New criteria for rockmass stability and control using integration of seismicity and numerical modelling


Research agency: CSIR
Project number: SIM 02 03 01
Date: January 2006
Executive summary

The main objective of the research project SIM 020301 was to investigate improved criteria for the evaluation of deep-level tabular mine layout design using integration of seismicity and numerical modelling. The final report covers three main areas of study. These areas address the basic formulation of rockmass stability assessment criteria, current practical strategies for integration of seismicity with numerical modelling, and future static and dynamic approaches to numerical modelling of mine seismicity, together with possible optimisation procedures for performing integration exercises.

Issues relating to the assessment of rockmass stability are examined in detail. An initial assessment is made of suggested parameters that can be evaluated from records of seismic activity in designated mining areas, together with the definition of measures of effective rockmass stiffness. It is argued that an explicit computation of ongoing changes in released energy, taking off-reef deformations into account, provides a direct method of assessing progressive seismic activity evolution as mining proceeds. The computation of the so-called Generalised Energy Release (GER) can be accomplished by extending the generic displacement discontinuity method that has been used routinely in the mining industry for many years. This extension is relatively easy to accomplish in simplified, two-dimensional plane strain analyses but is much more difficult for general three-dimensional layout configurations. Examples are used to motivate the utility of the GER concept in layout stability evaluation and to indicate how this may also be used to monitor stability transitions in sub-critical crack growth that may precede rapid deformations.

A practical method for the integration and reconciliation of observed and simulated seismic activity is then outlined. A simplified procedure, based on the MINF computer code for layout evaluation, is proposed, where no feedback of observed seismic activity for adapting the model properties is attempted. The program MINF, and a new integration tool called MINSINT, can be used to estimate the amount of seismicity expected from any sequence of planned mining, based on past seismicity and past mining and the planned mining. At this stage, the effects of changing geological conditions are not modelled. The MINF and MINSINT programs have been integrated as a suite to test the concepts of large-scale back-analyses of mining and seismicity, covering mined areas of about one square kilometre or more in extent. The program MINSINT performs the integration of seismicity and modelling, and estimates the amount and distribution of expected seismicity within planned areas of mining.

Finally, a number of future developments for the integration of seismic activity with numerical modelling and stability assessment are described in Chapter 4, with more detail provided in two appendices. The development of a prototype computer code to simulate three-dimensional rock damage and fracture evolution is described. The test code provides the capability of nucleating a series of crack growth elements at designated positions in space (seed points). The key assumption is made that rock damage zones can be simulated as an assembly of non-intersecting but mutually interactive circular crack elements. It is found that this approach is able to simulate the formation of non-trivial, but simplified, problems such as the formation of coherent shear band structures and the development of plausible fracture pattern orientations that arise near the edges of lead-lag and crush pillar configurations in tabular layouts. However, it is also found that a number of restrictions encumber the application of the current test code, including difficulties with numerical solution convergence in certain cases.

Integration of numerical modelling with seismicity, as well as with other underground observations can be considered as an optimisation problem. The first step of the integration process is an optimisation of the model parameters to minimize the differences between the models of past mining steps and that of the observed seismic data during that mining. A genetic algorithm approach has been tested for a simplified stope model using successive solutions generated by the MINF program with different values of Young’s modulus and strength parameters. The observed data is considered to be the closure at a point in the model and the
genetic algorithm was able to find an optimal solution which lies within the input resolution. A "synthetic annealing" optimisation approach was also investigated. This considers a sequence of runs where the intact strength is decreased by some specified interval and the other parameters are varied within certain ranges until there is a match between observed and predicted behaviour.

Various approaches for integrating dynamic numerical modelling with seismic measurements are also presented. The ultimate dynamic integration is for a numerical code to correctly project the dynamic rockmass response throughout a mined region, simply given inverted seismic parameters such as seismic moment, stress drop or moment tensor solutions. It is shown that this approach requires considerable further development due to uncertainties and limitations in both models and seismic data. A second form of dynamic integration is to use models to test and improve seismic inversions. This approach was used to evaluate the effect of mined out areas on traditional seismic inversions such as moment, magnitude, source radius, stress drop, and moment tensor. Synthetic seismograms generated from models were inverted using standard seismic techniques. Results are presented where the inverted source magnitude and mechanism differs considerably depending on whether a stope was present in the model. Integration of dynamic modelling with the seismic inversion process could therefore lead to improved inversions and a better knowledge of the source mechanisms. A third form of dynamic integration is in the use of active seismic velocity scans. Active velocity scans would provide useful diagnostics on the degree of fracturing for mining applications, and dynamic models provide a means of interpreting this fracturing. These velocity scans could also provide an important technique for evaluating and optimising models of fracture zone development. Examples are presented of simulated velocity scans and the effect of fracturing is shown to be detectable in the resulting waveforms.

In summary, the main achievements of this project are:

(1) A review of stability criteria and the proposal of a generalised energy release criterion (GER) which includes off-reef damage processes.

(2) The formulation of a new integration methodology and its embodiment in a program called MINSINT.

(3) The evaluation of a novel method for simulating three-dimensional fracture growth and seismic damage processes

(4) The evaluation of what have been termed future technologies, including methods of automating integration as well as various forms of dynamic integration.
Acknowledgements

Sponsorship of this work by the Mine Health and Safety Council through the SIMRAC Rock Engineering research programme, is gratefully acknowledged. Members of the SIMRAC Rock Engineering expert committee have given useful feedback during workshops and encouraged us to move forward with the predictive model presented in the Proposed Integration Method. In particular, Duncan Adams has helped both in technical feedback and in the resolution of practical problems. Driefontein Consolidated are thanked for permission to use their data and to Kevin Riemer and Ricardo Ferreira for assistance in preparing the data. We benefited from discussions with our Miningtek colleagues, especially Ray Durrheim, Gökhan Güler, Terry Hagan, Jan Kuijpers, Francois Malan and Fernando Vieira. We also benefited from comments by anonymous reviewers of papers.
# Table of contents

<table>
<thead>
<tr>
<th>Section</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Executive summary</td>
<td>2</td>
</tr>
<tr>
<td>Acknowledgements</td>
<td>4</td>
</tr>
<tr>
<td>Table of contents</td>
<td>5</td>
</tr>
<tr>
<td>List of figures</td>
<td>8</td>
</tr>
<tr>
<td>List of tables</td>
<td>15</td>
</tr>
<tr>
<td>Glossary of abbreviations</td>
<td>16</td>
</tr>
<tr>
<td>1 Introduction</td>
<td>17</td>
</tr>
<tr>
<td>2 Criteria to assess rockmass stability</td>
<td>18</td>
</tr>
<tr>
<td>2.1 Seismic parameters</td>
<td>18</td>
</tr>
<tr>
<td>2.2 Analogues for seismic behaviour</td>
<td>19</td>
</tr>
<tr>
<td>2.2.1 Block-slider models</td>
<td>19</td>
</tr>
<tr>
<td>2.2.2 Self-organising criticality</td>
<td>21</td>
</tr>
<tr>
<td>2.3 Simplified deformation mechanisms and models</td>
<td>21</td>
</tr>
<tr>
<td>2.3.1 Damage in a bracket pillar region</td>
<td>21</td>
</tr>
<tr>
<td>2.3.2 Simulation of fracture zone damage using non-intersecting fracture clusters</td>
<td>23</td>
</tr>
<tr>
<td>2.4 Assessment of rockmass stability</td>
<td>27</td>
</tr>
<tr>
<td>2.4.1 Classical stability analysis</td>
<td>27</td>
</tr>
<tr>
<td>2.4.2 Stiffness and energy release as measures of system stability</td>
<td>27</td>
</tr>
<tr>
<td>2.4.3 Statistics of energy release increments in mining a parallel-sided panel</td>
<td>29</td>
</tr>
<tr>
<td>2.4.4 Incremental extraction of three longwall mining panels</td>
<td>30</td>
</tr>
<tr>
<td>2.4.5 Cap stress model</td>
<td>37</td>
</tr>
<tr>
<td>2.5 Grid size effects, slip-weakening and fracture nucleation zone size</td>
<td>39</td>
</tr>
<tr>
<td>2.5.1 Grid size effects</td>
<td>39</td>
</tr>
<tr>
<td>2.5.2 Slip-weakening and fracture nucleation</td>
<td>41</td>
</tr>
<tr>
<td>2.5.3 Slip-weakening on multiple, overlapping fault structures or random mesh assemblies</td>
<td>46</td>
</tr>
<tr>
<td>2.6 Stope closure and ride as an indication of rockmass stability</td>
<td>51</td>
</tr>
<tr>
<td>2.7 Sub-critical crack growth simulation</td>
<td>55</td>
</tr>
<tr>
<td>2.8 Generalised Energy Release as a criterion for layout stability assessment</td>
<td>59</td>
</tr>
<tr>
<td>3 Proposed Integration Method</td>
<td>60</td>
</tr>
<tr>
<td>3.1 Introduction: practical integration work</td>
<td>60</td>
</tr>
<tr>
<td>3.2 Integration concepts</td>
<td>60</td>
</tr>
<tr>
<td>3.2.1 Assumptions</td>
<td>62</td>
</tr>
<tr>
<td>3.2.2 Methodology</td>
<td>63</td>
</tr>
<tr>
<td>3.3 Modelling seismic action</td>
<td>64</td>
</tr>
<tr>
<td>3.4 Measuring the seismic response to mining</td>
<td>65</td>
</tr>
<tr>
<td>3.5 Estimate future seismicity</td>
<td>69</td>
</tr>
<tr>
<td>3.5.1 Numerical modelling only</td>
<td>69</td>
</tr>
<tr>
<td>3.5.2 Previous seismicity only</td>
<td>70</td>
</tr>
<tr>
<td>3.5.3 Integration of seismicity and modelling</td>
<td>70</td>
</tr>
<tr>
<td>3.6 Case study</td>
<td>71</td>
</tr>
</tbody>
</table>
3.6.1 Selection of seismic events ................................................................. 71
3.6.2 Identification of “working areas” ...................................................... 72
3.6.3 Analysis of predictions using the three methods ....................... 73
3.6.4 Conclusions and discussion ............................................................ 77

3.7 Introduction to the programs MINF and MINSINT ................................. 77
3.7.1 MINF .................................................................................................. 78
3.7.2 MINSINT ........................................................................................... 79

3.8 Working with MINF and MINSINT ...................................................... 79
3.8.1 Running MINF and MINSINT .......................................................... 79
3.8.2 Editing input files using DFTWrap .................................................. 80
3.8.2.1 DFTWrap Step-by-step ................................................................. 80

4 Development of future technologies ..................................................... 85
4.1 Development of three-dimensional models to simulate seismic activity and rock
failure processes ...................................................................................... 85
4.1.1 Mesh-free concepts in tabular excavation stress analysis .............. 86
4.1.2 Simulation of three-dimensional fracture and damage processes .... 88
4.1.3 Simplified strategy for simulation of multiple fracture clusters ....... 89
4.1.4 Case studies and examples ............................................................... 91
4.1.4.1 Circular sliding crack with adjacent wing cracks ....................... 91
4.1.4.2 Shear fracture growth from random seed points ....................... 94
4.1.4.3 Simulation of fracture growth near tabular excavations .......... 100
4.1.5 Conclusions on 3D fracture growth models ................................. 107
4.2 Other forms of integration ................................................................. 107
4.2.1 Optimisation processes for integration ........................................... 107
4.2.2 Integration of seismic observations with dynamic numerical modelling .................................................................................. 110
4.2.3 Improving seismic analysis through integration with dynamic numerical
models .................................................................................................. 112
4.2.3.1 Description of models ................................................................. 113
4.2.3.2 Calculation of source parameters .............................................. 114
4.2.3.3 Results ..................................................................................... 119
4.2.3.4 Conclusions ........................................................................... 123
4.2.4 Integrating modelling with active seismic measurements .............. 124

5 Conclusions ............................................................................................. 130

Publications ................................................................................................. 132

References ..................................................................................................... 133

Appendix A Further MINF and MINSINT Information .................................. 138
A.1 Units and sign conventions for MINF and MINSINT ........................... 138
A.2 Seismicity input file (“fnseis”) ............................................................... 138
A.3 Brief description of MinView3D ........................................................... 140
A.4 Features of MINF not used in the integration procedure ................. 141
A.4.1 Large (1024 by 1024) mining problems (MINF_LARGE) ........... 141
A.4.2 Huge (2048 by 2048) mining problems (MINF_HUGE) ............. 141
A.4.3 Soft seam, for coal mining (MIN_COAL) ..................................... 141
A.4.4 Shallow mining with free surface, for coal mining (MIN_COAL) .... 141
A.4.5 Seismicity generation, for deep mining (MINF) .............................. 141
A.4.6 Slippery reef-parallel planes (MINF) ........................................... 141
A.4.7 Identification of isolated pillars & APS listing (MINF) ................. 142
A.4.8 Pillar failure (MINF) ....................................................................... 142
Appendix B Integration via automated optimisation procedures ........................................ 145
B.1 Introduction ............................................................................................................... 145
B.2 Fitness functions ..................................................................................................... 145
B.3 Optimisation methods ............................................................................................. 149
B.3.1 Genetic algorithms ............................................................................................. 149
B.3.2 Simulated annealing .......................................................................................... 150
B.4 Example of integration using genetic algorithm .................................................... 150
B.5 Integration concepts using simulated annealing ..................................................... 154
B.6 Conclusions ............................................................................................................ 158
B.7 References ............................................................................................................. 159
Appendix C Project Proposal .......................................................................................... 161
List of figures

Figure 2.1  Plot of simulated energy release increments recorded for a 20 m x 20 m sliding block with two slip resistance states corresponding to friction angles of 35 degrees and 10 degrees respectively. The upper surface of the block is displaced horizontally at a constant rate and has a fixed, uniform vertical displacement of 0.032 m. The lower surface slides on an elastic base ........................................ 20

Figure 2.2  Cumulative energy release increment distributions for three different sliding blocks. The cumulative distributions of the energy release increments do not follow simple power law relationships but exhibit complex stick-slip movements 20

Figure 2.3  Mining up to a fault indicating a simplified failure mechanism near the stope-fault intersection point ................................................................................................ 22

Figure 2.4  Slip profile on a fault intersecting a stope, examining the effect of a simplified “day lighting” damage mechanism .................................................................................. 22

Figure 2.5  Comparison between different constitutive properties assigned to fracture zone elements showing the energy release increments corresponding to each fracture growth step in the simulation of failure of the bracket pillar region ahead of a panel face................................................................. 23

Figure 2.6  Effect of fault overlap on energy release, compared to a single fault of 100 m length. ................................................................................................................. 25

Figure 2.7  Fracture growth pattern after a single mining step of 1 m with only shear growth seeds and non-intersecting cracks .............................................................................. 25

Figure 2.8  Fracture growth pattern after a single mining step of 1 m with combined shear and tension growth seeds and non-intersecting cracks. ..................................... 26

Figure 2.9  Fracture growth pattern after a single mining step of 1 m with combined shear and tension fracture initiated within a random Delaunay mesh.............................................. 26

Figure 2.10 A square excavation with potential adjacent slip planes at different distances from the edge of the excavation. .......................................................................................... 26

Figure 2.11 Energy release quantities computed from deformations occurring on each of a set of fractures located at increasing distances from the edge of a 40 m x 40 m square excavation ........................................................................................................... 28

Figure 2.12 Variation of shear slip stiffness computed from deformations occurring on each of a set of fractures located at increasing distances from the edge of a 40 m x 40 m square excavation ........................................................................................................... 28

Figure 2.13 Energy release increment size distribution in simulated mining of a single parallel-sided panel. ................................................................................................. 29

Figure 2.14 Detailed extraction sequence of three longwall panels. ........................................ 30

Figure 2.15 Simulated mining sequence of three longwall panels with final spans of 140 m and final pillar sizes of 40 m ................................................................................................. 30

Figure 2.16 Cumulative fracture damage zone developed after four extraction steps of the right hand longwall panel. ................................................................................... 31

Figure 2.17 Cumulative fracture damage zone developed after five extraction steps of right hand longwall panel. Extensive formation of additional fracturing is observed to occur in the right hand pillar region .................................................... 31

Figure 2.18 Cumulative fracture damage zone developed after five extraction steps of the left hand longwall panel. ................................................................................................. 32

Figure 2.19 Cumulative fracture damage zone developed after six extraction steps of the left hand longwall panel. Extensive formation of additional fracturing is observed to occur in the left hand pillar region as well as some additional fracturing in the “remote” region of the previously mined right hand panel ........................................ 33
Figure 2.20 Cumulative (final) fracture damage zone developed after seven extraction steps of the left hand longwall panel. ................................................................. 33
Figure 2.21 Incremental fracture length mobilised at each stage of mining ......................... 34
Figure 2.22 Cumulative moment plotted against cumulative span mined showing intermittent accelerations in the slip movement on mobilised fractures .............................................. 35
Figure 2.23 Incremental moment changes associated with cumulative mining span. The peak values associated with significant accelerations in fracture growth are highlighted. 35
Figure 2.24 Incremental energy release changes associated with cumulative mining span. The peak values associated with significant accelerations in fracture growth are highlighted. ................................................................. 36
Figure 2.25 Apparent volume as a function of cumulative mined span ................................ 36
Figure 2.26 Seismic stiffness modulus as a function of cumulative mined span .................... 37
Figure 2.27 Average stress in two pillar regions between three longwall panels .................. 38
Figure 2.28 Average stress in pillar regions when no off-reef failure is permitted ................ 38
Figure 2.29 Energy release increments .............................................................................. 39
Figure 2.30 Effect of tessellation density on simulation of mobilised fracture length in sequential panel mining simulation ................................................................. 40
Figure 2.31 Effect of tessellation density on cumulative moment in sequential panel mining simulation ........................................................................................................................................................................ 40
Figure 2.32 Effect of tessellation density on cumulative energy release in sequential panel mining simulation ................................................................. 41
Figure 2.33 Slip profiles on an incrementally loaded fault (element size = 0.2 m) showing loss of stability at a critical nucleation length of 6.8 m ......................................................... 42
Figure 2.34 Residual cohesion on incrementally loaded fault (element size = 0.2 m) showing loss of stability at a critical nucleation length of 6.8 m ......................................................... 43
Figure 2.35 Slip profiles on incrementally loaded fault (element size = 5.0 m) ....................... 43
Figure 2.36 Residual cohesion on incrementally loaded fault (element size = 5.0 m) .......... 44
Figure 2.37 Cumulative energy release in three-longwall mining problem for various choices of the effective critical slip nucleation length ................................................................. 45
Figure 2.38 Cumulative energy release plotted against cumulative scaled moment to indicate the effective stress drop in the three-longwall mining problem for various choices of the effective critical slip nucleation length ................................................................. 45
Figure 2.39 Final slip profiles computed on three parallel fault planes ................................. 47
Figure 2.40 Residual cohesion on three parallel fault planes ................................................ 48
Figure 2.41 Energy release for various horizontal fault planes subjected to incremental shear loads ...................................................................................................................................................................................... 48
Figure 2.42 Activated cracks at peak horizontal load of 220 MPa (Step 3) ............................. 49
Figure 2.43 Activated crack pattern at peak horizontal load of 230 MPa (Step 4) ................. 49
Figure 2.44 Cumulative energy release and moment profiles for incremental loading of random crack assembly (average element length = 0.6 m) ........................................ 50
Figure 2.45 Energy release increments plotted against scaled moment increments for random crack assembly loading ................................................................................................................................................................................................. 50
Figure 2.46 Schematic mining layout with footwall event occurring parallel to the face of the centre panel ...................................................................................................................................................................................... 52
Figure 2.47 Schematic mining layout with footwall event occurring parallel to the face of the centre panel – representative cross section ................................................................................................................................................................................................. 52
Figure 2.48 Dip closure profiles parallel to the stope face before and after a face-parallel event in the stope footwall. Two post-event profiles are shown corresponding to assumed fault sliding friction angles of 30 degrees and zero degrees respectively. ................................................................................................................................................................................................. 53
Figure 2.49 Incremental closure profiles plotted along a line parallel to and 7 m from the central stope face................................................................................................ 53
Figure 2.50 Incremental strike ride component plotted along a line parallel to and 7 m from the central stope face................................................................................................ 54
Figure 2.51 Skin stress component parallel to the stope face ($\tau_{xx}$) plotted along a strike line, showing strong induced tensile stress in the footwall, corresponding to the footwall face-parallel face “event”. ........................................................................ 54
Figure 2.52 Strike oriented skin stress plotted along strike showing induced compressive stresses in both the hanging-wall and the footwall............................................... 55
Figure 2.53 Simulation of sub-critical crack growth. Cumulative slip on a fault, with a given peak load, as a function of time (arbitrary time units). .................................................. 56
Figure 2.54 Cumulative energy release on a slipping fault, as a function of time, showing rapid onset of unstable slip (arbitrary time units). ........................................... 57
Figure 2.55 Detail of energy release increments in each time step near the onset of unstable slip (arbitrary time units). ........................................................................... 57
Figure 2.56 Detail of energy release increments in sub-critical slip phase. The magnitude of the release increments is much smaller than in Figure 2.55 (arbitrary time units). 58
Figure 2.57 Effect of driving stress on the sub-critical slip velocity before the onset of unstable slip behaviour (arbitrary time units).................................................. 58
Figure 3.1 Proposed components of a software package designed to implement integration (from GAP 603). ................................................................................. 60
Figure 3.2 A truncated version of Figure 3.1 to illustrate the components used in the MINF/MINSINT suite. ............................................................................. 61
Figure 3.3 Cartoon for simple weather forecast............................................................ 61
Figure 3.4 Elastic strain energy released........................................................................ 62
Figure 3.5 A sketch illustrating (a) distribution of a seismic event onto the grid squares that are currently being mined, followed by (b) distribution of values in currently mined areas onto planned mining areas........................................................................ 66
Figure 3.6 Smoothing functions. Each is circularly symmetrical. (a) “A” accounts for location error. Expanding functions “A” through “B” or beyond accounts for a region of influence that contains a sufficient number of events and grid elements. (b) Conical function to account for event size..................................................... 67
Figure 3.7 A sketch illustrating the process of attributing a seismic event to the area currently being mined. The process is described in the text. ............................. 68
Figure 3.8 Mining and seismicity (M~2) ............................................................................. 71
Figure 3.9 Size distribution of events identified as blast events, mining-induced events and all events. .................................................................................................................. 72
Figure 3.10 An example of extraction in the three months from November 2001 shown as contours, the larger seismic events as opaque balls and mining steps up to August 2001 as outlines...................................................................................... 73
Figure 3.11 Predicted and observed seismic moment, GN-m per grid element.................... 74
Figure 3.12 Comparison of the observed number of events per mined element with the predicted number of elements by each of three model approaches. The ellipse in (c) is drawn around some of the data that were over-predicted. ......................... 75
Figure 3.13 Comparison between the observed number of events per mined element and source radius time apparent stress, MPa-m. .................................................. 76
Figure 3.14 Flow-chart of integration procedure. ................................................................ 78
Figure 3.15 Main window of DFTWrap............................................................................ 81
Figure 3.16 File open dialogue box used to select the DFT file........................................ 82
Figure 3.17 Data structure – top level node in tree.......................................................... 82
Figure 3.18 DFT file format error message. ........................................................................ 82
Figure 3.19  First tier of the branches of the tree structure. .................................................... 83
Figure 3.20  Individual variables of a branch in the tree structure. .................................................... 83
Figure 3.21  Range error message. ........................................................................................ 84
Figure 3.22  Navigating to the directory containing MINF and MINSINT. .................................................... 84
Figure 3.23  The About dialogue. ........................................................................................ 84
Figure 4.1  Example of mesh-free analysis of tabular layout problem. .................................................... 87
Figure 4.2  Closure contours for layout problem (solution from 3DIGS). .................................................... 87
Figure 4.3  Mesh-free solution of tabular layout problem using a local quadratic variation
moving least-squares estimate of the closure at each node point shown in Figure
4.1. ........................................................................................................................................... 88
Figure 4.4  Circular crack with adjacent seed points. The crack is parallel to the x-axis and is
inclined at an angle of approximately 26.6 degrees to the z-axis. (The circular
outline is depicted approximately using a twelve-sided polygon and
appears to be elliptical since the crack is inclined at an angle to the z-axis). .... 92
Figure 4.5  Wing cracks nucleated in tension adjacent to a sliding circular crack (edge view
along the x-axis, parallel to the crack planes). ........................................................................... 92
Figure 4.6  Wing cracks nucleated in tension adjacent to a sliding circular crack (oblique
view). ........................................................................................................................................ 93
Figure 4.7  Conjugate shear crack initiation from seed points when far-field x and y-direction
stress is compressive. .................................................................................................................. 93
Figure 4.8  Fracture growth from random seed points in a uniform stress field. Seed point
positions are indicated by crosses. ............................................................................................. 95
Figure 4.9  Fracture growth from random seed points in a uniform stress field with uniformly
distributed crack radius values varying between 2 m and 5 m. (View along y-axis
− intermediate principal stress direction). ................................................................................. 95
Figure 4.10 Fracture growth from random seed points in a uniform stress field with uniformly
distributed crack radius values varying between 2 m and 5 m. View along x-axis
(minor principal stress direction) showing the distribution of different crack sizes. .......... 96
Figure 4.11 Fracture growth from random seed points in a uniform stress field with uniformly
distributed crack radius values varying between 1 m and 5 m. (View along y-axis
− intermediate principal stress direction). ................................................................................. 96
Figure 4.12 Fracture growth from random seed points in a uniform stress field with uniformly
distributed crack radius values varying between 1 m and 5 m. View along x-axis
(minor principal stress direction) showing the distribution of different crack sizes. .......... 97
Figure 4.13 Fracture growth from random seed points in a uniform stress field with
exponentially distributed crack areas corresponding to radius values varying
between 1 m and 5 m. (View along y-axis − intermediate principal stress direction)................. 97
Figure 4.14 Fracture growth from random seed points in a uniform stress field with
exponentially distributed crack areas corresponding to radius values varying
between 1 m and 5 m. View along x-axis (minor principal stress direction) showing the distribution of different crack sizes. .... 97
Figure 4.15 Histogram of the radius values corresponding to the selected crack growth
elements shown in Figure 4.11 and Figure 4.12, compared to the uniform
distribution frequency of the parent seed population. ............................................................... 98
Figure 4.16 Energy release increments, plotted for each crack growth initiation step,
corresponding to the uniform seed radius distribution crack pattern shown in
Figure 4.11. ........................................................................................................................................ 99
Figure 4.17 Cumulative net energy release increments, plotted on log-log scales for the crack steps, corresponding to the uniform seed radius distribution crack pattern shown in Figure 4.11.

Figure 4.18 Development of fractures initiated at fixed-radius seed growth points ahead of a lead-lag panel mining configuration – plan view.

Figure 4.19 Development of fractures initiated at fixed-radius seed growth points ahead of a lead-lag panel mining configuration – face-parallel view.

Figure 4.20 Development of fractures from random seed points with a random distribution of radius values ranging from 0.25 m to 1.5 m (plan view).

Figure 4.21 Development of fractures from random seed points with a random distribution of radius values ranging from 0.25 m to 1.5 m (face-parallel view).

Figure 4.22 Fracture pattern developed with fixed radius seed elements and with cohesion and slip-weakening values specified on the activated crack elements (plan view).

Figure 4.23 Fracture pattern developed with fixed radius seed elements and with cohesion and slip-weakening values specified on the activated crack elements (face-parallel view).

Figure 4.24 Progressive pattern of fracturing in the formation of a crush pillar – initial face position.

Figure 4.25 Progressive pattern of fracturing in the formation of a crush pillar – mining step 1.

Figure 4.26 Progressive pattern of fracturing in the formation of a crush pillar – mining step 2.

Figure 4.27 Progressive pattern of fracturing in the formation of a crush pillar – mining step 3.

Figure 4.28 Progressive pattern of fracturing in the formation of a crush pillar – mining step 4.

Figure 4.29 (a) MINF model of a deep level stope showing the point (x) where the closure is compared. (b) Comparison of average fitness (solid line) of the solutions (diamonds) with number of generations.

Figure 4.30 (a) Effect of intact strength on run time of analysis. b) Effect of intact strength on cumulative moment in MINF simulations of seismic activity.

Figure 4.31 a) WAVE Model geometry b) effect of source rise time on waveform in near-field for S- shaped source time function.

Figure 4.32 a) Mine plan with box showing WAVE model region. (‘*’ denotes the event and ‘■’ the receiver) b) Frequency domain comparison of observed and modelled waveforms for step source time function with 1m and 5m grid sizes.

Figure 4.33 Model of the fault-plane, pillar and stope. The dashed lines indicate the model boundaries.

Figure 4.34 Ideal shapes of displacement, velocity and acceleration spectra showing the corner frequency.

Figure 4.35 Representation of the nine possible couples of the moment tensor M. The directions of the force and arm of the couple are denoted by the indices i and j, respectively (from Aki & Richards, 1980).

Figure 4.36 Comparison of velocity seismograms showing difference in frequency content (site 105, models 1A and 1B).

Figure 4.37 Comparison of velocity seismograms showing S-wave amplification (site 197, models 2A and 2B).

Figure 4.38 Radiation patterns and fault plane solutions of punch pillar models 1A and 1B.

Figure 4.39 Radiation patterns and fault plane solutions of crush pillar models 2A and 2B.
Figure 4.40 Geometry for the fractured pillar models. (a) Three-dimensional view of fractured pillar showing positions of source and receiver and dimensions. (b) Plan cross-sections through four pillar models with different sized regions of fracturing with fractured “annulus” varying from (i) 0 m (elastic) (ii) 0.4 m (iii) 0.85 m (iv) 1.3 m. In all cases the fractures range in size from 0.2 m to 0.6 m with a crack density of 0.1.

Figure 4.41 Results for the four pillar models showing the effect of different sizes of the fractured region on wave-speed. (a) Time-domain waveforms (b) Fourier phase-difference relative to the source waveform. (i), (ii), (iii) and (iv) refer to the four models in Figure 4.40b.

Figure 4.42 Cross-section through the three-dimensional fractured hangingwall model.

Figure 4.43 Waveforms received in the hangingwall at 0 m, 2 m, 4 m, 6 m, 8 m, 10 m, 12 m and 14 m from the source, with a time window of 5 ms. $P_{th}$, $S_{th}$ and $R_{th}$ are the theoretical arrival times for the $P$, $S$ and Rayleigh waves. Amplitudes aren’t shown but are scaled according the peak in each trace and reduce with increasing distance.

Figure 4.44 Waveforms as for Figure 4.43, but with amplitudes zoomed ten-fold. $P_{th}$, $S_{th}$ and $R_{th}$ are the theoretical arrival times for the $P$, $S$ and Rayleigh waves. $P_{frac}$ is the $P$-wave arrival for the fractured model, while $P_2$ is a second $P$-wave arrival which after a certain distance arrives before the direct wave.

Figure A.1 Appearance of Windows control panel for MinView3D.

Figure A.2 Sketch of the strength of edge elements as a function of distance from the face or abutment.

Figure A.3 Contours of EDGE from a hypothetical mining layout. The small numbers of contours on the small pillars indicates that they will only be made weaker than the larger pillars.

Figure B.1 Gutenberg – Richter plot of cumulative number of events against moment for the five years of mining on a deep level stope.

Figure B.2 The cumulative moment for 3.9 years of mining on a deep level stope showing the effect of removing small magnitude events.

Figure B.3 Relationship between the cumulative moment and the minimum magnitude events included into the summation for five years of mining on a deep level stope.

Figure B.4 Schematic of response of a DD element to mining in MINF, showing the relative magnitudes of the input parameters.

Table B-1 Examples of genes in binary universe and example of crossover and mutation.

Figure B.5 MINF model of a deep level stope showing the point (x) where the closure is compared.

Figure B.6 Comparison of average fitness (solid line) of the solutions (diamonds) with number of generations.

Figure B.7 The average fitness (equivalent to the difference between observed closure and predicted closure) as a function of the average input Young’s modulus per generation.

Figure B.8 Example of fault plane discretised into squares with each having a random friction angle between 30° and 40°. A typical gene (for dark grey shaded block) would be “6;8;38”, i.e. column number; row number; friction angle, assuming that cohesion is zero.

Figure B.9 Effect of intact strength on cumulative moment in MINF simulations of seismic activity.

