne, 2002; Diederichs et. al., 2002). Also the Hoek-Brown peak rock mass strength envelope based on the intact rock properties and applying a GSI of 60, (representative for this rock mass), is plotted for comparison. The significant points that were identified from the microseismicity have also been indicated. At the Golden Giant mine, similar to the complimentary study at the Williams mine, (Chapter 3), there is relatively little change in induced linear elastic stress from initiation to disassociation. The major difference between the two analyses is the speed of the failure from coalescence to disassociation at Golden Giant. One reason for this is that, unlike at the Williams mine where the majority of the mining induced stress increase was due to mining some distance away from the failure, here the mining is directly adjacent to the cluster, and the mining of the 462-S5 stope causes a significant jump in stress of 7 MPa after the main flurry of activity had already initiated (Figure 4.17b). The proximity of the blasting and this small, but sudden jump in stress level during coalescence and localization, may be the reason for acceleration in the failure process. The other reason is that this region was at a higher stress state, such that once the critical fracture density was reached the fracture process became unstable. The key damage stages for both analyses are summarised in Table 4.2, with average stress levels, s-parameter and equivalent constant deviatoric stress term of $\sigma_1-\sigma_3/\sigma_c$ based on the Hoek-Brown Brittle parameters, with $m=0$. The stress path for one of the Williams cases has been displayed on Figure 4.17a and b. It can be noted that there is a relative increase in the point of initiation, coalescence and disassociation at Golden Giant over Williams Mine, and although a minor flurry of activity initiated in Oct 1995 (Nickson et. al., 1998), the seismicity abated and did not pick up again until the mining of the destress slot, when the first significant flurry occurred. However, the level of the damage limit for initiation is still within the range identified empirically by Martin et. al., (1999) and observed by Falmagne (2002) and Diederichs et. al., (2002), (i.e. damage initiation is observed to occur based on average linear elastic stresses for $\sigma_1-\sigma_3 = (0.4 \pm 0.1) \sigma_c$.)

If the damage limit observed from the Williams case study of interaction/coalescence and localization occurring at $0.3\sigma_c$ was applied as a predictor of yield for this zone, (a reasonable assumption based on equivalent rock masses and loading conditions), then the point in time that the failure should have initiated is predicted earlier than when it occurred by a number of
years, unless the parameters are increased to the upper observed limit of $0.37\sigma_c$ identified here (Figure 4.18).

### Table 4.2  Comparison of determined damage levels using the Hoek-Brown Brittle parameters based on linear elastic modelling between Golden Giant mine and Williams mine Slice 2, Sec 9435E (Chapter 3) $\sigma_c = 175\text{MPa}, m = 0$ (note: $\sigma_i = \sigma_3 + (m\sigma_3 + s)^{0.5}$)

<table>
<thead>
<tr>
<th></th>
<th>Golden Giant Mine</th>
<th>Williams Mine (Slice 2)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Date</td>
<td>$\sigma_1$ (MPa)</td>
</tr>
<tr>
<td>First minor flurry</td>
<td>Oct '95</td>
<td>84</td>
</tr>
<tr>
<td>Initiation</td>
<td>Mar '02</td>
<td>110</td>
</tr>
<tr>
<td>Coalescence/ Localization</td>
<td>Feb '03</td>
<td>113</td>
</tr>
<tr>
<td>Disassociation</td>
<td>Jun '03</td>
<td>120</td>
</tr>
</tbody>
</table>

It can be seen that the yield zone for the lower initiation damage limit occurs as early as 1998-12, while for the upper limit the zone is not predicted to move into complete yield until 2003-12. Diederichs (2000) and Diederichs et. al., (2002), proposed a damage thresholds based on a modified Tresca formulation which has been used by a number of researchers to represent the more significant confinement dependence on the initiation stress than suggested by Martin (1997) taking the form of $\sigma_1 = A\sigma_3 + B\sigma_3$ (where: $A = 0.4 \pm 0.1$ and $B = 1$ to 2). Here a fit for this data, (although very limited), would give values of $A=0.22$ and $B=1.5$ (Figure 4.17a). Note that this relationship passes through the intercept of the rock mass unconfined strength based on the equivalent M-C criteria. This is a similar relationship to that defined by Diederichs et. al. (2002), who defined $A=0.25$, $B=1$ for the case of initiation of instability of secondary stope pillars using linear elastic modelling at the Brunswick Mine.

If the classical Hoek-Brown parameters for the peak rock mass strength are applied, then for this linear elastic modelling they are deemed too conservative, as failure is never predicted in the region of intense seismic activity (Figure 4.18e & f). However, for non-linear modelling in which stress is shed from failed regions these peak parameters appear appropriate (Chapter 3 and 5).

The main difference for this analysis over Williams mine is that the confinement, $\sigma_3$, is almost double that of Williams, and although the rate of increase of $\sigma_1$ is equivalent for both cases, at Golden Giant there is a slightly higher increase in the rate of induced confinement $\sigma_3$ (Figure 4.17b).
Figure 4.18 Comparison of contours of factor of safety (FOS) on section 10480E through the core of the EOS cluster, for the Hoek-Brown Brittle parameters (a) & (b) initiation damage limit $0.3\sigma_c$ compared to (c) & (d) initiation damage limit $0.37\sigma_c$. (e) & (f) Hoek-Brown rock mass peak strength parameters.
The confinement is greater here primarily due to the mining geometry, with a solid abutment above the region while at the Williams mine above the sill the region is mined. This increased confinement may explain the delayed reaction at Golden Giant, however, it must pointed out that in determining appropriate damage limits, the variation in the rock strength, modelling accuracy and stress averaging can easily effect this calibration over this limited range and this empirically determined limit can only really be used as a 'rough indicator'. In order to more accurately determine more valid damage limits, a far greater number of case studies would be required; this was not the objective of this research. Additionally, linear elastic modelling cannot account for stress shedding resulting from regions where the rock mass is failing.

Based on observations during the eventual mining of this zone, (primarily mining of the 462-S5 stope), the overall post-peak behaviour is deemed to be brittle strain softening in nature. During the drilling of this zone there were no significant issues associated with spalling and loss of drill holes or short drill holes due to toe crushing which had been experienced on the first few stopes of the destress slot. The general comments regarding the drilling of this zone after inferred failure were that the ground appeared destressed and drilling was easier than expected, (Toppi, pers. com., 2005). This ground was, however, highly fractured as noted earlier, since during mining the upper lift of the S4 stope, between the 4633 and 4620 levels, the S3 stope caved substantially after the first void blast to within 5 m of the overcut (Curry, pers. com., 2005). Additionally, it was noted in late 2003 and early 2004, that the heavy stress spalling and floor heave in the 4620 level above this zone had stabilized. However, additional stress fracturing and minor loose with more intense seismicity initiated on the 4633 level in early 2004, and this area was noted to be taking load and required upgraded shotcrete support (Mercer, 2005). Only strain softening behaviour will result in the lowering of the stress within the zone, effectively shedding stress to more competent regions.

This is contrary to what might be predicted based the location of the linear elastic stress path in relation to the brittle-ductile transition zone, \((\sigma_1/\sigma_3 <= 3.4)\), defined by Mogi (1966) for intact polycrystalline rock. As discussed in Chapter 3, direct interpretation of Mogi’s behavioural classification when using linear elastic modelling, as suggested by Falmagne (2002), may not be applicable as local strain softening around openings and deeper failure can influence the induced stress field such that the stresses determined from the linear elastic modelling are only a guide and not representative when significant failure occurs (i.e. the rock mass may have never reached these stresses). This doesn’t suggest that Mogi is wrong but underlines the limitations of linear elastic modelling to identify post peak failure mode.
Additionally based on the observations of Hoek and Brown (1980), from limited laboratory triaxial tests on samples with planes of weakness, or that had been previously broken, it was identified that the transition zone did not hold and without published guidelines suggested a transition zone of $\sigma_1/\sigma_3 \leq 2$ (Hoek, 2004). As can be seen in Figure 4.17a the stress paths in this case lie above this ‘disturbed’ brittle-ductile transition zone. This again underlines the need for further research into this area.

An alternate interpretation is that the formation of this macrofracture shear structure occurs under ductile conditions, creating shear banding at the centre of the confined pillar zone. There is a change in the strength state, from ductile to brittle due to an overall change in the pillar geometry from short and squat creating significant confinement and ductile behaviour to a more slender pillar with brittle behaviour (Kaiser pers. comm., 2009). This is not a change in the material behaviour but a change in the strength state due to geometrical influences. The effect of the change in the geometry should be investigated further but is outside the scope of this thesis.

Numerical evidence for a strain softening response for a moderately confined region (20 to 30 MPa) based on non-linear modelling of near field rock mass displacements is however supported, as only application of non-linear brittle-plastic post peak parameters could approximate the dilation magnitudes and depth of local failure (Crowder et. al., 2006; Chapter 3). Further, in another simplified study of non-linear parameters to this region of the confined pillar, only brittle-plastic post peak parameters, could create similar shear patterns as identified from the PCA technique and fault plane solutions (Chapter 5) and a reduction in induced core pillar stress to below the initiation stress, as was physically observed during mining.
4.3.2.4 Observations of Temporal Changes in Source Parameters

In order to identify any significant changes in the source parameters, it is necessary to perform smoothing of the raw data to remove the inherent spiky, (log normal), response of individual events. This can be accomplished by averaging the data over time periods. Here it was found that to provide enough smoothing without compromising potential temporal changes, a 50 event moving average was required (Note: this is a backward looking average of the last 50 events, moved to the next sequential event).

The temporal variations with standard errors for strength estimates of Moment magnitude, $M$, seismic moment, $M_o$, radiated seismic energy, $E_o$ and stress estimates, apparent stress, $\sigma_a$, static stress, $\Delta \sigma$, and dynamic stress, $\Delta \sigma_d$ from May 2002, until March 2005, are plotted in Figure 4.19 and Figure 4.20. The measure of the source complexity, $\Delta \sigma_d/\Delta \sigma$ and the $E_o/E_p$ ratio, source radius, $r_o$ (Madariaga), apparent radius, $r_a$ (based on the apparent volume) are plotted in Figure 4.21 and Figure 4.22. A statistical summary of the events broken down into three definite periods, of pre-interaction, coalescence/localization and post localization including disassociation are presented in Table 4.3. Prior to April 2002 the system was in transition, and only selected triaxials were available, (generally less than two), hence this data was ignored. Additionally, on July 20 2002, 20:58:21.66, (during the pre-interaction period, Table 4.3), a large event, occurring with 3 other moderate events, was recorded with a moment magnitude $-0.18$ ($M_o = 5.28e8$ N.m) and a high energy value of $8e3$ J. This was found to locate at the bottom of the EOS cluster, within 1 m of the footwall drive. This event was removed from the temporal analysis, due to the anomalous effect on the averaged trend of $M_o$ and particularly $E_o$, and can be termed an outlier based on the distribution compared to the other events. However, for the scaling relationships discussed later in section 4.2.5 this event was included in the analysis.
Figure 4.19 Temporal variation of source parameters (EOS – Cluster) using a 50 event moving average for, (a) Moment magnitude, (b) Seismic moment, (c) Seismic Energy.
Figure 4.20 Temporal variation of source parameters (EOS – Cluster) using a 50 event moving average for, (a) Apparent Stress, (b) Static Stress Drop (Madariaga Model), (c) Dynamic Stress Drop (Madariaga Model).
Figure 4.21 (a) Temporal variation of Source Complexity ($\Delta \sigma_d / \Delta \sigma$), for the EOS Cluster using a 50 event moving average. Estimation of fracture geometry from a cross section on the 10480E plane through the cluster of events based on location, source radius and PCA calculated orientation of events for (b) the period prior to interaction, 04/18/2002 to 01/18/2003 and (c) the period during coalescence and localization (high ellipticities), 03/05/2003 to 04/30/2003 (from Chapter 5).
Figure 4.22 Temporal variation of source parameters (EOS – Cluster) using a 50 event moving average for, (a) $E_s/E_p$ ratio, (b) Source radius (Madariaga), (c) Apparent radius.
Table 4.3 Statistical summaries of source parameters for the EOS cluster.

### Pre-Interaction, 04/19/2002 to 01/18/2003 (n=308)

<table>
<thead>
<tr>
<th></th>
<th>M</th>
<th>$M_0$ (Nm)</th>
<th>$E_0$ (J)</th>
<th>$\sigma_a$ (Pa)</th>
<th>$\Delta \sigma$ (Pa)</th>
<th>$\Delta \sigma_d$ (Pa)</th>
<th>$E_d/E_p$</th>
<th>$r_0$ (m)</th>
<th>$r_a$ (m)</th>
<th>$\Delta \sigma_d / \Delta \sigma$</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Mean</strong></td>
<td>-1.12</td>
<td>2.93e07</td>
<td>166.18</td>
<td>9.42e04</td>
<td>2.61e06</td>
<td>2.74e06</td>
<td>1.90</td>
<td>1.69</td>
<td>3.47</td>
<td>1.07</td>
</tr>
<tr>
<td><strong>Std.D.</strong></td>
<td>0.23</td>
<td>3.27e07</td>
<td>460.36</td>
<td>9.47e04</td>
<td>2.45e06</td>
<td>3.23e06</td>
<td>0.87</td>
<td>0.19</td>
<td>0.55</td>
<td>0.42</td>
</tr>
<tr>
<td><strong>Median</strong></td>
<td>-1.17</td>
<td>1.76e07</td>
<td>33.30</td>
<td>6.34e04</td>
<td>1.90e06</td>
<td>1.93e06</td>
<td>1.79</td>
<td>1.67</td>
<td>3.39</td>
<td>0.91</td>
</tr>
<tr>
<td><strong>Min.</strong></td>
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<td>2.05e06</td>
<td>0.24</td>
<td>3.18e03</td>
<td>1.40e05</td>
<td>1.12e05</td>
<td>0.51</td>
<td>1.08</td>
<td>2.12</td>
<td>0.45</td>
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<tr>
<td><strong>Max.</strong></td>
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<td>2.76e08</td>
<td>5700.00</td>
<td>6.84e05</td>
<td>2.00e07</td>
<td>3.24e07</td>
<td>5.80</td>
<td>2.26</td>
<td>5.53</td>
<td>3.56</td>
</tr>
<tr>
<td>07/20/02</td>
<td>-0.18</td>
<td>5.28e08</td>
<td>8080.00</td>
<td>5.06e05</td>
<td>1.52e07</td>
<td>2.91e07</td>
<td>2.24</td>
<td>2.09</td>
<td>4.99</td>
<td>1.91</td>
</tr>
</tbody>
</table>

### Localization, 02/17/2003 to 03/31/2003 (n=707)

<table>
<thead>
<tr>
<th></th>
<th>M</th>
<th>$M_0$ (Nm)</th>
<th>$E_0$ (J)</th>
<th>$\sigma_a$ (Pa)</th>
<th>$\Delta \sigma$ (Pa)</th>
<th>$\Delta \sigma_d$ (Pa)</th>
<th>$E_d/E_p$</th>
<th>$r_0$ (m)</th>
<th>$r_a$ (m)</th>
<th>$\Delta \sigma_d / \Delta \sigma$</th>
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<tbody>
<tr>
<td><strong>Mean</strong></td>
<td>-1.29</td>
<td>1.50e07</td>
<td>55.74</td>
<td>5.02e04</td>
<td>1.62e06</td>
<td>2.50e06</td>
<td>1.95</td>
<td>1.66</td>
<td>3.56</td>
<td>1.73</td>
</tr>
<tr>
<td><strong>Std.D.</strong></td>
<td>0.16</td>
<td>2.00e07</td>
<td>264.47</td>
<td>7.00e04</td>
<td>1.80e06</td>
<td>2.38e06</td>
<td>1.72</td>
<td>0.22</td>
<td>0.60</td>
<td>0.68</td>
</tr>
<tr>
<td><strong>Median</strong></td>
<td>-1.34</td>
<td>9.83e06</td>
<td>7.25</td>
<td>2.50e04</td>
<td>9.40e05</td>
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<td>1.69</td>
<td>1.66</td>
<td>3.57</td>
<td>1.67</td>
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<tr>
<td><strong>Min.</strong></td>
<td>-1.64</td>
<td>3.42e06</td>
<td>1.18</td>
<td>7.20e03</td>
<td>3.05e05</td>
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<td>0.58</td>
<td>1.13</td>
<td>1.90</td>
<td>0.66</td>
</tr>
<tr>
<td><strong>Max.</strong></td>
<td>-0.38</td>
<td>2.70e08</td>
<td>4150.00</td>
<td>8.40e05</td>
<td>1.68e07</td>
<td>2.14e07</td>
<td>26.36</td>
<td>2.32</td>
<td>6.93</td>
<td>12.67</td>
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### Post Localization, 04/01/2003 to 09/30/2003 (n=539)

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<th>$M_0$ (Nm)</th>
<th>$E_0$ (J)</th>
<th>$\sigma_a$ (Pa)</th>
<th>$\Delta \sigma$ (Pa)</th>
<th>$\Delta \sigma_d$ (Pa)</th>
<th>$E_d/E_p$</th>
<th>$r_0$ (m)</th>
<th>$r_a$ (m)</th>
<th>$\Delta \sigma_d / \Delta \sigma$</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Mean</strong></td>
<td>-1.27</td>
<td>1.53e07</td>
<td>42.31</td>
<td>4.21e04</td>
<td>1.40e06</td>
<td>2.03e06</td>
<td>1.72</td>
<td>1.73</td>
<td>3.74</td>
<td>1.65</td>
</tr>
<tr>
<td><strong>Std.D.</strong></td>
<td>0.16</td>
<td>1.70e07</td>
<td>155.77</td>
<td>5.10e04</td>
<td>1.58e06</td>
<td>1.95e06</td>
<td>0.63</td>
<td>0.20</td>
<td>0.52</td>
<td>0.91</td>
</tr>
<tr>
<td><strong>Median</strong></td>
<td>-1.31</td>
<td>1.08e07</td>
<td>7.90</td>
<td>2.42e04</td>
<td>8.59e05</td>
<td>1.36e06</td>
<td>1.59</td>
<td>1.73</td>
<td>3.77</td>
<td>1.56</td>
</tr>
<tr>
<td><strong>Min.</strong></td>
<td>-1.81</td>
<td>1.95e06</td>
<td>0.13</td>
<td>2.13e03</td>
<td>9.99e04</td>
<td>1.25e05</td>
<td>0.42</td>
<td>1.09</td>
<td>2.10</td>
<td>0.64</td>
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<tr>
<td><strong>Max.</strong></td>
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<td>1.78e08</td>
<td>2040.00</td>
<td>3.89e05</td>
<td>1.25e07</td>
<td>1.68e07</td>
<td>6.79</td>
<td>2.26</td>
<td>5.40</td>
<td>14.26</td>
</tr>
</tbody>
</table>

### Overall, 04/19/2002 to 09/04/2005 (n=2761)

<table>
<thead>
<tr>
<th></th>
<th>M</th>
<th>$M_0$ (Nm)</th>
<th>$E_0$ (J)</th>
<th>$\sigma_a$ (Pa)</th>
<th>$\Delta \sigma$ (Pa)</th>
<th>$\Delta \sigma_d$ (Pa)</th>
<th>$E_d/E_p$</th>
<th>$r_0$ (m)</th>
<th>$r_a$ (m)</th>
<th>$\Delta \sigma_d / \Delta \sigma$</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Mean</strong></td>
<td>-1.25</td>
<td>1.85e07</td>
<td>77.03</td>
<td>5.60e04</td>
<td>1.76e06</td>
<td>2.36e06</td>
<td>1.87</td>
<td>1.69</td>
<td>3.60</td>
<td>1.54</td>
</tr>
<tr>
<td><strong>Std.D.</strong></td>
<td>0.19</td>
<td>2.47e07</td>
<td>321.39</td>
<td>7.14e04</td>
<td>2.00e06</td>
<td>2.43e06</td>
<td>1.18</td>
<td>0.20</td>
<td>0.55</td>
<td>0.71</td>
</tr>
<tr>
<td><strong>Median</strong></td>
<td>-1.31</td>
<td>1.10e07</td>
<td>9.28</td>
<td>2.86e04</td>
<td>1.06e06</td>
<td>1.59e06</td>
<td>1.69</td>
<td>1.69</td>
<td>3.59</td>
<td>1.48</td>
</tr>
<tr>
<td><strong>Min.</strong></td>
<td>-1.81</td>
<td>1.95e06</td>
<td>0.13</td>
<td>2.13e03</td>
<td>9.99e04</td>
<td>8.53e04</td>
<td>0.42</td>
<td>0.93</td>
<td>1.90</td>
<td>0.45</td>
</tr>
<tr>
<td><strong>Max.</strong></td>
<td>-0.36</td>
<td>2.88e08</td>
<td>6350.00</td>
<td>8.40e05</td>
<td>2.22e07</td>
<td>3.24e07</td>
<td>26.36</td>
<td>2.39</td>
<td>6.93</td>
<td>14.26</td>
</tr>
</tbody>
</table>
4.3.2.4.1 Source Mechanisms observed from $E_s/E_p$ ratios

Before summarizing temporal changes, there are some general important points regarding this data that must be discussed. Of significance are the relatively low $E_s/E_p$ ratios that were encountered throughout the failure process. These ranged from 0.48 to 26, however, only 1% of the events had ratios >5, these occurring during localization. The average $E_s/E_p$ ratio was around 1.9, and 4% had ratios < 1.0, (pure tensile failures, Urbancic, 1991). Low $E_s/E_p$ ratios in a mining setting are not uncommon, however, and from other studies it has been observed that up to 2/3 are less than 10, (Gibowicz et. al., 1991; Urbancic, 1991). Another example of low $E_s/E_p$ ratios is presented by van Aswegan and Butler (1993), who note in their evaluation of macroseismic mine induced events, ($M > 1.0$), that ‘slips’, (shear events), were encountered for $E_s/E_p > 1.0$ while ‘pops’, (tensile events), for $E_s/E_p < 1.0$, the statistical mode of their $E_s/E_p$ ratios being 2.5. This indicates firstly that these events are non-pure shear, and have a significant positive dilatation or non-shear component, suggesting a combined shear-tensile failure, (Urbancic, 1991), inline with the concept of fracture creation, and not slip on pre-existing structure. These are expected however, to have a definite shear or double-couple component, due in part to the fact that it was possible to determine a double-couple solution based on first motion polarities, (Chapter 5), indicating that shear motion and slip has to be occurring. The second point to make is that all of the events appear, from the surface, to be very similar in nature and size, and related to the formation of this macrofracture shear structure developed under stress and with confinement.

4.3.2.4.2 Temporal Changes in Strength and Stress Estimates

As can be seen, the average $M$, $M_o$, $E_o$, $\sigma_a$, $\Delta \sigma$ and $\Delta \sigma_d$ all follow almost identical temporal paths (Figure 4.19 and Figure 4.20). However, it can be observed that $E_o$, tends to have the greatest variance and standard errors, the trend being more jagged and affected often by a couple of high energy events. The other strength and stress estimates show predominantly more stable values and may be preferred for temporal monitoring, than straight $E_o$ values.

During 2002 there is a gradual building in the magnitude of the events (-1.8 to –1.02), and this has been postulated to be a function of the gradual loading of the zone through mining induced stress. The average linear elastic maximum principal stress, $\sigma_1$, at the core of the cluster increased from May 2002 to Feb 2003, (just before the extraction of the 462-S5 stope, the final blast was on 02-22-2003, 5 days after the flurry started), by only 3 MPa, to 113 MPa, while, $\sigma_3$,
was relatively stable, increasing from 45 to 47 MPa. The actual mining of the 462-S5 stope created a more significant jump in $\sigma_1$ of 7 MPa, and as previously mentioned may be one reason that this failure propagated relatively quickly, (it must be noted that these are linear elastic stress levels and cannot represent the true stresses that occur once failure of the rock mass has initiated, however they do give an indication of the loading environment). Prior to the major flurry of activity, resulting probably from redistribution of stress from the mining of the 462-S7 stope on the other side of the slot, the magnitudes of the events begin to fall gradually, (Figure 4.19 or Figure 4.25 for more detail), from the peak on January 18, 2003 until the beginning of February 8, 2003, (e.g. a drop in M from $-1.02$ to $-1.11$). This may be a result of the first regional interaction of the events. Prior to this the fractures were relatively isolated in space, and thus required more energy to propagate with free radiation of energy, until they start to interact making fracture propagation easier and radiating less energy due to consummation through increased damage. This is the same time that the PCA technique identifies a relatively stable trend, oriented in the direction of maximum shear stress.

This potential interaction is supported by the change in the events source complexity, (Figure 4.21a), in which for the first period up to just prior to the peak magnitude, the ratio of $\Delta \sigma / \Delta \sigma$ is relatively stable at around 1.0 indicating homogenous fractures. In late December 2002 there is a noticeable increase in the ratio above 1.0, even prior to the drop in the average magnitudes and indicates the start of a potentially inhomogeneous source model. Here it is felt that this is not due to asperity breakage, as suggested for larger magnitude events, (Trifu et al., 1995), but could be due to the more energetic fracturing of rock bridges between adjacent fractures (interaction), causing an increase in the peak acceleration as seen by the increase in the dynamic stress beyond that of the static stress level (Figure 4.20c). This interaction appears to start before the clustering seismic density, (an estimate of potential fracture interaction), is reached. The events were plotted as circular ellipsoids, sized to their respective source radii, and applying the PCA derived orientations for each event, which are similar to their fault plane solutions (Chapter 5). Then sections were taken through the core of the cluster, (here for section 10480 E), to determine the potential fracture network developed. As can be seen in Figure 4.21b for the period prior to 'interaction', and in Figure 4.21c, following intense activity period during 'localization' the fracture network appears to be representative of that suggested by the source geometric complexity. This physically provides a strong confirmation that the change in the source complexity and reduction of the source strength is indicative of interacting fractures developing as a macrofracture structure, based on an en echelon fracture geometry.
Just prior to the peak of activity on February 17 2003 there is a slight increase in the event rate starting on February 8 2003, (Figure 4.14b), which results in a more substantial fall in the average strength and the stress estimates of the events in terms of moment magnitude from –1.1 to –1.35, well beyond the standard deviation or standard errors. This has been postulated as the more regional onset of coalescence and start of localization with the formation of the macrofracture structure as defined by the PCA method. After this point, unstable fracture propagation appears to dominate, the strength and stress estimates staying low and never approaching the pre-localization levels and being dominated by the proliferation of smaller magnitude events/fractures. This suggests that this rock mass has predominantly failed at this point due to increased fracturing and has strain-softened as a result of inferred dilation. Because the distribution of magnitudes, moments, energy and stress parameters are distinctly not normal but more log normally distributed, (Figure 4.23), the observation of the temporal change in the average values is representative of a shift in the population mean, (Figure 4.23, Table 4.3). These overlapping distributions would make temporal variations difficult to define unless this form of statistical data smoothing using a moving average is applied.

Simser and Falmagne (2004) have also identified changes in the mean apparent stress, and a similar shift in the population mean, for a rock mass that was originally highly stressed producing high apparent stresses, to one which was physically destressed by cutting off the maximum principal stress through mining of a large destressing mass blast (Figure 4.24). They equated the drop in apparent stress and energy index, (ratio of the measured energy of an event to the average cluster energy from the log E₀ versus M₀ scaling relationship), to the reduction on induced stress. Here the reduction in stress estimates is felt to be due to failure of the rock mass through brittle strain softening behaviour. In their study, there was an observed drop in the average apparent stress from 6 kPa to 2.17 KPa, a difference of around 4 kPa or a 63% reduction (Figure 4.24), and from linear elastic modelling the mass blast produced a reduction in $\sigma_1$ from 75 to 35 MPa, or 40 MPa reduction in induced stress, (Coulson et. al. 2002). Here at the Golden Giant under higher stress conditions, (also using a higher frequency range of microseismic system), the apparent stress reduced from 94 kPa to 42 kPa, a difference of around 52 kPa, or a relative reduction of 55% indicating the potential for a similar reduction in induced stress by the natural processes of failure through strain softening of around 35 MPa. Note the linear elastic stress drop from the maximum at the point of disassociation (120 MPa) to the minimum 1995 initiation state (84 MPa) is 36 MPa (Table 4.2).
Figure 4.23 Comparison of distribution of Seismic Moments, for pre-interaction, coalescence/localization and post localization (disassociation), indicating a shift in the population mean. (a) Seismic Moment and (b) Log(Seismic Moment).
Figure 4.24 Identification of a shift in the population mean of Apparent Stress at Brunswick Mine (after Simser and Falmagne, 2004). Note Upper right view is a plot of the events apparent stress and location in the South Bulk Ore zone before the mass blast, while the upper left denotes the position of the mass blast and a sample of the event recorded in the period immediately after. The distribution curve for the period following is based on a greater sample than displayed.
With regard to the significant increase in ellipticity found from the PCA method, this appears to signify full formation of the macrofracture structure. It is interesting that around the peak ellipticity there is a minor but definite increase in the $E_s/E_p$ ratios, (2% > 5 and up to 26), suggesting that fracture interaction may allow for more pure shear type events, or for sliding on the interconnected fracture surfaces already formed during this period. This could suggest that the low $E_s/E_p$ ratios (< 5) relate primarily to fracture creation, while during localization the higher $E_s/E_p$ ratios may relate to slip on the previously formed fractures themselves or the creation of larger fractures. Following localization and prior to the point of disassociation the $E_s/E_p$ ratios fall back to a relatively low value (average ~ 1.8).

Based on a more limited data set, for the analysis microseismicity in a sill pillar prior to a large 2.9 Mn event at the Strathcona Mine, Trifu and Urbancic (1996), also noticed a minor decrease in the dynamic stress ($\Delta \sigma_d$) of 2 MPa, (similar to this case), corresponding to a minor increase in the ellipticity from a PCA of the clustering locations. Ellipticity raised from 4 to 7 and then dropped to 4 after 130 events, with an increase in the dynamic stress to the previous levels. The authors suggested that this was due to the coalescence through failure of joints to form a larger failure plane on which the 2.9 Mn event was initiated. It is interesting to note that although they draw a conclusion that the PCA trend and fault plane solutions correspond roughly to the predominant joint set, examination of mapped structure was different by 15° to 20° in dip and 25° to 40° in strike. Here it is felt that this is not the coalescence of joints but the coalescence of fresh fractures in the direction of maximum shear (Chapter 5).

The point of disassociation identified from the PCA is not clearly seen in a change in the source parameters apart from a minor increase in the moment and stress estimates to a relatively stable value following the second point of disassociation. This could be construed as a result of stress shedding from the region that has failed, however, this also occurs at the same time as the final blast of the 450-Q1/Q2 stope, which would also potentially cause a redistribution in stress, although it was felt that this pillar had failed prior to mining this lift. It should be recognized that although certain regions become aseismic due to possible large dilations, (>50 mm based on experience at Williams Mine), thus affecting the group behaviour and fracture interaction, the fracturing process away from these areas appears to continue. However, this fracturing seems to occur at the event strength and stress levels associated with a post peak rock mass, in which the fractures are interacting. Further, with regard to the point of complete aseismicity that occurs in May 2005 for the core of the cluster around the 460-S2 stope, there appears to be no change in the source parameters prior to this, with the averages staying at
their post peak levels, although a slight increase is seen following this but for events mainly associated to the far eastern abutment (Figure 4.25b).

4.3.2.4.3 Temporal Changes in Source Size

The temporal change in the source radius was found to be not as anticipated, as during loading the average source radius, \( r_0 \), decreases with an increase in the moment, (Figure 4.19 and Figure 4.22), and is to some extent is counter intuitive, for as the stress increases then the size of the fractures might also be anticipated to increase as here, based on the linear elastic modelling, confinement is not increasing substantially. What is compelling regarding the source radius is that the apparent radius, \( r_a \), based not on the corner frequency and being model independent, exactly shadows the source radius through temporal changes, and suggests that these two parameters are not mutually independent variables in this case. This behaviour of the source radius during loading is distinctly non-self similar behaviour, (i.e. self similar behaviour is defined as an increase in the magnitude resulting in a increase in the source radius for the same stress drop and identified for earthquakes). Non-similar behaviour of small earthquakes and mine induced microseismicity has been interpreted as a source effect, involving either an upper limit to radiated frequency due to either a characteristic fault length (joint persistence), or dependence of stress on the seismic moment, (Gibowicz et. al., 1991).

Here it is interpreted as a function of the later for the pre-coalescence phase, the dominant frequencies being potentially a function of the increasing stress regime. Andrew’s (1986) notes that in the determination of the corner frequency by the integration of the events displacement and the velocity spectra, (Appendix A3), is by definition the predominant frequency in random vibration theory. This might infer that for a higher stress, there may be greater clamping and hence a stiffer rock mass, resulting in higher frequencies, and thus smaller source radii, similar to the analogy of tightening the membrane screws on a percussion drum. In the post peak the rock mass is less stiff, resulting in lower frequencies, hence larger source radii. Physically, if the definition of source radius for these low \( E_s/E_p \) ratio events is correct, this also might indicate clamping of the rock mass, (an increase in the normal load), reducing the effective fracture surface, but increasing the strength of the events or energy radiated from the fractures as they develop prior to interaction.

Prior to the start of the peak flurry of activity the source radius increases relatively dramatically with the decrease in moment, (Figure 4.25a) (suggesting a larger rupture surface available), and after coalescence appears to increase and decrease with the rebound of the seismic moment, suggesting short-lived self-similar behaviour as the localization forms (Figure 4.25a).
However, after this, around the point of disassociation, there is no correlation in the relative changes of these parameters (Figure 4.25b). The average source radius appears to become unstable at a slightly larger value following localization and disassociation, which could be interpreted as the availability of larger rupture surfaces through fracture interaction. Based on the dominant frequency scenario, this could also be interpreted as a result of slightly lower frequencies radiated due to the failed and dilating ground, (i.e. lower stress state, and lower
stiffness of the failure rock mass), resulting in lower dominant frequencies. Overall, the
maximum source radius could be associated with joint spacing, as fractures that form acutely to
the A-set joints may terminate at a dominant joint. However, as the general spacing of A-set is
< 1m, this means that these fractures would have to cross A-set joints. Fractures with a
diameter of 3 to 4 m, also seem reasonable based on observation of back failures.

