CHARACTERIZATION AND ANALYSIS OF DISCONTINUOUS SUBSIDENCE ASSOCIATED WITH BLOCK CAVE MINING USING ADVANCED NUMERICAL MODELLING AND INSAR DEFORMATION MONITORING

by

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Abstract

While block caving presents an economic means to develop lower grade ore deposits, it often leads to significant ground deformations threatening the safety of overlying mine infrastructure. For guidance on relationships between caving depth and surface subsidence, a comprehensive cave mining database was developed and the database clearly shows caving-induced surface deformations tend to be discontinuous and asymmetric due to large movements around the cave being controlled by geologic structures, rock mass heterogeneity and topographic effects.

Also shown is that as undercut depth increases, the magnitude and extent of the caved zone on surface decreases. Numerical modelling conducted in a benchmark study testing several different numerical methods (finite-element, distinct-element, FEM-DEM with brittle fracture and 3-D finite-difference) indicates this is only the case for macro deformations and the lateral extent of smaller strain deformations increases as a function of undercut depth, which indicates caution should be taken against relying on existing empirical design charts for estimates of caving-induced subsidence where small strain subsidence is of concern, as the empirical data does not properly extrapolate beyond the macro deformations.

In addition, sophisticated 3-D numerical modelling was investigated as a means of predicting the extent and magnitudes of caving-induced surface subsidence. Results from a back analysis of the cave-pit interactions at the Palabora mine were used to constrain the rock mass properties and far-field in-situ stresses derived from field characterization data. The “best fit” set of input properties obtained was then used for forward modelling. Further calibration was performed using high-resolution InSAR monitoring data. The close fit achieved between the predictive 3-D numerical model and InSAR monitoring data demonstrates the significant value of InSAR calibrated 3-D numerical models.

Collectively, the results of this research help to further the characterization, assessment and understanding of block-caving subsidence, by addressing existing limitations in the use of empirical and numerical subsidence analysis methods. The limitations and uncertainty arising from mine site data are described, specifically the representation of mine geology, rock mass properties, in-situ stresses and cave propagation, together with means to constrain these
inputs and calibrate sophisticated 3-D numerical models through back analysis and integration with InSAR data.
Preface

Chapter 2 “Empirical investigation and characterization of surface subsidence related to block cave mining” was co-authored by Kyuseok Woo, Dr. Erik Eberhardt, Dr. Davide Elmo, and Dr. Doug Stead. As the lead author, Kyuseok Woo built the block caving database using materials available in the public domain, carried out the data analyses, and prepared the manuscript. Dr. Eberhardt provided guidance in the development of the database and reviewed the manuscript and provided editorial comments. Dr. Elmo provided guidance in the development of the numerical models used to support and provide insights into the trends detected.

Chapter 3 “Benchmark testing of numerical capabilities for modelling the influence of undercut depth on caving-induced subsidence” was co-authored by Kyuseok Woo, Dr. Erik Eberhardt, Dr. Doug Stead, and Dr. Davide Elmo. Mr. Woo, Dr. Eberhardt, Dr. Stead, and Dr. Elmo jointly identified and designed the research program, which was developed as part of a competitive research contract with the Centre for Excellence in Mining Innovation (CEMI). CEMI specified the problem geometry and several input constraints. Mr. Woo performed numerical modelling, completed the analytical studies, and prepared the manuscript. Dr. Eberhardt, Dr. Stead, and Dr. Elmo reviewed the manuscript and provided editorial comments.

Chapter 4 “Integration of field characterization, mine production and InSAR monitoring data to constrain and calibrate 3-D numerical modelling of block caving-induced subsidence” was co-authored by Kyuseok Woo, Dr. Erik Eberhardt, Dr. Doug Stead, Dr. Bernhard Rabus, and Dr. Alex Vyazmensky. Mr. Woo, Dr. Eberhardt, and Dr. Stead identified and designed the research program. Dr. Vyazmensky provided data and input from Rio Tinto, to supplement that collected at the mine site by the research team. Dr. Rabus directed and carried out the processing of the InSAR data, Mr. Woo performed the research, data analyses and 3-D numerical modelling, and prepared the manuscript. Dr. Eberhardt, Dr. Vyazmensky, Dr. Rabus, and Dr. Stead reviewed the manuscript and provided editorial comments.

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Dedication

I dedicate this dissertation to my family, especially...

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Chapter 1: Introduction

1.1 Problem Statement

As a low cost underground mass mining method, block caving is increasingly being favoured by a number of mining companies targeting low-grade orebodies at depth or transitioning from open pit to underground operations due to the economic and production benefits the method provides. Mass mining through block caving, however, may lead to significant ground deformations and subsidence on surface, for which the extent and magnitudes must be accurately predicted to protect mine infrastructure and minimize environmental impacts (e.g. Moss et al., 2006). Damage, destruction and/or replacement of surface and underground infrastructure due to block caving induced subsidence are often the cause for additional capital and operation expenditures as well as for environmental degradation.

To better manage this potential risk, numerous studies have been undertaken to characterize and assess caving-induced subsidence. Most of these are based on empirical procedures that relate rock mass quality to subsidence break angles (e.g. Whittaker & Reddish, 1989; Laubscher, 2000), but otherwise neglect the influence of geology in producing discontinuous and asymmetric deformations. Geologic structures, anisotropy and rock mass heterogeneity can each significantly influence the caving and subsidence process, leading to poor predictions of subsidence if not properly considered. The unexpected failure of an 800-m high pit slope above the block caving operation at Palabora in South Africa (Fig. 1.1) is a key example of an unanticipated risk that as a worst case scenario threatened worker safety and could have resulted in the loss of the production and ventilation shafts for the mine, which would have led to the financial shutdown of the mine. Fortunately the shafts were not affected, but the failure has increased dilution and sterilization of ore, as well as chronic concerns as operations continue.

Palabora, together with other recent ground-control failures involving mass mining operations (e.g. Northparkes, Kidd Creek, etc.), highlights the need for the use of sophisticated 3-D numerical modelling to properly account for the complex interactions between geology, surface topography, cave propagation and caving-induced subsidence. In
turn, this necessitates improved understanding and means to characterize the ground conditions and constrain the modelling results. With the industry now moving towards the next generation of "super" block cave mines (Resolution, Oyu Tolgoi), the long-term success of these operations will be heavily dependent on proper cave management and prediction of caving-induced ground deformations.

1.2 Thesis Objectives

This study aims to further the characterization, assessment and understanding of block-caving subsidence dynamics by addressing limitations in current empirical and numerical methods used for caving-induced subsidence prediction. To do so, several thesis objectives are defined as listed below. These involve steps that link an empirical-based database developed to help define key influences and interactions that affect caving-induced subsidence with advanced numerical modelling strategies that account for these interactions, integrated with remote-sensing monitoring techniques as a means to constrain sophisticated 3-D models.

The primary objectives of this thesis are:

i. Develop a comprehensive database of in-situ measured subsidence information from block cave mining operations from around the world to be used to identify and investigate the key factors and ground interactions influencing subsidence behaviour; provide relationships correlating geological information to subsidence observations as a means of guiding and constraining future numerical analyses; provide a framework on which to analyze the effects of geology in promoting asymmetry and discontinuous caving-induced subsidence; and develop empirical guidelines based on the diverse information compiled in the database to aid in the initial scoping and projection of expected subsidence as a function of undercut depth, for future block caving projects.

ii. Carry out benchmark testing of several different numerical techniques to investigate their abilities and limitations with respect to modelling block caving subsidence for a range of undercut depths. The comparison is being carried out for a conceptualized
problem involving a porphyry type deposit, incorporating faults, several different lithologies and varying rock mass properties.

iii. Develop a detailed 3-D numerical model for an active operation to better understand block-caving subsidence mechanisms and to develop much-needed guidance on the use of advanced numerical modelling focusing on means to incorporate geological complexity and rock mass heterogeneity using advanced 3-D continuum modelling techniques, and investigate the use of satellite-based Interferometric Synthetic Aperture Radar (InSAR) imaging as a means to monitor block-caving subsidence and constrain complex 3-D numerical models.

1.3 Thesis Structure

This thesis consists of an introduction chapter, three key research chapters, and a conclusion followed by several appendices. Chapter 1 introduces the problem and explains the thesis objectives and literature review carried out. The next three chapters elaborate on the procedures and findings of the research conducted to achieve the stated objectives.

Chapter 2 describes the development of a block caving database, including descriptions of the cases examined, and the corresponding caving, fracture initiation and subsidence angles as a function of undercut depth. Empirical relationships are established and bias within the database discussed. Also included are a number of Google Earth satellite images, analyzed to characterize asymmetric subsidence patterns seen above different caving operations and correlated to basic information in the database. Detailed information on the different data sources used to develop the database is provided in Appendix B.

Chapter 3 reports benchmark testing of different numerical methods based on a conceptual block caving example. These are used to test the strengths and limitations of each method in assessing the relationship between caving depth and subsidence. A comparison is drawn between numerical assumptions regarding the treatment of geology (continuum versus discontinuum), dimensionality (2-D versus 3-D), and means to simulate the mining process. The comparative analysis was carried out for assumed undercut depths of 500 to 2000 m in 500 m intervals.
Chapter 4 examines the problem of managing parameter uncertainty in complex 3-D numerical models, and works towards developing a constrained and calibrated 3-D subsidence model for the Palabora mine in South Africa. RADARSAT-2 InSAR monitoring data collected specifically for this study was used together with a back analysis of a 2005 caving-induced pit slope failure, to forward model the predicted subsidence arising for one year’s production (2009-2010).

Finally, Chapter 5 summarizes the findings for Chapters 2 to 4 and details the key conclusions, major contributions, and topics for further research arising from this thesis.

1.4 Literature Review: Subsidence Induced by Block Caving

1.4.1 Overview

Underground mining creates voids that through extraction and collapse can cause the lowering of the ground surface. Following numerous experiences in the 19th century where mining-induced subsidence resulted in damage to overlying mine infrastructure, railways, roads and homes, formal studies of subsidence were initiated (National Coal Board Subsidence Engineering Handbook, 1975; Kratzsch, 1983; and Whittaker & Reddish, 1989). These early studies laid the foundation for future subsidence research by explaining the basic mechanisms of subsidence. Today’s subsidence investigations cover structural geology, geomechanics, surveying, mining and property law, and mining methods and techniques (Singh, 1992).

Underground mining methods can be classified based on their support conditions as shown in Fig.1.2 (Brady & Brown, 2006). While the principles of subsidence are similar regardless of the mining method, surface deformation shows distinctive characteristics depending on which mining method is employed. Selection of the mining method is typically based on the geometry of the orebody, its grade and the geomechanical properties of the ore and the country rock (Villegas, 2008). Mining-induced subsidence can be viewed as taking place in two phases: an active subsidence phase and a residual subsidence phase. Active subsidence follows the advance of the working face or cave back and forms 90 to 95 percent of the total subsidence in most cases. Residual subsidence takes place after mining has
ceased through continued collapse, loss of support and migration of old mine workings. Residual subsidence typically continues for a year but in some cases it progresses for decades; new subsidence in exhausted mining areas is mostly residual subsidence.

Two types of subsidence have been observed: continuous and discontinuous (Brady & Brown, 2006). Continuous subsidence involves a smooth lowering of the ground surface. Discontinuous subsidence is characterized by abrupt changes and large vertical drops localized over smaller surface areas. Continuous subsidence is often viewed as a large trough, whereas discontinuous subsidence involves scarps, sinkholes, and tension cracks. In the past, subsidence studies were largely focused on continuous subsidence driven by investigations related to longwall coal mining where subsidence is approximately continuous. With an increasing interest in caving methods for hard rock mining, research focus is being shifted to discontinuous subsidence.

The block cave mining method (Fig.1.3) was originally used to mine orebodies where rock is fractured and weak, and thus prone to caving. More recently though, it has been applied to hard rock where high induced stresses facilitate rock fracturing (Carrasco et al., 2004). Block caving requires intensive preparation and development work, but its cost is the lowest amongst the different underground mining methods. The term “block caving” is used here to include both block and panel caving; panel caving is a variant where the orebody is divided into panels which are mined sequentially. Not included in the use of this term is sublevel caving.

Block caving is usually applied to large scale extraction of various metals and minerals from steep to vertical orebodies, with vertical dimensions (i.e. block heights) exceeding 100 metres. The ore block is undercut by blasting, allowing the "back" to fracture, fragment and cave. The broken ore is then extracted to create a void and initiate further caving. Whereas a sub-vertical crater eventually forms on surface through this caving process, a shallower profile may develop in weaker near-surface rocks. Factors identified by Flores & Karzulovic (2004) as influencing the steepness and geometry of the cave include rock mass quality, geological structures, and draw management.
1.4.2 Caving Subsidence Mechanism

A conceptual model describing the caving subsidence progress has been summarized by Abel & Lee (1980) as follows (Fig.1.4):

a. As ore is excavated from the drawpoints, undercutting and cave propagation upward from the extraction level and the initial collapse of the cave back begins.

b. The consequence of the upward cave propagation is the thinning of the overlying cap rock. When the thinning reaches the point that the cap rock cannot transfer its load to the adjacent solid cave walls, measurable ground subsidence occurs. The cap rock begins to deflect toward the caved rock below.

c. Caved ore production continues and tension cracks begin to open and subsidence increases on the surface. The result is a trough-shape subsidence profile.

d. Surface deformation further progresses and a circular collapse structure called “glory hole” is formed roughly at the center over the caved area.

e. If extraction continues, the rock adjacent to the caved zone slides along joints or faults, or topples into the open crater, growing the collapse structure laterally.

Van As et al. (2003), examining subsidence above block caving operations, defined several distinct zones of deformation as follows (Fig.1.5):

- **Caved Zone:** A zone of active caving typically situated above the undercut footprint and usually manifested as a collapse crater. The contents of this zone are caved ore and waste rock, ranging in size from large boulders to fines. The extent of the caved rock zone is defined by the angle of break, or caving angle, which is the inclination of the line drawn from the edge of extraction level to the in-situ/intact rock
boundary. The angle of break increases with depth and is sub-vertical in strong rocks (if no significant persistent dipping discontinuities are present), and less inclined where mining depths are shallow or overburden rocks are weak.

- **Fractured Zone:** An irregular broken surface with scarps, large open tension cracks, and large blocks undergoing shear-rotation and toppling towards the caving zone. The extent of this zone is defined by the fracture initiation angle, which is the inclination of the line drawn from the edge of the undercut to the limit of surface cracking.

- **Continuous Subsidence Zone:** An area of small-strain, continuous deformations. Lupo (1998) found that significant continuous surface subsidence can develop as far as 250 metres away from the limits of the fractured zone. The maximum extent of this zone is defined by the angle of subsidence, which is the inclination of the line drawn from the edge of the undercut to the limit of surface deformations.

The boundaries between these different zones are generally obvious under normal conditions (Gilbride et al., 2005), but are less obvious when soil cover, waste dumps or erosion features cover part of the surface. Some block caving mines only show the caved rock zone. Break angles are more commonly reported with respect to the extent of surface deformation, however there are different definitions of break angle. Karzulovic et al. (1999) explained it as the angle measured from horizontal of the straight line drawn from the edge of the undercut to the edge of the crater. Brady & Brown (2006) defined the break angle as being measured as a straight line drawn from the edge of the undercut to the farthest crack on surface. The subsidence limit angle, when reported, is defined as the angle measured from horizontal of the straight line drawn from the edge of the undercut to the farthest measured deformation (Gilbride et al., 2005).
1.4.3 Factors Influencing Subsidence Development

Mining-induced surface subsidence is the product of the complicated interactions between various factors. Crowell (2001), focussing on room and pillar coal mines, suggested the following controlling factors:

- **Height of mined-out area**: Vertical subsidence increases proportionally to the height of the mined-out block and typically does not exceed the height of the mine void.

- **Width of unsupported mine roof**: The area of subsidence and the width of unsupported roof has a positive relationship. Approximately, the potential area of subsidence can be estimated by adding the extraction area to the area defined by the angle of draw.

- **Thickness of overburden**: Negative correlation exists between vertical subsidence and the depth or thickness of overburden. Since the rock bulks when it collapses, vertical surface subsidence tends to be smaller when the extraction area is deeper.

- **Competency (strength) of rock**: Subsidence is related to the strength or competency of the overlying cap rock.

- **Pillar dimensions**: As pillar dimensions increase, subsidence decreases. When the size of the pillars is smaller, the probability of pillar crushing or punching, and thus roof collapse is higher.

- **Hydrology**: Fluctuating groundwater level can affect subsidence. Roof collapse becomes more likely if the roof rock is repeatedly saturated and if flowing water erodes softer rock pillars.

- **Fractures/joints**: When discontinuities exist, the mine roof is weakened and subsidence becomes more likely to occur. The subsidence could even go beyond the limit of the mined area due to discontinuities.
• **Time:** When depth of mining and competency of the overburden rock increases, subsidence can fully develop over a longer period time.

More specific to block caving, Brown (2003) identified a number of features of the orebody, local geology and surface topography that can influence subsidence development, including:

• Dip and geometry of the orebody

• Depth of mining and the associated in situ stress field

• Strengths of the caving rock mass and of the rock and soil cover

• Presence of a slope at surface (i.e. irregular surface topography)

• Nature of major intersecting geological features such as faults and dikes

• Previous surface mining (i.e. deep open pit)

• The accumulation of caved rock or placement of fill in the formed crater

• Presence of nearby underground excavations

Van As et al. (2003) reported general trends regarding caving-induced subsidence as follows:

• When a mining face encounters a significant discontinuity with moderate to steep dip, movement will occur on the fault regardless of the caving angle.

• A stepped crack morphology will result where the fault daylights at surface. If mining is only on the hangingwall side of the fault, the limits of surface movement will coincide with the fault.

• If the fault dip is steeper than the cave angle, the extent of surface subsidence will be reduced. Likewise, if the fault dip is less than the cave angle, the extent of surface subsidence will be increased.
Flores & Karzulovic (2002) list the relative degree of influence (high or moderate) of several factors on subsidence, as shown in Fig.1.6, based on their analysis of data from 18 block cave mining operations.

1.4.4 Mirco- and Macro-Subsidence

Subsidence above block caving operations can be divided into micro- and macro deformations (Butcher, 2005). Micro deformations include those detected as tilting ground, small strains and/or vertical and horizontal displacements detectable through deformation monitoring (see subsidence zone in Fig.1.5). Although relatively small compared to macro deformations, they can still be significant in causing differential displacements of several centimetres, which in turn can affect the structural integrity of strain sensitive structures (e.g. those built with concrete). Lupo (1998) reported that micro deformations have been observed up to 250m from the perimeter of a collapse crater. Macro deformations involve those ground movements that are visually detectable such as the opening of tension cracks, development of scarps, fracturing and break back of the surface above and around the cave’s footprint, and breakthrough of the cave itself to form a large crater (see caved and fractured zones in Fig.1.5).

The zone of macro deformation around or above a caving operation can have serious implications in terms of safety, damage to surface infrastructure and environmental damage. Based on experiences from caving operations in Australia and South Africa, Butcher (2006) suggested that the ground immediately above the cave footprint will subside, cave and collapse, and thus any excavations or surface infrastructure situated within the footprint will ultimately be destroyed. Predicting the zone of macro deformation with fair precision is also critical in terms of project viability as macro deformations and ground collapse into the cave can result in dilution of the ore being extracted. Dilution is one of many challenges confronting mining operations requiring extra expenditures to deal with waste material that can degrade the value of the ore below the cutoff grade.
1.5 Literature Review: Block Caving Subsidence Analysis

There are three main approaches to caving-induced subsidence prediction: empirical, analytical, and numerical. While empirical methods are heavily relied upon in block caving geomechanics, the use of numerical modelling has opened up new opportunities to investigate the factors governing subsidence and thus develop improved prediction methodologies.

1.5.1 Empirical Design Chart

Empirical methods are based on comparisons of data and experience collected from historical and existing operations, to which the new mine being designed is compared to. They represent a quick, simple to use tool, which yields fairly satisfactory results. However, as Bahuguna et al. (1993) noted, they can be limited by site-specific data bias and are not based on rock mechanic principles. Most of the published empirical methods provide guidance in the value of maximum subsidence, $s_{\text{max}}$, based on a large number of measurements carried out for cases having similar geological and mining conditions. Such assessments become less reliable if there is an absence or deficiency in data of earlier observations in areas having similar conditions (Bahuguna et al., 1993). The most commonly used empirical method for estimating subsidence parameters in cave mining is Laubscher’s method (Laubscher, 2000). Laubscher proposed a design chart (Fig.1.7a) that relates the predicted cave angle to the MRMR (Mining Rock Mass Rating), density and height of the caved rock, and mine geometry (minimum and maximum span of a footprint). Fig.1.7b presents a worked example of Laubscher’s method. Laubscher’s chart does not consider the effect of geological features like faults which may cause the cave angle to steepen or flatten depending on their dip. In addition, the method only applies to the caving zone and not the full extent of subsidence, which is an important parameter in cave mine design (Flores and Karzulovic, 2004). Overall, the application of Laubscher’s method requires sound engineering judgement, geotechnical expertise and experience in similar geotechnical settings.
1.5.2 Analytical Methods

Analytical methods generally include closed-form solutions and limit equilibrium procedures. Most of the published closed-form solutions for subsidence prediction draw upon continuum mechanics and elasticity theory, and are therefore more applicable to problems involving continuous subsidence. The force and/or moment balance techniques employed by limit equilibrium solutions on the other hand, can be applied to simplified discontinuous subsidence problems.

Hoek (1974) developed a limit equilibrium model for the analysis of subsidence generated by the exploitation of an inclined orebody, using the sublevel caving method. This model aimed to predict the subsidence due to progressive hangingwall failure with increasing mining depths. The variables considered in Hoek’s analysis are shown in Fig.1.8. The solution assumes that a planar shear failure is formed from the undercut to a surface tension crack and that the failure occurs under static conditions. The effect of the caved material is also taken into account.