Figure B.10 Cumulative moment with mining steps for different magnitudes of stress drop with an intact strength of 225 MPa.
Figure B.11  Effect of intact strength on run time of analysis. ............................................... 156
Figure B.12  Mine plan and seismicity simulated using MINF. .................................................. 157
Figure B.13  Closure time curves for different values of cap stress. ........................................... 157
### List of tables

<table>
<thead>
<tr>
<th>Table</th>
<th>Description</th>
<th>Page</th>
</tr>
</thead>
<tbody>
<tr>
<td>Table 2-1</td>
<td>Seismic parameters defined for a circular shear crack.</td>
<td>18</td>
</tr>
<tr>
<td>Table 2-2</td>
<td>Energy release increments (MJ/m) for different fracture growth rules</td>
<td>24</td>
</tr>
<tr>
<td>Table 2-3</td>
<td>Comparative simulations varying the element size and effective slip nucleation length.</td>
<td>44</td>
</tr>
<tr>
<td>Table 2-4</td>
<td>Summary of parallel fault configuration analyses.</td>
<td>46</td>
</tr>
<tr>
<td>Table 3-1</td>
<td>Six different ways in which the seismicity may be quantified in MINSINT</td>
<td>67</td>
</tr>
<tr>
<td>Table 3-2</td>
<td>Symbols used for array operations and types.</td>
<td>69</td>
</tr>
<tr>
<td>Table 3-3</td>
<td>Number of blast and induced events greater than three values of Moment-Magnitudes</td>
<td>72</td>
</tr>
<tr>
<td>Table 3-4</td>
<td>Some parameters that might be used to quantify the “size” of a seismic event.</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Scaling is in terms of $r_0 =$ source radius and $\tau =$ seismic stress (apparent stress or static stress drop)</td>
<td></td>
</tr>
<tr>
<td>Table 4-1</td>
<td>Model description.</td>
<td>113</td>
</tr>
<tr>
<td>Table 4-2</td>
<td>Source parameters.</td>
<td>120</td>
</tr>
<tr>
<td>Table 4-3</td>
<td>Source contributions.</td>
<td>120</td>
</tr>
<tr>
<td>Table A-1</td>
<td>Units used for internal storage and for reporting results.</td>
<td>138</td>
</tr>
<tr>
<td>Table A-2</td>
<td>Corrections made to header line of seismic catalogue file.</td>
<td>138</td>
</tr>
<tr>
<td>Table A-3</td>
<td>List of header titles and their meaning.</td>
<td>139</td>
</tr>
<tr>
<td>Table A-4</td>
<td>The following parameters are used to define the pillar strengths described above.</td>
<td>144</td>
</tr>
<tr>
<td>Table B-1</td>
<td>Examples of genes in binary universe and example of crossover and mutation.</td>
<td>152</td>
</tr>
</tbody>
</table>
# Glossary of abbreviations

<table>
<thead>
<tr>
<th>Abbreviation</th>
<th>Definition</th>
</tr>
</thead>
<tbody>
<tr>
<td>ERR</td>
<td>Energy Release Rate</td>
</tr>
<tr>
<td>ESS</td>
<td>Excess Shear Stress</td>
</tr>
<tr>
<td>VESS</td>
<td>Volume Excess Shear Stress</td>
</tr>
<tr>
<td>ER</td>
<td>Energy Release. Refers to strain energy release associated with a particular area of mining, as used in MINF / MINSINT.</td>
</tr>
<tr>
<td>APS</td>
<td>Average Pillar Stress</td>
</tr>
<tr>
<td>MINF</td>
<td>The Boundary element code enhanced for this project. MINF is short for &quot;MINing simulation using Fourier transforms.</td>
</tr>
<tr>
<td>MINSINT</td>
<td>The code developed in this project to perform mining and seismicity integration. MINSINT is short for MINing and Seismicity INTegrator.</td>
</tr>
<tr>
<td>BIC</td>
<td>Bushveld Igneous Complex</td>
</tr>
<tr>
<td>GER</td>
<td>Generalised Energy Release</td>
</tr>
<tr>
<td>VCR</td>
<td>Ventersdorp Contact Reef</td>
</tr>
<tr>
<td>PPV</td>
<td>Peak Particle Velocity</td>
</tr>
</tbody>
</table>
1 Introduction

The potential benefit of combining available records of mine seismic activity with an interpretation of this information using numerical modelling simulations of large-scale rock deformation has been recognised for a number of years in several SIMRAC-funded research projects. Specifically, project GAP 603 (Mendecki et al., 2001) attempted an initial investigation of fundamental aspects of the integration of seismic monitoring with numerical modelling. Part of the integration concept considered in project GAP 603 was the possibility of using ongoing records of seismic activity to adapt the structure of a numerical modelling tool for future assessments of expected mining induced seismic events. The major difficulty with this approach is in modifying the model. The direct superposition of observed events, represented as oriented slip patches, is not necessarily compatible with the intrinsic stress fields arising from an existing equilibrated state of the model. Highlighted seismic events that are found to be incompatible with the model state should, ideally, be used to infer appropriate changes to the structural environment such as the presence of geological features or average rockmass strength properties.

It is also apparent that the numerical models currently available for the analysis of tabular layout mining problems are not capable of performing the self-adaptive and non-linear analyses demanded by a general seismic monitoring/integration procedure. One objective of the current project SIM 02 03 01 is therefore to investigate improvements to tabular layout modelling tools that can be used for more detailed, explicit simulations of off-reef damage processes. Simplified forms of such a tool are available for two-dimensional plane strain analyses and are proposed for the simulation of full three-dimensional failure analysis. Simultaneously, a practical computational approach is proposed that can be applied to reconcile observed seismic activity with stress changes generated by currently available computer programs for large-scale tabular mine analysis. The simplified analysis tool can be employed to highlight incompatibilities between observed and predicted seismic activity, and can also be used to suggest the necessary structure (in the form of fault features, dykes or rockmass properties) that is required to be incorporated into detailed models of explicit inelastic deformation and seismic activity. Some possible techniques for the optimisation of this process are investigated.

It is recognised that implicit goals of the integration of seismic activity with numerical modelling include the assessment of large-scale rockmass stability transitions as mining proceeds, as well as the assessment of long-term seismic activity in a given shaft region. An important issue is to formulate an appropriate criterion for the evaluation of proposed tabular mine layouts. A detailed investigation is carried out to assess the potential of using an adapted energy release criterion for this purpose. This approach is generically compatible with existing displacement discontinuity boundary element methods but could be applied, in principle, to any computational procedure that is used for explicit simulation of rockmass damage.
2 Criteria to assess rockmass stability

This chapter investigates issues related to the evaluation of rockmass stability. It firstly assesses suggested parameters that can be evaluated from the seismic record. It then shows that explicitly computing the ongoing changes in released energy, including off-reef deformations, provides a direct method of assessing the evolution of seismic activity as mining proceeds. This results in the proposal for the computation of the so-called Generalised Energy Release (GER) as a useful criterion in the evaluation of layout stability. The criterion is motivated through plane strain analyses using the computer code DIGS (Napier, 1990; Napier, 2003). The extension to three-dimensional analysis is investigated in Chapter 4.

2.1 Seismic parameters

The detailed numerical simulation of failure and slip movements near deep level excavations in both space and time is currently not possible for large-scale mining regions. However, it is desirable to develop simulation tools that can provide insight into the mechanisms of seismic activity and that can assist in the interpretation of seismic data. Certain seismic parameters have been proposed (e.g. Mendecki, 1997) to track ongoing seismic activity in the rockmass and to assist in the identification of potentially hazardous conditions. It is important to determine whether these parameters can be interpreted in terms of numerical models. At the simplest level, some insight into the mechanistic nature of these parameters can be obtained by considering the prototypical process of slip across a circular (“penny shaped”) crack.

Consider the particular case when a constant stress drop \( q \) occurs across a flat crack of radius \( a \) in a rockmass with shear modulus \( G \). The average slip \( D \) across the crack, seismic moment \( M_0 \) and released energy, \( E \) are given in the following table together with some "seismic parameters" that are defined in terms of \( q, M_0 \) and \( E \).

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Formula</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average slip, ( D ) (m)</td>
<td>( D = \frac{Cqa}{\pi G} )</td>
</tr>
<tr>
<td>Moment, ( M_0 ) (MJ)</td>
<td>( M_0 = Cqa^3 )</td>
</tr>
<tr>
<td>Energy release, ( E ) (MJ)</td>
<td>( E = \frac{Cq^2a^3}{2G} )</td>
</tr>
<tr>
<td>Apparent stress, ( \sigma_A )</td>
<td>( \sigma_A = \frac{GE}{M_0} = \frac{q}{2} )</td>
</tr>
<tr>
<td>Apparent volume, ( V_A )</td>
<td>( V_A = \frac{M_0}{2\sigma_A} = \frac{M_0}{q} = C a^3 )</td>
</tr>
<tr>
<td>Seismic stress, ( \sigma_S )</td>
<td>( \sigma_S = \frac{2GE}{M_0} = 2\sigma_A = q )</td>
</tr>
<tr>
<td>Seismic strain, ( \varepsilon_S )</td>
<td>( \varepsilon_S = \frac{M_0}{2G \Delta V} = \frac{q}{2G} \text{ if } \Delta V = V_A )</td>
</tr>
</tbody>
</table>

(The constant \( C \) is equal to \( \frac{16(1-v)}{3(2-v)} \) for a circular crack with Poisson’s ratio equal to \( v \)).
If the so-called "seismic stiffness modulus", $k_s$, is defined to be the seismic stress, $\sigma_s$, divided by the seismic strain, $\varepsilon_s$, then $k_s = 2G$ if $\Delta V = V_A$, the apparent volume. For an isolated crack, the seismic stiffness modulus therefore corresponds to twice the shear modulus of the host material. This interpretation may, however, be modified if the crack is strongly coupled to the response of surrounding fractures and depends also on how the volume parameter, $\Delta V$, is assigned.

The direct crack “stiffness” can be defined to be the stress drop, $q$, divided by the average slip, $D$. Using the equations in Table 2-1, this gives:

$$k_s = \frac{q}{D} = \frac{\pi G}{Ca}.$$  \hspace{1cm} [2.1]

This formula indicates that the “mechanical” stiffness of the crack is inversely proportional to the characteristic source dimension, $a$. According to this interpretation, $k_s$ would be directly related to the corner frequency $f_0$ estimated for a simple circular shear crack. These rather simple relationships are deduced from a purely static analysis and have to be assessed carefully in relation to the numerical simulation of seismic behaviour.

A further consideration in the integration of seismic behaviour with numerical modelling is whether explicit dynamic behaviour needs to be modelled. Such modelling is extremely complex in the analysis of large-scale rockmass behaviour. In particular, a number of uncertainties exist as to how wave propagation dispersion effects should be treated in a numerical procedure and the extent to which the rockmass can be represented as a simple elastic medium. These problems have been treated to some extent in SIMRAC project GAP 601b (Napier et al., 2002) in the development and application of the finite difference computer program WAVE. However, many problems relating to the assessment of rockmass stability can be treated without requiring explicit simulations of dynamic deformation processes.

### 2.2 Analogues for seismic behaviour

#### 2.2.1 Block-slider models

A number of evocative analogues to seismic behaviour have been studied in the past. These include the well-known “sand pile” model developed by Bak et al. (1988) to demonstrate the concept of “self-organised criticality” and block-slider models proposed by Burridge & Knopoff (1967) in which chaotic motions of spring-connected slider blocks are manifested. An important characteristic of the block-slider models is that the sliding resistance is assumed to decrease as the block velocity increases, leading to chaotic stick-slip motions. In the context of static simulations (where inertial effects are negligible), it is of interest to examine whether this behaviour can also be reproduced from the extremely simplified assumption of just two sliding resistance states, corresponding to “static” and “dynamic” friction coefficients. This model has been implemented in the DIGS code for evaluation. Figure 2.1 shows the stick-slip response for one particular block, indicating intermittent episodes of sliding as the upper surface of the block is uniformly displaced parallel to the lower sliding surface. The cumulative statistics of the energy increment distribution, together with the response for two other blocks, subjected to different loading conditions or having a different size, are plotted in Figure 2.2. The plot shows that the response signals are not simple power law functions, but follow complex, “chaotic” incremental movements that depend on the block shape and loading arrangement. It is interesting that the stick-slip response, revealed in Figure 2.1, can be generated from an extremely simple static/dynamic friction state switching rule without requiring the explicit inclusion of inertial effects. This has relevance to the simulation of more complex mining step sequences in regions of pre-existing discontinuities or mining-induced fracture initiation.
Figure 2.1  Plot of simulated energy release increments recorded for a 20 m x 20 m sliding block with two slip resistance states corresponding to friction angles of 35 degrees and 10 degrees respectively. The upper surface of the block is displaced horizontally at a constant rate and has a fixed, uniform vertical displacement of 0.032 m. The lower surface slides on an elastic base.

Figure 2.2  Cumulative energy release increment distributions for three different sliding blocks. The cumulative distributions of the energy release increments do not follow simple power law relationships but exhibit complex stick-slip movements.
2.2.2 Self-organising criticality

One of the key unifying concepts available for the general description of deep level mine seismic activity is the Gutenberg-Richter relationship with an apparently universal “b-value” that is \( b \approx 1 \). The concept of “self-organized criticality” (SOC) originating from the work of Bak et al. (1988) has been proposed as an explanation for this universal behaviour and has been studied intensively as a prototypical model of seismic behaviour. Turcotte (1999) pointed out that SOC models fall into three classes: (a) the “sand pile” model of Bak et al., (b) chaotic block-spring slider models and (c) “forest fire” models. Turcotte also proposed a cascade model to provide an explanation of the statistics of the three model types and illustrated the potential of constructing global hazard maps to assess systematic patterns of seismic activity. However, it is not clear how these three elementary model analogies can be used directly in the case of deep level mining. The closest analogy to the sand pile model is Spottiswoode’s face softening model (Spottiswoode et al., 2003). One fundamental issue that differentiates mining activity from these simple models is the loading process. In the case of deep mining, loading is activated by the incremental enlargement of excavations. The most hazardous region tracks the advancing mining faces as new discontinuities are created (“burst” fractures or extension cracks), and as pre-existing faults and joints are activated. More complex stress transfer processes occur in old worked out areas and across large-scale geological features. Consequently, it is difficult to create hazard maps that delineate regions of specific seismic activity.

2.3 Simplified deformation mechanisms and models

2.3.1 Damage in a bracket pillar region

It is of practical interest to determine whether complex failure processes can be replaced by simplified failure mechanisms in certain cases. One such case is the interaction between a stope and a fault that intersects the stope horizon. A specific analysis was carried out in which a vertical fault was located 10 m ahead of the face of a parallel-sided longwall panel having a span of 400 m. Two methods of representing the fracture zone ahead of the stope face were considered. In the first case a simplified failure mechanism was assumed to comprise a single “burst” fault running from the stope face to a point 30 m above the reef plane. In the second case, it was assumed that a more complex damage zone was present in a 15 m by 30 m region adjacent to the stope-fault intersection point, as shown in Figure 2.1. The damage region was assumed to be covered by a random mesh of discontinuity elements (Delaunay tessellations) within which fracture growth was initiated in sequential steps. Two different sets of constitutive properties were assigned to the fault and to the random mesh discontinuities. In the first set, the cohesion on each discontinuity was assumed to be zero and the friction angle was assumed to be 30 degrees. In the second set, the cohesion on each discontinuity was assumed to be 25 MPa and the initial friction angle, prior to failure, was assumed to be 40 degrees. The residual friction angle was set at 10 degrees.

The fault slip profile for each case is shown in Figure 2.2. It is interesting to note that the simplified slip mechanism, with the single discontinuity connecting the fault to the stope, agrees remarkably well with the explicit random mesh damage model for both sets of constitutive properties. It appears that the low residual friction angle of the second set (10 degrees) controls the ultimate extent of slip on the fault for both the simplified and the detailed failure zone simulation. It is also of significance that the effective “day lighting” of the fault into the stope can be accommodated over a relatively extended region. These results are also important in considering the performance, and potential failure, of bracket pillars. Figure 2.3 compares the energy release increments that arise in the sequential activation of elements of the random mesh region for each set of constitutive properties during the progressive formation of the failure zone in the bracket pillar region. This comparison clearly shows a different energy release pattern (large energy release increments) for the case in which both cohesion loss and friction weakening occurs.
Figure 2.1  Mining up to a fault indicating a simplified failure mechanism near the stope-fault intersection point.

Figure 2.2  Slip profile on a fault intersecting a stope, examining the effect of a simplified “day lighting” damage mechanism.
2.3.2 Simulation of fracture zone damage using non-intersecting fracture clusters

The “forward” numerical simulation of anticipated seismic activity in three-dimensional mine layouts remains a significant challenge. Currently, the most practical approach to this problem is the use of on-reef “softening” logic in the form of the “cap stress” model developed by Spottiswoode (Spottiswoode et al., 2003). The development of three-dimensional damage simulation capabilities is a direct aim of the present work. However, it is not clear whether random tessellation mesh schemes, developed in two dimensions, can be extended to three-dimensional procedures. In particular, triangular displacement discontinuity elements defined by space-filling tetrahedral tessellations can intuitively be expected to cause deformation “lockup” and lead to diffuse regions of damage if they are used to model large-scale fault slip processes. A question worth further investigation is whether 3D Voronoi polyhedra can be used as an alternative tessellation structure. However, an alternative procedure is investigated first. This approach is based on the proposition that off-reef stope damage can be simulated effectively by means of multiple non-intersecting fracture “patches”. This scheme requires that each fracture patch has a relatively smooth curvilinear surface topography. The crack slip profile on the surface is assumed to be represented by a low-order variation of the slip or by an opening displacement discontinuity distribution across the crack patch. The overall damage in the region of interest is assumed to be simulated by using a suitable assembly of clusters of these non-intersecting patches. A potential advantage of this approach is that each patch can, in principle, be incrementally extended in such a manner as to avoid violating local “oversize” element specifications in relation to the Uenishi-Rice critical slip limit, discussed in Section 2.5.

It is recognised that a number of severe disadvantages may be associated with the proposed scheme. The most obvious criticism is whether a collection of disconnected crack segments is capable of representing the equivalent slip and energy release behaviour of an equivalent
macro fault feature. This problem can be partially investigated by considering some simplified overlapping crack geometries. Consider first the case of two overlapping, planar fault segments separated by a distance of 1 m. Figure 2.1 shows the computed energy release corresponding to different extents of fault overlap. A similar trend arises for the equivalent “volume” of fault slip (or moment) indicating that the two overlapped segments become essentially “equivalent” to the single segment when the overlap exceeds 50 per cent of the overall crack length. A more demanding test is the case of several rows of collinear crack/bridge segments. It can again be shown, by numerical experimentation, that if the crack to bridge length ratio exceeds 4 and if the separation distance is less than one per cent of the crack length, the moment and energy release of the equivalent system reaches at least 95 per cent of the values for a single sliding crack/fault. A yet more demanding test is to consider the full-scale damage evolution in the vicinity of a mining face. The development numerical codes to carry this out in 3D are discussed in Chapter 4. Some preliminary assessments can be made in the case of 2D plane strain configurations. Consider, first, the case of a single stope panel of length 55 m extending to the centre of a 10 m x 10 m region in which random seed points are located. The seed points are located at the mid points of each edge of a random Delaunay triangle mesh. (The same mesh is used in a further comparison described later). Additional seed points are located on the surface of the tabular opening. Fractures are allowed to extend from each seed point but intersection of the growing fractures is not permitted. (This analysis is carried out using the original crack growth version of the DIGS computer code). Figure 2.2 illustrates the cumulative growth pattern that is developed following the initial, sudden, introduction of the 55 m stope and a second mining step in which the stope is advanced by 1 m. Fracture growth is permitted to occur according to a “shear” growth rule that positions growth elements in the local direction of maximum excess shear stress. The resulting fracture pattern, when both shear and tension growth modes are permitted is shown in Figure 2.3. As would be expected, a greater number of growth sites are activated in Figure 2.3 than in Figure 2.2. The fracture growth simulation was repeated using the original Delaunay triangle mesh with incremental selection and activation of elements from the mesh. The resulting fracture pattern is shown in Figure 2.4 which has a broadly similar pattern to that of Figure 2.2 and Figure 2.3, but with a number of differences in detailed features. The differences in energy release values corresponding to Figure 2.2 to Figure 2.4 are summarised in Table 2-1. Step 1 refers to the fracture growth arising after the initial 55 m panel is equilibrated, and Step 2 refers to the fracture growth increments corresponding to a face advance step of 1 m.

Table 2-1 Energy release increments (MJ/m) for different fracture growth rules

<table>
<thead>
<tr>
<th>Step</th>
<th>Figure 2.2</th>
<th>Figure 2.3</th>
<th>Figure 2.4</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>2.28</td>
<td>1.83</td>
<td>6.57</td>
</tr>
<tr>
<td>2</td>
<td>5.09</td>
<td>4.70</td>
<td>3.60</td>
</tr>
</tbody>
</table>

The energy release increments reported in Table 2-1 indicate that in the second 1 m mining step there is some variability in the energy release (as would be expected from a random system) but there is no indication that the energy release increments corresponding to the non-intersecting growth simulations are less than the increments arising in the random mesh simulation of Figure 2.4. The energy release increment is in fact lower in this case.

It appears that the non-intersecting growth model proposed here holds some promise for large-scale off-reef damage simulation. The differences highlighted, particularly in Figure 2.2 and Figure 2.3, can be ascribed mainly to the “shear only” growth mode rule applied in the case of the non-intersecting fracture model. The proposed scheme to perform explicit off-reef growth simulations, using clusters of circular-shaped discontinuity elements near the edges of 3D tabular mine layouts, is described in Chapter 4.
Figure 2.1  Effect of fault overlap on energy release, compared to a single fault of 100 m length.

Figure 2.2  Fracture growth pattern after a single mining step of 1 m with only shear growth seeds and non-intersecting cracks.
Figure 2.3  Fracture growth pattern after a single mining step of 1 m with combined shear and tension growth seeds and non-intersecting cracks.

Figure 2.4  Fracture growth pattern after a single mining step of 1 m with combined shear and tension fracture initiated within a random Delaunay mesh.
2.4 Assessment of rockmass stability

2.4.1 Classical stability analysis

The stability properties of dynamical systems, described by sets of ordinary differential equations, can be analysed using a number of well-known methods (e.g. Jordan & Smith, 1999). In the case of differential equation systems with constant coefficients associated with the dependent variables, it is sufficient to determine the eigenvalues of the system matrix in order to determine the asymptotic stability of the system. If the eigenvalues are unique and are all negative, the system is asymptotically stable. Non-unique eigenvalues and more complex non-linear equation structures demand correspondingly more sophisticated techniques to analyse the system stability. Although it may be possible in principle to apply these methods to problems of non-linear rock failure simulated by finite element plasticity models, the number of degrees of freedom in such models will be extremely large. Consequently, it is of great importance to investigate whether alternative approaches can be used to assess system behaviour during the course of a simulation exercise. It is also important to define how “stability” in the context of seismic damage simulation should be understood. In the present study it is proposed that the rockmass is deformed in a number of evolutionary equilibrium configurations induced by mining activity, and that the most important requirement is to measure the “magnitude” of the transition between each successive state in some manner. It is suggested that this can be accomplished by determining a so-called “generalised energy release (GER)” increment corresponding to each equilibrium state. A stability analysis is then understood to imply the assessment of the magnitudes, patterns and statistics of the energy transition increments that arise in any proposed mining sequence. This approach provides a direct measure of the performance of a given or proposed mining sequence, but requires an explicit simulation of the representative damage processes that occur as mining proceeds. The direct damage simulation may be computationally demanding.

2.4.2 Stiffness and energy release as measures of system stability

The concept of loading system stiffness (LSS) has been proposed as a parameter to evaluate rockburst potential (Wiles, 2002) in preference to the application of a criterion referred to as the local energy release rate (LERR). This proposal is motivated by the concern that the LERR is proportional to the square of the mean stress and that computed values of LERR scale inappropriately when attempting to assess rockburst potential. However, this concern appears to be unfounded. A detailed framework for the calculation of energy changes in a fractured rockmass has been documented previously (Napier, 1991). The definition of LERR (given by Wiles, 2002) is very similar to the quantity termed “available” energy discussed by Napier and is also related to energy quantities previously derived by Salamon (1984). The “available” energy (Napier, 1991) can be used to compute the energy release in any incremental change in the closure and off-reef fracture processes corresponding to a mining step increment. The net “energy release” increment, \( \Delta ER \), is computed to be the difference between the “available” energy change, \( \Delta W_d \), and the energy dissipated, \( \Delta W_D \), by processes such as frictional sliding, fracture and backfill compression. It is suggested that \( \Delta ER \) is, in fact, a more appropriate measure of damage potential than the local stiffness measure (LSS). Consider, for example, potential damage near the edge of a large, square excavation caused by slip on any one of a series of slip planes located at increasing distances from the edge of the excavation as shown in Figure 2.1. The “available” energy, \( \Delta W_d \) (corresponding to LERR), dissipated energy, \( \Delta W_D \), and net released energy, \( \Delta ER \), are plotted for each slip plane position in Figure 2.2. This shows clearly that \( \Delta ER \) decreases away from the edge of the excavation indicating a decrease in damage potential as intuitively anticipated. This trend is opposite to the conclusion reached by Wiles for the analysis of LERR in the proximity of a large excavation. Figure 2.3 depicts the corresponding change in the loading system stiffness, LSS, computed for each sliding discontinuity by dividing the average shear stress drop by the average slip. This does show an increase in stiffness of approximately 12 per cent, which is inferred to indicate a decrease in
damage potential (Wiles, 2002). However, it appears that the computation of net released energy does, in fact, provide a useful indication of damage and stability/instability transitions, provided the actual damage mechanism is explicitly represented in the computation. The requirement to include the explicit damage process applies equally to the application of concepts such as LSS. Further, the computation of LSS seems, in general, to require an appropriate analysis of full system component interactions, as furnished for example by the computation of a global stiffness matrix in the analysis of pillar layout stability. It also requires a possibly subjective choice of the exact region boundaries where the stiffness computation is to be performed.

Figure 2.1 A square excavation with potential adjacent slip planes at different distances from the edge of the excavation.

Figure 2.2 Energy release quantities computed from deformations occurring on each of a set of fractures located at increasing distances from the edge of a 40 m x 40 m square excavation.
Figure 2.3 Variation of shear slip stiffness computed from deformations occurring on each of a set of fractures located at increasing distances from the edge of a 40 m x 40 m square excavation.

2.4.3 Statistics of energy release increments in mining a parallel-sided panel

The use of net released energy as a measure of damage and stability/instability transitions in tabular mining problems is also compatible with the overall frequency statistics established for seismic event magnitudes. Re-plotting data from previous studies of incremental mining through a random mesh of potential crack positions, shows that energy release increments do, in fact, display a power law distribution, as shown in Figure 2.1. In this case, the cumulative frequency slope of the logarithms of energy release increments is less than unity. However, the exact implications of carrying out the analysis in a two-dimensional cross section, compared to full three-dimensional damage simulation, requires further investigation.

Figure 2.1 Energy release increment size distribution in simulated mining of a single parallel-sided panel.
2.4.4 Incremental extraction of three longwall mining panels

An indication of the utility of using GER as an assessment of extraction stability can be obtained by considering the global behaviour of seismic activity in the mining of a sequence of three longwall panels. In this case, a central panel is mined initially with a final span of 140 m. This corresponds nominally to the East Driefontein 5 Shaft layout shown in Figure 2.1. Two further panels with spans of 140 m are mined adjacent to the first panel leaving 40 m pillars between each panel as indicated schematically in Figure 2.2. A coarse-scale random mesh is used to simulate the progressive development of failure in the vicinity of each longwall panel. Mining is carried out symmetrically from the centre of each longwall panel with the overall span increasing by 20 m in each step (10 m on each side). Figure 2.3 and Figure 2.4 show the overall accelerated change in the fracture extent when the right hand longwall span is increased from 80 m to 100 m. Similarly, Figure 2.5 and Figure 2.6 show the overall extensive increase in the number of mobilised fractures when the left hand longwall span is increased from 100 m to 120 m. It can be seen that there is a significant “coalescence” of fracturing in the pillar region to the right of the central longwall panel and in the pillar region to the left of the central longwall panel during these two incremental span increases. Figure 2.7 shows the final fracture pattern after mining has been completed with each panel span being extracted to 140 m.

![Longwall mining layout - Reef coordinates](image)

*Figure 2.1 Detailed extraction sequence of three longwall panels.*
Figure 2.2  Simulated mining sequence of three longwall panels with final spans of 140 m and final pillar sizes of 40 m.

Figure 2.3  Cumulative fracture damage zone developed after four extraction steps of the right hand longwall panel.
Figure 2.4  Cumulative fracture damage zone developed after five extraction steps of right hand longwall panel. Extensive formation of additional fracturing is observed to occur in the right hand pillar region.

Figure 2.5  Cumulative fracture damage zone developed after five extraction steps of the left hand longwall panel.
Figure 2.6  Cumulative fracture damage zone developed after six extraction steps of the left hand longwall panel. Extensive formation of additional fracturing is observed to occur in the left hand pillar region as well as some additional fracturing in the “remote” region of the previously mined right hand panel.

Figure 2.7  Cumulative (final) fracture damage zone developed after seven extraction steps of the left hand longwall panel.
Figure 2.8 is a plot of the incremental changes in the mobilised fracture length as a function of the cumulative mined span. This quantity cannot be inferred easily from monitored seismic data. Moment changes are routinely inferred from observations of seismic activity. Figure 2.9 shows a plot of the cumulative moment as a function of the cumulative span. In this case, significant increases in moment are observed prior to cumulative spans of 240 m and 400 m. This is emphasised in Figure 2.10 where the incremental moment changes are plotted against the cumulative mined span. Large local peak values are observed to arise at spans of 240 m and 400 m corresponding to the accelerated fracture coalescence behaviour that is illustrated in Figure 2.3 and 2.17, and in Figure 2.5 and 2.19, respectively. Qualitatively similar increases were noted in the real moment changes observed during actual mining operations. It is also seen in Figure 2.11 that the incremental changes in the energy release values are strongly peaked at the critical cumulative mined spans of 240 m and 400 m.

It is of some interest to plot the inferred “apparent volume” and the “seismic stiffness modulus” (SSM) increments as a function of the cumulative mined span. The apparent volume plot is shown in Figure 2.12. This exhibits a peak value at the critical span of 240 m but, interestingly, continues to increase after the cumulative span exceeds the critical value of 400 m. For uniform stress drop values, the apparent volume can be expected to be roughly proportional to the seismic moment but this is not always the case as can be seen by comparing Figure 2.12 and Figure 2.10. Figure 2.13 is a plot of the simulated “seismic stiffness modulus” (SSM) as a function of cumulative span. The SSM can be shown to be equal to \( 2G \frac{V}{V_{m}} \) where \( G \) is the material shear modulus, \( V_{m} \) is the “apparent volume” and \( V \) is a selected representative volume size in the region of interest. Since \( G \) and \( V \) are constant, the SSM plotted in Figure 2.13 is effectively the inverse of the apparent volume values plotted in Figure 2.12. In particular in the last mining step, when the final span is increased to 420 m, the stiffness shown in Figure 2.13 remains very low. However, the plot of the incremental energy release, shown in Figure 2.11, highlights very clearly the two main episodes of “accelerated” fracture coalescence and expected increase in overall seismic activity. Importantly, the final span increase from 400 m to 420 m is not associated with high seismic activity. Consequently, in the present example, low values of the seismic stiffness modulus are not necessarily associated with high levels of seismic activity.

![Figure 2.8 Incremental fracture length mobilised at each stage of mining.](image)
**Figure 2.9** Cumulative moment plotted against cumulative span mined showing intermittent accelerations in the slip movement on mobilised fractures.

**Figure 2.10** Incremental moment changes associated with cumulative mining span. The peak values associated with significant accelerations in fracture growth are highlighted.
Figure 2.11 Incremental energy release changes associated with cumulative mining span. The peak values associated with significant accelerations in fracture growth are highlighted.

Figure 2.12 Apparent volume as a function of cumulative mined span.
2.4.5 Cap stress model

A “cap stress” model (Spottiswoode et al., 2003) provides a simplified method of simulating damage processes in three-dimensional tabular layout problems. It is of interest to assess the extent to which the cap stress concept may be used to simulate actual off-reef damage processes. Some insight into this question can be obtained by examining the average pillar stress levels that arise in the mining of the three successive longwall panels, described in Section 2.4.4, where off-reef damage is simulated explicitly.

Figure 2.1 shows the average pillar stress that arises in the positions where the final pillars will be located as the raises are mined in the indicated sequence. Each raise is mined symmetrically from the centre, in increments of 10 m at each side of the span. The total mining step span increment is 20 m. Figure 2.2 shows the corresponding estimated pillar stress if no off-reef failure is permitted to occur. Figure 2.3 shows the energy release increments that arise during each 20 m extraction increment, with and without off-reef damage being permitted. It is clear from Figure 2.1 and Figure 2.2 that the average stress in the final pillar positions is, in fact, lower than the stress that arises if no off-reef damage occurs. Accordingly, it is appropriate to “cap” the peak stress that can occur adjacent to mined panels as embodied in the simplified model. In addition, the incremental level of seismic activity, measured by the energy release increments, is reduced when no off-reef damage is allowed to occur. This will to some extent be compensated for if cap stresses are introduced which will tend to increase closure values in excavation regions.

Figure 2.13 Seismic stiffness modulus as a function of cumulative mined span.
Figure 2.1  Average stress in two pillar regions between three longwall panels.

Figure 2.2  Average stress in pillar regions when no off-reef failure is permitted
Figure 2.3 Energy release increments.

2.5 Grid size effects, slip-weakening and fracture nucleation zone size

2.5.1 Grid size effects

An important issue in the assessment of rockmass instability is to understand how the choice of the numerical model mesh size affects simulated seismic behaviour. An assessment of the effect of the random mesh density was carried out by repeating the longwall mining simulation described in Section 2.4.4 using a finer tessellation grid. This was achieved by constructing a separate random mesh using a Voronoi tessellation with an internal triangulation structure. The overall mesh “length” covering a sectional area of 7600 m², perpendicular to the longwall panel axes, was increased from 22502 m to 39475 m, a factor of 1.75. The resulting cumulative mobilised fracture length, cumulative moment and cumulative energy release are plotted as a function of the cumulative span mined in Figure 2.1 to Figure 2.3, respectively. These results show that the cumulative mobilised fracture length and moment are relatively insensitive to the mesh density but that the cumulative energy release is slightly increased as the mesh density is increased. This result demonstrates that the simulation exercise carried out in Section 2.4.4 is relatively insensitive to the random mesh density. This can be ascribed partly to the inclusion of slip-weakening logic in the DIGS code.
Figure 2.1  Effect of tessellation density on simulation of mobilised fracture length in sequential panel mining simulation.

Figure 2.2  Effect of tessellation density on cumulative moment in sequential panel mining simulation.
2.5.2 Slip-weakening and fracture nucleation

Recent work carried out by Uenishi & Rice (2003) has demonstrated the existence of a universal nucleation length when a linear slip-weakening law is used. The stability analysis of Uenishi & Rice shows that the nucleation length, $h_n$, is given by the following relationship:

$$h_n = 1.158 \frac{G}{W}$$  \hspace{1cm} [2.2]

where $G$ is the material shear modulus and $W$ is the slip-weakening parameter. In the simulations described in Section 2.5.1 (shown in Figure 2.1 to Figure 2.3), $G = 29.167$ GPa and $W = 10,000$ MPa/m, giving a nucleation distance of 3.38 m. The element sizes used in the Delaunay and Voronoi internal meshes are 6.0 m and 7.25 m, respectively. The element sizes are therefore coarser than the minimum size that would be able to resolve the indicated nucleation distance. To evaluate the effect of element scale relative to the nucleation length in a random tessellation mesh, some further results are shown in Figure 2.1 to Figure 2.4 for the activation of a single fault disturbed by a locally peaked driving shear stress. The weakening parameter, $W'$, is chosen to be 5,000 MPa/m in this case giving a nucleation length of 6.8 m. Figure 2.1 shows the equilibrium slip profiles on the fault when the driving shear stress is increased from 10 MPa to 13.2 MPa in increments of 0.2 MPa. The cohesive strength is 5 MPa and the friction angle is set to 45 degrees. The element size is set initially to 0.2 m.