Overall the source radius behaviour is complex and is not as easily interpreted. Although the
overall source dimensions based on the Madariaga model are reasonable, the shortcomings of
this simplified 2D quasi-dynamic circular fault model for a pure shear rupture mechanism are
that the model does not account for the high frequencies that result from an inhomogeneous
rupture surface, extensile fracture generation or the influence of confining stress on the rupture
plane. This is an area requiring more definitive research, and although Cai et. al.’s (1998)
tensile model captures some of these shortcomings, the resultant source size from this model
tends to be too small, (source radii are ~ 20% of the Madariaga model), to suggest interaction,
coalescence and yield to occur over the volume required to cause the observed failure
(Appendix A1). Thus, it is important to recognise that the source radius or corner frequency,
(predominant frequency), is a function of not only the rupture size but also the stress conditions
relating to the stiffness of the rock mass.

4.3.2.4.4 Spatial Independence of Temporal Source Parameter Variation

The question arose as to whether the temporal variation in these source parameters could
have been a function primarily of their location. Events occurring on the far eastern abutment,
were noted to be generally larger or triggered more sensors and were more dominant prior to
the major flurry of activity. To investigate this the events were positionally filtered to only
include those events West of the raise, (i.e. West of section 10490E, see Figure 4.12a, for
location of the raise), occurring in stope regions 460-S2 to 460-S4. As can be seen from Figure
4.26, the general trends are similar to the analysis of the entire cluster with an increase in
strength and stress estimates during loading, at a reduced rate, up to the point of interaction,
followed by a significant drop in strength and stress estimates, the source radius being
inversely proportional until localization.
Figure 4.26 Temporal variation of source parameters for only events West of the raise (Sec 10490E) in the EOS – Cluster for, (a) Moment magnitude, (b) Seismic Energy, and (c) Source Radius. Note the trend follows the pattern as for the entire cluster, however, the peak Magnitudes and Energies, prior to Yield are reduced (Total # of event is 1782 versus 2761 for the entire data set).
The main difference is that the average peak moment magnitudes, and energies are not reached prior to the point of interaction or ‘yield’, which does indicate that higher magnitudes events are present closer to the abutment, as would be expected from the higher stresses associated with this region. However, as with the application of the PCA method to entire cluster, there is no real basis for removing these events from the analysis and they form a description of the fracturing process for this pendant pillar as a whole.

4.3.2.5 Seismic Scaling Relationships and b-value

4.3.2.5.1 Evaluation of Scaling Relations

As previously discussed from the comparison of the temporal changes in seismic moment versus source radius, non-similar behaviour appears to prevail. The classical basis for self-similar behaviour, (Aki, 1967), of small to large magnitude events is determined from the scaling relationship by plotting the seismic moment versus the source radius logarithmically with bounds of constant static stress drop. It is well recognised that large magnitude pure shear events, (M > 0 and Es/Ep > 10-20), follow a self–similar scaling relationship showing an increase in source radius with a complimentary increase in the seismic moment for a constant stress drop and is used as a semi-predictive tool in earthquake seismology allowing simplification of the source model to be the same for small and large earthquakes. However, for microseismic events, (M < 0, Es/Ep < 10), identified in many mines, researchers have found that these events typically follow non-similar behaviour with stress drop a function of the seismic moment over a restricted source radius range, (Urbancic, 1991; Gibowicz et. al., 1991; Mercer, 1999), as discussed previously. In order to more thoroughly investigate this and see if there are identifiable changes from pre-interaction to post interaction, this relationship was investigated for this cluster by plotting the moments versus source radius, for 2002 and 2003, and by identifying the events on a monthly basis to observe any changes (Figure 4.27a and b). It can be seen that there is no significant change in the non-self similar behaviour between the two years. Although the averaged values appeared to show self-similarity during the localization phase, (see previous), this is not recognisable when the raw values are plotted, and infers that non-similar behaviour predominates during the failure process, as at any one time the events cross lines of constant stress, indicating the dependence of the stress drop on the seismic moment. This may indicate that the short self-similar behaviour observed following localization, could be a coincidence of the averaging, as the seismic moment is varying very little during this time.
Figure 4.27  Scaling relationships. (a) and (b) Seismic Moment, \( M_o \), versus Source Radius, \( r_o \), with lines of constant Static Stress Drop. (c) and (d) Seismic Energy, \( E_o \), versus Seismic Moment, \( M_o \), with lines of constant Apparent Stress and trend line.
What is noticeable however, is a reduction in the seismic moment and slight increase in the source radius from 2002 to 2003 as was identified more readily from the average temporal variation. Also, the single large events recorded during July 2002 stands out as an outlier, significantly larger in moment and energy than any of the other events.

Additional scaling relations have been determined for the total seismic energy versus seismic moment, compared to lines of constant apparent stress for events in 2002 and 2003, again differentiating events by month (Figure 4.27c and d). As can be seen, the overall trend line, which is the same in both plots, but not a regressed line, crosses lines of constant stress indicating non-similar behaviour. In 2003, as identified from the temporal analysis, there is a reduction in both energy and moment as a result of the localization, with apparent stress also reducing. The energy index, (Mendecki, 1993), was not calculated for this data set, primarily due to the identified greater variance in the observed energy values over the other parameters. Since the data appears to be distributed about the trend line during both years and apart from the shift down along the trend line in 2003, it is anticipated that there would not be a significant change.

In other the source parameter studies of sill pillar regions in Canada such as at Strathcona Mine, (Urbancic, 1991), and Creighton Mine, (Mercer, 1999), using a similar monitoring frequency range, with similar mining induced stress levels, the source radius, (based on the Madariaga model) was determined to range from 0.9 to 2.1 m identical to the range seen here, for microseismic events having similar $E_s/E_p$ ratios, (generally < 10), with almost identical scaling relationships, (for $M_o = 1e5$ to $1e8$ Nm, and $M_o = 1.2e7$ to $1.8e9$ Nm respectively), showing non-similar behaviour. Additionally, in a more limited but controlled study by Gibowicz et. al. (1991), at the Underground Research Laboratory, (URL), Canada, the analysis of microseismicity developed within 5 to 10 m of the shaft walls during the sinking, also determined non-similar behaviour for small events of magnitude between $-3.6$ to $-1.9$ ($M_o = 3.98e3$ to $1.4e6$ Nm), with source radii, (Madariaga Model), ranging from 0.3 to 0.7 m. Gibowicz et. al., (1991) have also noted non-similar behaviour at other mines in South Africa and Poland. Another study at the URL, ‘the mine by experiment’, (Collins and Young, 1999), showed the opposite behaviour compared to the other studies, indicating self-similar behaviour for very small seismic events between a magnitude range of $-4.2$ to $-2.9$, ($M_o = 5e2$ to $4.4e4$), and source radii, (Madariaga Model), ranging from 0.13 to 0.5 m. These events however, differ from the other studies in that they generally exhibited $E_s/E_p$ ratios > 10, indicating more pure shear type events where self-similarity might be observed. In a similar study of the mine by experiment at the URL, (Cai et. al., 1998), over a broader range of $E_s/E_p$ ratio from 1 to < 20,
the source radii, (Madariaga Model), was found to be between 0.12 to 1.5 m, however, no
determination as to the scaling relation was made. Thus, for environments involving $E_s/E_p$ ratios
> 10, generally for earthquakes and large magnitude mine induced events far from openings,
where confinement inhibits dilation, it is expected that self-similar behaviour might be obtained.
However, for environments that exhibit predominantly $E_s/E_p$ ratios < 10, in which a significant
tensile non-shear component is present indicating a moderate confinement regime, then non-
similar behaviour is often observed.

Urbancic (1991) suggested that the apparent non-similarity in scaling was attributed to a high
level of clamping on intersecting pre-existing structural features. This is postulated as due to
increased stress concentrations acting to resist movement along features, and a combination of
the characteristic fault length and stress dependence on the seismic moment. Here, because of
the lack of pre-existing features with the orientation identified from the PCA technique and fault
plane solutions, (Chapter 5), these microseismic events are interpreted as fracturing of the
intact rock, and it is felt that much of the microseismicity that occurs in the mining environment
is attributed not to slip on pre-existing features but the development of fresh fractures, which in
some cases may be controlled by the joint structure, but here is controlled more by the stress
regime. This means that we could be dealing with two entirely different processes; in
earthquake and macro seismic mine events, shear slip (Mode II)$^4$ is occurring predominantly on
large pre-existing structural features (faults) in a system which is controlled by friction, while at
the microseismic level, the process is predominantly through the actual fracturing of the rock
with slip occurring on the newly created fractures, (Mode I and Mode II). In the later case,
direction of propagation appears to be controlled through the cohesional strength of the
material (Chapter 5). The source size is a function of the confinement, material properties of
the rock and the rock mass structure. That being said, one cannot ignore the fact that the
system and observable frequency range can influence the observed behaviour, and limiting the
frequencies due to noise effects will undoubtedly influence the maximum observable source
radius. However, here the maximum source radius observed during the monitoring of the entire
shaft region was found to be 2.96 m, (corner frequency, $f_c = 660$ Hz). Theoretically, the largest
potential observable source radius could be around 6.5 m, (based on $f_c = 300$ Hz, note the low
frequency cut-off is 250 Hz), indicating that there is a finite rupture dimension here that is not a
function of the system.

$^4$ Mode I = fractures developed under tensile conditions or opening mode, Mode II = fracture developed under in-plane pure shear
conditions or sliding mode, and Mode III = fracture developed through anti-plane shear or tearing mode.
### 4.3.2.5.2 Evaluation of b-values

Traditionally, b-values, which are a measure of the slope of the G-R, (Gutenberg-Richter, 1944), log-linear frequency-magnitude distribution, \( \log(N(M)) = a - bM \), have been commonly used in earthquake seismology as a potential precursory parameter for large fault slip events, (Urbancic and Young, 1992). Additionally, b-values of different clusters have been used for hazard analysis to identify the potential for large seismic events, i.e. a low b-value, (large negative value), corresponding to a high hazard, as there is an increased likelihood of large events, (Hudyma et al., 2002), or to characterize differences in event frequency and magnitude between clusters of events based on stress regime, (Young and Collins, 2001). In the study of acoustic emissions during uniaxial and triaxial laboratory testing, b-values have been used to determine changes in the rate and amplitude of events during failure, (Scholz, 1968; Lockner, 1993; Thompson et al., 2006).

The overall G-R frequency-magnitude distribution for events in the EOS cluster from April 18 2002 to May 8 2005, is plotted in Figure 4.28a for cumulative and non-cumulative frequencies, while differences in the distribution during the periods of pre-interaction, localization and post localization are shown in Figure 4.28b, (see Table 4.3 for time range). The main reason for analysis of cumulative versus non-cumulative is to identify if the dataset is statistically complete, which would mean that the slopes of both analyses are the same, (Trifu pers. com., 2008). Also, non-linearity of the slope may suggest non-similar scaling relations. Comparison of the non-cumulative to the cumulative distributions indicates similar slopes in the pre-interaction stage, but slightly different slopes following localization. During all periods these slopes are non-linear, the non-linearity being amplified in non-cumulative distribution. This non-linear behaviour further suggests that the stress regime has a non-similar scaling behaviour, (Urbancic and Young, 1992), as identified in the previous section. The difference in the slopes (cum. to non-cum.), post localization might suggest a statistically incomplete dataset. When dealing with non-similar behaviour, this however may be an incorrect assumption. The main objective here is to see if there is a reasonable temporal change in the b-value, and whether there is agreement with identified changes noted from the PCA technique and the key source parameters.

In order to evaluate the temporal changes in the b-value, a 50 event moving window, (sample every 50 events and shift 1 event plotted against the date/time of the last event of the sample), was used over the magnitude limits of −1.3 to −0.4, to evaluate the change in the predominantly linear portion of the slope based on the entire dataset (Figure 4.28a).
Figure 4.28 (a) Log-linear Frequency magnitude distribution, and magnitude range over which b-values (slope of the line) were calculated for all data in the EOS cluster (b) Frequency –magnitude for data for Pre-Interaction, Localization and Post-Localization (see Table 4.3) (c) Temporal variation of b-value for a 50 event moving window (backward looking) for cumulative and non-cumulative b-values, (d) Event rate for the EOS cluster.
As anticipated the overall temporal variation is similar to the observed changes in the source parameter strength and stress estimates (Figure 4.28c). However, the most significant decrease in the b-value from 1.8 to 0.4, (cumulative), does not occur until the start of an increase in event rate on February 12 2003, (Figure 4.28d). Although a minor change in the b-values can be noted in late December 2002 from the non-cumulative plot, it is not statistically as significant as say the change in source geometrical complexity. The sudden decrease in the b-value again identifies the increase and proliferation of smaller magnitude to large events, and the b-value during localization is on average around 1.0, but fluctuates in a similar fashion with the source parameters. Following the second point of disassociation the b-value increases to a level of around 1.3 to 1.5, but as with the source parameters not achieving the pre-peak values.

Legge and Spottiswoode, (1987; cited Hudyma et. al., 2002), have noted that seismic sources that are dominated by fault-slip mechanisms, (pure shear), tend to have low b-values (< 1.0), while seismic source that are dominated by stress-related failure mechanisms tend to have intermediate or high b-values (generally > 1.0). Here during the failure process the b-values transcend these ranges, and it is felt that this generalization of source mechanisms may be an over simplification of observations, considering that here we do not see a significant sustainable increase in the E_s/E_p ratios (>10) following localization. From a hazard perspective, the likelihood of a large magnitude shear slip event, (M > 0), would be expected in the pre-interaction and loading period based on the larger b-value. Although no large shear slip event occurred, the largest event recorded in the cluster did occur during this time frame, (i.e. July 20 2002, M = -0.18), but based on the E_s/E_p ratios and its proximity to the footwall drift, this event still has a large tensile component, and is more tensile in nature.

Thompson et. al. (2006) in their study of acoustic emission (AE) during the failure of triaxial specimens, determined b-values based on the frequency-amplitude distribution of AE, (note: for most ultrasonic high frequency sensors, the exact response of the sensor in terms of displacement, velocity or acceleration is not known, so a true magnitude cannot be calculated). They identified generally a slight increase or no change in the b-value during the loading phase up until yield prior to interaction, (e.g. 2.4 to 2.5), followed by a gradual drop in b-value during interaction and yielding in the pre-peak, (e.g. 2.5 to 1.5), with a significant fall during fracture propagation in the post peak during coalescence and localization, (e.g. 1.5 to 0.5), for tests that were not AE feedback controlled. This response is similar to that identified here at the rock mass scale, prior to disassociation.
4.4 Summary and Discussion

Often in mine evaluations of microseismicity, the identification of temporal changes in source parameters is hampered by inadequate data due to either relatively large arrays or limited analysis of source parameters. Complicating the analyses, there may be a mixture of failure modes, such as tensile spall or axial splitting created in stope pillars and adjacent to openings, or the result of caving of excavation backs causing a mixture of processes of probably shear driven stress fracturing and tensile fracturing related to the readjustment following caving. Here the main key to the relative success in identification of temporal changes in source parameters is: firstly, a well constrained and dense microseismic array surrounding the volume of interest; secondly, a stable mining geometry with a gradual building of the induced stress regime to a stress level to causing complete failure of a large volume of the rock mass. The geology and rock mass is relatively simple, with no major structures such as faults or secondary shears in which slip on large pre-existing features can create readjustments in the local stress regime. Finally the mode of failure analysed here is very specific and relatively pure in terms of the formation of a macrofracture shear structure. This failure forms under theoretically moderate confinement, which is controlled through a combination of induced stress and excavation geometry, without any caving, but probably with allowable dilation (bulking) occurring in the back of the filled stopes below the floor of the 4620 level drift above, and was monitored during the complete failure process from initiation to aseismic behaviour. The PCA technique, combined with an estimation of the yield strength from the clustering event density, was instrumental in defining the potential strength state of the rock mass and direction of failure, so that temporal changes in source parameters could be investigated further to increase our understanding of the potential fracturing process.

4.4.1 Limitations of the PCA Technique

There are limitations to the effectiveness of the PCA technique. The two case studies investigated, (Williams and Golden Giant), represent similar stress driven failures under relatively confined conditions, which were geometrically controlled in lateral extent, and are relatively pure, predominantly single mechanism failures. When the PCA technique was applied to a cluster of events in the Q1/Q2 pillar, observation of a relatively stable vertical north-south trend was identified, which was interpreted to be potentially similar to axial splitting. However, as the region was complicated by in situ raises, which were also seismicity attractors, the group behaviour was probably controlled by combinations of the more vertical pillar geometry and the
raises. The analysis then becomes more complicated by the volumetric distribution of seismicity, and significant increases in the ellipticity were not seen.

Also, on the west side of the destress slot above the 4620 level, the analysis was complicated it is thought by different failure mechanisms created through a similar shear style stress driven failure for the far western part of the cluster, but complicated by another mode of failure as a result of potential tensile extension fracturing parallel and adjacent to the destress slot and also above the back of the caved Q4/5/6 region. This makes the PCA derived planes less stable and difficult to interpret, and was not included in the analysis here. The average $E_s/E_p$ ratios for the events in this cluster had approximately 20% with ratios less than 1.0 indicating a tensile mechanism for a large proportion of the events.

An additional limitation to the PCA technique is expected if the events are transient in time due to a progressively changing geometry or stress shadowing by mined excavations, or if extensive caving occurs creating mixed modes of mechanisms. In the Williams study it was noted that the distribution of events was quite sensitive to sudden dilations (bulking) in the immediate back of the drift below the cluster, and to potential stress redistribution created from portions of the sill pillar failing. This is not to say that the PCA technique in these cases cannot give valuable information regarding the three-dimensional changes in group behaviour, but the interpretation will not be as straightforward as identified in this research for the relatively pure macrofracture shear type failure identified.

In order to obtain significant stability and identifiable changes in the trends, the failure has to create enough data to be statistically significant, (at least 500 events). Additionally, the failure needs to occur over a larger area than the location resolution of the system, in this case at least 5 times the location accuracy, (4 m), relating to 20 m, the diameter of the spatial window, $D$, determined from the event inter-distances.

The best method for identifying the trends of the events group behaviour is by plotting the events on lower hemisphere stereographic projections. In plotting the mean strike and dip of the population densities with time, caution should be noted for trends that strike North-South, switching by what appears to be $360^\circ$. In these cases observation of the dip and dip direction of the PCA derived planes is preferred. Using a continuous central moving PCA, (CMPCA), confirms more readily the points in time of temporal variations in the trends, however, caution should be used in interpreting the exact values of the mean orientation when there is a mixture
of different trends, the linearly averaged trends being generally less in strike and dip than observation of the mean pole concentration on the stereographic projection.

4.4.2 Limitations of Source Parameters

One key limiting factor in the evaluation of the source parameters was the relatively small number of triaxial sensors, which also tended to have a large amount of low frequency noise, probably generated as a function of electromagnetic radiation (EMR) from mine installations such as vent fans, electrical sub-stations and general noise that is present in the mining environment. This required limiting the low frequency bandwidth to a level that could affect the determined source radii and magnitude of the strength and stress parameters. However, based on other research and the respective energy levels identified here, non-similar behaviour is still anticipated. Trifu and Shumila (2006), however, have identified that not accounting for limited frequency bandwidths and the presence of noise in recordings can result in an underestimate or overestimate of spectral levels for large and small magnitude events respectively. Here corrections would potentially make the temporal changes identified more distinct. Although exact magnitudes could be marginally affected over the frequency range measured, we are more interested in the ‘relative’ average temporal changes and not the exact magnitudes. Ideally, more triaxial sensors spaced a distance of 100 m from the observation area may have greatly improved estimation of the source parameters, and allowed for potential analysis of the moment tensors, but even with this basic array the source parameters were able to give a useful insight into the fracturing process.

4.4.3 Linear Elastic Stress Modelling

This brittle strain-softening failure process is distinctively non-linear, and can only be accurately modelled using non-linear continuum, discontinuum or combined models, (FEM/DEM – Finite element/discrete element). Without further verification of post–peak parameters calibrated to physical dilations, such models can give drastically variable results, as in general most models become very unstable when a brittle strain-softening state is assumed, (Chapter 5; Andrieaux, pers. comm., 2007). However, linear elastic three-dimensional modelling can be used as a first approximation, in combination with the Hoek-Brown brittle failure parameters defining damage limits, or limiting factors of safety, to obtain a rough idea of the region that is expected to yield. In these studies, (Williams Mine and Golden Giant Mine), fracture initiation, has been found to start around an average deviatoric stress ($\sigma_1-\sigma_3$) of 0.3 to 0.37 of the uniaxial compressive
stress ($\sigma_c$), with coalescence and localization occurring at virtually the same stress levels. The point of disassociation in both the Williams and the Golden Giant cases was found to occur at only slightly higher stress levels of 0.375 to 0.4$\sigma_c$ respectively, (an increase in elastic deviatoric stress of around 8 and 5 MPa respectively). As identified in this Chapter, because of the generally slow, almost monotonically increasing loading rates, the difference in a required stress level to pass a damage limit of 0.3 to 0.37$\sigma_c$, can result in a predictive yield over a range of 3 years of mining. However, it should be pointed out that this is based on a limited number of stress history cases, and it was identified by Diederichs et. al., (2002) in the study of the stability of 50 secondary pillars, that cases were not always consistent with the damage limits. Other factors that can greatly effect the prediction of yield based on these limits are the discretization of the numerical model (although here efforts were made to use two different boundary element models to obtain elastic stress estimates within 2% of one another), accurate representation of the far field stress state and estimation of the uniaxial strength of the material. The later of these can, in some cases vary considerably, effecting the estimate of the damage limit. The other major issue with using linear elastic models is that they cannot allow for failure and stress shedding, and in both studies it must be recognised that regions outside of the areas analysed were also failing. Once fracture coalescence/ localization occurs the process also becomes non-linear. However, that being said the linear elastic modelling is essential in providing insight into the potential loading conditions, and when combined with observational data at individual sites, is useful tool for a first approximation of rock mass yield and failure to a postulated residual strength.

As identified in Chapter 3, the average confinement that was calculated for the core regions of the seismic clusters is relatively high, placing the stress path below the brittle-ductile line of $\sigma_1/\sigma_3 < 3.4$, indicating that potential ductile or strain hardening behaviour may occur. The response of the rock mass, observed through reduction of mining induced stress to a level below the initiation stress (i.e. no failure or squeezing of boreholes and easy drilling of the stope with no stress effects), however indicated apparent strain-softening behaviour. Direct interpretation of Mogi’s behavioural classification when using linear elastic modelling may therefore not be applicable. Regional failure could significantly affect the redistributed stress regime and there is some evidence, (Hoek and Brown, 1980), to suggest that a lower limit of $\sigma_1/\sigma_3 < 2$ might be more applicable to disturbed or fractured rocks. This needs to be researched in greater detail.

An alternate interpretation as mentioned previously is that the formation of this macrofracture shear structure occurs under ductile conditions, creating shear banding at the centre of the
confined pillar zone. There is a change in the strength state, from ductile to brittle due to an overall change in the pillar geometry from short and squat creating significant confinement and ductile behaviour to a more slender pillar with brittle behaviour (Kaiser pers. comm., 2009). This is not a change in the material behaviour but a change in the strength state due to geometrical influences.

4.4.4 Large Magnitude Events

At Golden Giant, unlike the Williams mine, there were no large magnitude (M > 0) seismic events, although the theoretical stress level was equivalent and higher than the stresses identified in the Williams footwall failure. One reason that no large events occurred may be due to the relatively smaller region in which the failure and mining redistributed stresses occurred in. In the Williams sill pillar region the gradual failure of the footwall region and mining of sill stopes from east to west occurs over a distance of 300 m. This causes the stress to be redistributed from East to West, creating a large but gradual stress front. This stress front may act on larger pre-existing structures in the footwall, causing stress build up on asperities and resulting in classical pure shear slip failures as occurs in earthquake seismology. At the Golden Giant, particularly in the shaft pillar region, the mining induced stress is very gradually redistributed to either side of the destress slot to the abutments over a smaller area by the expanding pyramid mine sequence, and produces relatively short stress front in comparison. Additionally, the region is not confined by mined geometry above as in the case of the sill pillar at Williams, and allows for stress to be more easily distributed to a large volume above.

On a broader scale in the history of Golden Giant, other than a number of small pillar related bursts, (< 1.0 Mn), only one large event, (1.9 Mn), was experienced in August 2001, high up on the 4700 level Q9 area related to the #1 fresh air raise (McMullan, 2001). One key difference between the mining sequence utilised at Golden Giant than at the neighbouring mines is that the mine predominantly practiced mining by end slicing causing a gradual redistribution of stress to an abutment, and not by mining primary-secondary stopes which create pillars. Although notable pillars were created between mining blocks, which were found to be problematic from stress later on in the mine life, (e.g. Q14/15 pillar and 6/7 West pillars, Coulson and Rizkalla, 1998), these were generally mined from the bottom up with no mining above to confine stress. However, that being said, the Golden Giant mine may have been fortunate in terms of regional geology, and managed to mine some of the largest volumes prior to regional stresses being great enough to cause the large magnitude events, as at the Williams and David Bell mines. Although, a large event did not occur in the shaft pillar region
during the mining of the destress slot, (beside the –0.18 M event in the loading pre-interaction stage), based on all the evidence and similarity of the stress driven failure to the Williams mine, it would not have been unexpected if one had occurred in the abutments, as even with the possible explanations above, our understanding of the true genesis of these large events is very limited.

4.5 Conclusions

The PCA technique combined with an estimation of the yield strength from the clustering event density was instrumental in defining the potential strength state of the rock mass and direction of failure, so that temporal changes in source parameters could be investigated further to increase an understanding of the fracturing process, and fracture interaction during coalescence. The failure analysed here is relatively ‘pure’ in that it is a stress driven failure of a confined and geometrically stable region, and based on three-dimensional linear elastic modelling is loaded almost monotonically in the horizontal direction.

- The application of the PCA technique to this cluster showed that a stable trend [075°, 50°] in PCA derived planes is not identified until the start of fracture interaction, determined primarily from an increase in the source complexity postulated as interaction of fractures, the high energy acceleration being attributed to the failure of intact bridges between fractures. This occurs well before a critical clustering density engulfs the region, (which is a rough estimate based on the source radius), and a drop in strength and stress estimates is noticed. This is postulated as due to isolated groups of fractures starting to interact. Prior to this pre-interaction phase no stable PCA trend is formed, the group behaviour being relatively volumetric and dispersed, while source strength and stress estimates appear to increase with loading while source radii decrease, indicating potential clamping of predominantly tensile rich fractures or a increase in the stiffness of the rock mass. The scatter in the PCA derived trend during this period does not necessarily mean that fractures are not propagating in the direction of the later formed dominant trend, which corresponds to the direction of maximum shear (Chapter 5), but the overall group behaviour is not localized and cannot be resolved with the PCA technique.

- Once more significant interaction and coalescence starts, (the point of true yield), the average source strength and stress estimates begin to fall, (i.e. M → −1.02 to −1.10 and $\sigma_a \rightarrow 0.135$ to 0.11 MPa), while the average source radii start to increase, ($r_o \rightarrow 1.58$ to
1.61 m). This drop in strength occurs as fractures interact more, requiring less energy to propagate and also radiating less energy; however, a larger fracture surface is potentially generated, this being non-similar behaviour. After a critical density of fractures is formed, due to an increased rate of formation of smaller magnitude events, a more significant reduction in the strength and stress estimates of events, (i.e. $M \rightarrow -1.1$ to $-1.35$ and $\sigma_a \rightarrow 0.11$ to $0.032$ MPa), which is statistically significant and beyond the standard errors, occurs indicating greater interaction and coalescence. It is at this point that strong localization begins with a significant increase in the ellipticity of the PCA derived planes (> 25), with the dominant PCA derived trend becoming more stable. The average source radius at this point also increase significantly, ($r_o \rightarrow 1.61$ to 1.75 m). This is postulated as the formation of a macrofracture shear structure, which is not a single feature, but interpreted as a process zone of en echelon fractures of the intact rock, and based on fault plane solutions is probably not shear slip on pre-existing joints, as the density of the closest features, the C-set joint, is not present and their orientations are much flatter. The idea that this is predominately fracturing of intact rock is also suggested by the relatively low $E_s/E_p$ ratios (< 2.5), indicating a strong tensile component to the mechanisms. During the period of high ellipticity, (strong localization), the amount of shear energy, or $E_s/E_p$ ratio, appears to increase with the largest values being recorded during this time, (average $E_s/E_p \rightarrow 2.2$ to 2.9, max.= 26), and may indicate that more shear slip style movement is occurring on the structure of interconnected fractures. Based on similarities to laboratory analysis of acoustic emissions (Lockner, 1993; Thompson et. al. 2006) this is interpreted to represent the start of post-peak behaviour, which based on the observations of the physical reaction of the rock mass to subsequent mining, is interpreted as strain softening behaviour.

- The key significance of the drop in the source strength and stress estimates prior to the flurry of activity, indicates that pre-cursory changes relating to interaction and yield could potentially be observed, suggesting the prediction of this failure. Note that we are suggesting the prediction of impending event/fracture localization and not the prediction of large magnitude events. However, the gradual drop in strength and stress estimates prior to the start of the flurry on February 8 2003, is only marginally statistically significant based on the standard errors, (i.e. a drop in moment magnitude from $-1.02$ to $-1.10 = -0.08$ while the standard errors are ±0.04). However, the measure of the source complexity appears to show a more significant increase prior to this, a result of higher accelerations being generated through energetic sub-events which may be related to the interaction of fractures postulated through the failure of rock bridges.
between fractures, and should be a parameter that is investigated more thoroughly in other studies. If we consider that this failure at the rock mass scale is very similar to observations of confined failure of small scale laboratory tests, it is not surprising that these precursory observations exist, as noted by Martin (1997) from laboratory testing of Lac du Bonnet granite, “Although the peak strength represents the ultimate capacity of the sample, the damage process, which ultimately leads to brittle failure, initiates long before the peak strength is reached”.

• From the investigation of the PCA technique at the Williams mine (Chapter 3), it was identified that a breakdown in the stable dominant trend of the associated group behaviour occurs when patches of the rock mass become aseismic, and were found to correlate to large displacements through shear or dilations, (10 to >50 mm), of the rock mass measured at the periphery of cluster of events or macrofracture structure and at depth, (7.5 to 9 m), away from the excavation. It was postulated that this was the point at which regions of the rock mass had completely failed and reached their residual strength, the large associated dilations causing a strong strain-softening behaviour with redistribution of stress, and allowing subsequent easier mining behind this failed zone. This was termed the point of disassociation, and in this study, although no direct displacements were measured, aseismicity in the region also at the periphery of the cluster below the 4620 level with combined floor heave of the drift were noted at the time of disassociation and it is felt that at this point a substantial volume of the rock mass had completely failed to its residual strength.

• After the point of disassociation the PCA trend becomes unstable, although fracturing at a reduced rate continues in regions that have not completely dilated. The overall group behaviour is no longer associated over the extent of the previously defined process zone due to the high level of fracturing and postulated dilated, (stress relived), regions. This makes the observed group behaviour distinctly different and variable, and in this analysis, as in the other similar failures at Williams mine, the trends switches between the dominant trend with a strike and dip of [075, 50] and [260, 65] similar to the A-set and other less stable orientations. Although the latter trend is similar to the A-set orientation it is not thought that slip is occurring on this structure based on fault plane solutions, (Chapter 5), but may be due to the activation of conjugate fracture sets or other shear structure allowing for greater degrees of freedom. Alternatively the second observed trend may be simply due to the vertical expression of the group behaviour following the trend of the orebody. Inter-fracture communication and the influence of
each event on its neighbours is significantly reduced from the previous large area due to the patches of failed/dilated or sheared rock mass resulting in the decrease in ellipticity. From the source parameters no clear or distinctive change could be seen apart from a minor increase in strength and stress estimates to more stable post peak reduced levels, potentially as a result of redistribution in stress.

- In this study the failure process occurs relatively quickly over a period of a few months once initiated, through a major increase or flurry of activity, while at the Williams mine, a number of flurries generally were seen, often related to or connected with large magnitude events. It is felt that at Williams a more gradual failure process over the period of a number of years occurred, primarily related to a slightly more gradual increase in the loading rate, based on the linear elastic modelling of the stress histories. Here, due to the higher stress state, once a critical fracture density had been reached the fractures appear to propagate unstably, increasing the rate of the failure process.

- Linear elastic three-dimensional modelling can be used as a first approximation, in combination with the Hoek-Brown brittle failure parameters defining damage limits, or limiting factors of safety, to obtain a rough guide to regions that are expected to yield. In this study fracture initiation, coalescence and localization was determined to occur at equivalent stress levels, at a deviatoric stress term ($\sigma_1 - \sigma_3$) of around 0.37$\sigma_c$, (i.e. $m=0$, $s=0.14$), and the point of disassociation was found to be slightly higher at 0.4 $\sigma_c$, (i.e. $m=0$, $s=0.16$). Comparing this analysis to the Williams mine study, (Chapter 3), the initiation damage limit appears to show some confinement dependence, and the linear damage limit is better approximated by applying a modified Tresca envelope ($\sigma_1 = A\sigma_c + B\sigma_3$) with a fit to the data determined for $A=0.22$ and $B=1.5$.