This model was used to analyze the subsidence at the Grängesberg iron ore mine in Sweden and showed that the break angle is highly influenced by the rock mass properties, the mining depth, and the depth of caved material (Hoek, 1974). The basic method proposed by Hoek (1974) has evolved as subsequent researchers also investigating discontinuous subsidence introduced their own modifications.

The first modification was made by Brown & Ferguson (1979) in order to consider sloping ground surface and groundwater pressure in the tension crack and in the shear plane. The method was applied at the Gath’s mine, Rhodesia. In their analysis, a sloping upper surface in the hangingwall decreased the break angle and increased the tension crack depth. Although it was mentioned that during the rainy season subsidence accelerates, its effect was not evaluated. Another factor analyzed was the 3-D nature of the crater. Based on field observations, it was concluded that when the radii of curvature of the walls is small, higher break angles can be expected.
Kvapil et al. (1989) also extended Hoek’s analysis to define discontinuous subsidence due to block and panel caving. This model is based on the recognition that progressive failure occurs in both the hangingwall and footwall, and allows the prediction of subsidence effects from caving operations involving very steep orebodies. This model was applied to evaluate the subsidence at El Teniente mine which was using block and panel caving.

Karzulovic (1990) extended Brown and Ferguson’s model to predict the evolution of the subsidence crater at the Rio Blanco mine. The theoretical break angle calculated was adjusted considering local factors such as the presence of faults and the amount of broken material in the crater. This adjustment was required to obtain improved agreement with a database of observed values for rock masses of similar quality.

Lupo (1996) also extended Hoek’s model to account for the failure of the hangingwall and footwall around an inclined orebody. This model considers the failure of the hangingwall using the limit equilibrium equations derived by Hoek (1974), but also considers an active earth pressure coefficient due to the effect of the draw of broken ore. For the footwall, the limit equilibrium equations derived by Hoek (1970) for excavated slopes in open pit mines are used. This approach was used in the Kii runavaara mine, Sweden and gave fair agreement with failure observations of the hangingwall. However, the model was not able to predict the behaviour of the footwall (Henry & Dahnér-Lindqvist, 2000); most of the mine infrastructure is located above the footwall.

Flores & Karzulovic (2004) summarized the evolution of limit equilibrium to predict caving-induced discontinuous subsidence, as shown in Table 1.1. Although these provide a means to estimate the angle of break, one of the key parameters required for mine planning, the limit equilibrium techniques do not take into account complex rock structure, in-situ stresses or stress-strain relationships, thereby restricting their predictive capabilities.

### 1.5.3 Numerical Modelling

An adequate representation of the rock mass is required for a numerical model to properly capture its physical and engineering response to mining. Hudson & Harrison (2000) describe a rock mass as being either Continuous, Homogeneous, Isotropic and Linear Elastic
(CHILE) or largely Discontinuous, Inhomogeneous, Anisotropic, and Non-Elastic (DIANE), with most rock masses being associated with the latter. The complex geology and its long history of formation make rock masses a difficult material for mathematical representation via numerical modelling. The most commonly applied numerical methods for rock mechanics problems are summarized in Table 1.2.

Amongst these, continuum codes like the finite difference programs FLAC (Itasca, 2007) and FLAC3D (Itasca, 2009) or the finite element program ABAQUS (Dassault Systèmes, 2009; Fig.1.9a) have been the most widely used for modelling block caving subsidence (e.g., Singh et al., 1993; Van As et al. 2003; Sainsbury et al. 2008; Beck & Pfitzner, 2008). To a lesser degree, distinct element codes like UDEC and 3DEC (Fig.1.9b) or hybrid methods like that used by Vyazmensky et al. (2010b) to incorporate brittle fracture processes, have been used. The choice of method varies on a case-by-case basis, ranging from selection based on simplicity and familiarity in the modelling method to specific selection to incorporate a key process like brittle fracturing.

Table 1.3 summarizes published accounts of numerical analysis of surface subsidence and block cave/open pit interactions. The modelling study by Flores & Karzulovic (2004) is arguably the first attempt after Laubscher (2000) to provide general guidance for subsidence analysis. Their study involved conceptualized FLAC/FLAC3D modelling of surface subsidence associated with block caving, including the varying of rock mass properties and consideration of the effects of an open pit with varying pit and undercut level depths. Based on their modelling results, complemented by limit equilibrium analysis, a series of design charts were developed correlating angle of break and zone of influence of caving with undercut level depth and crater depth for varying rock mass quality. Fig.1.10 shows one example of the design charts they developed. The validity of these charts though, has yet to be confirmed through mining experience.

More recently, Vyazmensky (2008) applied a hybrid finite-element/discrete-element (FEM/DEM) approach to the modelling of block caving subsidence. This technique incorporates brittle fracture capabilities enabling the modelling of a continuum passing to a discontinuum in response to changes in stress and strains. Vyazmensky et al. (2010a)
examined preferential rock fragmentation between an ore column and surrounding host rock, together with the influence of geological structures on the development of caving-induced subsidence. In a related study, Vyazmensky et al. (2010b) investigated the interaction between a propagating block cave and overlying open pit slope, highlighting the importance of rock bridges and their incremental failure through cave-pit interactions which led to the progressive failure of the Palabora pit slope (Fig. 1.11).

Overall, with the exception of Flores & Karzulovic (2004), all reviewed accounts were focused on back analysis or predictive modelling for specific mine sites. It appears that there is a need for comprehensive modelling attempts to evaluate the general principles of surface subsidence development in block caving settings. Brown (2003) recommended using a combination of empirical, analytical and numerical methods for subsidence predictions. He suggested that a preliminary estimate of the angle of break be derived using Laubscher’s chart and calibrated against observed break angles in similar mining settings (i.e. empirical). The estimated angle of break should then be checked against limit equilibrium approaches (i.e. analytical). The estimated value of the angle of break can be adjusted considering local geological features and the amount of broken material in the crater. Finally, numerical methods should be used to confirm the estimate of the angle of break and to estimate the stresses and displacements induced in the rock mass around the caved zone.

As noted by Flores & Karzulovic (2004), only numerical models allow the full extent of the influence zone to be predicted (including micro-deformations). Therefore, limit equilibrium analyses must be complemented with numerical models for the prediction of the extent of the influence zone. Given the significant cost implications of locating major excavations and infrastructure beyond the influence of the underground caving operation, it is well worth the effort of using numerical modelling to ensure that the empirical and/or analytical methods are not overly conservative in their predictions (Van As et al., 2003).
<table>
<thead>
<tr>
<th>Author(s)</th>
<th>Observations and Applications (Improvements)</th>
</tr>
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<tbody>
<tr>
<td>Hoek (1974)</td>
<td>This limit equilibrium model is aimed to subsidence prediction due to progressive hangingwall failure with increasing mining depth. It was developed to predict the surface subsidence at Grängesberg mine in Sweden. This became the basis for subsequent limit equilibrium models.</td>
</tr>
<tr>
<td>Brown &amp; Ferguson (1979)</td>
<td>Extended Hoek’s limit equilibrium model to account for a sloping surface and groundwater pressures in the tension crack and on the shear plane. This model was used to evaluate the progressive failure of the hanging wall at Gath’s mine in Rhodesia.                                                                винтическая месау.</td>
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<tr>
<td>Kwapil et al. (1989)</td>
<td>Used Hoek’s limit equilibrium model to include the progressive failure occurring in both hanging wall and foot wall in a very steeply dipping orebody. This model was applied at El Teniente mine, in Chile, to evaluate the subsidence generated by underground block and panel caving operations.</td>
</tr>
<tr>
<td>Karzulovic (1990)</td>
<td>Used Brown and Ferguson’s limit equilibrium model to predict the discontinuous subsidence associated with block caving at Rio Blanco mine in Chile. This model was developed to evaluate subsidence in a vertical orebody.</td>
</tr>
<tr>
<td>Herdocia (1991)</td>
<td>Proposed a simplified geometrical model for the calculation of geometrical factors affecting the stability of hanging walls in an inclined ore body using sublevel caving method. This limit equilibrium model was used to evaluate the hanging wall stability at Grängesberg, Kiruna and Malmberget mines, in Sweden.</td>
</tr>
<tr>
<td>Singh et al. (1993)</td>
<td>Carried out a study using numerical models (FLAC) to analyse the progressive development of fractures in the hanging wall and footwall with increase in mining depth in sublevel caving. They postulated a conceptual discontinuous subsidence model for an inclined orebody using sublevel caving method. The analysis was carried out at Rajpura Dariba mine, in India, and Kiruna mine, in Sweden, where steeply dipping ore bodies are extracted by sublevel caving.</td>
</tr>
<tr>
<td>Lupo (1996)</td>
<td>This model considers the failure of the hangingwall using the limit equilibrium equations derived by Hoek (1974) but considering an active earth pressure coefficient, and the limit equilibrium equations derived by Hoek (1970) for excavated slopes in open pit mines to analyse the footwall. The use of an active earth pressure coefficient is intended to include the effect of the movement of broken rock during draw. This method was applied to the conditions at the Kiruna mine, in Sweden.</td>
</tr>
<tr>
<td>Karzulovic et al. (1999)</td>
<td>Performed a study using Karzulovic’s model (1990) to predict the evolution of the horse-shoe shaped subsidence crater at El Teniente mine in Chile, and numerical models (FLAC) to assess the extent of the influence zone. The angle of break value so calculated was adjusted to take into account of local factors such as the presence of faults and the amount of broken material in the crater. The numerical models were calibrated against field observations to define the limits of the influence zone.</td>
</tr>
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Table 1.2. Most commonly applied numerical methods for rock mechanics problems.

<table>
<thead>
<tr>
<th>Continuum methods</th>
<th>Discontinuum methods</th>
<th>Hybrid continuum / discontinuum models</th>
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<tr>
<td>• Finite Difference Method (FDM)</td>
<td>• Discrete Element Method (DEM)</td>
<td>• Hybrid FEM/BEM</td>
</tr>
<tr>
<td>• Finite Element Method (FEM)</td>
<td>• Distinct Element Method (DEM)</td>
<td>• Hybrid FEM/DEM</td>
</tr>
<tr>
<td>• Boundary Element Method (BEM)</td>
<td>• Particle Flow Code (PFC)</td>
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Table 1.3. Numerical studies of surface subsidence

<table>
<thead>
<tr>
<th>Author(s)</th>
<th>Code</th>
<th>Type of analysis</th>
</tr>
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<tbody>
<tr>
<td>Singh et al. (1993)</td>
<td>FLAC</td>
<td>Site specific: Rajpura Dariba and Kiruna mines</td>
</tr>
<tr>
<td>Karzulovic et al. (1999)</td>
<td>FLAC</td>
<td>Site specific: El Teniente mine</td>
</tr>
<tr>
<td>Reported by Van As (2003)</td>
<td>FLAC 3D</td>
<td>Site specific: Northparkes mine</td>
</tr>
<tr>
<td>Cavieres et al. (2003)</td>
<td>3DEC</td>
<td>Site specific: El Teniente mine</td>
</tr>
<tr>
<td>Flores &amp; Karzulovic (2004)</td>
<td>FLAC &amp;</td>
<td>Conceptual</td>
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<tr>
<td></td>
<td>FLAC 3D</td>
<td></td>
</tr>
<tr>
<td>Gilbride et al. (2005)</td>
<td>PFC 3D</td>
<td>Site specific: Questa mine</td>
</tr>
<tr>
<td>Brummer et al. (2006)</td>
<td>3DEC</td>
<td>Site specific: Palabora mine</td>
</tr>
<tr>
<td>Elmo et al. (2007)</td>
<td>FLAC 3D</td>
<td>Site specific: San Manuel mine</td>
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<tr>
<td>Villegas (2008)</td>
<td>Phase2,</td>
<td>Site specific: Kiruna mine</td>
</tr>
<tr>
<td></td>
<td>PFC 2D</td>
<td></td>
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<tr>
<td>Sainsbury et al. (2008)</td>
<td>FLAC 3D</td>
<td>Site specific: Palabora mine</td>
</tr>
<tr>
<td>Beck &amp; Pfitzner (2008)</td>
<td>ABAQUS</td>
<td>Site specific: Several mines</td>
</tr>
<tr>
<td>Vyazmensky et al. (2010b)</td>
<td>ELFEN</td>
<td>Site specific: Palabora mine</td>
</tr>
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<td>Elmo et al. (2010)</td>
<td>ELFEN</td>
<td>Site specific: Cadia East</td>
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<tr>
<td>Sainsbury et al. (2010)</td>
<td>FLAC 3D</td>
<td>Site specific: Grace Mine</td>
</tr>
</tbody>
</table>
Fig. 1.1. 2005 failure of the northwest wall at Palabora following breakthrough of the cave into the pit floor. Photograph courtesy of Rio Tinto Technical Services.

Fig. 1.2. Classification of underground mining methods (from Brady & Brown, 2006)
Fig. 1.3. Illustration of the block cave mining method (from Hamrin, 1982)

Fig. 1.4. Conceptual model of block caving-induced subsidence and corresponding points from associated text above (from Van As et al., 2003, based on Abel & Lee, 1980).
**Fig. 1.5.** Subsidence deformation zones as defined by Van As et al. (2003)

**Fig. 1.6.** Relative importance of the main causes of subsidence in underground mines by caving methods suggested by Flores & Karzulovic (2002)
Fig. 1.7. Laubscher’s (2000) Mining Rock Mass Rating (MRMR) system, an empirical design chart, for assessing cave angle (angle of break) based on measurable geological parameters.
Fig. 1.8. Idealized model used in limit equilibrium analysis of progressive hangingwall caving proposed by Hoek (1974). After Flores & Karzulovic (2004).

**Symbols**

- $H_1$: Previous mining level depth
- $H_2$: Current mining level depth
- $H_s$: Depth to the caved material surface
- $H_c$: Caved material height
- $z_1$: Previous tension crack depth
- $z_2$: New tension crack depth
- $\psi_0$: Dip of the orebody
- $\psi_b$: Break angle
- $\psi_{p1}$: Inclination of the previous failure plane
- $\psi_{p2}$: Inclination of the new failure plane
- $\phi_w$: Friction angle between caved material and rock wall
- $\theta$: Inclination of the line of action of $T$
- $W_c$: Weight of the caved material
- $W$: Weight of the potentially unstable block
- $T$: Lateral force due to $W_c$ on the potentially unstable block
- $T_c$: Lateral force due to $W_c$ on the footwall
- $\gamma_c$: Unit weight of the caved material
- $\gamma$: Unit weight of the rock mass
- $c$: Cohesion of the rock mass
- $\phi$: Angle of friction of the rock mass
Fig. 1.9. (a) Finite-element modelling of cave-pit interactions (Beck & Pfitzner, 2008), and (b) 3DEC modelling of cave-pit interactions (Brummer et al., 2006).
Fig. 1.10. Design chart for estimating the angle of break in a transition from open pit to underground mining by block/panel caving for rock masses of different geotechnical quality and undercut level (UCL) depths in the range from 600 to 1700 metres; Poor to Fair: GSI=30-50, Fair to Good: GSI=50-70, and Good to Very Good: GSI=70-90 (GSI=Geological Strength Index) (Flores & Karzulovic, 2004).
Fig. 1.11. Cave-pit interaction in the Palabor mine simulated by hybrid finite-element/discrete-element modelling technique (Vyazmensky et al. 2010b). (a) NW-SE section of the Palabora mine; (b) Pit slope deformation at cave breakthrough for model P1 (with a tensile strength increase of 100% in the foskorite and 150% in the micaceous pyroxemite); (c) Pit slope deformation at 40% ore extraction for model P1; (d) Pit slope deformation at cave breakthrough for model P2 (with a tensile strength increase of 150% in the foskorite and 200% in the micaceous pyroxemite); and (e) Pit slope deformation at 40% ore extraction for model P2.
Chapter 2: Empirical Investigation and Characterization of Surface Subsidence Related to Block Cave Mining

2.1 Introduction

Block caving is increasingly being favoured as a mining method for maximizing Net Present Value (NPV) from large, lower grade ore bodies, especially as companies target deeper resources or transition underground from open pits that have reached the end of their mine life. As a mass mining method, block caving results in significant ground collapse and extensive surface deformations. Yet despite having been in use for more than 100 years, there has been limited research conducted regarding the impact of caving on surface subsidence. Of concern is the locating of mine infrastructure on surface or the impact ground deformations may have on protected areas neighbouring the mine property. Damage of surface infrastructure, together with increased dilution due to larger than expected caving angles, are often the cause for additional capital and operation expenditures.

To better understand and assess these potential geo-risks, a comprehensive database has been developed based on a thorough review of all available (i.e. public domain) sources reporting subsidence values related to both historic and present-day cave mining operations (including block, panel and sublevel caving). Empirical databases provide a means to learn from case histories, discover causal relationships between different contributing factors, establish guidelines for design, and to help provide a starting point to undertake more sophisticated analyses like numerical modelling. One of the most commonly cited is Laubscher’s method (Laubscher, 2000). Laubscher proposed a design chart (Fig.1.7) that relates the predicted cave angle to the rock mass quality (defined using the Mining Rock Mass Rating, or MRMR), density of the caved rock, height of the mined block and mine geometry (minimum and maximum span of a footprint). The resulting prediction by default assumes symmetry; i.e., the caving angle is equally projected from all points around the

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1 Woo, KS. Eberhardt, E. Elmo, D. Stead, D. Empirical investigation and characterization of surface subsidence related to block cave mining. (To be submitted)
perimeter of the undercut. The application of Laubscher’s method requires sound engineering judgement and a full consideration of the geological and geotechnical setting in which it is being applied.

The caving angle referred to by Laubscher is defined by Van As et al. (2003) as the angle of the line extending from the edge of the extraction level to the edge of the zone of active caving (Fig.2.1). The caved zone is usually located directly above the undercut footprint and thus is characterised as having the greatest surface disturbance, usually manifested as a crater filled with broken, irregular blocks. Van As et al. (2003) also defined two further subsidence zones and corresponding angles: the fracture initiation angle and subsidence angle (Fig.2.1). The fracture initiation angle is the angle measured from horizontal of the line extending from the edge of the extraction level to the edge of the zone of fracture (or zone of active movement). This zone encompasses all obvious surface deformations adjacent to the caved zone, typically characterized by large radial cracks and rotated and toppling blocks. The angle of subsidence marks the outer most zone and the limits of measurable surface deformations on surface. These are generally described as elastic or continuous non-elastic strains, with vertical displacements greater than 2 mm.

The empirical database presented here was developed to more fully examine the relationships between these zones of surface subsidence and depth of undercut, together with the key factors that influence them. Data relating to geology, topography, orebody type and undercut geometry were specifically targeted to analyze their effects in promoting asymmetry and discontinuous caving-induced subsidence. Where key relationships are revealed, illustrative numerical models are used to help draw conclusions to guide preliminary assessments during the planning stages of future new mining projects.

2.2 The UBC Block Caving Subsidence Database

A thorough search of the published literature, university theses, and government reports (e.g. U.S. Bureau of Mines) was carried out leading to a cave mine database for empirical analysis and characterization of caving-induced surface subsidence. The database is populated by more than one hundred cave mining operations throughout the world including both historic mines that have ceased to operate and those still producing. A tabular
format adopted for the database is designed to systematically display diverse basic information on a mine including its location, undercut depth and geology, combined with measurements related to macro- and micro-strain surface deformations. Although the study was primarily directed towards block and panel caving operations, data from sub-level caving operations were also collected.

2.2.1 General Trends

Fig.2.3 shows the breakdown of cave mines by continent, mining method, and resource mined. The majority of operations reported are North American (Fig.2.2a), although these are mostly historic involving the iron mines of Michigan, where the method was first developed, the copper mines of Arizona, and asbestos mines of Quebec. Currently developing or operating cave mines are more globally distributed between South America, Asia, Australia, Africa and North America. When it comes to mining method, 62 percent of the cases involve block caving with 19 percent using sublevel caving to adapt to steeply dipping orebodies of narrower width (Fig.2.2b). Grouped with sub-level caving are mines that combined sub-level caving with similar methods like top slicing and shrinkage stoping. The reported use of two caving methods in tandem – block caving plus sublevel caving for example – were found where it was advantageous to optimize the operations relative to variations in the shape of the orebody. As for minerals produced by these mines (Fig.2.2c), copper and gold form the majority at 29 and 15 percent, respectively, followed by asbestos (9%) and diamond (9%). The large number of copper-based caving operations reflects the favorability of block and panel caving for mining low grade copper porphyry ore deposits.

Based on these data, two interesting trends are evident. Fig.2.3 shows the changing trend in block height being caved. Before 1950, block caving was typically applied to block heights between 20 and 100 m at a time, employing multiple lifts of increasing depth where the height of the ore column was greater. However, this trend has transitioned to larger block heights exceeding 100 m to reduce development costs as confidence has been gained in draw sequencing practices that help minimize dilution by steering and maintaining cave propagation within the ore column. In step with increasing block heights being mined, undercut depths are likewise increasing. Fig.2.4 shows the range of undercut depths prior to
1950 as being 100 to 300 m, gradually increasing to depths at present of 600 m or deeper. Similarly, the size of the undercut (i.e. in plan view) has also increased as operations moves towards developing large panel caves instead of smaller blocks.

2.2.2 Caving-induced Subsidence Data

The use of block caving was first reported in 1895 in the Michigan iron and copper mines where large blocks of ore were undercut, allowing the ore to mine itself under gravity and crush through comminution to a size suitable for handling (Bucky, 1945). Soon after, the economic advantages gained by the method were being tempered by reports of its impact on surface and the need to better understand the factors controlling ground movements to help safeguard against property damage and loss of life (Crane, 1929). Several detailed studies were carried out with these and other historic mines, but given the total number of mines populating the UBC database, those directly reporting subsidence measurements are actually few in number.