It can be seen from Figure 2.1 that the activated region (non-zero slip region) on the fault "jumps" discontinuously from a length of 6.8 m to 12 m when the applied load is increased from 10.8 MPa to 11.0 MPa. This corresponds, within the resolution of the chosen element size, to the theoretical nucleation size predicted by the stability analysis of Uenishi & Rice. Effectively, the loading system stiffness decreases as the mobilised slip length increases. At the critical mobilised length of 6.8 m the loading system stiffness matches the slope, $W'$, of the slip weakening parameter exactly and the slip becomes unstable. Figure 2.2 shows the
corresponding profiles of residual cohesion, indicating very clearly the abrupt loss of cohesion as the applied peak load is increased from 10.8 MPa to 11.0 MPa (between load increment 5 and load increment 6).

These simulations were repeated with the fault element size set to 5.0 m and are plotted in Figure 2.3 and Figure 2.4, respectively. (Element collocation points are explicitly highlighted as “diamond” symbols in these plots). It is apparent that the coarse element simulations mask the precise transition from stability to instability and also yield further clustered instabilities as the fault load is increased. The effect of over-sized elements has been pointed out previously by Rice and is obviously important in the integration and simulation of synthetic seismic behaviour. It is also important to note that it is still unclear whether the slip-weakening parameter $W$ is a basic material property of the rock and whether this parameter scales with the size of the fault system under consideration. The value that has been used for $W$ in the studies shown in Figure 2.1 to Figure 2.3 is 10 000 MPa/m. Laboratory scale estimates of $W$ indicate that the value may be of the order of 200 000 MPa/m, yielding an instability nucleation length of 0.15 m if the shear modulus is of the order of 30 000 MPa.

**Figure 2.1**  *Slip profiles on an incrementally loaded fault (element size = 0.2 m) showing loss of stability at a critical nucleation length of 6.8 m.*
Figure 2.2  Residual cohesion on incrementally loaded fault (element size = 0.2 m) showing loss of stability at a critical nucleation length of 6.8 m.

Figure 2.3  Slip profiles on incrementally loaded fault (element size = 5.0 m).
In view of these results, it is of interest to determine the sensitivity to element size and slip weakening parameters in the simulation of the three longwall panels discussed in Section 2.4.4. Various parameter combinations were investigated as summarised in Table 2.3. The ratio of the critical nucleation length to the average grid size is shown in the last column of Table 2.3. A ratio that is less than one indicates oversized element lengths relative to the critical nucleation length. The cumulative energy release is plotted as a function of the mining step number in Figure 2.137. Each mining step corresponds to an increase in the cumulative mined span of 20 m. The values plotted in Figure 2.5 indicate large differences in cumulative energy corresponding to the various combinations of modulus, weakening parameter and average element length. Unfortunately, computational constraints prevented the case with small element size (the last row of Table 2.3) being run for the full sequence of mining steps. Figure 2.6 shows a plot of the cumulative energy release against the cumulative scaled moment (moment divided by twice the shear modulus). This gives an indication of the intrinsic stress drop associated with the slip events in each simulation. It is interesting to note that the stress drop slope is very similar for three of the parameter combinations but is significantly different in the case where the modulus is artificially large (row two of Table 2.3). In this case, very little off-reef fracturing developed during the mining simulation. Consequently, it is apparent that different combinations of the weakening parameter and the basic element size can affect the absolute levels of cumulative energy release. However, this does not appear to affect the relative energy release accelerations associated with fracture coalescence episodes and, also, does not affect intrinsic stress drop characteristics unless unnatural rock modulus values are introduced.

### Table 2.1 Comparative simulations varying the element size and effective slip nucleation length.

<table>
<thead>
<tr>
<th>Model</th>
<th>Young’s Modulus (GPa)</th>
<th>Slip-weakening parameter, W (MPa/m)</th>
<th>Nucleation Length, ( h_n ) (m)</th>
<th>Avg. (max) grid size (m)</th>
<th>( h_n : ) Avg. grid size</th>
</tr>
</thead>
<tbody>
<tr>
<td>Base case</td>
<td>70</td>
<td>10 000</td>
<td>3.38</td>
<td>7.25 (15.73)</td>
<td>0.47</td>
</tr>
<tr>
<td>High modulus</td>
<td>1 400</td>
<td>10 000</td>
<td>67.6</td>
<td>7.25 (15.73)</td>
<td>9.3</td>
</tr>
<tr>
<td>Low slip-weakening</td>
<td>70</td>
<td>1 000</td>
<td>3.38</td>
<td>7.25 (15.73)</td>
<td>4.76</td>
</tr>
<tr>
<td>Small elements (limited run)</td>
<td>70</td>
<td>10 000</td>
<td>3.38</td>
<td>1.25 (2.0)</td>
<td>1.9</td>
</tr>
</tbody>
</table>
Figure 2.5 Cumulative energy release in three-longwall mining problem for various choices of the effective critical slip nucleation length.

Figure 2.6 Cumulative energy release plotted against cumulative scaled moment to indicate the effective stress drop in the three-longwall mining problem for various choices of the effective critical slip nucleation length.
2.5.3 Slip-weakening on multiple, overlapping fault structures or random mesh assemblies

The studies described in Section 2.5.2 have addressed the question of the stability transition of a single fault with an increasing shear load applied over a limited area of the fault. Theoretical analysis of this case, carried out by Uenishi & Rice (2003), indicates that the critical nucleation distance, \( h \), for the onset of unstable slip behaviour is proportional to the ratio \( G/W \), where \( G \) is the material shear modulus and \( W \) is the slope of the (assumed) linear relationship between cohesion and slip. The Uenishi-Rice analysis considered only a single fault plane. Clearly, it is of interest to examine the response of both multiple fault planes and random crack assemblies. A special version of the DIGS program has been used to facilitate the specification of general primitive stress variations in the form

\[
P_{ij} = c_0 + c_1 y + c_2 z + c_3 y^2 + c_4 yz + c_5 z^2,
\]  

[2.3]

where \((y, z)\) is a general field point and \( P_{ij} \) is a general component of the primitive stress tensor. A number of simulations were performed on flat, parallel faults, centred on \( y = 0 \), with a series of incremental applied shear load steps. The normal stress component was varied as \( c_0 + c_3 y^2 \) with the minimum "clamping" effect at \( y = 0 \). In these cases, the parameters \( G \) and \( W \) were set to 30 GPa and 5 000 MPa/m respectively. The maximum cohesion and friction angle on each fault was set to 25 MPa and 45 degrees, respectively. The runs, together with the chosen spacing between the parallel fault planes, are summarised in Table 2-1.

<table>
<thead>
<tr>
<th>Run</th>
<th>Number of Planes</th>
<th>Spacing</th>
<th>Number mobilised</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>1</td>
<td>0 m / 1 m</td>
<td>1</td>
</tr>
<tr>
<td>2</td>
<td>2</td>
<td>0 m / 2 m</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>2</td>
<td>0 m / 4 m</td>
<td>2</td>
</tr>
<tr>
<td>4</td>
<td>2</td>
<td>0 m / 2 m / 4 m</td>
<td>2</td>
</tr>
<tr>
<td>5</td>
<td>3</td>
<td>0 m / 2 m / 4 m</td>
<td>2</td>
</tr>
</tbody>
</table>

Table 2-1 Summary of parallel fault configuration analyses.

Figure 2.1 shows the final slip profile for the case of Run 5 in which three parallel planes are spaced 2 m apart. The bracketed numbers in Figure 2.1 indicate the relative spacing position of each plane. Figure 2.2 shows the corresponding values of the residual cohesion on each plane. The main, perhaps surprising, point of interest is that in this case only two of the three planes are mobilised following the initial unstable transition when cohesion is rapidly lost. The central plane is not in fact mobilised, as can clearly be seen in Figure 2.1 and Figure 2.2 by examining curve (2). It is also of interest to note the asymmetrical slip profile of the outermost planes (0 and 4) in Figure 2.1. Figure 2.3 presents a summary of the cumulative energy release for each of the runs in Table 2-1. This diagram shows very clearly the "step" transition from the initial stable state to the subsequent equilibrated state in each case. The magnitude of the energy step at the onset of instability is a subtle function of the spacing of the planes. In addition, it appears that the number of planes that are mobilised is affected by the inter-plane spacing, S. It may be possible to relate the number of planes mobilised to both \( S \) and \( G/W \) or to the dimensionless quantity, \( SW/G \).

It is also unclear how the onset of instability is affected if multiple random crack positions are loaded. This case is examined by considering a random mesh tessellation of approximately
4000 crack elements with an average length of 0.6 m, distributed over a rectangular region 50 m by 15 m in extent. A variable compressive load is imposed along the major axis of this region with a peak load at the central position. A constant compressive load of 10 MPa is imposed in the orthogonal direction. The major peak load was increased in a series of five steps from 200 MPa (just above the theoretical Coulomb strength of 180 MPa at a confining load of 10 MPa) to 240 MPa. Figure 2.4 and Figure 2.5 show the mobilised fracture pattern at loads of 220 MPa and 230 MPa, respectively. The cumulative energy release and the cumulative slip moment are plotted as a function of the loading steps in Figure 2.6. It can be seen that the stability transition occurs between steps 3 and 4, which is reflected by the relatively extensive increase in the number of mobilised fractures shown in Figure 2.4 and Figure 2.5, respectively. This transition is most strongly identified by the energy release plot shown in Figure 2.6. Figure 2.7 is a plot of the incremental energy release values against the incremental slip moment values, scaled by the shear modulus. This gives an “effective” stress drop that appears to be of the order of 7 MPa.

The simulations carried out here indicate that the simple nucleation length analysis of Uenishi & Rice needs to be enhanced to treat multiple fault plane or random fracture assemblies. Nevertheless, it appears that tracing the energy release history of incremental load steps can be used as a robust measure to identify characteristic stability/instability transitions. This reinforces the potential utility of general energy release (GER) as a criterion to assess stability transitions in the simulation of rockmass damage. Tests on multiple 3D cluster assemblies are described in Chapter 4 to test the feasibility of carrying out large-scale explicit fracture simulations in 3D and to identify whether on-reef damage simulation, computed using the MINF code, can be used as an effective simplification of explicit off-reef damage simulation.

![Figure 2.1](image-url)  
**Figure 2.1** Final slip profiles computed on three parallel fault planes.
Figure 2.2  Residual cohesion on three parallel fault planes.

Figure 2.3  Energy release for various horizontal fault planes subjected to incremental shear loads.
Figure 2.4  Activated cracks at peak horizontal load of 220 MPa (Step 3).

Figure 2.5  Activated crack pattern at peak horizontal load of 230 MPa (Step 4).
Figure 2.6  Cumulative energy release and moment profiles for incremental loading of random crack assembly (average element length = 0.6 m).

Figure 2.7  Energy release increments plotted against scaled moment increments for random crack assembly loading.
2.6 Stope closure and ride as an indication of rockmass stability

A further objective of the project SIM 020301 is to assess the feasibility of integrating closure observations with numerical modelling for the evaluation of stope stability. Malan et al. (2003) have obtained limited data on stope closure behaviour during seismic events (SIMRAC project GAP 852). This data shows qualitatively that the stope closure measured at a number of stations parallel to the stope face, increases rapidly following a seismic event, with smaller continuous increases over a period of several hours or longer. In addition, it has been observed that the closure increase may be largest near the location of the event and may be accompanied by falls of ground in this region.

Investigations into mechanisms to explain this behaviour have been initiated using the 3DIGS computer code in which bi-quadratic variation square elements are used to facilitate the computation of surface-parallel stress components. Figure 2.1 shows a simplified panel layout in which a seismic event (rock burst) is simulated by allowing slip to occur on a fault rupture plane parallel to and intersecting the stope face of the central panel in the layout. The fault plane is at right angles to the reef plane and, in the particular case studied, was assumed to be located in the footwall. A section view is shown in Figure 2.2. Figure 2.3 shows the profile of stope closure before and after the event along a line running parallel to the central panel face and 7 m behind the face. Two post-event closure profiles are shown, corresponding to assumed fault friction angles of 30 degrees and zero degrees respectively. Figure 2.4 is a plot of the differential closure before and after the event, and Figure 2.5 is a corresponding plot of the differential ride component in the strike direction (the y-axis direction in Figure 2.1 and Figure 2.2). Figure 2.4 and Figure 2.5 show clearly that the peak value of the differential closure and ride, occur at a dip position that is opposite the epicentre of the event source. It can be proposed, therefore, that measured closure profiles may provide a useful indirect indication of event hypocentres near the stope face. More importantly, the relative sign of the strike ride component (or tilt direction of the closure meter) should provide an indication of whether the event has occurred in the hangingwall or in the footwall near the stope face. In the case shown in Figure 2.5, the negative sign of the strike ride component (the y-axis component in Figure 2.1) indicates that the event occurred in the footwall. A positive strike ride increment would indicate, conversely, that the event occurred in the hangingwall region ahead of the stope face. These observations motivate the utility of including ride (directional tilt) measurements to supplement current instruments that are used to monitor continuous closure movements.

A further observation of great importance is to note the induced change in the surface-parallel stress in the excavation following the simulated seismic event. Figure 2.6 shows the induced dip stress component (the $\tau_{xx}$ component relative to the axis system shown in Figure 2.1) plotted along a strike line intersecting the centre of the middle panel in Figure 2.1. This plot shows a dramatic increase in the surface-parallel induced tensile stress in the dip (x-axis) direction in the footwall ("-" side) of the stope and a minor change in the surface-parallel induced tension in the hangingwall ("+" side). Conversely, if the event were to have occurred in the hangingwall region, strong tensile stresses would be induced in the excavation hangingwall, with corresponding destabilisation. It should be noted that support equipment that is designed to give a preferential strike-oriented coverage may be rendered ineffective in controlling ground falls induced by dip-oriented tensile stresses. Also, the induction of strong tensile stresses parallel to the stope face in the footwall, may provide a mechanism to explain anecdotal observations of prop and stick-support "punching" into apparently "solid" ground sometimes observed after rock burst events. Figure 2.7 shows that the induced stress in the strike direction is generally compressive in the open stope region in both the hangingwall and the footwall.

In conclusion, an entirely new potential destabilisation mechanism has been identified. This mechanism can be quantitatively and qualitatively tested using measured face-parallel closure profiles and observations of ground fall positions. These results need to be supported by further field observations, including the development of directional tilt measuring devices to supplement existing closure meter equipment.
Figure 2.1  Schematic mining layout with footwall event occurring parallel to the face of the centre panel.

Figure 2.2  Schematic mining layout with footwall event occurring parallel to the face of the centre panel – representative cross section.
Figure 2.3  Dip closure profiles parallel to the stope face before and after a face-parallel event in the stope footwall. Two post-event profiles are shown corresponding to assumed fault sliding friction angles of 30 degrees and zero degrees respectively.

Figure 2.4  Incremental closure profiles plotted along a line parallel to and 7 m from the central stope face.
Figure 2.5  Incremental strike ride component plotted along a line parallel to and 7 m from the central stope face.

Figure 2.6  Skin stress component parallel to the stope face ($\tau_{xx}$) plotted along a strike line, showing strong induced tensile stress in the footwall, corresponding to the footwall face-parallel face “event”.
Figure 2.7  

Strike oriented skin stress plotted along strike showing induced compressive stresses in both the hanging-wall and the footwall.

2.7 Sub-critical crack growth simulation

Time-dependent tabular stope closure and fault slip has been investigated in previous studies using the DIGS fracture simulation program. It has been demonstrated in this previous work that observed time-dependent behaviour could be simulated by employing a simple slip “relaxation” rule applied to active discontinuities or faults. This slip rule postulates that the rate of slip is proportional to the difference between the shear stress, $\tau$, acting on the discontinuity and the shear resistance, $\rho$. Specifically,

$$\frac{\partial D_s}{\partial t} = \kappa (\tau - \rho) \quad [2.4]$$

where $D_s$ is the extent of slip, $t$ is the time and $\kappa$ is a proportionality constant (“fluidity”).

The shear resistance, $\rho$, is assumed to be the sum of a friction resistance term and a cohesion term. The cohesion is allowed also to be a function of the extent of slip (slip-weakening behaviour). A somewhat unsatisfactory aspect of this model is the treatment of the transition from “creep like” behaviour to rapid, unstable slip. This transitional state is obviously linked closely to the replication of intermittent stability “jumps” and corresponding seismic energy release effects.

In order to address the transition behaviour, it is postulated that Equation [2.4] is applied only when the slip extent $D_s$ is less than a given critical value, $D^*_s$. The inclusion of this feature allows the simulation of so-called “sub-critical” crack growth where fractures are observed to extend even when the applied load is below the apparent strength of the material. Most observations of this phenomenon have been made on tension-induced fractures in laboratory scale samples. (It is observed as well that temperature and moisture effects can play a strong role in determining the effective sub-critical crack growth rate). The laboratory observations are usually described by equations of the form...
\[ v/v_0 = \alpha \exp(\beta K_I) \]  
\[ v/v_0 = aK_I^n. \]

In these expressions \( v \) is the observed crack velocity, and \( v_0, \alpha \) and \( \beta \) are material parameters. \( K_I \) is the sub-critical stress intensity factor, which is lower than the fracture toughness \( K_{IC} \).

Some simple 2D tests have been carried out using DIGS to determine whether the use of Equation [2.4], with a limiting critical slip parameter, \( D^{*}_s \), can yield qualitatively similar results to the empirical behaviour described by Equations [2.5] and [2.6]. Figure 2.1 shows the cumulative mobilised slip length on a fault subjected to a fixed shear force and having a parabolically varying normal (clamping) stress profile. The initial peak ESS value is 2 MPa and the critical slip value, \( D^{*}_s \), is chosen to be 0.5 mm. The initial fracture mobilisation (sub-critical) phase, followed by rapid slip instability, is shown clearly in Figure 2.1. Figure 2.2 depicts the corresponding cumulative energy release profile with the rapid transition from “stable” to “unstable” slip. A more detailed view of the rapid energy release phase is given in Figure 2.3 that shows a peak value at time step 122 followed by some “Omori like” decay values. Some interesting precursory energy release activity can also be observed prior to the peak value at step 122. It is also of some interest to observe the energy release pattern during the sub-critical period (prior to time step 120), using a magnified energy scale, as shown in Figure 2.4. Finally, Figure 2.5 indicates the effect of increasing the applied shear stress on the effective slip velocity during the sub-critical period. The corresponding increases in average slip velocity are qualitatively similar to the empirical behaviour suggested by Equation [2.6], with some modifications to convert \( K_I \) to an equivalent critical energy release value.

In summary, it appears that the critical slip extent concept is potentially an appropriate way of capturing sub-critical slip behaviour and for combining the slip law given by Equation [2.4] with rapid instability transitions. This will be incorporated in the prototype seismic simulation tool that is described in Chapter 4.

**Figure 2.1** Simulation of sub-critical crack growth. Cumulative slip on a fault, with a given peak load, as a function of time (arbitrary time units).
Figure 2.2  Cumulative energy release on a slipping fault, as a function of time, showing rapid onset of unstable slip (arbitrary time units).

Figure 2.3  Detail of energy release increments in each time step near the onset of unstable slip (arbitrary time units).
Figure 2.4  Detail of energy release increments in sub-critical slip phase. The magnitude of the release increments is much smaller than in Figure 2.55 (arbitrary time units).

Figure 2.5  Effect of driving stress on the sub-critical slip velocity before the onset of unstable slip behaviour (arbitrary time units).
2.8 Generalised Energy Release as a criterion for layout stability assessment

A long-standing criterion for the assessment of tabular mine layout stability is the use of the energy release rate measure (ERR) proposed many years ago by Cook et al. (1966). This quantity is traditionally evaluated as part of the solution provided by tabular stress analysis codes such as MINSIM. The ERR values computed in this manner, using linear elastic solutions, require careful interpretation, and generally are not able to provide information relating to local stability/instability transitions that occur as mining proceeds. However, it is proposed that evolutionary mining and fracture generation steps can be considered to be represented by a series of punctuated transitions between successive equilibrium states in the rockmass. The original energy balance concept proposed by Cook et al. is that the difference between the energy supplied by the system loading forces and the energy taken up in the form of strain energy and dissipative processes, such as frictional sliding, can be used as a measure of “unaccounted” kinetic energy that can, in turn, cause damage during the transition from one equilibrium state to another.

The main problem is the ability to assess the magnitude of the “unaccounted” energy during each transition. This is dependent on the specific nature of the off-reef damage extent and the energy absorbed in sliding on pre-existing discontinuities in the rockmass. Provided some computational means is available to simulate these off-reef inelastic deformations, it is argued that the residual energy that is not explicitly accounted for represents a useful measure of the “degree of instability” of any simulated transition state or sequence of states. This energy quantity is therefore termed the “Generalised Energy Release” (GER) and is proposed as a criterion for the assessment of the stability of evolving layout sequences. The application of the GER measure has been illustrated in many of the examples discussed in Chapter 2. It should also be noted that GER can be very easily computed if the off-reef damage processes are represented by assemblies of interacting displacement discontinuity elements where both the traction values and the discontinuity sliding or opening components are known before and after each transition between successive states.
3 Proposed Integration Method

3.1 Introduction: practical integration work.

Mine layout designs for controlling seismicity and rockburst damage are currently based on use of the criteria ERR, APS and ESS for long- and medium-term planning. In the medium term (of the order of months), these criteria are applied to design final pillar layouts and to guide changes in planning when the knowledge of geological conditions improves. Many recent publications have proposed various approaches to mine layout design that integrate modelling with seismicity for better estimating the future seismic risk. Probably the best source of such papers is the proceedings of the 5th and 6th Rockburst and Seismicity in Mines symposia. The use of seismicity to estimate rock strength was studied by Wiles et al. (2001), Beck & Brady (2001) and by Côté et al. (2001). Lachenicht et al. (2001) presented detailed quantitative analyses of seismicity resulting from fault slip. Hoffmann et al. (2001) and Spottiswoode (2004, 2005) compared the spatial and temporal distribution of observed seismicity to the amount expected from stress or energy changes within the elastic rock mass.

3.2 Integration concepts

The project GAP 603 was the first SIMRAC project to specifically address the potential of seismic-modelling integration. Figure 3.1 is taken from the GAP 603 final report (Figure 1.1, p7) and represents a proposed flow chart of a possible software package that can be designed to implement the general principles of integrating real seismic data with mining-oriented numerical modelling.

![Figure 3.1 Proposed components of a software package designed to implement integration (from GAP 603).](image-url)
Figure 3.2 A truncated version of Figure 3.1 to illustrate the components used in the MINF/MINSINT suite.

The approach used in this chapter is simpler than that advocated by GAP 603 (Figure 3.1) and is shown in reduced form in Figure 3.2. The proposed basic philosophy is similar to that followed by meteorologists when considering changing weather conditions, as sketched in the cartoon in Figure 3.3.

Figure 3.3 Cartoon for simple weather forecast.

The approach described here was first proposed by Spottiswoode (2004b, 2005). It is basically similar to the simple model for weather forecasting shown in Figure 3.3 in that both data (the weather today) and a model (a change is expected from what is happening elsewhere) are involved. In the following sections, reference is made to Energy Release (ER), referring to some
model of energy released in any identified area. In this study, the simplest form of energy release, namely the elastic strain energy release is used (Figure 3.4). This is applied separately to each grid point mined. The classical Energy Release Rate (ERR) would be the area under the triangle marked ER in Figure 3.4 if the area mined is not included in the calculation.

$$ER = \frac{\text{Stope closure after mining} \times \text{area mined}}{\text{Stress before mining}}$$

Figure 3.4 Elastic strain energy released.

3.2.1 Assumptions

The integration model proposed herein (Figure 3.2) is less sophisticated and less complicated to apply than that proposed in GAP 603 (Figure 3.1) in that the feedback loop of imposing observed seismic deformations on the model is not applied. The feedback loop places demands on interpretation of seismic data that are extremely difficult to meet, namely interpreting each significant event in terms of the correct spatial distribution of deformations so that the models can then identify regions in the rock that are due to fail or overdue for failure.

Not only does the correct spatial distribution of deformations require very accurate hypocentral locations, but also an accurate knowledge of the three-dimensional distribution of the amount and direction of fault-slip. This is especially true of larger events that fail rock hundreds of metres in extent. This in turn requires appropriate corrections for velocity structure, issues such as dilatancy associated with fault formation and the effect of stope closure and ride on the recorded seismograms. Only preliminary work has been done to fully invert for the full seismic source in mines (e.g. Dzhafarov, 1998). Even determination of the zero-order moment tensors, in which no three-dimensional source geometry is obtained, is the subject of active research in mines (e.g. Andersen (Linzer) & Spottiswoode, 2001 and Milev et al., 2005). An indication of how far mine systems are from this ideal world is that wave velocities are normally considered to be constant for P- and S-waves across entire mines, even when there is very clear evidence of velocity variations. For example, Vertical Seismic Profile (VSP) logs show wave velocities in Ventersdorp lava that are substantially faster than wave velocities in Witwatersrand quartzites and shales. Contrasts in velocities are used in reflection seismology (e.g. Vibroseis) to map the VCR.

Seismic deformation is typically distributed across a mine by assuming that each source spreads spherically from the hypocentre (e.g. Hoffman et al., 2001). However, earthquakes, and presumably mine seismic events, often spread asymmetrically from the hypocentre. For example, the slip causing the Banda Aceh mega-thrust earthquake of December 26, 2004 (Vigny, 2005) extended much further to the north than to the south. The report by Vigny (2005) also showed other complex behaviour of this earthquake in terms of its aftershocks and inferred aseismic slip region.
The question as to the relevance of asymmetric or complex slip distribution immediately arises. Basically, assuming symmetrically distribution of slip could result in assumed distributions of deformations that are severely in error.

The following assumptions are made in this study:

1. The slip distribution should be defined in terms of the source location and strength, as with the symmetrical model, but also conditioned by the modelled distribution of expected slip. In MINSINT, circularly symmetrical distribution values placed on reef are multiplied by the expected distribution provided by the model at each grid element.

2. Over short periods of time seismicity is highly irregular, particularly in terms of larger events. However it is assumed here that seismicity can be viewed as a continuous process on the scale of months, in which the amount of seismicity in any area varies according to some modelled seismicity rate. This assumption is supported by the nearly linear relationship that has often been found between cumulative seismicity and strain energy release that is reported, for example, in the Remnant Project final report (Sellers et al., 2005).

3. Geological conditions do not vary rapidly in time or space from one mining step to another. This is not valid for faces that breast onto dykes or faults in which case a sudden increase in seismicity can be expected. If the standard recommended practice of maintaining an angle of more than 30° between the face and a geological discontinuity is followed, the location of the feature and resulting seismicity will migrate along the face and a steady state situation can be reached in which the amount of seismicity associated with a geological feature remains approximately the same.

4. Virgin stresses are also locally similar. For example, the k-ratio is assumed to vary less over a scale of tens of metres than over thousands of metres. This assumption is in line with normal geostatistical experience with regard to grade, for example.

5. The total amount of seismicity is proportional to the elastic strain energy release. This is the simplest approach and has been adopted here while the integration methodology is being developed. It is envisaged that other measures of modelled energy release could be adopted at a later stage. Examples are capped ER (Spottiswoode, 1999), GER (Section 2.8 in this report) or estimated seismic energy release (Spottiswoode, 2001).

### 3.2.2 Methodology

The methodology applied for MINSINT is now described.

1. Identify mine planning options for future mining in an area.

2. Model seismic action (details in Section 3.3 below), taking into account:
   
   a. Multi-step mining geometry of previous mining, at monthly intervals;
   
   b. Multi-step mining geometry of planned mining;
   
   c. Grid generation to convert mine outlines to a grid of squares that are mined out at different times, or left unmined;
   
   d. Geological information: stratigraphy, intrusions and discontinuities (not considered in the current study); and
   
   d. Numerical modelling code to generate seismicity estimates. In this study, ER = strain energy release is used, as estimated by the MINF code.
3. Measure seismic response to mining (details in Section 3.4 below):
   a. Grid the seismicity by projecting events onto the nearest square grid squares;
   b. Smooth patterns of modelled ER within mined areas during each mining step; and
   c. Smooth patterns of observed seismicity within mined areas during each mining step.

4. Estimate future seismicity on planned areas of mining (details in Section 3.5 below):
   a. Estimate ER on planned mining areas by migrating data from past mining;
   b. Estimate seismicity on planned mining areas by migrating data from past seismicity;
   c. Estimate seismicity ratio in areas of planned mining, defined by:
      \[ \text{Seismicity ratio} = \frac{\text{previous seismicity}}{\text{previous ER}}; \]
   d. Multiply seismicity ratio by expected ER on areas of planned mining, generating a pattern of expected seismicity on gridded areas; and
   e. Group gridded data into panels or contiguous mining areas.

5. Assess whether the level of expected seismicity in each area is acceptable. If not, return to step 1 above and re-plan.

3.3 Modelling seismic action

There have been a number of ways in which numerical models simulate the amount of seismic “action” that takes place. The earliest measure of seismic action was ERR multiplied by the area mined (Cook et al., 1964). McGarr (1976) and McGarr & Wiebols (1977) suggested that the total seismic moment \(\Sigma M_0\) should be approximately equal to \(G\Delta V_E\), where \(G\) = modulus of rigidity and \(\Delta V_E\) is the change in volume by elastic convergence. Milev & Spottiswoode (1997) extended this relationship to include a scaling factor \(\gamma_E\) that was assumed to be of geological origin.

Ryder (1988) proposed the use of Excess Shear Stress (ESS) on geological faults as being a suitable predictor of total seismicity. Spottiswoode (1988) suggested an extension to the ESS concept by including shear stress and changes in shear in the entire rock mass. This became known as VESS (Volume Excess Shear Stress) (e.g. Jager and Ryder, 1999, p53).

In Section 2.8, a model of GER (Generalised Energy Release) is proposed. GER includes the ER mentioned above and also energy released through fracturing.

Step 2 of those listed in Section 3.2.2 above is now explained in more detail.

2. Model seismic action
   a. Multi-step mining geometry of previous mining, at monthly intervals.

   Ideally this is obtained directly from CAD systems used on the mine. In practice, difficulties with date-flagging and polygon closure typically require that mine plans are manually digitised. This is the most labour-intensive stage of the entire MINSINT process.
b. Multi-step mining geometry of planned mining

Outlines of planned mining on CAD systems are currently being used to define mining patterns for MINSIM. In the case study presented in section 3.6 below, the forward modelling was tested on past data, all of which was manually digitised.

c. Grid generation to convert mine outlines to a grid of squares that are mined out at different times, or left unmined

This requires approximation of the mining outlines. The error from this approximation can be reduced by reducing the grid size used by MINF. This can be easily achieved by using the ZOOM function (Spottiswoode et al., 2000). In the case study presented here, a grid size of 11 m was considered to be sufficient.

d. Geological information: stratigraphy, intrusions and discontinuities

Geological information is not considered in the current study. As mentioned, it is assumed that a good approach angle to and through discontinuities is maintained.

e. Numerical modelling code to generate seismicity estimates.

In this study, ER is the strain energy release, as estimated by the MINF code

3.4 Measuring the seismic response to mining

As previously pointed out, seismic data needs special processing before it can be compared in spatial detail with modelled results. In particular, the “strength” of each seismic event needs to be distributed over the same square grids that are mined at the time of each event, as defined for the numerical simulations.

This comparison methodology has been developed considerably during this project to include spatial comparisons between past and future modelling and seismicity on an on-reef square grid. This method has two steps (Figure 3.1):

1. The location accuracy of mine seismic events in South African gold mines is typically tens of metres, whereas the stress peak on faces and abutments is only several metres wide. This disparity in dimensions makes it reasonable to shift each seismic event onto the face. This is achieved by distributing each event onto areas of mining with a smoothing function that is set to zero where no energy release is expected. A further smoothing function is applied to emulate the source dimension of each event. This has the effect of spreading the seismic deformation of bigger events over a bigger area than smaller events. Smoothing functions are shown in Figure 3.2 and their application described in more detail below.

2. One-to-one spatial correspondence and quantitative comparisons between past and future data is obtained by shifted past modelling and seismicity on to areas of planned mining, using the same modified Gaussian function. Seismicity is measured in terms of the parameters listed in Table 3-1. The values at each element for which mining is planned are based on a minimum number of mined elements and seismic events, 20 elements and a weighted total of 20 events in the simulations that follow.
Figure 3.1 A sketch illustrating (a) distribution of a seismic event onto the grid squares that are currently being mined, followed by (b) distribution of values in currently mined areas onto planned mining areas.
Seismicity can be quantified in various ways (e.g. Table 3-1). The measure that relates best to the total amount of deformation is seismic moment, expressed as $\Sigma M_0$, the summed seismic moment. However, as a few large events account for most of the total seismic moment, a simple count of the number of recorded events in a given area over a given time period will provide a more even statistical base. In the case study it is argued that the best measure might be one that may be related to the potential for rockburst damage.