- The stress path determined in this study, using linear elastic modelling, lies significantly below the brittle-ductile transition line (Mogi, 1966). As discussed in Chapter 3, Mogi’s line may not be appropriate as a behavioural guideline to determine the post peak behaviour of the rock mass if these stresses are based on linear elastic modelling. The overall behaviour observed at Golden Giant was perceived to be brittle strain softening due to the removal of stress effects. Additionally, the suggested limit proposed by Hoek and Brown (1980), of $\sigma_1 / \sigma_3 < 2.0$, may be more appropriate for this kind of calibration and needs to be investigated more thoroughly through laboratory testing.
• An alternate interpretation is that the formation of this macrofracture shear structure occurs under ductile conditions, creating shear banding at the centre of the confined pillar zone. There is a change in the strength state, from ductile to brittle due to an overall change in the pillar geometry from short and squat creating significant confinement and ductile behaviour to a more slender pillar with brittle behaviour (Kaiser pers. comm., 2009). This is not a change in the material behaviour but a change in the strength state due to geometrical influences. The effect of the change in the geometry should be investigated further.
CHAPTER 5

FOCAL MECHANISM STUDY OF MINE INDUCED MICROSEISMIC EVENTS DURING VARIOUS STAGES OF FAILURE OF A CONFINED ROCK MASS IN RELATION TO THE PRE TO POST-PEAK STRENGTH CONDITIONS

5.1 Introduction

In the previous Chapters, (Chapter 3 and Chapter 4), two major studies have been carried out of relatively controlled failures of the rock mass through progressive, almost monotonic loading of modelling linear elastic principal stresses, in which it was identified that that the rock mass was driven well into the post peak strength state, causing ‘complete failure’ to a residual strength state by eventual strain softening behaviour based on the response of the rock mass to subsequent mining. In these studies a temporal analysis of the spatial distribution of microseismicity was performed using the principal components analysis (PCA) technique, to obtain three-dimensional trends in the group behaviour of the events.

Two very similar failure processes were identified and analysed at two of the mines; the Williams Mine – analysis of temporal PCA with conventional displacement instrumentation (Chapter 3) and the Golden Giant Mine – analysis of temporal PCA with temporal changes in source parameters (Chapter 4). It was postulated that, in these specific cases, the predominant failure process was through the formation of a macrofracture shear structure created by an en-echelon interacting fracture network under confinement.

The PCA technique was shown to be valuable in identifying the potential changes in strength states of the rock mass for these cases, by making comparison to the formation of similar structures recorded through acoustic emissions of triaxial testing of intact rock in the laboratory, (Lockner et. al., 1992; Lockner, 1993; Thompson et. al., 2006), based on the assumption that events are inter-related to one another through the stress regime and fracture generation zone in a planar manner.
The main area of interest, investigated further in this Chapter, that was geometrically stable, (i.e. had no caving of the rock mass, and no development of excavations in the immediate vicinity), is the high stress region above the back of previously mined stopes in the east abutment of the Destress Slot region, the East of Slot (EOS) cluster at the Golden Giant mine, (Figure 5.1 and Chapter 4). This is one of the regions at the camp that was monitored through the full failure process from microseismic initiation, coalescence and localization (Chapter 1) to aseismic behaviour before being mined easily without previously observed stress effect or squeezing of boreholes. This cluster was analysed using the PCA technique, with analysis of source parameters and showed significant temporal changes during the failure process. The former have been summarized in Figure 5.2 and Figure 5.3, and is used as a temporal guide in this Chapter for the investigation of focal mechanisms during the failure process. The summarised behaviour found from this analysis was that, prior to interaction and coalescence of events/fractures, no specific trend could be identified (Figure 5.2a). Following interaction and coalescence however, a strong and stable trend, (Figure 5.2b), oriented [076,52] in what appears to be the direction of maximum shear, (close to 45° from the maximum induced principal stress direction), formed which stayed stable for 6 months, or half of the 3000 events recorded in this cluster, up until what has been termed the point of disassociation. The trend switches in strike and dip to [254, 84] and then is unstable switching between the previously dominant trend [076,52] and [255,65] into 2004 and 2005. This is the potential mobilization of conjugate shears or other structures allowing for greater degrees of freedom on the rock blocks. At this time it was identified that patches of the rock mass become aseismic, related to potential large dilations or shear, based on evidence at the Williams mine, (Chapter 3), and cause the previously observed dominant group trend to change and destabilize (Figure 5.2c and d). It is at this point that evidence suggests that part of the rock mass has failed to close to the residual strength, the dilation causing strain softening behaviour.

The temporal changes of the average PCA derived trends and average ellipticity can be seen in Figure 5.3a. It was noticed that, following an increase in the microseismic activity, (Figure 5.3b), and the proliferation of small magnitude interacting events associated with what is considered to be the yield strength of the rock mass, there was a significant increase in the ellipticity, indicating localization of events to this macrofracture structure. Based on an analogy to laboratory triaxial testing discussed previously, this probably represents the peak strength of the rock mass. This infers that all the subsequent events that are recorded are from a rock mass that is in the post peak strength condition, probably undergoing gradual strain softening through additional rock mass degradation.
Figure 5.1 All recorded events during 2003 in the shaft pillar area, showing the location of the destress slot and shaft and the east of slot (EOS) cluster of events analysed in this chapter.

Figure 5.2 Yearly temporal analysis of PCA for EOS cluster (a) Source locations and plot of PCA poles in 2002 – Pre-Interaction, (b) Locations and PCA poles 2003 – Interaction, Coalescence, Localization and Disassociation, (c) Locations and PCA poles 2004 – Post Disassociation and (d) Location and PCA poles 2005 – Post Disassociation.
Figure 5.3  (a) Temporal variation in mean PCA derived planes for EOS cluster from May 2002 – July 2005, showing variation in mean strike and dip, and ellipticity per temporal window plotted against time of the last event in the window. Indicated for points [1] to [5] are estimation of the strength states of the rock mass during failure. Also indicated are the time periods A to E in which samples of events where analysed using first motions. (b) Event rate in EOS cluster and stopes mined (e.g. S5 mined between 4620 and 4600 levels is named 460-S5)
In this study, at the point of disassociation the strength of the ellipticity reduces to the background level, with the strike and dip of the trend of the PCA planes remaining unstable during what is postulated to be close to the residual strength of the rock mass.

It was also identified from the source parameter study that these events had relatively low \( \frac{E_s}{E_p} \) ratios, (a measure of the potential source mechanism), with around 95% of the events having ratios between 1.0 and 5 (Chapter 4). This indicates that these events are non-pure shear, and have a significant tensile component suggesting a combined shear-tensile failure (Urbancic, 1991, Gibowicz et al., 1991) which may be inferred as representative of fracture creation, rather than pure slip on pre-existing structure. An increase in the \( \frac{E_s}{E_p} \) ratios to greater than 10 (suggesting more pure shear failure) was noticed, however only during the period of localization identified from the PCA. The estimated strength states of the rock mass during the failure process are indicated in Figure 5.3a, and more details on the Detress Slot and analysis of this cluster using the PCA technique and temporal changes in source parameters can be found in Chapter 4.

The main question arising from the formation of this macrofracture structure is whether the events are associated with joint structures or are fresh fractures with different orientations? Additionally, do the focal mechanisms change temporally and does the orientation of the fault plane solutions change with similar trends as defined by the PCA? To answer these questions the source mechanisms of sub-sets of the events taken at periods during the failure process were investigated using P-wave first motions to identify fault plane solutions. These have been compared to geology, the PCA derived trends and the induced stress field to understand their formation and whether they change during the failure process. From linear and non-linear modelling the orientation of fractures is analysed, in relation to the mining induced stress orientation and magnitudes, to identify the most likely orientation and initiation condition to form these en echelon fractures using a ubiquitous joint analysis approach. Additionally, non-linear modelling is used to investigate the post peak behaviour and the conditions necessary to form similar shear style failures.

Based on the findings of the focal mechanism study determined in this Chapter, source parameters and the PCA, a rudimentary fracture geometry has been postulated. This is based on experimental work of Hoek and Bieniawski (1965), similar to the sliding crack model of Kemeny and Cook (1991), and is used to observe the potential fracture network developed. This has been compared to visual observations of other mining induced shear structures (Gay
and Ortlepp, 1979) and to laboratory shear box tests with the development of ‘Riedel fractures’ (Cho et. al., 2008) as visual observation was not possible here.

## 5.2 Calibration of Microseismic Array for First Motion Studies

Details of the microseismic array located around the Destress Slot region are reviewed in Chapter 4. The primary array used in this analysis can be seen in Figure 5.4, and consists of 4 triaxial sensors and 26 uniaxial sensors within the shaft region with another 25 uniaxial sensors distributed outside of this region for mine wide coverage. For the focal mechanism study covered in this Chapter, manual picking of P- and S-wave arrivals was performed. Although the online system can automatically pick P-wave arrivals, locate and calculate source parameters, the accuracy of the first motion is often not adequately recognised. The online locations were used as the basis for the determination of group behaviour using the PCA technique and source parameters, (Chapter 4), and it was found that average differences between automatic, (with some manual picks), and full manual picking were around 2 m for locations and ±5% offset for source parameters for the events in the cluster analysed here.

In order to evaluate the source mechanisms the exact orientation of sensors must be known. Although the original survey’s of sensors accurately located them, information on their orientation was not retained. This required a detailed resurvey by the author of sensors in the shaft pillar array to within ± 5° dip and azimuth, and reconstruction from drilling layouts of some 25 sensors outside of the shaft array, calculating the directional cosines for all functioning sensors. In order to check that the correct sensor polarities were being applied, 48 blast events consisting of 40 development blasts and 8 stope blasts from various locations within the shaft pillar area were analysed to see the distribution of polarities. For an explosion, the resultant first motions should all have +ve polarites, (Chapter 2), the correct sign being translated from the first motion and corrected for the orientation of the sensor in relation to the source. Often sensors may be incorrectly wired, may not have accurate orientations noted or structures close to the sensor location can create reflections, (Trifu pers. com., 2006), and thus this procedure is essential in establishing a base line for polarities. It was noted during the initial analysis that the majority of the uniaxial sensors conformed well, with 15%, (8 out 50 sensors including mine wide sensors), requiring adjustment of polarities. Additionally, some of the uniaxial sensors (~5%) did show some polarity dependence based on the position of the blast in relation to the sensor location. This is thought to be primarily due to the orientation of the accelerometers in relation to the direction of the blast, having close to 90° incidence angles, and to potential site effects, (i.e. discontinuities affecting the signal). These sensors were noted as being less
confident and had their polarities adjusted to minimise mismatch between the blasts, and would be dropped from the fault plane solutions if in discord with neighbours.

![Array Feb 2003, uniaxial (circle) and triaxial (triangle) sensors (a) View North, (b) View East and (c) Rotated Plan View looking down the orebody.](image)

Additionally, the triaxial sensors, based on their surveyed coordinates and orientation, were found to rotate poorly, with large divergence angles between theoretical and actual locations, suggesting that the sensors may be rotated in the boreholes even though a tilt switch was used for installation. The orientations of the four sensors were recalculated by perturbing the orientation, to achieve maximum P- and S-wave amplitudes of the rotated waveform components, (P, SV and SH), and then tested for orientation consistency by comparison of pairs of sources and sensors (Shumila, 2007). From this study two of the sensors, (13 and 10 see Figure 5.4a), were found to obtain greatest amplitudes if rotated around the z-axis, (bore hole orientation), by 90°, suggesting incorrect orientation at installation. After correction these sensors showed relatively stable rotations, with minimum divergence angles between rotated theoretical and actual blast locations (mean of 2° to 5°). The other two sensors, (16 and 33, the former found to be rotated around the z-axis by 45°, the later with switched channels), showed greater variance in divergence angles for all blast locations from this process. These sensors were noted to be less reliable in the evaluation of the first motions, but were used if reasonable
agreement existed between the rotated theoretical and actual event location. The reasons for the greater variance could not be thoroughly ascertained and is most likely a result of site conditions.

Based on the blast analysis, it was apparent that the majority of sensors should give reasonable determination of polarities. Considering that there is relatively good focal sphere coverage for the area of interest, fault plane solutions should be possible based on first motions of uniaxial sensors alone, if a double-couple mechanism is present. However, due to the issues in triaxial orientation and having only four sensors, calculation of moment tensor inversion for stable solutions using the general model, (Trifu and Shumila, 2002a, Chapter 2), was not possible. Even with the addition of uniaxial sensors, (Trifu and Shumila, 2002b), to increase determination of the solution quality, stable moment tensor solutions using the general model could not be achieved. However, much can still be learned from fault plane solutions based on first motion studies.

### 5.3 Focal Mechanism Analysis

#### 5.3.1 Method and Data Selection

Determination of focal mechanisms by first motions is made on the basis of a double-couple force system characterised by a quadrilateral seismic radiation pattern, (Chapter 2), and is based on the idealized Mohr stress circle in that shear, (orientation of maximum shear stress), occurs at 45° to the applied couple (cohesion assumed = 0). By using the distribution of the polarities in azimuth and take-off angle around the event location, information is obtained on the wave radiation from the source. The measured P-wave polarities recorded at the sensors are projected onto the focal sphere, and translated to an equal area lower-hemisphere stereonet to give the fault plane solution (Urbancic, 1991). If a double-couple mechanism, (representing shear-slip), on a plane is present then, based on the relative distribution of polarities, the faulting type can be inferred. Details of different faulting types are discussed in Chapter 2. Two possible planes, (nodal planes), are defined based on the double-couple, which are the fault plane and the auxiliary plane. Identification of which of the nodal planes is the actual fault plane must be made with additional information. This assessment can be made through identification of clusters of nodal poles with a knowledge of the stress regime, based either on a fundamental mechanical understanding of the likely shear plane based on the loading conditions, or using stress inversion techniques, (Gephart and Forsyth, 1984; cited Urbancic et. al., 1993), but not used here, and/or comparison to geological structural
information, and comparison to the PCA poles. As previously mentioned Urbancic et. al., (1993), identified correlation of fault plane solutions to geological joint structure, stress inversion and the dominant PCA trend at two mines sites, the Strathcona Mine and Mines Gaspé, (the later based on fault plane solutions identified by Connors et. al, 1993). Thus, it is expected that there is the potential for similar correlations here, although as will be identified, these trends correlate not to the geology but to the principle direction of fracture propagation.

Determination of the nodal planes is made automatically through use of a commercial package, Visual Fault plane solutions (VFps - ESG, 2006), based on the maximum-likelihood algorithm as described by Brillinger et. al. (1980) and Udias et. al., (1982) and discussed in more detail in Urbancic (1991). In order to apply the algorithm, it is required to iteratively perturb the initial model until a convergence criteria is met to maximize the likelihood function. Here it was found that the simplex algorithm provided the most stable results, over the Gradient Davidson-Fletcher-Powell or simple grid search algorithms (ESG, 2006).

The quality of the fault-plane solutions is directly related to the relative source-sensor positioning and the number of polarity readings. From synthetic testing, (Urbancic, 1991), it has been suggested that a minimum of 15 sensors is required to provide adequate solutions based on reasonable focal sphere coverage and a number of incorrect polarities. In order to maximize the chances of successful solutions here, a criteria of events that triggered a minimum of 20 sensors was used for most of the analysis. This allows for either dropping completely polarities that could not be easily determined due to signal to noise ratios, (generally 1 to 3 polarities), or had low confidence, (generally 2 to 3 polarities), but still obtaining enough to produce a reasonable solution. During the monitoring period of this cluster, from 2002 until 2005, a total of 2882 events were recorded. As can be seen in Figure 5.5a and b, by setting the criteria to 20 sensors triggered it appears to show 3 separate clusters, which are connected together by events triggering fewer sensors, (these regions may be due to shadowing effects related to excavations). Here efforts have been concentrated on analysis of the core events in the centre of the cluster, (Figure 5.5b), which it is felt defines the macrofracture shear structure. The events that are above this core cluster are an extension up dip based on the dominant PCA trend. Activity in this region however is only present up to the point of disassociation, (June 2006, Figure 5.2c and Figure 5.3a), and it was felt that if there was a change in the faulting mechanisms then this would be also reflected in the core cluster which was present over a longer time frame. With regards to the apparently separate cluster to the east of the footwall raise, this is close to the backfilled abutment stope where it might be expected that the mechanisms are different from the core cluster and are not analysed here.
Figure 5.5 Example of selection criteria for fault plane analysis (a) Online location of events in the EOS cluster from Jan 1, 2003 to April 30, 2003 (1050 events) covering periods B and C (b) Same period but for events triggering a minimum of 20 sensors (342 events total) and selected events at the core of the cluster (100 events) with selected events only in the view west (c) Relocated events in cluster after re-picking of P- and S-waves. Note colour legend is based on the number of sensors hit.
Table 5.1 Summary of event filtering for determination of good quality fault plane solutions at the core of the EOS cluster

<table>
<thead>
<tr>
<th>Period</th>
<th>Date</th>
<th>Trigger Criteria</th>
<th>Total # of Events in Entire Cluster for Period</th>
<th>Total # of Events in Entire Cluster at Criteria</th>
<th>Total # of Events At Criteria and Core for FP Solns.</th>
<th>Avg. # Of Triggers</th>
<th>Avg. Number Of Polarities Used For FP Solns.</th>
<th>Avg. Score (% fit)</th>
<th>State</th>
</tr>
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<tbody>
<tr>
<td>A</td>
<td>04/19/2002</td>
<td>&gt;15 hits</td>
<td>330</td>
<td>272</td>
<td>15</td>
<td>18</td>
<td>16</td>
<td>73</td>
<td>Initiation</td>
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<tr>
<td></td>
<td>01/31/2003</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Pre-interaction</td>
</tr>
<tr>
<td>B</td>
<td>02/12/2003</td>
<td>&gt;20 hits</td>
<td>599</td>
<td>105</td>
<td>15</td>
<td>23</td>
<td>18</td>
<td>71</td>
<td>Interaction/Coalescence</td>
</tr>
<tr>
<td></td>
<td>03/04/2003</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Pre-localization</td>
</tr>
<tr>
<td>C</td>
<td>03/06/2003</td>
<td>&gt;20 hits</td>
<td>386</td>
<td>210</td>
<td>85</td>
<td>22</td>
<td>18</td>
<td>73</td>
<td>Peak Ellipticity Starts</td>
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<tr>
<td></td>
<td>04/30/2003</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Localization</td>
</tr>
<tr>
<td>D</td>
<td>06/15/2003</td>
<td>&gt;20 hits</td>
<td>330</td>
<td>101</td>
<td>59</td>
<td>22</td>
<td>19</td>
<td>76</td>
<td>Disassociation</td>
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<tr>
<td></td>
<td>12/30/2003</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Post Disassociation</td>
</tr>
<tr>
<td>E</td>
<td>01/01/2004</td>
<td>&gt;20 hits</td>
<td>118</td>
<td>92</td>
<td>49</td>
<td>24</td>
<td>21</td>
<td>75</td>
<td>Post Disassociation</td>
</tr>
<tr>
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<td>04/16/2004</td>
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<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Post Disassociation</td>
</tr>
<tr>
<td></td>
<td>Total</td>
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<td>780</td>
<td>243</td>
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</tbody>
</table>

The strength states of the rock mass that have been estimated based on the PCA are indicated in Figure 5.3a, and are used as a guide to observe potential temporal changes in the focal mechanism over time. It was found that applying the minimum 20 sensor triggering criteria was too harsh in 2002, and this was softened to a minimum of 15 sensors triggered. The main reason for this is that the array was increased in 2003, with the addition of one more triaxial sensor (#16) on January 10, 2003 (Figure 5.4), and the addition of 4 uniaxial sensors on February 21, 2003 on the 4600 and 4620 levels (Chapter 4). Five samples of events were selected from the same core region as identified in Figure 5.5b, and were temporally selected based on the inferred strength states of the rock mass. The time intervals of periods, (A to E), are illustrated in Figure 5.3a and summarised with number of events in the sub-set, average number of polarities and inferred strength state in Table 5.1. Generally, the sub-sets represented approximately 1/3 of the population of the entire cluster for the same time periods. Based on the criteria, all events that occurred in the core region were analysed for the first motion study with the exception of Period C, in which it was noted that the same mechanisms existed for May to June 2003, however, due to time constraints of the research only the first period was sampled and is felt to be representative. All the events selected had the P-wave first motions and S-wave first motions re-picked manually, were relocated, (typical shift in event locations illustrated in Figure 5.5b to c), and fault plane solutions determined. Polarities that had low confidence were flagged, and not included in the fault plane solution. The average
percentage scores, (percentage of polarities that fit the determined model; note low confidence polarities reduce the score but do not affect the determined model), are also indicated in Table 5.1, and tended to be around 75% with a standard deviation of 10%. Only fault plane solutions that had scores greater than 60% were retained, this resulted in a loss of around 7 solutions out of 250 or 3%. This relatively low rejection rate results from the high triggering criteria resulting in good focal sphere coverage.

Comparison of the source parameters from these reduced sub-sets with the population based on mostly automated P-wave picking, indicates that similar behaviour exists in terms of temporal decrease in the events average strength during interaction (Figure 5.6). It can be noted that the decrease in seismic moment associated with fracture interaction, (Chapter 4), occurs later in this sub-set starting at the peak activity level, suggesting that interaction may have started at the edges of the entire cluster first and propagated to the centre. The drop in seismic moment for the sub-set, occurs at the same time as the increase in sustained high ellipticity and stable PCA trend suggesting localization. This is thought to be the peak entering into the post peak strength of the rock mass. It should be remembered that setting a filtering criteria of greater than 20 triggers may affect the distribution, however, the sampled sub-set were found to have similar means and standard deviations to the entire cluster (Table 5.2). For investigation of the first motions this sub-cluster appears to be representative of the population.

Table 5.2 Comparison of EOS Cluster Population Statistics to Extracted Sub-set for Fault Plane Solutions

<table>
<thead>
<tr>
<th>Overall, 04/19/2002 to 09/04/2005 (n=2761)</th>
<th>M</th>
<th>M₀ (Nm)</th>
<th>E₀ (J)</th>
<th>σᵦ (Pa)</th>
<th>Δσ (Pa)</th>
<th>Δσᵈ (Pa)</th>
<th>Eₐ/Eₚ</th>
<th>rₑ (m)</th>
<th>rₑ (m)</th>
<th>Δσᵈ/Δσ</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>-1.25</td>
<td>1.85e07</td>
<td>77.03</td>
<td>5.60e04</td>
<td>1.76e06</td>
<td>2.36e06</td>
<td>1.87</td>
<td>1.69</td>
<td>3.60</td>
<td>1.54</td>
</tr>
<tr>
<td>Std.D.</td>
<td>0.19</td>
<td>2.47e07</td>
<td>321.39</td>
<td>7.14e04</td>
<td>2.00e06</td>
<td>2.43e06</td>
<td>1.18</td>
<td>0.20</td>
<td>0.55</td>
<td>0.71</td>
</tr>
<tr>
<td>Median</td>
<td>-1.31</td>
<td>1.10e07</td>
<td>9.28</td>
<td>2.86e04</td>
<td>1.0e06</td>
<td>1.59e06</td>
<td>1.69</td>
<td>1.69</td>
<td>3.59</td>
<td>1.48</td>
</tr>
<tr>
<td>Min.</td>
<td>-1.81</td>
<td>1.95e06</td>
<td>0.13</td>
<td>2.13e03</td>
<td>9.99e04</td>
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<td>0.42</td>
<td>0.93</td>
<td>1.90</td>
<td>0.45</td>
</tr>
<tr>
<td>Max.</td>
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<td>2.88e08</td>
<td>6350.00</td>
<td>8.40e05</td>
<td>2.22e07</td>
<td>3.24e07</td>
<td>26.36</td>
<td>2.39</td>
<td>6.93</td>
<td>14.26</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Overall Sub-set, 04/19/2002 to 04/16/2004 (n=243)</th>
<th>M</th>
<th>M₀ (Nm)</th>
<th>E₀ (J)</th>
<th>σᵦ (Pa)</th>
<th>Δσ (Pa)</th>
<th>Δσᵈ (Pa)</th>
<th>Eₐ/Eₚ</th>
<th>rₑ (m)</th>
<th>rₑ (m)</th>
<th>Δσᵈ/Δσ</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>-1.37</td>
<td>1.62e07</td>
<td>63.97</td>
<td>6.03e04</td>
<td>1.86e06</td>
<td>2.85e06</td>
<td>1.90</td>
<td>1.60</td>
<td>3.39</td>
<td>1.58</td>
</tr>
<tr>
<td>Std.D.</td>
<td>0.18</td>
<td>2.00e07</td>
<td>252.12</td>
<td>6.74e04</td>
<td>1.90e06</td>
<td>2.99e06</td>
<td>0.76</td>
<td>0.20</td>
<td>0.53</td>
<td>0.34</td>
</tr>
<tr>
<td>Median</td>
<td>-1.42</td>
<td>1.11e07</td>
<td>11.10</td>
<td>3.38e04</td>
<td>1.11e06</td>
<td>1.88e06</td>
<td>1.82</td>
<td>1.60</td>
<td>3.42</td>
<td>1.60</td>
</tr>
<tr>
<td>Min.</td>
<td>-1.69</td>
<td>3.80e06</td>
<td>0.50</td>
<td>4.33e03</td>
<td>2.12e05</td>
<td>1.97e05</td>
<td>0.64</td>
<td>1.14</td>
<td>2.09</td>
<td>0.75</td>
</tr>
<tr>
<td>Max.</td>
<td>-0.52</td>
<td>2.00e08</td>
<td>2800.00</td>
<td>4.64e05</td>
<td>1.42e07</td>
<td>2.44e07</td>
<td>8.22</td>
<td>2.62</td>
<td>4.71</td>
<td>3.37</td>
</tr>
</tbody>
</table>
Figure 5.6  Comparison of temporal variation of seismic moment for the (a) the entire EOS cluster of events for 2003 and (b) the sub-set of events used for the first motion study at the core of the cluster and using a filtering criteria of greater than 20 sensors triggered.
5.3.2 Overall Faulting Mechanism, Comparison to Modelled Stress Orientation

For the time from February 2003 to May 2003, spanning periods B (interaction) and C (coalescence and localization), samples of typical fault plane solutions can be observed in Figure 5.7. For most solutions during this time frame, there is relatively good focal sphere coverage. Out of the 97 events with acceptable scores, (3 events dropped), the majority of the events present reverse faulting mechanisms (82%), with the next most prevalent mechanism being normal faulting (12%), with some minor strike-slip or dip-slip faulting (6%). As can be seen the normal and particularly the strike-slip faulting appear to be positionally related to the edge of the sub-set, in line with the footwall (FW) raise, although a number of normal faults are seen, early on in the core region of the cluster. Although no events are located close to the raise, it may be that past failure, predominantly identified in 1995 (Chapter 4), could have altered the immediate stress field in this region affecting the fracture mechanisms.

The stress axis determined from the 243 events (Periods A to E) that were processed to determine the focal mechanisms have been compared to the mining induced stress tensor orientation of the principal stress axis, ($\sigma_1$, $\sigma_2$, $\sigma_3$), calculated using a three dimensional boundary element linear elastic model, (Examine3D; Rocscience 2007), of the mining geometry (Chapter 4). The stress orientation was calculated at the location of events for periods B and C of which the poles representing the trend and plunge are displayed in Figure 5.8a. From the linear elastic modelling it was found that the orientation of these induced elastic stresses do not vary significantly over time from 1993 until mid 2005, when the S2 stope at the core of the cluster was mined, (Figure 5.2d), although the magnitude of the average induced maximum principal stress increases almost monotonically with time (Chapter 4). Thus, this smaller modelled sub-set of data, with stresses calculated at the appropriate time can be considered to be representative of the overall induced stress orientation, up until the time of localization, when the failure of the rock mass would potentially alter the induced stress orientation.

As can be seen from the comparison in Figure 5.8, there is a strong correlation between the orientation of the stress axis of the focal mechanisms and that of the model induced principal stresses. This has been summarized in Table 5.3. It should be noted that the prevalent reverse faulting mechanism dominates the maximum pole concentrations of the stress axes. The compression P-axis is found to have an almost identical orientation to the maximum induced principal stress, ($\sigma_1$), (Figure 5.8a & b).
Figure 5.7  Typical fault plane solutions determined for events in Periods B and C (February 22 2003 to May 30 2003) associated with the stress state of interaction and coalescence/localization. Note Blue triangle are compressional (+ve) and Yellow triangles are tensile (-ve) first motions, while squares are uncertain polarities.

Also, there is strong correlation of the tension T-axis oriented vertically to the that of minimum induced principal stress ($\sigma_3$), while the null B-axis is more dispersed but agrees relatively well with the intermediate principal stress ($\sigma_2$), related to the strike of the orebody and hence mining geometry. The agreement of the stress axis with linear elastic modelled induced stresses has also been identified by Connors et al. (1993), Bird (1993), Sampson-Forsythe (1994) and postulated by Urbancic (1991), when the predominant failure mechanism of reverse faulting due to high horizontal stress is present. The correlation of stress axis to the principal stress direction has also been noted in earthquake seismology when a statistically significant number of fault-plane solutions can be obtained, (Zoback and Zoback, 1980). This indicates that we can add another level of confidence to the determined fault plane solutions, and identifies that the orientation of the modelled stress tensor can be determined with a fair degree of accuracy from the focal mechanism study of microseismicity.
Figure 5.8  Comparison of (a) linear elastic induced stress tensor orientation based on event locations of the fault plane sub-sets for February 2003 to May 2003 (Periods B and C) to the stress axis of all 243 focal mechanism for (b) the P-axis, the compression axis equivalent to $\sigma_1$, (c) the B-axis, the null axis equivalent to $\sigma_2$, (c) and the T-axis, the tension axis, equivalent to $\sigma_3$.

Table 5.3  Comparison of Linear Elastic Induced Principal Stress Orientations with Overall Stress Axis Orientations Determined from Fault Plane Solutions

<table>
<thead>
<tr>
<th></th>
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<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>$\sigma_1$ or P-axis</td>
<td>[171,02]</td>
<td>[171,02]</td>
<td>[358,10]</td>
</tr>
<tr>
<td>$\sigma_2$ or B-axis</td>
<td>[081,10]</td>
<td>[254,03]</td>
<td>[093,28]</td>
</tr>
<tr>
<td>$\sigma_3$ or T-axis</td>
<td>[272,80]</td>
<td>[044,82]</td>
<td>[250,60]</td>
</tr>
</tbody>
</table>
5.3.3 Overall Faulting Mechanisms, Comparison to Geology and PCA

The mapped joint features that have been identified in the region are displayed for comparison in Figure 5.9a, the dominant set being the foliation A-set oriented parallel to the orebody, [262,60] ([Strike Right-Hand-Rule, Dip from horizontal]), with the next dominant set the B-set oriented perpendicular and vertically to the orebody [344,86]. There is also a flat lying set, the C-set [085,25], however, this set tends to be sparse even when vertical mapping is performed and, although used for structural analysis, was found not to be prevalent in the region analysed here. For a more detailed discussion on the geology of the region see Chapter 4.

All of the probable fault planes have been plotted in Figure 5.9c, with their associated auxiliary planes in Figure 5.9d. The probable fault plane solutions for each individual mechanism are plotted separately in Figure 5.10, along with plots of all nodal planes for each mechanism.

The most probable orientation of the reverse faulting mechanism [076,47] was chosen on the relatively strong agreement with the orientation identified by the PCA technique [076,52]. From analysis of the most likely orientation to fail based on the stress conditions, discussed later in this Chapter, the PCA orientation is just slightly more likely to fail than that of the conjugate auxiliary orientation [244,42] identified in Figure 5.10e. Also, from underground observations of stress fracturing associated with back failures at this site and the adjacent sites, this general trend of fractures propagating approximately perpendicular to the A-set foliation appears more appropriate. Here the included angle between the fault plane and the A-set foliation is around 70°, whereas if the auxiliary plane orientation were chosen the included angle would be 20°, and this type of fracture interaction appears unlikely, and has not been observed. Solutions that were reverse faulting and did not correspond to the dominant PCA trend were designated ‘reverse other’, and the most probable fault plane was chosen based on pole clustering identified from the temporal analysis (see next section).

The most probable orientation of normal faulting mechanisms, [263,56], were selected based on a similar orientation to the steeply dipping A-set foliation, [262,60] (Figure 5.9). Mechanically this faulting mechanism would indicate that the local maximum principal stress (P-axis) would have to be oriented vertically, but based on the modelled stress orientation this is not the case and might suggest local stress heterogeneity. Evidence to support the steeply dipping orientation was taken primarily from consideration of the radiation pattern emitted by this rupture orientation.
Chapter 5  Adam Lee Coulson, Doctor of Philosophy, 2009  242

**a. Joint Mapping**

Table 5.4 Comparisons of Overall Fault Plane Orientations with PCA and Geology (Strike/R/Dip)

<table>
<thead>
<tr>
<th>Fault Mechanism</th>
<th>Fault Plane</th>
<th>Aux. Plane</th>
<th>PCA</th>
<th>Joint Mapping</th>
</tr>
</thead>
<tbody>
<tr>
<td>Reverse (PCA)</td>
<td>[076,47]</td>
<td>[244,42]</td>
<td>[076,52]$^2$</td>
<td>[085,25] C-set</td>
</tr>
<tr>
<td>Reverse Other$^1$</td>
<td>[183,42], [057,42]</td>
<td>[045,46], [195,60]</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Normal (A-set)</td>
<td>[263,56]</td>
<td>[039,40], [119,41]</td>
<td>[255,65]$^3$</td>
<td>[262,60] A-set</td>
</tr>
<tr>
<td>Normal Other$^1$</td>
<td>[276,36], [060,14]</td>
<td>---</td>
<td></td>
<td>[085,25] C-set</td>
</tr>
</tbody>
</table>

1 Other fault plane orientations seen after the point of disassociation (see temporal analysis).
2 Dominant PCA trend identified during coalescence and localization.
3 Alternate PCA trend identified after the point of disassociation.