This is reflected in earlier databases on subsidence related to mass mining (Table 2.1). Flores and Karzulovic (2002) carried out the first benchmark study as part of the International Caving Study Stage II (ICS-II), citing 242 break angles measured at various depths from 11 block, panel and sublevel caving operations. Most of these involved operations that transitioned to underground from open pit mining. For scoping and prefeasibility use, they suggest typical caving angles of >45° and >60° for MRMR values <70 and >70, respectively. Van As et al. (2003) systematically tabulated information for a number of mines including rock type, ore body dip, depth, caving angle, and angle of subsidence. Their treatment included 19 caving operations together with data from several stoping and room and pillar operations. A similar compilation was reported by Tetra Tech (2006) providing caving angle and angle of draw (defined as the greatest extent of affected ground). They note that only 20% of the mines they reviewed experienced unexpected subsidence, with most anomalies arising from geologic structure such as faults.

Although these existing databases provide a starting point and means of verification, original sources were consulted for each entry to review and extract the data first hand. One of the limiting factors of the previous databases is the consideration of only those sources
that report subsidence data directly. This was seen to involve only 5 percent of the caving operations populating the UBC database. Closer inspection of the different published sources for each mine property revealed that in many cases, detailed cross-sections were provided that contained indirect information relating to the disturbance on surface caused by caving. In many cases, a caving angle could be measured from a scaled map or section and in some cases, a fracture initiation angle. The use of indirect data increased the number of mine properties accounted for to 44, with the number for block and panel caves (28) tripling those reported in previous databases. Furthermore, in several cases, multiple observations were provided for the same mine property, either for multiple blocks or different mine levels again almost doubling the number of data points considered (see totals in parentheses in Table 2.1).

2.3 **Database Analysis: Caving and Fracture Initiation Angles**

From the database, a subset of 47 direct and indirect subsidence observations were analyzed to determine the caving and fracture initiation angles for each. These are reported in Appendix A. References are provided for each data entry and a detailed background description for each is provided in Appendix B. Excluded from the analysis were those operations involving caving into a deep open pit. Where several angles are reported for different stages of cave development, only the greatest values (worst case) are reported. Emphasis was also placed on data provided in the form of cross-sections or plan view maps showing the extent of caving, surface cracks, or subsidence (see Appendix B for examples of data sources used). It was found that in many cases what was reported as a break angle or caving angle by the author(s) was actually the angle of draw (90° minus caving angle) as estimated underground, as opposed to that considering the propagation of the cave to surface and the corresponding angle of its surface expression. Based on the definitions in Fig.2.1, these were corrected where required.

**Figs.2.5 and 2.6** plot the caving and fracture initiation angles determined as a function of undercut depth for the entire dataset including sub-level operations. **Fig.2.5** shows a rather wide range of caving angles among the sub-level caving mines as the dip of the orebody causes a large variation between the caving angle seen on the footwall side and that seen on the hangingwall. To eliminate the influence of orebody dip specific to sub-level caving
operations, these were excluded from subsequent analyses. **Fig.2.7** and **Fig.2.8** shows the relationships between caving and fracture initiation angles versus undercut depths in block and panel caving operations. Caving angles are generally seen to vary between 70 and 95°, where angles greater than 90° indicate overhanging angles (i.e. the extent of the zone in question fall within the footprint of the undercut). Angles for fracture initiation are broader and generally vary from 55 to 80°.

### 2.3.1 Influence of Topography

**Fig.2.9** shows the relationship between caving angle and surface topography. The surface topography considered is classified based on visual observation into two groups: generally flat (regular) topography and irregular topography where the mine is situated beneath a mountain peak(s) or slope/flank. Although the trend is varied, in general, the influence of a more irregular topography is seen to result in lower caving angles as well as a larger range in angles measured. A larger range in angles signifies a greater degree of asymmetry in the subsidence profile. This is reflected in Google Earth satellite images collected for the different mine sites in the database. Those for caving operations under relatively flat topography, for example Northparkes (**Fig.2.10a**), tend to show more symmetry in the shape of the caving zone on surface, whereas those under mountainous topography, for example Henderson (**Fig.2.10b**), tend to be more irregularly shaped.

The influence of topography can also be clearly demonstrated using comparative numerical models. Typical surface profiles, relative to the location of the undercut beneath, were derived based on inspection of those in the caving database. These were then examined using the 2-D finite-element code Phase2 (Rocscience Software, 2009). All input parameters were kept the same, including a conceptualized geology involving a joint network of varied persistence and spacing, two bounding faults to either side of the undercut, and several geological units assigned typical rock mass properties. An orthogonal joint pattern was adopted so as to not introduce asymmetry through dipping joints. The undercut depth was kept approximately the same in each model (1500m), as was the block height caved (500m). Simulation of caving was undertaken by incrementally changing the properties of the elements above the undercut from those of rock to those of caved rock. A horizontal to
vertical stress ratio of 2 was assumed. Full details of the model setup are reported in Chapter 3, and are only presented here in a summarized form for illustrative purposes.

The modelling results show that when assuming a flat topography (Fig.2.11a), both the caving and subsidence angles are similar on both sides of the undercut (i.e. symmetry). A similar result is obtained where the topography is irregular but approximately symmetrical relative to the position of the undercut (Fig.2.11b). This case represents a caving operation directly beneath a mountain peak with sloping flanks at different angles. The presence of the slopes above either side of the undercut results in a broader caving zone compared to the flat topography case. Fig.2.11c-e represents scenarios where the undercut is located beneath different slope configurations. The influence of a slope on the caving and subsidence angles to the left and right of the undercut is clearly visible for these different cases, with the upslope side experiencing notably more subsidence. As the cave propagates towards surface, it undermines the slope on the uphill side promoting gravity driven down slope movements towards the cave. Thus, the empirical and numerical analyses show that symmetric surface conditions generally lead to symmetric subsidence patterns; whereas, asymmetric surface conditions in the form of a sloping surface above the undercut results in cave-surface interactions that draw cave propagation in the uphill direction resulting in asymmetric subsidence. A similar observation was made by Benko (1997) who conducted a study investigating the influence of a slope on surface subsidence above a longwall coal mine. According to Benko, surface subsidence above longwall operations where the topography is relatively flat tends to be symmetric while surface subsidence above mines where a slope is present shows a pattern of greater subsidence developing in the upper part of the slope.

As for the influence of faults, in cases where the surface topography above the caving area is symmetric (Fig.2.11a, b), the area of subsidence exceeding 5 metres (see contours color coded in blue) does not extend beyond the fault interfaces. The faults effectively constrain/limit the large-strain subsidence. Where an irregular topography is present (Fig.2.11c-e), however, the area of subsidence exceeding 5 metres does extend beyond the boundary faults. This indicates a greater influence of topography on surface subsidence despite any limiting influence the faults may present. A similar observation was made by
Vyazmensky et al. (2010a) who conducted an extensive investigation of the influence of faults on block caving induced surface subsidence.

### 2.3.2 Influence of Orebody Characteristics

Data on the site geology for the different cases populating the database were limited to that provided in the different sources consulted. For a number of these, there was no geology data reported and an alternative source was used instead to obtain basic geological information for the given block cave mine property. The lack of detailed data prevented any extensive analysis into the influence of geological factors on caving angle, and instead, correlations were drawn using the only information that was consistently provided – that of the ore resource being mined. Further development of the database to populate it with more detailed geological data may make it possible to better clarify and separate relationships between undercut depth, caving angles and geological influences. However, for the purpose of the analysis carried out in this study, the ore body resource was used as a simple proxy for mine geology.

**Fig.2.12** plots the relationship between undercut depth versus caving angle for block and panel caving operations as a function of the resource being mined. In general, diamond, iron, nickel and asbestos operations are seen to have steeper caving angles signifying a smaller impact footprint on surface. This is due in part to the typical shapes of these orebodies, which tend to be narrow and vertical, combined with strength contrasts between the weaker ore being caved and the stronger host rock. For example, diamonds are predominantly mined from vertical kimberlite pipes. These are typically intruded into a stronger host rock, meaning that caving tends to follow the boundaries of the vertical orebody resulting in steep and symmetric caving angles. Symmetry in the caving angles is signified in **Fig.2.12** by a narrow range of caving angles, with those for diamond kimberlite and upturned bedded iron deposits rarely varying by more than 10°. In contrast, copper operations generally involve porphyry deposits that are more irregular in shape and have less contrast between the strength of the ore and host rock. As such, the caving angles can vary from 90° on one side of the undercut to 65° on the other side.
Using ore body type and mineral resource as a proxy for geology, Fig.2.12 shows that site geology has a significant influence in promoting asymmetry in caving-induced displacements. Similar to the influence of topography, the influence of geology is observable in the Google Earth satellite images in the UBC database (Fig.2.13a and Fig.2.14a). Fig.2.13a is the satellite image for the Kimberley diamond mine (kimberlite pipe). The caving zone shown in the Google Earth image is approximately symmetric in shape centered by a glory hole, which agrees with the symmetric geological distribution surrounding the kimberlite pipe illustrated in the geology cross section in Fig.2.13a. This can be compared to Fig.2.14a, which shows the outline of caving for the San Manuel copper mine. In this case the caving zone is highly irregular consistent with the asymmetric geological makeup of the mine geology presented in Fig.2.14b. These observations are consistent for like cases in the database.

2.4 Discussion: Influence of Undercut Depth

A thorough examination of the block and panel caving subsidence data compiled shows that the distribution of data is heavily weighted towards caving angles and macro deformations, Very little data is reported on the extent and magnitudes of smaller strain surface deformations (also known as micro deformation). This empirical bias towards macro-deformations is largely a function of the measurement resolution available at the time of the investigation. The majority of the detailed investigations reporting on caving-induced ground deformations are more than 50 years old, and as such, rely heavily on visual mapping observations and low-resolution levelling surveys. Furthermore, the focus of the reported investigations was primarily placed on the area immediately above the undercut, thus characterizing the caving zone, and in some cases extending the survey outwards towards the edges of mine property, thus characterizing the fracture initiation zone.

To examine the potential impact of this sampling bias better, specifically with respect to the influence of undercut depth on the extent of surface subsidence, a series of conceptualized numerical models were developed. To be able to fully compare both discontinuous zones of macro-deformations (caving and fracture initiation angles) and small
strain micro-deformations (subsidence angles) a hybrid FEM-DEM approach incorporating brittle fracture capabilities was adopted using the commercial code ELFEN (Rockfield Software, 2009). ELFEN allows for the representation of the pre-caving geological domain as a continuum populated by discrete fractures representing a brittle fracture network, that then may undergo subsequent fracturing in response to the stresses and strains induced through undercutting and cave propagation. The technique has been shown by Vyazmensky et al. (2010a) as being well-suited for capturing important block-caving mechanisms, including preferential rock fragmentation within the ore column and influence of geological structures on cave development and surface subsidence. Full details of the models shown here are provided in Chapter 3, but in summary for illustration of the effects of undercut depth, involve: a joint network of varied persistence and spacing, two bounding faults to either side of the undercut, and several geological units assigned typical rock mass properties. The assumed presence of two bounding faults was adopted as it facilitates the basic examination of how faults influence small-strain subsidence in 2-D numerical analysis. In addition, Vyazmensky et al. (2008) already analyzed the influence of shallow dipping faults on caving and fracture initiation angles, which were examined as a function of the distance between the caving area and the faults. Accordingly, this study chose steeply dipping faults and a vertical joint set to control the effect of faults and joints highlighting the role of undercut depth. As before, an orthogonal joint pattern was adopted so as to not introduce asymmetry through the presence of inclined dipping joints. A horizontal to vertical stress ratio of 2 was assumed, and caving was simulated for a block height of 200 m.

Fig.2.15 shows the results for undercut depths of 500 and 2000 m with undercutting from right to left. In both cases, the zone of caving is largely constrained by the presence of the bounding faults. For the 500 m deep undercut, where the 200 m high ore column represents a 40% extraction ratio, the impact on surface involves a caving zone extending from the edges of the undercut to the bounding faults (i.e., between 75 and 90°). For the 2000m deep undercut, the caving angles are actually overhanging (>90°). This can be explained by the lower extraction ratio (10% extraction) when assuming the same height of the ore column being caved (200 m), and therefore a less extensive cave developing and daylighting at surface. This agrees with the observations in the UBC block caving database.
where the caving angle is seen to increase with undercut depth and the two cases involving the deepest undercuts (> 1000 m) involve overhanging caving angles.

However, when considering the extent of smaller strain deformations (<1 m), the opposite is true and the subsidence angle decreases with increasing undercut depth. For the 500 m deep undercut, the subsidence angle only partly extends beyond the bounding faults and is not significantly different from the caving and fracture initiation angles. In contrast, the zone of subsidence for the 2000 m undercut extends well beyond the bounding faults and is much farther reaching.

This has important practical implications. If the location of critical infrastructure, or similarly a hazard assessment of the extent of caving-induced ground deformations, is based on empirical data then these will be biased towards observations of large-scale ground disturbance and collapse and would suggest that the impact of caving on surface is reduced for deeper undercuts. However, smaller-strain subsidence (< 1 m) may be of equal concern and its extent actually increases with undercut depth. These results therefore caution against relying on existing empirical design charts and databases for estimating the extent of caving-induced subsidence where small strain subsidence is of concern as the data being relied on does not properly extrapolate beyond the macro deformations (i.e. caving angles) that make up the majority of the observations.

2.5 Conclusions

A detailed and comprehensive database of cave mining operations and caving-induced ground deformation observations has been developed to guide empirical relationships between caving depth and its impact on surface. The data shows that asymmetry in caving-induced subsidence is prevalent and largely controlled by topography and geology of the ore deposit and host rock. Where design calculations are carried out using methods that assume, directly or indirectly, symmetrical ground deformations relative to the projection of the undercut footprint at surface, caution must be taken so as to not under-predict their magnitudes and reach.
The availability and quality of subsidence data was also seen to be deficient as little attention has been paid to the measurement of subsidence angles compared to caving angles. The data on caving angles suggests that as undercut depth increases, the magnitude and extent of the caved zone on surface decreases. However, numerical modelling results indicate that the opposite is true with respect to smaller strain deformations (< 1 m) and that subsidence angles increase and are farther reaching with increasing undercut depths. The results therefore caution against relying on existing empirical design charts and databases for estimating the extent of caving-induced subsidence where small strain subsidence is of concern, as the data being relied upon does not properly extrapolate beyond the macro deformations (i.e. caving angles) that make up the majority of the observations. Thus, with the new generation of deep block/panel caving projects being planned, and the higher geo-risk profiles being carried due to the capital investments and development times required, the need is clear for more detailed measurements to better understand cave-surface interactions as a function of undercut depth and potential asymmetry.
**Table 2.1.** Comparison of previous databases reporting subsidence data for mass mining operations

<table>
<thead>
<tr>
<th>Mining Method</th>
<th>Number of Operations (Total Observations)</th>
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<tr>
<td>Block &amp; Panel Caving</td>
<td>9 (229)</td>
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<tr>
<td>Sublevel Caving/Shrinkage Stoping</td>
<td>1 (4)</td>
</tr>
<tr>
<td>Open Stope Caving</td>
<td>-</td>
</tr>
<tr>
<td>Caving (Unspecified)</td>
<td>-</td>
</tr>
<tr>
<td>Other Stoping (Sublevel, Cut &amp; Fill)</td>
<td>-</td>
</tr>
<tr>
<td>Room and Pillar</td>
<td>-</td>
</tr>
<tr>
<td>Unspecified</td>
<td>1 (9)</td>
</tr>
<tr>
<td>Total</td>
<td>11 (242)</td>
</tr>
</tbody>
</table>
Fig. 2.1. Definition of block caving deformation zones as defined by Van As et al. (2003)
Fig. 2.2. General breakdown of cave mining data in database by: (a) regional distribution (b) mining method (BC = block cave, PC = panel cave, SC = sublevel cave), and (c) resource mined. Reported are the relative percentages followed by the total number of cases in parentheses. Symbols in each legend are ordered from highest percentage to lowest.
Fig. 2.3. Breakdown of block heights being mined for the block caving cases in the database, showing a trend towards the mining of larger blocks.
Fig. 2.4. Breakdown of undercut depths associated with the block caving cases in the database showing a trend towards the development of deeper undercuts.
Fig. 2.5. Undercut depth versus caving angle for block, panel, and sub-level caving operations. Each line segment represents the range in caving angles measured from different sides of the undercut; the greater the range the higher the degree of asymmetry.
Fig. 2.6. Undercut depth versus fracture initiation angle for block, panel, and sub-level caving operations. Each line segment represents the range in fracture initiation angles measured from different sides of the undercut; the greater the range the higher the degree of asymmetry.
Fig. 2.7. Undercut depth versus caving, fracture initiation and subsidence angles for block caving operations. Each line segment represents the range in angles measured from different sides of the undercut; the greater the range the higher the degree of asymmetry.
Fig. 2.8. Undercut depth versus caving, fracture initiation and subsidence angles for panel caving operations. Each line segment represents the range in angles measured from different sides of the undercut; the greater the range the higher the degree of asymmetry.
Fig. 2.9. Undercut depth versus caving angle for global block and panel caving operations, colour coded according to the general characteristics of the surface topography. Each line segment represents the range in caving angles measured from different sides of the undercut; the greater the range the higher the degree of asymmetry.
Fig. 2.10. Influence of topography observed visually in Google Earth satellite images: (a) flat topography (Northparkes mine, Australia), and (b) irregular topography (El Teniente mine, Chile). Surface subsidence area is marked by dashed line.
Fig. 2.11. Finite-element modelling of influence of topography on caving-induced subsidence assuming typical surface profiles visually identified in the UBC database. The same geological inputs used for the comparative models in Chapter 3 were applied. In order of increasing degree of assymetry (cave-surface interactions) relative to the position of the undercut below, these are: (a) flat topography, (b) mountain peak, (c) rising slope ending in a mountain peak, (d) rising slope, and (e) slope with plateau. Note that grey shaded area approximates zone of caving.
Fig. 2.12. Undercut Depth versus caving angle for global block and panel caving operations, colour coded according to the resource being mined. Each line segment represents the range in caving angles measured from different sides of the undercut; the greater the range the higher the degree of asymmetry.
Fig. 2.13. Symmetric surface subsidence observed in association with a vertical kimberlite pipe - Kimberley diamond mine: (a) Google Earth satellite image, and (b) geological cross section (De Beers, 2008).
Fig. 2.14. Asymmetric surface subsidence observed in association with a copper porphyry ore deposit - San Manuel mine: (a) Google Earth satellite image, and (b) geological cross section (Sandibak, 2004).
Fig. 2.15. Caving-induced brittle fracture and subsidence for undercut depths of: 500 m (top), and 2000 m (bottom). The continuous black lines to the left and right of the undercut are bounding faults.
Chapter 3: Benchmark Testing of Numerical Methods for Modelling the Influence of Undercut Depth on Caving-Induced Subsidence

3.1 Introduction

Block caving represents a low cost underground mass mining method with production tonnages that are competitive with those from open pit operations. However, the method results in significant disturbance to the surface environment. Damage to and subsequent replacement of surface and underground infrastructure due to differential caving-induced ground movements are often cause for additional capital and operation expenditures as well as for environmental degradation. These include both large scale deformations, defined by the caving and fracture initiation zones (Van As et al., 2003), and small-strain deformations, defined by the continuous subsidence zone (Fig.2.1). The latter is difficult to directly measure and empirical guidelines are focused on the former. Accordingly, the analysis of subsidence is often based on numerical modelling.

The complexity of geology and its brittle tectonic overprint (jointing, faulting, etc.) make rock masses a difficult material for mathematical representation via numerical modelling. Numerical techniques vary widely in their representation of geology ranging from continuum methods (e.g. finite element, finite difference), where the presence of fabric is treated implicitly through constitutive models and input properties resulting in a more computationally efficient solution especially in 3-D, to discontinuum methods (e.g. distinct element) that explicitly account for the presence of geological structures, but because of this added complexity, are more computationally demanding. A coupling of these two techniques (FEM-DEM) represents a further degree of sophistication, and computational effort, in which a continuum domain populated by discrete fractures may undergo subsequent fracturing in response to changing stresses and strains. These methods and their application are discussed in detail by Stead et al. (2006).

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2 Woo, KS. Eberhardt E. Elmo, D. Stead, D. Benchmark Testing of Numerical Capabilities for Modelling the Influence of Undercut Depth on Caving-Induced Subsidence. (To be submitted)
Due to their faster run times, continuum codes such as the finite difference programs FLAC and FLAC3D or the finite element program ABAQUS have seen the most widespread use in the modelling of block caving subsidence (Van As et al., 2003; Singh et al., 1993; Sainsbury et al., 2008; and Beck et al., 2008). To a lesser degree, distinct element codes like UDEC and 3DEC or coupled FEM-DEM codes like ELFEN have been used (Vyazmensky et al., 2010b and Elmo et al., 2010). Yet ELFEN has been shown by Vyazmensky et al. (2010b) to be particularly well-suited for capturing important block-caving mechanisms, including preferential rock fragmentation within the ore column and influence of geological structures on cave development and surface subsidence. Ultimately, the choice of method varies on a case-by-case basis, ranging from selection based on simplicity and familiarity in the modelling method to specific requirements to incorporate a key process such as brittle fracturing.

In this study, benchmark testing of several different numerical techniques was carried out to investigate their abilities and limitations with respect to modelling block caving subsidence for a range of undercut depths with added consideration of the influence of structural geology. Reviews of historic and current caving operations by Chapter 2 have shown that the average undercut depth has increased from 200 m to deeper than 600 m over the past 50 years. In effect, as near surface resources become exhausted caving operations are moving deeper and developing larger caves, raising questions as to their influence on the surface environment. The comparison carried out here focuses on a conceptual problem geometry involving a porphyry type deposit, incorporating faults, several different lithologies and varying rock mass properties. The undercut depths tested vary from 500 to 2000 m in increments of 500 m using the 2-D continuum code Phase2, 3-D continuum code FLAC3D, 2-D discontinuum code UDEC, and hybrid FEM-DEM brittle fracture code ELFEN.