**Table 3-1 Six different ways in which the seismicity may be quantified in MINSINT**

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Unit</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>no.</td>
<td>1</td>
<td>Number of events: each event is given equal weight.</td>
</tr>
<tr>
<td>rad_tau</td>
<td>MPa m</td>
<td>Source radius times stress drop: a measure of damage potential (Spottiswoode, 2001).</td>
</tr>
<tr>
<td>area_tau</td>
<td>MPa m$^2$</td>
<td>Source area times stress drop: a measure of damage potential</td>
</tr>
<tr>
<td>Va</td>
<td>m$^3$</td>
<td>Apparent volume</td>
</tr>
<tr>
<td>Mo</td>
<td>GN m</td>
<td>Seismic moment: a measure of source “strength”</td>
</tr>
<tr>
<td>Energy (En)</td>
<td>MJ</td>
<td>Seismic energy</td>
</tr>
</tbody>
</table>
In this section, the approaches sketched above are expanded into mathematical terms, using the following definitions, where modelled and seismic data are expressed as two-dimensional arrays of values projected onto reef. These definitions were developed especially for this study. It expected that they will change somewhat as this methodology is developed further.

A: Area 1 mined, m$^2$
E: Energy release when mined, J
S: Seismicity, units as per Table 3-1
G: Modified Gaussian function (Figure 3.2a)
C: Conical function (Figure 3.2b)

Two types of smoothing functions are used here:

Modified Gaussian functions (Figure 3.2a) are used to “correct” the seismic location errors by attributing each event to the nearest areas of mining that may be considered to be the most reasonable source location. These functions are also used to reduce the effect of small sample statistics, while quantifying seismicity and modelled energy release.

Conical functions (Figure 3.1b) are used to spread seismic events according the expected source size, as listed in the mine’s seismicity catalogue. The conical function is similar in shape to the asperity profile preferred by Lachenicht et al. (2001).

Each of the arrays in the previous list can be modified in various ways and will be indicated as such by use of the following superscripts:

1 Bold characters are displayed to motivate the choice of the single-character abbreviations.
C: Current value at each element
M: Current value spatially migrated to the next step.
P: Predicted value in the next step
A: Actual value in the next step.

Table 3-2 Symbols used for array operations and types.

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>~</td>
<td>Smoothed array</td>
</tr>
<tr>
<td>*</td>
<td>Convolution operator.</td>
</tr>
<tr>
<td>*G</td>
<td>Smoothes using a modified Gaussian function</td>
</tr>
<tr>
<td>*G∥A∥</td>
<td>Smoothes using a Gaussian function, only onto the area mined at this step</td>
</tr>
<tr>
<td>*G∥E∥</td>
<td>Smoothes using a Gaussian function, weighted according to the energy released in each element</td>
</tr>
<tr>
<td>*C∥E∥</td>
<td>Smoothes using a Conical source function, weighted according to the energy released in each element</td>
</tr>
</tbody>
</table>

Using these symbols and conventions, Figure 3.1a and b can be described by:

\[
\tilde{S}^C = \left( S^C \ast G∥E^C∥ \right) \ast C∥E^C∥
\]  \[3.1\]

3.5 Estimate future seismicity

Three approaches to designing macro mine layouts are expanded into simple mathematical terms in this section and then tested in the case study analysed in the following section.

3.5.1 Numerical modelling only

As mentioned above, this is the normal procedure for macro layout design. A generic approach is initially used to ensure that the layout satisfies the set criteria. During the early stages of the design of dip pillars, raise spacing, spans and pillar sizes are set. The pillar positions are then adjusted to avoid mining the larger geological discontinuities, but usually without reducing pillar widths. More ground is commonly left behind than was originally planned (e.g. Klokow et al., 2003).

The ideal numerical modelling will predict the amount of seismicity directly. However, this is not achieved and if the model is allowed to be incorrect by some factor that can be estimated from previous seismicity and modelling in the general area, then future seismicity can be estimated from

\[
S^P = E^P \times \left( \sum S^C / \sum E^C \right).
\]  \[3.2\]

However, this is very demanding on the accuracy of local detail in the numerical model. A more realistic formulation is:
\[ \tilde{S}^p = \tilde{E}^p \times (\sum S^c / \sum E^c) , \quad [3.3] \]

where the predicted energy release is smoothed over itself as follows:

\[ \tilde{E}^p = E^p \times G \| A^p \| . \quad [3.4] \]

This equation is easy to apply because it is not necessary to reallocate values from past mined areas on to planned mined areas.

### 3.5.2 Previous seismicity only

The hazard posed by tectonic earthquakes may be judged in terms of the likely incidence of strong ground motion. A definition of seismic hazard in mines has been proposed by Kijko et al. (1998) and is based on the probability of events above certain magnitudes occurring within a certain time period.

This hazard definition assumes that the future rate of seismicity will be the same as the past rate of seismicity. This might be true for a large production area, but will, in general, not be true locally as the mining and geological conditions continually change. An immediate improvement in accuracy can be expected if corrections are made for the area to be mined, as is described in this report.

If it is assumed that the current rate of seismicity per area mined will persist into the future, then:

\[ \tilde{S}^p = \tilde{S}^M \quad [3.5] \]

where spatial migration takes place through

\[ \tilde{S}^M = \tilde{S}^c \ast G \| E^p \| . \quad [3.6] \]

Geological discontinuities have a strong influence on seismicity. The correct practice is to keep an angle of more than 30° between the face and such features. This results in a concentrated area of excess seismicity following these features as the face advances. The meaning of “current” mining and seismicity in Equation [3.6] is then broadened to include “nearby” mining and seismicity when selecting seismic data. The extent of influence is expanded from each element to be mined until a sufficient number of previously mined elements and seismic events are included. In this study, 20 previous elements and events were considered sufficient. As previously mentioned, the counting of previous events was based on their distributed values. In actuality most previously mined elements contained fractional contributions from many events.

### 3.5.3 Integration of seismicity and modelling.

The “ultimate” integration model will consist of an advanced numerical model that will continually adapt according to the actual response of the rockmass to mining.

In Chapter 2.8 an approach to the numerical modelling is described in which the Energy Release associated with the change in elastic strain energy with mining is considered as the sole driver of seismicity. This approach involves full simulation of failure in three dimensions. In this study, no off-reef inelastic effects are simulated or considered and Energy Release is obtained by MINF with elastic modelling without applying any cap stress.

It may also be assumed that the amount of seismicity changes according to the change in the elastic strain energy released. This can be expressed as:
The above relationship simply means that future seismic hazard per area mined is expected to be equal to past seismic hazard multiplied by the ratio of future ERR to past ERR.

## 3.6 Case study

The study area at No 5E shaft, Driefontein Gold Mine, covers an area of about 1000 m by 1400 m in extent in which seven raises were wholly or partially mined. The mine plan was digitised from 1999 to 2002 using the program MinPlan from the MinSim2000 suite of programs. The mine plan was drawn at quarterly intervals for face positions from 1999 to August 2002 and then every month until December 2002. The MinPlan program creates files that contain the pattern of mining over a range of time steps as digitised off the mine plan. The “size” of the simulations was 128 by 128 square elements, each 11 m on a side. These files were then used as input to the program MINF which solves for the elastic convergence, stress and elastic energy release. Figure 3.1 shows the digitised mining outlines and the location of larger seismic events.

![Figure 3.1  Mining and seismicity (M>~2)](image)

### 3.6.1 Selection of seismic events.

The mine catalogue included numerous small events that were associated with off-reef development (footwall haulages and cross cuts to reef). Richardson & Jordan (2002) showed that events associated with advancing development had Moment-Magnitude M(Mo) below 0.5 in a study of nearby mines. They labelled these events type A events and interpreted them as “fracture-dominated” rupture events, as against the normal type B “friction-dominated” slip events. Several lines of evidence were presented in Cichowicz et al. (2004) that these events were the actual development blasts. They will be called “blasts” in this report.

Since this present study relates to seismicity induced by stoping and not by tunnel development, these blast events need to be excluded from the analysis. The simplest way to exclude development events would be to exclude all events with M(Mo) below 0.5. Unfortunately, doing so would exclude in excess of 80% of the recorded events.

Following the suggestion of Richardson & Jordan (2002), the development blast events were identified as those with M(Mo) less than 0.5, that located within 100 m of another events and that occurred within 80 s of another event.
Table 3-1. Number of blast and induced events greater than three values of Moment-Magnitudes

<table>
<thead>
<tr>
<th></th>
<th>M&gt;-0.5</th>
<th>M&gt;0.0</th>
<th>M&gt;0.5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Blast</td>
<td>26425</td>
<td>5652</td>
<td>49</td>
</tr>
<tr>
<td>Induced</td>
<td>17082</td>
<td>12198</td>
<td>7284</td>
</tr>
<tr>
<td>Total</td>
<td>43507</td>
<td>17850</td>
<td>7333</td>
</tr>
</tbody>
</table>

Figure 3.1 Size distribution of events identified as blast events, mining-induced events and all events.

This division of events into “blast” and “induced” events is shown in Table 3-1 and Figure 3.1. An increasing proportion of all events are likely to have been blast events as the threshold magnitude drops below 0.5. In the process, the “hump” in the frequency-magnitude distribution in the range moment-magnitude range from 0.0 to 0.5 that is visible in the graph for the entire data set disappears.

The 12198 events with M greater than 0.0 that were likely to be induced events are used in this study.

3.6.2 Identification of “working areas”

The methodology outlined above was applied to the Driefontein data set. An example of mining and seismicity within part of the area during a single time step is shown in Figure 3.1. Three distinct regions of active mining can be identified. These will be termed “working areas”. In the normal mining operation these working areas would probably be lumped together into a single “polygon” for seismic analysis to avoid misallocating events into the wrong polygon. In this analysis, the influence of events that locate between the working areas, such as A, B and C in Figure 3.1 are split between these areas according to their proximity to current mining.

An algorithm was implemented to identify such regions for each mining step. This algorithm was based on the regions being separated by at least three elements that were not being mined at that time.
3.6.3 Analysis of predictions using the three methods

Observed seismicity is compared to seismicity predicted using the three methods described above. Data for the 99 working areas that included at least ten mined elements are now shown in Figure 3.1, Figure 3.2 and Figure 3.3.

Table 3-1 Some parameters that might be used to quantify the “size” of a seismic event. Scaling is in terms of $r_0 =$ source radius and $\tau =$ seismic stress (apparent stress or static stress drop).

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Scales as</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>Energy</td>
<td>$r_0^3 \times \tau^2$</td>
<td>The largest events strongly dominate the action.</td>
</tr>
<tr>
<td>Moment (Mo)</td>
<td>$r_0^3 \times \tau$</td>
<td>Large events are still dominant. Largest events probably extend furthest from reef horizon.</td>
</tr>
<tr>
<td>Apparent Volume (Va)</td>
<td>$r_0^3$</td>
<td>Proposed by Mendecki et al (in Ryder and Jager, 2002) as a measure of the 3-dimensional extent of inelastic deformation. However the important role of stress drop in rockburst damage (Cichowicz, 1997) is ignored.</td>
</tr>
<tr>
<td>at</td>
<td>$r_0^2 \times \tau$</td>
<td>~ $Mo/r_0$ Reduces effect of expansion of source normal to reef. Retains seismic stress.</td>
</tr>
<tr>
<td>Rad_taut</td>
<td>$r_0 \times \tau$</td>
<td>~ $Mo/r02$ Dominated by small events.</td>
</tr>
<tr>
<td>Unit (No)</td>
<td>1</td>
<td>Dominated by the smallest events.</td>
</tr>
</tbody>
</table>
In Figure 3.1, the total observed seismic moment in each region is compared to the seismic moment predicted using the integration model. Note the dominant contribution of the few largest events, which resulted in the actual observed total moment being under-predicted (Y>X) or as over-predicted (Y<X), depending whether the larger events are future or past events.

![Integration model diagram](image)

**Figure 3.1 Predicted and observed seismic moment, GN-m per grid element.**

Using the number of events eliminates any dominance of the effect of any event on any other. Figure 3.2 shows the comparison between predicted and observed number of seismicity per area mined. An inspection of this figure shows, *inter alia*, that:

1. The most noticeable feature in this figure is the fact that almost all predictions were under-estimates. In other words, the seismic event rate almost always increased with time in any working area. This occurred because the span, and therefore the face stresses and values of ERR, increased.

2. The data distribution shown in Figure 3.2c is similar to that shown in Figure 3.2b. The data are about evenly spread across the “perfect” line, labelled “Y=X”. This means that the ERR correction implied by Equation [3.7] provided, on average, an appropriate correction for the deficit apparent in Figure 3.2b.

If this approach were to be used for deciding whether to continue mining in an area or not, the bounding line labelled “Y=X+1” provides a conservative upper limit to the impending seismicity. Working places that were predicted to result in less than one event per element did indeed generate this small seismicity rate.

The data points within the dashed ellipse in Figure 3.2c were the most severe examples of over-prediction. This will be contrasted later with a similar region in Figure 3.3c.
Figure 3.2 Comparison of the observed number of events per mined element with the predicted number of elements by each of three model approaches. The ellipse in (c) is drawn around some of the data that were over-predicted.
Predicting the number of events is of limited value because most of these events are of negative magnitude and do not cause any damage. As shown in Figure 3.1 the opposite extreme of using seismic moment to predict the amount of seismic moment is ineffective. A suitable compromise is measuring each seismic event as source radius multiplied by apparent stress \((r_0 \times \tau)\), labelled “rad\_tau” in Figure 3.3. This parameter was chosen in an attempt to represent both the extent (source radius, or \(r_0\)) and the intensity of strong ground motion. The intensity of strong ground motion is represented by stress drop \(\tau\).

Figure 3.3  Comparison between the observed number of events per mined element and source radius time apparent stress, MPa-m.
The small weighting given to larger events by applying a “strength” of \( r_0 \times \tau \) gives better results for the seismicity-related predictors, as seen in Figure 3.3b and c. In particular, the number of working places that had their seismicity severely over-predicted decreased, as can be seen by fewer events falling in the elliptical region of Figure 3.3c compared to a similar region in Figure 3.2c.

### 3.6.4 Conclusions and discussion

This section describes the principles of a methodology, and makes reference to a computer program, for an integrated analysis of both rock mass modelling and observed seismicity to predict, or forecast, the amount of seismicity that is likely to occur over the immediate future of a month or quarter in any working area.

The results show the potential benefits of this type of integration work for mine layout planning. The techniques presented here for associating observed seismicity with modelled seismicity are new at the time of writing and further developments of the method and more detailed analysis will undoubtedly lead to better interpretation of the likely response, in time and space, of seismicity to mining.

Although the results are not sufficiently convincing to be implemented, they are encouraging. As this is the first analysis using such a methodology, there will obviously be much scope for improvement, even in the short term. A partial list would include:

- **Modelling:** There are a number of ways to introduce simplified approximations that simulate actual rock failure (e.g. Spottiswoode, 2001). These should provide better insights into the rock mass response at high levels of ERR and over time.
- **Seismic data:** Seismic locations can be improved using methods described in Cichowicz et al (2004) and expanded by Spottiswoode & Linzer (2005).
- **Statistical processing:** Very simple methods have been used in this study. For example, all working areas have been given equal weight and equal symbol sizes in Figure 3.1, Figure 3.2 and Figure 3.3, even though larger areas behaved better. Improved methods will be essential before this work can be used for quantitative hazard analysis.

### 3.7 Introduction to the programs MINF and MINSINT.

The programs MINF and MINSINT can be used for estimating the amount of seismicity expected from any sequence of planned mining, based on past seismicity and mining and the planned mining itself. At this stage, the effects of changing geological conditions are not modelled. In the simplest sense, such modelling could be performed by multiplying the ER within or near geological features by some factor. Spottiswoode (1988) showed that dykes and faults increased seismicity in their vicinity by a factor of about 3.5.

The program MINF is a boundary element program similar to MinSim2000. It originated in early work by Ortlepp & Spottiswoode (1982) and has been further developed under projects GAP 612c, GAP 722 and the current project SIM-020301. MINF can now perform a number of functions, but the focus in this manual is on its use together with the MINSINT code in performing integration of seismicity and modelling. MINSINT was developed for project SIM-020301.

These programs have been integrated as a suite to test the concepts and criteria mentioned above by doing large-scale back-analyses of mining and seismicity, where large-scale means areas of about 1000 m in extent. Since June 1999, MINF also has the capability of generating
and analysing simulated seismicity. This feature and several others are listed in section 3.7.1 with a brief explanation in Section A.4.

The program MINSINT performs the integration of seismicity and modelling and estimates the amount and distribution of expected seismicity within planned areas of mining.

![Flow-chart of integration procedure.](image)

**Figure 3.1 Flow-chart of integration procedure.**

The recommended integration procedure is shown as a flow chart in Figure 3.1. Two additional programs are included for convenience, namely MinPrep and MinView3D. These programs form part of the MinSiM2000 suite of programs. Although their use is recommended, they are not essential for the integration. MinPrep is not described in detail here. A brief description of MinView3D is provided in Section A.3.

Work is underway for CADSMINE to generate CXX files. This will initially apply only to planned mining.

The relevant functions of MINF and MINSINT are now described. The program MINFLIST was previously described for GAP 722 and has been omitted in this report for simplicity.

### 3.7.1 MINF

MINF is a MinSim2000-type boundary element solver for large numbers of grid elements on a single reef or multiple parallel reefs. It reads MinSim2000 pattern files. Two reefs may be integrated using up to 512 by 512 square elements. The solution consists of the on-reef convergence, ride and stress at each mining step and the strain energy changes associated with mining, at each element. The solution is written to PXX files in extended MinSim2000 format. Off-reef stresses and displacements on reef-parallel benchmark sheets may also be calculated.
MINF has also been developed to solve for:

1. Large (1024 by 1024) and huge (2048 by 2048) mining problems.
2. Soft seam, for coal mining.
3. Shallow mining with free surface, for coal mining.
4. Seismicity generation, for deep mining (*).
5. Slippery reef-parallel planes (*).
6. Identification of isolated pillars and APS listing (*).
7. Pillar failure and pillar runs (*).
8. Boussinesq solution, for dam loading (*).

Items 4 to 7 are flagged with (*) to indicate that these problems can be solved with the “standard” version of MINF, while items 1 to 3 require specially compiled versions of MINF. These options are briefly described in Appendix A.3. The use of common blocks for sharing memory across MINF and MINSINT precludes the use of dynamic memory allocation, resulting in separate executable files (EXEs) to handle different size of problems.

3.7.2 MINSINT

MINSINT performs all the steps described in points 3 and 4 above and covered in Sections 3.4 and 3.5 above.

3.8 Working with MINF and MINSINT

3.8.1 Running MINF and MINSINT

MINF and MINSINT are console application programs. It is recommended that they be configured using the Windows program DFTWrap. They may also be started from a DOS prompt or by a batch file that can be invoked from Windows Explorer. The batch file command or prompt command takes the form:

```
PRNAME [prob]
```

where PRNAME is either MINF or MINSINT; and

```
prob
```

is the problem name as used in the MinSim2000 pattern files.

The files prob.ciN or prob.cNN describe the mining pattern. The prob field above can take up to 100 non-blank ASCII characters. Normally, prob will consist of only a few characters; the 100-character space is provided specifically to allow DFTWrap to run MINF and MINSINT in any directory on disk.

If prob is not provided on the command line, the user is prompted for one by the program. All user-defined parameters are read from the file prob.DFT. DFTWrap facilitates user-friendly editing of the file prob.DFT and can launch MINF or MINSINT.
3.8.2 Editing input files using DFTWrap

A number of input files that contain information related to the mining geometry, seismic data and modelling parameters are required by MINF and MINSINT. These files are all located in the same directory on disk and have the same filename (which is also referred to as the “job name”), but different extensions. The file containing the model parameters is known as the DFT file, and has the extension .DFT.

The task of editing the DFT files for MINF and MINSINT previously entailed the use of a text editor such as “Notepad”. This process was both laborious and prone to an assortment of errors. In order to alleviate some of these shortcomings, Lindsay Linzer has programmed a generalised “wrapper”, called “DFTWrap”, that reads the DFT files containing the parameters needed to run MINF and MINSINT. DFTWrap can also be used to edit Aura DFT files (“Aura” is the seismological processing software written by CSIR Miningtek), because the files have the same structure as the MINF and MINSINT DFT files.

DFTWrap is a Windows graphical user interface (GUI) program that displays the information contained in the DFT file and allows the values of the input variables to be edited. In addition, the program checks the format of each line in the DFT file. Errors are reported (together with a line number) to the user by means of a Windows message box. Once changes have been made, the new input values are saved back to the DFT file. To help prevent typographical errors, the program checks that the new values are within a prescribed range and are of the correct type. MINF or MINSINT can then be run in the usual manner or launched from the “Run” menu item of the wrapper. DFTWrap can also be used to run the Seismic Explorer, the database module of Aura.

The program was written in Delphi 5 and consists of a single executable named DFTWrap.exe that can be run from the Windows Explorer by clicking the icon. It is a general program, designed to read DFT files having a prescribed format, and the program create certain GUI elements dynamically.

The following section provides a step-by-step description of the process of editing a DFT file using DFTWrap.

3.8.2.1 DFTWrap Step-by-step

When the program is launched, the first screen shown to the user is illustrated in Figure 3.1. The user selects the DFT file of interest using the “File” → “Open DFT File” menu items or clicking the button on the toolbar. Either of these user actions will launch a “File Open” dialogue box, as shown in Figure 3.2, and the user will be required to navigate to a DFT file. In the example, the DFT file is named “Demo.DFT”.

Once the relevant DFT file has been selected, the program creates a data structure containing all the information represented in the DFT file and populates a tree structure (Figure 3.3) with this information. If an error in the format of the DFT file is encountered, an appropriate message will be displayed onscreen, indicating the line number of the error (Figure 3.4). The user then has the option of loading the DFT file into the default Windows text editor (e.g., “Notepad”) using the button on the toolbar, or opening the file from Windows Explorer. The text file can also be opened via the “File Open” dialogue box (Figure 3.2) by navigating to the file using the functionality provided by the dialogue box, clicking the right-hand mouse button and selecting the “Open With” option on the context menu.

Clicking on the top node of the tree expands the first tier of the branches (Figure 3.5) to reflect groups of variables and parameters. Clicking on the branches of the tree will expand the tree further and the individual variables of each group will be displayed (Figure 3.6). At this stage, a grid containing the variable names, values, default values and comments will be displayed on
the right-hand side of the main window. The variable values can be edited by means of the keyboard and mouse. As each value is edited, the program checks whether it is of the correct type and lies within the specified range of values. If not, a message box describing the error message is displayed, and the newly edited value is set to its default value (Figure 3.7).

The program can be run in two modes: a “beginner” mode that only displays simple variables, and an “expert” mode that allows all the variables to be edited. These modes can be accessed using the “Settings” menu item. The screendump in Figure 3.5 was created in beginner mode. In this figure, several of the tree nodes are not preceded by plus signs, indicating that the variables of these nodes are only for advanced users.

Once the user has edited the variable values, the data structure is saved back to the DFT text file using the “File” → “Save As” menu item or the button on the toolbar. The text file can also be saved with a different name.

MINF or MINSINT can then be run, either in the usual manner, using the “Run” → “MINF” or “Run” → “MINSINT” menu items, or by clicking on the green or blue button on the toolbar. The red button is to run the Seismic Explorer program. The first time an attempt is made to launch MINF or MINSINT, the user is prompted by a dialogue box (Figure 3.8) to navigate to the directory containing the appropriate executable. The path to the executable is then stored in the Windows registry for future runs.

The program version and general information is displayed on the “About” dialogue, accessed using the “About” menu item on the main program window (Figure 3.9).

![Figure 3.1 Main window of DFTWrap.](image)
Figure 3.2  File open dialogue box used to select the DFT file.

Figure 3.3  Data structure – top level node in tree.

Figure 3.4  DFT file format error message.
**Figure 3.5** First tier of the branches of the tree structure.

**Figure 3.6** Individual variables of a branch in the tree structure.
Figure 3.7 Range error message.

Figure 3.8 Navigating to the directory containing MINF and MINSINT.

Figure 3.9 The About dialogue.
4 Development of future technologies

It is important to continue to develop technologies which can improve mine design and stability analysis. This chapter shows where the technologies described in chapter 2 and chapter 3 are headed, and how 3D fracture growth models are now feasible, and could ultimately be integrated into the analysis and design of mining layouts. The chapter also covers pilot studies into potential technologies which could lead to more advanced integrated analysis by including additional seismic measurements, dynamic models and automated optimisation.

Chapter 3 developed a methodology for integrating observations of seismicity with fairly simplistic but large-scale models of energy release. The methodology can in principle be applied to more complex models using more advanced criteria for assessing stability. In particular, Chapter 2 proposes a “Generalised Energy Release” (GER) criterion, which is particularly well suited for computation in models using explicit representations of off-reef fracturing. However, explicit fracture models are computationally intensive and the useful modelling of 3D mining layouts is currently not feasible. Two possibilities exist, however, for meaningful layout assessment using 3D models of explicit fracture growth. The first is to produce somewhat simpler models of the fractures and the fracturing process. The second is to make use of high performance parallel computing, a trend which has developed in 3D computational physics as it becomes increasingly economically viable. Truly large-scale 3D fracture models could make a major impact to stability assessment, and both options should be pursued. This project has investigated the former approach, using simplified models of fracturing to enable 3D analyses. The present chapter covers this novel method and its application to fracture development in a lead-lag geometry and to the evolution of a crush pillar.

The development of models representing off-reef failure and their integration with observed seismicity has been the fundamental thrust of this project. At this stage only the simplest, most stable inversions from seismic recordings (viz. magnitudes, rate and locations) have been contemplated for comparison with models. Ultimately, one can envisage integrating fracture models with more complex seismic inversions, including the mechanism of failure, the size of fracture, and the stress drop. If methods of observation can be improved, then observations and models could also be constrained to have an equivalent density and depth of fracturing. Seismic observations and inversions themselves could perhaps also be improved through integration with dynamic numerical models, where the consistency of inversions could be tested, and algorithms optimised. Finally, methods of automated optimisation could be introduced to enhance the integration process. The second part of this chapter covers some pilot investigations during this project into these novel areas.

4.1 Development of three-dimensional models to simulate seismic activity and rock failure processes

The simulation of rock failure processes is often approximated in two-dimensional space by considering the in-plane movements that occur in representative cross-sections. A number of typical applications of this approach are illustrated in Chapter 2. It is, of course, essential to develop techniques that can allow an assessment of inelastic failure mechanisms in full three-dimensional space. This goal is difficult to achieve for a number of reasons. The most immediate difficulty is simply the fact that most numerical schemes require some form of computational grid that spans the region of interest. The computational effort is correspondingly proportional to $L^2$ in two-dimensional analyses, and to $L^3$ in three-dimensional problems, where $L$ is the characteristic problem dimension. If the representative grid size is $g$ then the cost of performing a three-dimensional, relative to a two-dimensional, failure analysis is of the order of $L/g^3$. For a relatively fine zoning, the factor $L/g$ can be of the order of 1000, which may simply prohibit the direct application of the chosen computational scheme in three dimensions. A potential resolution of this difficulty is offered by current developments in parallel computation.
that will inevitably become increasingly accessible. Nevertheless, there remains a considerable challenge to devise methods that can provide useful simplifications to full-space problems.

Studies were carried out to investigate the potential for introducing an alternative “mesh-free” solution scheme specifically for tabular mine layout analysis. Inter-element stress singularities can be eliminated if the displacement discontinuity density is continuous and possesses a continuous tangent (is differentiable) at all points within the crack surface. It is difficult to ensure that this condition is met when the discontinuity surface is covered with arbitrarily shaped, convex, polygonal elements such as squares or triangles. One solution to this problem is to construct a local representation of the surface density using a “moving” least-squares fitting procedure. Preliminary attempts to assess the viability of introducing such a method for the efficient analysis of induced seismic effects near tabular excavation edges were investigated. One of the main shortcomings of the method is the limited effective size of the local region within which simple polynomial basis functions can be used to represent the discontinuity density. Some form of surface partitioning or tessellation is necessary to achieve a sufficiently accurate representation of the discontinuity values over the whole surface. It is logical to seek an approach that allows the least-squares estimation procedure to be applied within a limited local region and to compute supplementary influence “corrections” from all discontinuity partitions or elements external to the designated local region. Smooth estimates of the surface stress at arbitrarily close, neighbouring surface positions can be achieved by means of a “taper” function that “blends” the local least-squares region with the external region influences. However, it is found that near the edge of the excavation, large errors can arise in estimated stress values. These discrepancies are found to decay very rapidly away from the crack surface. The least-squares technique can also be used for the efficient computation of far field influences (“lumping”) to improve the solution efficiency of large-scale problems. A full component implementation of the method could be useful in general layout solutions but these possibilities have not been pursued in the present study.

4.1.1 Mesh-free concepts in tabular excavation stress analysis

Figure 4.1 illustrates an artificial tabular layout outline containing regularly spaced node points at which local estimates of the closure profile are established using a moving least-squares fitting scheme. The estimated closure values are shown in Figure 4.3 and can be compared to the values obtained using a conventional element-based solution shown in Figure 4.2. It can be seen that the mesh-free solution closure contours are qualitatively in good agreement with the element-based solution contours. Small differences in detail can be noted close to excavation edges and corners.
**Figure 4.1**  Example of mesh-free analysis of tabular layout problem.

**Figure 4.2**  Closure contours for layout problem (solution from 3DIGS).
Closure contours (mesh-free expansion order = 5)

Figure 4.3  Mesh-free solution of tabular layout problem using a local quadratic variation moving least-squares estimate of the closure at each node point shown in Figure 4.1.

4.1.2 Simulation of three-dimensional fracture and damage processes

In order to carry out simulations of likely stability changes in the course of mine excavation it is necessary to represent both inelastic rock failure behaviour and dynamic wave propagation effects. These goals can only be partially achieved by current computational tools. It is useful to consider an explicit hierarchy of capabilities that are required, presented here in increasing order of difficulty.

1. Computation of displacements and induced stress changes that arise from the successive introduction of excavation steps in a linear, elastic medium. This is the easiest goal to achieve and a number of computer simulation codes are available for this purpose. (Specific examples are the MINSIM, MAP3D and MSCALC codes).

2. Excavation in a linear, elastic medium with the capability of simulating wave propagation effects in the medium. This form of analysis can be carried out using finite element, finite difference and boundary element techniques. Caution has to be exercised in interpreting possible spurious results caused by numerical “dispersion” effects. Currently, the computer code WAVE (SIMRAC project GAP 601b) is probably the most useful model for this form of analysis in relation to tabular layout and fault slip problems.

3. Simulation of material damage and scale effects. Fracture initiation and coalescence behaviour is strongly linked to the development of numerical methods that can represent complex geometrical failure patterning in two and three dimensions. Numerical simulation strategies for non-linear behaviour generally follow either the formulation of a continuum,
plasticity or damage mechanics description of material failure or some form of distinct element or particle assembly analysis. In addition, a number of procedures have been developed where crack segments are selected in sequence from an assembly of pre-defined discontinuity positions (for example in SIMRAC projects GAP 029 and GAP 601b). The representation of fracture growth processes by chaining successive displacement discontinuity boundary elements end to end, in calculated orientations, can be carried out relatively easily in two dimensional plane strain problems. The incorporation of slip-weakening models for discontinuity slip also enables scale effects to be modelled (SIMRAC project GAP 601b).

In summary, the modelling of three-dimensional fracturing can be achieved only in limited regions of interest using available finite element or distinct element schemes. These restrictions can be addressed, to some extent, if rock damage can be simulated using a limited number of discrete, problem-dependent fracture growth surfaces. An elegant method to represent the influence of such surfaces on the surrounding material is the displacement discontinuity boundary element method. In the current research project (SIM 020301) some effort has been made, therefore, to explore the possibility of using this approach to simulate three-dimensional fracture and seismic damage processes. The motivation for this is to determine whether a generally useful approach can be developed for the investigation of fracture patterning and stability behaviour in three dimensions. At this stage, no attempt is made to extend the method to the simulation of dynamic, inertia-controlled wave propagation effects.

Although the analysis of individual crack or slip surface movements is easy to calculate using the boundary element displacement discontinuity method, it is far more difficult to construct a robust procedure for three-dimensional fracture initiation and growth processes. This requires, in particular, an understanding of the growth mode and the necessary logic to control both the evolution of the fracture front and potential intersections of the crack front with pre-existing discontinuities. It is still a matter of research to identify the nature of mode II (in-plane) fracture growth and, indeed, whether this can be properly represented by appropriate macro-constitutive properties that are equivalent to the complex micro failure events that make up an emerging shear band structure. Equally importantly, very little information is available on the fundamental physical processes that determine effective mode III (“tearing” mode) shear strength and propagation properties and the precise conditions under which mode III shear propagation occurs. For example, petal-like wing cracks can be formed in tension, and parallel to the major loading direction, in positions that are adjacent to the points of maximum mode III slip in inclined penny-shaped cracks in brittle materials.

It is apparent that any construction of three-dimensional fracture growth surfaces will depend on a number of ad hoc and intuitive assumptions concerning both the material strength parameters and the failure growth directions. Once these assumptions have been made, it is necessary to devise a suitable computational scheme to represent the geometric evolution of the crack front and to control the possible intersection of the front with pre-existing crack or excavation surfaces. No attempt has been made in the present study to formulate such a scheme. However, in order to make progress, a simplified modelling procedure has been investigated. This simplified scheme appears to be capable of providing some insights into the processes of evolutionary three-dimensional fracture patterning accompanying progressive mining steps.

### 4.1.3 Simplified strategy for simulation of multiple fracture clusters

A special purpose prototype computer code has been written to allow the simplified simulation of 3D damage by avoiding explicit evolution of complex fracture surfaces and fracture front tracking. Instead, such features are approximated by means of multiple, overlapping (but non-intersecting) clusters of circular cracks or by three-dimensional triangular crack element tessellations. The crack clusters can, if necessary, be constrained to lie on individual, pre-defined fault planes or curved surfaces. The prototype code employs displacement discontinuity influence kernels for both circular disc-shaped elements and triangular-shaped elements. The slip or crack opening functions over the circular elements can be controlled using up to five
internal collocation points. Constant or linear variation shape functions are available for the
evaluation of slip/opening distributions over the triangular elements. A simulation exercise is
carried out by defining a tabular mining outline, and required mining steps, using triangular
element tessellations. A distribution of random seed point positions is specified in a volume
covering a particular region of interest. The detailed simulation steps are as follows.