**b. PCA Planes 2003 only**

**c. All Probable Fault Planes**

**d. All Aux. Fault Planes**

Figure 5.9  Comparison of poles plotted on lower hemisphere stereographic projections determined from (a) Joint mapping, (b) PCA derived planes for 2003 only for the EOS cluster, (c) All probably Fault planes (243) based on stress and clustering for April 2002 to April 2004 (Periods A to E) of sampled sub-sets and (d) All other nodal (Auxiliary) planes.
Figure 5.10  (a) Joint Mapping with grouping of all determined fault plane solutions for April 2002 to April 2004 (Periods A to E) of sampled sub-sets for (b) all probable reverse faults, (c) all probable normal faults and (d) all probable strike-slip faults, and (e) all reverse fault nodal planes, (f) all normal nodal planes and (g) all strike-slip nodal planes with auxiliary planes noted.
It was found that, generally, these normal events triggered more sensors than the reverse faulting mechanisms, especially triggering sensors located much deeper in the mines footwall and above in the hangingwall outside of the dense shaft array. If a steeply dipping rupture plane were considered over the auxiliary sub-vertical nodal plane, then a radiation pattern emitted vertically makes a better sensor triggering logic, based on theoretical radiation patterns (Chapter 2). Additionally, review of the all of the nodal planes for the normal faulting mechanism identifies that based on this small dataset, there is no significant clustering of potential auxiliary planes, but there is pole clustering for an orientation related to the A-set (Figure 5.10f). A small number of normal mechanisms were identified that did not fit with this model, but had either a near vertical East-West striking plane or a sub-horizontal East-West plane. In this case the sub-horizontal plane was chosen as the more probable fault plane due to less clamping.

The small number of strike-slip faults that were observed in this sampled sub-set, were generally not pure strike-slip events as in defined in Chapter 2, but were similar to the typical solution shown in Figure 5.7, a cross between dip-slip and strike-slip. The most probable fault plane was identified as a sub-vertical plane, [180,89] corresponding most closely to the B-set joint orientation [344,86] or sub-vertical plane [010,47], which may correspond to the minor D-set, [354,51], as identified in Figure 5.10d. Additionally, a minor cluster of planes was found to be oriented [253,66] this being similar to the A-set orientation of [262,60]. These other rupture plane directions may represent readjustment of the rock mass with fracturing controlled by pre-existing features. The auxiliary plane to these orientations is a sub-vertical to vertical East-West striking plane, [106,83], as indicated in Figure 5.10g. This would be an unlikely fracture and shear direction due to clamping from the maximum principal stress.

The determined orientations of the probable fault planes for each mechanism, their auxiliary plane orientation and relationship to joint structure are summarised in Table 5.4. As has been noted there is a strong correlation between the determined orientation of the fault plane solutions of the individual events/fractures and that of the group behaviour identified during coalescence and localization to a macrofracture structure identified from the PCA technique. This indicates that the group behaviour of the events and the overall direction of the event cloud is a function of the individual event/fracture direction. As the closest joint set to this orientation is the sparse C-set, which is at a significantly shallower dip, and from the ubiquitous joint analysis would be unlikely to fail under shear, (Section 5.4.1), combined with the knowledge that these events are characterised by a low $E_s/E_p$ ratio, (< 5), suggests that the formation of this macrofracture shear structure is through the development of new fracture
growth and not shear slip on pre-existing joint structure. The macrofracture structure is postulated as being developed through coalescence of en-echelon, fractures of the intact rock. The fact that reasonable fault plane solutions were possible also indicates that, although these events are characterised by a large tensile component, there is still a significant amount of shear-slip behaviour occurring during the propagation process, and it was found that the mean distribution of first motion polarities were 50% tensile, (+ve first motions). Thus, these events could not be considered purely tensile in nature, but are considered to have a combined shear-tensile mechanism. For similar $E_s/E_p$ ratio events studied at the Strathcona Mine, this mechanism was also postulated by Urbancic (1991).

From identification of the normal faulting mechanism with that of the A-set and the fact that the application of the PCA technique to the entire cluster indicates a similar direction after the point of disassociation, it was originally envisioned that potentially this normal faulting mechanism may become more dominant in the post-peak strength state of the rock mass, giving rise to the instability in the PCA trend switching strike and dip to the lower quadrant. However, as will be identified in the next section, this was not the observed response.

### 5.3.4 Temporal Changes of Faulting Mechanisms

The most probable fault plane orientations based on the previous discussion, have been plotted for the 5 selected periods, (A to E), in Figure 5.11 to observe if there is a change in their distribution and orientation similar to that identified by the PCA technique. Here all fault mechanisms, regardless of type, are plotted on the same stereonet for each period. In addition to the orientation the relative quantity of the various mechanisms has been summarised in Table 5.5 and pictorially in Figure 5.12. As can be seen from Figure 5.11, for periods A, B and C, covering pre-interaction, interaction and localization, the trend of the fault plane solutions stays relatively stable, dominated by the reverse faulting mechanism oriented similarly to the dominant PCA trend. This is still maintained at the point of disassociation, but the strike of these reverse planes starts to spread and rotate slightly from a mean strike of $074^\circ$ to $057^\circ$ in Period E, indicating the potential rotation of the stress field. Additionally, although relatively minor, after the point of disassociation, (Period D), there is the development of reverse faults with a different orientation from those determined to be similar to the dominant PCA trend, a drop from 80% to 60%, (Table 5.5 and Figure 5.12). A cluster of these is oriented [189,39], with other more random orientations noted. However, that being said, there is no substantial change in the propagation direction of these fractures prior to interaction and through the point of disassociation. The significance of this will be discussed later.
A. Pre-Interaction

B. Interaction

C. Coalescence and Localization

D. Point of Disassociation

E. Post Disassociation

Figure 5.11 Temporal breakdown of all probable fault planes for the various periods (a) Pre-interaction, April 19 2002 to January 31 2003 - Period A, (b) Interaction, February 12 2003 to March 4 2003 – Period B, (c) Coalescence and Localization, March 6 2003 to April 30 2003 – Period C, (d) Point of disassociation and post disassociation, June 15 2003 to December 30 2003 – Period D, and (e) Post disassociation, January 1 2004 to April 16 2004 – Period E.
Figure 5.12  Visual summary of the proportion of changing focal mechanisms determined for the various temporal periods A to E.
Table 5.5 Summary of focal mechanism type determined based on the most probable fault plane solutions for the various temporal periods.

<table>
<thead>
<tr>
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<tbody>
<tr>
<td>Type #</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
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</tr>
<tr>
<td>Reverse PCA</td>
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<td>10</td>
<td>67</td>
<td>34</td>
<td>30</td>
<td>167</td>
</tr>
<tr>
<td>Reverse Other</td>
<td>0</td>
<td>0</td>
<td>3</td>
<td>14</td>
<td>12</td>
<td>29</td>
</tr>
<tr>
<td>Normal A-set</td>
<td>8</td>
<td>4</td>
<td>4</td>
<td>4</td>
<td>0</td>
<td>20</td>
</tr>
<tr>
<td>Normal Other</td>
<td>1</td>
<td>1</td>
<td>3</td>
<td>2</td>
<td>4</td>
<td>10</td>
</tr>
<tr>
<td>Strike Slip</td>
<td>2</td>
<td>0</td>
<td>6</td>
<td>5</td>
<td>3</td>
<td>16</td>
</tr>
<tr>
<td>Total # of FP’s</td>
<td>37</td>
<td>15</td>
<td>83</td>
<td>59</td>
<td>49</td>
<td>243</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
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<th></th>
<th></th>
<th></th>
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<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Reverse PCA</td>
<td>70%</td>
<td>67%</td>
<td>81%</td>
<td>58%</td>
<td>61%</td>
<td>69%</td>
</tr>
<tr>
<td>Reverse Other</td>
<td>0%</td>
<td>0%</td>
<td>4%</td>
<td>24%</td>
<td>24%</td>
<td>12%</td>
</tr>
<tr>
<td>Normal A-set</td>
<td>22%</td>
<td>27%</td>
<td>5%</td>
<td>7%</td>
<td>0%</td>
<td>8%</td>
</tr>
<tr>
<td>Normal Other</td>
<td>3%</td>
<td>7%</td>
<td>4%</td>
<td>3%</td>
<td>8%</td>
<td>4%</td>
</tr>
<tr>
<td>Strike Slip</td>
<td>5%</td>
<td>0%</td>
<td>7%</td>
<td>8%</td>
<td>6%</td>
<td>7%</td>
</tr>
<tr>
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<td>100</td>
<td>100</td>
<td>100</td>
<td>100</td>
</tr>
</tbody>
</table>

What can be identified is that during the period of coalescence and localization, these reverse faults dominate the fracture development, upwards of 80% (Table 5.5 and Figure 5.12), forming the macrofracture structure or process zone. This gives an understanding to why the ellipticity determined through the PCA increases significantly in this period. As these en-echelon fractures interact the group behaviour becomes more planar and dominated by this mechanism, creating the relative stability in the dominant PCA trend. Throughout the failure process reverse faulting dominates, which is to some extent not surprising given the relatively horizontal orientation of the maximum principal stress.

The normal faults, having a similar orientation as the A-set, can be seen as the minor cluster of poles in the lower quadrant of the stereonet (Figure 5.11). These are more dominant during the period prior to interaction (Period A) and at interaction (Period B), but become less dominant during coalescence and localization, with the normal faults with this orientation becoming weaker in the post disassociation stage. The percentage of all the normal mechanisms drops from around 25% pre-interaction/interaction to around 10% post localization for this sub-cluster.

The strike-slip faults that were determined stay at a relatively constant rate during the failure process at around 5% to 8% of all the mechanisms, (Table 5.5 and Figure 5.12), and due to their low numbers do not form a substantial clustering of poles unless analysed separately as in Figure 5.10d. As previously mentioned, these events appear to be closely related to the FW raise, (Figure 5.7), and if the probable mechanism is related to fracture propagation in line with the B-set, these events may be related to the fracturing around the raise or a region of the raise
that had previously failed and are not associated with the predominant macrofracture shear structure formation identified through the reverse faulting mechanism. Alternatively, as previously mentioned, these events along with the normal faulting events, may be related to readjustment of the rock mass during the failure process.

5.3.5 Summary of Behaviour and Comparison to Other Sites

Based on this analysis, although it must be recognised that we are only observing around 1/3 of the events at the core of the cluster and only dealing with those that triggered a large number of sensors, it appears that even before interaction and localization occurs, in the pre-interaction stage following fracture initiation, individual fractures are propagating in the direction identified by the dominant PCA trend at localization. From the analysis of the group behaviour using the PCA technique during this time, when the fractures are occurring relatively randomly throughout the rock mass, there is no identification of a stable trend. The stability in the dominant PCA trend does not start to occur until interaction of the fractures starts, as was identified from the analysis of the source parameters showing a significant fall in the strength of the events (Chapter 4). During localization there is strong agreement between the direction of the individual rupture direction of the reverse faults and that determined from the group behaviour as defined by the PCA. The point of disassociation determined from the PCA analysis results from the region of the rock mass up dip of this core cluster becoming aseismic, probably as the result of significant rock mass dilation, (as noted by floor heave in the drift above), and resulting in probably significant strain softening behaviour. It appears that, even though a significant proportion of the rock mass has failed to what would be considered it’s residual strength, in regions that still maintain local confinement but are also probably strain softened, fracture propagation still continues in the direction that initiation started in. Thus, the stable PCA trend that occurred following interaction is disrupted due to the aseismicity and becomes unstable, as fracture communication over a broad region is now inhibited. The observed group behaviour of the fractures is now probably more related to the orebody geometry, giving rise to an orientation similar to the A-set, which strikes parallel to the orebody.

Originally, it was envisioned that shear on the A-set might be occurring following the point of disassociation, and would potentially result in an increase in normal faulting with this orientation due to a rotation in the stress field. However, this is not the case as the normal faulting appears to reduce significantly at the point of coalescence and localization. That being said, it must be recognised that this sub-set of data may not capture all the mechanisms of the events that
trigger a fewer number of sensors. The fact that normal mechanisms are occurring in this strongly horizontally dominated stress field, suggests that there must be some stress heterogeneity or a readjustment of the rock mass. Observation of normal faulting mechanisms have also been identified in similar mining induced stress fields such as at the Strathcona Mine, (Urbancic, 1991), in which, although the majority of faulting mechanism were determined to be reverse (~ 60 to 70%), a significant proportion of normal mechanisms were also identified (30 to 40%). Similarly, from a focal mechanism study of a sill pillar under high horizontal stress at the Lockerby Mine, (Sampson-Forsythe, 1994), it was identified that again the majority of the events fit a reverse faulting mechanism with the addition of normal faulting. Of interest in this study, it was found that prior to, and during a major increase in seismic activity related to partial mining of the sill pillar, (Figure 5.13), approximately 60% of the events in the sill were reverse and 40 % normal faulting mechanisms. In the post active period following mining activity, during which time the seismic rate reduced, the percentage of normal faults also significantly reduced down to 20%, with 80% being reverse. In Figure 5.13 the stress axis determined from the reverse faults in the sill pillar and linear elastic modelled stress orientations indicated along with the nodal planes, can be seen. There is similarity between the orientation of these fault planes and the stress orientation, (although at a flatter angle), and those identified here, these events possibly also forming a macrofracture shear structure. However, in this Lockerby study as with others, (Bird, 1993; Urbancic et. al., 1993; Connors et. al., 1993; Coulson, 1996; Mercer, 1999), there was a pre-occupation in trying to fit the most likely fault planes to mapped joint structure; this may lead to inaccurate interpretation of the actual behaviour. The joint structure in some case does undoubtedly control some of the fracturing. It is felt however that the dominant fracture propagation direction should be taken at face value, as has been identified in studies at the URL, (Feigner and Young, 1992, Collins, et. al., 2002; Reyes-Montes, 2004), in the case when the structure does not exist. Fracturing of the intact rock and creation of new fractures must then occur. In these studies it was found that, based on limited data, the stress axis of the source mechanisms generally aligned with the induced principal stress direction.

Here, the structure does not exist with a similar trend or density to warrant trying to fit the fault plane solutions to the nearest structure, and in the following sections investigation of the predominant direction in relation to the stress field and material properties will be investigated.
Figure 5.13  Lockerby Sill Pillar Study (after Sampson-Forsythe, 1994), determined P-, B- and T-axis for 194 reverse faulting mechanisms in the Sill, with model induced linear elastic stress tensor orientations indicated. Reverse Fault plane solutions with probable fault planes oriented in a similar fashion to Golden Giant.
5.4 Stress Analysis

5.4.1 Ubiquitous Joint Analysis

In order to correlate the significance of the trend determined from the principal components analysis and the fault plane solutions, a ubiquitous joint analysis is performed similar to that carried out in Chapter 3 at the Williams mine. As previously discussed, in this type of analysis the joints do not interact with the model, but are used to calculate the normal stress and maximum shear stress developed on planes of a certain orientation, and by comparison of these to one another identify the most likely to fail, (Coulson, 1996; Falmagne, 2002; Diederichs et al., 2002; Connors et al., 1993, Samson-Forsyth, 1994). As the linear elastic model over predicts the stress magnitudes, as failure cannot develop, it is more important to compare the relative magnitudes than to interpret the absolute values. Here, as the maximum principal stress strike is predominantly perpendicular to the plane of the orebody, with the minimum principal stress vertical, and the predominant PCA or fault planes, C-set and A-set striking parallel to the orebody, the problem has been simplified to a two dimensional transverse plane bisecting the 4600-S2 stope at the core of the cluster at section 10480E, (see Figure 5.1). The model analysed here is the basis for the two dimensional non-linear finite element model using Phase², (Rocscience, 2006), discussed in the next section. Here the simplified model is run over multiple mining steps with application of the mine wide stresses from the three dimensional boundary element model, (Chapter 4), to the tractions on the exterior boundary of the 2D model, (see Crowder and Bawden, 2005 for details of application), run in both a linear elastic mode and non-linear mode applying brittle-plastic post peak parameters determined from back analysis of actual displacements at the Williams Mine, (Chapter 3; Crowder et al., 2006), as a first approximation and discussed in the next section. The linear elastic ubiquitous joint analysis can also be performed using the 3D boundary element model, however, for time and ease of analysis the Phase² 2D model was used.

A similar analysis is applied following the methodology discussed in Chapter 3, and the three different material models which were reviewed with the ubiquitous joint analysis, using a Mohr-Coulomb model, (Eqn. 5.1), are summarised again below:

i. Cohesion = 10 MPa, Friction = 30° – equivalent to the rock mass strength and upper most strength of a joint with strong infill material, assuming a simultaneous mobilization of cohesion-friction,

ii. Cohesion = 0 MPa, Friction = 30° – equivalent to a joint or fracture that is pre-existing and not healed, and
iii. Cohesion = 10 MPa, Friction = 0° – Used to determine the lowest strength possible based on purely cohesional failure, assuming the non-simultaneous application of friction and cohesion.

Mohr-Coulomb Criterion:

\[ \tau = C + \sigma_n \tan \phi \]  

[5.1]

where: \( \tau \) = the shear strength (MPa), \( C \) = the material cohesion, \( \sigma_n \) = the normal stress to the plane and \( \phi \) = the internal friction angle

From the linear elastic run the stress conditions were observed at the mining step relating to the start of localization in February 2003, however, as the stresses vary little from 2002 until 2005 any mining step would produce similar results. The normal, \( \sigma_n \), and maximum shear stresses, \( \tau \), for planes of various orientation can be calculated and are plotted in Figure 5.14 as joint factor of safety, (FOS=joint shear strength/shear stress). The three orientations in this two dimensional plane that are important are the C-set, (inclined south at \( \sim 0^\circ \) to 25\(^\circ\); local orientation [090, 0 to 25]), the dominant PCA trend, (inclined south at \( \sim 40^\circ \) to 60\(^\circ\); local orientation [090, 45]) and A-set (inclined north at \( \sim 55^\circ \) to 65\(^\circ\); local orientation [090, 60]). It can be seen here, based on the conventional simultaneous mobilization of cohesion and friction, that visually overall the most likely orientations to fail are from the upper limit of the potential C-set, (Figure 5.14b), and the orientation related to the fault plane solution or the PCA, (Figure 5.14c), over a perfectly horizontal plane or a plane oriented similar to the A-set (Figure 5.14a & b respectively).

In order to see the effect of the variation of the orientation and material strength model on the FOS, an analysis box was placed at the location of the core of the seismic cluster, (Figure 5.14), and the average FOS of all the query points has been determined for each model and orientation and plotted in Figure 5.15a. From this plot, the minima of the FOS lines gives the most likely orientation to fail for the three different models. Both the models with application of friction, (models i. and ii.), give minima at lower orientations than the PCA or predominant fault plane orientation, [090, 25] and [090, 20], respectively, appearing to indicate the C-set orientation. Only the purely cohesive model gives a minima that approaches this orientation [090, 40]. If the average shear and normal stress path histories for the two orientations of [090,25] and [090,45] based on the query lines indicated in Figure 5.14c are plotted, this also appears to indicate that the lower dipping plane is slightly more likely to fail (Figure 5.15b).
Figure 5.14  Linear elastic ubiquitous joint analysis based on the stress state at 02-2003 above the 4600 level David Bell Stopes on a section (10480E) at the core of the EOS cluster, view looking West. Plots of joint or plane factor of safety (FOS) for the Mohr-Coulomb joint model of $\Phi = 30^\circ$ and $C = 10$ MPa. (a) FOS for plane oriented horizontally (Strike R/ Dip [090,00]), (b) FOS for plane oriented sub horizontally, simulating the C-set orientation [090,25], (c) FOS for plane oriented sub vertically, simulating the dominant PCA or Fault Plane orientation [090,45], also indicated are the ubiquitous joint planes oriented at 25° and 45° used to determine the shear and normal stress path histories and (d) FOS for plane oriented sub horizontally, simulating the A-set orientation [270,60].
Figure 5.15  Linear Elastic Ubiquitous Joint analysis based on the stress state at 02-2003 for the EOS region plotting (a) variation of mean factor of safety (FOS) versus joint orientation, measured from the horizontal counter clockwise (CCW), for three material models of i. $\Phi = 30^\circ$ and $C = 10$ MPa, ii. $\Phi = 30^\circ$ and $C = 0$ MPa and iii. $\Phi = 0^\circ$ and $C = 10$ MPa. Note also indicated is the predominant range of the PCA and Fault plane trend (b) Shows the shear and normal stress path histories on two joint orientations (see previous figure) from 12-1989 to 10-2003.
This contrasts to the analysis performed for a similar failure at the Williams Mine, (Chapter 3), in which the PCA trend was identified as definitely the most probable orientation to fail over the C-set, based on all three models, but with the cohesive model having the best fit. One key difference that was identified here is that in this 2D linear elastic model the maximum principal stress direction indicated by the tensor orientation, (plotted as crosses in Figure 5.14), is not homogenous or purely horizontal as in the case at the Williams mine. It was found from investigation of non-linear modelling, (discussed in the next section), in which failure can occur, that the stress tensor orientations in the region rotate to become homogenous and horizontal, (identified for all non-linear modelling parameters plastic and brittle), primarily due to localized failure occurring above the back of the David Bell stopes. These stopes were mined originally in 1993, and based on the seismic history the region directly above these stopes, (3 to 5 m), was always aseismic during the monitoring period, indicating that local failure probably occurred at the time of mining, (note this is not by caving but probably by pervasive extensional ‘tensile’ fracturing).

Hence, based on this, the ubiquitous joint analysis was also performed on the base case non-linear model, using post peak brittle-plastic parameters determined through back analysis of near field displacement at the Williams Mine, (Case 1 – Table 5.6), for the mining step prior to the development of shear failure at the core of the cluster, (12-1995). Note again here, as we are not sure of the exact model determined stress magnitudes, a comparative analysis between the various orientations and joint models is more appropriate. The joint FOS for the same orientations analysed in the linear elastic model are displayed for the joint model with simultaneous mobilization of friction and cohesion, in Figure 5.16a to d. As can be noted, the rotation in the principal stress field, becoming more horizontal, now makes a plane oriented [090,45] slightly more likely to fail than the sub-horizontal C-set [090,25], (Figure 5.16c versus Figure 5.16b). Comparison of the average joint FOS for the three joint models can be seen in Figure 5.17a, and it can be noted that the minima are shifted to the right due to the slight stress rotation. Here again, for the two models with application of friction, (models i. and ii.), the minima give an orientation that is slightly less than that observed from the PCA and fault plane solutions, at [090, 40] and [090,35] respectively, but greater than the C-set. The best fit is obtained from the purely cohesive model, which has a minima at [090,50], identical to the observation of the direction indicated from the reverse fault plane solutions and the dominant PCA trend at localization. It is interesting to note that, due to regional failure in the non-linear model, the lowest FOS region for different joint models corresponds roughly with the core region in which the microseismicity developed. The lowest FOS region was found for the cohesive only model, at a plane orientation of [090,50] in Figure 5.16e.
Figure 5.16 Non-Linear (Parameter Set 1 – Table 5.6) ubiquitous joint analysis based on the stress state at 12-1995 on a section (10480E) at the core of the EOS cluster, view looking West. Plots of joint or plane factor of safety (FOS) for the Mohr-Coulomb joint model of $\Phi_r = 30^\circ$ and $C = 10$ MPa. (a) FOS for plane oriented horizontally (Strike R/ Dip [090,00]), (b) FOS for plane oriented sub horizontally, simulating the C-set orientation [090,25], (c) FOS for plane oriented sub vertically, simulating the dominant PCA or Fault Plane orientation [090,45], also indicated are the ubiquitous joint planes oriented at 25$^\circ$ and 45$^\circ$ used to determine the shear and normal stress path histories and (d) FOS for plane oriented sub horizontally, simulating the A-set orientation [270,60] (e) Minimum FOS contour for orientation [090,50], for the purely cohesive material model.
Figure 5.17  Non-Linear (Parameter Set 1 – Table 5.6) Ubiquitous Joint analysis based on the stress state at 12-1995 for the EOS region plotting (a) variation of mean factor of safety (FOS) versus joint orientation, for the three material models of i., ii. and iii. Note also indicated is the predominant range of the PCA and Fault plane trend (b) Shows the shear and normal stress path histories on two joint orientations (see previous figure) from 12-1989 to 10-2003.
Analysis of the FOS for these models indicates that the second orientation observed from the PCA [255, 65] similar to the A-set would be less likely to fail as it is more strongly clamped. The direction of the conjugate based on this analysis would be oriented [270, 40], at a considerably lower dip than was observed from the PCA. This may indicate that if conjugate shears are developed in the other direction, then the PCA was not able to identify them clearly, potentially due to the dominance of primary direction of the macrofracture structure and the complexity of the fracturing process. Minor conjugate directions from the PCA were observed at the point of disassociation (see Figure 5.24 d), however, the overall second orientation is skewed by the combined behaviour. Alternatively the second observed trend may be simply due to the vertical expression of the group behaviour following the trend of the orebody.

The average shear and normal stress paths for histories for two investigated orientations, [090, 25] and [090, 45], based on the query lines indicated in Figure 5.16e are plotted in Figure 5.17b. For this non-linear model, it can be seen that the maximum shear stress is developed early in 1993 after the mining of the top David Bell stopes, and due to regional failure, (based on an imposed brittle model criteria), decreases significantly in 1996 as shear failure develops in the core of the cluster with a similar orientation. The reason why this failure develops considerably earlier than reality will be discussed in the next section but emphasises a limitation of the 2D modelling approach. Comparison of the two stress paths indicates, a plane oriented [090,45] is more likely to fail than the other orientation based on the cohesional strength, but is only slightly more likely to fail when friction is considered.

### 5.4.1.1 Rationalization of the Macrofracture Orientation Based on Stress

Based on laboratory testing it is known that the angle of the failure plane, (shear zone) $\beta$, in relation to the maximum principal stress orientation can be approximated. For brittle rocks, at low confinement, this angle can be close to zero with the proliferation of extension fracturing parallel to the applied load. With an increase in confinement the angle can increase up to 30° to 35° as extension fracturing is inhibited and can be approximated based on the well known Mohr-Coulomb criterion discussed above, here expressed in terms of 2D principal stresses:

$$\sigma_1 = 2C \tan(\beta) + \sigma_3 \tan^2(\beta) \quad [5.2]$$

$$\beta = 45 - \frac{\phi}{2} \quad [5.3]$$

where: $\beta$ = the angle between the failure surface or rupture surface and the maximum principal stress.
Although, as Hoek and Brown (1980) point out, this formulation is an over-simplification as it does not adequately describe the behaviour of natural occurring anisotropy, but here as the loading direction is around 60° to 70° to the foliation (A-set) the material can be considered to be close to it’s peak non-anisotropic strength. Also, Hoek and Brown (1980) proposed a slightly more complex formulation based on the Hoek and Brown failure criterion and Balmers equation, however, for simplicity the Mohr-Coulomb criterion will be discussed. The failure direction identified from the fault plane solutions and the dominant PCA trend appears to define the direction of maximum shear stress, and indicates that the direction of fracture propagation is related primarily to cohesive failure of the material, and that friction at the time of initiation is close to zero. This suggests a similar concept as proposed by Martin and Chandlier (1994), based on work by Schmertmann and Osterberg (1960), that the during laboratory cyclic damage testing of Lac du Bonnet Granite it was identified that at the start of yield, the crack damage point, the strength of the sample is dominated by the cohesive component with the friction initially being zero. As damage increases cohesion is gradually lost and friction is mobilized, the peak friction being reached once most of the cohesion has been lost. If we consider that the rock mass strength is based on the concept of the weakest link, then if the direction identified from the PCA and fault plane solutions is not related structurally or due to material anisotropy, but to fracturing of the intact rock, this may indicate that internal friction of the material is not relevant to the formation of the direction of the macrofracture structure under shear conditions. If internal friction (~30°) were considered, then theoretically, (based on Equation 5.3), the fault plane solution should have occurred at a lower angle of around 28° to 38° based on the P-axis plunging slightly to the south (2°), but here the fault plane solutions have a dip that varies between 40° to 60° from horizontal, (38° to 58°) to the P-axis or the maximum principal stress direction.

The concept of the macrofracture (PCA) and individual fractures (fault planes) orientations occurring in the direction of maximum shear is fitting based on earthquake seismology. In earthquake seismology it has been recognised that the P- and T-axis should not necessarily equate to the principal stress direction if pre-existing structure exists, and could be in error of 35° to 40°. However, for fresh faults generated in intact rock, the P- and T- axis are commonly assumed to be at 45° to the nodal planes at the point of maximum shear. In the evaluation of the principal stress direction in the conterminous United States based on fault plane solutions, Zoback and Zoback (1980), made the assumption the P- and T-axis lie at 45° to the nodal planes and that the various trends due to pre-existing faults would tend to cancel out. This again implies that the frictional strength of the material does not affect the rupture direction especially in the creation of fresh fractures. Where it is thought that the overall material friction
does come into play is in controlling the tortuosity of the rupture, the amount of dilation, and the amount of slip on the rupture plane. Additionally, the amount of dilation is also related to the confinement and the potential for extension fracturing postulated as occurring during fracture interaction. Here these are not pure shear events, as from their source parameters the $E_s/E_p$ ratios are generally less than 5.0, suggesting that a combined shear-tensile ruptures with a definite tensile (dilational) component.

Diederichs (1991), performed discrete element modelling (PFC$^{2d}$) of laboratory scale uniaxial and triaxial samples compared to lab testing of Lac du Bonnet Granite, (Martin and Chandelier, 1994). In order to calibrate the models to the laboratory strength and behaviour, the particle bond strength was controlled by the shear strength to normal strength, (S/N), ratio and for most of the modelling this ratio was 4 or greater, resulting in suppression of shear cracks over tensile normal crack formation. These analyses would give equivalent strengths and shear zone (macrofracture) orientations, $\beta = 26^\circ$ to $30^\circ$ for unconfined and 60 MPa confinement respectively, to the laboratory tests, even when an inter-particle friction angle after failure of $45^\circ$ was used. If the particle bond strength ratio is relaxed, (S/N < 1), then shear cracks dominate, and the strength of the samples was significantly reduced at higher confinement with the formation of the shear zone or macrofracture structure at an orientation of $\beta = 45^\circ$. This may be similar to what is seen at the rock mass scale, and points out that the behaviour seen in the laboratory in terms of the orientation of shear zones may not be directly applicable at the rock mass scale.

5.4.2 Non-Linear Modelling and Estimated Post Peak Parameters

In this case study at the Golden Giant mine in the analysis region no direct measurements of local or distant rock mass deformation were made during the time of the failure, unlike the similar study performed at the Williams Mine (Chapter 3). Hence, it is not possible to perform explicit modelling, and detailed parametric back analysis to match the modelled displacements with those measured in the field. At the Williams mine a detailed parametric study was carried out for a drift below a region that had considerable microseismic activity develop in the form of a similar macrofracture structure as discussed earlier. From this study, it was identified that the best, (although not exclusive), parameters to fit the near field displacement in the back of a highly stressed drift, were brittle-plastic parameters using a non-associated flow rule for the residual strength with the incorporation of dilation to enable a volume change post peak in the continuum model (Chapter 3). The peak or failure surface was determined from the rock mass properties based on the generalized Hoek and Brown failure criteria (Hoek et. al., 2002). It should be pointed out that the this model can only exhibit brittle-plastic behaviour, and not
gradual strain softening; i.e. once the stress level of the peak failure envelope has been reached the material properties of the failed element fall to the post-peak residual strength immediately, and follow a non-associated flow rule along the residual envelope (See Figure 5.19). True strain softening would create a more controlled and gradual fall to the residual level but requires a constitutive model such as that proposed by Hajiabdolmajid et. al. (2002). These parameters were, however, found to fit the near field, low confinement ‘bulking’ region surrounding the drift with reasonable agreement in recorded displacements, (Crowder et. al., 2006, Chapter 3), although, it was not possible to maintain model stability to create failure in the more distant rock mass, due to boundary instability issues (Chapter 3). Here it was hoped that this might be overcome, however, other issues that will be discussed arose.

Although no field deformation information exists at Golden Giant, it is possible to make observation of the behaviour, based on the mining experience and the microseismicity. It is postulated, similar to the Williams case study, that strain-softening or strain-weakening behaviour occurs following localization of the seismicity, (peak to post-peak state), and more significant strain-softening occurs at the point of disassociation. The observational evidence for this is that, prior to localization, significant stress effects were noticed in terms of stress spalling and back failure in the 4620 level drift above. Additionally, it was known that the stresses were greater than the initiation stress, (also determined through linear elastic modelling, Chapter 4), sufficient to cause onset of significant microseismicity and failure of drill holes through dog earing, and crushing or offsetting of the most distant toes of the holes identified during mining of the first few stopes of the destress slot. However, following the significant flurry of microseismic activity in this cluster, significant floor heave in the 4620 L, (~0.5 m), occurred around the same time as the point of disassociation, with a large proportion of the upper part of the rock mass engulfed by the microseismic cloud becoming aseismic, and assumed to fail to a residual strength. The dilations seen in the 4620 L floor heave, based on an analogy to the Williams case, indicate strain-softening behaviour, (i.e. stress can only build again, once this dilated rock mass is compressed enough to close dilated fractures). The stress in this region must have decreased substantially, to at least a level below the initiation threshold, as during mining of the stopes in this region, (4600-S3 and S2 stopes), no significant drilling problems were encountered, and it was reported by the mine personnel that these stopes were significantly easier to extract than anticipated, (Toppi, pers. com., 2005). If we assumed that the linear elastic stresses prior to failure were achievable, this would equate to a decrease in stress from 120 MPa, (prior to disassociation), to 84 MPa, (initial initiation), a reduction of 35 MPa (Chapter 4). Incidentally, during the monitoring of the shaft during mining of the destress slot to shadow the shaft, the stress reduction created by the destress slot was modelled linear
elastically to reduce between 20 to 30 MPa. Vibrating wire stress cells installed within the shaft, had only limited stress reduction monitoring capacity, and although they started to follow the modelled stress, saw a stress reduction of only 5 to 10 MPa. However, based on the reaction of the shaft and no observable stress deterioration following subsequent mining, the linear elastic stress reduction to a level below the initiation stress was determined to be reasonable, (McMullan et. al., 2004; Bawden, 2003).