3.2 Numerical Modelling Methodology

3.2.1 Model Geometry

One of the larger and deepest block/panel caving projects currently in development is the Resolution Copper Mine near Phoenix, Arizona. Fig.3.1 provides a cross section of the Resolution mine (Rio Tinto, 2008), showing that the undercut and production levels will be
located approximately 2,000 metres beneath surface. The conceptual geometry used for this study was specified by the Centre for Excellence in Mining Innovation (CEMI), Sudbury, Ontario, as part of a sponsored benchmark study, and is illustrated in Fig.3.2. Outlined in the 3-D panel caving geometry are several faulted lithologies, with the faults located adjacent to the undercut, similar to that experienced at Resolution (Fig. 3.1). The undercut is extensive and divided into three panels (A, B and C), mined from south to north (Panel A toward Panel C). The width of each panel is 500 m. For 2-D modelling, section A-B was used.

The external limits of the model were determined based on preliminary elastic modelling of the lateral extent of measureable surface subsidence for a 2500 m deep undercut. This resulted in a 2-D section 10,500 m in length and 4200 m in depth. For the 3-D geometry, the east-west width of the model is 10,000 m and the north-south width is 8000 m. The depth is 3000 m.

3.2.2 Numerical Modelling Software Used

Four different numerical modelling codes were applied to the benchmark problem: Phase2, FLAC3D, UDEC and ELFEN (Rocscience Software, 2009; Itasca Consulting Group, 2010; Itasca Consulting Group, 2009; and Rockfield Software, 2009). Phase2 is a 2-D finite-element code, the governing equations for which are based on continuum solid mechanics. Although simulation of discontinuities is possible through joint elements, these are restricted to small strain movements in which the pairing between two contacting joint interfaces does not change. In finite-element treatments applying joint elements, once interconnectivity between the two sides of a joint is established upon meshing, it remains unchanged throughout the solution process, despite any displacements that occur (Riahi et al., 2010). Phase2 was used here to model the problem in Fig.3.2 with and without joint elements and the results were compared to understand the influence of small-strain joint movements on the modelling of surface subsidence. Modification of the continuum model to include a discrete fracture network (DFN) involved two joint sets of variable spacing and persistence dipping at 0 and 90 degrees (Fig.3.3). These were only applied from the undercut level up to the surface and extending outwards at an angle of 40 degrees from the edges of the undercut. The joint angles of 0 and 90 degrees were selected to minimize the degree of asymmetry that would be
introduced through the DFN to enable a more direct comparison to models without the DFN. Vyazmensky et al. (2010a) demonstrated the importance of inclined joints on influencing the lateral extent and asymmetry of surface subsidence.

Continuum modelling was subsequently extended into 3-D using the finite-difference code FLAC3D (Fig.3.4). While 2-D codes are effective in analyzing cross-sections pertaining to an area of interest, this requires that a plane strain assumption is reasonably applicable. A typical block caving mine operation does not necessarily fit this condition. As both a 3-D and continuum analysis tool, FLAC3D does not easily allow the introduction of joints into the model. Doing so requires each discrete feature to be manually specified through matching interfaces, entailing significant pre-processing preparation time. In this case, only the large bounding faults were inserted into the FLAC3D model, with the added simplification of using vertical interfaces (Fig.3.4). Beyond the continuum treatment of the problem, all further model runs involving discontinuum techniques were modelled in 2-D given the prohibitive pre-processing and computing times required for 3-D.

The influence of joint sets was further analyzed using UDEC, a 2-D discontinuum analysis tool based on the distinct-element method. UDEC models the problem domain as an assemblage of interacting deformable blocks that simulates either the quasi-static or dynamic response to loading of a rock mass containing multiple, intersecting joint structures. This approach improves on the inclusion of joint elements in a continuum treatment like Phase2 as the joints included in a UDEC model can undergo large displacement offsets in shear or through opening and block rotation. Blocks between the joints can be modelled as either elastic or elasto-plastic deformable materials, and joints can also be modelled using a number of constitutive relationships. Here a simple Coulomb slip model was applied to the joints; blocks were modelled as elasto-plastic. Fig.3.5a,b shows the model geometry used in which the joint spacing varies from 10 m in the area of interest above the undercut and simulated cave, to 50 m towards the model boundaries. Because of the versatility and ease of UDEC in representing a joint network, an alternative geometry was also modelled in which the 0 and 90 degree joint pattern is superimposed with a second coarser pattern of joints at 45 and 135 degrees (Fig.3.5c) to increase the degrees of freedom for block slip and vertical subsidence.
The last numerical technique employed in this benchmark study involved the hybrid FEM-DEM brittle fracture code ELFEN. Here a similar geometry was used as in the Phase2D model with respect to the DFN imported into the continuum problem domain (Fig.3.6) (Eberhardt et al., 2011). This code enables the modelling of brittle fracture initiation and propagation within a continuum finite-element domain populated by discrete fractures through adaptive remeshing of the problem during time-stepping. ELFEN was used for explicitly modelling 2-D cave propagation through brittle fracturing as ore is extracted. This allows ELFEN, unlike the other codes, to visually depict the caving and fracture initiation angles directly.

Aside from their inherent differences, efforts were made to ensure these different codes were tested with identical input parameters and boundary conditions as much as possible.

3.2.3 Rock Mass Properties

Depth dependent rock mass properties were derived using the UCS, $m_i$ and GSI values reported in Table 3.1. These were provided by CEMI in association with this study. Scaling was carried out using Hoek et al.’s (Hoek et al., 2002) relationships based on the Hoek-Brown failure criterion. Because not all codes have fully implemented a Hoek-Brown constitutive model, the Hoek-Brown values were converted to Mohr-Coulomb by fitting an average linear Mohr-Coulomb relationship to the non-linear Hoek-Brown envelope (Table 3.2). This was done for a range of minor principal stress values with an upper bound of $\sigma'_{3\text{\text{max}}}$. The estimation of $\sigma'_{3\text{\text{max}}}$ for the problem geometry is complicated by differences in the stress field above the undercut, which involves extensive relaxation, and that away from the cave, which is more in line with the far-field in-situ stresses. To vary the rock mass strength properties as a function of depth, $\sigma'_{3\text{\text{max}}}$ values were estimated for each geological unit using stress values calculated by means of a Phase2 finite-element elastic stress analysis. Calculations were performed for each undercut depth assuming the presence of an undercut filled with caved material with properties that result in approximately 0.5 m of surface subsidence. This allows for a minor degree of stress relaxation above the cave back in assessing $\sigma'_{3\text{\text{max}}}$. Because the stresses vary between the top and bottom of each geological
unit, the higher $\sigma'_{3\text{max}}$ values in the stress range were adopted; the justification for this is that the cave must first propagate up through the bottom of each geological unit and therefore the rock mass properties derived for the higher $\sigma'_{3\text{max}}$ values are more appropriate. Tensile strength ($T_{rm}$) was estimated assuming a 25% tensile cut-off applied to the theoretical Mohr-Coulomb value and the rock mass deformation modulus, $E_{rm}$, was calculated using Hoek and Diederichs’ (Hoek et al., 2006) empirical relationship. A value of fracture energy ($G_f$) of 43 J/m² was applied for brittle fracturing in the ELFEN modelling.

Joint strength properties for the joint elements and discrete elements were assigned assuming a linear Mohr-Coulomb slip criterion. The North and South faults, the DFN joints and, in the case of ELFEN, any new fractures generated during the simulation of caving were given the same values: a joint cohesion of zero and joint friction angle of 30°. It should be noted that specific fracture sets will generally have markedly different surface morphologies and hence different shear strengths. However, this has not been considered in the current models to maintain similarity between the comparisons of the different codes. The mechanical contact forces that govern the interactions between discrete elements in UDEC and ELFEN can be loosely defined as the forces that are required to prevent blocks from interpenetrating. Contact forces are realised at contacting nodes and are evaluated by considering the relative kinematics of surface entities. The enforcement of these constraints is established using either a prescribed joint stiffness or penalty method, based on which proportionality between the degree of constraint violation and the degree of corrective measure is assumed. A normal stiffness/penalty of 4 GPa/m and tangential stiffness/penalty of 0.4 GPa/m were assumed.

While values reported in Table 3.2 were applied to all of the Phase2, UDEC and ELFEN models, an exception was made for the FLAC3D models where, to account for the larger mesh size, the rock mass properties were assumed to be lower by 75%. The full details of the material properties used in this benchmarking study are provided in Appendix C.

3.2.4 In-situ Stresses

The initial stress conditions were implemented as outlined by CEMI in a background document they provided (Fig.3.7) (Eberhardt et al., 2011). The vertical stress is defined as
gravitational loading with a rock unit weight of 27 kN/m³. The maximum horizontal stress \( \sigma_{H\text{max}} \) is set as 1.9 times the vertical stress and is aligned north-south. This corresponds to the in-plane direction for the 2-D model representations. The minimum horizontal stress \( \sigma_{H\text{min}} \), aligned east-west, is set as 1.2 times the vertical stress. For the 2-D models, this was used as the out-of-plane stress condition.

The external boundaries in all cases were set as zero displacement boundaries normal to the boundary, except for that representing the top surface. This was left as a free boundary.

### 3.2.5 Simulation of Draw and Caving

The simulation of cave propagation is of prime importance given its direct relationship with the caving-induced deformations being modelled. For the continuum methods, caving was implemented through an implicit approach where the geometry of the cave is built into the model, as opposed to explicitly modelling caving and cave propagation. This is a key limitation of applying continuum techniques to block caving problems. The implicit approach employed in the Phase2 and FLAC3D modelling assumes a cave geometry at several different points in its overall development over time, and incrementally changing the corresponding element properties from those of the ore to fragmented rock. A simplified cave geometry was assumed involving caving in 50 m increments up to a total block height of 200 m. Caving is initiated in Panel A and progresses through two stages (100 m cave height) before the next panel is initiated (Fig.3.8). This is continued for each panel until the cumulative height of caved rock area reaches 200 m.

Table 3.2 includes the material property assumed for the caved rock. As the material responsible for the relaxation and deformation of the surrounding rock, it was found to have a significant influence on the magnitude of the modelled displacements. To avoid numerical errors related to severe mesh distortion, the caved rock material was modelled as an elastic material, using reduced elastic properties to account for the reduced deformation modulus that would be expected for caved rock, as well as allowances for the presence of a small air gap. This procedure requires redefinition of the initial stress state within the modelled cave material to coincide with the self-weight of the caved material and not the locked in tectonic stresses initially prescribed for the ore and host rocks.
Simulation of the propagating cave in UDEC was carried out in a similar fashion. Although the discontinuum treatment of the problem domain allows for blocks in the undercut to be deleted, thereby more directly simulating the actual mining process, the contact detection algorithm used by the code is unable to create new contacts for detached/falling blocks. The algorithm is able to create new contacts when motion occurs along a discontinuity (i.e. shear), enabling large displacements to develop in the model. However, the UDEC data structure was designed specifically to model compact rock masses (i.e., tightly packed blocks) and not multiple blocks that lose contact with neighbouring blocks.

ELFEN modelling of the draw and caving process was carried out explicitly following procedures by Elmo et al. (2010). The algorithm removes all meshed elements whose centroids are located within a specified region, in this case corresponding to the undercut/production level. An iterative process is used such that the removal of elements is repeated continuously at a given numerical time step in order to return the specified draw rate (Fig.3.9). No hang ups are currently simulated at the extraction level since the simulated draw is designed to gradually fracture the rock passing through the extraction level (i.e., an ideal draw scenario is assumed in the model).

3.3 A Comparison of Numerical Analysis Results

3.3.1 Continuum Analysis (2-D versus 3-D)

Fig.3.10 shows the results from a 2-D continuum elasto-plastic analysis using the Phase2 finite-element program. Results are plotted as vertical displacement contours applying a minimum threshold of one metre. These can be compared against results obtained for the models that include the small-strain joint elements (Fig.3.11). The comparison reveals that the lateral extent of subsidence is similar for the shallower undercut depths, but increases more noticeably in the jointed models as the undercut depth increase (i.e. 3% and 8% wider for the 1500 m and 2000 m deep undercuts, respectively). This tendency can be attributed to the weaker rock mass conditions associated with the inclusion of the joint sets.
**Fig.3.12** presents the FLAC3D modelled subsidence. In general, the magnitude of subsidence seen in the 2-D continuum modelling (Phase2) is greater than that in the FLAC3D modelling. This is largely due to the 2-D plane strain condition and the added restraint provided by the third dimension in the FLAC3D model. Rocscience (2009) also cautions that the stress changes calculated based on the plane strain assumption show some exaggeration if the out-of-plane dimension of the excavation is not at least five times greater than the largest in-plane cross-sectional dimension. This is because the stress flow around the boundaries of the excavation is not taken into consideration. A final factor for the reduced subsidence magnitudes in FLAC3D is the absence of the joint sets; this is despite the reduced rock mass properties used to account for the need to use larger elements. Consequently, subsidence occurs to a lesser extent (approximately 75% less) in the FLAC3D model compared to the 2-D analyses. The larger mesh size also likely limited the extent of elasto-plastic yielding in the model. As a result, a lower bound contouring threshold of 0.25 m was used in **Fig.3.12** as opposed to the 1 m cutoff used in the other plots of the benchmarking results. Comparison can be made by observing the outline of the 1 m upper bound threshold used in **Fig.3.12** in blue.

The FLAC3D results for the 500 m deep undercut show that the modelled extent of subsidence, when a block height of 200 m is reached in Panel A (with block heights of 150 and 100 m in panels B and C, respectively), is limited by and bound by the faults on either side of the undercut. As the undercut depth increases, the subsidence profile widens and the influence of the bounding faults diminishes. This agrees with the observations made in the 2-D Phase2 results. It is also noteworthy that the shape of the subsidence profile elongates in the north-south direction as the undercut depth increases. This coincides with the longer axis of the panels being modelled (**Fig.3.2**), with the effect becoming more pronounced as the undercut depth increases. In addition, the varying ratio of panel size to undercut depth with caving propagation seems to contribute to asymmetric subsidence.

### 3.3.2 Discontinuum Analysis

The results from the UDEC discontinuum analysis are presented in **Figs.3.13** and 3.14. As previously noted, two different joint geometries were considered for the UDEC
discontinuum analysis. Fig.3.13 presents the subsidence results for the joint configuration similar to that used for the other 2-D modelling methods (0 and 90 degrees). These can be compared to the results in Fig.3.14, which shows the results assuming the same pattern of 0 and 90 degrees superimposed with a second joint pattern dipping at 45 and 135 degrees. In Fig.3.13, the subsidence zone is about 30% narrower than that seen in the previous model results. This is due to the continuous persistence of the orthogonal joint sets used, specifically that for the vertical joint set. Given the large strain slip afforded by the distinct element formulation, most of the rock mass response is therefore concentrated on these joints directly above the growing block cave. Hence, the lateral extent of the vertical subsidence is reduced. In contrast, the extent of subsidence in Fig.3.14 is significantly increased as the 45 and 135 degree joint angles promote freer ground movement (slip) in the lateral direction. Fig.3.14 shows this response to increase with increasing undercut depth. For the 2000 m undercut, the subsidence extends so broadly that it approaches and slightly interacts with the right boundary. The asymmetry observed in these contours is related to the direction of caving (Panel A to the south being developed first before progressing towards the north and Panels B and C).

The >5 m subsidence contour suggests that the other effect of the 45 and 135 degrees joint sets, relative to the 0 and 90 degree joint sets, is that although the lateral extent of subsidence increases, the magnitudes of subsidence that develop over the cave are reduced. These reduce further as a function of undercut depth, with plots for the 1500 and 2000 m deep undercuts showing that the saturated vertical displacements (> 5 m) does not extend to surface. In all cases (Fig.3.14), the results indicate that the zone of modelled caving is limited by the presence of the bounding faults despite the presence of the 45 and 135 degree joint sets; the extent of the smaller strain subsidence (1 m contour threshold) is seen to increase as a function of undercut depth despite the presence of the bounding faults.

3.3.3 Discontinuum with Brittle Fracture Results

The influence of undercut depth on subsidence using the hybrid FEM-DEM code ELFEN is captured in Fig.3.15. Brittle fracturing induced by caving is mostly limited
between the north and south faults. In contrast, the pattern of subsidence is similar to those in the other numerical models, widening with increasing undercut depth.

The key consideration in comparing the results for the different undercut depths is that the “volume” of excavated ore (or block height) in each model is the same. Since the column of rock above the undercut increases with increasing undercut depth, this means that the extraction ratio is decreasing. In the case of the 500 m deep undercut, 40% of the rock column above the undercut is caved. For the 2,000 m deep undercut, only 10% of the rock column is caved. Consequently, the modelling results for the 500 m deep undercut clearly show the importance of incorporating a "realistic" fracture network in the analysis. The natural fractures are shown to accommodate most of the caving-induced extensional strains. This, as well as the panel size to undercut depth ratio, changes as the undercut depth increases. The comparison of the subsidence profiles clearly indicates that the subsidence zone increases as a function of increasing undercut depth. The zone of caving remains within the bounding faults and its extent actually decreases with increasing undercut depth. This is due to the decreasing extraction ratio (i.e. draw height to overburden height for an equal volume of draw). In the shallow undercut, caving angle, fracture initiation angle, and subsidence angle are all similar, but in the deeper undercut, subsidence angle tends to be significantly smaller.

3.4 Discussion

The caving, fracture initiation and subsidence angles, as defined by Van As et al. (2003) (Fig.2.1), are reported in Fig.3.16-19 relative to their measurement on the south and north sides of the undercut. In agreement with the definitions by Van As et al. (2003), caving angles are the largest at around 90° or over followed by fracture initiation angles and subsidence angles. The caving angles identified in the Phase2, FLAC3D, and UDEC models were measured based on the yield zone with shear indicator. In the ELFEN models, the caving angles were directly measured. Fracture initiation angles in the FLAC3D models were assumed to be the mid-values between the caving and subsidence angles, since no tension yield indicators occur in FLAC3D modelling. This is because for shallower undercut depths, the fractured zone is observable on the surface and therefore the fracture initiation angle can
be explicitly measured, but for deeper undercut depths, the fractured zone does not appear on
the surface making the fracture initiation angle not possible to measure. One possible way to
estimate the fracture initiation angle in deeper undercut depths is to interpolate the fracture
initiation angle explicitly measured in shallower undercut depths based on the empirically
identified trend in Chapter 2 where the influence of caving activities on the surface
decreases as the undercut depth increases. But the influence of joint sets, especially their
directions, on the fractured zone is uncertain as demonstrated in the UDEC modelling results.
Accordingly, the mid-point value between caving and subsidence angles was determined to
be an appropriate approximation of fracture initiation angle because fracture initiation angle
is usually between caving and subsidence angles. The fracture initiation angles measured
such in the FLAC3D models turned out to be similar to the fracture initiation angles
identified in the other modelling results supporting the decision to take a value between the
caving and subsidence angles. In the Fig. 3.16-19, caving angles and fracture initiation
angles show similar trends. In the UDEC models, the fracture initiation angles were defined
based on the point in the joints where normal stiffness (Kn) or shear stiffness (Ks) equal zero.
In the ELFEN models, the fracture initiation angles were directly measured. For the
subsidence angles, a 1-m cut-off was applied.

3.4.1 Trends with Depth

For the Phase2 results (Fig.3.16), the general trend shows the caving, fracture
initiation and subsidence angles decreasing as a function of depth (indicating a widening of
each related zone with increasing undercut depth). However, the shape of these curves is
convex with the rate of decrease in angle between each depth increment diminishing with
depth (i.e., as the panel width-to-undercut depth ratio (P/D) decreases). In part this is due to
the continuum assumption and interconnectivity of the elements across which the
incremental cave growth is simulated, combined with the increasing overburden across which
the caving-induced strains can be redistributed as the undercut depth increases (or the P/D
ratio decreases). The higher confinement provided by the higher stresses at depth (and
increase in rock mass strength accounted for in the models) would also contribute towards
limiting the extent of subsidence. The results also indicate that the presence of the boundary
faults have little to no significance. The differential between the caving angles and the
subsidence angles is 20-25 degrees for the undercut depth of 500m, and 45-50 degrees for the undercut depth of 2000m.

Results from the 3-D continuum analysis carried out with FLAC3D (Fig.3.17), show similar trends to those from the 2-D continuum analysis. The shape of the curves for the caving, fracture initiation and subsidence trends with depth are likewise convex, but not as strongly so. Here the added dimension and degrees of freedom enable caving-induced strains to be further redistributed resulting in narrower zones of influence on surface. The differential between the caving and subsidence angles is 25 degrees for the undercut depth of 500m and 37 degrees for the undercut depth of 2000m. The 3-D nature of the code, the absence of joint sets, and the larger mesh size appear to contribute to these smaller differences in angles. It should be noted again that the fracture initiation angles in the FLAC3D results were estimated based on the caving and subsidence angles.

Transitioning to a discontinuum approach where a system of joints are free to open, close and slip in response to the caving-induced strains, the trends derived from UDEC (Fig.3.18) are less consistent (i.e. more irregular) but also generally show a convex shape. In the case of the 0 and 90 degree joint sets, the caving and fracture initiation angles do not vary significantly regardless of undercut depth. The presence of the meso-scale joint network together with the bounding faults appears to have some influence on the different angles, with angles at 500 and 1000 m having similar values. This influence then diminishes as the undercut depth further increases, and the curves revert to following a convex trend. It is not possible though to separate the influence of the faults from that of the joint network for the 0 and 90 degree joint set model. For the orthogonal case with added 45 and 135 degree joint sets (Fig.3.18), the trend of the subsidence curve is noticeably more convex, with the subsidence angles sharply decreasing from 500 to 1000m but then less so below this. The caving and fracture initiation angles increase before reversing, implying constraint (against widening) by the bounding faults. The differential between caving and fracture initiation angles is 10-20 degrees for the undercut depth of 500m and 35-57 degrees for the undercut depth of 2000m. This greater extent of differences appears influenced by the joint sets and discontinuum formulation, which allows large strains to develop along the fault and joint
contacts, whereas the continuum formulation is only limited to small strain displacements along the joint elements used.