1. Specify the geometric outline of all tabular excavations and mining step increments. Specify
random seed point positions or three-dimensional space-filling tessellations.

2. Solve for the closure and ride distributions in all excavations.

3. Compute the closure tensor components induced at each seed point and determine the
principal stress values and directions. Let the principal stress components be designated as
$\sigma_1$, $\sigma_2$ and $\sigma_3$ where it is assumed that $\sigma_1$ is the most compressive principal stress value
and $\sigma_3$ is the least compressive principal stress value.

4. Check whether a circular crack element can be introduced at the seed point. The crack is
assumed to be sub-parallel to the direction of the intermediate principal stress component,
$\sigma_2$. If $\sigma_3$ is in tension, assume that the crack orientation is perpendicular to the $\sigma_3$ direction.
If $\sigma_3$ is in compression, assume that the crack will initiate in a shear mode and will be
inclined at an angle of $\pm \alpha$ to the direction of the $\sigma_1$ principal stress direction, where $\alpha = \pi/4 -
\phi/2$ and where $\phi$ is a specified internal angle of friction. One of the two conjugate directions $\pm
\alpha$ is chosen at random or is selected to ensure that the crack will be aligned as closely as
possible to a specified preferential growth direction.

5. Determine the failure potential at the circular crack centroid or at the centroid of a potential
triangular crack element with a pre-defined orientation. The failure potential is expressed as
the “distance” to a given failure envelope.

6. If a crack can be initiated at the seed point, save the crack in a ranked list of potential failure
elements. Check that the potential crack does not intersect existing seed cracks or
excavation surfaces.

7. Finally, select a specified number of new elements from the ranked list of potential crack
elements and re-solve the entire assembly allowing the additional elements to interact with
one another and with existing excavation-defining elements.

8. The crack selection procedure is repeated from step 3 for a specified number of growth
cycles and the entire computational sequence is repeated from step 1 for as many mine
excavation steps as are required.

Results from the test code developed to perform the solution procedure described in steps 1 to
8 are presented in the next section. It is also possible to allow creep relaxation to occur in
selected growth elements according to a specified relaxation rate law. This feature is not
pursued in the current study. It is important to note that no attempts have been made to
optimise the run time efficiency of the prototype test program by means of “lumping” techniques
or special iterative solution schemes. These refinements are not warranted at the current stage
of development of the test code. However, it is found that iterative instabilities can sometimes
be encountered when certain element proximity tolerances are violated. It is also found that the
simple slip weakening and tension weakening rules that have been implemented can cause
iterative difficulties if reverse sliding or oscillating crack opening-closing load cycles arise. These
deficiencies need to be addressed in further developments of the test program.
It is possible to allow fixed orientation triangular or circular elements in a manner similar to the seed point selection procedure. In this case, the element edges are fixed, and stress values are evaluated at the element centroid to determine whether fracture can be initiated within the element surface. This facility effectively allows a “tessellation” approach, using Delaunay tetrahedral structures or Voronoi polyhedra, to be implemented as an alternative to the selection of random circular, non-intersecting seed elements. Further investigation is required to assess whether such an approach is viable for the simulation of three-dimensional failure processes.

4.1.4 Case studies and examples

A number of test cases have been evaluated to illustrate some of the features and limitations of the prototype crack growth simulation code.

4.1.4.1 Circular sliding crack with adjacent wing cracks

To illustrate the complex nature of fracture growth in three dimensions it is of interest to consider the case of a single circular crack that is inclined at an arbitrary angle of 26.6 degrees to a far-field compressive stress of 100 MPa. The friction angle of the sliding crack surfaces is assumed to be 10 degrees and the load direction is assumed to be parallel to the z-axis of a local coordinate system. The crack radius is assumed to be 10 m and two seed points are located adjacent to the crack x-axis diameter as illustrated in Figure 4.1. Two cracks of radius 2 m are allowed to initiate at the seed points. The orientation of these cracks is indicated in Figure 4.2 and Figure 4.3 and is found to be perpendicular to an induced tensile stress of approximately 10 MPa in the y-axis direction.

Figure 4.4 shows the fracture orientations that arise at the seed points if the far-field x and y stress components are set to -10 MPa and -15 MPa, respectively (negative signs indicate compressive stresses). In this case, the principal stress components at each seed point are all compressive, and conjugate shear initiation orientations are assumed. The choice of which conjugate direction to choose is made at random unless a preferential growth direction is specified. In the example shown in Figure 4.4, both directions happen to have been chosen and the intermediate principal stress component direction is along the x-axis. The far-field intermediate principal stress direction is along the y-axis. It is also very important to note that the crack initiation angle depends sensitively on the seed point position. In the case of the inclined circular crack, it is found that strong tensile stress values can be induced near the edge of the crack in the position where the mode III sliding movements are at a maximum.
Figure 4.1 Circular crack with adjacent seed points. The crack is parallel to the x-axis and is inclined at an angle of approximately 26.6 degrees to the z-axis. (The circular crack outline is depicted approximately using a twelve-sided polygon and appears to be elliptical since the crack is inclined at an angle to the z-axis).

Figure 4.2 Wing cracks nucleated in tension adjacent to a sliding circular crack (edge view along the x-axis, parallel to the crack planes).
**Figure 4.3**  *Wing cracks nucleated in tension adjacent to a sliding circular crack (oblique view).*

**Figure 4.4**  *Conjugate shear crack initiation from seed points when far-field x and y-direction stress is compressive.*
4.1.4.2 Shear fracture growth from random seed points

It is of interest to consider the progressive nucleation and evolution of circular fractures initiated at random points in a specified region. Consider, in particular, the case of a cubic region of side 40 m, having 5000 randomly distributed seed points. It is assumed that the principal stress directions coincide with local $x,y,z$ Cartesian axes and that $\sigma_x = -20$ MPa, $\sigma_y = -40$ MPa, $\sigma_z = -200$ MPa, respectively. Figure 4.1 illustrates the successive growth of 120 circular seed elements, viewed along the intermediate principal stress direction (y-axis). Each element is assumed to have a fixed radius of 2 m with a contact friction angle of 30 degrees and zero cohesion. The evolutionary growth pattern is seen to follow the two conjugate Coulomb growth directions inclined at $\pm 30$ degrees to the principal (z-axis) loading direction.

It is of interest to compare the growth pattern shown in Figure 4.1 to the pattern that evolves when the crack elements have different sizes. Figure 4.2 shows the clear formation of a shear band structure that evolves when the seed elements have random, uniformly distributed radius values ranging from 2 m to 5 m. This suggests that it is probably important to allow a range of fundamental seed element sizes to be selected in order to simulate shear band evolutionary growth processes. Figure 4.3 shows a view of the shear band along the minor principal stress direction (x-axis) and illustrates the different element sizes comprising the macro band. A second example, with seed point crack radius values uniformly distributed from 1 m to 5 m is shown in Figure 4.4 and Figure 4.5. It is very interesting to observe that the coherent shear band structure is retained in Figure 4.4 but that the overall shape of the slip “patch”, exhibited in Figure 4.5, appears to be more “circular” than the slip area shown in Figure 4.3. Clearly the seed point size distribution plays a significant role in determining the characteristics of macro-shear localisation features. As a further example, a different radius distribution was considered in which it was assumed that the seed element areas, $A$, were exponentially distributed following a cumulative distribution function of the form

$$F(A) = 1 - \exp(-\beta (A - A_0)). \quad [4.1]$$

The parameter $\beta$ was chosen to have a value of 0.749 m$^{-2}$, and $A_0$ is the area of the smallest allowed element having a radius of 1 m. The fracture growth patterns are shown in Figure 4.6 and Figure 4.7 exhibiting some detailed differences to the corresponding patterns shown for the uniform seed radius distribution of Figure 4.4 and Figure 4.5.

It is very important to note that the distribution of crack sizes that are selected from the parent seed population, to make up the final shear band structure, do not correspond to the size distribution of the parent population. An analysis of the selected crack seed sizes, for the seed growth simulations shown in Figure 4.4 and Figure 4.5, is given in Figure 4.8. This histogram indicates that the selected seed frequency distribution tails off rapidly. A relatively greater frequency of small-sized elements is selected for growth than in the parent population. The reasons for this are not completely clear, but may occur because after large cracks have been nucleated, space is then only available for smaller crack sizes to be nucleated in the regions between the large crack positions. This is controlled by the intersection logic of the test code which prevents a selected crack from being activated if it is determined to be too close to pre-existing cracks.

The incremental energy release values corresponding to each seed growth step are plotted in Figure 4.9. The so-called “available” energy (Napier, 1991) represents the difference between the energy supplied by the system loading forces and the change in stored strain energy. The net “released” energy corresponds to the difference between the “available” energy and the energy dissipated by friction sliding. These energy increments are distributed in a complex manner. Figure 4.10 is a plot of the cumulative net released energy and exhibits some hint of power-law distribution behaviour.
**Figure 4.1** Fracture growth from random seed points in a uniform stress field. Seed point positions are indicated by crosses.

**Figure 4.2** Fracture growth from random seed points in a uniform stress field with uniformly distributed crack radius values varying between 2 m and 5 m. (View along y-axis – intermediate principal stress direction).
Figure 4.3  Fracture growth from random seed points in a uniform stress field with uniformly distributed crack radius values varying between 2 m and 5 m. View along x-axis (minor principal stress direction) showing the distribution of different crack sizes.

Figure 4.4  Fracture growth from random seed points in a uniform stress field with uniformly distributed crack radius values varying between 1 m and 5 m. (View along y-axis – intermediate principal stress direction).
Figure 4.5  Fracture growth from random seed points in a uniform stress field with uniformly distributed crack radius values varying between 1 m and 5 m. View along x-axis (minor principal stress direction) showing the distribution of different crack sizes.

Figure 4.6  Fracture growth from random seed points in a uniform stress field with exponentially distributed crack areas corresponding to radius values varying between 1 m and 5 m. (View along y-axis – intermediate principal stress direction).
Random seed growth
(Exponential area distribution; x-axis; Rmin = 1, Rmax = 5)

Figure 4.7 Fracture growth from random seed points in a uniform stress field with exponentially distributed crack areas corresponding to radius values varying between 1 m and 5 m. View along x-axis (minor principal stress direction) showing the distribution of different crack sizes.

Figure 4.8 Histogram of the radius values corresponding to the selected crack growth elements shown in Figure 4.11 and Figure 4.12, compared to the uniform distribution frequency of the parent seed population.
Figure 4.9  Energy release increments, plotted for each crack growth initiation step, corresponding to the uniform seed radius distribution crack pattern shown in Figure 4.4.

Figure 4.10  Cumulative net energy release increments, plotted on log-log scales for the crack steps, corresponding to the uniform seed radius distribution crack pattern shown in Figure 4.4.
4.1.4.3 Simulation of fracture growth near tabular excavations

The main goal for the development of an explicit method for the simulation of fracturing is to enable an assessment to be made of progressive fracture development and evolving stability of tabular excavation sequences. Two particular examples are considered to illustrate the performance of the prototype test code for the simulation of three-dimensional fracture formation. The first example considers the simple case of the junction between two longwall mining panels where the face of one panel “leads” the second panel by a distance of 6 m. Random seed points are specified in the hangingwall region surrounding the lead-lag configuration. Special circular “edge seed” elements are also allowed to initiate from specified positions on the edge of the excavation. All seed elements have a fixed radius of 1 m and are initiated sequentially. The friction angle is assumed to be 10 degrees and the cohesion at the seed point is assumed to be zero. A series of 120 growth steps are simulated resulting in the fracture pattern shown in Figure 4.1. (The fractures are viewed in plan, in the direction perpendicular to the plane of the excavation, together with the projection of the background seed point positions, designated by small crosses). Quite extensive fracturing is observed ahead of the stope faces with a predominance of fracturing occurring ahead of the lagging face. It is also of interest to observe the general fracture direction “curving” through the lead-lag junction. This pattern is qualitatively similar to fracture patterns observed in small-scale laboratory tests such as those in SIMRAC project GAP 601b (Napier et al., 2002). Figure 4.2 shows the fracture pattern viewed in a direction parallel to the longwall panel faces. The fracture dip directions, parallel to the panel faces, tend to be inclined into the solid region ahead of these faces. These orientations depend on the fracture initiation mode (assumed to be shear in the present case) and on the assumed intrinsic friction angle when failure is initiated. The simulation was repeated using seed points with a uniform random distribution of radius values ranging from 0.25 m to 1.5 m. The results are shown in Figure 4.3 and Figure 4.4 and are very similar to the case with a uniform seed radius shown in Figure 4.1 and Figure 4.2, except that the fracture pattern seems to follow the lead-lag face direction changes more closely.

It is also of interest to examine the effect of the constitutive properties of the material in relation to the fracture initiation pattern. A further simulation run was carried out using an intrinsic cohesion value of 12 MPa and friction angle of 30 degrees. A residual friction angle of 30 degrees was specified and the cohesion breakdown was assumed to follow a linear decay law, with respect to the slip extent, having a slope of 6 000 MPa/m. The results of this simulation are shown in Figure 4.5 and Figure 4.6. It is apparent that more fractures are now initiated over the mined region and that the dip angle of the fractures is altered significantly.
Figure 4.1 Development of fractures initiated at fixed-radius seed growth points ahead of a lead-lag panel mining configuration – plan view.

Figure 4.2 Development of fractures initiated at fixed-radius seed growth points ahead of a lead-lag panel mining configuration – face-parallel view.
Figure 4.3 Development of fractures from random seed points with a random distribution of radius values ranging from 0.25 m to 1.5 m (plan view).

Figure 4.4 Development of fractures from random seed points with a random distribution of radius values ranging from 0.25 m to 1.5 m (face-parallel view).
Figure 4.5 Fracture pattern developed with fixed radius seed elements and with cohesion and slip-weakening values specified on the activated crack elements (plan view).

Figure 4.6 Fracture pattern developed with fixed radius seed elements and with cohesion and slip-weakening values specified on the activated crack elements (face-parallel view).
As a second example, consider the progressive development of fracturing as a crush pillar is formed by extending a given mining face adjacent to the pillar in a series of mining steps. A face advance of 2 m is used in each step and the seed point intrinsic cohesion and friction values are again assumed to be 12 MPa and 30 degrees respectively. The cohesion weakening slope is assumed to be 6 000 MPa/m. The seed radius is fixed and chosen to be 1 m. In addition, fixed vertical elements are specified from 0 m to 2 m above the reef plane, along the edges of the successive face positions in an attempt to simulate near-face failure effects. The development of the progressive fracture pattern, as the crush pillar emerges from the stope face, is depicted in the sequence of plan-view diagrams shown in Figure 4.7 to Figure 4.11. As the mining steps advance, additional fractures are developed ahead of the active mining face and additional fractures are observed to form over the solid region occupied by the emerging pillar. However, no evidence is given of a sudden acceleration of fracturing at a particular mining step, corresponding to a critical length of the pillar. This is probably due to the rather coarse size of the seed cracks (radius of 1 m), relative to the width of the formed pillar (3 m). The simulation of edge damage using the vertical elements is also unsatisfactory and precludes the detailed development of local fracturing above the pillar. Further effort is required to investigate the effect of seed size distributions and improved face edge fracture simulation.

![Crush pillar (Step 0)](image-url)

*Figure 4.7  Progressive pattern of fracturing in the formation of a crush pillar – initial face position.*
Figure 4.8 Progressive pattern of fracturing in the formation of a crush pillar – mining step 1.

Figure 4.9 Progressive pattern of fracturing in the formation of a crush pillar – mining step 2.
**Figure 4.10** Progressive pattern of fracturing in the formation of a crush pillar – mining step 3.

**Figure 4.11** Progressive pattern of fracturing in the formation of a crush pillar – mining step 4.
4.1.5 Conclusions on 3D fracture growth models

This chapter gives a broad outline of a novel method that has been proposed and evaluated for the simulation of three-dimensional fracture growth processes near the edges of tabular mine excavations. The evaluation of the method has required the construction of a prototype test computer program. The test code provides the capability of nucleating a series of crack growth elements at designated positions in space (seed points). The slip and opening displacement components arising on nucleated crack elements are solved using an iterative procedure. It is found that this approach is able to simulate the non-trivial formation of a coherent shear band structure and can also provide an indication of plausible fracture pattern orientations that arise near the edges of simplified tabular layout configurations. This is illustrated by the application of the procedure to the genesis of fracturing near a lead-lag stope panel configuration and to the evolution of fracturing in the vicinity of an emerging crush pillar. Many further examples can be envisaged such as the generation of fracturing in the vicinity of a mined remnant pillar or the formation of damage zones between superimposed pillars on parallel reef planes.

Although the proposed method for fracture growth simulation exhibits many encouraging features, a number of limitations must be noted. In particular, unstable (non-converging) iterative behaviour can arise when crack growth elements are initiated in unfavourable positions. Some checks have been implemented in the code to limit the proximity of interacting elements. General rules and improvements to the iterative procedure are still required. More importantly, it would seem that the nucleation of fixed-size crack elements at random positions in space, is too restrictive to allow damage structures to evolve fully. This restriction can possibly be addressed by employing very dense sets of interacting, variable sized crack elements. The testing of this possibility is computationally extremely demanding and will require improvements to the currently implemented iterative approach. An alternative strategy is to consider the possibility of allowing extended growth of each crack element from the initial fracture position.

The current test code allows for the future investigation of alternative 3D space filling tessellations such as the use of Voronoi polyhedra, provided suitable element mesh generators are available. However, more radical possibilities should also be investigated such as the explicit simulation of evolving, curvilinear crack front growth and intersection.

4.2 Other forms of integration

During the course of this project, a number of other aspects relevant to stability and integration have been studied. This section covers four of these areas which could contribute significantly to the future of integrated modelling. These are: optimisation processes for automated integration; integration of seismic observations with dynamic numerical modelling; improving seismic analysis through integration with dynamic numerical models; and integrating modelling with active seismic measurements.

4.2.1 Optimisation processes for integration

Integration of numerical modelling with seismicity, as well as with other underground observations, can be considered as an optimisation problem. Seismic data provides information about the rockmass that is very useful in determining the strength and the effect of mining-induced stresses. However, the seismicity is historical and has to have been observed previously. The data cannot easily be applied to plan future mining and to determine the effect on the rockmass because the mining will have to be performed to generate the seismicity. Numerical modelling is able to provide a means for evaluating different mining scenarios without having to actually mine the ore body. However, the ability of the model to correctly predict the mining-induced hazard will strongly depend on the formulation of the model, and the correct choice of input parameters. Integration is a procedure that will, hopefully, align the model input parameters to the real rockmass using the historically observed data to provide predictions that may be closer to the actual behaviour. It can then be assumed that, at least for a short time
period, the forward predictions of the model will reflect the future observations more closely. Thus, the first step of the integration process is an optimisation of the model parameters to minimize the difference between the modelling of past mining and the seismic data observed during that mining. This is usually done by brute force methods of selecting various “typical” parameters and adjusting these between runs. However, a proper integration scheme should have the back-prediction built in and should be able to run variations of models and automatically find the best fit parameters. This would be done using an optimisation process.

There are many different optimisation methods. Cherkaev (2001) discusses classical direct search-for-optimum methods, such as Golden Mean, Conjugate Gradients, Modified Newton Method, methods for constrained optimisation, including Linear and Quadratic Programming, genetic algorithms that mimic evolution, and stochastic algorithms that account for uncertainties in mathematical models. Genetic algorithms and simulated annealing methods were considered in this study. More details of the procedures and examples used to illustrate the concepts are provided in Appendix B.

To define the optimisation problem, a fitness or cost function must be determined that is to be minimized. In many optimisation processes, the function to be optimised is known implicitly. However, in the integration process, the seismic input and the modelled output depend on the way in which the mining is carried out, the type of model, and the resolution of the data. Thus, it is necessary to explicitly evaluate the fitness function for each option of mining. This may prevent the application of many optimisation procedures, especially those that require knowledge of the gradient of the fitness function to predict more optimal solutions.

A genetic algorithm approach may be useful to perform optimisation without an explicit fitness function. It mimics an evolutionary process where a sample population of potential solutions is selected and tested against the fitness function. The most suitable solutions are then “bred” in order to provide new solutions. This process continues for a number of generations, with some random mutation allowed so that other possible solutions may arise. To test the genetic algorithm, a simple model of a stope was developed, based on the site studied in GAP 604 (Figure 4.1a). A computer code was written which could modify the input file for MINF to change the Young’s modulus and strength parameters. The code was linked to a genetic algorithm code so that the genetic algorithm could pass new input parameters into MINF and then launch MINF. The output from MINF is then read by the genetic algorithm code. The genetic algorithm must maximize a function so that the selected cost function is the negative of the least-squares norm of the difference between the predicted and the observed values. The observed data is considered to be the closure at a point in the model and the genetic algorithm was able to find an optimal solution (Figure 4.1b), which lies within the input resolution.

Considerably more work is required to extend this optimisation strategy to the full integration problem. One area where genetic algorithms would be especially powerful is in the determination of non-uniform material properties. For example, by modelling a fault as a plane of displacement, the friction angle and cohesion could be varied locally to simulate various frictional regimes and asperities.

With the genetic algorithm approach, the run time for each trial layout increases significantly with decreasing strength. Each seismic event is considered sequentially, so increased seismicity increases the run time (Figure 4.2a). Thus, a “synthetic annealing” optimisation approach may be more efficient computationally. This approach considers a sequence of runs where the intact strength is decreased by some specified interval and the other parameters are varied within certain ranges until there is a match between observed and predicted behaviour. A simulated annealing approach was implemented but not tested on actual data. Initial studies suggested that the use of cumulative moment alone will not provide a unique solution since a situation with a low cap stress and low stress drop (more and smaller events) may produce the same moment as a model with a higher cap stress and higher stress drop, but fewer events. More complex fitness criteria are thus required.
Appendix B. A much more detailed analysis of integration via automated optimisation procedures is given in Section 4.2. The above constitutes a summary of the relevant issues, and a description of the best optimisation process, and the answer will depend on the type of model and data that is being used in the integration. Considerable research is also required to find the best compromise is to use simple models with few, but critical parameters that can be run fairly quickly. Considerable research is also required to find the best optimisation process, and the answer will depend on the type of model and data that is being used in the integration. The above constitutes a summary of the relevant issues, and a much more detailed analysis of integration via automated optimisation procedures is given in Appendix B.

Figure 4.1 (a) MINF model of a deep level stope showing the point (x) where the closure is compared. (b) Comparison of average fitness (solid line) of the solutions (diamonds) with number of generations.

Figure 4.2 (a) Effect of intact strength on run time of analysis. b) Effect of intact strength on cumulative moment in MINF simulations of seismic activity.
4.2.2 Integration of seismic observations with dynamic numerical modelling

A currently important theme in rock engineering is to integrate the information provided by observed seismic activity with numerical models which can assess the rockmass response to future mining scenarios. This optimisation process aims to improve the prediction of rockmass behaviour around a mining excavation (Mendecki et al., 2001; Spottiswoode, 2001). The simplest integration methods apply visual comparisons of contour plots or differential maps derived from numerical models against those derived from seismic interpretation software (e.g. Lachenict et al., 2001, and other papers at the same symposium). More advanced, “passive”, methods permit the seismic event to be applied as an induced load or deformation at the observed location (e.g. Lachenict et al., 2001, Sellers & Napier, 2001). These methods are very sensitive to the location accuracy and require assumptions about the magnitude and direction of the co-seismic deformation (Sellers & Napier, 2001). Alternatively, “active” methods such as the Point Kernel approach (Sellers & Napier, 2001) and the MINF cap model (Spottiswoode, 2001) allow the model to produce the seismicity and repeatedly alter the model input parameters to match the observed seismicity, However, these methods depend strongly on the way in which the seismicity is simulated and on the frequency of the comparisons.

The above studies of integrating modelling with seismicity have considered seismic events as quasi-static loading, i.e. applied as a time-independent deformation, with the only consideration of time being the inclusion of mining steps at daily, weekly or monthly intervals. However, seismicity and rockbursts are dynamic events that cause the propagation of seismic waves through the rockmass and can induce rockburst damage in excavations that are often distant from the source (Durrheim et al., 1997). Thus, it would seem that the concept of integration should be extended to include the dynamic characteristics of seismic events and rockbursts. This section covers preliminary investigations into appropriate criteria that are required for a dynamic integration procedure. It investigates the manner in which inverted seismic parameters such as seismic moment, stress drop or moment tensor solutions should be interpreted in order that a numerical code can properly predict the dynamic rockmass response. To minimize the difference between observations and predictions, it is necessary to properly understand how to relate the model output to the seismic observations of the ground dynamics. These issues were investigated using case studies comparing modelled elastodynamic ground motions and with recorded ground motions from events on South African gold mines. Details were presented in Kataka et al. (2003) and results are summarised in this section.

A key issue in the simulation of seismic activity relates to the accuracy and the interpretation of the seismic source location, dimension, magnitude and deformation. Studies of seismic data from underground stopes have highlighted that there can be considerable deviations in the source parameters inverted from different geophone positions. Seismic techniques are used to select the most likely parameters, but this poses difficulties in matching observed and modelled waveforms. The seismic events for a three-year period on a series of tabular stopes in a deep level gold mine in South Africa were studied to evaluate the variability of the data to determine its suitability for input into passive integration schemes. The average location inaccuracy of 17 m is equivalent to 2 months face advance and illustrates that the events may be reported to be some distance from where they actually occurred. Clearly, significant care is needed in applying the seismic events in a numerical model using a “passive” integration approach, where the model is physically altered by the induced seismicity. Study of the data indicates that the deviation of the observed moment, source radius and stress drop, which are vital for determining the input loading, can be as great as the value of the parameter itself. Fortunately, the errors are lower for the larger events, which are associated with the most damage. Errors in location, magnitude and direction can lead to incorrect assumptions about the input deformations and hence force the model further from, not closer to, the future reality.

Numerical investigations into the issues connected with dynamic integration were made using the elastodynamic finite difference program WAVE (Hildyard, 2001). These provided a better understanding of the passive integration approach for improving the ability of numerical models...
to reproduce observed dynamic rockmass behaviour. A study of a vertical slip event in a square domain (with and without a tabular stope), as shown in Figure 4.1a indicated that the modelled seismogram is sensitive to the grid size, the distance to the source and the source model. The source-time model was either an instantaneous step function or an S-curve, and the amplitude and rise time were varied. The effect of rise time is shown in Figure 4.1b. The observed source parameters and moment tensors can be back calculated to some extent from the model output. The models showed that seismic inversion for the source differed when waveforms included the effects of the mining layout. The effect of further geological complexity and fractured rock increases the waveform complexity, as is observed underground. Thus, an iterative approach using such a model with explicit fracturing may increase the accuracy of the seismic inversion and hence improve the success of the "passive" type of integration.

The program WAVE was then used to simulate an observed seismic event in a limited portion of the mine, as shown in Figure 4.2a, to assess how closely the simulated waveform matched the observed waveform. For dynamic integration it is important to ensure that the numerical wave compare well with the observed seismograms in both the time and frequency domains, and Figure 4.2b shows the comparison in the frequency domain. This event was recorded on the mine seismic system and on a triaxial geophone in a small-scale seismic system attached to the hangingwall of the adjacent stope (Andersen (now Linzer) et al., 2002). Only the small-scale system output could be considered due to the limited model grid size.

![Figure 4.1](image)

**Figure 4.1** a) WAVE Model geometry b) effect of source rise time on waveform in near-field for S-shaped source time history.
These preliminary studies indicated that it is important to represent the rockmass with a sufficiently fine grid, and to correctly represent the source dimensions, frequency content and slip directions in order to obtain meaningful correlations with observed seismological parameters. Fracturing surrounds the stope and in order to properly model the dynamic effects in the stope, the model must include a fracture model that correctly affects the wave propagation. In back analyses of laboratory scale test samples containing fractures and in well-controlled in-situ experiments, Hildyard (2001) found that the observed waveforms could be well-reproduced. However, when applied to a larger scale underground experiment simulating a rock burst using a controlled blast (Hagan, 2001), the simulated and recorded waveforms compared well for a small calibration blast but not for the large blast (Hildyard and Milev, 2001). The differences were attributed to problems in data acquisition, a poor source model, or the inelastic behaviour of the medium. Thus, these variations indicate that the dynamic integration process will not necessarily be unique and will depend strongly on the input source, and the modelling of damage and fracturing.

Integration techniques for correlating the observed mine seismicity and the stress state predicted by numerical models require quantitative methods and criteria for determining the success of the match that must still be developed. Currently, however, a dynamic integration procedure that can directly input observed mine events to condition the modelled rockmass is impractical due to the large amount of data, poor understanding of source models and nonlinear effects such as damage and fracturing. The difference in time scales between the daily face advances, monthly planning updates and millisecond event durations must also be overcome. Memory and runtime constraints also limit the turnaround time and hence the practical application to large-scale integration. The less routine application of these models in comparison with recorded data from selected events remains important, as it provides a method of understanding the near-field behaviour and damage, which cannot be provided by the far-field observations. Further calibration studies and case histories should be conducted to develop the understanding of dynamic rockmass behaviour. An active method that permits dynamic failure of the rockmass and induced damage to accumulate as a result of the propagation of high amplitude seismic waves through the rockmass, coupled with regular comparison to the observed seismicity, must still be investigated as an integration approach.

4.2.3 Improving seismic analysis through integration with dynamic numerical models

The previous section shows that significant developments are required before simulated waveforms can routinely match measured waveforms. Nevertheless, less ambitious forms of dynamic integration could have significant benefit. In particular, using dynamic modelling in conjunction with seismic recordings, has the potential to improve seismic analyses and inversions as well as providing a means of determining what these far-field recordings indicate about the near-field region.

In the integration methodology of Chapter 3, the only seismic parameters used are the rate and distribution of seismicity, moment and energy, and the locations. Section 3.4 even proposes shifting seismic locations to mining faces due to errors in the seismic locations. Other inverted seismic parameters such as source size, stress drop and mechanism (moment tensor), could usefully be integrated with models, particularly explicit fracture models. However, the uncertainties in these inversions are considered to large for current integration strategies.

One area where dynamic models could improve these inversions of seismic parameters, is to establish how different aspects of the medium affect the accuracy of different seismic inversions.
– for example, the influence of the stope layout or fracturing. It is feasible to create simulations in models, and apply the same routines used to invert recorded far-field waveforms, to invert the simulated waveforms. Since, (in this case) the source mechanism is known in detail, the accuracy of the seismic inversions can be evaluated, or an understanding gained of what the seismic inversions reveal. This could lead to improved inversion methods or at least increased confidence in the seismic inversions.

The models first need to be tested on a purely homogenous solid medium, to ensure that the simulated waveforms are suitable for seismic inversion – i.e. if modelled waveforms in an homogenous medium cannot be correctly inverted, then this probably indicates limitations in the model. (For example, a problem with the frequency content, or a problem with the model not covering a sufficiently large volume). Conversely, if these simple cases can be correctly inverted, then the models can be used to evaluate the effects of more complicated geometries, or more complex media, on seismic inversions. These investigations are in their infancy, but it can be envisaged that, in the longer term, such modelling could form an essential part of an integrated seismic inversion process.

As a first study, this work investigated the influence of the stope on seismic inversions of the scalar moment, the source radius, stress drop and moment tensor mechanism. This was evaluated for both a slip and a crush mechanism. The source parameters were computed from waveforms 'recorded' by 'triaxial geophones' placed spherically around the pillar source. An important practical objective was to determine whether the source parameters calculated from recorded waveforms are due to a combination of the stope and pillar sources, rather than being only related to the pillar.

### 4.2.3.1 Description of models

Models were constructed of punch- and crush-type pillar failure. The influence of the stope was isolated by either including or removing the stope effect in the models. In all of the models, the pillar had dimensions of 20 m x 20 m and was positioned within a 60 m x 60 m stope. In the punch models, a slip-plane having dimensions 20 m x 20 m was explicitly included in the model. The four models are described in Table 4-1.

#### Table 4-1  Model description.

<table>
<thead>
<tr>
<th>Type</th>
<th>Name</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>Punch</td>
<td>1A</td>
<td>No stope, 20 m x 20 m vertical fault.</td>
</tr>
<tr>
<td>Punch</td>
<td>1B</td>
<td>Stope, 20 m x 20 m vertical fault situated on the edge of pillar, fault-plane daylights into horizontal stope.</td>
</tr>
<tr>
<td>Crush</td>
<td>2A</td>
<td>No stope, 20 m x 20 m with vertical stress drop.</td>
</tr>
<tr>
<td>Crush</td>
<td>2B</td>
<td>Stope, 20 m x 20 m with vertical stress drop in centre of stope.</td>
</tr>
</tbody>
</table>

Figure 4.1 shows the geometry of the models. The models used a 360 x 360 x 360 grid (47 million grid points), having a grid spacing of 1 m. The punch models consisted of a vertically oriented slip plane with dimensions of 20 m x 20 m. A shear stress drop of 50 MPa, with a rise-time of 1.1 ms, was imposed over the slip plane. The rupture initiated at a central point at the top of the slip plane, and propagated along the plane at a rate of 90 per cent of the S-wave velocity. In the model with a horizontal stope, the slip plane daylighted the stope. The crush-pillar models imposed a similar 50 MPa stress drop in vertical stress over a 20 m x 20 m region. The rupture initiated at the corner of this region and propagated at 90% of P-wave velocity. For the model with a horizontal stope, the region of stress drop was situated in the centre of the stope.
Forty-eight triaxial geohones were placed around the source centre, half of them at 100 m from the source and the other half at 160 m from the source. The geophones were positioned to provide complete coverage of the focal sphere in three dimensions, as far away as possible from the source so that recordings could be made in the far field. Seismic inversions typically neglect the near-field terms. However, seismic recordings are sometimes thousands of metres from the source, which is currently not feasible in the models. Nevertheless, a distance of 160 m is approximately 16 times the source radius, and the near-field contribution would normally be considered negligible.