The base case for the non-linear modelling is application of the post-peak parameters determined at the Williams case study. It was found from this study that using a Mohr-Coulomb failure criterion for the peak and residual strength indicated more strongly defined shear development similar to that observed from the microseismicity. Preliminary modelling of the geometry was performed by applying the 3D boundary element stresses, at each mining step, to the boundary of the non-linear model as tractions to simulate the induced stress from global mining. In this preliminary model the base case parameters, (Table 5.6 - Parameter Set 1), were applied to the entire domain, and showed distinctive shear zones develop in a similar orientation to that identified from the PCA and fault plane solutions (Figure 5.18b). However, it was found that this shear develops much earlier than seen in reality, and the model was found to become unstable at the subsequent mining step due to failure propagating to the boundary of the model. This results from sill pillar failure occurring in the David Bell stopes below in 12-1995 (Figure 5.18a). What is poignant regarding this is that in 1995, when David Bell mined these stopes, significant failure in the sill is thought to have occurred, the high stresses causing the blast to be frozen, due to incomplete detonation due to stress offsetting of boreholes. This was the same time that a small flurry of seismic activity occurred at Golden Giant on the 4600 level (Nickson et. al., 1998).

In order to build a stress history and failure history essential for a non-linear stress path, it is necessary to follow the mining sequence, which requires excavating the stope geometry at the appropriate mining step and backfilling the stopes. In these models an appropriate backfill stiffness, (Table 5.6), was used but only allowed to behave linear elastically reducing computation time. The key issue here is that in this more complex model geometry over a simple drift geometry, the 2D plane strain model assumes the excavation be infinite out of plane, this is a correct assumption for the final geometry, but not for the sequential mining, as in reality the stope extraction is of a smaller volume, with reduced deformations. The 2D linear elastic runs of this geometry and sequencing create similar stress distributions to the 3D boundary element model, however, for the non-linear modelling, the deformation created on excavation of the stopes causes substantially more failure to occur than is observed in reality.
This causes the failure to develop at a mining step significantly earlier than expected. Additionally, applying brittle-plastic parameters to the entire domain of this large model can create numerical instability with failure propagating to the model boundaries. In order to try and increase stability of the model two approaches were adopted. Elastic material softening prior to excavation was already being used to aid stability; i.e. the elastic modulus is gradually reduced over mining steps before complete extraction by, 66%, 33%, 25%, 10%, 5% and 2% allowing the model to converge at each stage, (Crowder et. al., 2006).

Figure 5.18  (a) Geometry of Phase² model for section 10480E with boundary tractions applied. (b) Non-linear model using parameter set 1 (base case Brittle-Plastic analysis) applied over the entire domain (here the domain was not perfectly plastic), showing development of similar shear at the core of the EOS cluster, at mining step 12-1995.
Table 5.6 Summary of Non-linear Parameters Analysed.

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<th>Post Peak – Residual Strength</th>
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<td>( s )</td>
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<td>Back Fill Properties – Linear Elastic(^6)</td>
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</tr>
</tbody>
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Notes:

\(^1\) Based on limited triaxial testing of Hemlo core from Golden Giant Mine (Queen’s, 1994), UCS and Tensile strength (MPa) based on testing of core by various facilities (Chapter 3).

\(^2\) Simplified intact parameters used in this and other modelling studies (Crowder et. al., 2006)

\(^3\) Rock mass strength (Peak Strength) based on Hoek et. al.(2002) with \( D=0 \) and \( a=0.5 \) and best post peak Brittle-Plastic parameters determined at Williams Mine (Crowder et. al., 2006; Chapter 3)

\(^4\) Rock mass equivalent Mohr-Coulomb parameters based on Hoek-Brown parameters\(^3\) using RocLab (Rocscience, 2002; Hoek et. al. 2002). Note cohesion was increased from 10 to 14 MPa, and tensile strength from 5 to 10 MPa, in this study to increase stability.

\(^5\) Material properties applied outside of the analysis region in the domain (Figure 5.18 – blue region)

\(^6\) Backfill properties (Sand fill assumed equivalent to Paste fill) included for completeness, but material run as linear elastic based on \( E=20 \text{ MPa} \) and \( v = 0.3 \) (le Roux, 2004)

\( E_i \), Intact Modulus = 55 GPa

\( E_m \), Rock mass Modulus = 17.8 GPa (based on Hoek et. al.(2002) with \( D=0 \))

\( v \), Poisson’s Ratio = 0.25
Instead of completely mining the stopes, application of the backfill properties were performed after the material had been softened to 25% of its modulus. However, as the modulus of the backfill (0.02 GPa) is still substantially lower than reduced elastic properties at this stage, (i.e. 4.45 GPa), this only reduced the deep failure slightly. The other approach to create global model stability was to define a zone of interest for adjustment of various post peak parameters, but for the region outside of this to the boundary, to only allow the rock mass to fail perfectly plastically, (Figure 5.18a). This allows for continuity of displacements, and maintains the stress level as if the domain was linear elastic, but limits the failure in the bulk region of the domain away from excavations. This was successful in achieving global model stability in mining steps beyond failure in the region of interest. However, this model manipulation does mean that actual displacement magnitudes in the region may not be representative.

This parametric study investigates the effect of brittleness, on the level of stress developed at failure with the identification of the development of shear zones in the area of interest. The base assumption is that the rock mass peak parameters are reasonably appropriate to characterise the strength of the rock mass, as was found at the Williams Mine study. In the model a 3 m zone around the excavation is defined, this is the local ‘bulking’ zone identified from modelling and monitored displacements (Chapter 3). In this zone the post peak parameters are maintained based on the base case brittle-plastic parameters determined from back analysis. The region outside of this zone for separate models had the residual strength increased from this base case to a perfectly plastic solution. The modelling parameters analysed on this basis are summarised in Table 5.6 and the peak and post peak envelopes graphically displayed in Figure 5.19. The dilation was assumed to vary linearly, from a maximum value of 16.7° determined for the base case to a value of zero, (i.e. no dilation and no volume change), for the perfectly plastic case. This is based on the assumption that as the brittleness decreases then so does the effect of dilation, based on the concept that as we move away from the low confinement region of the excavations, to the deeper field rock mass where the confinement increases, the rock mass brittleness decreases and dilation is inhibited, (Cundall et. al., 2003). It was found in the Williams case study that decreasing the dilation allowed for more pronounced shear development, as this reduces the increase in confinement post yield. However, to see if the dilation has an effect on the shear zone deformation of the perfectly plastic case the maximum dilation was also applied.
Figure 5.19 Comparison of Peak failure envelopes and Post-Peak residual envelopes used in the parametric analysis using Phase². Based on Table 5.6. Also, shown is a typical element failure path for brittle-plastic post peak behaviour.

Originally, these modelling parameters were going to be adjusted to analyse actual displacement measured above a drift to the north of the zone and above on the 4633 level which were monitored from 2004 to 2005. However, because of limitations of the 2D model, only general comments regarding the relative resultant behaviour of different post peak parameters will be discussed.

The maximum principal stress contours for each of the 6 models based on the parameters sets in Table 5.6, are displayed in Figure 5.20. In these plots shear failure of elements is identified as red crosses (x) and tensile failure as filled red circles (o). In all of the brittle-plastic models (Figure 5.20 a to d), dominant shear failure develops at the same mining step of 1996-06, at the core of where the seismicity develops, and has a general inclination of 45° to 50°. In the two perfectly plastic models, one with dilation set to zero, (Figure 5.20e), and the other with dilation set to 16.7°, the development of shear failure does not occur at any orientation matching that identified from the PCA or reverse fault plane orientations.
Adding dilation to the perfectly plastic model does little to alter the behaviour. Considering the stress magnitudes after failure of the zone, as the brittleness is reduced then the zone of higher stress increases. In the perfectly plastic models the stress of course does not reduce at all, giving a stress distribution similar but slightly higher than the linear elastic case (Chapter 4). This indicates that, as we know the stress must have fallen below the initiation threshold to allow ease of mining, then this mode of behaviour is unrealistic, and only some form of brittle
failure and strain-softening behaviour will give reasonable and more realistic results. The resultant stress levels after failure for residual strengths in parameter set [3] and [4] (Figure 5.20c & d), are probably still too high and would indicate that a realistic stress reduction can be seen for a residual strength related to the base case or a slightly higher residual strength (parameter set [2]).

Although this non-linear modelling was not totally successful, primarily due to the time mismatch from the modelling sequence, and should not be explicitly interpreted as absolute values to create similar displacements, it does indicate that the post peak constitutive behaviour of the rock mass here, in which similar shear failure develops in the model compared to the microseismicity and formation of a macrofracture shear structure, could be through strain-softening behaviour, while perfectly plastic behaviour is not representative of the eventual failure state of the pillar. In order to remove the limitations developed from this 2D model, full 3D non-linear modelling would be more appropriate, and it would be envisioned that greater model stability could be achieved (Andrieux, pers. com., 2007). This would then allow for better calibration of the post peak parameters, through back analysis of actual displacements if they could be obtained. With a stable 3D model it would also be interesting to investigate the change in the strength of the rock mass through degradation by effectively reducing the strength of the rock mass in line with the microseismic clustering.

One major issue with all continuum modelling, is the need to determine a model appropriate dilation based on the friction term, in order to match the field deformations (Chapter 3). This dilation may or may not be directly equivalent to the real dilation of the material, but this parameter needs to be directly related to the developed confinement in the model, as setting absolute values for the entire continuum is incorrect. Additionally, as previously mentioned application of a volumetric dilation is unrealistic under low confinement close to excavation surfaces, where the dilation is unidirectional towards the drift in the form of bulking (Kaiser et al., 2000). The other physical issue is that the true amount of dilation resulting from these postulated combined shear-tensile ruptures needs to be actually measured for improved calibration. In view of some of the limitation of continuum modelling, the author feels that greater progress in understanding the constitutive behaviour of rock masses should be made with looking at synthetic rock masses using Distinct element formulation in PFC (Itasca, 2008) or Continuum/Discontinuum models such as ELFEN (Rocfield Software Ltd.) and (FEM/DEM - Munjiza and Latham, 2004), however, these will undoubtedly develop their own issues to resolve.
Beck (2008), promotes the use of multi-scale 3D strain softening, dilatant material models, incorporated into the finite element code of Abaqus©, to replicate displacements and failure behaviour of complex mine geometries over a broad range of scales. This type of modelling requires detailed displacement measurements over various scales and stress states, to be able to calibrate post peak parameters through iterative modelling runs and setting displacement objective criteria for models at various levels (Beck, pers. comm., 2007). Unfortunately little of the details of these modelling problems have been reported in the literature, and for most practitioners this modeling is currently not practical or affordable.

5.5 Fracture Network Analysis

In an attempt to visualize the formation of the macrofracture structure network and fracture interactions during the failure process a simple fracture model and geometry is proposed. It is postulated that part of the dilatational energy developed in these low $E_s/E_p$ ratio ruptures, (i.e. these have a strong tensile, $E_p$, component with a depletion in shear wave energy, $E_s$), may partially come from development and propagation of extension fractures. Crack interaction has been modelled by a number of authors through the primary concept of the sliding crack model developed based on Linear Elastic Fracture Mechanics (LEFM), (e.g. Kemeny and Cook, 1991). Martin (1997) notes that this model shows a relationship between extension fracture lengths and the ratio of $\sigma_3/\sigma_1$ similar to what Hoek and Bieniawski (1965) measured in the laboratory for a series of experimental tests on annealed glass plates (Figure 5.21a). Although Hoek and Bieniawski’s fracture geometry assumes an ‘open’ Griffiths elliptical fracture, which has not been seen microscopically, (Hajiabdolmajid, 2001), this is the only fundamental experimental work that physically tries to determine the length of extension fractures formed in a stress field in a brittle material.

In reality it can be considered that the opening of the fractures will result from the tortuosity of the shear rupture plane. This will also result in a dilatational component from the rupture as well as the dilatation produced from propagation of the extension fractures. It can be seen from Figure 5.21a that, for both models under exceedingly low confinement, extension fracture lengths (c) can propagate to larger lengths exceeding the sliding crack diameter ($c_o$), in the direction of the maximum applied load. A noted difference is that in the sliding crack model these extension fractures have a maximum length as $\sigma_3/\sigma_1 \rightarrow 0$ of $c/2c_o = 1.5$, and extension fractures are totally inhibited with increased confinement at $\sigma_3/\sigma_1 = 0.2$. 
Figure 5.21 (a) Single isolated crack geometry, based on a Griffiths elliptical crack with tensile extension wings (after Hoek and Bieniawski, 1965) compared to the sliding crack model (after Kemeny and Cook, 1991). (b) side view of proposed fracture geometry, (c) plan view and (d) 3D view.

The experimental work of Hoek and Bieniawski, indicates that at low confinement, \( \sigma_3 / \sigma_1 \to 0 \), the extension fractures have a length that may exceed this, and as the confinement increases, \( \sigma_3 / \sigma_1 > 0.2 \), the extension fractures still form but at a finite length \( c / 2c_0 \to 0.05 \). This finding is more useful and more realistic as it allows for the formation of small extension wings to occur for a single isolated fracture at high confinement. These extension fractures would be expected to grow as a result of fracture interaction where the local confinement field is perturbed and reduced due to other cracks. This local reduction in confinement for en-echelon fractures has been identified in simple linear elastic modelling by Diederichs (2000), as a means to identify that the ‘effective confinement’ is reduced in the presence of open fractures by the applied stress less a crack closure pressure. The key point is that once the cracks have formed, the...
local confinement field normal to the cracks is reduced changing the stress state of the rock mass.

A rudimentary fracture geometry based on Hoek and Bieniawski’s (1965) work, is determined from analysis of the linear elastic stress states at the core of the seismic cluster prior to localization, ($\sigma_1 = 110$ MPa and $\sigma_3 = 48$ MPa), and from the analysis of the non-linear stress state prior to regional yield, forming a worst case scenario for extensile fracturing ($\sigma_1 = 65$ MPa and $\sigma_3 = 5$ MPa). Thus, based on these stress states extension fractures for a single isolated fracture of unit radius could vary in length from 0.12 to 0.22 m respectively. Realistically the difference is negligible, and a length of 0.2 m is assumed primarily to give an indication of where the extension fracture would be established. Based on this, a rough fracture geometry for a 1 m radius rupture is displayed in Figure 5.21b to d. For simplicity it is assumed that the rupture plane is circular, however, in reality the trace of this plane might be oval in shape, the long axis relative to the shear direction. Hoek and Bieniawski (1965) determined, based on applying the work of Griffiths (1924) and McClintock and Walsh (1963), that the critical crack orientation, $\Psi_c$, at which the maximum and minimum stresses are induced near the crack tip can be found from equation [5.4] below.

$$\cos 2\Psi_c = \frac{\sigma_1 - \sigma_3}{2(\sigma_1 + \sigma_3)}$$

[5.4]

Here, based on the linear elastic and non-linear stress regimes, this would equate to 39° and 32° respectively, the former being closer to what is observed from the microseismicity. For simplicity here and based on the orientation of the dominant PCA trend and reverse fault plane solutions to the principal stress direction, the basic orientation of the crack in relation to the principal stress is taken as 45°, such that the extension fracture of the simplified geometry propagate in the direction of the principal stress.

This fracture geometry has been scaled to the determined Madariaga source radius obtained for each event and plotted for all of the analysed fault plane events, (Figure 5.22a); (Periods A to E), rotated to the orientation determined from the most probable fault plane solutions, (Note the extension fracture tips are only applied to the reverse faulting mechanisms). This can be compared to the plotting the same fracture geometry, sized again to the source radius, plotted for all of the events during localization (Period C), using the PCA derived plane orientations (Figure 5.22b). Period C is chosen as this is the time in which greatest agreement exists between the PCA and fault plane orientations. In three-dimensions, these en-echelon fractures visually form a planar looking structure when rotated.
Figure 5.22  (a) All fault planes  (b) PCA derived planes during localization Period C.
As already noted, by plotting the different failure mechanisms, it appears that the normal faults are distributed generally throughout the core of the cluster of events and to the east of the raise, while the majority of the strike slip events appear to be directly in line with the raise. As already identified these strike slip events may not be associated with the development of the macrofracture structure, but may be related more to north south fracturing related to the raise. There is a general agreement between the overall geometry of the macrofracture structure determined based on the individual fault plane orientations and that of the PCA derived orientations based on the group behaviour.

To more readily see the development and potential interaction of the event/fractures, sections have been cut though the core of the clustering events. The fracture geometry can be seen for the fault plane solutions in Figure 5.23a for the analysed sub-set, and in particular the time of localization, Period C (Figure 5.23b). It can be seen that, particularly for Period C on section 10480E at the mid point of the cluster, these events form a relatively flat structure, produced by en-echelon fracturing. This analysis for the fault plane solutions is based on just the sub-set of data determined to give the best success for good solutions. In order to see the potential macrofracture formation based on all the events in the cluster, isolated fracture geometry has been imported based on the PCA derived planes for each of the time periods, and sections cut at the same points to see the relative changes. It should be remembered that outside of the time of fracture interaction, (Period B), and coalescence/localization, (Period C), the fracture plane orientation may not reflect the individual fracture directions, as the PCA technique observes the group behaviour. The development for the macrofracture structure and the increase in ellipticity, however, can easily be visualized (Figure 5.24a to d). It can be seen in the pre-interaction state, following fracture initiation, that the fractures are isolated and relatively randomly distributed through the domain (Figure 5.24a). Following the point of interaction these fractures start to coalesce, and give a stronger definition to the identified dominant PCA trend (Figure 5.24b). At the point in time that there is a significant increase in the ellipticity, (Figure 5.3a), during Period C, the events in the core of the cluster coalesce and interact more closely, (increase in the fracture density), with the definite localization on section 10480 E to a planar macrofracture structure as can be seen in Figure 5.24c. After the point of disassociation, (Period D), the fractures do not develop over the same regional extent, up and down dip, and appear to be distributed more sparsely, with the group behaviour having a more volumetric than planar structure resulting in the reduction in ellipticity. There is also the observance of possible conjugates oriented at low angles (Figure 5.24d), however, the overall averaged group behaviour being steeper.
Figure 5.23 Visualization of fracture network based on fault plane solutions for (a) Sections cut through all fault planes (b) Sections cut through fault planes recorded during localization (Period C)
Figure 5.24a & b  Visualization of fracture network based on PCA derived planes for (a) Sections cut through PCA derived planes for Period A – pre-interaction and (b) Period B – Interaction to Localization.
Figure 5.24 c & d  Visualization of fracture network based on PCA derived planes for (c) Sections cut through PCA derived planes for Period C – Localization and (d) Period D – Disassociation to Post Disassociation.
It can also be noted that the relative proximity of some of these fractures is almost coincidental. Previously it was suggested that due to the low $E_s/E_p$ ratios that these events were not shear slip on pre-existing structure but fresh fracture formation, (Chapter 4), which is probably true for the majority of the events. If some of the events are occurring at virtually the same location, there may be reactivation on previously fractured rupture planes. It would be expected that the $E_s/E_p$ ratios of these events should therefore increase, and although higher ratios were noted during localization, overall the mean $E_s/E_p$ ratios fall after interaction (Chapter 4). This may suggest that the reason for the lower ratios after interaction is due to dilatational effects, (i.e. dilation of the rock mass post peak, which would result in strain-softening). In order to investigate this more thoroughly, actual observation of both changes in the fractured volume and the stress level would be required. Alternatively, as discussed later, these fractures may be related to tensile fracture formation in the form of Riedel shears (Cho et. al., 2008).

Ideally, it would have been beneficial to also obtain a physical analysis of the fractures from diamond drill core before and after the formation of this structure. Unfortunately the region was mined and the mine closed before this could be done. However, from a detailed mapping study of a ‘fresh mining induced’ fault zone by Gay and Ortlepp (1979), some strong parallels in the structure can be drawn. In this study a shear zone was identified and mapped in detailed by ramping up the shear zone from the footwall. It was thought that this shear zone was the origin of two large magnitude events, (3.4 and 2.1 $M_R$), resulting from high induced stress at the mining front in the East Rand Proprietary Mine, S.A.. It was found that this normal fault zone, (shear displacement of up to 10 cm were observed), was made up of smaller, en echelon shear planes, which were connected by subsidiary conjugate shear and extension fractures (Figure 5.25a and b). If the fracture geometry that was determined on section 10480 E for the PCA determined structure during localization, (Figure 5.24c), is rotated to the same stress orientations as Gay and Ortlepp’s fault, (note the reverse faults become normal faults if rotated by 90°), there is considerable agreement in the structure if the inferred extension fractures are extended, and there is also strong similarity in the spacing, size and orientation of these shear fractures. Of interest also, is that these macrofracture shear structures and sub-shears are oriented at approximately 45° to the principal stress axis, confirming to some extent the observation that these stress related fault orientations are based on a cohesional failure and are not related to the material friction angle. Also, this suggests that here we are observing the genesis of a larger fault zone developed from the microseismic activity. Unfortunately in the case of Gay and Ortlepp (1979), no microseismic system existed to record the smaller events that probably occurred to develop their shear zone.
Associated with Large 3.4 and 2.1 M events

After Gay and Ortlepp, 1979

Figure 5.25  (a) Mapping of a normal fault in South Africa (After Gay and Ortlepp, 1979, (b) Location of the normal shear in relation to mining front (c) Rotated image of PCA derived planes on Section 10480 E, image rotated clockwise 90° about the B-axis (σ₂–axis) to align stresses with the Gay and Ortlepp’s fault.

Also, if this macrofracture shear structure had been developed over a larger region, similar to the Williams Mine footwall macrofracture shears, then large magnitude events could have possibly been initiated. A number of large events did occur at Williams but no direct link to the orientation of the macrofracture structure could be made. Trifu and Urbancic (1996), suggested coalescence of mapped joints as the likely cause of a large magnitude event at the Strathcona
mine, while here it has been identified that coalescence and localization are related primarily to new fracture creation, and not the result of pre-existing structure.

Additional evidence that this macrofracture shear structure is developed based on en echelon fractures could be inferred based on the laboratory direct shear testing carried out on a brittle synthetic rock by Cho et. al., (2008), who's interpretation is based on the Riedel Experiment (Riedel, 1929; Tchalenko, 1970). In the experiment originally on clays, en echelon shears and their conjugates are referred to as Riedel Shears (R and R’ respectively) as indicated in Figure 5.26. The Riedel shears form at an angle to the shear direction of approximately $\Phi/2$, where $\Phi$ is the internal angle of friction. These Riedel shears have been observed in natural shear zones in brittle rocks on the microscale (Figure 5.26c) and on the macroscale of earthquake shears (Figure 5.26d).

![Figure 5.26](image)

**Figure 5.26** (a) Diagram of the Riedel experiment, (R) Riedel Shears, (R’) conjugates Riedel shears (after Tchalenko, 1970). (b) Discrete Riedel Shear fractures observed in clay experiments (i.e. $\Phi$ is internal angle of friction) and Skempton’s principal displacement fracture connecting Riedel shears (after Cho et. al., 2008) (c) Tectonic shear zone in diorite, UCS=212MPa, showing a natural shear zone (after Cho et. al., 2008), and (d) Dasht-e Bayaz earthquake fault (after Tchalenko, 1970).
Figure 5.27  Development of a shear fracture for a 1.85 MPa normal stress test on PFC synthetic rock (after Cho et. al., 2008)

Cho et. al. (2008) performed shear experiments on a synthetic rocks and in order to understand the shear fracture development performed calibrated numerical simulations using clumped discrete element modelling (PFC). One of the higher normal confinement simulations is shown in Figure 5.27. Their fracture development interpretation is based on the Riedel clay shear box experiments of Tchalenko (1970), shown in Figure 5.28a, in which before and at the
Figure 5.28 (a) Fracture pattern developed at each stage of stress-strain curve on overconsolidated clay (after Tchalenko, 1970 modified by Cho et. al., 2008). At peak shear strength (Stage A), the first Riedel shear appear at 12° to the horizontal. (Stage B) Riedel shear are extended and a few new shears are generated at about 8° to horizontal. (Stage C) New shears named “P shears” appear at an inclination of – 10°. (Stage D) Principal displacement surface is formed. (b) PCA derived planes indicating direction of shear development at localization. (c) Fault plane solution derived planes indicating direction of fractures at localization.

peak shear stress (Stage A) en echelon Riedel shears form at an inclination $\Phi/2$ to the direction of shearing. After the peak shear stress (Stage B in Figure 5.28a), as shear deformation increases these Riedel shears extend. Before the shear stress approaches the residual strength, (Stage C in Figure 5.28a) the Riedel shears become kinematically impossible and a new type of fracture called Thrust shears (also known as “P shears”) develops which is approximately symmetrical to the Riedel shears but in the opposite direction. As the shear stress approaches the residual strength (Stage D Figure 5.28b), the Riedel and P shears coalesce and further displacement forms the principal shear plane (Cho et. al., 2008).

Cho et. al. (2008) identified similar formation from the PFC modeling, however, with the formation of the R and P shears starting to occur pre-peak (Figure 5.27). At peak there is a
significant occurrence of R type fractures with some minor occurrence of R’ and P. At stages C and D approaching the residual strength there is a significant increase in the R type fractures showing definite en echelon formation, with also increases in the number of R’ and P type fractures. From the laboratory test it is interesting to note that the R’ fractures were only observed in the latter stages of the test as the residual shear stress was approached. Additionally, it was identified that these fractures in the modelling formed primarily through tensile failure (Figure 5.27), and not shear failure, with modeled ratios of Tensile/Shear cracks going from a ratio of 2.0 in the pre-peak to 3.0 to 4.8 in the post peak. If we draw a direct comparison to the development of microseismicity in this macrofracture structure, the $E_s/E_p$ ratios were always predominantly lower than expected indicating a strong tensile component to the events. The ratio was seen to gradually increase pre-peak or pre-coalescence, and only a few large $E_s/E_p$ events were noticed at localization (inferred as the peak strength). Following localization and into what is considered the post peak, there was a marginal reduction in the $E_s/E_p$ ratio to lower than the pre-peak average. It was originally anticipated that these fractures would become more shear enriched following localization but this was found not to be the case. This indicates that the model that assumes shear slip for microseismic events needs to be reviewed, and although the tensile model (Cai et. al., 1998) gives rupture radii that appear too small, these findings indicate that we may be preferentially dealing with tensile fracturing, which could be similar to the development of Riedel shears.

If we consider that the events in this macrofracture structure are the development of Riedel shear in tension, then it could be inferred that these are tensile en echelon fractures with the resultant deformation occurring as shear and recorded by the polarities of the seismic system to obtain fault plane solution. The PCA direction and the predominant fault plane solution during localization are giving the overall direction of shearing (Figure 5.28b and c), the point of disassociation may be related to Stage C (Figure 5.28a) when the development of Riedel fractures become kinematically impossible and conjugate R’ fractures are formed with “P shears”. This change in the behaviour and creation of greater degrees of freedom results in dilation and the breakdown of the strong dominant trend. It can be observed from (Figure 5.28b and c) that we are dealing with a visually similar structure to that observed in the Riedel experiment and observed in the formation of shear zone on a larger scale (Gay and Ortlepp, 1979; Tchalenko, 1970). One important point to note however, is that the Cho et. al., (2008) experiments were based on relatively low normal loading (1 to 1.85 MPa) in comparison to the theoretical normal loading observed from the ubiquitous joint analysis which is in the range of 20 to 30 MPa. Further research into the potential formation of this shear structure using
clumped discrete element modelling with PFC may be able to shed more light on the actual development but was outside the scope of this thesis.

5.6 Conclusions

The fault plane solutions of 243 representative events, taken at various time frames during the progressive failure of a confined region of the rock mass that was postulated to fail through the formation of a macrofracture shear structure, were analysed to observe the fault plane orientations and mechanism. It was found that the overall mechanisms were mainly reverse faulting (82 %) with some normal faulting (12 %) and minor strike-slip or dip-slip faulting (6 %). The later is thought not to be related to the macrofracture shear structure but to a raise in the footwall due to their location. Pre localization it was found that the normal faulting events were more significant at around 34 %, but at the point of localization significantly reduced to only 9 % with 85% of the events having a reverse mechanism.

- Analysis of the mainly reverse faulting events showed significant strong correlation in orientation to the dominant trend determined at localization from a principal component analysis (PCA) of the clustering events; the mean fault plane orientation identified as [076,47] compared to [076, 52] from the PCA method. This strong correlation between the fault plane solution of the individual events/fractures and that of the group behaviour indicates that the overall direction of the event cluster is a function of the individual event/fracture direction, the events coalescing and localizing to the macrofracture structure orientation.

- The orientation of the reverse fault plane solutions is at a greater dip than the nearest sparsely populated joint set, (C-set), and indicates that these events are the generation of fresh fracture through intact rock and not slip on pre-existing features. The $E_s/E_p$ ratios of these events is low, (< 5), and indicates that, although there is a strong tensile component, these events are probably combined shear-tensile ruptures, as focal plane solutions were possible. The dilatational (tensile) component has been postulated as partially the formation of extension fracturing, which may be expected to increase during fracture interaction, but also some dilation of the fracture, related to the generation of the rupture surface.

- Alternatively, it may well be that these fractures are tensile ruptures in the form of Riedel shears, based on the findings of Cho et. al., (2008).
• Comparison of the fault plane stress axis with induced stress tensor orientations calculated from three-dimensional linear elastic modelling of the mining geometry were also found to be in agreement, indicating additional confidence in the solutions.

• The orientation of the individual reverse faulting mechanisms was found to be relatively stable during all phases of the failure process, from pre interaction, interaction, localization to post disassociation. This indicates that, although the PCA technique cannot resolve a stable orientation prior to event interaction, the fractures have already started to initiate in the direction of failure at localization, but are relatively randomly distributed throughout the region.

• Post disassociation, the dominant PCA trend changes and becomes unstable, switching between the dominant trend and one that either represents the trend of the orebody (A-set orientation) or the poorly resolved formation of conjugates or P shears formed as previous fracture development becomes kinematically impossible, creating greater degrees of freedom. This change in behaviour is also identified as being related to partial aseismicity of the cluster of events in regions of the rock mass that are inferred to be dilated and strain softened. Because the predominant fault plane orientation is maintained, this suggests that fractures are still propagating in the direction of the localization at the core of the cluster, but the overall group behaviour is no longer as strongly associated to this same orientation.

• Both of the two previous points identify the advantage of performing first motion studies, in that without confirmation of the mechanisms, it could have been possible to misinterpret the fracture directions if the PCA trends had been taken as face value. However, based on this study, confidence is gained in confirming the direction and formation of the macrofracture structure at localization, and as first motion or moment tensor analyses are exceedingly labour intensive, the PCA technique is a useful and efficient tool to interpret confined stress driven failures. Additionally, there are significant advantages to studying the group behaviour of clusters of events, primarily the identification of localization through the ellipticity and also the point of disassociation, which, through the study of the source parameters or fault plane solutions alone, does not give an indication in the change in the group behaviour as patches of the rock mass fail.
• From a ubiquitous joint analysis, the orientation of the fracturing based on the fault plane solutions and the dominant PCA trend at localization, appears to be related primarily to the direction of maximum shear. This is determined based on a joint material model with cohesion only, which is more likely to fail in shear over the range of orientations observed, than if friction is simultaneously applied to the joint model. The orientation of the C-set is less likely to fail in shear than the observed fracture direction. This suggests that the direction of this fresh fracture propagation is related to cohesional failure of the material and that friction is only mobilized once the fractures have formed.

• The simplified two-dimensional non-linear modelling indicates that similar shear development and a drop in the stress level to below the initiation stress could be achieved by application of post peak brittle-plastic parameters, resulting in strain softening behaviour, while perfectly plastic behaviour is not representative of the eventual failure state of the pillar. The base case parameters, determined from back analysis of near field failure at the Williams mine, appear to give a reasonable representation of the shear development at the core of the seismic cluster, and provide sufficient brittleness to cause the stress level to fall below the initiation stress level. If a perfectly plastic model is applied, there is of course no decrease in the stress level and no shear development as observed through the microseismicity.

• In order to improve the non-linear modelling stability when applying brittle-plastic dilatant post peak parameters and correct representation of the progressive mining stress history, this geometry should be modelled using at least a three-dimensional continuum model. But, as identified in Chapter 3, this would require more detailed displacement monitoring, in order to determine the correct dilation to apply post peak, and it is recommended that a variable dilation parameter based on confinement is incorporated into the routine. Additionally, application of unidirectional dilation in low confinement regions close to the drift opening where bulking predominates should be investigated as this may also impact on the progressive development of shears further away.

• A simplified fracture geometry based on a Griffith crack model with extension fracture wing’s, sized to the source radius, was used to observe the fracture network created. Observations based on sections taken through the core of these fractures at localization indicated the formation of a relatively spatially compact macrofracture shear structure
made up of en echelon fractures. This fracture geometry is similar to that identified through underground mapping of a large mining induced shear by Gay and Ortlepp (1979), and the development of a shear band based on Riedel shears (Tchalenko, 1970; Cho et. al., 2008). It is postulated that what is being observed here, through the microseismicity, is the formation of a fault zone with the potential to produce large magnitude events. As the failure formation studied at the Williams mine (Chapter 3), is similar to the one studied here, this could suggests that the formation of the macrofracture shear structure could possibly be the primary site for the genesis of the large magnitude events at the Williams mine. The main detractor from this theory is that the observed damage in the development related to the large events, was proposed to be on a structure more distant than the core of the seismic event density on the 9390 Level, further to the south and above (Bawden and Jones, 2005), but may have been related to similar macrofracture shear development on the level above.