A completely different trend was observed for the ELFEN results (Fig.3.19), which showed more concave-shaped curves. In general, the caving and fracture initiation angles show an increasing trend with undercut depth, indicating that the influence of caving and its extent at surface diminishes as the undercut depth (and extraction ratio) decreases. This trend is consistent with that observed in the UBC block caving database (Chapter 2) and that reported by Van As et al. (2003). However the opposite is observed with respect to the trend of the subsidence angles. As the undercut depth increases, the subsidence angles are seen to decrease more indicating a significantly wider zone of impact on surface. This difference is likely due to the difference in how caving is simulated, as ELFEN’s unique mesh fracture, block delete (of mined caved material) and block contact update capabilities enable a more direct simulation of the block cave mining process generating more realistic results. Even so, it should be noted that the subsidence angles estimated for the undercut depth of 2000m is similar to those obtained from the other modelling results.

### 3.4.2 Caving Angle

Caving angles approximated by the different numerical techniques applied were compared. The zone of caving is observed directly above the undercut area in each case. The saturated nature of the vertical contours when applying an upper bound plotting interval makes the estimation of the caving angle relatively easy. In Phase2, these contour plots were supplemented with plots of the plasticity indicators to help define the zone of caving, with added weighting given to the distribution of shear indicators (Fig.3.20). In the FLAC3D models too, the caving angles were measured based on the yield zone with shear indicator (Fig.3.21).

A similar technique was used to define the caving zone in the UDEC models (Fig.3.22); the caving angle was estimated based on yield pattern and vertical displacement.
contours. In the ELFEN models, the caving angles were measured directly based on the zone of completely detached blocks that develop and form the cave.

**Fig.3.23** and **3.24** plot the caving angles measured on the north side and the south side of undercut, respectively, for each modelling technique. The different results show similar trends with most indicating overhanging angles relative to the footprint of the undercut. The results though can be ambiguous about the influence of undercut depth on caving with the exception of the ELFEN results which show the caving angle increasing (meaning a smaller zone of disturbance on surface) with increasing undercut depth. Asymmetry is seen in the results between the north and south sides of the undercut. This is due to the initiation of caving in the simulations beginning on the south side (Panel A), meaning a more advanced state of cave development, before progressing to Panel B and then Panel C.

The comparison of caving angle between the Phase2, FLAC3D results and UDEC (with vertical joint sets) reveal that the enabling of large strain slip along the joints in UDEC result in a near vertical caving angle, as would be expected given their persistence. Because the orthogonal joints somewhat restrict the lateral extent of ground movements (concentrating them in the vertical direction instead), the caving angles measured in UDEC signify a broader caving zone on surface relative to that identified in the Phase2 and FLAC3D results (which show overhanging angles for the caving zone). A further consequence of the persistent orthogonal joints is that the caving angle does not vary significantly with undercut depth. This changes when the inclined joint sets (45 and 135 degree) are added resulting in lower caving angles (broader zone on surface). This observation implies that the direction of the joint sets (i.e. rock mass fabric) is a key consideration in terms of subsidence pattern as shown by Vyazmensky et al (2010a).

Through this benchmarking comparison, it is suggested that ELFEN appears to provide more realistic results, and most correct in producing caving angles that consistently increase as a function of undercut depth; i.e. the extent of the caving zone on surface decreases with decreasing extraction ratios.
3.4.3 Fracture Initiation Angle

In Phase2, the fracture initiation angle was estimated based on the distribution of tensile yield indicators on surface observed in the plasticity plots (Fig.3.20). An exception is the 2000 m deep undercut in the Phase2 modelling where no tension yield indicators occur. This perhaps indicates that fracturing outside the caving zone does not occur due to the smaller strains resulting from the smaller extraction ratio.

In UDEC (Fig.3.25), the fracture initiation angle was defined by joints on surface which show opening or slip (normal stiffness (Kn) or shear stiffness (Ks) becomes zero). In ELFEN, fracture initiation was again estimated directly based on the development of fractures at surface in the models. Comparing the different results, specifically the fracture initiation angles to the north and south of the undercut (Fig.3.26 and 3.27), some sensitivity is again seen due the interaction between the faults and panel caving sequence. Fracture initiation angles measured in the UDEC are approximately 10° lower than the caving angles from the same model (the zone of fracture is broader than the zone of caving). Again, the UDEC models with orthogonal jointing show angles that are near vertical due to the significant influence their high persistence has on slip concentrations and therefore where surface fracturing develops. In the UDEC modelling with the inclined joint sets, meanwhile, the fracture initiation angle shows a pattern of decreasing angles with undercut depth as fracturing/slip extends to the bounding faults, before reversed for the 2000 m deep undercut showing that the reduced extraction ratio results in a diminished extent of fracturing on surface. Based on the modelling results, it is observed that the fracture initiation zone does not extend beyond the bounding faults.

In the ELFEN modelling, the fracture initiation angle increases with increasing undercut depth, similar to the caving angle trend. This again confirms that the influence of the bounding faults is to both promote and limit the extent of fracture initiation for the shallower undercuts (maximum cave-fault interaction). For the deeper undercuts, and smaller extraction ratios and caving-induced strains at surface, the influence of the faults significantly decreases. Interestingly, the fracture initiation angle measured in the Phase2 model decreases as undercut depth increases, which is different from the other models. This
suggests that the influence of the faults in Phase2 is less significant, likely due to the small-strain limitations of the joint element formulation.

In the FLAC3D modelling, the fracture initiation angle was approximately 10-15° smaller than the caving angle and tends to increase as the undercut depth increases.

### 3.4.4 Subsidence Angle

The angle of subsidence was estimated based on the vertical displacement contours defined by a 1 m lower bound cut off (Fig.3.28 and 3.29). In the UDEC model with inclined joints, the angle of subsidence although greatest sharply drops off for the deeper undercuts. The implication of this observation is that the angle of the joint sets dominates the development of the subsidence zone regardless of undercut depth, with the lowest angle attained coinciding with the dip of the joints. In the ELFEN model, the angle of subsidence decreases with increasing undercut depth indicating that unlike the caving and fracture initiation angles, the bounding faults do not significantly influence or limit the lateral extent of smaller-strain subsidence. The subsidence angles measured in the ELFEN models further imply a non-linear trend with the extent of subsidence markedly increasing as undercut depths decrease. This perhaps suggests that the bounding faults do have some influence (minor) on limiting small-strain subsidence for the shallower undercut depths, but not so for the deeper undercuts. A noteworthy observation from the comparison of subsidence patterns identified by UDEC (with inclined joint sets) and ELFEN is that the subsidence angles measured in the undercut depths of 500 and 2000 m do not differ but the difference in the subsidence angle measurements in the 1000 and 1500 m undercut depths are considerable. Movement of individual blocks on the inclined joints in response to caving appears to be more active with the shallower undercuts, having a more pronounced influence on the surface subsidence.

In the case of the ELFEN results, the influence of the brittle fracture network on the lateral extent of subsidence tends to be less significant for the shallower undercuts, but that with greater undercut depths (and higher stresses), brittle fracture activity away from the immediate area above the undercut increases. The trend of subsidence angles with increasing undercut depth between ELFEN and the UDEC model with vertical joints is similar. This
suggests that the influence of large strain slip along joints in the UDEC models decreases with increasing confining stresses at depth, as would be expected based on a Coulomb slip law.

For the FLAC3D results, where estimates of the caving and fracture initiation angles are limited by the smaller vertical displacements that develop, the subsidence angles are seen to decrease from 500 to 1000m, but then remains unchanged for the deeper undercuts. This indicates that the subsidence area increases in a consistent ratio in relation to the undercut depth increase. The subsidence angle could be consistent probably because the caving height is constant.

In summary, each set of models: Phase2, UDEC, ELFEN and FLAC3D, each show a similar trend with the extent of subsidence increasing (decreasing subsidence angles) with increasing undercut depth.

3.5 Conclusions

Results are presented from a benchmark study examining the relationship between the extent of caving, large-strain fracture initiation and small-strain subsidence as a function of undercut depth using four different numerical techniques: 2-D finite-element (Phase2), 3-D finite difference (FLAC3D), 2-D distinct element (UDEC) and 2-D FEM-DEM with brittle fracture (ELFEN). All models showed a similar response with the extent of caving decreasing with increasing undercut depth (resulting in caving angles greater than 90° indicating an overhanging condition with respect to the undercut footprint). A similar decrease in the extent of macro-deformation in the form of the fracture initiation zone, was also seen to decrease with undercut depth. These responses can be explained by decreasing extraction ratios with increasing undercut depth as the simulation of caving in the models maintained the same block height of 200 m (i.e., a constant block height relative to an increasing rock column height above the undercut with increasing undercut depth corresponds to a decreasing extraction ratio). In most cases, the presence of bounding faults on either side of the undercut limited the lateral extent of caving and fracture initiation.
Differences in the results between the different numerical methods applied emphasize the importance of carefully defining the key objectives of the modelling together with the factors that are most important, while exercising good modelling practice when selecting a numerical modelling tool. Often the needs for fast model setup and run times are counter to those for accurate representations of the rock mass fabric (through the inclusion of DFN’s) and caving mechanics. Overall, the 2-D hybrid FEM-DEM approach allowing stress-induced brittle fracturing between joints appeared to provide the more dependable and realistic results. In addition, the contact detection algorithm used in ELFEN allows caving to be more intuitively (explicitly) simulated through a block deletion procedure which appears to work better than changing the material properties to those of caved rock as the cave advances. Ideally, using a 3-D FEM-DEM brittle fracture code would seem to incorporate all of the key requirements and needs. Computationally, though, this is prohibitively expensive with current computer and software capabilities. Where the need for a 3-D analysis is the overriding factor, a 3-D finite-element or finite-difference code (e.g. FLAC3D) represents the most viable option at present.
Table 3.1. Rock mass properties used for conceptual benchmark model testing, as defined by the Centre for Excellence in Mining Innovation (CEMI)

<table>
<thead>
<tr>
<th>Lithology</th>
<th>UCS [MPa]</th>
<th>mi [GPa]</th>
<th>E [GPa]</th>
<th>RMR89</th>
<th>GSI</th>
<th>Block length (m)</th>
<th>Joint conditions Jc</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rhyolite</td>
<td>205</td>
<td>25</td>
<td>60</td>
<td>49</td>
<td>44</td>
<td>0.35 m</td>
<td>0.7</td>
</tr>
<tr>
<td>Quartz – Monzodiorite</td>
<td>140</td>
<td>20</td>
<td>50</td>
<td>43</td>
<td>38</td>
<td>0.35 m</td>
<td>0.5</td>
</tr>
<tr>
<td>Sandstone and Siltstone</td>
<td>125</td>
<td>18</td>
<td>40</td>
<td>52</td>
<td>47</td>
<td>0.5 m</td>
<td>0.7</td>
</tr>
<tr>
<td>Biotite Granodiorite</td>
<td>145</td>
<td>22</td>
<td>55</td>
<td>43</td>
<td>38</td>
<td>0.35 m</td>
<td>0.5</td>
</tr>
</tbody>
</table>

Note: UCS – Unconfined Compressive Strength, mi – the intact rock parameter, E – Young’s Modulus, RMR89 – Rock Mass Rating at 1989, and GSI - Geological Structure Index; Poisson’s ratio used is 0.25.

Table 3.2. Depth-dependent Mohr-Coulomb rock mass properties derived for the 500 to 2000 m deep undercut models.

<table>
<thead>
<tr>
<th>Lithology</th>
<th>σ_{3\text{max}} (MPa)</th>
<th>E_{rm} (GPa)</th>
<th>c_{rm} (MPa)</th>
<th>\varphi_{rm} (°)</th>
<th>T_{rm} (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rhyolite</td>
<td>7.5-17.5</td>
<td>12.6</td>
<td>3.3-5.9</td>
<td>46-52</td>
<td>0.6-1.2</td>
</tr>
<tr>
<td>Quartz-Monzodiorite</td>
<td>30</td>
<td>7.0</td>
<td>6.5</td>
<td>34</td>
<td>1.8</td>
</tr>
<tr>
<td>south of South fault</td>
<td>5-20</td>
<td>7.0</td>
<td>1.9-4.9</td>
<td>38-49</td>
<td>0.4-1.2</td>
</tr>
<tr>
<td>Sandstone and Siltstone</td>
<td>45-50</td>
<td>10.2</td>
<td>9.1-9.7</td>
<td>31-32</td>
<td>2.5-2.8</td>
</tr>
<tr>
<td>south of South fault</td>
<td>30-35</td>
<td>10.2</td>
<td>6.9-7.7</td>
<td>34-35</td>
<td>1.8-2.1</td>
</tr>
<tr>
<td>Biotite Granodiorite</td>
<td>10-40</td>
<td>7.7</td>
<td>3.2-8.2</td>
<td>33-44</td>
<td>0.7-2.3</td>
</tr>
<tr>
<td>north of North fault</td>
<td>40</td>
<td>7.7</td>
<td>8.2</td>
<td>33</td>
<td>2.3</td>
</tr>
<tr>
<td>south of South fault</td>
<td>-</td>
<td>0.26</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

Caved Rock

Note: - indicates data not applicable or not available.
Fig. 3.1. Generalized E-W cross section through the Resolution Deposit (Rio Tinto, 2008), on which the conceptual geometry for this benchmarking study, as defined by the Centre for Excellence in Mining Innovation (CEMI), was loosely based.
Fig. 3.2. Conceptual panel cave mining geometry for benchmark testing specified by the Centre for Excellence in Mining Innovation (CEMI): (a) plan view, and (b) A-B section. Panel dimensions are 500 x 1000m.
Fig. 3.3. Phase2 model geometry for the 2000 m deep undercut simulation, showing the orthogonal orientation, and variable spacing (20-50m) and persistence (40-170m) of joint elements introduced above the undercut. For scale, the model is 10,500 m in length and 4200 m in depth.
Fig. 3.4. FLAC3D model geometry: (a) in plan view, with semi-transparency to show projection of the bounding faults through the model (dark blue), and (b) north and south section showing the fault interfaces in dark blue.
Fig. 3.5. (a) UDEC model geometry for the 2000 m deep undercut simulation, showing, (b) orthogonal joint pattern and (c) an orthogonal joint pattern superimposed with a second pattern at 45 and 135 degrees. Note that the lower close-up views are 250 by 250 m.
Fig. 3.6. ELFEN model geometry for the 2000 m deep undercut simulation, showing the extended discrete fracture network used. Note that this is the same DFN as used for the Phase2 models. (Eberhardt et al., 2011)
Fig. 3.7. In situ stress information for benchmark testing as specified by CEMI: $S_{H_{\text{max}}}$ is north-south section and $S_{H_{\text{min}}}$ is east-west section (Eberhardt et al., 2011).
Fig. 3.8. Method used to simulate draw for Phase2D, FLAC3D and UDEC model.

Fig. 3.9. Method used to simulate draw for ELFEN model, involving a block deletion algorithm.
**Fig. 3.10.** Phase2 continuum subsidence results for undercut depths of 500 to 2000 m. Vertical displacement contours are plotted with a 1 m minimum cut off.
Fig. 3.11. Phase2 continuum subsidence results with the inclusion of joint elements, for undercut depths of 500 to 2000 m. Vertical displacement contours are plotted with 1 m minimum cut off.
Fig. 3.12. FLAC3D subsidence results from 500 to 2000m after 200,000m³ (0.25m cut off): Surface section (top), north-south Section (bottom).
Fig. 3.13. UDEC subsidence results from 500 to 2000m (1m cut off) - with orthogonal joint sets.
Fig. 3.14. UDEC subsidence results from 500 to 2000m (1m cut off) - with 45 and 135 degrees joint sets.
Fig. 3.15. ELFEN subsidence results from 500 to 2000m after 200,000m$^3$ (1m cut off).
Fig. 3.16. Estimated angles plotted as a function of undercut depth at north (top) and south (bottom) fault side of undercut using the 2-D continuum finite-element code Phase2. Shown are the different trends for the caving, fracture initiation and subsidence angles with and without the inclusion of joints modelled using small-strain joint elements.
Fig. 3.17. Estimated angles plotted as a function of undercut depth at north and south fault side of undercut using the 3-D continuum finite-difference code FLAC3D. Shown are the different trends for the caving, fracture initiation and subsidence angles.
Fig. 3.18. Estimated angles plotted as a function of undercut depth at north (top) and south (bottom) fault side of undercut using the 2-D discontinuum distinct-element code UDEC. Shown are the different trends for the caving, fracture initiation and subsidence angles modelled assuming two different joint patterns: orthogonal jointing at 0 and 90 degrees dip and 45 and 135 degrees dip. Joints are modelled explicitly in UDEC allowing for large strain slip, opening and closing along each joint interface.
Fig. 3.19. Estimated angles plotted as a function of undercut depth at north and south fault side of undercut using the 2-D FEM/DEM brittle fracture code ELFEN. Shown are the different trends for the caving, fracture initiation and subsidence angles, modelled using the “mesh delete” and contact updating capabilities of ELFEN to more directly simulate the block cave mining process.
Fig. 3.20. Phase2 continuum subsidence results with the inclusion of joint elements, for undercut depths of 500 to 2000 m. Vertical displacement contours are plotted with 1 m minimum cut off. The inside lines define the caving angle measured based on shear point and the outside lines define the fracture angle measured based on the tension point.
Fig. 3.21. FLAC3D plasticity indicator, red line-caving angle threshold.
Fig. 3.22. UDEC plasticity indicator: orthogonal joint pattern, red line-caving angle threshold.
Fig. 3.23. Estimated caving angle plotted as a function of undercut depth at north fault side of undercut.

Fig. 3.24. Estimated caving angle plotted as a function of undercut depth at south fault side of undercut.
Fig. 3.25. Open joint (UDEC): (a) orthogonal joint pattern and (b) an orthogonal joint pattern superimposed with a second pattern at 45 and 135 degrees, red line-fracture initiation angle threshold.
Fig. 3.26. Estimated fracture initiation angle plotted as a function of undercut depth at north fault side of undercut.

Fig. 3.27. Estimated fracture initiation angle plotted as a function of undercut depth at south fault side of undercut.
Fig. 3.28. Estimated subsidence angle plotted as a function of undercut depth at north fault side of undercut.

Fig. 3.29. Estimated subsidence angle plotted as a function of undercut depth at south fault side of undercut.
Chapter 4: Integration of Field Characterization, Mine Production and InSAR Monitoring Data to Constrain and Calibrate 3-D Numerical Modelling of Block Caving-Induced Subsidence

4.1 Introduction

The use of block caving to mine deep, massive, low grade orebodies is often favoured by the mining industry given its merits in terms of safety, tonnages produced and costs that can match those of open pit operations. As an underground mass mining method, however, significant ground surface deformations often develop. If these are not properly assessed and accounted for, they may threaten the integrity and safety of overlying mine infrastructure. To better manage such risks, detailed engineering studies are undertaken to characterize the ground conditions and provide input for empirical and numerical design calculations.

Empirical relationships are generally used for preliminary scoping calculations, for example to estimate caving angles based on rock mass quality (Laubscher, 2000). These do not explicitly account for the influence of stress-strain interactions and geological heterogeneity that may significantly affect the ground deformation profile. Instead, investigators have increasingly turned to advanced 2-D and 3-D numerical modelling to improve the assessment and understanding of block-caving subsidence dynamics and surface-underground interactions (Karzulovic et al., 1999; Gilbride et al., 2005; Brummer et al., 2006; Beck et al., 2008; Vyazmensky, 2008; and Vyazmensky et al., 2010b).

As in any modelling study, the results depend on the assumed initial conditions and material properties assumed. These can vary greatly in accordance with the geological heterogeneity and variability encountered on site, with ranges of input properties being more likely than a single value. Furthermore, the numerous surface and underground interactions

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3 Woo, KS. Eberhardt, E. Rabus, B. Stead, D. Vyazmensky. A. Integration of field characterization, mine production and InSAR monitoring data to constrain and calibrate 3-D numerical modelling of block caving-induced subsidence. (In Review)
involved – both spatial and temporal – result in a complex 4-D problem that challenges even the most sophisticated numerical models.

The resulting model and parameter uncertainty necessitates that the models be constrained and calibrated in order to gain confidence in their output. This paper examines these issues based on detailed back and forward analyses of the Palabora block caving operation in South Africa. The first part of this paper focuses on the back analysis of a large 800 m high pit slope failure that occurred in response to block caving activities below the pit. This was used to constrain a 3-D finite-difference model against the range of rock mass strength values and in-situ stress ratios derived from field investigation data. The second part of this paper reports the use of these “best fit” input properties to forward model the caving-induced subsidence at Palabora for the period 2009-2010. High resolution satellite-based Interferometric Synthetic Aperture Radar (InSAR) data was used to further calibrate the 3-D numerical model developed. Together, the results presented demonstrate the potential of InSAR as a means to calibrate sophisticated 3-D subsidence models, improving our ability to assist mining companies to execute safe, economic and sustainable mining practices.

4.2 Palabora Case History

The Palabora copper mine is a large, 30,000 tonne/day block cave operation located in the eastern half of Limpopo, South Africa’s northern most province. Underground mining was commenced in April 2001 after transitioning from an earlier open pit operation with target production being achieved in May 2005. The dimensions of the pit are approximately 800 m deep and 1650 to 1900 m in diameter, with slope angles ranging from 37° in the upper half of the pit to 58° in the lower, more competent lithologies. The block cave undercut level is approximately 1200 m below surface, 400 m below the pit floor (Fig.4.1). The production level below the undercut consists of 20 cross-cuts spaced across a footprint 650 m long by approximately 250 m wide (Moss et al., 2006).