![Figure 4.1 Model of the fault-plane, pillar and stope. The dashed lines indicate the model boundaries.](image)

### 4.2.3.2 Calculation of source parameters

The seismic moment $M_0$ is calculated for each geophone site $i$, channel $j$ and wave-phase $k$ using:

$$M_{0ijk} = 4\pi \rho v^3 R \frac{\Omega_{0ijk}}{f_{bijk}}$$

[4.2]
where the $\rho$ is the density of the rockmass (using the average value of 2700 kg/m$^3$), $v$ is the wave-speed of the P- or S-phase (using $v_1 = 6610$ m/s for P-waves and $v_2 = 3865$ m/s for S-waves), $R$ is the hypocentral distance, $f_{\theta \phi}$ accounts for the radiation pattern ($f_{\theta \phi_1} = 0.37$ for P-waves and $f_{\theta \phi_2} = 0.57$ for S-waves (Spottiswoode & McGarr, 1975) and $\Omega_0$ is the integrated displacement, $\Omega_0 = \int u dt$, calculated from the low frequency displacement spectral plateau for windows over the P- and S-phase. Equation [4.2] is based on the formulation given in McGarr (1984), but the notation has been extended to include the site, channel and wave-phase indices.

The moment for each phase $k$ of the event, for each geophone site $i$, is the vector sum of the moments calculated for each channel:

$$M_{0ik} = \sum_{j=1}^{3} M_{0ijk}$$

[4.3]

The total moment for an event is the sum of its P- and S-moments, averaged over all geophone sites:

$$M_0 = \frac{1}{2N} \sum_{i=1}^{N} \sum_{k=1}^{2} M_{0ik}$$

[4.4]

where $N$ is the number of active geophone sites.

The corner frequency was read off the velocity spectrum. Ideally, the corner frequency would be picked at the positions marked in Figure 4.1. However, in practice, the corner frequency can be difficult to recognise from synthetic seismograms because of the trade-off between grid size and frequency content of the signal.

![Figure 4.1](image)

**Figure 4.1** Ideal shapes of displacement, velocity and acceleration spectra showing the corner frequency.

The source size and stress drop calculations are based on the Brune model (Brune, 1970, 1971). The source radius $r$ is calculated from the corner frequency, $f_o$, using:

$$r = \frac{k \beta}{2\pi f_o}$$

[4.5]

where $\beta$ is the shear wave velocity and $k$ is a constant equal to 2.34 (Brune, 1970).

Finally, the stress drop $\Delta\sigma$ is calculated from
\[ \Delta \sigma = \frac{7}{16} \frac{M_0}{r^3} \] \hspace{1cm} [4.6] 

where \( M_0 \) is the total moment of the event (Equation [3]).

The Hanks-Kanamori Moment-Magnitude relationship was used to compute the magnitude \( m \) from the scalar moment \( M_0 \):

\[ m = (\log_{10} M_0 - 9.1)/1.5 \] \hspace{1cm} [4.7] 

where the units of \( M_0 \) are Nm.

The moment tensor of each event was computed using the Moment Tensor Inversion Toolbox (MTIv5.exe) written by Lindsay Linzer at CSIR Miningtek (Andersen, 2001). Absolute (i.e. single event) moment tensor inversion methods were applied. The input data consisted of the moments for each site, channel and wave-phase, calculated using Equation [4.2].

The seismic moment tensor \( \mathbf{M} \) provides a set of point forces that describes the source mechanism and allows the slip-plane orientation and direction to be determined. The moment tensor \( \mathbf{M} \) is defined by a combination of force couples and dipoles as:

\[
\mathbf{M} = 
\begin{bmatrix}
M_{xx} & M_{xy} & M_{xz} \\
M_{yx} & M_{yy} & M_{yz} \\
M_{zx} & M_{zy} & M_{zz}
\end{bmatrix}
\] \hspace{1cm} [4.8] 

where each element \( M_{ij} \) of the matrix represents the force couples or dipoles as shown in Figure 4.2.

**Figure 4.2** Representation of the nine possible couples of the moment tensor \( \mathbf{M} \). The directions of the force and arm of the couple are denoted by the indices \( i \) and \( j \), respectively (from Aki & Richards, 1980).

Since the moment tensor is symmetrical due to considerations of static equilibrium, only six of the nine components need to be determined. The symmetry also allows the moment tensor to
be diagonalised and described uniquely in terms of its corresponding eigenvectors and eigenvalues (Jost & Hermann, 1989):

\[
M = \begin{bmatrix}
a_{1x} & a_{2x} & a_{3x} \\
a_{1y} & a_{2y} & a_{3y} \\
a_{1z} & a_{2z} & a_{3z}
\end{bmatrix}
\begin{bmatrix}
e_1 & 0 & 0 \\
e_2 & 0 & 0 \\
e_3 & 0 & 0
\end{bmatrix}
\begin{bmatrix}
a_{1x} & a_{1y} & a_{1z} \\
a_{2x} & a_{2y} & a_{2z} \\
a_{3x} & a_{3y} & a_{3z}
\end{bmatrix}
\]

[4.9]

where the elements \( e_1, e_2 \) and \( e_3 \) are the eigenvalues of the moment tensor \( M \), and the eigenvectors are of the form \( (a_{kx}, a_{ky}, a_{kz}) \). The three eigenvectors represent the axes of the source along which the principal forces act. The sizes of the eigenvalues denote the magnitude of these forces, and the sign indicates their direction (conventionally, a negative force points towards the source).

The diagonalised moment tensor \( M_D \) in Equation [4.9] is:

\[
M_D = \begin{bmatrix}
e_1 & 0 & 0 \\
0 & e_2 & 0 \\
0 & 0 & e_3
\end{bmatrix}
\]

[4.10]

To aid interpretation, the diagonalised moment tensor, \( M_D \), is usually decomposed into an isotropic part (equivalent to the product of a scalar and the identity matrix), and a deviatoric part:

\[
M_D = \frac{1}{3} \begin{bmatrix}
tr(M) & 0 & 0 \\
0 & tr(M) & 0 \\
0 & 0 & tr(M)
\end{bmatrix}
+ \begin{bmatrix}
e_1 - \frac{1}{3}tr(M) & 0 & 0 \\
0 & e_2 - \frac{1}{3}tr(M) & 0 \\
0 & 0 & e_3 - \frac{1}{3}tr(M)
\end{bmatrix}
\]

[4.11]

\[
M_D = \frac{1}{3} \begin{bmatrix}
tr(M) & 0 & 0 \\
0 & tr(M) & 0 \\
0 & 0 & tr(M)
\end{bmatrix}
+ \begin{bmatrix}
e^*_1 & 0 & 0 \\
0 & e^*_2 & 0 \\
0 & 0 & e^*_3
\end{bmatrix}
\]

[4.12]

where \( tr(M) \) is the trace of the moment tensor, and is equal to the sum of the eigenvalues:

\[
tr(M) = e_1 + e_2 + e_3
\]

[4.13]

The trace is a measure of the volume change at the source and the sign of \( tr(M) \) gives the direction of motion relative to the source with positive outwards. For example, a \( tr(M) \) that is negative indicates an implosion.

The deviatoric term is calculated by subtracting \( \frac{1}{3}tr(M) \) from each of the eigenvalues:

\[
e^*_k = e_k - \frac{1}{3}tr(M)
\]

[4.14]

where \( e^*_k \) are the deviatoric eigenvalues. The first term on the right hand side of Equation [4.12] describes the isotropic part of the moment tensor, which corresponds to volume changes in the
medium. The second term describes the deviatoric part of the moment tensor, which corresponds to the non-volume-change slip movements in the medium.

The percentage contribution of the isotropic component, \( \%ISO \), to the full tensor can be calculated from the trace of the normalised Euclidean moment tensor (i.e. the eigenvectors are orthonormal and have a length equal to 1) using the formulation given by Silver & Jordan (1982):

\[
\%ISO = \frac{tr(M)}{tr(M)} \cdot \frac{M_I^2}{M_T^2}
\]  

[4.15]

where \( M_T \) is the scalar moment of the moment tensor and is computed from the nine moment tensor elements using:

\[
M_T = \frac{1}{2} \sqrt{\sum_{i=1}^{3} \sum_{j=1}^{3} M_{ij}^2}
\]  

[4.16]

and \( M_I \) is computed from the trace:

\[
M_I = \frac{tr(M)}{\sqrt{6}}
\]  

[4.17]

The deviation, \( \varepsilon \), of the seismic source from that of a pure double-couple is expressed by Dziewonski et al. (1981) as the ratio of the minimum to maximum deviatoric eigenvalue:

\[
\varepsilon = \frac{e_3^*}{e_1^*},
\]  

[4.18]

where \( |e_1^*| \geq |e_2^*| \geq |e_3^*| \). \( \varepsilon = 0 \) for a pure double-couple source, and \( \varepsilon = 0.5 \) for a pure compensated linear vector dipole (CLVD). The percentage of double-couple contributions, \( \%DC \), to the deviatoric moment tensor can be calculated from \( \varepsilon \) using:

\[
\%DC = 100 \cdot (1 - 2\varepsilon)
\]  

[4.19]

as described in Jost & Hermann (1989). The percentage of CLVD contributions, \( \%CLVD \), to the deviatoric moment tensor can be calculated from \( \varepsilon \) using:

\[
\%CLVD = 200 \varepsilon
\]  

[4.20]

(Jost & Hermann, 1989). The sum of \( \%DC \) and \( \%CLVD \) should be 100.

The R-ratio is another measure of the nature of the moment tensor introduced by Feigner & Young (1992). Ratio R is defined as:

\[
R = \frac{tr(M)}{\left| tr(M) + \sum_{k=1}^{3} e_k^* \right|} \cdot 100
\]  

[4.21]
where \( e_i^* \) are the deviatoric eigenvalues (Equation [13]). If \( R > 30 \), the event is considered to be tensile; if \(-30 \leq R \leq 30\), the event is a shear event; if \( R < -30 \) the event is implosive.

The diagonalised deviatoric moment tensor can be decomposed into a variety of other eigenvalue combinations representing simple arrangements of equivalent forces. These decompositions are used to characterise how well an equivalent body-force system of the moment tensor describes a particular seismic source. However, these decompositions are not unique and do not generally correspond to any readily recognisable physical attributes of the source. The simplest decomposition is therefore used in this study, i.e. the isotropic-deviatoric decomposition (Equation [4.11]). The other values used are the scalar measures described by Equations [4.19] and [4.20].

Finally, the orientation and sense of slip of two planes fitted to the nodal planes of the radiation pattern are computed from the deviatoric moment tensor. This is termed the “fault-plane solution” and is described by strike, dip and rake. Two solutions are calculated for each deviatoric moment tensor because the nodal planes of a shearing source are orthogonal and produce indistinguishable radiation patterns, i.e. right-lateral motion on the fault plane would produce a P-wave radiation pattern indistinguishable from that which results from left-lateral slip on the auxiliary plane.

Calculating the parameters outlined above in this section is not as simple as the theory seems to imply because these equations are only valid in the far field (as a rule of thumb, the far field is at least six wavelengths away from the source). Conceptually, one requires the element size to be as small as possible to give good resolution and accuracy in the models, but simultaneously one requires the total grid size to span a large enough volume such that seismograms can be recorded sufficiently far from the source to be considered far field. Computer memory and computation speed are the limiting factors, and one has to reach a compromise between resolution and recording in the far field. As a result, the intermediate field is visible for the larger sources (e.g. a 60 m x 60 m stope) whereas the far field dominates for the smaller source models (e.g. a 20 m x 20 m fault slip source).

### 4.2.3.3 Results

Table 4-1 shows the results of the source parameter inversions computed from velocity seismograms recorded around the source. These parameters were calculated at 160 m from the source. Table 4-2 shows the source contributions calculated from the moment tensor of the pillar punch and pillar crush models.

#### 4.2.3.3.1 Pillar punch model source parameters

There are no significant differences between the moments and magnitudes calculated for the punch models, with and without a stope (Table 4-1). The main effect of the stope is to introduce higher frequencies into the seismograms. This is evident in the shift of corner frequency to a higher value when the punch model with no stope (1A) is compared with the punch model with a stope (1B). Figure 4.1 shows two velocity seismograms, recorded at the same geophone, illuminating the difference in frequency content.

Before discussing the source radius and stress drop parameters computed for the modes, it is important to mention that these calculations are highly dependent on the corner frequency. Selecting the corner frequency from the seismograms was difficult and ambiguous, and as a result, parameters dependent on accurate corner frequencies should be viewed as unreliable.

Source areas computed from the source radii are 568 m² and 302 m² for models 1A and 1B respectively (equation used: \( area = \pi r^2 \)). The actual source models had an area of 441 m². The stress drops calculated for the punch models were 58 MPa and 110 MPa. The actual stress drop applied in the model was 50 MPa.
Table 4-1  Source parameters.

<table>
<thead>
<tr>
<th>Model description</th>
<th>Mo</th>
<th>Mag.</th>
<th>fo</th>
<th>r</th>
<th>Δσ</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>[N.m]</td>
<td>[Hz]</td>
<td>[m]</td>
<td>[MPa]</td>
<td></td>
</tr>
<tr>
<td>Punch 1A No stope</td>
<td>3.23E+11</td>
<td>1.61</td>
<td>107.1</td>
<td>13.4</td>
<td>58</td>
</tr>
<tr>
<td>1B Stope</td>
<td>2.44E+11</td>
<td>1.52</td>
<td>146.9</td>
<td>9.8</td>
<td>110</td>
</tr>
<tr>
<td>Crush 2A No Stope</td>
<td>7.42E+11</td>
<td>1.85</td>
<td>146.8</td>
<td>9.8</td>
<td>340</td>
</tr>
<tr>
<td>2B Stope</td>
<td>1.45E+12</td>
<td>2.04</td>
<td>72</td>
<td>20.0</td>
<td>79</td>
</tr>
</tbody>
</table>

Mo = Moment (calculated using Equations [4.2], [4.3] and [4.4])
Mag. = Moment magnitude calculated from Moment using Equation [4.7]
fo = Corner frequency calculated from the velocity spectra
r = source radius calculated from the velocity spectra
Δσ = Stress drop calculated using Equation [4.6].

Table 4-2  Source contributions.

<table>
<thead>
<tr>
<th>Model description</th>
<th>%ISO</th>
<th>%DC</th>
<th>%CLVD</th>
<th>R-ratio</th>
<th>Strike</th>
<th>Dip</th>
<th>Rake</th>
</tr>
</thead>
<tbody>
<tr>
<td>Punch No stope (1A)</td>
<td>0.4</td>
<td>90.9</td>
<td>9.1</td>
<td>7.0 (shear)</td>
<td>(a) 101.8°</td>
<td>(b) 271.6°</td>
<td>(a) 2.0°</td>
</tr>
<tr>
<td></td>
<td>0.1</td>
<td>98.5</td>
<td>1.5</td>
<td>-2.1 (shear)</td>
<td>(a) 199.0°</td>
<td>(b) 92.5°</td>
<td>(a) 4.5°</td>
</tr>
<tr>
<td>Crush No Stope (2A)</td>
<td>39.6</td>
<td>29.0</td>
<td>71.0</td>
<td>-46.5 (implosive)</td>
<td>(a) 291.2°</td>
<td>(b) 127.9°</td>
<td>(a) 43.4°</td>
</tr>
<tr>
<td></td>
<td>96.7</td>
<td>53.9</td>
<td>46.1</td>
<td>-85.7 (implosive)</td>
<td>(a) 66.8°</td>
<td>(b) 160.4°</td>
<td>(a) 61.9°</td>
</tr>
</tbody>
</table>

%ISO = percentage of isotropic (volume change) component to the full tensor calculated using Equation [4.15].
%DC = percentage of pure double-couple contribution to the deviatoric (non-volume change) moment tensor, calculated using Equation [4.19].
%CLVD = percentage of compensated linear vector dipole to deviatoric moment tensor, calculated using Equation [4.20]. %DC and %CLVD should add up to 100%.
R-ratio = nature of the source, Equation [4.21].
Strike, Dip, Rake = fault-plane solution fitted to the deviatoric portion of the moment tensor, i.e. orientation of the nodal planes of the radiation pattern.
4.2.3.3.2 Crush pillar model source parameters

In contrast to the punch pillar models, there is a significant difference between the moments and magnitudes calculated for the crush models, with and without a stope (Table 4-1). The moment calculated for the crush source with no stope (2A) was $7.42 \times 10^{11}$ Nm (equivalent to a magnitude of 1.85) whereas the moment computed for the crush source with a stope (2B) was approximately twice as large, at $1.45 \times 10^{12}$ Nm (magnitude of 2.04). Inspection of the waveforms shows significant S-wave amplification in the crush pillar model where the effect of the stope is included (2B) (Figure 4.1).

There is also a shift in corner frequency towards the lower frequencies when the crush model with a stope (2B) is compared with the mode with no stope (2A). The corner frequency of model 2A was 147 Hz, whereas that of model 2B was 72 Hz. The lower corner frequency corresponds to a larger source area, indicating the stope is contributing to the total deformation. Source areas computed from the source radii are 302 m² and 1256 m² for models 2A and 2B respectively, whereas the actual source models had an area of 441 m². The stress drops calculated for the crush models 2A and 2B were 340 MPa and 79 MPa, respectively. The actual stress drop applied in the model was 50 MPa.

As mentioned previously, source parameters calculated from the corner frequency should be treated with caution because of the uncertainty in identifying the corner frequency.

4.2.3.3.3 Punch pillar mechanisms

The isotropic component of the moment tensor (i.e. the volume change component) for both punch models is very low (\%ISO < 1 \%) indicating a very small volume change at the source (Table 4-2). The source mechanisms of both models are dominated by the double-couple component (\%DC > 90 \% in both cases), indicating that the source mechanism is dominated by shearing. The R-ratio, another measure of the nature of the moment tensor, supports the shearing model since both R-ratios lie in the range of a shear event. (If $R > 30$, the event is considered to be tensile; if $-30 \leq R \leq 30$, the event is a shear event; if $R < -30$ the event is implosive).
Despite the stope introducing higher frequencies into the waveforms, the P- to S-wave ratios remained consistent, and the polarities were not severely affected, because the overall source mechanism for the models with and without the stope are very similar. The stope, however, does cause a slight change in the fault-plane solution. The fault plane solution of the model without a stope matches the WAVE model (i.e. dip-slip faulting with a vertical fracture plane having an EW strike), whereas the model with a stope shows a slightly poorer match. The radiation patterns and fault plane solutions of these models are shown in Figure 4.2.

4.2.3.3.4 Crush pillar mechanisms

The isotropic component of the moment tensor for the crush model without a stope (2A) was moderately high (\%ISO ~ 40 \%) whereas that of the crush pillar with a stope was extremely high (\%ISO ~ 97 \%, Table 4-2). This is associated with a large volume change at the source. The R-ratios of both crush models indicate implosive source models (\( R < -30 \)) indicates an implosion), indicating that closure in the vicinity of the source has taken place. This volumetric closure was higher for the crush pillar with a stope (2B) than that with no stope (2A).

The deviatoric component of the crush pillar with no stope (2A) was dominated by a CLVD mechanism (a force system where the compressional force has a relative magnitude of two and the two tensional forces each have a relative magnitude of one). CLVD sources are still a topic
of debate in the literature with opinion being divided over whether the CLVD can exist in reality or not. In contrast, the deviatoric component of the crush pillar with a stope (2B) consisted of almost equal portions of \( \%DC \) and \( \%CLVD \).

The fault-plane solutions of models 2A and 2B are significantly different from one another, and do not match the fault plane of the models, which is a pressurised horizontal crack for both cases. The radiation patterns and fault plane solutions of these models are shown in Figure 4.1. The overall mechanisms of both crush pillars are different, indicating the stope played a large role in the overall mechanism.

![Radiation patterns and fault plane solutions of crush pillar models 2A and 2B.](image)

**Figure 4.1** Radiation patterns and fault plane solutions of crush pillar models 2A and 2B.

### 4.2.3.4 Conclusions

Traditional seismic inversions were made from simulated waveforms from models with and without a stope – i.e. the moment, magnitude, source radius, stress drop, and moment tensor were calculated using standard techniques. Unfortunately, the waveforms were not sufficiently far field for such analyses (the assumption on which conventional seismic inversions are based), and the window lengths had to be set with caution. Near-field contributions were identifiable in the waveforms even for source models excluding the influence of the stope, while the frequency effects due to the stope should only become dominant at distances many times the stope span. Choosing the corner frequency for the waveforms at this distance was particularly difficult and unstable, and parameters heavily dependent on this (source size and stress drop) should be considered unreliable. In spite of the presence of the near-field contribution, the moment and magnitude inversions were found to be reasonably reliable.

In spite of the above limitations, the models showed that:

- For both slip and crush type mechanisms, the stope has a very significant influence on the seismograms, both in amplitude and frequency content.
- The inverted moment is larger due to the presence of the stope. This was unequivocal in the case of inversions for the crush source (magnitude 2.04 versus 1.85), but was not conclusive for the slip source.
- The inverted source size was much larger than the actual pillar size for the case of the crush model including the influence of the stope. This is a very important result, but should be treated with caution, as the source size calculation is heavily dependent on the choice of the corner frequency, which was not considered accurate.
- The inverted source mechanisms were affected by the presence of the stope. The case of the slip source showed a slight change in the fault-plane solution due to the stope. The case of the crush source showed a significant change in the fault-plane solution due to the stope.
In addition, the isotropic component (a measure of volumetric closure at the source) of the crush source was significantly higher in the case where a stope was included in the model.

It is believed that extending this study to provide waveforms at sufficiently far distances will more conclusively confirm that the far-field inversions are significantly influenced by the stope itself. This has fundamental significance for interpreting the underlying cause of a seismic event from far-field data. Obtaining truly far-field waveforms is not yet supported by the current Wave technology, although features to support this are being addressed.

4.2.4 Integrating modelling with active seismic measurements

Techniques to quantitatively measure the fracture zone are essential to the development of improved stability criteria. One reason is simply that the nature and state of fracturing around stopes and tunnels has a fundamental influence on their stability and hence on support requirements. A second reason, of great importance to this project, is that if advanced models of fracture zone evolution are to be developed and used, then measurements of fracture zones are essential to validate and improve such models. Measurement techniques are unfortunately under-developed, or at least not widely applied in mining rock mechanics. Of the methods available, seismic techniques are particularly attractive due to the direct mechanical interaction of seismic waves with fracturing. They are also attractive since, as will be shown, their interaction with fracturing can be well simulated within models.

While major advances are anticipated in the ability to model fracture zone development in 3D mining geometries, the question remains whether such model "predictions" can be tested and adapted against some form of measurements. Although seismicity rate and magnitude provide some measure of how the fracture models relate to observations, it is desirable that the models should match up in a number of ways, including mechanisms of failure, sizes and orientations of fractures, stress drop during failure, and, in particular, the density of fracturing. One possibility is to use seismic monitoring to identify the formation of fractures and the location and size of fractures, and to relate these to fracture development in the model. This would require a very high level of accuracy in the seismic inversions which would in turn require a dense, very localised network covering a volume of the order of metres to tens of metres, and an appropriately high frequency range. Such passive tomography is highly desirable for individual research experiments. However, it is not suited to regular measurement of fracture zone development.

An alternative is to use active velocity scans. Typically, these will not provide detailed information on individual fractures, but can provide information on the overall nature of the fracturing, including fracture density, orientation and size. In addition, the simulated fracture zones can, in fact, be probed in the same manner, through simulated velocity scans in dynamic models. This appears to be a convenient way in which simulated fracturing in mining geometries can be compared to, and modified to be similar to, that of real mining geometries.

One approach to applying active seismic monitoring would be to use high frequency ultrasonic tools being developed for use in nuclear waste depositories, such as the micro-velocity probe (Maxwell et al., 1998). This probe is used to log the variation in wave-speed along a borehole using ultrasonic sensors spaced up to 30 cm apart. It could for example be used to log the changes in velocity in the fracture zone of stopes, determining the degree of fracturing and the distance over which the fracture zone extends into the hangingwall. It would however be cheaper to use existing low frequency seismic monitoring tools and a CSIR-Miningtek project has been conducted to determine how successfully such tools can be applied, before investigating ultrasonic techniques.

A paper was written recently on the use of seismic wave propagation to diagnose the state of fracturing (Hildyard et al., 2005). The main aim of the paper was to demonstrate that active seismic measurements could have an impact in helping solve rock engineering problems experienced in the South African mining industry. The paper amalgamated work from a number
of sources such as Hildyard (2001), Simrac project GAP 601b, an EU funded project “Omnibus” (Pettitt et al., 2003), and a CSIR project on blocky hanging wall behaviour.

The above paper (Hildyard et al., 2005) presents a variety of experimental and numerical results where seismic waves show clear differences in wave-speed and amplitudes due to different degrees of fracturing. Results are shown from the “Omnibus” project which was commissioned to develop new ultrasonic technologies for diagnosing fracturing for nuclear waste repositories. In this work, models were used to investigate assemblies of cracks where the effects of crack density and crack size on waveforms are shown to be coupled. The results indicate that these effects can be decoupled in the frequency domain, where the Fourier amplitude aids in estimating crack size, while the low frequency phase-difference has a direct relationship to crack density. Models were also shown to aid in the interpretation of waveforms by isolating geometric effects from the effects due to cracks. Two rock engineering problems experienced in the South African mining industry were then investigated through numerical examples. These demonstrate that determining whether ‘crush’ pillars have failed and determining the degree of fracturing in the hangingwall of stopes, may be possible through the use of active seismic surveys.

This section presents the results of simulated velocity scans applied to two examples of rock engineering problems in South African mines. Hildyard et al. (2005) should be referred to for more detail. The first application is for velocity scans to help determine the degree or extent of fracturing in pillars. One case of great importance to platinum mines is to be able to determine whether a “crush” pillar has failed correctly. Crush pillars are small pillars intended to fail while close to the face. Incorrect design of these pillars can mean that the core of the pillar remains intact and fails more violently at a later stage when the mining face is well ahead of the pillar.

Numerical models were constructed to determine whether this could feasibly be investigated using existing equipment. The assumptions were a hammer-type source generating frequencies around 2 kHz, which is the current high-frequency cut-off of the filtering in the monitoring equipment. Figure 4.40 shows the model geometry for a 6 m by 3 m pillar, with a source and receiver investigating a 3m path through the centre of the pillar. Four models investigated different sizes of the fracture zone – an unfractured elastic pillar and pillars with a fractured skin of 0.4 m, 0.85 m and 1.3 m – this last case being almost fully fractured. In all cases the fractures ranged in size from 0.2 m to 0.6 m with a crack density of 0.1. Figure 4.41a shows the effect of the fracturing on the time domain waveforms. Clear differences in the arrival-times can be observed as the fractured region is increased. Figure 4.41b shows that the effects on Fourier phase difference are unambiguous, and that the techniques of sections 2 and 3 would be useful in such analyses. The primary effect for these ranges of fracture sizes and frequencies are on wave-speed. Higher frequency investigations would allow the attenuation effects to also be studied, potentially improving both the time-domain and frequency-domain interpretations.
Figure 4.1  Geometry for the fractured pillar models. (a) Three-dimensional view of fractured pillar showing positions of source and receiver and dimensions. (b) Plan cross-sections through four pillar models with different sized regions of fracturing with fractured "annulus" varying from (i) 0 m (elastic) (ii) 0.4 m (iii) 0.85 m (iv) 1.3 m. In all cases the fractures range in size from 0.2 m to 0.6 m with a crack density of 0.1.
Figure 4.2  Results for the four pillar models showing the effect of different sizes of the fractured region on wave-speed. (a) Time-domain waveforms (b) Fourier phase-difference relative to the source waveform. (i), (ii), (iii) and (iv) refer to the four models in Figure 4.40b.

A second important mining application is in the investigation of jointing and fracturing in the hangingwall of stopes, which is of obvious significance for evaluating support requirements. Figure 4.42 shows a cross-section through a model constructed to evaluate simple attempts to measure velocity in this fracture zone. The hangingwall of the stope was assumed to have two parting planes with the initial 1.25 m of the fracture zone having a crack density of 0.2 and the next 1.25 m a crack density of 0.1. Again, based on the equipment likely to be used in initial experiments, the assumptions were a hammer-type source with vertical impact generating frequencies of around 2 kHz. Receivers were spaced at 2 m intervals from the source and recorded vertical motion.

Figure 4.3  Cross-section through the three-dimensional fractured hangingwall model.

Receiver waveforms are shown in Figure 4.43 for a fractured and unfractured hangingwall, along with the theoretical P-wave, S-wave and Rayleigh wave arrival times for an unfractured material (5740 m/s, 3510 m/s and 3170 m/s respectively). The P-waves in this test are clearly swamped by the shear and Rayleigh waves, and care will need to be taken in such measurements not to choose the P-wave arrival much later than its true arrival, and hence infer a much greater degree of fracturing. Figure 4.44 shows these same waveforms zoomed tenfold.
The arrival-times for the unfractured model are consistent with theory. The P-wave in the fractured model is slowed by approximately 18%. The model identifies a second problem to be aware of for this geometry before such measurements are naively interpreted. The arrival at 14 m is earlier than would be expected. The waves can travel through the highly fractured region into faster regions and then back through the fractured region to the receiver. At some distance, the average wave-speed for such a path is faster than the direct path. For this geometry, measurements at 14 m and beyond will under-estimate the wave-speed and hence under-estimate the degree of fracturing. The above behaviour is well understood in standard refraction surveys and is used to yield the depth of layers. However, correct interpretation of results will require an appropriate density of the receivers.

Figure 4.4 Waveforms received in the hangingwall at 0 m, 2 m, 4 m, 6 m, 8 m, 10 m, 12 m and 14 m from the source, with a time window of 5 ms. $P_{th}$, $S_{th}$ and $R_{th}$ are the theoretical arrival times for the P, S and Rayleigh waves. Amplitudes aren’t shown but are scaled according the peak in each trace and reduce with increasing distance.
Figure 4.5 Waveforms as for Figure 4.43, but with amplitudes zoomed ten-fold. $P_{th}$, $S_{th}$ and $R_{th}$ are the theoretical arrival times for the P, S and Rayleigh waves. $P_{frac}$ is the P-wave arrival for the fractured model, while $P_2$ is a second P-wave arrival which after a certain distance arrives before the direct wave.

The above numerical examples demonstrate that active velocity scans would provide useful diagnostics on the degree of fracturing for mining applications. Velocity scans could also be a useful technique for evaluating and optimising models of fracture zone development. For this to be realised, advances are required in the physical techniques and equipment, as well as the capabilities of dynamic numerical models to simulate velocity scans through complex fracturing.
5 Conclusions

The highlights of this project include the following:

- A review of stability criteria and the proposal of a criterion (GER) which includes off-reef damage processes.
- The formulation of a new integration methodology and its embodiment in a program called MINSINT.
- The evaluation of a novel method for simulating three-dimensional fracture growth and seismic damage processes.
- The evaluation of what have been termed future technologies, including methods of automating integration as well as various forms of dynamic integration.

A Generalised Energy Release criterion (GER) is proposed as a measure of the stability of transitions that occur in any simulated sequence of mining steps. The application of the GER measure has been illustrated in a number of examples presented in Chapter 2. The GER criterion can be very easily computed if the off-reef damage processes are represented by assemblies of interacting displacement discontinuity elements where both the traction values and the discontinuity sliding or opening components are known before and after each transition between successive, equilibrium states.

An integration methodology has been formulated and implemented in the computer programs MINF and MINSINT to predict the amount of seismicity that will be likely to occur in any working area over a planned production period of one month or one quarter. The results show the potential benefits of this type of integration work for mine layout planning. Future seismicity per area mined is assumed to be similar to past seismicity adjusted by the change in the amount of modelled deformation, where ERR was used as a measure of modelled deformation. The techniques presented here for associating observed seismicity with modelled seismicity are still very new, and further developments of the method and more detailed analyses will undoubtedly lead to better interpretation of the likely response, in time and space, of seismicity to mining.

A general description is given of a novel method that has been proposed and evaluated for the simulation of three-dimensional fracture growth and seismic damage processes near the edges of tabular mine excavations using a prototype, test computer program. The test code provides the capability of nucleating a series of crack growth elements at designated positions in space (seed points). The slip and opening displacement components arising on nucleated crack elements are solved using an iterative procedure. It is found that this approach is able to simulate the formation of simple shear band structures, and can also provide an indication of plausible fracture pattern orientations that arise near a lead-lag stope panel configuration and during the evolutionary formation of fracturing in the vicinity of an emerging crush pillar. Additional examples of interest, such as the generation of fracturing in the vicinity of a mined remnant pillar or the formation of damage zones between superimposed pillars on parallel reef planes, warrant further study.

It should be noted that the proposed method for fracture growth simulation exhibits several shortcomings. Unstable (non-converging) iterative behaviour can arise when crack growth elements are initiated in unfavourable positions. The nucleation of fixed-size crack elements at random positions in space may be too restrictive to allow damage structures to be fully evolved, and the possibility of allowing extended growth of each nucleated crack element from the initial fracture position should be considered.

Although this project focussed primarily on the integration of static models with measured seismicity, the potential of various forms of integration of dynamic models was also evaluated. The ultimate dynamic integration would be for a numerical code to properly predict the dynamic rockmass response in mined out areas simply given inverted seismic parameters such as
seismic moment, stress drop or moment tensor solutions. This concept was evaluated but its practical application was found to require considerable further development due to uncertainties and limitations in both models and seismic data. More research and development is required to create data which is better suited to testing and modifying these models, and to use back-analysis to improve the accuracy and capabilities of the models themselves. Nevertheless, the potential benefits include wider knowledge of in-stope motions which is fundamental to damage prediction and rockburst support requirements and hence this area of research should not be neglected.

A second form of dynamic integration is to use models to test and improve seismic inversions. This was investigated, to evaluate the effect of mined out areas on traditional seismic inversions such as moment, magnitude, source radius, stress drop, and moment tensor. The stope was found to have a very significant influence on the seismograms. The inverted moment was larger due to the presence of the stope, and the inverted source mechanisms were affected by the presence of the stope. It is proposed that the integration of dynamic modelling with the seismic inversion process would lead to improved inversions and a better knowledge of the source mechanisms. These concepts require further development and analysis.