- It was not possible in this study to perform moment tensor inversion due to the limited number of triaxial sensors. The advantages of moment tensor inversion are that the proportion of isotropic (tensile or compressive – volume change), components versus double-couple (pure shear), and deviatoric (complex shear) components of the events can be evaluated as well as fault plane solutions determined. It would be anticipated that during the failure process there should be some significant changes in the general model ratios, similar to that observed in laboratory testing of triaxial samples using AE (Thompson, et. al., 2005), although it must be recognised that the stress path in the laboratory especially when using servo controlled feedback is different from the loading path in the field. Based on this study, it would be recommended that a minimum 8 triaxial sensors are placed approximately 100 m distant from the area of interest.
CHAPTER 6

CONCLUSIONS AND RECOMMENDATIONS

6.1 General Comments

At the commencement of this study it was envisioned that the rock mass would react more plastically, (close to perfectly plastic), with an increase in confinement away from the openings, the microseismicity being more enriched in shear wave energy. However, the evidence found in the two case studies, (Chapter 3, Chapter 4 and Chapter 5), does not support this. The observed eventual behaviour of these confined failures showed definite strain softening or strength weakening behaviour, resulting in a drop in the stress level to below the stress initiation level, allowing for easy drilling and stope extraction in what was deemed stress relaxed regions. This is also supported by the study of the source parameters at the Golden Giant mine (Chapter 4) which indicate, based on the $E_s/E_p$ ratios, that the events that were recorded in the development of the macrofracture shear structure, had a strong dilatational (tensile) energy component, suggesting a combined shear-tensile rupture. This dilation is also supported by the field displacement instrumentation, which after large deep dilations or shear (distant from the excavation boundary) occurred, caused changes in the group behaviour of the seismicity indicated from the point of disassociation identified by the PCA method (Chapter 3). This point of disassociation is believed to be related to the formation of conjugate fractures or shearing which must develop kinematically creating greater degrees of freedom, and results in patches of the rock mass becoming aseismic, which are inferred to be related to a dilated and strain or strength softened rock mass at close to it's residual strength.

The non-linear modelling that was performed in this study (Chapter 3 and Chapter 5), although not completely successful in an explicit sense at modelling the macrofracture shear structure development, did indicate that the only way that the similar shear structure could develop would be with the application of brittle-plastic post peak parameters. Additionally, a perfectly plastic model does not result in the required stress shedding, to be able to lower the stress state to below the stress initiation level.
The modelling and the displacement instrumentation suggest that two environments exist. The near field-bulking zone surrounding the immediate excavation is probably characterised by stronger brittle behaviour with increased dilation. The more distant confined regions that failed, results from the development of a macrofracture shear structure or process zone, in which smaller dilations are probably occurring during the formation in the post peak, but become more brittle with sudden increased dilations occurring when the rock mass fails to close to it’s residual strength after the point of disassociation. Undoubtedly, without the presence of the excavations this behaviour would not have been the same, and suggests that in the presence of openings, where the majority of microseismic events are recorded, the rock mass will generally react in a strain softening manner for relatively hard rocks. This can only be investigated more thoroughly by performing more routine studies similar to this.

These interpretations are based on assuming that the final state of the rock mass observed from the mining behaviour (i.e. the absence of and inferred stress reduction post peak) has resulted from strain softening behaviour. An alternate interpretation is that the formation of this macrofracture shear structure occurs under ductile conditions, creating shear banding at the centre of the confined pillar zone. There is a change in the strength state, from ductile to brittle due to an overall change in the pillar geometry from short and squat creating significant confinement and ductile behaviour, to a more slender pillar with brittle behaviour (Kaiser pers. comm., 2009). This is not a change in the material behaviour but a change in the strength state due to geometrical influences. The effect of the change in the geometry should be investigated but was outside the scope of this research.

Analysis of the temporal changes in the spatial trend of the seismicity using the principal components method, combined with an estimate of the clustering density, appears to show strong potential for determining the state of the regional rock mass as it progresses through the various stages of failure (Chapter 3 and Chapter 4). It is important to recognize that the type of regional rock mass failure that this method was applied to in these studies is developed under conditions of progressive almost monotonic loading of principal stresses, (i.e. failure of a confined rock mass, without caving). In stress failures also involving caving where the geometry is continuously changing, it is expected that the development of fracturing may complicate the trend of seismicity.

The basis for determination of the strength states of this confined failing rock mass is based on an analogy of the microseismicity to laboratory triaxial testing of intact rock samples with monitoring of acoustic emissions (Chapter 1). The assumption made is that fundamental
mechanics of failure at various scales are similar, and it is the magnitude and amplitude of
the deformations that are scale dependent. The objective of this research was to see if the
stages of failure can be identified from the microseismic analysis, field displacement monitoring
and numerical modelling to understand the post peak behaviour of confined failures of the rock
mass, from initiation to aseismic behaviour. This has not been studied in detail previously and
was found to be relatively successful. The major conclusions and contributions based on the
observations of the failures studied in this research are discussed in the following sections and
more detailed conclusions can be found at the end of each case study chapter (Chapter 3,
Chapter 4 and Chapter 5).

6.2 Microseismic Analysis

Through analysis of the three dimensional group behaviour of the seismicity using the PCA
technique, combined with analysis of source parameters and faulting mechanisms, it has been
observed that following seismic initiation, (dispersed flurries of seismic activity), and prior to
interaction of events, no stable dominant trend in the group behaviour can be clearly observed
(Chapter 4 and Chapter 5). Based on the focal mechanisms (Chapter 5), however, this does
not mean that fractures are randomly oriented, but are randomly distributed through the domain
of the high stress and high confinement region. The direction of these ruptures or fractures of
the intact rock, which are predominantly reverse faulting mechanisms, are oriented in the
direction of maximum shear stress, suggesting failure of the material is based primarily on
cohesional strength determined from ubiquitous joint modelling at both sites (Chapter 3 and
Chapter 5). The frictional component of strength is probably mobilized immediately following
rupture, and the strong dilatational (tensile) component of the events is postulated as the
formation of limited extension fracturing, (while the events are isolated in space) or these
fractures maybe the formation of Riedel shears (Chapter 5). During this time period as the
principal stresses gradually increase, there is also a gradual increase in the source parameters
strength and stress estimates, with a decrease in the source size suggesting clamping.

As these fractures start to interact, noted by a reduction in the average events source
parameter strength and stress estimates (Chapter 4), greater stability in the PCA derived
planes occurs, with an orientation of the group behaviour matching that of the individual event
focal mechanisms, both oriented in the direction of maximum shear stress (Chapter 5). This
fracture interaction is most clearly noted in the analysis of the source geometric complexity
based on the average dynamic to static stress drop ratio, going from 1 to > 1 and is suggested
as a result of more energetic sub-events occurring due to the failure of rock bridges between interacting fractures (Chapter 4). This may be the point of true yield, however, it becomes more significant as a critical fracture density is reached, observed through the development of the clustering density, (Chapter 2, Chapter 3 and Chapter 4), of 5 events per 125 m$^3$ voxel, (based on an average Madariaga source radius of 1.85 m, to develop potential fracture interaction/ coalescence based on the clustering index). As this critical density is reached, there is a significant decrease in the average event strength/stress estimate and an increase in source size, with the proliferation of these smaller magnitude events, and more easily propagating, interacting fractures dominating the failure process (Chapter 4). This is postulated as the peak strength of the rock mass. Subsequently, the coalescence of these fractures significantly localize to form a macrofracture shear structure or process zone, noted from the PCA technique by an increase in the average ellipticity from around 5 to > 15 in both studies (Chapter 3 and Chapter 4). This is thought to be the start of more regional strain softening with probably greater extension fracture propagation, the events recorded subsequently to this time occurring in the post peak strength state of the rock mass. The source parameter strength and stress estimates (Chapter 4) remain at this lower post peak strength state and do not appear regain to the maximum magnitudes observed pre interaction. An increase in the $E_s/E_p$ ratios to greater than 5 is observed with the formation of this macrofracture structure, (although in this case limited to the peak of the average PCA ellipticity), indicating more shear behaviour may be occurring through either remobilization of previously developed fractures or greater shearing over the interconnected fracture surface.

During this localization the correlation of the PCA derived planes with the a higher percentage of reverse faulting mechanism is greatest (Chapter 5), and suggests that this macrofracture shear structure or process zone is developed through fresh en echelon interacting fractures of the intact rock and is not shear slip on pre-existing structures. When the fracture network is observed, based on fitting a simplified fracture geometry based on a Griffith crack model with extension fractures to the event locations sized to the source radius, a relatively compact and planar structure is noted at localization (Chapter 5). The observed fracture network is comparable to observations of fault zones related to large magnitude seismic events, and may be the same as the formation of Riedel shears en echelon. This indicates that these small mining induced microseismic events may be related to the genesis of a fault zone similar to those observed in earthquake seismology.

The onset of continuous instability in the dominant PCA derived trend, following a significant period of stability, has been termed in this research the point of disassociation, when the spatial
group behaviour of events no longer follows the orientation of fracturing indicated by the individual fault plane solutions. This occurs in both case studies (Chapter 3 and Chapter 4) when patches up dip of the macrofracture structure become aseismic, (no longer produce recordable seismicity at the monitoring frequency), and is postulated to occur with the formation of minor conjugate fractures, creating greater degrees of freedom of rock block movement. From observation of field displacement instrumentation (Chapter 3), this occurs following relatively sudden and large dilations, (at least 10 to >50 mm), at depth away from openings, (7.5 to 9 m), in the region of aseismic behaviour, this being the first time to the authors knowledge that dilations associated to aseismic behaviour have been recorded. The point of disassociation has been postulated as the state in which large regions of the rock mass have undergone significant strain softening, caused by dilation, and is the point in which the rock mass is at, or approaching it's residual post peak strength. Of interest is the fact that the average source parameters of the remaining clustering events show only a slight increase in magnitude of strength and stress estimates, possibly related to redistribution of stress, but stay predominantly at the post peak averages following coalescence and localization (Chapter 4). Also, the fault plane solutions and mechanisms (Chapter 5), do not vary significantly following disassociation, suggesting that the fracturing process still continues in regions not fully strain softened, and which may still have some confinement. Only by observation of the group behaviour, which was combined with displacement monitoring in Chapter 3, is the point of disassociation observed, and indicates the importance of the PCA technique in determining the postulated residual strength state of the rock mass.

These observations of initiation, interaction, coalescence, localization and disassociation are analogous to observations of confined triaxial testing of laboratory samples with monitoring of acoustic emissions (Chapter 1). The key benefits of the PCA technique is that it gives a three-dimensional picture of the behaviour of the events or fractures to one another and, based on the analogy to laboratory testing, can be used in the case of confined failures such as those studied in this research, to obtain a reasonable approximation to the strength state of the rock mass.

The seismic density and the clustering density, based on the clustering index (Chapter 2), give an insight into the point of fracture interaction and the yield point of the rock mass (Chapter 3 and Chapter 4), but is limited and a rough approximation, as it has to be based on cumulative history of the region undergoing failure which is not always available. The PCA technique, however, is not reliant on the cumulative history, and can be used to take a snap shot of the behaviour, making it more conducive to real-time monitoring.
The source parameters provide a useful insight of the fracture interaction up to the peak strength, however, in this study once unstable fracture propagation and coalescence has occurred, these do not appear to provide more information based on the overall average of the clustering events (Chapter 4). Likewise the fault plane solutions confirm the concept of en echelon fresh fracture propagation in the formation of these macrofracture structures (Chapter 5). The dominant PCA trend observed at localization in the study at Golden Giant conforms strongly to the individual fracture directions, which continue in regions that do not become aseismic. The key draw back to rigorous focal mechanism studies is the significant time required in determining the first motions. The advantage of the PCA technique is using the 50 event sampling method and temporal analysis is to greatly reduce the time for observation of the orientation of this type of failure.

6.3 Numerical Modelling

A considerable amount of time and effort was taken in this research to build a full three dimensional mine model encompassing all thee mines to obtain firstly an accurate representation of the mining geometry in relation to the mine induced microseismicity, but to also provide a relatively fine picture on a month to month basis over a number of years of the linear elastic stresses that were developed (Chapter 3 and Chapter 4), and used also for sub-modelling using non-linear analysis (Chapter 3 and Chapter 5). Although the time involved is not represented in terms of the discussion in this thesis, without this three-dimensional model less would be understood with regards to the stresses and seismicity developed in relation to the mine openings. For any mine analysis of this kind, a three dimensional model of the mine geometry of not only the stope excavations, but also the critical infrastructure development, is crucial.

Based on the postulated strength state, determined from the microseismic analysis, three-dimensional linear elastic modelling can be used as a first approximation to determine regions that are in failure. It was observed that initiation of seismic activity and yield based on coalescence and observed localization occur at almost the same stress level (Chapter 3 and Chapter 4). Based on the Hoek-Brown Brittle failure parameters, \( m=0 \), to determine linear damage limits, fracture initiation, coalescence and localization was found to occur at a stress level, based on the deviatoric stress term \( (\sigma_1 - \sigma_3) \), between 0.3 to 0.37\( \sigma_c \), \( s= 0.09 \) to \( 0.14 \), \( \sigma_c= 175 \text{ MPa} \), at the Williams and the Golden Giant sites respectively. The modelled linear elastic confinement, \( (\sigma_3) \), was on average, 25 MPa and 45 MPa respectively. This initiation/localization
damage limit shows some confinement dependence and, by applying a modified Tresca envelope, \((\sigma_1 = A\sigma_c + B\sigma_3)\), a fit to the data is obtained with \(A=0.22\), \(B=1.5\). The point of disassociation was found in both cases to be at a slightly higher damage limit of 0.375 to 0.4\(\sigma_c\) for the Williams and the Golden Giant studies (Chapter 3 and Chapter 4). The key observation is that, although the linear elastic modelling can act as a tool for first approximation of the potential rock mass damage limits, the minor differences in the stress levels and the variation observed at both sites means that, without some form of observed calibration, the timing of failure may be over or under predicted, based solely on the variation of the uniaxial rock strength (Chapter 4). Additional factors that could effect this prediction are: the discretization of the model, application of the correct \textit{in situ} stress field, and stress shedding caused by failure of the rock mass outside of the region of interest. Thus, without calibration, only potential regions of failure can be determined and not the exact timing. It was also identified that, when using linear elastic modelling, Mogi’s Brittle-Ductile transition zone of \(\sigma_1/\sigma_3 < 3.4\), may not be appropriate as a behavioural guideline to determine the eventual post peak behaviour of the rock mass. The overall behaviour observed at Williams and Golden Giant mines was perceived to be brittle strain softening due to the removal of stress effects. Additionally, the suggested limit proposed by Hoek and Brown (1980), of \(\sigma_1/\sigma_3 < 2.0\), may be more appropriate for the rock mass (Chapter 3 and Chapter 4), or disturbed laboratory samples, but this needs further investigation.

From the simplified 2D non-linear modelling, the post peak parameters that could achieve similar shear failure development and a drop in the stress level to below the initiation stress, were a brittle-plastic dilatant model, simulating strain softening behaviour (Chapter 3 and Chapter 5). For linear elastic modelling the peak rock mass strength, based on the Generalized Hoek-Brown failure (GBH) criterion or equivalent Mohr-Coulomb (MC) criterion, using the GSI as a strength reduction factor, under predicts the extent of failure. For non-linear modelling however, based on the back analysis of the near field, (bulking zone), calibrated to measured displacements, these criteria can be used successfully with appropriate post peak parameters to give reasonable estimation of the displacements (Chapter 3). In this near field bulking zone it was found that the best post peak parameters, based on the GHB criterion, were \(m_r = 0.1m_b\), \(s_r = s_b\) and \(dil = 0.33m_b\) (\(m_b=2.397\), \(s_b=0.0117\), \(a=0.5\)). The parameter of dilation has a great impact on the development of shear away from the bulking zone, and reduction in the dilation parameter creates an increase in the shear development (Chapter 3). Additionally, using the equivalent MC parameters with a tensile cut-off of 5 to 10 MPa and dilation parameter equivalent to 0.33\(m_b\), provided improved development of shear at the orientations observed
from the microseismicity. If a perfectly plastic model is applied, there is of course no
decline in the stress level and no shear development as observed through the
microseismicity.

There is a real need to determine a confinement dependent dilation parameter, as a single
value cannot represent failure in both the near and far field when using continuum modelling.
This can only be achieved with more accurate measurement of displacements in both the
bulking zone and the core of the macrofracture zone. Any non-linear modelling needs to have
these measurements to calibrate the model not just to the depth of failure but also to the
displacements developed such that appropriate support can be placed. Additionally, application
of unidirectional dilation in low confinement regions close to the drift opening where bulking
predominates should be investigated as this may also impact on the progressive development
of shears further away.

6.4 Contributions

The main contributions of this study are listed as follows:

- Definition of the concept of the point of disassociation based on changes in the stable
group behaviour of events/fractures defined from the PCA technique, through the
development of conjugate fractures, and/or the occurrence of partial aseismic
behaviour.

- Measurement of deep dilation or shearing from field instrumentation prior to the point of
disassociation creating potential strain softening and patches aseismic behaviour.

- Identification of the formation of a macrofracture shear structure (fault) developed through
en echelon fracturing for a highly stressed but confined rock mass.

- Identification that these individual fractures propagate in the direction of maximum shear
stress, and control the directional development of the macrofracture structure.

- Determination that these fractures are related primarily to cohesional failure of the
material, being fresh fracturing and not slip in pre-existing structural features.

- Development of a simplified seismic event clustering density which is volume
independent to determine a first approximation to the point of interaction and ‘true yield’
of the rock mass.

- Development of a methodology using the principal component analysis technique and
seismic clustering density to identify failure stages of initiation, interaction (yield
strength), coalescence and localization (peak to post peak strength), and disassociation
(approaching residual strength) of a brittle hard rock confined rock mass.
- Identification of an increase in source parameter strength and stress estimates up to the point of interaction, suggesting loading to the peak strength and significant decrease in strength just prior to coalescence and localization.

- Identification that at coalescence and localization the proliferation of smaller magnitude events dominates the post peak response, with average strengths not able to attain pre-interaction magnitudes.

- Identification that fracture interaction may well be best identified in the specific case analysed, through observation of estimates of the source complexity.

- Identification that the source radii of these microseismic events that form the macrofracture structure are inversely proportional to the source strength estimates, and the dominant frequency may be related to stress and strength state of the rock mass, resulting in non-similar behaviour.

- Based on the low Es/Ep ratios (< 5.0) found in the cluster of events that form this macrofracture structure, that even under confinement these events still have a significant dilatational (tensile) component suggesting a combined shear-tensile rupture and are probably related to fresh fracturing of intact rock and may be the formation of Riedel shears.

- Identification that at localization and coalescence fault plane solutions of the individual ruptures planes strongly correlate to the dominant PCA derived trend of the group behaviour, with the individual fracture direction based on the induced stress orientation being predominantly reverse faulting mechanisms and is maintained throughout the failure process.

- Identification that at relatively high confinement that the post peak response of the rock mass is suggested to be strain softening or strength weakening, and that a ductile (perfectly plastic) constitutive model is inadequate to develop similar shear oriented failure or a reduction in the stress level.

- Identification that use of the intact rock Brittle-Ductile transition definition of $\sigma_{1}/\sigma_{3} < 3.4$ may not be representative of the eventual behaviour if based on linear elastic modelling of stresses and that a behaviour transition definition of $(\sigma_{1}/\sigma_{3} < 2.0)$, may be more appropriate.

- Corroboration that three-dimensional modelling using linear elastic modelling and application of the Hoek-Brown Brittle failure parameters are useful as a first approximation to determine damage limits and regions of potential rock mass yield.
6.5 Recommendations and Future Work

The combined analysis of induced microseismicity with field displacement monitoring instrumentation was essential in determining the potential strength state of the rock mass. Given the cost of these instruments relative to the potential value added in understanding the behaviour of the rock mass, these should be routinely implemented in high stress mines.

6.5.1 Seismic Event Density and Clustering Density

- The definition of the clustering density here was based on the average source radius of around 1.85 m and based on consideration of the clustering index, potential interaction and ‘yield’ can occur at an event density of 5 events per 5 x 5 x 5 m voxel. This is a useful starting point and relatively good first approximation. Improvements in this technique and calibration of clustering index could be made if the clustering index is calculated in each voxel cumulatively with time, but based on individual event source radii. This could easily be contoured to identify regions of high interaction.

6.5.2 Principal Component Analysis (PCA) Technique

- The PCA technique applied to observe the spatial and temporal behaviour of clustering seismic events, was found to greatly enhance the observations that were determined in this research. As this is a non-cumulative analysis, there is greater potential for application of this method on a real time basis. It is strongly recommend as tool to analyse different types of failures in other rock masses, to determine the full effectiveness of observations of the group behaviour of clustering events, even if not associated to similar stress driven macrofracture structures.

- The main PCA parameters determined to obtain smoothing, without compromising sensitivity of temporal variations, was found for a minimum temporal window size of 50 events and applying a spatial window, D, based on either 75% of the cumulative inter event distance, or the mean inter distance plus one standard deviation.

- As a starting point for other analyses it is expected that these parameters should work reasonably well, although if source locations and temporal density of seismicity are significantly different the effect of both should be evaluated.

- It should also be noted that, in order to obtain significant stability and identifiable temporal variation, the failure has to create enough data to be statistically significant, (at least 500 events). Attaining, enough microseismic data is thought to be not just a function of the failure, but the sensitivity of the microseismic array. The sensor density and location accuracy were found to be just adequate at the Williams mine, however, improvements
in both could be made by reducing the mean inter sensor distance ranging from 50 to 100 m used at the Williams mine, to be in the range of 20 to 50 m, used at the Golden Giant mine.

- The primary method used in these studies to identify the correct dominant orientation was through use of lower hemisphere stereographic projections of 50 event temporal window samples. The other method employed was a continuous central moving PCA (CMPCA), in which observation of the time of temporal variations was more easily defined (Chapter 3 and Chapter 4 and defined in Chapter 2). Caution should be used, however, in interpreting the exact values of the mean orientation of PCA derived planes from this method if there is a mixture of different trends at the same time. These linearly averaged trends were generally less in strike and dip than observation of the mean pole concentration based on the stereographic projection. It is recommended that this CMPCA could be more effectively evaluated than the time consuming process of plotting the individual 50 events samples by application of an automatic pole density algorithm, as proposed by Hammah and Curran (1998), for the automatic identification of joint sets.

### 6.5.3 Seismic Source Parameters

- Essential to improved analysis of the rock mass failure using microseismic data is a dense microseismic system with inter-sensor spacing of 20 to 50 m, providing good focal sphere coverage, and with a minimum of 6 but preferably 8 triaxial sensors around the zone of interest at a distance of about 100 m from the centre of the analysis region. This would provide additional benefits not only to the sensitivity of the system and mean source parameter calculation, but also to investigate the components of moment tensor inversion.

- Individual sensor attenuation corrections would also be advised to homogenise the magnitudes recorded. Even without this it has been shown in this study that much can still be gained from the averaged relative changes.

- One key area of the source parameters that needs further research is the effect of confinement on the corner frequency and hence the source radius model. It has been noted here that if the corner frequency, as determined from Andrew’s method, is related to the predominant frequency, then it may be expected that this frequency should increase with an increase in the stress state, and conversely reduce as the stress state is relaxed, as occurs due to strain softening behaviour. Also, the source radius model is
based on the concept of a homogenous double-couple event, represented by a circular rupture plane. Based on the events recorded in this study, the $E_s/E_p$ ratios suggest a combined shear-tensile mechanism that during fracture interaction is geometrically inhomogeneous and complex and may not be correctly represented by this simple model. Although the Madariaga model does appear to give a realistic fracture dimension, it is unclear what role the rock mass system plays on the recorded frequencies, and this needs further research. Additionally, if these ruptures are tensile fractures (Cho et al., 2008), greater investigation into improved development of a tensile model similar to Cia et al. (1999) needs to be performed.

- In order to relate the source model to actual dilations measured in the field, greater monitoring through and around the core seismic cluster, needs to be performed with displacement and stress monitoring instrumentation. Only through measuring the actual physical reaction of the rock mass through the stages of failure into post peak, will the proposed behaviour of strain softening following localization be confirmed, and can only be implied here based on the evidence presented.

### 6.5.4 Focal Mechanisms

- It was not possible in this study to perform moment tensor inversion due to the limited number of triaxial sensors. The advantages of moment tensor inversion are that the proportion of isotropic, (tensile or compressive – volume change), components versus double-couple, (pure shear), and deviatoric, (complex shear), components of the events can be evaluated as well as fault plane solutions determined. It would be anticipated that during the failure process there should be some significant changes in the general model ratios, similar to that observed in laboratory testing of triaxial samples using AE (Thompson, et al., 2005).

### 6.5.5 Numerical Modelling

- The two dimensional non-linear modelling performed in this study suffered from significant limitations of oversimplification of the problem, (Chapter 5), and instability created at the boundaries of this hybrid model when pushed to the limit (Chapter 3 and Chapter 5). To be able to explicitly model the failure process in high stress mining environments to overcome these problems, use of full three-dimensional (3D) non-linear modelling, using either continuum models, (KECK Code – UoT 2007; FLAC3D - Itasca 2008; ANSYS - ANSYS Inc., 2008; ABAQUS – SIMULIA Inc. 2008), or discontinuous/
discrete models, (3DEC, PFC$^{3D}$ – Itasca, 2008, ELFEN – Rocfield Software ltd., 2008), is required. These latter packages, at present, suffer from size limitations and computing power requirements, requiring at present either supercomputer power, or hybrid modeling, (which again can result in boundary issues of the sub-model Beck (2008)). However, it is envisioned, that much can be learned in terms of behaviour from these discrete element models especially PFC$^{3D}$ or even as a staring point PFC$^{2D}$, using a synthetic rock mass (Pierce et. al., 2009). One of the key advantages of this discrete element modelling is that it does not require the development of complex constitutive models.

• For continuum modelling, although greater stability may be achieved in three-dimensions, for brittle-plastic modeling calibration of the post peak dilation is essential in exhibiting the right behaviour and displacements, (see recommendations in the next section). It is suggested that this dilation parameter should reduce with confinement to achieve a realistic residual dilatational flow rule, and application of unidirectional dilation in low confinement regions close to the drift opening where bulking predominates should be incorporated.

• A strain-softening model based on cohesion-weakening/ friction-strengthening, (Hajiabdolmajid, 2001) may be more appropriate and would have been interesting to investigate in this research had time permitted.

• 3D linear elastic modelling and calibration to observed brittle damage limits, although not able to model true displacement or potential increases in stress due to stress shedding from failed regions, still acts as one of the best, and most easily implemented tools, in comparison to non-linear modeling. Caution should however be used in applying Mogi’s brittle-ductile behavioural classification (Mogi, 1966) to calibration. The Williams mine study, (Chapter 3), and the complementary study at the Golden Giant mine, (Chapter 4), suggests that the rock mass, which has inherent discontinuities, should not be compared to or classified by Mogi’s transition zone for crystalline intact rock as suggested by Falmagne (2002) and Diederichs (2000). More research into the influence of discontinuities on the effect the transitional behaviour from brittle to ductile should be performed to confirm Hoek-Brown’s (1980) finding’s, where a lower limit of $\sigma_1/\sigma_3 = 2$ over 3.4, may potentially be more applicable to the rock mass or disturbed laboratory samples. Use of polyaxial testing with the measurement of acoustic emissions (Young, 2007) should help to advance this research.
• An alternative interpretation is that the failure initiated under ductile conditions, and is seen as eventual strength weakening due to a change in the strength state from ductile to brittle. This is not a change in the material behaviour but a change in the strength state due to geometrical influences. This occurs through a change in the geometry of the pillars (gradual failure at the excavation boundaries) which results in a reduction in the strength made more significant at the point of disassociation by creation of conjugate shearing propagating to a free surface (Kaiser pers. comm., 2009). The effect of the change in the geometry and a reduction in the strength state should be investigated further.

6.5.6 Overall Monitoring Recommendations

Finally, in order to better understand the role of dilation of these confined failures and to develop an improved constitutive model, continuous measurement of displacements and stress throughout the failure domain, not just at the periphery of the zone, with the combined monitoring of microseismicity is required.

• In this research, although field displacements were monitored, the instruments were not ideally oriented to monitor the potential dilations that are anticipated at the core of the seismic clusters. It is essential, as indicated from the non-linear modelling, in order to model the post peak behaviour of this strain softening and dilating rock mass, that actual displacements and dilations are required to calibrate the models. In order to evaluate the deformations in situ, measurement of displacements using either SMART Cables or MPBX instruments should be monitored continuously and automatically. Also, complimentary to these instruments, time-domain reflectometry (TDR) cables, (Coulson and Plummer, 2000), would aid in identifying the location of dilations between the MPBX nodes. Both of these instruments should be fanned through a potential failure region. In the petroleum industry the use of surface located micro sensitive tiltmeters has shown promise in measuring volume change of oil reservoirs during extraction (Maxwell et. al., 2008). Incorporation of these tiltmeters in short boreholes close by an anticipated failure, may add value in understanding the regional volume changes that occur. Another option, to ascertain regional volume changes, would be to resurvey excavations that are above, below, and in regions that are failing.
• Complimentary to deformation monitoring would be to perform diamond drilling to obtain core from one of these stress fractured regions which could also be used to observe the microfracture structure of these 'failed' regions. Borehole logging of these holes using geophysical logging to obtain a velocity profile and material changes, as well as a visual identification of fractures using a borehole camera, would also significantly aid in confirming the fracture network, and observation of dilated fractures if they exist.

• Conducting velocity surveys could additionally enhance the observational potential of the development of failed or dilated rock mass.

• Also missing from this study is measurement of the exact drop in stress following this strain softening behaviour and this needs to be studied and not inferred. Physical monitoring of stress in high stress environments is not, however, a trivial matter, (Kaiser et. al., 2000). It has been noted in a number of studies based on destressing of regions of the rock mass through mining, (Simser and Falmagne, 2002; McMullan et. al, 2004), that vibrating wire stress cells have inadequate response to relaxation and are better suited to more robustly measure stress increase. One of the best stress measuring devices for determining the full stress tensor change are CSIRO hollow inclusion cells, (Worotniki and Walton, 1976), which will monitor relaxation relatively well. Large borehole deformations under high stress, however, can cause these sensitive strain measurement devices to fail, generally through cracking of the epoxy bond to the rock. Thus, measuring stress, which is essential to accurately measure the effect of strain softening or strength weakening, needs to be researched more thoroughly.

Only through measuring the physical reaction of the rock mass through the stages of failure into the post peak, will the proposed behaviour of strain softening following localization be confirmed, and can only be implied here based on the evidence presented.
A Philosophical thought!

“As we progress up the stair case of discovery we make little steps, these small steps progress us forward and upward. There is often a tendency in haste to jump a few steps at a time, however, we must be vigilant as this is often what causes us to trip and fall.”

Adam Coulson (2009)
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APPENDIX A

A1  Seismic Event Density Parameter Evaluation
A2  Principal Component Analysis Parameter Evaluation
A3  Source Parameter Calculation and Equations
APPENDIX A1

Seismic Event Density Parameter Evaluation
A1 SEISMIC EVENT DENSITY PARAMETER EVALUATION

A1.1 Overview of The Clustering Index

In order to use the seismic density evaluation effectively to determine the potential start of fracture interaction, (the point of true yield), called here the clustering density, analysis of the clustering index has been used to determine a probable event density in which events/fracture may be interacting. Falmagne (2002) developed a modified clustering index function, (Clf), (Figure A1.1a and b), based on work by Lockner et. al. (1992), to try to quantify the point of interaction and coalescence ($\sigma_{cd}$) of events, thus identifying the point of yield of the rock mass in analogy to failure of laboratory samples. The clustering index function (Clf), is based on the inter event distance and degree of interaction dependent on the source size (inelastic deformation volume) and is used to determine the cluster index, (ClI), this being the effect of all historical neighbouring events (Figure A1.1c). Note, Falmagne (2002), inverted the relationship of Lockner et. al., (1992), to get around the problem of numerical instability resulting from events occurring at the exact same location, as the authors event data had locations recorded only to the nearest metre, and thus coincidence of events was a common occurrence. In the absence of source radii, (normally determined from triaxial data), Falmagne (2002), fitted an estimated source radius based on the number of sensors hit for an event, from comparison at another mining operation with actual source radii data. Even with this very crude scaled source size, the cumulative clustering index (Falmagne, 2002) was used to spatially identify regions where the rock mass was yielding, and for the site in the case study, the amount of overbreak, (caved rock), that occurred roughly correlated to the intensity of the cumulative cluster index when a comparison of both was made to modelled 3D linear elastic stress. Another indicator called the degradation index, the ratio of the cumulative apparent volume (after Mendeki, 1994) to the cumulative cluster index, (this relation being similar to Mendeki’s, (1994), seismic damage parameter), was analysed on a temporal basis with the cumulative cluster index. It was noted (Falmagne, 2002) that the damage index could be used to identify cases when there was an increase in microseismicity for a given volume without clustering (prior to coalescence) and periods when clustering and coalescence becomes intense and the rock mass degradation was at its highest, and more likely to result in caving, large seismic events or yield and stress redistribution. One major issue with carrying out cumulative analyses is that it is reliant on having a full history of the activity and generally any changes in seismic event rate will lead to changes in the parameters. The other limitation with this study was that is was found that the magnitude of the parameters and the point of change, was site specific due to the definition of...
the volumes of observation and thus calibration to a particular site or some form of normalisation, are required (Falmagne pers. com., 2005).

Reyes-Montes (2004) used a technique of relative location based on the method of Gibowicz and Kijko (1994), to improve the location accuracy of events before analysing the cluster index. The method uses a master event at the centre of mass of a cluster of events and then relocates all the other events in the cluster based on the velocity structure of the master. This can remove location inaccuracies due to an incorrect velocity model or the presence of excavated voids (Reyes-Montes, 2004) and can improve the definition of linear planar structures that fit geological and source mechanism observations (Saccorotti et al., 2002). Reyes-Montes (2004) applied this method on the microseismic data generated around the tunnel sealing experiment (TSX) at the Underground Research Laboratory (URL), to improve the definition for the analysis of the clustering index.

The clustering index is used here to assess the potential for events to be interacting based on the seismic event density and estimated source radius. This is used to estimate the clustering density, being an estimate of the point of interaction and yield of the rock mass (Section A1.3). Although the use of relative location may have greatly aided in improving the overall relative location accuracy, because of the complexity and time involved in processing the data, this avenue was not followed.