Three years after the initiation of caving, cracking was observed in the northwest wall of the pit. This evolved over several months into major movements and eventually failure of the 800 m high pit wall shortly after breakthrough of the cave into the bottom of the pit (Fig.4.2). The failure extended 300 m beyond the outer perimeter of the pit, affecting access
and haul roads, tailings, water and power lines, water reservoirs and a railway line (Moss et al., 2006). Fortunately, other critical mine infrastructure were not affected.

Moss et al. (2006) remarked that the failure at Palabora revealed deficiencies in our understanding of cave-pit interactions. Subsequently, a number of studies were carried out applying sophisticated numerical modelling to back analyze the Palabora failure (Brummer et al., 2006; Sainsbury et al., 2008; Vyazmensky, 2008; and Vyazmensky et al., 2010b). Brummer et al. (2006) first examined the failure mechanism using the 3-D distinct element code 3DEC to model the influence and kinematic control of major geological structures. Their findings show that the observed movements were potentially caused by wedges formed by pervasive joints that daylight into the cave region below the pit.

Another key study was that by Vyazmensky (2008), who applied a hybrid finite-element/discrete-element modelling approach that simulates brittle fracturing within a network of non-persistent discontinuities. His work highlights the importance of joint set orientation on influencing the direction of cave propagation, as well as the importance of rock bridges and their incremental failure through cave-pit interactions leading to the progressive failure of the Palabora pit slope (Vyazmensky et al., 2010b).

Although the preceding back analyses have made significant contributions to our understanding of the pit slope failure mechanism associated with block caving at Palabora, Moss et al. (2006) also stress that given the level of up-front investment in a block cave, it is extremely important to develop reliable predictive tools. This emphasizes the need for reliable, well constrained and calibrated forward analyses of block caving induced subsidence.

4.3 Model Constraint and Calibration through Back Analysis

4.3.1 Palabora 3-D Model Geometry

A detailed 3-D geological model was constructed integrating digital mine plans (Fig.4.1) with mine geology data (Fig.4.3). The Palabora Igneous Complex (Palabora Mining Company Limited, 1976) consists of a succession of sub-vertical pipe-like bodies of alkaline and ultramafic rocks that have intruded into a host Archean granite. The copper ore body
occurs near the centre of the complex and is a composite vertical intrusion of micaceous pyroxenite subsequently intruded by foskorite and carbonatite. This body has then been cut by the emplacement of steeply dipping dolerite dykes.

The detailed geological model was used to develop a 3-D numerical model of the pit and undercut (Fig. 4.4) using the commercial finite difference code FLAC3D (Itasca Consulting Group, 2009). Implementation of the lithological units was limited in part by the resolution of the mesh; 15x15x15 m elements were used to model the primary area of interest meaning that lithological domains with widths less than 15 m were not resolved. This includes the dolerite dykes. The external boundaries of the model measure 4000 x 4000 m in plan and 2000 m in depth. A gradational mesh with larger elements extending towards the outer model boundaries was employed. The 3-D mesh was iteratively tested and modified to limit numerical errors resulting from boundary effects and element shape and size.

4.3.2 Modelling of Caving Influence

The representation of the block cave and its development in the model is of prime importance given its direct relationship with the caving-induced deformations being modelled. This was implemented through an approach where the geometry of the cave is built into the model, as opposed to explicitly modelling caving and cave propagation. The latter would have required considerable effort within the continuum framework to model the caving process accurately, alternating between small strain calculations applying a seismogenic/yield zone approach to model cave propagation and large strain calculations to model the corresponding ground deformations. The former approach involves assuming the cave geometry at several different points in its development over time, and incrementally changing its properties from those of the ore body rocks to fragmented rock. This requires special consideration of the material properties of caved rock, and the redefinition of the induced stress state within the modelled cave material.

Adopting the implicit cave geometry approach, a procedure was developed for determining the cave geometry at any given point in time (Fig. 4.5). This involved extracting, for each time interval, the tonnages for each draw point from the mine production data, applying a 20% swell factor, and calculating the cave height above each draw point. The
resulting 3-D cave shape was modified according to the mine microseismic data for the same period.

4.3.3 Rock Mass Properties

Rock mass properties were derived for each of the key lithologies represented in the FLAC3D models (Fig. 4.3). This involved compiling data from mine geotechnical reports in which laboratory testing and rock mass characterization data were reported. Because these reports span several different testing and field characterization campaigns (over a period of more than 25 years), all data was carefully reviewed and evaluated to establish lower and upper bound values which are the minimum and maximum values reported, respectively (Table 4.1). This work was supplemented by field-based assessments made by the UBC-SFU team during fieldwork carried out at the mine in 2008. The major lithological units represented in the model, and their rock mass characteristics, are summarized in Appendix E. The rock mass descriptions and rock strengths provided are compiled from several modelling, laboratory testing, and field measurement sources. Not included in the model are the dolerite dykes, described as being composed of very strong rock, the jointing and several large fault zones.

From these, rock mass shear strength properties were estimated for use with a Mohr-Coulomb strain softening constitutive model. Most practitioners have more experience and therefore an intuitive feeling for the physical meanings of cohesion and friction on which the Mohr-Coulomb criterion is based. Accordingly Mohr-Coulomb rock mass shear strength properties for the use in the FLAC3D numerical modelling were derived through empirical procedures based on GSI, RMR and Q (e.g. Hoek et al., 2002). Several empirical procedures exist to derive Mohr-Coulomb rock mass shear strength properties, one of the more commonly used being Hoek et al.’s (2002) conversion of Hoek-Brown to Mohr-Coulomb achieved by fitting an average linear relationship to the non-linear Hoek-Brown envelope for a range of minor principal stress values with an upper bound of $\sigma_{3\text{max}}'$. Analytical relationships are provided for estimating $\sigma_{3\text{max}}'$, however, these do not apply to block caving. Consequently, Hoek (2007) recommends that caving analyses be carried out based either on
Hoek-Brown or Mohr-Coulomb parameters, assessed independently, but not on the conversion of one to the other.

The applicability of these scaling relationships is noted here, but is also weighed against their required use: to simply provide an initial estimate of the rock mass shear strength properties, the values for which will then be varied and refined through the back analysis calibration exercise. Accordingly, the Hoek-Brown to Mohr-Coulomb conversion procedure was deemed adequate and a $\sigma'_{3\text{max}}$ value of 10 MPa was estimated for the conversion based on preliminary modelling of the stresses that develop between the cave and pit walls. Table 4.1 reports the corresponding ranges of equivalent rock mass cohesion and friction angle values for the lower and upper bound values established for each rock mass unit represented in the FLAC3D model. These are in general agreement with values used in previous studies by the mine’s consultants (see Table 4.3). The lower- and upper-bound values were next tested through a parametric analysis using the FLAC3D model to test the sensitivity of the modelled response to the material properties used.

4.3.4 In-situ Stresses

Data for the in-situ stress boundary conditions were compiled based on regional stresses reported in the published literature and mine specific in-situ stress studies. Regional stress data reported in the World Stress Map database (Heidbach et al, 2008) and those in a study of South African mining areas (Stacey et al., 1998) suggest that stress ratios in the region fall somewhere between 0.5 and 1.5 for the major horizontal to vertical stress ratio, and 0.5 to 1.0 for the minor horizontal to vertical stress ratio. These ranges bracket values obtained from in-situ stress measurements at Palabora (SRK Consulting Group, 1992a; SRK Consulting Group, 1992b; and Gash, 1999), where investigations in 1992 and 1999 reported ratios ranging from 0.5 to 1.2 (Table 4.2). The variability seen in Table 4.2 is not uncommon for in-situ stress measurements, and may be due to local effects caused by major faults and/or geological heterogeneity.

A number of reports presenting results from numerical modelling for Palabora were further reviewed to see how the in-situ stress boundary conditions were treated for different aspects of the pit and cave designs (Table 4.3). Early studies assumed an in-situ stress ratio
$K_o$ of 1.0 (Martin et al., 1991), or assumed $K_o$ to be 1.5 to 2.0 in the upper 200 m of the pit, decreasing linearly to about 1.1 at a depth of 2000 m (SRK Consulting Group, 1991). More recently, numerical analyses have used a $K_o$ varying between 1.5 and 2.0 based on findings from the Mass Mining Technology (MMT) project of the International Caving Study.

Based on this review, several different in-situ stress assumptions were tested in step with the model calibration for the best fit set of rock mass properties.

4.3.5 Back Analysis and Model Calibration

FLAC3D modelling of the 2005 northwest wall failure (Fig.4.2) was carried out to back analyze and constrain the material properties and in-situ stresses to be used for subsequent forward modelling. Cave-pit interactions were modelled starting from the time of initial underground production in 2002 to the time of failure in 2005 in one year increments (Fig.4.5). Full implementation of the caving simulation involved first initializing the stresses in the ore zone (i.e. calculated according to the depth of the host rock and acting horizontal in-situ stresses), and then modelling the advancement of the cave by changing the material properties of the ore to those of the caved rock in step with the upward propagation of the cave. A key step in this process is that the initialized stresses in the elements representing the caved material must be reset with each modelled advance of the cave to correspond with the self-weight of the caved rock and not the initial tectonic stresses.

Average rock mass properties and in-situ stress ratios from the ranges compiled in Tables 4.1 and 4.2, respectively, were tested and then varied depending on the closeness of the fit achieved between the modelled displacements and outline of the northwest wall failure. Several limitations in the modelling approach applied here must be noted. First, the presence of both meso-scale jointing in the northwest wall and major faults in its proximity would have played a significant role in the caving-induced slope failure process. These are not considered in the FLAC3D continuum representation of the slope. The results are also limited by the minimum element size, which influences the ability for a failure surface in the model to localize and develop. Due to these limitations, it is not possible to explicitly model the pit wall failure that occurred. Instead, the comparative analysis carried out relied on the
distribution of caving-induced displacements, specifically those arising from strain softening, as the measure to compare the different back analysis model runs.

Results from the back analysis clearly showed a varied response for the different rock mass properties tested. Fig. 4.6 compares the caving-induced displacements modelled assuming a set of average rock mass properties (Fig. 4.6a) and those assuming the lower bound properties (Fig. 4.6b). The latter shows increased displacements in the northwest wall of the pit that approximately coincide with the northwest wall failure. Table 4.4 reports the calibrated rock mass properties judged visually as providing the best fit for the back analyzed northwest wall failure. These include testing of different strain softening thresholds at which strength degradation through brittle fracturing would begin (correlating induced plastic strains with reduced post-peak rock mass strengths). Results using the Mohr-Coulomb strain softening constitutive model produced significantly improved results over models solved assuming a simpler Mohr-Coulomb elasto-plastic constitutive model. Model calibration suggested that the influence of strain softening was most important for the pyroxenite and glimmerite units.

Also included in Table 4.4 are the material properties assumed for the caved rock. As the material responsible for the relaxation and deformation of the surrounding rock, it was found to have a significant influence on the magnitude of the modelled displacements. A detailed search of the literature proved unsuccessful in finding values for caved rock; several papers were found reporting values for broken rock as used in the construction of rockfill dams and for mine backfill and these were used as an initial starting point for the back analysis. However, these required additional calibration. To avoid numerical errors related to severe mesh distortion, the caved rock material was modelled as an elastic material, using reduced elastic properties to account for the reduced deformation modulus that would be expected for caved rock, as well as allowances for the presence of a small air gap.

Model calibration of the best-fit material properties was carried out in step with calibration of the assumed far-field in-situ stress boundary condition (Fig. 4.7). These showed that a uniform stress field where the NS and EW horizontal stresses equal the vertical stresses (i.e. $K_o=1$) provided the best fit to the outline of the northwest wall failure (Fig. 4.8).
stress field is in agreement with those in the World Stress Map database (Heidbach et al., 2008), those reported by Stacey and Wesseloo (1998) and values reported in Tables 4.2 and 4.3. The contours in Fig. 4.8 adopt a vertical displacement cut-off of 3 cm, which is the National Coal Board’s minimum threshold (National Coal Board, 1975) for damage to surface infrastructure and an approximate indicator for the appearance of surface fractures (i.e. brittle fracturing of the rock mass). Given the continuum treatment of the problem, the closeness of the fit achieved suggests that the spatial positioning of the undercut beneath the pit was an important factor influencing the northwest wall failure. The 3-D model results clearly show that the interaction between the developing cave and open pit above was more pronounced for the northwest wall than any other area of the pit. These directed the cave towards the north where it undermined the toe of the slope eventually resulting in failure.

4.4 InSAR Monitoring as a Means to Constrain Modelling Displacements

Space-borne Synthetic Aperture Radar (SAR) involves the use of satellite-based microwave radar to remotely observe characteristics of ground terrain. With repeated orbits and image capture (referred to as stacks), Interferometric SAR (InSAR) data can be processed to resolve 3-D information of surface deformations by analyzing differences in the phase between waves being transmitted and received by the satellite (Zebker et al., 1994). Ground deformations can be detected on the scale of centimetres to millimetres for a surface area resolution of several square metres using these techniques. This ability to detect shape changes in a surface area with significant resolution provides a means to monitor mining-induced differential strains, including small strain (<1 %), that develop across an irregular surface topography.

Inspection of the results in Fig. 8 shows that the modelled displacement field extends to the east of the northwest wall failure. Although this at first may appear to suggest a minor mismatch between the model and observed extent of failure, subsequent integration with RADARSAT-1 data shows excellent agreement. Fig. 4.9 shows the InSAR measured displacements from the analysis of RADARSAT-1 images (8-m surface area resolution) for Palabora recorded for the two-year period following the 2005 northwest wall failure. The data shows that most of the measured displacements are concentrated along the north wall,
especially towards the east along its crest. These would appear to be related to the instability of the pit wall in response to the cave breakthrough. Behind the pit rim and around the mine area in general, little subsidence is detected during this period. Comparing these displacements to those modelled in Fig. 8, both the areas of extent and magnitude of displacement are in close agreement. This provides an additional degree of confidence in carrying the back analyzed input values forward for subsequent analysis of the current state of caving-induced subsidence.

4.5 Forward-Modelling of Caving-Induced Displacements (2009-2010)

4.5.1 Modifications to Model Geometry

The back analysis of the 2005 pit slope failure provided an important initial step in calibrating and constraining the FLAC3D subsidence model. However, the failure also represents a major change in the 3-D geometry of the mine model, including a localized deviation of the cave. These were accounted for in a modified 3-D model directed towards a forward analysis of caving-induced ground deformations for the period March 2009 to March 2010. Built into the model were the changing cave geometries for 2006 to 2010 (Fig.4.5) together with the presence of the pit wall failure debris (Fig.4.10). Thus, this modified model represents a continuation of the back analysis model incorporating the influence of the northwest wall failure.

The pit wall failure debris was assigned the same properties as the caved material (i.e. broken rock). In addition, the stress conditions in the failed zone were reassigned to represent those of the collapsed ground (i.e. gravity loading instead of the $K_o = 1$ in-situ stress condition used throughout the rest of the model). The presence of the failure debris also required special consideration in the construction and implementation of the post-failure cave geometries. As can be seen in Fig.4.2, a significant amount of slide debris sits above and on top of the pit floor bottom where the cave has broken through. Moss et al. (2006) estimate the failure to be approximately 100 million tonnes and note that the potential exists for the slide/waste material to move at a faster rate than the ore rock within the cave as the cave is pulled due to differences in block size between the two (i.e. mechanical sieving). Thus, the production data used to project the changing volume of the cave after 2005 would likely
include both caved rock (associated with the changing cave geometry) and slide debris from surface entering the cave (not associated with the changing cave geometry). To correct for this, Digital Terrain Models (DTMs) and QuickBird satellite images for different periods between 2005 and 2009 were compared to approximate changes in volume of the rockslide material on surface and therefore that entering the cave. This analysis indicated that it is unlikely that debris was entering the cave prior to January 2006. Subsequent to this, however, an average volume of 2 million m$^3$ per year was resolved as entering the cave and was corrected for in generating the cave geometries. The caving intervals analyzed include: March 2006, March 2008, March 2009, September 2009, and March 2010, where the last three intervals coincide with the beginning, mid-point and end of targeted RADARSAT-2 SAR data captures carried out as part of this study.

4.5.2 RADARSAT-2 Deformation Monitoring

Through a partnership between the Canadian Space Agency, MDA Systems, the University of British Columbia and Simon Fraser University, targeted RADARSAT-2 data was collected to monitor mining-induced ground deformations at several sites, including Palabora. Launched in December 2007, RADARSAT-2 is Canada’s second-generation commercial SAR satellite, capable of providing surface area resolutions approaching 2x2 m. Together with improvements in data processing and inversion of the phase components (atmosphere, height error and displacement corrections), significant gains have been made in the robustness of the InSAR solution (Rabus et al, 2009).

RADARSAT-2 data was collected for Palabora during the period March 2009 to March 2010, in 28 day intervals. The images were taken in ascending as well as descending modes. The processed InSAR deformations were then used to compare with the results from the FLAC3D forward modelling results.

4.5.3 FLAC3D Forward Modelling Results and Comparison with InSAR Data

Figs.4.11 and 4.12 show the FLAC3D forward modelling results for the caving-induced displacements for Palabora between March 2009 and March 2010. These predict total displacements on the order of 5-15 mm along parts of the upper pit and crest, and 15-30
mm just above the bottom of the pit. **Fig.4.12** shows that the cave interaction with surface will have the greatest influence on the west pit wall with minor movement of the north wall. **Fig.4.13** shows the ascending and descending InSAR data for the same period. These show that most of the displacements (20-40mm) correspond to ongoing activity to the east of the 2005 northwest wall failure (primarily near the crest) and to the west extending across half of the west wall. Otherwise, displacements are less than 10mm for this time period along the east and south pit walls.

**Figs.4.14** and **4.15** present the side-by-side comparison of the FLAC3D results and corresponding ascending and descending InSAR images for the 2009-2010 data gathering period. Using the input properties derived from the back analysis, close agreement was achieved with respect to the spatial extent of the displacements. However, the displacement magnitudes predicted by the FLAC3D model are approximately 20% lower than those seen in the InSAR data. A better fit of the measured displacement magnitudes was subsequently achieved through minor calibration of the model in the form of decreasing the plastic strain threshold for strain softening for the pyroxenite from 0.01 to 0.005. Further comparisons of the calibrated FLAC3D subsidence model and the InSAR measured vertical displacements are shown for several targeted areas around the open pit, including the main access shaft (**Fig.4.16**), the ventilation shaft (**Fig.4.17**) and the crest above the west wall (**Fig.4.18**). Also included are the vertical displacements calculated from the geodetic monitoring data, which show very good agreement.

The subsidence predicted by the FLAC3D model for the period of 2009-2010 closely fits the InSAR measurements during the same period time, proving the significant accuracy and reliability of the FLAC3D prediction. As for the geodetic data, the geodetic measurements show more variability in the subsidence rate than the InSAR measurements. Geodetic data are direct measurements taken on site and thus presumed to be reliable. However, there are several sources of potential measurement errors. Firstly, geodetic data are not collected on a daily basis. Secondly, measurement data are collected only in selected points. Thirdly, if multiple agents collect the geodetic data, sampling errors could occur. The variability of the geodetic data possibly resulting from measurement errors can potentially trigger false concern. In fact, the geodetic for the ventilation shaft indicates caving-induced
movements as shown in Fig.4.17 but much of this is scatter. In the meantime, the InSAR measurements (and the FLAC3D prediction) in the ventilation shaft do not oscillate much around the mean value. These observations vindicate the significant value of the InSAR data and thus the InSAR calibrated FLAC3D model regarding subsidence prediction.

4.6 Conclusions

This study presents results from a detailed 3-D back and forward analysis of caving-induced displacements at the Palabora open pit/block cave mine in South Africa. Results from the back analysis of a 2005 pit slope failure were used to constrain uncertainty in the rock mass properties and far-field in-situ stresses, as well as to bring understanding to the problem with respect to the cave-pit interactions. The modelled outcome underscores the sensitivity and dependence of modelling results on the input required and thus the difficulty of using modelling results for predictive purposes. Agreement was found between the FLAC3D modelled zones of ground movement and the location of the 2005 northwest wall pit slope failure, as well as with RADARSAT-1 InSAR data from 2005.

A “best fit” set of input properties were obtained and used for forward modelling of the caving-induced subsidence occurring at Palabora for the period 2009 to 2010. Dedicated high-resolution InSAR data from the recently launched RADARSAT-2 satellite was collected for the same period (March 2009 to March 2010). Again, close agreement was achieved with respect to the spatial extent and magnitudes of the FLAC3D predicted displacements and those measured using InSAR, allowing further calibration of the model. These were further compared with geodetic monitoring data from the mine likewise showing very good agreement.

The close fit achieved between the predictive 3-D numerical model and InSAR data demonstrates the promise of InSAR as a means to calibrate and validate sophisticated numerical models, and thereby contribute to managing block caving associated subsidence hazards. The results demonstrate that satellite-based InSAR provides an effective means to identify and map spatial movements across a large open pit and beyond the pit limits. This ability is important for protecting key mine infrastructure located on surface, especially where subsidence caused by underground mass mining may be interacting with open pit
slopes and surrounding surface area above. Together, an advanced information product has been developed and demonstrated, integrating geology, geotechnical data sets, and 3-D numerical modelling with InSAR imagery to assist mining decision makers in their development of safe and efficient block caving and open pit mining operations.
### Table 4.1
Ranges of intact rock properties and rock mass rating values compiled from mine geotechnical reports and internal field assessments, and corresponding lower and upper bound rock mass properties derived.