A third form was proposed for dynamic integration by integrating active seismic velocity scans with dynamic numerical modelling of the velocity scans in fracture models. Active velocity scans would provide useful diagnostics on the degree of fracturing for mining applications, and the dynamic models provide a means of interpreting this fracturing. These velocity scans could provide an important technique for evaluating and optimising models of fracture zone development. Examples were shown of simulated velocity scans and the effect of fracturing was found to be detectable in the resulting waveforms.

Some possibilities for the optimal integration of seismic activity with numerical modelling results have been investigated. These include a genetic algorithm approach applied to a simple model of a stope to optimise the selection of modulus and strength properties compatible with observed stope closure values. The stope response to particular modulus and strength values was analysed using the MINF computer code and a genetic algorithm was able to find an optimal solution within the input resolution. One area in which genetic algorithms would be especially powerful is in the determination of non-uniform material properties. For example, by modelling a fault as a plane of displacement, the friction angle and cohesion could be varied locally to simulate various frictional regimes and asperities. A second optimisation approach termed “synthetic annealing” has also been briefly examined. Initial studies suggested that the use of cumulative moment alone will not provide a unique solution since a situation with a low cap stress and low stress drop (more but smaller events) may produce the same moment as a model with a higher cap stress and high stress drop, but with fewer events. More complex “fitness” criteria are required to implement this procedure. Automation of the integration process would require considerably further work on these optimisation strategies and is expected to be time consuming on existing computers. The best compromise is to use simple models with few, but critical parameters that can be run fairly quickly.

This project has reviewed stability criteria and has proposed a generalised energy release as a suitable criterion which includes off-reef failure. It has also formulated a new practical integration methodology. The integration methodology is a starting point and should evolve with time as it is applied practically. The work on generalised energy release needs to be pursued in further projects, particularly in establishing numerical methods for practically evaluating generalised energy release in three-dimensional mine models. Much scope has also been established for further degrees of integration which have been termed future technologies in this report. Such future work can all be loosely grouped into a research theme on dynamic rock mass behaviour.
Publications


Note

1. This was jointly supported under a CSIR-sponsored project.
2. These two references were jointly supported by this project and SIM 040301: “Evaluation of the design criteria of Regularly Spaced Dip Pillars (RSDP) based on their in situ performance”.

132
References


Appendix A  Further MINF and MINSINT Information

A.1 Units and sign conventions for MINF and MINSINT

<table>
<thead>
<tr>
<th>Table A-1</th>
<th>Units used for internal storage and for reporting results.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Quantity</td>
<td>Internal storage</td>
</tr>
<tr>
<td>Stress</td>
<td>MPa</td>
</tr>
<tr>
<td>Displacement</td>
<td>m</td>
</tr>
<tr>
<td>Energy</td>
<td>MJ</td>
</tr>
<tr>
<td>Seismic moment</td>
<td>N·m</td>
</tr>
<tr>
<td>Volume</td>
<td>m³</td>
</tr>
</tbody>
</table>

Compressive stresses and stope closure DDs are positive.

The spatial coordinate system is defined as that used by the MinSim2000 spatial coordinate system. Seismic data are sometimes available in a different coordinate system from that used during the MinPlan (MinSim2000) digitising. The following generalised transformation may be used to obtain seismic X, Y and Z coordinates (x_S, y_S and z_S) from the original mining coordinates (x_M, y_M and z_M) used when digitising using MinPlan:

\[
x_S = R_{11} \times x_M + R_{21} \times y_M + R_{31} \times z_M + T_1 \\
y_S = R_{12} \times x_M + R_{22} \times y_M + R_{32} \times z_M + T_2 \\
z_S = R_{13} \times x_M + R_{23} \times y_M + R_{33} \times z_M + T_3
\]

The default is the trivial case of the same coordinates: \( R_{ij} = 1 \) for \( i=j \), \( R_{ij} = 0 \) for \( i \neq j \) and \( T_i = 0 \).

The following two cases of non-identical coordinate systems are the most common and corrections can most easily be made in the header line of the seismic catalogue file (Table A-2).

<table>
<thead>
<tr>
<th>Table A-2</th>
<th>Corrections made to header line of seismic catalogue file.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Difference in coordinate system</td>
<td>Correction in seismic catalogue header line</td>
</tr>
<tr>
<td>The X and Y coordinates are listed in reverse order.</td>
<td>Place header “Y” before header “X”</td>
</tr>
<tr>
<td>One of the signs has been switched, e.g. XS = -XM</td>
<td>Use –X or –Y as the column header.</td>
</tr>
</tbody>
</table>

A.2 Seismicity input file (“fnseis”)

Note: \( \text{xorm, yorm, zorm} = \) the mine coordinate values of the centre of the first element in the first row.

Format required for seismicity data:

Space or tab-separated headings containing combinations of the following fields.

Sample data:

<table>
<thead>
<tr>
<th>Magnitude</th>
<th>LocationX</th>
<th>LocationY</th>
<th>LocationZ</th>
<th>LocationError</th>
<th>EventDate</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.4</td>
<td>22816.9</td>
<td>-1788.1</td>
<td>6569.4</td>
<td>3.0</td>
<td>00/05/26 04:34:45 PM</td>
</tr>
<tr>
<td>2.0</td>
<td>22704.1</td>
<td>-2112.9</td>
<td>6527.2</td>
<td>31.0</td>
<td>00/06/04 02:37:56 PM</td>
</tr>
<tr>
<td>1.2</td>
<td>21876.4</td>
<td>-715.3</td>
<td>6260.1</td>
<td>12.0</td>
<td>00/06/04 10:49:26 PM</td>
</tr>
</tbody>
</table>
Table A-1  List of header titles and their meaning.

<table>
<thead>
<tr>
<th>Code(s)</th>
<th>Meaning</th>
<th>Unit</th>
<th>Note</th>
</tr>
</thead>
<tbody>
<tr>
<td>“YYMMDD” or “YYYYMMDD”</td>
<td>Date</td>
<td></td>
<td></td>
</tr>
<tr>
<td>“HHMM” or “HHMMSS”</td>
<td>Time</td>
<td></td>
<td></td>
</tr>
<tr>
<td>“YY_MM_DD”</td>
<td>Date</td>
<td>(1)</td>
<td></td>
</tr>
<tr>
<td>“HH_MM_SS”</td>
<td>Time</td>
<td>(1)</td>
<td></td>
</tr>
<tr>
<td>“EventDate”</td>
<td>Date and time</td>
<td>(2)</td>
<td></td>
</tr>
<tr>
<td>”x” or ”LocationX”</td>
<td>X location</td>
<td>m</td>
<td></td>
</tr>
<tr>
<td>”y” or ”LocationY”</td>
<td>Y location</td>
<td>m</td>
<td></td>
</tr>
<tr>
<td>”z” or ”LocationZ”</td>
<td>Z location</td>
<td>m</td>
<td></td>
</tr>
<tr>
<td>”-x” or ”-LocationX”</td>
<td>-X location</td>
<td>m</td>
<td>(3)</td>
</tr>
<tr>
<td>”-y” or ”-LocationY”</td>
<td>-Y location</td>
<td>m</td>
<td>(3)</td>
</tr>
<tr>
<td>”-z” or ”-LocationZ”</td>
<td>-Z location</td>
<td>m</td>
<td>(3)</td>
</tr>
<tr>
<td>&quot;Moment&quot;</td>
<td>Seismic moment</td>
<td>MN-m</td>
<td>(4)</td>
</tr>
<tr>
<td>&quot;Mo_SI&quot;</td>
<td>Seismic moment</td>
<td>N-m</td>
<td>(4)</td>
</tr>
<tr>
<td>&quot;Energy&quot;</td>
<td>Seismic energy</td>
<td>MJ</td>
<td>(4)</td>
</tr>
<tr>
<td>&quot;E_SI&quot;</td>
<td>Seismic energy</td>
<td>J</td>
<td>(4)</td>
</tr>
<tr>
<td>&quot;fo&quot;</td>
<td>Corner frequency</td>
<td>Hz</td>
<td>(4)</td>
</tr>
<tr>
<td>&quot;Radius&quot;</td>
<td>Event radius</td>
<td>m</td>
<td></td>
</tr>
<tr>
<td>&quot;Taus&quot;</td>
<td>Static stress drop</td>
<td>MPa</td>
<td></td>
</tr>
<tr>
<td>&quot;Magn&quot; or &quot;Magnitude&quot;</td>
<td>Event magnitude</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Error</td>
<td>Location error</td>
<td>m</td>
<td></td>
</tr>
</tbody>
</table>

Notes:

1) Any spacers, such as “/” or “:” are acceptable where “_” is written.

2) The EventDate should be in the format set for this PC.

3) The negative coordinate switches the sign of the coordinate value. This allows for the use of different coordinate systems between the MinSim2000 CXX file and the seismic data.

4) Individual data for P- and S-wave pulses may be used by adding a “P” or “S” suffix to each of these parameters.

The following fields must be provided:

1. One of the date fields

2. The seismic location in x, y, and z.

3. "Moment", "Mo_SI" or "Magnitude"

If no seismic moment is given, it is assumed that the Magnitude is a Moment-magnitude and the seismic moment is calculated using the standard Hanks-Kanamori relationship:

\[ M_0 = 10^{5.1 + 1.5 \cdot M} \]

If no other source parameters are given, it is assumed that \( \tau_a = 1 \) MPa.

Other parameters are obtained, where required, using the following relationships:

\[ \sigma_s = \tau_a \] [1]
\[ \sigma_s = \frac{7 \times M_0}{(16 \times r_0^3)} \]  
\[ \tau_a = \frac{G \times E_S}{M_0} \]

where \( G \) = modulus of rigidity = 30GPa

\( \sigma_s \) = static stress drop

\( \tau_a \) = apparent stress

\( M_0 \) = seismic moment

\( E_S \) = seismic energy

**A.3 Brief description of MinView3D**

MinView3D is a Windows-compatible commercial product that was written specifically for displaying data from the following files:

- MinView3D reads plans in “MLS”, “SEG” and “DGA” file formats. The MLS format is the file format used by MinSim2000. SEG format is a format created by the MinSim2000. Files in DGA format can be created by exporting CadsMine layouts.

- Gridded data (PXX)

- Seismicity data. The data is read from a suitably formatted text input file

Examples of graphics output are provided in several figures in the text. Figure A.1 illustrates the appearance of the user interface when setting up various graphics options.

**Sample set-up views in MinView3D**

**Figure A.1** Appearance of Windows control panel for MinView3D.
A.4 Features of MINF not used in the integration procedure

As mentioned in section 3.7.1, MINF has also been developed to solve for other classes of problems. Some of these are available using MINF as it is used in combination with MINSINT for integration. More than 160 large arrays of memory are required to perform all of MINF’s functions. The full functionality can only be achieved for 256 elements squared and four parallel planes. If some functions are disabled, many arrays may be reduced in size to a negligible amount. This has resulted in some functions being disabled in MINF itself and some special-purpose versions of MINF, as listed below.

A.4.1 Large (1024 by 1024) mining problems (MINF_LARGE)

1024 by 1024 single-reef mining problems can be solved using MINF_LARGE.EXE. The problem size is too large for MINSINT.

A.4.2 Huge (2048 by 2048) mining problems (MINF_HUGE)

A 2048 by 2048 single-plane solution is possible if only the normal convergence and stress are required, as in the case of APS estimates. Again, the problem size is too large for MINSINT.

A.4.3 Soft seam, for coal mining (MIN_COAL)

The use of Displacement Discontinuities for seam-type mining is only valid, in effect, if the product of the seam Young’s modulus (\(E_S\)) and seam height is very small compared to the product of the country rock modulus (\(E_C\)) and stope span. This assumption is valid for Witwatersrand gold mining and Platinum mining in the Bushveld Igneous Complex (BIC) in South Africa. It is not valid in coal mining and an adjustment for seam stiffness is needed. The simplest and most direct correction for finite seam thickness is the use of a soft seam for the coal to allow for seam compression. The net effect is to relax stresses at the coal face and to increase them within pillars. According to Ryder (pers. comm. 2005), the soft seam correction does not sufficiently allow the face to bulge into the mined-out areas.

MINF_COAL generates MinSim2000-compatible PXX files for plotting results using MINPOST.

A.4.4 Shallow mining with free surface, for coal mining (MIN_COAL)

The solution method used by MINF allows for full relaxation of normal and/or shear stresses on reef-parallel planes, without the need for iteration as is done in MinSim2000. The simplest application is a free surface in which both normal and shear induced stress are zero. MINF_COAL allows placement of the zero induced stress plane below true surface in an attempt to model competent roof rocks while allowing the surface weathered zone to deform freely.

A.4.5 Seismicity generation, for deep mining (MINF)

Seismicity generation, as described in GAP 722 and Spottswoode (1999, 2001) can be performed using MINF. DFTWrap must be run in expert mode to set the parameters needed for the seismicity generation.

A.4.6 Slippery reef-parallel planes (MINF)

By setting induced shear stress to zero, MINF can simulate the suggestions by Heasley and Chekan (1999) that the higher than elastic roof convergence in coal mines can be simulated by reducing the friction on seam-parallel planes to zero.
A.4.7 Identification of isolated pillars & APS listing (MINF)

MINF can identify unmined areas that are surrounded by areas of mining. One aspect of mine layout design typically involves not exceeding some value of Average Pillar Stress (APS). MINF numbers pillars and lists the APS at each mining step. Fields are added to the PXX files with pillar numbers and APS.

A.4.8 Pillar failure (MINF)

Squat pillars have traditionally been designed using a rule of thumb based on some maximum Average Pillar Stress (APS) that a pillar can sustain (Ryder & Jager, 2002, p270). This has been used for the design of regional stability pillars in which the width-to-height ratio is much greater than 10. Previous reports (e.g. Spottiswoode, 1997) have proposed that, if there is some maximum APS value that a pillar can sustain, then this can be simulated by limiting the stress on every unmined area to that value. MINF has since its inception in first had the feature of limiting the on-reef to some value (called SMAX1).

It is well known that pillars with width-to-height ratios of five or less fail at stress value much less than squat pillars. Application of a single value of strength for squat pillars of all sizes and shapes is unrealistic. Platinum mines use large pillars, including pot-holes, for regional stability and small crush or yield pillars for stope support. These crush or yield pillars must be large enough to prevent back break, but not so large that they burst. However, there have been some reports of pillars bursting even for pillars that might not have been described as being excessively wide. Two special features have been added to MINF to allow for weakening due to edge effects or to width-to-height ratio.

1. Edge weakening. On-reef strength increases with increasing distance from the stope.
2. On-reef strength increases with increasing pillar width-to-height ratio.

The cap stress model applied to MINF has previously always assumed that the strength normal to reef is the same in all unmined areas. Mine layout design is based on the understanding that large pillars will sustain 100’s of MPa while small pillars (W:H ratios < 2 to 5) crush before holding a stress of 10’s of MPa.

Both of these strength rules are being used in studies of pillar strength and failure in the Platinum mines. Details on the parameters used by MINF are given in the section “Application of pillar strengths in MINF” below.

A.4.8.1 Edge weakening

Pillars and abutments are surrounded by fractured rock, caused by the high initial elastic stress, but also by regions of low confining stresses close to the open stopes. Once failed, the rock can sustain lower stresses. Combinations of linear and quadratic increase in strength are allowed until a plateau is reached, as illustrated in Figure A.1. The edge logic was calibrated for a straight edge, but “sees” any face shape. A narrow pillar will be “seen” from both sides. When edge weakening is allowed (NERR = 2, 3 or 4), a field named EDGE is written to the PXX file for viewing in MinView3D. An example of the values of EDGE on part of a mine is shown in Figure 2.1.
**Figure A.1** Sketch of the strength of edge elements as a function of distance from the face or abutment.

**Figure A.2** Contours of EDGE from a hypothetical mining layout. The small numbers of contours on the small pillars indicates that they will only be made weaker than the larger pillars.

### A.4.8.2 Strength as a function of the width-to-height ratio

Narrow "crush" or "yield" pillars are commonly aligned strike-parallel, with holings between them. MINF will identify and number pillars. This feature was developed to calculate and report APS values and was recently extended to identify a characteristic width to height ratio for each pillar. Given that these pillars are generally elongated along strike and the widths are irregular, the width is estimated from:

Width, in elements = NINT(number of elements in pillar / number of elements along strike),

where NINT is the nearest integer value and is used in the strength arrays SMAX_I and SMAX_F.
A.4.8.3 Application of pillar strengths in MINF

Table A-1  The following parameters are used to define the pillar strengths described above.

<table>
<thead>
<tr>
<th>NERR</th>
<th>Strength of intact rock</th>
<th>Strength of failed rock</th>
</tr>
</thead>
<tbody>
<tr>
<td>0</td>
<td>Infinite</td>
<td>n/a</td>
</tr>
<tr>
<td>1</td>
<td>SMAX1</td>
<td>SMAX1</td>
</tr>
<tr>
<td>2</td>
<td>Linear to SMAX1</td>
<td>Linear to SMAX2</td>
</tr>
<tr>
<td>3</td>
<td>Quadratic to SMAX1</td>
<td>Quadratic to SMAX2</td>
</tr>
<tr>
<td>4</td>
<td>Linear to SMAX1</td>
<td>Quadratic to SMAX2</td>
</tr>
<tr>
<td>5</td>
<td>SMAX_I(1:10)</td>
<td>SMAX_F(1:10)</td>
</tr>
</tbody>
</table>

SMAX1 and SMAX2 are reached at a distance FHOR * stope width from the face.
Pillars are allocated a dip length = area / strike length, rounded to nearest integer value.

This is described in the MINF DFT file as follows.

**NERR** (0) Flag for using SMAX

= 0 for elastic solution

= 1 to apply the same cap stress to all elements.

> 1 to apply a cap stress that increases with distance from the face.

= 2 for a linear increase from zero to SMAX1 over FHOR * SW distance from the face before failure and a linear increase to SMAX2 after failure

= 3 for a quadratic increase from zero to SMAX1 over FHOR * SW distance from the face

= 4 for a linear increase before failure & quadratic after failure.

= 5 for strength = f(pillar width), using SMAX_I & SMAX_F.

**SMAX1**(1000) Cap stress limit before failure

**SMAX2**(100) Cap stress maximum limit after failure. SMAX2 = SMAX1 for plastic behaviour.

**FHOR**#### (2.0) Factor used with NERR for pillar strength.

When NERR = 2, 3 or 4, FHOR is used to reduce the cap stress. See NERR above.

**SMAX_I**(10*1000.) Strength of intact pillar = f(width in elements) for NERR=5

**SMAX_F**(10*1000.) Strength of failed pillar = f(width in elements) for NERR=5

A.4.9 Boussinesq solution, for dam loading (*).

This feature was written to estimate the stresses induced by a proposed dam to provide some insight into the potential for triggering an earthquake. A measure of the compactness of the MINF solution method is that this problem, called the Boussinesq solution, is solved in MINF using only 22 lines of code. A similar amount of code defines the surface loading conditions imposed by damned water or by mountains.
Appendix B  Integration via automated optimisation procedures

B.1 Introduction

Integration of numerical modelling with seismicity, as well as with other underground observations can be considered as an optimisation problem. The seismic data provides information about the rockmass that is very useful in determining the strength and the effect of mining-induced stresses. However, the data is historical and has to have been observed previously. The data cannot easily be applied to plan future mining and to determine the effect on the rockmass because the mining will have to be performed to generate the seismicity. Numerical modelling is able to provide a means for evaluating different mining scenarios without having to actually mine the ore body. However, the ability of the model to correctly predict the mining-induced hazard will strongly depend on the formulation of the model, and the correct choice of input parameters. The parameters are difficult to determine and may have considerable variability since they relate to the geological structure of the rockmass. They must often be obtained from laboratory tests and then modified for the underground situation with various heuristic rules. Integration is a procedure that will, hopefully, align the model input parameters to the real rockmass using the historically observed data, ultimately to provide predictions that are closer to the expected behaviour. Thus, the integration is an optimisation of the model parameters to minimize the difference between the model output and the observed seismic data.

B.2 Fitness functions

To define the optimisation problem, a fitness or cost function must be determined that is to be minimized. The solution space of the variables that affect the fitness function must be identified and then a suitable optimisation procedure must be followed that can efficiently evaluate the optimal set of input parameters. In many optimisation processes, the function to be optimised is known implicitly. However, in the integration process, the seismic input and the modelled output depend on the way in which the mining is carried out, the type of model, and the resolution of the data. Thus, it is necessary to explicitly evaluate the fitness function for each option of mining. This may exclude the application of many optimisation procedures, especially those that require knowledge of the gradient of the fitness function to find an optimal solution.

The observed seismicity can be considered to be a time series of events \( O(\beta_o, t_o) \) where \( \beta_o \) is a vector that describes the properties of the events and has \( n \) dimensions corresponding to the number of properties. These properties should include the position in three-dimensional space, dimensions, as well as some measures of magnitude and direction. Similarly, the predicted data should be defined by a time series \( P(\beta_p, t_p) \) where, in general the types of properties included in the vector \( \beta_p \) may be the same as or different from the observed properties \( \beta_o \). In general, even if the properties selected to be included in the vectors \( \beta_o \) and \( \beta_p \) are the same, the values of those properties will be different and the optimisation method must modify the model input parameters so that the model output matches the observed data as closely as possible.

Thus, the most general statement of the integration problem is that it is required to minimize some cost function \( f \) that expresses the difference between \( O(\beta_o, t_o) \) and \( P(\beta_p, t_p) \) for all times to the present. The cost function can be expressed as
\[ f = \min_{t \leq t_c} \left\| O(\mathbf{b}_o, t) - P(\mathbf{b}_p, t) \right\| \]

where \( \| \| \) denotes some suitable norm. This could, for example, be a least-squares norm such that

\[ \left\| O(\mathbf{b}_o, t) - P(\mathbf{b}_p, t) \right\| = \sqrt{\sum_{i=1}^{n} (\mathbf{b}_o^i - \mathbf{b}_p^i)^2 + (t_o - t_p)^2} \]

The vector \( \mathbf{a} \) that describes the properties of the observed seismic data should have components that relate to the position, dimension and magnitude of the event. The Brune (1970, 1971) model is most often used to relate the observed seismic waveforms to the source parameters. In this model, two independent parameters are required to characterize the dimension and the magnitude of the event. The most basic of these are the radius of the event to characterize the dimension and the moment to characterize the magnitude. Other parameters, such as stress drop can be determined using the Brune model. If a moment tensor solution (e.g. Andersen (Linzer) and Spottiswoode, 2001) can be obtained for the event, the dip and strike of the possible event plane can be determined (though not necessarily in a unique manner). Thus, the main parameters describing the seismic event are the x, y, z coordinates, the radius, and the moment. The vector of observed properties becomes a five dimensional

\[ \mathbf{b}_o = [x; y; z; r; M_o]^T \]  \[ \text{[B.3]} \]

or a seven dimensional vector

\[ \mathbf{b}_o = [x; y; z; r; M_o; \phi; \theta]^T \]  \[ \text{[B.4]} \]

where \( \phi \) is the strike and \( \theta \) is the dip of the penny shaped crack defining the event.

However, the model output will strongly depend on the modelling method and the input parameters. The model input parameters can be considered to be vector \( \mathbf{a} \) of dimension \( m \) such that

\[ \mathbf{a} \rightarrow \text{Model} \rightarrow P(\mathbf{b}_p) \]  \[ \text{[B.5]} \]

and so the optimisation process must vary the input parameter vector \( \mathbf{a} \) to obtain an optimal solution that matches the observed data.

The parameter space that can be varied to attempt matching the predicted response with the observed data is determined by the choice of the model and the decision regarding which input parameters are crucial to the evolution of the predicted seismicity.

For the simplest model, the input parameter vector \( \mathbf{a} \) has only a single dimension i.e. the rock strength \( \sigma_{-p} \). In the more complex model,

\[ \mathbf{a} = [\sigma_{-p}; f_d; f_r; f; \kappa]^T \]  \[ \text{[B.6]} \]

expressed in terms of the strength factors \( f_d = \sigma_{-d} / \sigma_{-p} \) and \( f_r = \sigma_{-r} / \sigma_{-d} \) such that \( 0 < f_d \leq 1 \) and \( 0 < f_r \leq 1 \). The confinement factor is \( f \) and the fluidity is denoted by \( f \geq 0 \). These factors could vary by position, but the number of possible positions will be limited by the grid size.
In a model such as the Point Kernel Method (Sellers & Napier, 2001), the model space can be populated with many potential sites that are assumed to be penny shaped cracks, so that the predicted properties vector

\[ \mathbf{\beta}_p = [x; y; z; r; M_o; \phi; \theta]^T \]  \hspace{1cm} \text{[B.7]}

has a one-to-one correspondence with the observed vector \( \mathbf{\beta}_o \) with dimension \( n = 7 \). The input parameter vector includes the position, orientation (dip and bearing), size (diameter), and, in addition, can contain at least the cohesion, friction angle, dilation angle, residual cohesion, and residual friction angle, for each position. Thus, the vector

\[ \alpha = [x, y, z, d, \theta, \Psi, c; \phi; \psi; c, \phi, \psi]^T \]  \hspace{1cm} \text{[B.8]}

has a minimum dimension of \( m = 12 \). For a sequence of \( N \) seismic events at \( M \) positions, assuming that events may occur on the same plane more than once, the function parameter space contains \( N \times m \) options that can be varied in the optimisation procedure.

This is an excessively large number of selections and simplifications are required. In addition, this method, in its current form, creates strong stress singularities that can cause numerical instability, and must be treated only as a step towards a numerical method that effectively predicts mining-induced seismicity. The predicted events are also determined by the input distribution of the site properties and there may be clusters of smaller events that could be considered to be a single large event so that the predicted cumulative frequency-magnitude response may not be able to be directly related to that observed.

The previous discussion assumes that the observed data \( O(\mathbf{\beta}_o, t_o) \) is well defined and is an accurate representation of the actual seismicity. However, the locations are not necessarily well known with an accuracy of less than the source dimension. In particular, because the geophones are usually positioned near the reef due to easy access, there is often a significantly higher location error in the vertical \( (z) \) direction. The frequency-magnitude relationships, represented by the Gutenberg-Richter power law distribution (Figure B.1), often show a limit to the number of small events caused by the limited resolution of the seismic system. Thus, there will always be a resolution limit to the observed data, and an optimisation of the complete time series as expressed in Equation [B.2] of the directly observed parameters as defined by Equation [B.4] becomes impractical. There is also the problem of resolution in the numerical models since time-related mining stages may take too long to analyze on a daily basis, and most likely are even unknown.

It is therefore necessary to interpret the observed data in a form that capture the main behaviour and the most important effects, without considering details which introduce excessive computational effort but which do not contribute significantly to the physical processes. For the integration process, a simplified optimisation can be achieved by comparing the observed moment with the equivalent synthetic moment created by the model.

The moment of the seismic event increases exponentially with the magnitude and so provides a measure that is relatively insensitive to the number of small events that are included. Figure B.2 shows a time series of cumulative moment for 1400 days and indicates little difference between the final cumulative moments when events less than magnitude 1.0 are removed. Figure B.3 shows the data set for 5.8 years and indicates that events of magnitude 2.0 and above (i.e. 0.6 % of the events) produced about 75 % of the total moment. Considering only the 6 % of events with a moment above magnitude 1.0 produces about 95 % of the total moment.
Figure B.1 Gutenberg – Richter plot of cumulative number of events against moment for the five years of mining on a deep level stope.

Figure B.2 The cumulative moment for 3.9 years of mining on a deep level stope showing the effect of removing small magnitude events.
Figure B.3  Relationship between the cumulative moment and the minimum magnitude events included into the summation for five years of mining on a deep level stope.

B.3 Optimisation methods

A genetic algorithm approach may be useful to perform such an optimisation procedure. It mimics an evolutionary process where a sample population of potential solutions are selected and tested against the fitness function. The most suitable solutions are then “bred” in order to provide new solutions. This process continues for a number of generations, with some random mutation allowed so that other possible solutions may arise. This method was tested on a simple mining problem.

There are many different optimisation methods. Cherkaev (2001) discusses classical direct search-for-optimum methods, such as Golden Mean, Conjugate Gradients, Modified Newton Method, methods for constrained optimisation, including Linear and Quadratic Programming, genetic algorithms that mimic evolution, and stochastic algorithms that account for uncertainties in mathematical models. Genetic algorithms and simulated annealing methods were considered in this study to demonstrate the principles, and the methods are described in the following two sections.

B.3.1 Genetic algorithms

Genetic algorithms were developed by Goldberg at Iowa University and are described in detail by Goldberg (1988). As an overview, Lucas (2004) states

“… we create a population of individuals, each represented by a chromosome (a collection of genes or characteristics) appropriate to the problem we are trying to investigate. Genes are usually implemented by a series of computer bits, enough to cover all available alternative values. The value of each gene is initially assigned randomly, within the available parameters. The population is then evaluated to determine how well each individual does the required task.

The best members of the population (typically the top 10 %) then become the parents for the next generation. The genes of the offspring are created by selecting two parents at random

---

---
and combining part of the chromosome of each, so parent gene combinations ABC mated with DEF may become two offspring ABF and DEC. This mimics the crossover or recombination of sexual reproduction in nature and is repeated for further pairs until a full second population is created. Random changes to individual bits are then made mimicking the natural role of mutation, at some desired rate (say 1 in 100 bits changed).

The resultant population is then evaluated as before, and the process is repeated as many times as desired until the required performance level has been achieved (or no further improvement seems possible). This technique can rapidly cover the space of all possible options and converge on a solution that is beyond the ability of all but the best human programmers, in areas where no conventional solution techniques exist.”

### B.3.2 Simulated annealing

Another method of optimisation that may be useful for the types of problems that occur in the integration process is simulated annealing. The method is described by Sandia (2005) as

“... a generalization of a Monte Carlo method for examining the equations of state and frozen states of n-body systems (Metropolis et al. 1953). The concept is based on the manner in which liquids freeze or metals re-crystallize in the process of annealing. In an annealing process a melt, initially at high temperature and disordered, is slowly cooled so that the system at any time is approximately in thermodynamic equilibrium. As cooling proceeds, the system becomes more ordered and approaches a "frozen" ground state at T=0. Hence, the process can be thought of as an adiabatic approach to the lowest energy state. If the initial temperature of the system is too low or cooling is done insufficiently slowly the system may become quenched, forming defects or freezing out in metastable states (i.e. trapped in a local minimum energy state).

The original Metropolis scheme was that an initial state of a thermodynamic system was chosen at energy E and temperature T, and holding T constant the initial configuration is perturbed and the change in energy dE is computed. If the change in energy is negative, the new configuration is accepted. If the change in energy is positive, it is accepted with a probability given by the Boltzmann factor exp \(-dE/T\). This processes is then repeated sufficient times to give good sampling statistics for the current temperature, and then the temperature is decremented and the entire process repeated until a frozen state is achieved at T=0.

By analogy the generalisation of this Monte Carlo approach to combinatorial problems is straight forward (Kirkpatrick et al., 1983; Cerny, 1985). The current state of the thermodynamic system is analogous to the current solution to the combinatorial problem, the energy equation for the thermodynamic system is analogous to the objective function, and ground state is analogous to the global minimum. The major difficulty (art) in implementation of the algorithm is that there is no obvious analogy for the temperature T with respect to a free parameter in the combinatorial problem. Furthermore, avoidance of entrainment in local minima (quenching) is dependent on the "annealing schedule", the choice of initial temperature, how many iterations are performed at each temperature, and how much the temperature is decremented at each step as cooling proceeds.”

### B.4 Example of integration using genetic algorithm

In order to be able to solve the large-scale mining problems efficiently, some simplifications are needed. The model may be a MINSIM or MINF type boundary element analysis where the reef is considered to be a tabular plane of boundary elements and the surrounding rock is treated as being purely elastic, in which case there is no explicit seismicity, but parameters such as ERR and ESS may indicate where seismic events could occur. Since there is no subsequent failure and stress transfer, the time series of seismic events will be always in error. Some simple modifications have been made to the MINF code (Spottiswoode, 1997, 1999) to permit the
rockmass strength to be exceeded in the reef plane so that subsequent failure can simulate seismic events. In this case the process followed is to allow the element that exceeds the failure envelope the most to fail. Element failure results in a stress redistribution process that may transfer stress to other elements, which can fail in turn. Once there are no more elements that can fail, the event is considered to be complete and its size can be determined. This model in its simplest form considers a perfectly plastic behaviour where the rock has a single value of maximum strength, independent of confinement. The stress on the failed element is reduced to the strength and remains at that value by creating additional closure on the failed element, to simulate the crushing of the rockmass at that point. The magnitude of the stress predicted will also depend on the elastic properties. A more complex response can be obtained by allowing the rockmass to soften from the failure stress $\sigma_p$ to a dynamic strength $\sigma_d$ during the modelled seismic event as shown in Figure B.4. Finally, the stress can relax to the residual strength $\sigma_r$ due to viscoplastic creep. The strength can be made to be dependent on the confinement produced by the minor principal stress parallel to the reef plane.

![Figure B.1 Schematic of response of a DD element to mining in MINF, showing the relative magnitudes of the input parameters.](image)

To test the genetic algorithm, a simple model of a stope was developed, based on the site studied in GAP 604. A computer code was written to be able to modify the input file for MINF to change the Young's modulus and strength parameters externally and to then run the modified analysis. The code was linked to a genetic algorithm code so that the genetic algorithm could prepare new input parameters for MINF and then execute MINF. The output from MINF is then read by the genetic algorithm code.

The objective of integration is to determine the most appropriate input parameters so that the observed historical record aligns with the predicted history. It can then be assumed, that for at least a short time period, the forward predictions of the model will reflect the future observations more closely. In an elastic boundary element analysis, the key input parameter is the Young's modulus. This test was therefore designed to back-predict the Young's modulus based on "observations" of closure at a point in a mine. The genetic algorithm must maximize a cost function, which was defined here to be the negative of the least-squares norm of the difference between the predicted and the observed closure values. The observed data is considered to be the closure at a point in the model defined by the “x” in Figure B.5. There were difficulties in
obtaining the exact closure at the point from the data in GAP 604 and so, for simplicity the “observed” closure was obtained from a numerical model with a Young’s modulus of 70 GPa.