**A1.2 Seismic Event Density and Effect of Voxel Size**

During this study a great amount of microseismicity was recorded at both the Williams mine, (Chapter 3), and Golden Giant Mines (Chapter 4). An example used here is of some 33,000 events recorded within the Williams mine sill pillar region (Figure A1.2). In order to get a more defined picture of where and to what intensity the microseismicity was occurring, a cumulative seismic event density algorithm was developed to visualize the most intense regions. This was achieved by counting the number of events that occur within a specific voxel (cubic) volume, evenly distributed over the domain. The algorithm developed utilises a coordinate mapping function to quickly correlate the event locations to the elemental cells, or voxels and increments the cell count for events that occur within that cell. This enables a large quantity of data to be analysed quickly. Choice of voxel size can have a large impact on the resolution of the seismic intensity. The effect of voxel size was investigated by increasing the cubic dimension from 2.5 m (15.625 m³), to 5 m (125 m³), to 10 m (1000 m³) and to 25 m (15626 m³), for recorded microseismicity in the sill pillar region at the Williams Mine collected from Sept 1999 to Feb 2005 (Chapter 3). A comparison on a single north-south grid plane cutting perpendicular to the
orebody for the most intense region can be seen in Figure A1.3 to Figure A1.6, where the voxels are coloured based on the event count to produce a pixelation plot, or are contoured using a linear regression algorithm in Examine3D®. As can be noted, the best resolution appears to come from a voxel dimension of 5 x 5 x 5 m. At half this dimension the distribution becomes more scattered and intense regions less pronounced, as the voxel dimension is increased to 10 m, smoothing of the data occurs, as more data off the plane is included in each voxel, while at a large voxel size of 25 m all resolution has been lost. From a practical standpoint a voxel size of 5 x 5 x 5 m, is often what is used as a Selective Mining Unit (SMU) for block modelling, when Kriging of grade data is performed for open stope mining, and physically can be visualized as the dimensions of an common excavated drift in section. This voxel size of 5 x 5 x 5 m (125 m³) is also quite useful, if the source radii are considered, to determine potential fracture interaction, based on an estimate of the clustering density.

A1.3 Effect of Mean Source Radius to determine the Clustering Density

There has been much speculation over the accuracy of source radii that have been recorded in the mining environment. Three models for the determination of the source radius or extent of the rupture surface are commonly applied in mine seismology. The Brune (1970) and Madariaga (1976) empirical 2D circular failure models, taken from earthquake seismology, are both based on the fall off of the high frequency spectra and the assumption that the rupture occurs through a pure-shear slip failure mode. The later model takes into account the fact that the spectral decay is dependent on the angle of observation to the rupture plane. The third model the, apparent radius, is based on an estimation of the volume of coseismic inelastic deformation called the apparent volume (Mendecki, 1994), which is assumed to be spherical in nature. A fourth model, the tensile source model, has been proposed by Cai et al. (1998) for low confinement regions close to openings based on the assumption that the primary mode of failure is tensile (Mode I) not pure-shear (Mode II). This is taken from observations of low S-wave to P-wave energy ratios (E_s/E_p) less than 10, for microseismic events close to openings, suggesting a non-pure shear failure mode (Urbancic and Trifu, 2000). This model is based on Griffith crack theory (Griffith, 1924) and the balance of potential and rupture surface energies, with the assumption that the unbalanced energy corresponds to the attenuated recorded seismic energy. One interesting aspect of this model, that is not included in the others, is that the crack length (source diameter) is influenced through the normal stress on the penny shaped rupture plane.

A comparison of these four models was made at the URL by Cai et al. (1998), Falmagne (2002), and Reyes-Montes (2004), indicating that the Madariaga, apparent radius and Brune
models result in relatively equivalent source radii ranging from 0.4 to ~1.5 m; the latter two models tend to have a mean source radius 100% larger than the Madariaga model. In contrast, the tensile source model results in source radii of the order 0.04 to 0.15 m, approximately a tenth of the rupture radius found for the other models. One issue with this model is that it has been noted that the radiated seismic energy is only a small proportion of the energy consumed during the rupture process (Kasahara, 1981), and this energy balance does not fully account of all of the energy losses, potentially resulting in small radii (Cai, et al., 1998). However, from visual observations of boreholes and slot excavation at the URL Mine-by tunnel (Martin et al., 1997) and TSX (Chandler et al., 2002) it was found that this rupture or fracture length agrees with observations (Cai et al., 1998; Falmagne, 2002; Reyes-Montes, 2004). Contrary to this Collins and Young (1999) found that the Madariaga model was more suited at the Mine-by tunnel, giving source radii of 0.13 to 0.51 m, which also fit with field observations.

Due in part to the difficulty in obtaining the normal stress on the rupture plane, which can only be determined through numerical modelling, Falmagne (2002) and Reyes-Montes (2004) suggested that the tensile model is most easily approximated through determination of the apparent radius and applying a 0.1 multiplier (90% reduction in dimension) to obtain the tensile source radius.

However, this model is probably only most applicable to microseismic events very close to excavation surfaces, and Gibowicz et al., (1991), and Urbancic (1991), have noted that it is not uncommon in the mining environment to have low E_s/E_p ratio’s < 10 with a significant tensile component but still found that the Madariaga model for source radii was in good agreement with independent underground observations and suggest a combined shear-tensile failure mode. Pure shear failures are expected at E_s/E_p > 20 (Boatwright and Fletcher, 1984; cited Urbancic, 1991).

As mentioned in Chapter 2 at the Williams mine there were little to no confident source parameters that could be determined (Chapter 3), however, at the adjacent Golden Giant mine (Chapter 4) online event source radii are determined using the Madariaga model (ESG, 2006). The source radii determined for a year of data at Golden Giant Mine around the shaft pillar area (~19,000 events) were found to be normally distributed, with mean source radius 1.85 m and ranged from 0.8 to 2.5 m with a standard deviation of 0.3 m (Figure A1.7a). It has been assumed that as rock mass is the same and under similar stress conditions using the same type of microseismic system that this average source radius is applicable for the Williams study. This source size seems reasonable for rupture away from openings and is similar in
range for the source radii assumed by Falmagne (2002) for the study at Lac Short, in which the author reverted back to a source radius based on the Madariaga Model. This source size does not seem unreasonable from observations made at the Hemlo camp. Additionally, through investigation of mining induced seismicity in Polish mines and Canada, Gibowicz (Gibowicz et al., 1991), has found that Madariaga’s model provides reasonable agreement with independent visual observations underground.

As stated earlier, based on work by Falmagne (2002) and Lockner (1993), the clustering index was developed to determine the point of fracture interaction leading to coalescence (σ_{cd}) and is an indication of true yield. For microseismic data if the source radii of adjacent events interact, then clustering or coalescence can be considered, and Falmagne (2002) proposed that events are coalescing if the Cluster Index (CIi) is greater than 0.5. A rough approximation to determine if events are coalescing and onset of yield occurring would be if at least 5 events of the mean Madariaga source size, are contained within the 125 m³ voxel (Figure A1.8a). Analysis of a worse possible case event distribution in a voxel, (four events at the corners and one at the centre all at the mean source radius), indicates the Cluster Index, CIi (Falmagne, 2002; Figure A1.1c), for the centre event would be CIi ~3.0 (Figure A1.8b). Even though the events are not touching (i.e. a pair of events have a CIf < 0.5) they are close enough to be potentially interacting. The corner events would have a CIi ~ 1.25 (Figure A1.8b), however, if we consider the events to be sized one standard deviation less than the mean, the centre event CIi would decrease to 1.5, and the corner events to 0.5.

If we now consider only four events (i.e. loss of the centre event), then for this configuration and source size, these corner events would have a CIi < 0.5 and thus would not be considered to be clustering (Figure A1.8b). The CIi of these event becomes greater than 0.5 when source radii of 1.7 m are obtained. Thus, based on this simplified analysis considering what happens within only one voxel of 125 m³ volume, it is possible to state that if 5 events with source radii distributed about the mean (1.85 m) occur within the voxel (125 m³), these events will probably be interacting and having a Cluster Index, CIi >0.5. The effect of this can be visualised in Figure A1.8c for the specified contour limits and Figure A1.8d, showing a potential rock mass yield region contained within the iso-surface contour of 5 events per voxel, this being the clustering density. It should be pointed out that this is a rough estimate, as the actual source rupture envelope in the Madariaga model is considered to be a 2D circular surface, however, by applying this limiting condition a reasonable approximation to the potential 'yield' front is achieved.
A comparison of the Madariaga source size has been made to the other source models, (the apparent radius, and the tensile source model) and is shown back in Figure A1.7 (Figure A1.1b,c, & d). The apparent radius, calculated from the online source parameters for the events from the Golden Giant Mine, indicate that this source radius is 100% larger than the Madariaga model, ranging from 1.5 m to 6.0 m with a mean of 4 m and standard deviation of 0.8 m. By applying a 90% reduction or 10% multiplier to reduce the apparent radius to be equivalent to the tensile source model (Falmagne, 2002), the resultant mean source radius is of course 0.4 m (Figure A1.7d). A multiplier of 22% (0.22) is required to achieve similar tensile source radii from the Madariaga model. In order to carry out the previous analysis to define a clustering density, the voxel size would have to be reduced to 1 x 1 x 1 m, if again the limiting condition of 5 events per cell is to be applied. The voxel density has been plotted in Figure A1.9 for a similar section as in Figure A1.3 to Figure A1.6, and has been contoured. As can be noted, most of the voxels are sparsely populated and have one event per cell and at most 2 events. This, although it does not rule out the possibility that pairs of events are interacting, based on this source size, only limited regions at the densest core of the seismic cloud would potentially be in yield. This does not fit with the findings of this research (Chapter 3 and 4) and thus suggests that the source size away from openings has to be considerably larger than the tensile source model would suggest for interaction, coalescence and yield to occur. Thus, the Madariaga model is felt to be more representative of the approximate rupture or fracture dimension.
Figure A1.1 Definition of the clustering index, (a) definition of the clustering index function (Clf), and (b) and classification of cracks according to their interaction based on their inter-distance (d_ij) (after Falmagne 2002; Reyes-Montes 2004), and (c) use and definition of the clustering index at the URL, comparison of clustering events (Cl_i ≥ 0.5) to surveyed notch outlines using the apparent radius model (after Falmagne, 2002).
Figure A1.2 Microseismicity recorded in the Williams mine sill pillar region analysed in this study from September 1999 to February 2005. The sill region analysed here is bounded from stope 28 to stope 16, and analysis slices 1 to 6 are 40 m wide spanning two stopes. Approximately 33,000 events were recorded and located during this time frame. Also, shown is longitudinal grid plane at section 9860N (b), used for seismic density plotting in section 4.1.
Figure A1.3  WM Sept 1999 - Feb 2005 – Seismic Density Section 9433.75E, Voxel Size = 15.6 m$^3$ (2.5x2.5x2.5 m) (a) location of grid plane, (b) pixelation plot of events density and (c) contoured events on the grid plane.

Figure A1.4  WM Sept 1999 - Feb 2005 – Seismic Density Section 9432.5E, Voxel Size = 125 m$^3$ (5x5x5 m) (a) location of grid plane, (b) pixelation plot of events density and (c) contoured events on the grid plane.
Figure A1.5  WM Sept 1999 - Feb 2005 – Seismic Density Section 9430.0E, Voxel Size = 1000 m$^3$ (10x10x10 m) (a) location of grid plane, (b) pixelation plot of events density and (c) contoured events on the grid plane.

Figure A1.6  WM Sept 1999 - Feb 2005 – Seismic Density Section 9437.5E, Voxel Size = 15625 m$^3$ (25x25x25 m) (a) location of grid plane, (b) pixelation plot of events density and (c) contoured events on the grid plane.
Figure A1.7  Comparison of source radius based on (a) Madariaga and (c) Apparent radius to (b)&(d) Tensile Radius. Data from adjacent Golden Giant Mine (Coulson, 2008b).
Figure A1.8  (a) Representation of worst case event location in a 125 m$^3$ voxel. (b) Calculation of clustering index (CIi) for a 5 x 5 x 5 m cube (125 m$^3$) with 5 events (blue line CIi for centre event and red line CIi for corner events), and 4 events only (green line CIi for corner events). (c) Contour of event density for 125 m$^3$ voxels with the lower contour limit set to 5 events and (d) 5 event per voxel iso-surface of all events recorded in the sill region.
Figure A1.9  WM Sept 1999 - Feb 2005 – Seismic Density Section 9432.5E, Voxel Size = 1 m³ (1x1x1 m) (a) location of grid plane, (b) pixelation plot of events density and (c) contoured events on the grid plane.
APPENDIX A2

Principal Component Analysis Parameter Evaluation
A2 PRINCIPAL COMPONENT ANALYSIS PARAMETER EVALUATION

As discussed in Chapter 2 the principal component analysis (PCA), is a statistical technique applied to determine the spatial trends in a cluster of seismic events. The main assumption of the technique is that events are inter-related to one another through the stress regime and geology, or in this case, the fracture generation zone (Chapter 3, 4 and 5). The trends being based on, inter-event or inter-fracture communication, it is necessary to treat the data in a time sequential manner. The details of the calculation are laid out in Chapter 2, and are summarised in Figure A2.10. In order to effectively use this method, a number of key parameters need to be investigated these are: the size of the spatial window, D, the sample size or temporal window size, N', and the effect of the sliding window, the later two used to identify temporal variation in the observed trends. Changes of these parameters act on the smoothing of observed trends and temporal variations. An analysis of the effects of changing these parameters on a dataset that showed strongly defined trends and temporal variation in them has been made to obtain the optimum compromise between smoothing and sensitivity of the technique.

A2.1 Effect of Varying the Size of the Spatial Window (D)

Although previous studies have suggested that a spatial window dimension, D value that captures the majority of the events should be used (Posodas et al., 1993; Urbancic et al., 1993; Coulson, 1996) the effect of varying this spatial value was investigated to verify the consistency of the results. The objective is to find a resolution that enables varying trends to be observed within clusters, while limiting the amount of scatter of the trends. This is similar to determining an appropriate resolution for observing the seismic event density. In the example shown here a cluster of 1333 events from the Williams mine (9390 Level Footwall - 2003) was analysed (Figure A2.11). This cluster showed a marked temporal change in the spatial distribution of events, the significance of these changing trends is discussed in Chapter 3. The dimension of this cluster is contained within a region, 40 x 35 x 35 m with the maximum inclined dimension of 45 to 49 m. From the event inter-distance distribution the PCA technique was applied to the cluster using varying values of D based on 10%, 25%, 50%, 75% and 100% of the CDF, corresponding to D values of 5, 10, 15, 20 and 50 m (Figure A2.10a). These have been rounded to the nearest integer for simplicity. Temporal windows where kept constant at 50 events, using a full 50 event shift of the temporal window and will be discussed in the next
section. All of the PCA derived planes with ellipticity > 2.5 where plotted on lower hemisphere projections using Dips© (Rocscience, 2005) for the varying D values (Figure A2.12).

As can be seen from Figure A2.12, when the D value is small, below 20 m (75% of the CDF), then there is a great amount of scatter in the poles, (eigenvectors), and major pole densities are not stable. These lower D values are close to the location accuracy of the microseismic system (±7-10 m) with an average location accuracy for this cluster being ± 5 m (Chapter 3), and also the small size of the spatial window means that only a small proportion of the events in any given temporal window are used for the analysis, i.e. the F–parameter < 50% (Figure A2.13c). At a D value of 20 m, three distinct population groups can be identified, and as the D value is increased to 50 m such that the spatial window covers 100% of the events, then the overall shape and main trend of the clustering events is increased, however, the other minor trends become less significant. Another way to determine the D value, is based on the mean inter-distance (15.9 m) plus one standard deviation (7.5 m). This again gives a similar distribution of poles to the D value based on 75% of the CDF (Figure A2.12d and e), however, the minor trends become again slightly less significant. Note also that the mean inter-distance of the cluster of events is 15.9, and the median value (corresponding D at 50% of the CDF) is very close at 15, indicating that the distribution can be approximated by a symmetric normal distribution (Weisstein, 2004).

If the PCA planes strikes, are plotted for each event versus time, using a 25 point moving average (average of the prior 25 points) to smooth the data (Figure A2.13a), the variance in the planes orientation measured by the standard deviation (n-1) also gives an indication of the stability and scatter. Note if smoothing using a moving average is not performed the resultant trace of the strike is very spiky and less easy to interpret. Through trial and error of other smoothing widths (5, 10, 25 and 50) a 25 point moving average was determined to show the best resolution without compromising identification of temporal changes. For the time period when there is a distinct change in the orientation of the average PCA planes (September to November, 2003) followed by stability for most values of D, the standard deviation of the strike using a 25 point moving window, indicates that at this time period, a D value of 20 has a lower overall value of standard deviation (Figure A2.13b). Analysis of the overall average standard deviations and average standard errors (Figure A2.13c) indicate that at this spatial dimension (20 m) there is a significant drop in these values, indicating greater stability in the determined PCA planes. Additionally, analysis of the overall average ellipticity, indicates that it is at a minima for a spatial dimension of 20. Note that in Figure A2.13a, the temporal change in
orientation of PCA planes is identified at all values of D, however, at the largest D value of 50 m, there is a change back to the overall cluster shape that is not seen at the lower values. Thus, based on these findings, and keeping the original objective in mind, a D value based on 75% of the inter-distance CDF (here 20 m), achieves sensitivity with relatively reduced scatter in results, and was chosen as the point of inflection of the average standard errors, and minima of the ellipticity.

A2.2 Effect of Varying the Size of the Temporal Window

In determining the size of the temporal window, the question arises, how many events are required to form a stable trend if a trend exists; that is, how many events (N’) should be evaluated in a temporal window? The objective again is to have enough sensitivity to observe temporal changes while minimising the error due to an inadequate sample size. Posadas et al. (1993) who analysed a series of 158 earthquakes (macroseismic events), used a sample size of 20 events based on the stabilisation of the average ellipticity for a given sample size, observed through gradually increasing the temporal window size from 3 to 54 events. However, the authors based their approach on identifying single planar fault structures. In the mining environment of this case study there is no expectation of single structures from geology, and the population of events within a cluster is in the thousands of microseismic events and not the hundreds. Coulson (1996) in analysis of microseismicity with event cluster sizes from 50 to 800 events, used a sample size (temporal window) of 100 events at a time, when the dataset permitted. In this case, however, it is important to use as small a sample size as possible in order to be able to recognise temporal changes and the point in time when they occur.

The same cluster of events previously analysed, (section A2.1), is evaluated by applying the PCA technique using a fixed spatial window D of 20 m (75% of the inter-distance CDF), and by changing the temporal window size, N’, from 10, 25, 50, 100, 250 and 500 events with equivalent sampling shift, to observe the effects on stability of the determined PCA planes and the amount of scatter to smoothing effects (Figure A2.14). Between a temporal window size of 25 to 100 events, the main three identified population groups tend to remain stable, with the amount of scatter decreasing with increasing sample size. At the lowest sample size of 10 events, the single main trend is still observed, but with significant amounts of scatter of the poles, while at a sample size greater than 100 events temporal smoothing occurs, again with loss of the minor trends.
If the individual determined event PCA planes are again plotted versus time, using a 25 point moving average (Figure A2.15a), then for the larger sample sizes of 250, and 500 events, the notable temporal variation in September 2003, is totally smoothed out, while being delayed, for the 100 event size. Review of the standard deviation of strike, using a smoothed 25 point moving window (Figure A2.15b), indicates that the standard deviations for this time period are minimal for the 50 event sample size, although lowest for the 500 event due to gross smoothing. Evaluation of the overall errors and the average ellipticity (Figure A2.15c), similar to Posadas, et. al. (1993), indicates that at a sample size of 50 events the average standard deviation, standard error and ellipticity, all start to plateaux to a more or less constant value. The F-parameter is constant for all windows at around 75%.

Thus, based on this analysis, a 50 event temporal window appears to give the most stable results with minimal scatter but still being sensitive enough to observe temporal changes in the spatial distribution of events or PCA derived planes. This sample size was also chosen as being convenient to assess the individual temporal windows (~50 poles) on lower hemisphere stereographic projections using Dips©. With much less than this number of PCA derived planes, identification of stable population groups can become difficult. Additionally, it is preferred to identify the mean trend using stereographic projection and a Fisher distribution, which has benefit when observing sparse data sets, (Rocscience, 2005). This method is much preferred for three dimensional data over straight averaging, as in Figure A2.13 and Figure A2.15, which leads to inaccurate identification of the average trend if more than one trend exists within the averaging window or there is a wide scatter of poles. However, in the analysis of using a continuous PCA (i.e. shifting the temporal window every event), discussed in the next section, a moving average was found to be the easiest way to identify more precisely the time of temporal changes.

### A2.3 Effect of Varying the Shift of the Temporal Window

Previous use of the PCA technique used a sliding temporal window with a full shift (Figure A2.10e) to analyse potential trends in the spatial distribution of microseismic events (Urbancic et al., 1993; Coulson, 1996), and was the primary method of choice in this thesis. However, it was noted during this study that, although the overall trend is observed in the individual temporal windows, often there could be swings in the trend from one sample set to another. In order to evaluate this, the degree of shift was investigated by moving the temporal window every event, to see the effect on the temporal changes. Three different sliding windows were investigated; i. perform the PCA method on an event using the next 49 events in the future and,
after one iteration to determine the PCA plane, shift the temporal window to the next event so that the sample set changes every event (continuous sampling); ii. perform PCA on an event with the sample of 25 events prior and 24 events in the future and after one iteration, shift the window to the next event, and iii. perform PCA on an event using the previous 49 events, shift to the next event. The results of this analysis compared to the 50 event sample, shift 50, base case (Figure A2.16a) indicate that there is little difference between four methods in terms of the overall plane population groups and mean trends (Figure A2.16b, c and d). Temporal analysis of the strike using a 25 point moving average, indicates that a central window (±25 events) is almost identical to the 50 event, shift 50 events method (Figure A2.16e), and has been called the continuous central moving principal component analysis (CMPCA). The other two methods follow the same temporal changes, however, these occur either slightly before or slightly after, for the forward and backward continuous 50 event samples respectively. A 50 event sample and shift of 50, plotting all the events for that sample makes it possible to easily identify the number of events that had ellipticity below the 2.5 maximum to minimum eigenvalue threshold, and were dropped, giving an indication that the cloud of microseismic events is becoming more spherical than planar in nature.

For all of the principal component analyses carried out in this thesis the size of the spatial window (D) was determined from corresponding dimension for 75% of the CDF of the inter-distance distribution, using a sample size (N’) of 50 events for the temporal window and shifting to the next sample set of 50 events. Evaluation for more accurate timing of temporal changes was however, also performed using the continuous central moving principal components analysis, (CMPCA) [i.e. PCA on an event with the sample of 25 events prior and 24 events in the future, and after one iteration, shift the window to the next event], which was showed to be most comparable to the base method.
Figure A2.10 Principal Components Analysis Technique. (a) determination of optimum spatial window size, D, based on the distribution of the cluster of events hypocentre inter-distances in Euclidian space, using the a Normal Distribution Cumulative Density Function. (b) Determination of the mean hypocentral location of the events that fall within the Spatial Window, situated on the event of interest, and (c) calculation of the spread matrix describing the variance of the hypocentres location to the mean hypocentral location, and (d) application of the principal components analysis through eigenvector and eigenvalue decomposition to produce and ellipsoid with strike and dip determined for the overall trend of the events surrounding the event of interest. (e) Application of the temporal sliding window, in the example shown, N' is 50.
Figure A2.11 Cluster of events above the 9390 L main footwall haulage drive, for PCA parameter analysis ‘Slice 2’, for all 2003 events (Coulson, 2008a).
Appendix A  
Adam Lee Coulson, Doctor of Philosophy, 2009  
337

Figure A2.12  
Effect of varying the size of the Spatial Window D, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the determined PCA planes and their orientation and distribution when plotted on a lower hemisphere stereographic projection, for D values of 5, 10, 15, 20, 23.4 and 50 m.
Figure A2.13  (a) Effect of varying the size of the Spatial Window D, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the temporal variation in strike of the determined PCA planes for each event. Note the temporal window size is fixed at 50 events, and the strike is plotted using smoothing, based on a 25 event moving window.  
(b). 25 point moving Standard Deviation during the major change in the predominant orientation of the PCA planes for varying Spatial Dimensions.  
(c) Relationship of overall (averaged) standard deviation, standard errors, ellipticity (x10) and F-parameter for the entire time period versus Spatial Window Dimension, D. At a D value of >20 m the variance starts to plateaux, and the ellipticity is at a minima.
Figure A2.14  Effect of varying the size of the Temporal Window, $N'$, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the determined PCA planes and their orientation and distribution when plotted on a lower hemisphere stereographic projection, for $N'$ values of 10, 25, 50, 100, 250 and 500 events.
Figure A2.15  Effect of varying the size of the Temporal Window, $N'$, on a cluster of events above the 9390 L Footwall drive in 2003, and result on the temporal variation in strike of the determined PCA planes for each event. Note the Spatial Window size is fixed at 20 m, and the strike is plotted using smoothing, based on a 25 event moving window. (b). 25 point moving Standard Deviation during the major change in the predominant orientation of the PCA planes for varying Temporal Window sizes. (c) Relationship of overall (averaged) standard deviation, standard errors, Ellipticity and F-parameter for the entire time period versus Temporal Window Size, $N'$. At a $N'$ value of >50 events the variance and average ellipticity starts to plateaux.
Appendix A

Adam Lee Coulson, Doctor of Philosophy, 2009

341

Figure A2.16 Effect of varying the temporal window shift (temporal window size fixed at 50 events) (a) to (d) on the stability of the PCA derived planes and (e) the temporal change in orientation of the strike of the planes with time using a 25 point moving average to smooth the data.
APPENDIX A3

Source Parameter Calculation and Equations
A3 SOURCE PARAMETER CALCULATION AND EQUATIONS.

A3.1 Automatic Calculation of Spectral Parameters (after Andrews, 1986)
The key source parameters specified in this section are found from the three fundamental spectral parameters observed in the frequency domain (Figure A3.17c): the low frequency spectral level or plateau, \( \Omega_0 \) (the basis for the seismic moment which is proportional to this and an average of the p- and S-wave contributions), the corner frequency, \( f_c \), the intercept of the lower frequency level and the slope of the high frequency spectral decay assuming a \( f^{-2} \) slope (the basis for the source radius which is inversely proportional and again an average of the P- and S-wave contributions), and the energy flux, \( J_c \), the integral of the squared ground velocity (the basis for radiated energy which is proportional to this, but the sum of the P- and S-wave contributions). In previous studies (Urbancic, 1991; Gibowicz et. al., 1991; Bird, 1993; Collins and Young, 1992) these parameters were primarily determined from the frequency domain manually. This generally results in a limited data set due to the time involved in processing. However, it has been found by Urbancic et. al., (1996), Mercer (1999) that use of the Andrews method (Andrews, 1986) incorporated into automatic calculation, provides an objective and stable methodology for determining source parameters with less than 4% difference from frequency domain calculated parameters, (i.e. removes practitioner bias and allows for fast real-time evaluation of source parameters).

Although the triaxial accelerometers employed at both the sites (Chapter 3 and 4), have a flat operating response of ± 3 dB, between 1 Hz and 5 kHz at 0.5 V/g, evaluation of the effect of low frequency cut-offs found that below a frequency of 150 Hz, low frequency noise became a contributing factor to degradation in some signals (Figure A3.17c), creating significant scatter in the evaluation of the spectral parameters from cut-offs above this. Thus a Butterworth band pass filter was applied between 250 Hz, and, 5 kHz, and as the majority of corner frequencies were found to lie between 400 and 1000 Hz, this system and filtering should be optimal. The full waveform data is sampled at a rate of 20 kHz (Nyquist rate), and a 205 ms window of all triggered sensors is stored to the database with determined arrivals, while locations and source parameters are stored to a separate database.

In the study at Golden Giant mine, as no in situ attenuation properties had been determined and because the source to triaxial sensor distances were small, in the range of 60 to 140 m, attenuation was not accounted for. From previous studies at this similar frequency range and source to sensor distance, (Gibowicz et. al., 1991, Mercer, 1999, Urbancic et. al., 1996), it has
been found that attenuation corrections are not significant versus standard errors for estimation of the spectral parameters, and a correction based on a Q value of 200 would result in a possible amplitude increases of 10 to 20%, which are not significant (Urbancic et. al., 1996).

In this research Andrews’s method employed in the time-domain, is used to determine automatically the spectral parameters, without the need to perform a fast fourier transform to the frequency domain, as depicted in Figure A3.17c. It is assumed by Andrews that the spectral decay beyond the corner frequency fits a $t^{-2}$ slope (Brune, 1970). Thus, the three key spectral parameters are:

**Low Frequency Spectral Level (Plateau):**

$$\Omega_o = \sqrt{4S_{D2}^{3/2} S_{V2}^{-1/2}}$$

**Corner Frequency:**

$$f_c = \frac{1}{2\pi} \sqrt{S_{V2} / S_{D2}}$$

**Energy Flux:**

$$J_c = S_{V2}$$

where $S_{D2}$ and $S_{V2}$ are integrals of the squared spectral displacement and velocity for both the $P$- and $S$- wave trains. From Parseval’s theorem, these integrals can be converted to the time integrals as follows:

$$S_{D2} = 2 \int_0^\infty D^2(t) \, dt,$$

$$S_{V2} = 2 \int_0^\infty V^2(t) \, dt$$

where $D^2 (t)$ and $V^2 (t)$ are the velocity and the displacement signals in the time domain, calculated by summing the squared double- and single-integrated $P$- and $S$-wave train acceleration components.
A3.2 Strength Parameters

**Moment magnitude** (Hanks and Kanamori, 1979):

\[
M = 2/3 \log_{10} M_o - 6
\]

Note: The Moment is calculated for each individual sensor and then an average of all is taken.

**Seismic Moment** (Aki, 1968):

\[
M_o = \frac{4\pi \rho c^3 R \Omega_o}{F_c}
\]

Note: the seismic moment is an average of the P- and S-wave contributions.

**Seismic Energy** (Snoke, 1987)

\[
E_c = \frac{4\pi \rho c R^2 J_c}{F_c^2} \langle F_c^2 \rangle
\]

\[
E_o = E_P + E_S
\]

Note: the radiated seismic energy is the sum of the P- and S-wave contributions.

where \( \rho \) is the density of the material (2700 kg/m\(^3\)), \( R \) is the source-receiver distance (m), \( c \) is the P- or S- wave velocity (e.g. 6096 m/s and 3500 m/s), and \( F_c \) is the radiation pattern coefficient, and \( \langle \rangle \) represents the average values. It has been assumed that the loss in energy from scattering and anelastic attenuation has been taken into account, and that the individual radiation pattern coefficients are similar to average values. Based on Boore and Boatwright, (1984) the coefficients for the P- and S- wave coefficients are 0.52 and 0.63 respectively.

**Wave Speed Relationship**:

\[
\frac{c_p}{c_s} = \sqrt{\frac{1 - \nu}{1/2 - \nu}} = \sqrt{3} \text{ for } \nu = 0.25
\]

where \( c_p \) and \( c_s \) are the P- and S-wave velocities, \( \nu \) (Poisson’s ratio) ranges from 0.2 to 0.3 for rocks, and assumes straight ray path linear elastic propagation of waves.
### A3.3 Size and Displacement Parameters

**Source Radius** (Madariaga, 1979):

\[ r_o = \frac{K_c c}{2\pi f_c} \]

Note: the source radius is an average of the P- and S-wave contributions.

where based on the Madariaga model the value of \( K_c \) is a function of the angle between the fault normal and the source-sensor direction, however, if no fault plane has been determined, the average values of the coefficients of 2.01 and 1.32, for the P- and S-wave respectively can be used with the appropriate wave velocity (Gibowicz et al., 1991; Urbancic, 1991). The Brune model is based primarily on the S-wave, using a coefficient, \( K_c \) of 2.34 (Brune, 1970).

**Apparent Volume** (Mendecki, 1993):

\[ V_a = \frac{M_o}{2\sigma_a} \]

**Apparent Radius** (Cai et al., 1998):

\[ r_a = \frac{\sqrt[3]{3V_a}}{4\pi} \]

**Asperity Radius** (McGarr, 1991):

\[ r'_a = K_s c_s \frac{v_{\text{max}}}{a_{\text{max}}} \]

where \( K_s \) is 2.34 for the Brune Model or 1.32 for the Madariaga model (used in this thesis), \( c_s \) is the S-wave velocity, \( v_{\text{max}} \) and \( a_{\text{max}} \) are the maximum recorded velocity (m/s) and acceleration (m/s²) from the root-mean-squared (rms) trace.