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Intact Rock Properties</th>
<th>Rock Mass Characteristics &amp; Properties</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Density (kg/m$^3$)</td>
<td>UCS (MPa)</td>
</tr>
<tr>
<td>Carbonatite</td>
<td>2760-4720</td>
<td>75-172</td>
</tr>
<tr>
<td>Foskorite</td>
<td>2850-4420</td>
<td>26-150</td>
</tr>
<tr>
<td>Pyroxenite</td>
<td>2800-3240</td>
<td>39-136</td>
</tr>
<tr>
<td>Glimmerite</td>
<td>3100</td>
<td>37</td>
</tr>
<tr>
<td>Fenite</td>
<td>2610-2730</td>
<td>133-340</td>
</tr>
<tr>
<td>Granite</td>
<td>3100</td>
<td>200-300</td>
</tr>
</tbody>
</table>

UCS = Uniaxial Compressive Strength; E = Young’s modulus; $m_i$ = Hoek-Brown intact rock parameter; RMR = Rock Mass Rating; GSI = Geological Strength Index; $c_{rm}$ = rock mass cohesion; $\Phi_{rm}$ = rock mass friction angle; $T_{rm}$ = rock mass tensile strength

### Table 4.2
Results from Palabora in-situ stress studies and measurement campaigns.

<table>
<thead>
<tr>
<th>Year (reference)</th>
<th>Method</th>
<th>Stresses determined</th>
</tr>
</thead>
<tbody>
<tr>
<td>1991 (National Coal Board, 1975)</td>
<td>Back analysis</td>
<td>$\sigma_{NS} = 2.0\sigma_V$ $\sigma_{EW} = 1.5\sigma_V$</td>
</tr>
<tr>
<td>1992 (SRK Consulting Group, 1992a)</td>
<td>Borehole slotter</td>
<td>$\sigma_{NS} = 0.65\sigma_V$ $\sigma_{EW} = 0.51\sigma_V$</td>
</tr>
<tr>
<td>1992 (SRK Consulting Group, 1992b)</td>
<td>CSIR triaxial</td>
<td>$\sigma_{NS} = \sigma_V$ $\sigma_{EW} = 0.97\sigma_V$</td>
</tr>
<tr>
<td>1999 (Gash, 1999)</td>
<td>CSIRO 12</td>
<td>$\sigma_1 = 46$ MPa ($16^o/323^o$) $\sigma_2 = 38$ MPa ($73^o/152^o$) $\sigma_3 = 36$ MPa ($4^o/051^o$)</td>
</tr>
</tbody>
</table>
Table 4.3. Horizontal to vertical in-situ stress ratios ($K_o$) used in different numerical modelling studies for Palabora.

<table>
<thead>
<tr>
<th>Year</th>
<th>Method</th>
<th>$K_o$</th>
<th>Notes</th>
</tr>
</thead>
<tbody>
<tr>
<td>1991</td>
<td>FLAC</td>
<td>1.0</td>
<td>Assumed for model calibration.</td>
</tr>
<tr>
<td>1991</td>
<td>UDEC</td>
<td>1.5-2.0</td>
<td>Assumed.</td>
</tr>
<tr>
<td>1995</td>
<td>FLAC3D</td>
<td>$\sigma_{NS} = 1$</td>
<td>Based on 1992 measurements (Table 2).</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>$\sigma_{EW} = 0.97$</td>
</tr>
<tr>
<td>1998</td>
<td>UDEC</td>
<td>1.0</td>
<td>Based on 1997 geotechnical review.</td>
</tr>
<tr>
<td>1999</td>
<td>UDEC</td>
<td>1.2</td>
<td>Based on 1999 measurements (Table 2).</td>
</tr>
<tr>
<td>2000a</td>
<td>UDEC</td>
<td>1.0</td>
<td>Calibrated and back analyzed using monitoring data for East Wall.</td>
</tr>
<tr>
<td>2000b</td>
<td>UDEC</td>
<td>0.5-1.2</td>
<td>Based on 1992 and 1999 measurements (Table 2).</td>
</tr>
<tr>
<td>2004</td>
<td>3DEC</td>
<td>1.0</td>
<td>Based on modelling report.</td>
</tr>
<tr>
<td>2005</td>
<td>3DEC</td>
<td>$\sigma_{NS} = 1$</td>
<td>Based on 1992 measurements (Table 2).</td>
</tr>
<tr>
<td></td>
<td></td>
<td>$\sigma_{EW} = 1$</td>
<td></td>
</tr>
<tr>
<td>2008</td>
<td>FLAC3D</td>
<td>$\sigma_{NS} = 2.0$</td>
<td>Based on MMT project results.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>$\sigma_{EW} = 1.5$</td>
<td></td>
</tr>
<tr>
<td>2009</td>
<td>FLAC3D</td>
<td>$\sigma_{NS} = 1.5$</td>
<td>Based on MMT project results.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>$\sigma_{EW} = 2.0$</td>
<td></td>
</tr>
</tbody>
</table>

Table 4.4. Back-analyzed FLAC3D rock mass properties providing the best fit to the observed outline of the northwest wall failure. K and G denote the Bulk and Shear modulus, respectively, as derived from estimates of the rock mass Young’s modulus and Poisson’s ratio.

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Elastic Properties</th>
<th>Rock Mass Strength Properties</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Density (kg/m³)</td>
<td>K  (GPa)</td>
</tr>
<tr>
<td>Carbonatite</td>
<td>3100</td>
<td>8</td>
</tr>
<tr>
<td>Foskorite</td>
<td>3100</td>
<td>4</td>
</tr>
<tr>
<td>Pyroxenite</td>
<td>3100</td>
<td>4</td>
</tr>
<tr>
<td>Glimmerite</td>
<td>3100</td>
<td>1</td>
</tr>
<tr>
<td>Fenite</td>
<td>3100</td>
<td>16</td>
</tr>
<tr>
<td>Granite</td>
<td>3100</td>
<td>20</td>
</tr>
<tr>
<td>Caved Rock</td>
<td>2300</td>
<td>0.2</td>
</tr>
</tbody>
</table>

K = Bulk Modulus; G = Shear Modulus; $c_m$ = rock mass cohesion; $\Phi_m$ = rock mass friction angle $T_m$ = rock mass tensile strength; $\varepsilon_p$ = plastic strain threshold for material strain softening (75-90% reduction in strength)
Fig. 4.1. Digital mine plans of the Palabora open pit and undercut geometries, showing their proximate location to one another. Modified after Moss et al., 2006.
Fig. 4.2. QuickBird image of the northwest wall failure at Palabora.
Fig. 4.3. Geological map for Palabora showing the key lithologies. After Piteau Associate Engineering LTD., 2005.
Fig. 4.4. FLAC3D model developed for back analyzing the northwest wall failure (covering the period 2002 to 2005). Shown is the detailed geology built into the model in plan and 3-D perspective views. Model dimensions are 4000 x 4000 m in plan and 2000 m in depth.
Fig. 4.5. Cave geometries for different time intervals implemented into the FLAC3D models
Fig. 4.6. Back analysis comparing caving-induced displacements assuming: (a) average, and (b) lower bound rock mass properties, based on the input ranges compiled in Table 4.1. Displacements are reported in metres.
Fig. 4.7. FLAC3D results for several different in-situ stress assumptions. Superimposed are the contours outlining the northwest wall failure. Displacements are reported in metres.
Fig. 4.8. Best fit FLAC3D model comparing modelled vertical displacements (greater than 3 cm) to the DEM outline of the North wall failure, in plan and along a north-south section through the centre of the northwest wall failure. NS and EW horizontal stresses are assumed to be equal to the vertical stresses ($K_o=1$). Displacements are reported in metres.
Fig. 4.9. RADARSAT-1 data for Palabora recorded for the two-year period following the 2005 northwest wall failure. The area highlighted by the white dashed box coincides with the extent of modelled displacements (greater than 3cm) shown in Fig. 4.8.
Fig. 4.10. Modified model geometry for forward modelling of caving-induced ground deformations (with respect to the slope failure) for the period 2009 to 2010.
**Fig. 4.11.** FLAC3D forward modelling results showing the caving-induced vertical displacements for the period March 2009 to March 2010 in plan view. Displacements are reported in metres.
Fig. 4.12. FLAC3D forward modelling results showing the caving-induced vertical displacements for the period March 2009 to March 2010: (a) N-S section, and (b) E-W section. Displacements are reported in metres.
Fig. 4.13. RADARSAT-2 data for Palabora for the period March 2009 to March 2010, showing: (a) ascending, and (b) descending InSAR measured vertical displacements. Points colour-coded with respect to downward movements are, Red: 20-40mm, Yellow: 10-20mm, Green: 0-10mm.
**Fig. 4.14.** Side-by-side comparison of FLAC3D modelling results with the ascending RADARSAT-2 data for the period March 2009 to March 2010. Downward subsidence magnitudes for the zoned regions correspond to: (1) 20-40mm, (2) 10-20mm, (3) 0-10mm, and (4) shadow.

**Fig. 4.15.** Side-by-side comparison of FLAC3D modelling results with the descending RADARSAT-2 data for the period March 2009 to March 2010. Downward subsidence magnitudes for the zoned regions correspond to: (1) 20-40mm, (2) 10-20mm, (3) 0-10mm, and (4) shadow.
Fig. 4.16. Comparison of FLAC3D modelled and InSAR and mine geodetic measured vertical displacements between March 2009 and March 2010, for a history point/survey prism located near the main access shaft.
Fig. 4.17. Comparison of FLAC3D modelled and InSAR and mine geodetic measured vertical displacements between March 2009 and March 2010, for a history point/survey prism located near the ventilation shaft.
Fig. 4.18. Comparison of FLAC3D modelled and InSAR and mine geodetic measured vertical displacements between March 2009 and March 2010, for a history point/survey prism located above the crest of the west wall.
Chapter 5: Thesis Discussion and Conclusions

This thesis presents the results from a detailed investigation examining the characterization and assessment of block caving-induced subsidence using empirical and numerical techniques. First, findings from an extensive block caving database were reported identifying and defining key influences and interactions that affect the symmetry and extent of large-strain discontinuous and small-strain continuous subsidence. Results were then presented from a benchmarking study employing several different numerical techniques in which different modelling assumptions were compared (continuum, discontinuum, 2-D, 3-D, etc.). Specific focus was placed on the influence of undercut depth on the extent of different classes of subsidence: caving zone, fracture initiation zone and continuous subsidence zone. Lastly, a detailed case study of the Palabora block cave mine in South Africa was presented in which field characterization and remote-sensing data were used to constrain an advanced 3-D numerical model that simulates the ground response and interactions with an overlying deep open pit over several years of cave development.

Together, the results from this research help to further: 1) the characterization, assessment and understanding of block-caving subsidence, and its evolution, by addressing existing limitations in the use of empirical and numerical subsidence analysis methods; 2) the understanding of the fundamental processes involved in the progressive caving and its expression at surface as small-strain subsidence or larger strain fracturing; and 3) the application of sophisticated 3-D numerical modelling techniques by detailing the limitations and uncertainty arising from mine site data, specifically the representation of mine geology, rock mass properties, in-situ stresses and cave propagation, and means to constrain these input and calibrate the models through back analysis and integration with high resolution monitoring data.

5.1 Summary

5.1.1 Empirical Investigation and Characterization of Block Caving Subsidence

A detailed and comprehensive database of cave mining operations and caving-induced ground deformation observations has been developed from an exhaustive search of
published literature, university theses, and government reports for guidance on relationships between caving depth and surface subsidence. Emphasis was placed on the examination of the relationships between the caved, fractured, and subsidence zones of surface subsidence and depth of undercut as well as the effects of geology, topography, orebody type and undercut geometry in promoting asymmetry and discontinuous caving-induced subsidence. From the database, a trend emerged showing the transition by mining companies to larger and deeper caving operations indicating the need to understand the effect of undercut depth on surface subsidence more fully. Direct and indirect subsidence observations in the database were analyzed to determine the caving and fracture initiation angles as a function of undercut depth.

The range of caving and fracture initiation angles for individual sites, derived from differences relative to which side of the undercut the measurement is made (north, south, etc.), were rather large signifying a significant degree of asymmetry in the subsidence profile. This was seen to be strongly influenced by topography, the mechanism for which was demonstrated by comparative numerical modelling using the 2-D finite-element code Phase2. The numerical results showed that generally symmetric surface conditions, whether it be a flat surface or a cave centered under the peak of a mountain, generally lead to symmetric subsidence patterns; whereas asymmetric topographic conditions, primarily in the form of a sloping surface, results in asymmetric subsidence. A secondary factor influencing asymmetry involved the ore body geology as observed in variations in caving angles as a function of the resource being mined. Diamond, iron, nickel and asbestos operations were seen to have steeper caving angles and more symmetry. In the case of diamond kimberlites and asbestos deposits, the high strength contrasts between the weaker ore being caved and the stronger host rock also work to promote symmetry. In contrast, copper and molybdenum operations involving porphyry-type deposits were seen to have a significant degree of asymmetry in their caving profiles due to the more irregular shape of these types of deposits. Furthermore, a smaller strength contrast generally exists between the ore and host rock for these types of deposits. Using ore body type and mineral resource as a proxy for geology, these results show that site geology in addition to topography has a significant influence in promoting asymmetry in caving-induced displacements. This was also evident where large faults and/or rock mass fabric also influenced the directionality of caving.
These influencing factors become an important point of consideration when undertaking any subsidence analysis, as many commonly used design tools (e.g. the use of empirical design charts), do not account for the influence of topography or geological variations. The result is that symmetrical predictions of the extent of caving are produced, leading to the possibility that these may under-or over-predict the extent and magnitudes of caving-induced subsidence. Where these are under-predicted, the risk to surface infrastructure and safety may be heightened, and dilution may be more problematic than expected affecting the economics of the operations.

Another important finding from the examination of the empirical database was that the data revealed a distribution heavily weighted towards the reporting of caving angles and macro-scale deformations (large open tension cracks and offset scarps). Very little data is reported on the extent and magnitudes of smaller strain surface deformations. To examine the potential impact of this sampling bias, specifically with respect to the influence of undercut depth on the extent of surface subsidence, a series of conceptualized numerical models were developed using the commercial FEM-DEM brittle fracture code ELFEN. The modelling results, based on undercut depths between 500 and 2000 m, show a decreasing footprint on surface with increasing undercut depth with respect to the caving and macro-scale fracturing zones. This agrees with trends observed in the empirical database. When considering the extent of micro deformations (<1m), however, the opposite is true and the subsidence zone increases with increasing undercut depth. Again, these results caution against relying on existing empirical design charts and databases which do not properly extrapolate beyond the macro deformations for estimating the extent of caving-induced subsidence where small strain subsidence is of concern.

5.1.2 Benchmark Testing for Block Cave Mining Using Numerical Analysis

A benchmarking study was performed comparing a 2-D finite-element continuum code (Phase2), a 3-D finite-difference continuum code (FLAC3D), a 2-D distinct-element discontinuum code (UDEC), and a 2-D hybrid FEM-DEM brittle fracture code (ELFEN). Tested was the ability of these numerical techniques in assessing large-strain macro-deformations (caving and fracture initiation zones) and small-strain micro-deformations
(continuous subsidence), and the trends of these measures as a function of undercut depth. A conceptual geometry and set of site conditions were used as defined by the sponsoring partner for the study, the Centre for Excellence in Mining Innovation (CEMI). These were based on the panel caving of a large porphyry type deposit, incorporating faults, several different lithologies and varying rock mass properties.

The first comparison involved the results from a 2-D elasto-plastic finite-element analysis in which a network of non-persistent small-strain joint elements were either included or not. The results show that the lateral extent of subsidence is similar for the shallower undercut depths, but increases more noticeably in the jointed models as the undercut depth increases. This trend can be attributed to the weaker rock mass conditions associated with the inclusion of the rock mass fabric (i.e. joint sets). These results were then compared to those derived in 3-D using FLAC3D. Here the magnitude of subsidence was seen to be considerably less than that in 2-D, pointing to the influence of the 2-D plane strain condition in concentrating more of the strain distribution (and therefore subsidence) in the plane of analysis. The FLAC3D results for the 500 m deep undercut show that the modelled extent of subsidence is limited by the bounding faults on either side of the undercut. The subsidence profile then widens and the influence of the bounding faults diminishes as the undercut depth increases. This agrees with the 2-D observations made in Phase2. It is also noteworthy that the 3-D shape of the subsidence profile elongates in the north-south direction more so than the east-west direction as the undercut depth increases. This coincides with the north-south cross section analyzed in the 2-D modelling.

Results from the 2-D distinct-element discontinuum analysis (UDEC) showed that although being able to explicitly represent the dip and dip direction of discontinuities is important, so is the accurate representation of joint persistence. The extent of the subsidence zone modelled in UDEC was directly related to the dip direction with a significantly narrower zone being modelled when the joint sets were oriented at 0 and 90 degrees compared to the 2-D continuum analyses where the same joint orientations were used. This was due in part to the large strain slip along discontinuities allowed in UDEC, relative to the small strain capabilities in the finite element analysis, and the fully persistent nature of the orthogonal joint sets used. Where jointing in UDEC was set to orientations of 45 and 135
degrees, the fully persistent nature of the discontinuities used results in a wider zone of subsidence matching the 45 and 135 degree dip angles.

Results from the hybrid FEM-DEM brittle fracture code ELFEN showed a pattern of subsidence similar to those in the other numerical models, but produced results that were interpreted as being more reliable than the others. As with the other methods, the caving angles were constrained by the bounding faults to the north and south of the undercut. While the different model results show similar trends with most indicating overhanging angles relative to the footprint of the undercut, the results are ambiguous with respect to the influence of undercut depth on caving with the exception of the ELFEN results which show the extent of caving at surface decreases with increasing undercut depth. This is due to the decreasing extraction ratio as the height of caving for each undercut depth was kept constant while the column of rock above the undercut across which the caving-induced strains are redistributing increases with increasing undercut depth. A similar result was seen for the zone of fracture initiation with the UDEC and ELFEN results showing that the fracture initiation zone does not extend beyond the bounding faults. In the ELFEN modelling, the fracture initiation zone at surface is again seen to decrease with increasing undercut depth. Thus, the ELFEN results confirm the influence of the bounding faults in limiting the extent of large-strain macro-scale deformations. As the depth of the undercut increases, the influence of the faults is less significant as the smaller extraction ratios limit the extent of surface disturbance.

The opposite was seen to be true for the small-strain continuous subsidence, even when applying large cut off threshold of > 1 m. Each set of models: Phase2, FLAC3D, UDEC and ELFEN, showed a similar trend with the extent of small strain subsidence increasing with increasing undercut increased. The bounding faults showed little influence in limiting the extent of subsidence. Thus the results of the benchmark study confirm earlier results cautioning against relying on or scaling subsidence assessments based on macro deformations where small strain subsidence is of concern.

Different characteristics of the different numerical methods applied, and resulting differences in results, emphasize the importance of carefully defining the key objectives of the modelling and factors that are most important. The selection of a modelling method often
involves the trade-off between fast model setup and run times, and accurate representations of the rock mass fabric and caving mechanics. Overall, the 2-D hybrid FEM-DEM approach allowing stress-induced brittle fracturing between joints appeared to provide the more dependable and realistic results. However, although a 3-D FEM-DEM brittle fracture analysis would seem to incorporate all of the required key elements for a block caving subsidence hazard assessment, the required computing times are still prohibitive. This will likely change, as computer hardware and software capabilities evolve. Today, where a 3-D analysis is required, continuum-based finite-element or finite-difference codes like FLAC3D seem to be most suitable.

5.1.3 Integration of Field Characterization, Mine Production and InSAR Monitoring Data to Constrain and Calibrate 3-D Numerical Modelling of Block Caving-Induced Subsidence

Results were reported from an advanced 3-D back and forward analysis of caving-induced displacements at the Palabora open pit/block cave mine in South Africa using the commercial finite difference code FLAC3D. The representation of the block cave and its development in the model was implemented through an approach where the geometry of the cave is built into the model. Rock mass properties were derived for each of the key lithologies. From these, rock mass shear strength properties were estimated for use with a Mohr-Coulomb strain softening constitutive model. Data for the in-situ stress boundary conditions were compiled based on regional stresses reported in the published literature and mine specific in-situ stress studies. A number of reports presenting results from numerical modelling for Palabora were further reviewed and several different in-situ stress assumptions were tested based on this review for the best fit set of rock mass properties.

First, FLAC3D modelling of the 2005 northwest wall failure at Palabora was carried out to back analyze and constrain the material properties and in-situ stresses to be used for subsequent forward modelling. Results from the back analysis clearly showed a varied response for the different rock mass properties tested and the “best fit” rock mass properties were obtained. Historic InSAR data from RADARSAT-1 was examined and used to further calibrate the model. The close agreement achieved on the spatial extent and magnitudes of
the FLAC3D predicted displacements and those measured using InSAR increased the degree of confidence in the validity of the input values used in the back analysis.

These were then applied to a forward analysis (with respect to the pit slope failure date) of subsidence over the block caving operation at Palabora for the period 2009 to 2010. Whereas the back analysis of the 2005 pit slope failure represents a substantial initial step in calibrating and constraining the FLAC3D model, the failure caused a major change in the 3-D geometry of the model and this was accounted for in a modified 3-D model for the forward analysis. The results from the FLAC3D forward modelling were then compared with InSAR deformation data from the recently launched RADARSAT-2 satellite collected for the same period (2009 to 2010), in 28 day intervals. A close fit was observed between the subsidence predicted by the FLAC3D model and the InSAR measurements using the back analysis properties and the assumptions of cave growth for the same time period. The close agreement achieved on the spatial extent and magnitudes of the FLAC3D predicted displacements and those measured using InSAR (and geodetic monitoring data from the mine) allowed a greater degree of confidence in the 3-D model as a hazard assessment tool. The value of the high resolution InSAR data was shown to be significant as a means to calibrate and constrain sophisticated numerical analyses and subsidence predictions.