Figure B.2  MINF model of a deep level stope showing the point (x) where the closure is compared.

A relatively coarse grid of 24 m was used to run the MINF analysis in the least possible time. The model is shown in Figure B.5. No mining steps were considered at this stage. The genetic algorithm was then run allowing the modulus to range from 30 GPa to 80 GPa, with 31 divisions between these limits. The modulus can be expressed as

\[ E = 30 + \frac{c}{(2^5-1)} \times 50 \text{ GPa} \]  

[B.9]

Where c defines the gene and is coded in a bitwise integer representation as shown in Table B.1. Thus, the resolution of the optimised solution will be 50/31, approximately 1.61 GPa. As shown in Table B.1, two genes are selected and then the crossover operation takes a specified part of one and the remainder of the other and makes a new gene. A given portion of the new genes is mutated (by flipping a single bit) to reach new areas of the solution space.

Table B-1  Examples of genes in binary universe and example of crossover and mutation.

<table>
<thead>
<tr>
<th></th>
<th>Value</th>
<th>Power of 2</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>4 3 2 1 0</td>
</tr>
<tr>
<td>Example</td>
<td>0 0 0 0 0</td>
<td></td>
</tr>
<tr>
<td>Example</td>
<td>31 1 1 1 1</td>
<td></td>
</tr>
<tr>
<td>Example</td>
<td>1 0 0 0 1</td>
<td></td>
</tr>
<tr>
<td>Gene 1</td>
<td>21 1 0 1 0</td>
<td></td>
</tr>
<tr>
<td>Gene 2</td>
<td>10 0 1 0 1</td>
<td></td>
</tr>
<tr>
<td>Crossover</td>
<td>22 1 0 1 0</td>
<td></td>
</tr>
<tr>
<td>Mutation</td>
<td>30 1 1 1 0</td>
<td></td>
</tr>
</tbody>
</table>
The relationship between the predicted Young’s modulus and the fitness of each solution is plotted against the number of generations in Figure B.6. The figure shows how the genetic algorithm investigates many input parameters, but focuses mainly on the optimal solution. There is no observable convergence to a single solution with increasing generations, but near-optimal solutions are predicted after a few generations. This may require more tuning of the genetic algorithm parameters or may just be a result of the way in which the algorithm operates. By plotting the fitness against the input parameters, in Figure B.7, the “most optimal” solution can be determined. In this case, the optimal solution is predicted to be 69.7 GPa, which is close to the “correct” value of 70 GPa. This is a very simple model and converges within 20 generations. All additional runs are wasted computer time. By better tuning of the parameters the algorithm should produce results that remain near the solution.

Figure B.3  Comparison of average fitness (solid line) of the solutions (diamonds) with number of generations.

Figure B.4  The average fitness (equivalent to the difference between observed closure and predicted closure) as a function of the average input Young’s modulus per generation.
Considerably more work is required to extend this optimisation strategy to the full integration problem. One area in which genetic algorithms would be especially powerful is in the determination of non-uniform material properties. For example, by modelling a fault as a plane of displacement, the friction angle and cohesion could be varied to simulate various frictional regimes and asperities. By modelling of mining towards the fault and comparison with observed seismic data, the true structure and properties of the fault could be determined and used for future simulations. In that case, each gene would consist of

- The row number;
- The column number;
- The friction angle; and
- The cohesion,

and the chromosome would be the total of all the genes (being $n \times m$ where $n$ is the number of rows and $m$ is the number of columns).

<table>
<thead>
<tr>
<th></th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
<th>8</th>
<th>9</th>
<th>10</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>32</td>
<td>35</td>
<td>37</td>
<td>32</td>
<td>33</td>
<td>32</td>
<td>38</td>
<td>39</td>
<td>35</td>
<td>35</td>
</tr>
<tr>
<td>2</td>
<td>33</td>
<td>38</td>
<td>31</td>
<td>39</td>
<td>36</td>
<td>38</td>
<td>36</td>
<td>34</td>
<td>33</td>
<td>33</td>
</tr>
<tr>
<td>3</td>
<td>34</td>
<td>37</td>
<td>31</td>
<td>35</td>
<td>39</td>
<td>34</td>
<td>33</td>
<td>35</td>
<td>37</td>
<td>34</td>
</tr>
<tr>
<td>4</td>
<td>31</td>
<td>30</td>
<td>34</td>
<td>37</td>
<td>39</td>
<td>36</td>
<td>37</td>
<td>30</td>
<td>30</td>
<td>37</td>
</tr>
<tr>
<td>5</td>
<td>36</td>
<td>33</td>
<td>31</td>
<td>33</td>
<td>38</td>
<td>32</td>
<td>31</td>
<td>33</td>
<td>32</td>
<td>37</td>
</tr>
<tr>
<td>6</td>
<td>37</td>
<td>31</td>
<td>30</td>
<td>37</td>
<td>31</td>
<td>35</td>
<td>37</td>
<td>38</td>
<td>31</td>
<td>36</td>
</tr>
<tr>
<td>7</td>
<td>31</td>
<td>34</td>
<td>34</td>
<td>34</td>
<td>31</td>
<td>30</td>
<td>31</td>
<td>34</td>
<td>32</td>
<td>34</td>
</tr>
<tr>
<td>8</td>
<td>35</td>
<td>38</td>
<td>34</td>
<td>33</td>
<td>39</td>
<td>38</td>
<td>33</td>
<td>35</td>
<td>35</td>
<td>34</td>
</tr>
<tr>
<td>9</td>
<td>37</td>
<td>34</td>
<td>34</td>
<td>32</td>
<td>36</td>
<td>34</td>
<td>35</td>
<td>31</td>
<td>36</td>
<td>39</td>
</tr>
<tr>
<td>10</td>
<td>35</td>
<td>34</td>
<td>31</td>
<td>32</td>
<td>37</td>
<td>34</td>
<td>34</td>
<td>31</td>
<td>39</td>
<td>39</td>
</tr>
</tbody>
</table>

*Figure B.5  Example of fault plane discretised into squares with each having a random friction angle between 30° and 40°. A typical gene (for dark grey shaded block) would be “6;8;38”, i.e. column number; row number; friction angle, assuming that cohesion is zero.*

The above examples have provided a limited understanding of the needs for optimisation schemes in the integration process. The genetic algorithm is an enhanced brute force method, and can take considerable number of generations and hence a large amount of computational time to solve the problem. This is a significant problem in the integration scenario since a solution of the numerical model, which can take a day or even longer, is required for each optimisation generation. Usually, the problem to be solved is an equation that can be solved in a very short time and so many millions of generations are possible in a practical timeframe. Other optimisation algorithms may be more useful and need to be investigated. Some further work will be done to prove some of these concepts, but a development of the complete optimisation strategy would exceed the time constraints of this project.

### B.5 Integration concepts using simulated annealing

Genetic algorithms take a long time to reach a solution, partly because of the need to solve the numerical model each time. Thus, further studies have been undertaken to evaluate the feasibility of performing the integration of seismic activity with numerical modelling as an optimisation problem using simulated annealing. The benefit of this process is most obvious for MINF runs using a cap stress to produce synthetic seismicity. The run times will increase with increased cap stress so by taking the cap stress as being equivalent to the temperature, the model can be run from highest cap stress to lowest so that the shortest run times will occur first. At each “temperature” i.e cap stress, the stress drop can be varied to determine which will produce a synthetic moment that most closely matches the observed moment.
To investigate these concepts, a series of numerical analyses of a deep-level longwall stope were undertaken with the MINF program to understand the effect of the input parameters on the output response. Having a better understanding of the relationship between input and output parameters makes it possible to select an appropriate optimisation scheme. The MINF parameters that need to be specified include the intact rock strength, the dynamic stress drop, the residual strength, and the healed strength.

A number of analyses were carried out using different values of intact strength. Figure B.9 shows how the intact strength affects the simulated cumulative moment and indicates that the moment increases if lower strength rock parameters are used. In this case greater reef-projected deformation occurs. If the strength is specified as being too low, the analysis may not converge as is shown for the run with a specified intact strength of 100 MPa. For each possible value of intact strength there are many combinations of the other strength parameters that can be considered. For example, the effect of the dynamic stress drop on cumulative moment is shown in Figure B.10 for an intact strength of 225 MPa. An optimisation scheme is required to take all the different parameter variations into account. This is possible with a so-called genetic algorithm approach, but the run time for each trial layout increases significantly with decreasing strength, as shown in Figure B.11. Each seismic event is considered sequentially so that increased seismicity increases the run time. Thus, a “synthetic annealing” optimisation approach, which considers a sequence of runs where the intact strength is decreased by some specified interval and the other parameters are varied within certain ranges until there is a match between observed and predicted behaviour, may be more efficient computationally.

Figure B.1 Effect of intact strength on cumulative moment in MINF simulations of seismic activity.
Figure B.2  Cumulative moment with mining steps for different magnitudes of stress drop with an intact strength of 225 MPa.

Figure B.3  Effect of intact strength on run time of analysis.
Figure B.4  Mine plan and seismicity simulated using MINF.

The mine plan and simulated seismic activity are shown in Figure B.12. The time-dependent closure calculated from runs with different intact strength values are shown in Figure B.13. Lower cap stresses result in a larger extent of failure and hence a higher deformation. Such data can provide an additional constraint on the seismic integration to help the optimisation routine select the correct material parameters if closure is measured directly in the stope.

Figure B.5  Closure time curves for different values of cap stress.

Simulated annealing was implemented, but not tested on actual data. Initial studies suggested that the use of cumulative moment alone will not provide a unique solution as a situation with a low cap stress and low stress drop (more and smaller events) may produce the same moment as a model with a higher cap stress and high stress drop but fewer events. This can be seen by comparing the range of moments produced for various stress drops at a constant cap stress.
as shown in Figure B.10, with the effect of cap stress at constant stress drop in Figure B.9. Thus, more complex fitness criteria are required.

### B.6 Conclusions

Integration of numerical modelling with seismicity, as well as with other underground observations can be considered as an optimisation problem. Seismic data provides information about the rockmass that is very useful in determining the strength and the effect of mining-induced stresses. However, the seismicity is historical and has to have been observed previously. The data cannot be easily applied to plan future mining and to determine the effect on the rockmass because the mining will have to be performed to generate the seismicity. Numerical modelling is able to provide a means of evaluating different mining scenarios without having to actually mine the ore body. However, the ability of the model to correctly predict the mining induced hazard will strongly depend on the formulation of the model and the correct choice of input parameters. Integration is a procedure that will, hopefully, align the model input parameters with those of the real rockmass using the historically observed data, in order to provide predictions that are closer to the actual behaviour. It can then be assumed that, for at least a short time period, the forward predictions of the model will reflect the future observations more closely. Thus, the first step of the integration process is an optimisation of the model parameters to minimize the difference between the output from the modelling of the historical and the seismic data observed during that mining. This is usually done by brute force methods of selecting various “typical” parameters and adjusting these between various runs. However, a proper integration scheme should have the back-prediction built in, and would be able to execute models and automatically find the best fit parameters. This would be done using an optimisation process.

To define the optimisation problem, a fitness or cost function must be determined that is to be minimized. In many optimisation processes, the function to be optimised is known implicitly. However, in the integration process, the seismic input and the modelled output depend on the way in which the mining is carried out, the type of model, and the resolution of the data. Thus, it is necessary to explicitly evaluate the fitness function for each option of mining. This may exclude the application of many optimisation procedures, especially those that require knowledge of the gradient of the fitness function to predict more optimal solutions.

A genetic algorithm approach may be useful to perform optimisation without an explicit fitness function. It mimics an evolutionary process where a sample population of potential solutions are selected and tested against the fitness function. The most suitable solutions are then bred in order to provide new solutions. This process continues for a number of generations, with some random mutation allowed so that other possible solutions may arise. To test the genetic algorithm, a simple model of a stope was developed, based on the site studied in GAP 604. A computer code was written to be able to modify the input to MINF to change the Young’s modulus and strength parameters. The code was linked to a genetic algorithm code so that the genetic algorithm could supply new input parameters to MINF and then execute MINF. The output from MINF is then read by the genetic algorithm code. The genetic algorithm must maximize a cost function that is the negative of the least-squares norm of the difference between the predicted and the observed values. The observed data is considered to be the closure at a point in the model, and the genetic algorithm was able to find an optimal solution that lies within the input resolution.

Considerably more work is required to extend this optimisation strategy to the full integration problem. One area in which genetic algorithms would be especially powerful is in the determination of non-uniform material properties. For example, by modelling a fault as a plane of displacement, the friction angle and cohesion could be varied locally to simulate various frictional regimes and asperities.

With the genetic algorithm approach, the run time for each trial layout increases significantly with decreasing strength. Each seismic event is considered in sequence so that increased
seismicity increases the run time. Thus, a “synthetic annealing” optimisation approach, that considers a sequence of runs where the intact strength is decreased by some specified interval and the other parameters are varied within certain ranges until there is a match between observed and predicted behaviour, may be more efficient computationally. Simulated annealing was implemented, but not tested with actual data. Initial studies suggested that the use of cumulative moment alone will not provide a unique solution since a situation with a low cap stress and low stress drop (more and smaller events) may produce the same moment as a model with a higher cap stress and high stress drop, but fewer events. Thus, more complex fitness criteria are required.

Much work is required to automate the integration process, and the integrated process will probably be time consuming on existing computers. The best compromise is to use simple models with few, but critical parameters that can be run fairly quickly. Considerable research is also required to find the best optimisation process, and the answer will depend on the type of model and data that is being used in the integration.

B.7 References


Kataka, M. O. Hildyard M.W. and Sellers E. J., Integration of seismic observations from deep level gold mines with dynamic numerical modelling.


1. PROJECT SUMMARY:

PROJECT TITLE: New criteria for rock mass stability and control using integration of seismicity and numerical modelling

PROJECT LEADER: J A L Napier

ORGANISATION: CSIR: Division of Mining Technology (An equal opportunity employer)

ADDRESS: P.O. Box 91230, Auckland Park, 2006

TELEPHONE: (011) 358 0000  TELEFAX: (011) 726 5405

PRIMARY OUTPUT: Identification of improved criteria for layout design and evaluation using integration of seismicity and numerical modelling.

HOW USED: Used to evaluate new and existing mining sequences and potential seismic activity and to assist in stability and risk assessment.

BY WHOM: Mine rock mechanics practitioners, seismologists, mine planning engineers and Chief Inspector of Mines.

CRITERIA FOR USE: Adaptive interpretation of seismic activity and forecasting of potential rock mass instability.

POTENTIAL IMPACT: Improvement in mine layout designs and excavation control through better means of assessing the response of mine excavation sequences.

<table>
<thead>
<tr>
<th>FUNDING REQUIREMENTS (R 000s)</th>
<th>YEAR 1</th>
<th>YEAR 2</th>
<th>YEAR 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>TOTAL PROJECT COST</td>
<td>1 623</td>
<td>1 651</td>
<td>1 790</td>
</tr>
<tr>
<td>TOTAL SUPPORT REQUESTED FROM SIMRAC</td>
<td>1 623</td>
<td>1 651</td>
<td>1 790</td>
</tr>
</tbody>
</table>

DURATION (YY/MM) 2002/04 TO 2005/03

SIMRAC SUB-COMMITTEE: AU/PT X COAL OTHER GENERIC
2. **PROJECT DETAILS**

2.1 **PRIMARY OUTPUT**

Identification of improved criteria for layout design and evaluation using integration of seismicity and numerical modelling.

2.2 **OTHER OUTPUTS (deliverables)**

1. Computer codes for the simulation of static and dynamic deformation in the rock mass.
2. Improved understanding of large scale rock mass behaviour and limits to certainty in mine design.

2.3 **ENABLING OUTPUTS**

<table>
<thead>
<tr>
<th>NO.</th>
<th>ENABLING OUTPUT</th>
<th>MILESTONE DATE</th>
<th>MAN DAYS</th>
<th>COST OF OUTPUT</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Phase 1</td>
<td>03/2003</td>
<td>315</td>
<td>1 623</td>
</tr>
<tr>
<td>1.1</td>
<td>Statement of current key uncertainties relating to the formulation of numerical procedures for performing integration of observed seismic activity with numerical modelling.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1.2</td>
<td>New and adapted numerical models for fault slip and seismicity.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1.3</td>
<td>Improved description of the mechanics of rock mass instability on faults and near stope faces.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>Phase 2</td>
<td>06/2004</td>
<td>368</td>
<td>2 098</td>
</tr>
<tr>
<td>2.1</td>
<td>Trial integration exercise of actual seismic observations and numerical modelling on an appropriate mine.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2.2</td>
<td>Ability to simulate realistic mechanisms of rock mass instability.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2.3</td>
<td>Industry interaction on proposed integration procedures.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3</td>
<td>Phase 3</td>
<td>03/2005</td>
<td>220</td>
<td>1 343</td>
</tr>
<tr>
<td>3.1</td>
<td>Recommended integration approach and software.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3.2</td>
<td>Effective criteria for layout and stability assessment based on integration precepts.</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3.3</td>
<td>Final report on integration methods and evaluation criteria.</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
### 2.4 METHODOLOGY

<table>
<thead>
<tr>
<th>NO. OF ENABLING OUTPUT</th>
<th>STEP NO.</th>
<th>METHODOLOGY TO BE USED TO ACCOMPLISH THE ENABLING OUTPUT (INDICATE STEPS/ACTIVITIES)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Phase 1:</td>
<td></td>
</tr>
<tr>
<td></td>
<td>1.1</td>
<td>Identify and document key uncertainties in the ability of numerical models to simulate rock mass behaviour near underground workings.</td>
</tr>
<tr>
<td></td>
<td>1.2</td>
<td>Develop and adapt numerical models to be applied to integration studies of fault slip and seismicity.</td>
</tr>
<tr>
<td></td>
<td>1.3</td>
<td>Assemble information relating to seismic activity and to other factors such as time-dependent stope closure to assess how this information can be used to develop an improved description of the mechanisms of rock mass stability and instability.</td>
</tr>
<tr>
<td>2</td>
<td>Phase 2:</td>
<td></td>
</tr>
<tr>
<td></td>
<td>2.1</td>
<td>Set up a trial integration exercise in an actual mine area for a period of some months, performing regular, adaptive modelling exercises to assimilate the observed activity and to assess incipient deformation potential.</td>
</tr>
<tr>
<td></td>
<td>2.2</td>
<td>Adapt simulation tools to exhibit explicit mechanisms of stable and unstable time-dependent behaviour based on principles of global system stiffness, non-linear mechanics and statistical physics.</td>
</tr>
<tr>
<td></td>
<td>2.3</td>
<td>Arrange industry workshops and seminars to obtain industry interaction on proposed scheme for integration of seismic activity and modelling.</td>
</tr>
<tr>
<td>3</td>
<td>Phase 3:</td>
<td></td>
</tr>
<tr>
<td></td>
<td>3.1</td>
<td>Assemble results from industry trial and workshops and recommend appropriate implementation approach.</td>
</tr>
<tr>
<td></td>
<td>3.2</td>
<td>Formulate effective criteria for layout and stability evaluation and forecasting.</td>
</tr>
<tr>
<td></td>
<td>3.3</td>
<td>Prepare final report on new evaluation criteria for rock mass stability and control.</td>
</tr>
</tbody>
</table>

**Key Facilities and Procedures to be used in the Project**

CSIR Mining Technology facilities and resources will be used for modelling work, 3-D graphics and other studies.
2.5 TECHNOLOGY TRANSFER

Industry workshops to train rock mechanics and seismologists in the use of the software will be held. The software will be compatible with pre- and post-processor software widely available to the industry.

3. FINANCIAL SUMMARY

3.1 Financial Summary

<table>
<thead>
<tr>
<th></th>
<th>YEAR 1</th>
<th>YEAR 2</th>
<th>YEAR 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Project staff costs</td>
<td>1 400</td>
<td>1 406</td>
<td>1 507</td>
</tr>
<tr>
<td>Other costs:</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Operating costs</td>
<td>24</td>
<td>14</td>
<td>20</td>
</tr>
<tr>
<td>Capital &amp; plant costs</td>
<td>0</td>
<td>18</td>
<td>0</td>
</tr>
<tr>
<td>Sub-contracted work</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Presentations and Papers</td>
<td>0</td>
<td>10</td>
<td>43</td>
</tr>
<tr>
<td>Value added tax*</td>
<td>199</td>
<td>203</td>
<td>220</td>
</tr>
<tr>
<td>TOTAL COST OF PROJECT</td>
<td>1 623</td>
<td>1 651</td>
<td>1 790</td>
</tr>
<tr>
<td>Less funding from other sources</td>
<td>1 623</td>
<td>1 651</td>
<td>1 790</td>
</tr>
</tbody>
</table>

* Only for VAT registered concerns
### 3.2 Project Staff Costs

Reflect Man Days and Costs separately

<table>
<thead>
<tr>
<th>NAME AND DESIGNATION</th>
<th>YEAR 1</th>
<th>YEAR 2</th>
<th>YEAR 3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>MD</td>
<td>COSTS</td>
<td>MD</td>
</tr>
<tr>
<td>J A L Napier</td>
<td>140</td>
<td>756</td>
<td>130</td>
</tr>
<tr>
<td>Project Manager</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>S M Spottiswoode</td>
<td>70</td>
<td>328</td>
<td>70</td>
</tr>
<tr>
<td>Technical consultant</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>E J Sellers</td>
<td>35</td>
<td>160</td>
<td>25</td>
</tr>
<tr>
<td>Technical consultant</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>M O Kataka</td>
<td>70</td>
<td>156</td>
<td>70</td>
</tr>
<tr>
<td>Research engineer</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>TOTAL (R 000s)</strong></td>
<td>1 400</td>
<td>1 406</td>
<td>1 507</td>
</tr>
</tbody>
</table>

### 3.3 OPERATING COSTS (Running)

<table>
<thead>
<tr>
<th>ACTIVITY/EQUIPMENT (Items above R10 000)</th>
<th>YEAR 1</th>
<th>YEAR 2</th>
<th>YEAR 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Travelling to the mines</td>
<td>5</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td>Computer maintenance and upgrading</td>
<td>5</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td>Miscellaneous / sundries</td>
<td>3</td>
<td>4</td>
<td>0</td>
</tr>
<tr>
<td>Software</td>
<td>11</td>
<td>0</td>
<td>10</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td>24</td>
<td>14</td>
<td>20</td>
</tr>
</tbody>
</table>
### 3.4 CAPITAL AND PLANT COSTS

#### (i) ITEMS TO BE PURCHASED OR DEPRECIATED FOR MORE THAN R10 000 PER ITEM

<table>
<thead>
<tr>
<th>Item</th>
<th>Year 1</th>
<th>Year 2</th>
<th>Year 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>High speed computer</td>
<td></td>
<td>18</td>
<td></td>
</tr>
<tr>
<td>Other miscellaneous items</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td>18</td>
<td></td>
</tr>
</tbody>
</table>

#### Chapter 1(ii) ITEMS TO BE MANUFACTURED WITH ASSEMBLED COST OF MORE THAN R10 000 INCLUDING MATERIAL AND LABOUR

<table>
<thead>
<tr>
<th>Item</th>
<th>Year 1</th>
<th>Year 2</th>
<th>Year 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Other miscellaneous items</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**TOTAL (i) and (ii)**
### 3.5 Sub-contracted Work

<table>
<thead>
<tr>
<th>SUB-CONTRACTOR</th>
<th>ACTIVITY</th>
<th>YEAR 1</th>
<th>YEAR 2</th>
<th>YEAR 3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**TOTAL**

### 3.6 Presentation and Papers

<table>
<thead>
<tr>
<th>ACTIVITY</th>
<th>COST (R000s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Presentations &amp; publications</td>
<td></td>
</tr>
<tr>
<td>Industry seminar/ workshop</td>
<td>10</td>
</tr>
<tr>
<td>Booklets</td>
<td>43</td>
</tr>
</tbody>
</table>

**TOTAL**

10 43

### 3.7 Other Funding

<table>
<thead>
<tr>
<th>ORGANISATION</th>
<th>NATURE OF SUPPORT/ COMMITMENT</th>
<th>AMOUNT (R000s)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
4. **MOTIVATION**

(Provide a clear and quantified motivation of justification for the proposal, as well as the main conclusions of a literature survey and the findings of related local and international research. The motivation should include a synthesis of previous work in the project area, both locally and overseas, why the project is proposed what the primary output will achieve and a cost benefit analysis, if applicable. Use continuation pages where necessary but in most cases it should be possible to clearly present the key data and arguments in the space provided.)

The recently completed SIMRAC project GAP603 (2001) and other studies have demonstrated the potential benefits of integrating numerical modelling with observed seismic activity in order to provide a data-driven basis for the adaptive assessment of rockburst hazards and the evaluation of mining sequences. It has also become apparent, from studies of time-dependent closure (SIMRAC project GAP601b), that the deformations arising in a mining panel are dependent on the global stiffness of the rock mass "system" (faults and mining induced fractures) surrounding the panel.

It is necessary to consider the explicit incorporation of integration features into currently used evaluation software and also to implement extensions to these tools to provide some ability to model dynamic deformations. This is still extremely challenging and it is likely that a number of short-term compromises have to be made to enable basic integration exercises to be completed. In this regard, it is necessary to document these limitations and key areas of uncertainty in the formulation of these models. In addition to this, it is recognised that the interpretation of observed rock deformations and seismic behaviour is related to the mechanics of the relevant "system" components and their mutual interaction. Some systematic means of describing this complex behaviour in terms of readily applied layout evaluation criteria is required that is analogous to the classic stability principles that have been proposed, for example, for the assessment of pillar layouts. This in turn will lead to improved methods for the assessment and solution of a wide variety of practical rock engineering problems such as remnant extraction, excavation layouts and implications of panel lead-lag sequencing and face advance rate.

These studies will be supported by available field data and by a “live” field trial in which a deep level mining sequence is tracked in a selected mining area and the observed seismic activity is integrated with prototype evaluation software. The results of this experiment will be used to modify proposed integration procedures and will be amplified by interaction with industry stakeholders through workshops and seminars. A final booklet outlining proposed evaluation criteria and guidelines for the application of computer tools that allow the integration of seismic observations with numerical modelling will be issued.)
5. CURRICULA VITAE OF PROJECT LEADER AND RESEARCH STAFF

5.1 SUMMARY INFORMATION

**Project Leader**

<table>
<thead>
<tr>
<th>NAME &amp; INITIALS:</th>
<th>Napier J. A. L.</th>
<th>AGE: 55</th>
</tr>
</thead>
<tbody>
<tr>
<td>QUALIFICATIONS:</td>
<td>BSc (Chem Eng) (Rand) 1967; MSc (Chem Eng) (Rand) 1971; Ph.D (Rand) 1980</td>
<td></td>
</tr>
</tbody>
</table>

**Principal Project Team Members**

<table>
<thead>
<tr>
<th>NAME &amp; INITIALS:</th>
<th>Spottswoode S.M.</th>
<th>AGE: 54</th>
</tr>
</thead>
<tbody>
<tr>
<td>QUALIFICATIONS:</td>
<td>BSc Stellenbosch, BSc Hons, PhD (Geophysics) WITS, 1980, Chamber of Mines Rock Mechanics Certificate.</td>
<td></td>
</tr>
<tr>
<td>SPECIAL AWARDS:</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>NAME &amp; INITIALS:</th>
<th>Ewan J. Sellers</th>
<th>AGE: 35</th>
</tr>
</thead>
<tbody>
<tr>
<td>QUALIFICATIONS:</td>
<td>BSc Civil Eng (Cape Town) First class Hons, MSc Eng (Cape Town), PhD (Cape Town), Pr Eng.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>NAME &amp; INITIALS:</th>
<th>Kataka M.O</th>
<th>AGE: 35</th>
</tr>
</thead>
<tbody>
<tr>
<td>QUALIFICATIONS:</td>
<td>BSc (Hons) (Nairobi) 1990; Certificate (seismology) (Uppsala, Sweden) 1993; Certificate (seismology) (IISEE-Tsukuba, Japan) 1995; MSc. (Nairobi) ; PhD (Geophysics) (Wits) awaiting examination results.</td>
<td></td>
</tr>
</tbody>
</table>
5.2 RELEVANT EXPERIENCE AND PUBLICATIONS (one page for each individual listed in 5.1)

NAME: Napier J. A. L.

RELEVANT EXPERIENCE:

20 years experience in rock engineering with emphasis on the development of stress analysis computer systems:

MINSIM-D - development and coding of solution and benchmark programs.
DIGS/3DIGS - Discontinuity Interaction and Growth Simulation program to predict fracture initiation and growth around openings in highly stressed material.

Responsible for coordinating rockmass behaviour research at COMRO and CSIR Division of Mining Technology in elastodynamics, micro-mechanics and stope fracture zone formation.

RELEVANT PUBLICATIONS:


RELEVANT PUBLICATIONS:


NAME: Spottiswoode S. M.

RELEVANT EXPERIENCE:

Work experience
32 years experience as a seismologist in the South African mining industry:
1970 - 78 at the Bernard Price Institute of Geophysical Research.
1978 - 84 at Blyvooruitzicht Gold Mine.
1984 - 85 at Rand Mines.
1986 - present at CSIR Mining Technology, previous COMRO, the Chamber of Mines Research Organization. Currently hold the position of Technical Consultant.

Leadership in scientific & technical innovations
Similarity of mine seismic events to earthquakes (PhD work, 1970’s).
First mine seismologist employed solely by a mine in SA (Blyvooruitzicht, 1978)
Developed the concept of Volume Excess Shear Stress (1986).
Led team to develop the COMRO PSS (1986-88)
Installed first in-stope geophones to understand the site response in stopes (1988)
Multiplet locations by waveform similarity (1997).
Extended ERR concept for mine layout design (1997).
Generate synthetic seismic catalogues that are very similar to observed seismicity (2000). This shows promise to totally replace classical mine layout design criteria.

SELECTED RELEVANT PUBLICATIONS (relevant to this project work only)


<table>
<thead>
<tr>
<th>NAME:</th>
<th>Spottiswoode S. M. (continued)</th>
</tr>
</thead>
</table>


SM Spottiswoode. Aftershocks and foreshocks of mine seismic events. 3rd international workshop on the application of geophysics to rock and soil engineering, GeoEng2000, Melbourne Australia, 2000.


NAME: Sellers, E.J.

RELEVANT EXPERIENCE:

Experience as researcher and lecturer in Department of Civil Engineering at University of Cape Town from 1988 to 1994. Work was carried out on development of a critical state model for clay (MSc Thesis) and on the formulation of an anisotropic damage model for rock (PhD Thesis). Has worked on COMRO projects as external collaborator while at UCT.

After joining CSIR Miningtek in 1995, he gained further experience in rock mechanics with the emphasis on micro-mechanics and numerical analysis of failure mechanisms in rock. Has been involved with research into the fracture processes around deep level stopes, integration of modelling and seismicity, blasting processes and the in situ stress state using underground observations and numerical models. He is also involved in the laboratory testing of rock samples and the development of specialised equipment for physical modelling experiments. He was project manager of the ELFEN numerical modelling program development as well as SIMRAC and other rock mechanics projects. He was promoted to technical consultant in 2000 and has been involved with a number of SIMRAC and consulting projects for stress measurement and underground instrumentation. He has written and co-authored over 80 papers and consultancy reports.

RELEVANT PUBLICATIONS:


Sellers, E. and Scheele, F. Finite element predictions of damage in Indiana Limestone, submitted to : 8th International Conference of International Association of Computer Methods and Advances in Geomechanics, Morganton, West Virginia, (H.J. Siriwardane ed). (also FRD/UCT CERECAM report No. 224.)


Sellers, E. and Scheele, F. Prediction of anisotropic damage in experiments simulating mining excavations in Witwatersrand Quartzite.


Sellers, E., Modelling of the influence of geology on the in-situ stress state, SIMRAC Interim Project report, GAP029, CSIR Division of mining technology, 1995

Sellers, E. and Turner, P.A. Modelling of pressure driven fracture propagation using DIGS, SIMRAC Interim Project report, GAP332, CSIR Division of mining technology, 1996

Sellers, E. A review of models for the propagation of seismic waves in the fractured rockmass around a stope, SIMRAC Interim Project report, GAP029, CSIR Division of mining technology, 1995


Roberts, D.P., Sellers, E.J. & Sevume, C, 1999 Numerical modelling of fracture zone development and support interaction for a deep level tunnel in a stratified rockmass. SARES 99, Johannesburg


with numerical modelling GAP(603). SIMRAC Symposium.


NAME: Kataka M.O

RELEVANT EXPERIENCE:

RELEVANT PUBLICATIONS:

Research Papers in Preparation
Kataka, M. O., and A., Cichowicz. The use of empirical green’s function with asperity model to synthesize ground motions of a large event (to be submitted to Bulletin of Seismological Society of America, 2002).
Kataka, M. O., and A., Cichowicz. Identification of fault planes for mining-induced earthquakes using a green’s function approach (to be submitted to Bulletin of Seismological Society of America, 2002)

Published Articles and Abstracts
Sellers, E and M. O., Kataka. Localization and scaling of acoustic emission A sources with mining induced seismicity. *(submitted to Journal Geophysical Research).*
6. **DECLARATION BY THE PROPOSING ORGANISATION**

I, the undersigned, being duly authorized to sign this proposal, hereewith declare that:

- The information given in this proposal is true and correct in every particular.

- This Organization has the basic expertise and facilities required for satisfactory completion of the project and will adhere to the program of activities as set out in this proposal.

- The costs quoted are in accordance with the normal practice of this Organization and can be substantiated by audit.

Signed on this __________ day of __________ 2002 for and behalf of

____________________________________________________________________________________

SIGNATURE: ____________________________________________________________________

NAME: _______________________________________________________________________

DESIGNATION: __________________________________________________________________

USE A CONTINUATION SHEET IF NECESSARY