**Average Displacement** (Aki, 1968):

\[ u = \frac{M_o}{\mu \pi r_o^2} \]

where \( \mu \) is the shear modulus and based on the wave equation for linear elastic wave propagation can be found from \( \rho c_s^2 \) and here is equal to 33.1 GPa.
Maximum Displacement (McGarr, 1991):

\[ D = 22.9R \frac{v_{\text{max}}}{c_s} \]  
(Madariaga Model \( K_s = 1.32 \)) or \( \)  
\[ D = 8.1R \frac{v_{\text{max}}}{c_s} \]  
(Brune Model \( K_s = 2.34 \))

Note: these are based on the S-wave components only

A3.4 Stress Release Estimates and Other Parameters

Static Stress Drop (Brune, 1970):

\[ \Delta \sigma = \frac{7M_o}{16r_o^3} \]

Where \( r_o \) is based on either the Madariaga or Brune source radius (Madariaga used in this research)

Apparent Stress (Wyss and Brune, 1968):

\[ \sigma_a = \frac{\mu E_o}{M_o} \]

Dynamic Stress Drop (after McGarr, 1991):

From McGarr, (1991, eqn. [20]):

\[ \Delta \sigma_d = \frac{1}{0.0312K_s^\frac{3}{2}F_s^B} \frac{F_s^B}{F_s^M} \rho R a_{\text{max}} = \frac{1}{0.0312 \times 1.32^3 \times 0.63} \rho R a_{\text{max}} \]

such that:

\[ \Delta \sigma_d = 12.61 \rho R a_{\text{max}} \]  
(Madariaga Model \( K_s = 1.32, F_s = 0.63 \))

or

\[ \Delta \sigma_d = 2.50 \rho R a_{\text{max}} \]  
(Brune Model \( K_s = 2.34, F_s = 0.57 \))

Note: these are based on the S-wave components only

Source Complexity (after Gibowicz and Kijko, 1994):

\[ \text{Source Complexity} = \frac{\Delta \sigma_d}{\Delta \sigma} \text{or} \frac{\Delta \sigma_d}{\Delta \sigma_{\text{rms}}} \]

where Source Complexity is determined to be a simple homogenous event (or based on this research an isolated fractures) for a ratio of 1.0, but for ratios >1.0, indicates a more geometrically complex inhomogeneous source model, possibly the result of asperity failure or interacting shear/fractures, creating high energy sub-events.
**Peak Velocity Parameter** (McGarr, 1991):

\[ PVP = Rv_{\text{max}} \]

**Peak Acceleration Parameter** (McGarr, 1991):

\[ PAP = \rho Ra_{\text{max}} \]

**Seismic Efficiency** (Wyss and Brune, 1968):

\[ \varepsilon_{\text{eff}} = \frac{\mu E_o}{\sigma_{\text{rms}} M_o} \]

where \( \sigma_{\text{rms}} \) is the root mean squared (RMS) stress drop, modified from Andrews (1986) for the Madariaga model as:

\[ \sigma_{\text{rms}} = \frac{2\pi \rho R f_c}{F_s K_s} \left( \frac{S_{a4}}{S_{a2}} \right)^{1/2} \]

Note: this is based on the S-wave components only

where \( f_c \) is the corner frequency of the S-wave, \( F_s \) is the S-wave radiation coefficient 1.32, and \( K_s \) is the 1.32 for the Madariaga model and the acceleration power spectrum is used to calculate the following integrals approximated in the time-domain as:

\[ S_{a2} = \int_0^\infty A^2(t)dt, \quad S_{a4} = \int_0^\infty A^4(t)dt \]

**Attenuation Correction** (Aki and Richards, 1980):

\[ A(R) = A_o \exp \left( -\frac{\pi R f_c}{cQ} \right) \]

where \( R \) is the source-sensor distance (m), \( f_c \) is the corner frequency (P- or S-), \( c \) is the wave speed (P- or S-), \( Q \) is the Q-value or quality factor for the given site found to range from 20 to 1000 (Spottswoode, 1993) for underground mines (typical value of 200 is often used) and can be found from the event location and the deviation in the displacement spectrum slope from –2 beyond the corner frequency, \( A_o \) is the signal amplitude at the source and \( A(R) \) is the observed signal amplitude at the sensor.

**Note:** In this study due to the small distances involved no attenuation correction was made to the signal amplitudes.
Figure A3.17  Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_o$), corner frequency, ($f_c$) and energy flux, ($J_c$), fitted with a Brune $f^{-2}$ spectral decay.
APPENDIX B

B1 Williams Mine Sill Pillar Microseismicity

B2 Williams Mine Sill Pillar Microseismic Cluster Analysis using the Principal Components Analysis (PCA) Technique
APPENDIX B1

Williams Mine Sill Pillar Microseismicity
Figure B1.1  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from September 29, 1999 to March 31, 2000. A total of 6,651 events. Section 9860 N used for seismic density contours shown in (b).

Figure B1.2  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from April 1, 2000 to December 31, 2000. A total of 6,662 events. Section 9860 N used for seismic density contours shown in (b).
Figure B1.3 Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2001 to December 31, 2001. A total of 5,301 events. Section 9860 N used for seismic density contours shown in (b).

Figure B1.4 Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2002 to December 31, 2002. A total of 3,876 events. Section 9860 N used for seismic density contours shown in (b).
Figure B1.5  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2003 to December 31, 2003. A total of 8,721 events. Section 9860 N used for seismic density contours shown in (b).

Figure B1.6  Williams mine sill pillar microseismicity, for block 3 Central (B3-C) from January 1, 2004 to December 31, 2004. A total of 1,830 events. Section 9860 N used for seismic density contours shown in (b).
APPENDIX B1 (CONTINUED)

Williams Mine Sill Pillar Microseismicity Plotted
Per Slice and Year
Figure B1.7  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 1 Events (1587). Note mining geometry is fixed at Dec 1999.

Figure B1.8  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 2 Events (2633). Note mining geometry is fixed at Dec 1999.
Figure B1.9  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 3 Events (2757). Note mining geometry is fixed at Dec 1999.

Figure B1.10  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 4 Events (2485). Note mining geometry is fixed at Dec 1999.
Figure B1.11  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 5 Events (2815). Note mining geometry is fixed at Dec 1999.

Figure B1.12  Williams Mine Microseismicity Sept 1999 - Dec 2000 – Slice 6 Events (899). Note mining geometry is fixed at Dec 1999.
Figure B1.13  Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 1 Events (954). Note mining geometry is fixed at Dec 1999.

Figure B1.14  Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 2 Events (2038). Note mining geometry is fixed at Dec 1999.
Figure B1.15  Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 3 Events (1254) . Note mining geometry is fixed at Dec 1999.

Figure B1.16  Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 4 Events (554) . Note mining geometry is fixed at Dec 1999.
Figure B1.17  Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 5 Events (383). Note mining geometry is fixed at Dec 1999.

Figure B1.18  Williams Mine Microseismicity Jan 2001- Dec 2001 – Slice 6 Events (87). Note mining geometry is fixed at Dec 1999.
Appendix B  Adam Lee Coulson, Doctor of Philosophy, 2009  362

Figure B1.19  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 1 Events (586). Note mining geometry is fixed at Dec 1999.

Figure B1.20  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 2 Events (1775). Note mining geometry is fixed at Dec 1999.
Figure B1.21  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 3 Events (990) . Note mining geometry is fixed at Dec 1999.

Figure B1.22  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 4 Events (266) . Note mining geometry is fixed at Dec 1999.
Figure B1.23  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 5 Events (130). Note mining geometry is fixed at Dec 1999.

Figure B1.24  Williams Mine Microseismicity Jan 2002- Dec 2002 – Slice 6 Events (110). Note mining geometry is fixed at Dec 1999.
Figure B1.25  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 1 Events (733). Note mining geometry is fixed at Dec 1999.

Figure B1.26  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 2 Events (3719). Note mining geometry is fixed at Dec 1999.
Figure B1.27  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 3 Events (3356) . Note mining geometry is fixed at Dec 1999.

Figure B1.28  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 4 Events (577) . Note mining geometry is fixed at Dec 1999.
Figure B1.29  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 5 Events (189). Note mining geometry is fixed at Dec 1999.

Figure B1.30  Williams Mine Microseismicity Jan 2003- Dec 2003 – Slice 6 Events (117). Note mining geometry is fixed at Dec 1999.
Figure B1.31  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 1 Events (227) . Note mining geometry is fixed at Dec 1999.

Figure B1.32  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 2 Events (823) . Note mining geometry is fixed at Dec 1999.
Figure B1.33  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 3 Events (553). Note mining geometry is fixed at Dec 1999.

Figure B1.34  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 4 Events (124). Note mining geometry is fixed at Dec 1999.
Figure B1.35  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 5 Events (61) . Note mining geometry is fixed at Dec 1999.

Figure B1.36  Williams Mine Microseismicity Jan 2004 - Feb 2005 – Slice 6 Events (80) . Note mining geometry is fixed at Dec 1999.
APPENDIX B2

Williams Mine Sill Pillar Microseismic Cluster Analysis using the Principal Components Analysis (PCA) Technique
Figure B2.1  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes.

Figure B2.2  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes.
Figure B2.3  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2002 to Dec 2002. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes.

Figure B2.4  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2003 to Dec 2003. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes.
Appendix B

Figure B2.5  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 1, Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes.
Figure B2.6 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes.

Figure B2.7 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2 Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes.
Figure B2.8  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Jan 2002 to Dec 2002. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 2 events. (d) PCA derived planes.

Figure B2.9  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Jan 2003 to Dec 2003. (a) Long section, (b) View west and (c) Plan view of slice 2 events. (d) PCA derived planes.
Figure B2.10  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 2, Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 1 events. (d) PCA derived planes.
Figure B2.11 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes.

Figure B2.12 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2001 to Dec 2001. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes.
Figure B2.13  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2002 to Dec 2002. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes.

Figure B2.14  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2003 to Dec 2003. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes.
Figure B2.15 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 3 Jan 2004 to Dec 2004. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 3 events. (d) PCA derived planes.
Figure B2.16  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes.

Figure B2.17  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2001 to Dec 2001 (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes.
Figure B2.18  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2002 to Dec 2002 (a) Long section, (b) View west and (c) Plan view of slice 4 events. (d) PCA derived planes.

Figure B2.19  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2003 to Dec 2003 (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes.
Figure B2.20 NO EVENTS - Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 4 Jan 2004 to Dec 2004 (a) Long section, (b) View west with SMART location and (c) Plan view of slice 4 events. (d) PCA derived planes.
Figure B2.21  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 5, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 5 events. (d) PCA derived planes.

Figure B2.22  Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 5 Jan 2001 to Dec 2001 (a) Long section, (b) View west with SMART location and (c) Plan view of slice 5 events. (d) PCA derived planes.
Figure B2.23 Stereonet of (PCA) derived planes for yearly clustering events above the 9390L FW haulage drive in slice 6, Sept 1999 to Dec 2000. (a) Long section, (b) View west with SMART location and (c) Plan view of slice 6 events. (d) PCA derived planes.
APPENDIX B2 (CONTINUED)

Williams Mine Sill Pillar Microseismic Cluster Analysis using the Principal Components Analysis (PCA) Technique

Slice 2 9390 Level FW events
50 Event Sub-Set Windows
Figure B2.24 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 0; 12/26/1999, (b) Temporal Window = 1; 02/05/2000 (c) Temporal Window = 2; 04/25/2000 (d) Temporal Window = 3; 06/09/2000. Note date is based on last event in temporal window.
Figure B2.25 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 4; 07/30/2000, (b) Temporal Window = 5; 08/08/2000 (c) Temporal Window = 6; 09/06/2000 (d) Temporal Window = 7; 12/17/2000. Note date is based on last event in temporal window.
Figure B2.26 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 8; 03/09/2001, (b) Temporal Window = 9; 04/28/2001 (c) Temporal Window = 10; 06/12/2001 (d) Temporal Window = 11; 06/28/2001. Note date is based on last event in temporal window.
Figure B2.27 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 12; 06/29/2001 Note jump in instrument 24_1 after 3.1 Mn event and slight change in seismicity, (b) Temporal Window = 13; 06/29/2001 (c) Temporal Window = 14; 07/01/2001 (d) Temporal Window = 15; 07/06/2001. Note date is based on last event in temporal window.
Figure B2.28 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 16; 07/15/2001, (b) Temporal Window = 17; 08/06/2001 (c) Temporal Window = 18; 09/03/2001 (d) Temporal Window = 19; 10/03/2001. Note date is based on last event in temporal window.
Figure B2.29  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 20; 11/30/2001 Note jump in instrument 24_1 at 4.5 m & change in seismicity, (b) Temporal Window = 21 12/30/2001 (c) Temporal Window = 22; 01/15/2002 (d) Temporal Window = 23; 01/23/2002. Note date is based on last event in temporal window.
Figure B2.30 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 24; 02/18/2002, (b) Temporal Window = 25; 04/09/2002 (c) Temporal Window = 26; 05/04/2002 (d) Temporal Window = 27; 05/11/2002. Note date is based on last event in temporal window.
Figure B2.31 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 28; 06/26/2002, (b) Temporal Window = 29; 08/24/2002 (c) Temporal Window = 30; 10/01/2002 (d) Temporal Window = 31; 11/21/2002. Note date is based on last event in temporal window.
Figure B2.32 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 32; 12/26/2002, (b) Temporal Window = 33; 12/31/2002 Note jump in instrument 26_1 of 55 mm at 4.5m depth also instrument 24_1 and 24_2 fail at depth on the other side, and cause a drop in the number of planes with ellipticity for the next two windows (c) Temporal Window = 34; 01/12/2003 (d) Temporal Window = 35; 02/06/2003. Note the events plotted in these last two figures are all the events in the temporal window Ellip> 1.0.
Figure B2.33  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 36; 03/12/2003, (b) Temporal Window = 37; 04/09/2003 (c) Temporal Window = 38; 04/30/2003 (d) Temporal Window = 39; 05/11/2003,
Figure B2.34 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 40; 05/14/2003, (b) Temporal Window = 41; 05/27/2003 (c) Temporal Window = 42; 05/30/2003 (d) Temporal Window = 43; 05/30/2003. Note date is based on last event in temporal window.
Figure B2.35 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 44; 05/31/2003, (b) Temporal Window = 45; 06/02/2003 (c) Temporal Window = 46; 06/10/2003 (d) Temporal Window = 47; 06/20/2003. Note date is based on last event in temporal window.
Figure B2.36 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 48; 07/07/2003, Note jump in instrument 26_2 of 16 mm at 9 m depth and point of disassociation for this cluster (b) Temporal Window = 49; 07/25/2003 (c) Temporal Window = 50; 08/28/2003 (d) Temporal Window = 51; 08/28/2003. Note date is based on last event in temporal window.
At the end of August over 50% of the events did not have an ellipse $>2.5$, this occurs for the windows 53 and 54 and indicates some change in the seismicity in the region. Note: this is around the same time as haulage is Paste Filled from 26 to 23-stope.

Figure B2.37 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 52; 08/28/2003, (b) Temporal Window = 53; 08/29/2003, Note drop in number of planes with ellipticity for this and the next window, same time as haulage is paste filled from stope 26 to 23 (c) Temporal Window = 54; 08/29/2003 (d) Temporal Window = 55; 08/30/2003. Note date is based on last event in temporal window.
Figure B2.38 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 56; 09/03/2003, (b) Temporal Window = 57; 09/16/2003, Note on the Sept 13 the 3.5 Mn event occurred and cause a permanent change in the seismicity as the back above the 9390 became aseismic, this could have been the failure of the back onto paste? (c) Temporal Window = 58; 10/02/2003 (d) Temporal Window = 59; 11/02/2003.
Figure B2.39 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 60; 12/31/2003, (b) Temporal Window = 61; 03/05/2004 (c) Temporal Window = 62; 04/15/2004 (d) Temporal Window = 63; 07/18/2004. Note date is based on last event in temporal window.
Figure B2.40 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 9390L FW haulage drive in slice 2, including location of SMART cables. (a) Temporal Window = 64; 01/31/2005, END OF ANALYSIS. Events with ellipses > 2.5 only => (b) Temporal Window = 35; 02/06/2003, compare event loc. to Fig. B2.32(d); (c) Temporal Window = 53; 08/29/2003, compare to Fig. B2.37(b); (d) Temporal Window = 54; 08/29/2003, compare to Fig. B2.37(c).
APPENDIX C

Williams Mine Sill Pillar SMART Cable and SMART MPBX
Response on the 9390 Level Haulage Drive
Cross cut intersections #16 to #27
Figure C1. a. Plan view of 9390 L sill pillar region, showing boundaries of seismic analysis slices and location of conventional instrumentation (SMART cables). b. Longitudinal view of conventional instrumentation and contours of depth of displacement that exceed 1 mm dilation, for three points in time, 12-2000, 12-2002 and 07-2003. Note instruments 17, 18, 19 and 20 showed negligible movement during the monitoring period. Also shown on (b.) depth of caving on 9415L as of 12-1999 following 2.6 Mn event (Yi, 1999)
Figure C2  Instrument 27 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
a. **Displacement vs Date/Time** - Smart Cable - 26x/c Intersection - 9390 Level

**Figure C3**  Instrument 26-1 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
**Figure C4**  Instrument 26-2 SMART cable response (a) nodal displacement in mm relative to the Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yyyy format.
Figure C5  Instrument 25 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C6  Instrument 24-1 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C7  Instrument 24-2 SMART cable response (a) nodal displacement in mm relative to the Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C8  Instrument 23 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C9 Instrument 22 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C10 Instrument 21-1 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C11 Instrument 21-2 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C12 Instrument 20 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C13  Instrument 19 SMART cable response (a) nodal displacement in mm relative to the head versus time. Note node 1 is at the collar, and the head is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C14  Instrument 18 SMART MPBX response (a) nodal displacement in mm relative to Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note : labelled dates are in mm-dd-yy format.
<table>
<thead>
<tr>
<th>Date/Time</th>
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</tr>
<tr>
<td>29-Apr-03</td>
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</tr>
</tbody>
</table>

**Figure C15**  Instrument 17 SMART MPBX response (a) nodal displacement in mm relative to Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
Figure C16 Instrument 16 SMART MPBX response (a) nodal displacement in mm relative to Node 1 versus time. Note the head is at the collar, and node 1 is at the toe of the hole, (b) displacement in mm versus depth along cable. Note: labelled dates are in mm-dd-yy format.
APPENDIX D

D1  Automatic versus Manual Picking.
D2  Effect of Individual Triaxial Sensors on the Average Calculation.
D1 AUTOMATIC VERSUS MANUAL PICKING.

D1.1 Automatic Picking and Location
In the automatic location algorithm, P-wave arrivals are determined from sampling of the squared signal to noise amplitude (here S/N=2.5), in the root mean squared (rms) seismograph. A simplex algorithm, employing the L2 norm is used to locate events (ESG, 2006) for a first pass, (P-wave velocity determined at the site 6096 m/s), then P-wave arrivals are re-picked over a smaller window with a tighter signal threshold, (min. P-wave voltage = 0.001V), rejecting sensors with large residual values and again relocating the events with the simplex algorithm. From the triaxial sensors, the P- and S- windows that are used to determine the spectral parameters are estimated as follows. For the P-window this starts from the P-wave arrival to the time corresponding to 60 cycles at the high frequency cut-off (5 kHz), or just before the theoretical S-wave arrival (based on the S-wave velocity 3500 m/s), if this coincides (Figure D1.1a). The S-wave window (if no pick has been made), goes from the theoretical S-wave pick to double the window length of the P-wave window. As can be seen in Figure D1.1a, sometimes the theoretical S-wave pick can be slightly late, hence it was important to evaluate the effect of manual versus automatic picks on the source parameters.

D1.2 Comparison of Online (Automatic Picks) versus Offline (Manual Picks) on Location and Source Parameter Calculations
From the study on the focal mechanism of the events at the Golden Giant mine (Chapter 5), a representative data set of 250 events spanning the time frame from May 2002 until May 2004, had P- and S- arrivals manually picked and source parameters recalculated to compare to the automatic online results for the same time period (Chapter 4). Figure D1.2a, b, & c, shows the comparison of the effect on location of the events, indicating that in general the majority of the events locate within 3 m of the original locations, this being lower than the location error due to the array. The differences in location are evenly distributed about the 1:1 relationship, indicating no bias in the automatic picking routine and the average absolute differences were found to be 1.6, 2.0 and 1.4 m, for Northing, Easting and Depth respectively. The improvement in the online location errors due to manual picking can be seen in Figure D1.2d, where by there has been a reduction in the arrival time residuals and hence location error due to more accurate picking of the P-wave arrivals. Based on this analysis of the automatic picks versus manual picks, it is felt that the automatic picks for location purposes are within an acceptable level and can be confidently used for further analysis.
The effect of accurate picking of the P- and S-wave windows on the source parameters was investigated. Comparisons of the automatic versus manual picking, for moment magnitude (M), seismic moment (M₀), radiated seismic energy (E₀), E₀/Eₚ ratio, source radius (r₀) and apparent stress (σₐ) are shown in Figure D1.3. As can be noted most parameters have relatively little scatter between automatic and manual picking, with an even distribution about the 1:1 relationship indicating no bias. For this subset of data and representative of the population, it can be noted that the M, M₀, E₀ and σₐ all have lognormal distributions, a general characteristic of microseismic data, and primary reason for plotting on log scales, with E₀ having the largest skew. While E₀/Eₚ ratio (bar a few outliers of probably a different mechanism) and r₀ appear to be more normally distributed, over a relatively tight range. This will be discussed in greater detail in the following section.

The regression coefficients based on either linear or log linear regression (regression of the log₁₀ of the data) of the comparative analysis, indicates relatively high coefficients of r², for most of the parameters with an average of 0.87 (Figure D1.3), the exception to this is the greater scatter in the E₀/Eₚ ratio. However, over such a small interval this is not significant and still implies relatively low ratios. The average slope offsets for M, M₀, E₀, E₀/Eₚ, r₀ and Δσₐ are 3%, 5%, 10%, 13%, 2% and 6% respectively. These results suggest reliable estimates of source parameters can be obtained based on the automatic picking routine.
Figure D1.1  Typical triaxial acceleration response for a source located during localization at the Golden Giant mine. (a) Event 03/11/2003 23:35:24.34, showing clear P- and S-wave separation and with manual picks (lines) and theoretical picks based on location (arrows) for P- and S-waves. (b) Rotated waveforms P, SV, and SH components. (c) Displacement spectrum (spectral amplitude versus frequency) for P-wave, bandwidth filtered from 250 Hz to 5 kHz, showing signal and noise, and three key spectral parameters, low frequency spectral level ($\Omega_0$), corner frequency ($f_c$) and energy flux ($J_c$), fitted with a Brune $f^{-2}$ spectral decay.
Figure D1.2 Comparison of Automatic picking versus manual picking of p-wave arrivals for a representative 250 event sub-set of the data in the EOS cluster, spanning from May 2002 to May 2004. (a) Northing, (b) Easting, and (c) Depth (axis range for all is 30 m). (d) Online location error based on the simplex algorithm and source to sensor distance assuming a linear path.
Figure D1.3  Comparison Automatic picking versus manual picking of p- and s- arrivals on source parameter calculations for the same previous data sub-set. (a) Moment magnitude ($R^2 = 0.82$), (b) Seismic moment ($R^2 = 0.87$), (c) Seismic energy ($R^2 = 0.94$), (d) $E_s/E_p$ ratio ($R^2 = 0.75$), (e) Source Radius ($R^2 = 0.85$) and (f) Apparent stress ($R^2 = 0.97$)
D2 EFFECT OF INDIVIDUAL TRIAXIAL SENSORS ON THE AVERAGE CALCULATION.

Generally, for basic mine interpretation of source parameters a minimum of 4 triaxial sensors are required to produce stable solutions for a small array (Hudyma and Brummer, 2007), however, for more in depth studies of focal mechanisms using moment tensor inversion, this number should be increased to 6 or 8 or include uniaxial sensors in the estimation (Trifu and Shumila, 2002). Uniaxial sensors can be included to calculate source parameters for a smaller triaxial array if good focal sphere coverage can be obtained, as was evident at Golden Giant (Coulson, 2008c). The source parameters were investigated based on the uniaxial sensors, which were found to show similar temporal changes in these parameters to the triaxial sensors (Figure D2.4e), with however, a substantial offset in the magnitude of the values when compared (Figure D2.4a and b). The $E_s/E_p$ ratios between the two analyses were comparable, although some degree of scatter can be seen (Figure D2.4c), which may be expected when comparing a three component signal to a single. However, these sensors have a different frequency response, are more sensitive to clipping and were found to be on the whole more influenced by noise\(^1\) than the triaxials creating greater scatter in mainly the corner frequencies and resultant source radii (Figure D2.4d).

Thus, more confidence was found in the triaxial sensor response and these are used as the basis for this study. The main reason for averaging source parameters over a number of sensors is to achieve adequate radial coverage, due to non-uniform seismic radiation patterns from the rupture plane, (Madariaga, 1976), and potential site effects (Urbancic et. al., 1996). At the Golden Giant mine, although the triaxial sensors are relatively well configured around the zone of interest, in order to make sure that the resultant temporal changes that are observed were not the influence of a single sensor, (especially the addition of triaxial sensor #16 during the monitoring), a rigorous analysis of the influence of each sensor was performed. This was achieved by selectively removing one of the triaxial sensors from the sensor file, and recalculating the source parameters for a temporal sub set of the events during which the most significant changes occurred from May 2002 to December 2003. The results of each analysis were compared to the full triaxial array, but it should be noted that in the data analysed often not all of the triaxials are employed in the online source calculation, largely due to the influence of either drilling noise close to a sensor or poor signal to noise (S/N) ratios for the very small events (i.e. generally sensor #10 the furthest away would often be the first to be dropped).

\(^1\) The analysis run combining the uniaxial and triaxial sensors was performed using a band pass filter between 100 Hz and 5000 Hz for the uniaxials and between 250 Hz and 5000 Hz for the triaxials. In hindsight it would have been preferable to have run the analysis at a universal low frequency cut-off of 250 Hz. This may have reduced some of the scatter in mainly the source radius.
Figure D2.5, Figure D2.6 and Figure D2.7 shows the results of comparison for seismic moment, energy and source radius. As can be noted a relatively good correlation can be made to the 1:1 relation for the majority of the range for most sub arrays with slight biases either positive or negative. However sensor #13, which is within 100 m of the cluster, shows a tendency to bias larger moments and energies for smaller magnitude events, although the corner frequencies observed through the source radius are less affected (Note: the variance at the low moment range is accentuated because of the log scale and is observed as an offset). This may be due to its orientation with respect to the ruptures, being along strike of the foliation, such that higher frequency signals are attenuated less than the other triaxials, or is in a more ideal location to observe the maximum non-uniform radiation pattern. Another potential influence for lower magnitude events is the influence of noise, and both triaxials #13 and #16, always had a 60 Hz overprint (a result of stray EMR), which, although removed by a band pass filter, the low S/N ratios (as observed in Figure D1.1) may bias the absolute magnitudes (Trifu and Shumila, 2006). In hindsight, individual sensor attenuation corrections (Mercer, 1999) may have harmonized the source parameters, closer to the average and the 1:1 relation. However, the largest influence is still probably the location of the sensor in relation to the source, and the added complexity required to determine a variable attenuation for each sensor was outside the scope of this thesis, with the emphasis on observing any relevant relative changes based on the run of mine online data.

Thus, it was important to make sure that the slight bias did not affect the temporal changes that were observed, and in this case we are more interested in observing the relative changes than accurate calculation of the absolute magnitude. The averaged temporal changes of $M_o$, $E_o$ and $r_o$ for all sub array cases (Figure D2.8a, b and c) indicates that all sensors show exactly the same trends, with varying amplitudes. The main concern that the addition of sensor #16 could have influenced the temporal changes was found not to be the case. Also, although sensor #13 does have an influence on the magnitude of the average change, it is not the cause of them. With regards to the addition of the 4 uniaxial sensors on February 21, 2003, it is noted that the temporal change in source parameters occurs significantly prior to, this starting around February 10, 2003, with seismic activity peaking on February 17, 2003, and indicates that these do not affect the change in source parameters. The significance of these temporal changes are discussed in Coulson (2008b); however, from this analysis confidence was been gained in the consistency of the data.
Figure D2.4 Comparison of source parameters automatically calculated with Triaxial sensors only versus Triaxial plus Uniaxial (note the uniaxial dominate the average), for (a) Moment Magnitude, Mo, (b) Radiated Seismic Energy, Eo, (c) Es/Ep ratio, and (d) Source Radius, ro. (e) Temporal change of average seismic moment, Mo, calculated with uniaxial and triaxial sensors (compare to Figure D2.8a).
Figure D2.5  Comparison of Seismic Moment, \( (M_o) \) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials. Note for the data set not all triaxials are always used.
Figure D2.6  Comparison of Radiated Seismic Energy, \( (E_o) \) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials. Note for the data set not all triaxials are always used, this results in no change for events that did not already include the sensor excluded (i.e. 1:1 relationship results).
Appendix D  Adam Lee Coulson, Doctor of Philosophy, 2009  432

Figure D2.7 Comparison of Source Radius, \( r_o \) for exclusion of one triaxial sensor from the array versus inclusion of all triaxials. Note for the data set not all triaxials are always used.
Figure D2.8  Comparison of temporal changes of the average source parameters for each 3 triaxial analysis dropping one sensor, using a 50 event moving average. (a) Seismic Moment, $M_o$, (b) Radiated Seismic Energy, $E_o$, and (c) Source Radius, $r_o$. 

APPENDIX E

Golden Giant Shaft Pillar Microseismicity
Figure E1  Golden Giant mine shaft pillar microseismicity, from June 22, 1994 to Dec 25, 1994. A total of 114 events. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25].

Figure E2  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1995 to Dec 31, 1995. A total of 661 events.
Figure E3  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1996 to July 07, 1996 (Note database missing events until the end of 1996). A total of 232 events.

Figure E4  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1997 to Nov 20, 1997 (Note: system had triggering issues –random triggers for Nov to Dec 1997). The development of the new East Return Air Raise is captured and location within 8 m. A total of 927 events.
Figure E5  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1998 to Dec 31, 1998. A total of 1819 events.

Figure E6  Golden Giant mine shaft pillar microseismicity, from Jan 1, 1999 to Dec 31, 1999. A total of 1415 events.
Figure E7  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2000 to Dec 31, 2000. A total of 1175 events.

Figure E8  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2001 to Dec 31, 2001. A total of 1141 events.
Figure E9  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2002 to Dec 31, 2002. A total of 7358 events. Events coloured by number of sensors triggered [blue = 5 to 15, magenta = 16 to 14, orange = > 25].

Figure E10  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2003 to Dec 31, 2003. A total of 13,034 events.
Figure E11 Golden Giant mine shaft pillar microseismicity, from Jan 1, 2004 to Dec 31, 2004. A total of 4527 events.

Figure E12 Golden Giant mine shaft pillar microseismicity, from Jan 1, 2005 to Sept 31, 2005. A total of 5720 events.
Figure E13  Golden Giant mine shaft pillar microseismicity, from Jan 1, 2005 to May 31, 2005. This is a partial breakdown of the previous figure to highlight the aseismicity directly below the 4620 L which started after the point of disassociation and May was the last period that events occurred in the EOS cluster around 460-S2 stope. A total of 3682 events.

Figure E14  Golden Giant mine shaft pillar microseismicity, from June 1, 2005 to Sept 23, 2005. Note before the 460-S2 stope is mined this region becomes completely aseismic. A total of 2039 events.
APPENDIX F

Golden Giant Mine Microseismic Cluster Analysis using the Principal Components Analysis (PCA) Technique

4600 level East of Slot (EOS) events
50 Event Sub-Set Windows
Figure F1  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = -8; 04/01/2002, (b) Temporal Window = -7; 04/21/2002 (c) Temporal Window = -6; 05/05/2002 (INITIATION) (d) Temporal Window = -5; 05/23/2002. Note date is based on last event in temporal window.
Figure F2  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = -4; 07/05/2002, (b) Temporal Window = -3; 09/14/2002 (c) Temporal Window = -2; 11/30/2002 (d) Temporal Window = -1; 12/27/2002. Note date is based on last event in temporal window.
Figure F3  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 1; 02/12/2003, (b) Temporal Window = 2; 02/15/2003 (c) Temporal Window = 3; 02/17/2003 (INTERACTION) (d) Temporal Window = 4; 02/17/2003. Note date is based on last event in temporal window.
Figure F4  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 5; 02/18/2003, (b) Temporal Window = 6; 02/20/2003 (c) Temporal Window = 7; 02/22/2003 (d) Temporal Window = 8; 02/23/2002. Note date is based on last event in temporal window.
Figure F5  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 9; 02/24/2003, (b) Temporal Window = 10; 02/27/2003 (c) Temporal Window = 11; 02/28/2003 (d) Temporal Window = 12; 03/02/2003. Note date is based on last event in temporal window.
Figure F6  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 13; 03/05/2003 (COALESCENCE AND LOCALIZATION), (b) Temporal Window = 14; 03/08/2003 (c) Temporal Window = 15; 03/13/2003 (d) Temporal Window = 16; 03/18/2003. Note date is based on last event in temporal window.
Figure F7  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 17; 03/25/2003, (b) Temporal Window = 18; 04/01/2003, (c) Temporal Window = 19; 04/07/2003, (d) Temporal Window = 20; 04/16/2003. Note date is based on last event in temporal window.
Figure F8 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 21; 04/28/2003, (b) Temporal Window = 22; 05/16/2003 (c) Temporal Window = 23; 05/26/2003 (d) Temporal Window = 24; 06/03/2003. Note date is based on last event in temporal window.
Figure F9  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 25; 06/14/2003, (b) Temporal Window = 26; 06/30/2003 (DISASSOCIATION) (c) Temporal Window = 27; 07/17/2003 (d) Temporal Window = 28; 08/24/2003. Note date is based on last event in temporal window.
Figure F10  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 29; 09/29/2003, (b) Temporal Window = 30; 10/18/2003 (c) Temporal Window = 31; 11/17/2003 (d) Temporal Window = 32; 12/30/2003. Note date is based on last event in temporal window.
Figure F11  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 33; 02/13/2004, (b) Temporal Window = 34; 03/29/2004 (c) Temporal Window = 35; 04/24/2004 (d) Temporal Window = 36; 06/09/2004. Note date is based on last event in temporal window.
Figure F12 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 37; 08/31/2004, (b) Temporal Window = 38; 11/17/2004 (c) Temporal Window = 39; 11/30/2004 (d) Temporal Window = 40; 12/07/2004. Note date is based on last event in temporal window.
Figure F13 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 41; 12/12/2004, (b) Temporal Window = 42; 12/30/2004 (c) Temporal Window = 43; 01/31/2005 (d) Temporal Window = 44; 02/12/2005. Note date is based on last event in temporal window.
Figure F14 Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 45; 03/19/2005, (b) Temporal Window = 46; 04/05/2005 (c) Temporal Window = 47; 04/18/2005 (d) Temporal Window = 48; 05/04/2005. Note date is based on last event in temporal window.
Figure F15  Stereonet and location of (PCA) derived planes for each temporal window of clustering events above the 4600 L EOS region. (a) Temporal Window = 49; 05/23/2005, (b) Temporal Window = 50; 09/04/2005. Note date is based on last event in temporal window.