5.2 Key Conclusions and Scientific Contributions

The following are the key contributions of this study:

1. The block caving database developed as part of this study serves as the most thorough developed to date and a substantial source of preliminary information that can be utilized by mines in assessing the feasibility of selecting block caving as the mining method. The results highlight a bias in published experiences towards trends that are only applicable for large strain macro deformations. For these, the correlation between the extent of the caving and fracture initiation zones on surface with undercut depth (seen as decreasing with undercut depth) provide a useful guidance in assessing the “fit” of a site as a block caving candidate. Caution must be exercised though where small strain micro deformations may have adverse effects.
2. The application of advanced numerical modelling in this study demonstrated the value of such tools in helping to fill data and knowledge gaps where empirical data may be limited or biased. Results examining the influence of undercut depth on small strain continuous subsidence, where little experience exists in its direct measurement, show that its extent increases with increasing undercut depth. Because this is the opposite of what is shown for macro-scale deformations, caution is highlighted for block caving operations involving deep undercuts where the impact of subsidence is assessed based on conventional assessments. Furthermore, whereas bounding faults may serve to limit the zones of caving and fracture initiation, this is not the case for the smaller strain subsidence zone. In deeper undercuts, the extent of the subsidence zone increases despite the presence of faults.

3. This study helps lay the groundwork on which to develop, calibrate and constrain 3-D numerical models directed towards assessing caving-induced subsidence patterns. The back analysis of subsidence and pit slope deformations at the Palabora mine in South Africa served to identify factors that contributed to the unexpected slope failure and helped constrain input parameters for an advanced 3-D forward model. The 3-D forward model was compared to and further calibrated using high resolution satellite InSAR monitoring data, demonstrating the value of this new data source. Together, the potential of a fully calibrated 3-D subsidence model as a predictive tool was tested and verified for its ability to assess the extent of caving-induced subsidence for time intervals as small as 6 months.

In summary, this study completed a comprehensive block caving investigation linking empirical and numerical assessments of caving, fracture initiation and subsidence zones, from feasibility studies for a new block caving project, to mine design and production planning (e.g. draw sequencing) to mitigate potential subsidence hazards. The value of advanced numerical modelling was demonstrated as a means to address data gaps in direct measurement of subsidence across areas farther reaching than the mine footprint or property. Advanced numerical modelling was able to define and explain the relationship between
undercut depth and subsidence angle, and the capabilities of 3-D numerical modelling as a subsidence prediction tool where care is taken to fully integrate and implement all available relevant data and to constrain and calibrate the model using high resolution, wide spatial coverage data such as that offered by satellite InSAR. Doing so enabled an advanced information product to be developed and demonstrated, integrating geology, geotechnical data sets, and 3-D numerical modelling with InSAR imagery to assist mining decision makers in their development of safe and efficient block caving and open pit mining operations.

5.3 Future Research

1. The UBC block caving database developed should be updated as new data sources become available. One such data set briefly analyzed is satellite imagery like QuickBird and Google Earth. These show the full surface pattern of macro-scale caving, which through the use of pattern recognition software, may lead to further means to quantify and classify macro-scale caving-induced surface deformations.

2. This study highlighted the practical value of 3-D continuum analyses using codes like (FLAC3D). This was able to produce relatively fast results, enabling multiple model runs to test parameter sensitivity, calibration tests, etc. However, this required numerous simplifications of the mine site geology specifically in the form of the explicit representation of geological structures. These have been shown through 2-D analyses as having an important effect in properly accounting for asymmetry that may arise. However, with continuous advances in computation speed and software pre- and post-processing capabilities, further research should explore the application of 3-D discontinuum codes like 3DEC or 3-D hybrid FEM-DEM brittle fracture analyses using EFLEN. Although these demand significantly more data processing and computing capacity, they allow an improved treatment of the rock mass fabric, heterogeneity and simulation of caving mechanics.

3. The modelling performed in this study demonstrated the value of considering post-peak behaviour through the application of a strain softening model. However, this could benefit from a more thorough examination of the sensitivity of the results to
different critical strain thresholds and application of more complex constitutive models that better account for the influence of depth and confinement on both the peak and post-peak behaviour (e.g. Hoek-Brown with strain softening).

4. As shown in the benchmarking study, the 3-D modelling results showed sensitivity to the full 3-D stress field initialized, especially for deeper undercut depths. Under high stress conditions, preliminary results suggest that subsidence tends to be more sensitive to the stress condition than to the rock mass fabric. Given the industry trend towards deeper undercuts, it is recommended that further numerical analysis be carried out on the importance and influence of the intermediate principal stress.

5. While the benchmarking study suggests the importance of considering small-strain subsidence in mine design, limitations and errors inherent in numerical modelling (e.g. related to mesh size effects) could be more significant for small-strain subsidence. Rigorous examination of such errors especially in relation to the mesh size needs to be performed to further validate the argument of the importance of small-strain subsidence.

6. Lastly, all FEM-DEM brittle fracture modelling was carried out using a Mohr-Coulomb brittle fracture constitutive model. It is anticipated that a newly developed Hoek-Brown brittle fracture constitutive model will soon be implemented in ELFEN, which will allow for a better simulation of depth dependent rock mass properties.
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Appendix A: Derived Subsidence Data used to Develop Empirical Database
A.1 Subsidence data for block cave operations caving to surface exclusive of those caving into an open pit. Angles are reported as ranges where asymmetry occurs between the hangingwall and footwall sides of the ore body. Angles greater than 90° refer to overhanging angles.

<table>
<thead>
<tr>
<th>Mine (location)</th>
<th>Orebody/Lift</th>
<th>Mining Period Reported</th>
<th>Undercut Depth (m)</th>
<th>Caving Angle</th>
<th>Fracture Initiation Angle</th>
<th>Angle of Subsidence</th>
<th>Data Confidence/Comments</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Alaska-Juneau (Alaska, USA)</td>
<td>North</td>
<td>1923-1944</td>
<td>450</td>
<td>88-93</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section with scale bar). No information is given on mining operations.</td>
<td>Petrilloand Hilbelink, 1999</td>
</tr>
<tr>
<td></td>
<td>South</td>
<td>1923-1944</td>
<td>255</td>
<td>52-82</td>
<td>52-62</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with mine levels and depths; assumed to be drawn to scale). Early stage of cave development and production depicted. Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Lower angle in range corresponds with uphill side of sloping surface.</td>
<td>Torres et al, 1981</td>
</tr>
<tr>
<td>Andina Rio Blanco (Chile)</td>
<td>Panel I (Block 1)</td>
<td>1970-1980</td>
<td>135</td>
<td>85-89</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with mine levels and depths; assumed to be drawn to scale). Early stage of cave development and production depicted. Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Lower angle in range corresponds with uphill side of sloping surface.</td>
<td>Flores and Karzulovic, 2002</td>
</tr>
<tr>
<td>Andina Rio Blanco (Chile)</td>
<td>Panels I &amp; II</td>
<td>1978-1995</td>
<td>390</td>
<td>61-76</td>
<td>-</td>
<td>-</td>
<td>Good (cross-section with scale bar). Caving and subsidence on surface partly concealed by thick blanket of glacial till. Cave propagation and boundaries partly controlled by vertical dykes. Marginal (subsidence map with scale bar; depths determined from secondary information and used to calculate angles). Boundary level drifts used to control the lateral extent of caving. Marginal (cross-section with scale bar; full limits of caving zone not shown but reported in text). Undercut is located under steep topography.</td>
<td>Boyum, 1961</td>
</tr>
<tr>
<td>Athens (Michigan, USA)</td>
<td>Blocks 1-4 and 1 &amp; 2, Lift 2</td>
<td>1919-1951</td>
<td>670</td>
<td>84-94</td>
<td>80-90</td>
<td>-</td>
<td>Marginal (subsidence map with scale bar; depths determined from secondary information and used to calculate angles). Boundary level drifts used to control the lateral extent of caving. Marginal (cross-section with scale bar; full limits of caving zone not shown but reported in text). Undercut is located under steep topography.</td>
<td>Hardwick, 1959</td>
</tr>
<tr>
<td>Bagdad (Arizona, USA)</td>
<td>West</td>
<td>1937-1947</td>
<td>265</td>
<td>84-90</td>
<td>72-86</td>
<td>-</td>
<td>Marginal (cross-section with scale bar; full limits of caving zone not shown but reported in text). Undercut is located under steep topography.</td>
<td>Weisz, 1958</td>
</tr>
<tr>
<td>Mine (location)</td>
<td>Orebody/Lift</td>
<td>Mining Period Reported</td>
<td>Undercut Depth (m)</td>
<td>Caving Angle</td>
<td>Fracture Initiation Angle</td>
<td>Angle of Subsidence</td>
<td>Data Confidence/Comments</td>
<td>Source</td>
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<tr>
<td>Climax (Colorado, USA)</td>
<td>Phillipson Level</td>
<td>1940-1945</td>
<td>145</td>
<td>86-95</td>
<td>61-95</td>
<td>-</td>
<td>Good (cross-section with scale bar). Lower angles in ranges correspond with uphill side of sloping surface and retrogressive slumping towards cave. Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Mining of thick interval of coal using multiple lifts. Neighbouring section mined by top slice method.</td>
<td>Vanderwilt, 1949</td>
</tr>
<tr>
<td>Corbin (B.C, Canada)</td>
<td>No. 6 Mine – West</td>
<td>1917-1934</td>
<td>80</td>
<td>60-83</td>
<td>-</td>
<td>-</td>
<td></td>
<td>Warburton, 1936</td>
</tr>
<tr>
<td>Crestmore (California, USA)</td>
<td>Stanley Bed – Block 1A</td>
<td>1930-1954</td>
<td>60</td>
<td>70-90</td>
<td>55-88</td>
<td>-</td>
<td>Good (cross-section with scale bar). Vertical cutoff stopes excavated on all four sides of block to control caving angles. Caving on footwall side of block extended beyond this to align with a dipping fault (lower angle in fracture initiation range). Good (cross-section with scale bar). Undercut is located under a steep slope; lower caving angle in range corresponds to the uphill side.</td>
<td>Long and Obert, 1958</td>
</tr>
<tr>
<td>Inspiration (Arizona, USA)</td>
<td>Transfer Block</td>
<td>1947-1963</td>
<td>70</td>
<td>85-105</td>
<td>-</td>
<td>-</td>
<td>Good (cross-section with scale bar). Caving of small transfer block following transition from block caving to open pit mining. Good (subsidence map and cross-section with scale bars). Periphery of caving zone marked by single, continuous, steep-wall face with little to no change in subsidence outside this area.</td>
<td>Hardwick, 1963</td>
</tr>
<tr>
<td>Jenifer (California, USA)</td>
<td>Jenifer</td>
<td>1952-1957</td>
<td>160</td>
<td>81-95</td>
<td>81-95</td>
<td>81-95</td>
<td>Marginal (cross-section without scale bar but with mine levels and depths; assumed to be drawn to scale). Steeper angle in ranges coincides with caving parallel to footwall of dipping orebody; lower angles occur on uphill side of caving zone.</td>
<td>Obert and Long, 1962</td>
</tr>
<tr>
<td>Mine (location)</td>
<td>Orebody/Lift</td>
<td>Mining Period Reported</td>
<td>Undercut Depth (m)</td>
<td>Caving Angle</td>
<td>Fracture Initiation Angle</td>
<td>Angle of Subsidence</td>
<td>Data Confidence/Comments</td>
<td>Source</td>
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<tr>
<td>Lake Superior (Michigan, USA)</td>
<td>Marquette range (Case C; Types D-F)</td>
<td>?</td>
<td>450</td>
<td>80</td>
<td>70</td>
<td>-</td>
<td>Poor (schematic cross-section without scale bar; assumed to be roughly drawn to scale). No direct indication given of mining depth. Bedrock is covered by a thick blanket of glacial till that partly obscures the caving and subsidence zones at surface. Good (cross-section with scale bar). Caving limits controlled by vertical boundary drifts. Caving on one side extends into the “Main Orebody” previously mined by sublevel caving. Lower angles are sub-parallel to foliation of schist. Good (cross-section with scale bar). Caving limits controlled by vertical boundary drifts. Angles reported to have flattened considerably since 1929 measurements.</td>
<td>Crane, 1929</td>
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<td>Miami (Arizona, USA)</td>
<td>Low Grade Orebody</td>
<td>1926-1929</td>
<td>195</td>
<td>71-84</td>
<td>50-73</td>
<td>-</td>
<td></td>
<td>MacLennan, 1929</td>
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<tr>
<td></td>
<td>Stope 11</td>
<td>1928-1929</td>
<td>195</td>
<td>83-92</td>
<td>76-87</td>
<td>-</td>
<td></td>
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<tr>
<td>Miami (Arizona, USA)</td>
<td>Main/Low Grade Orebodies</td>
<td>1910-1939</td>
<td>195</td>
<td>62-84</td>
<td>40-70</td>
<td>-</td>
<td></td>
<td>Fletcher, 1960</td>
</tr>
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<td></td>
<td>720 Levels</td>
<td>1910-1958</td>
<td>300</td>
<td>60-69</td>
<td>47-56</td>
<td>-</td>
<td></td>
<td></td>
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<tr>
<td>Northparkes (Australia)</td>
<td>E26 Lift 1</td>
<td>1993-2000</td>
<td>450</td>
<td>84-88</td>
<td>-</td>
<td>-</td>
<td>Collapse of crown pillar related to change in geology resulted in near-vertical cave angles. Lift also caved into the bottom</td>
<td>Duffield, 2000</td>
</tr>
<tr>
<td>Mine (location)</td>
<td>Orebody/Lift</td>
<td>Mining Period Reported</td>
<td>Undercut Depth (m)</td>
<td>Caving Angle</td>
<td>Fracture Initiation Angle</td>
<td>Angle of Subsidence</td>
<td>Data Confidence/Comments</td>
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<tr>
<td>Questa (New Mexico, USA)</td>
<td>Goathill</td>
<td>1983-2000</td>
<td>300</td>
<td>70-85</td>
<td>51-84</td>
<td>32-81</td>
<td>Good (subsidence map with scale bar; depths determined from secondary information and used to calculate angles). Lower angle in ranges coincide with deformations related to shallow rockslide movements undercut by cave. Marginal (subsidence map with scale bar; depths determined from secondary information and used to calculate angles). Subsidence not developed enough to allow measurement of caving or fracture initiation angles.</td>
<td>Gilbride et al, 2005</td>
</tr>
<tr>
<td>San Manuel (Arizona, USA)</td>
<td>South Orebody, Lift 1</td>
<td>1956-1960</td>
<td>420</td>
<td>64-95</td>
<td>53-95</td>
<td>-</td>
<td>Good (cross-sections and subsidence map with scale bars).</td>
<td>Gilbride et al, 2005, Buchanan and Buchella, 1960</td>
</tr>
<tr>
<td></td>
<td>South Orebody, Lift 1</td>
<td>1956-1962</td>
<td>420</td>
<td>56-90</td>
<td>56-66</td>
<td>-</td>
<td>Good (subsidence map with scale bar). Final reporting of subsidence for mining of South orebody, Lift 1. Good (cross-sections and subsidence map with scale bars). Active subsidence contained within established boundaries for Lift 1, with only minor activity outside this periphery.</td>
<td>Thomas, 1971</td>
</tr>
<tr>
<td>San Manuel (Arizona, USA)</td>
<td>North Orebody, West</td>
<td>1959-1970</td>
<td>390</td>
<td>78-86</td>
<td>66-74</td>
<td>-</td>
<td>Good (subsidence map with scale bar). West and East blocks separated from one another by a 200 m pillar. Good (subsidence map with scale bar). West and East blocks separated from one another by a 200 m pillar.</td>
<td>Wilson, 2000</td>
</tr>
<tr>
<td>Shabani (Zimbabwe)</td>
<td>52 &amp; 58</td>
<td>1987-1999</td>
<td>630</td>
<td>75-83</td>
<td>-</td>
<td>-</td>
<td>Inclined undercut dipping at</td>
<td></td>
</tr>
</tbody>
</table>
A.2 **Subsidence data for panel cave operations caving to surface exclusive of those caving into an open pit. Angles are reported as ranges where asymmetry occurs between the hangingwall and footwall sides of the ore body. Angles greater than 90° refer to overhanging angles.**

<table>
<thead>
<tr>
<th>Mine (location)</th>
<th>Orebody/Lift</th>
<th>Mining Period Reported</th>
<th>Undercut Depth (m)</th>
<th>Caving Angle</th>
<th>Fracture Initiation Angle</th>
<th>Angle of Subsidence</th>
<th>Data Confidence/Comments</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Climax (Colorado, USA)</td>
<td>600 Level</td>
<td>1945-1980</td>
<td>325</td>
<td>72-74</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with mine levels and depths; assumed to be drawn to scale). Continuation following transition from block to panel caving. Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Blasting used to induce caving beyond limits of previously mined stopes.</td>
<td>Vera, 1981</td>
</tr>
<tr>
<td>Creighton (Ontario, Canada)</td>
<td>23 Level</td>
<td>1951-1955</td>
<td>420</td>
<td>90</td>
<td>-</td>
<td>-</td>
<td></td>
<td>Brock et al., 1956</td>
</tr>
<tr>
<td>Creighton (Ontario, Canada)</td>
<td>1900 Level</td>
<td>1951-1963</td>
<td>420</td>
<td>74-88</td>
<td>62-79</td>
<td>55</td>
<td>Good (subsidence map with measured angles together with cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale).</td>
<td>Dickhout, 1963</td>
</tr>
<tr>
<td>El Teniente (Chile)</td>
<td>South 1 (Ten 1 Sur)</td>
<td>1940-1980</td>
<td>510</td>
<td>60-88</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Undercut is positioned under a steep slope; lower caving angles occur on the uphill side.</td>
<td>Ovalle, 1981, Ovalle, 1981, Kvapil et al., 1989</td>
</tr>
<tr>
<td>El Teniente (Chile)</td>
<td>North 4 (Ten Norte)</td>
<td>1960-1980</td>
<td>540</td>
<td>70-80</td>
<td>-</td>
<td>-</td>
<td>Poor (empirical chart of caving angle versus depth based on numerical modelling and field observations; no depth is reported for the undercut but can be estimated from other sources).</td>
<td></td>
</tr>
<tr>
<td>El Teniente (Chile)</td>
<td>Regimiento 4</td>
<td>1982-1998</td>
<td>250</td>
<td>82-87</td>
<td>-</td>
<td>-</td>
<td></td>
<td>Brown, 2003</td>
</tr>
<tr>
<td>El Teniente (Chile)</td>
<td>Esmeralda</td>
<td>1997-2001</td>
<td>800</td>
<td>65-77</td>
<td>58-67</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with mine levels and depths, together with empirical chart of caving angle versus depth). Undercut is positioned under a steep</td>
<td>Rojas et al., 2003</td>
</tr>
</tbody>
</table>

approximately 30° from horizontal.
<table>
<thead>
<tr>
<th>Mine (location)</th>
<th>Orebody/Lift</th>
<th>Mining Period Reported</th>
<th>Undercut Depth (m)</th>
<th>Caving Angle</th>
<th>Fracture Initiation Angle</th>
<th>Angle of Subsidence Data</th>
<th>Confidence/Comments</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Grace (Pennsylvania, USA)</td>
<td></td>
<td>1958-2004</td>
<td>750</td>
<td>80-110</td>
<td>70-86</td>
<td>Marginal (subsidence map showing limits of surface cracking; angles estimated based on average depth of undercut). Inclined panel cave (20-30°). Cave breakthrough only occurred above half of the undercut, facilitated by a steeply dipping fault. Good (cross-section with scale bar together with subsidence map). Vertically spaced boundary cutoff drifts together with steeply dipping faults contribute to vertical nature of cave.</td>
<td>Sainsbury et al, 2010</td>
<td></td>
</tr>
<tr>
<td>Henderson (Colorado, USA)</td>
<td>8100 Level Panel 1</td>
<td>1976-1983</td>
<td>1050</td>
<td>90-98</td>
<td>86-92</td>
<td>Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale) Marginal (cross-section without scale bar but with mine levels and depths; assumed to be drawn to scale). Caving zone depicted prior to and after air blast collapse of cave back. Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Undercut beneath steep hill. Caving limits controlled by vertical boundary cutoff stopes (shrinkage stopes). Considerable blasting required to aid caving.</td>
<td>Brumleve and Maier, 1981, Stewart et al, 1984</td>
<td></td>
</tr>
<tr>
<td>Henderson (Colorado, USA)</td>
<td>7700 level</td>
<td>1976-2000</td>
<td>1150</td>
<td>90-100</td>
<td>-</td>
<td>-</td>
<td>Rech et al, 2000</td>
<td></td>
</tr>
<tr>
<td>Salvador (Chile)</td>
<td>Inca West</td>
<td>1994-2000</td>
<td>700</td>
<td>70-76</td>
<td>-</td>
<td>-</td>
<td>Escobar, 2000</td>
<td></td>
</tr>
<tr>
<td>Urad (Colorado, USA)</td>
<td>1100 Level</td>
<td>1967-1969</td>
<td>150</td>
<td>90</td>
<td>82</td>
<td>-</td>
<td>-</td>
<td>Kendrick, 1970</td>
</tr>
</tbody>
</table>
### A.3 Subsidence data for sublevel caving/shrinkage stoping/top slicing operations caving to surface exclusive of those caving into an open pit.

Angles are reported as ranges where asymmetry occurs between the hangingwall and footwall sides of the ore body. Angles greater than 90° refer to overhanging angles.

<table>
<thead>
<tr>
<th>Mine (location)</th>
<th>Orebody/Lift</th>
<th>Period Reported</th>
<th>Undercut Depth (m)</th>
<th>Caving Angle</th>
<th>Fracture Initiation Angle</th>
<th>Angle of Subsidence</th>
<th>Data Confidence/Comments</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cambria Jackson (Michigan, USA)</td>
<td>260 Sublevel</td>
<td>1942-1945</td>
<td>350</td>
<td>86-94</td>
<td>71-72</td>
<td>-</td>
<td>Good (cross-section with scale bar). Cave propagation through competent diorite sill capping weak hematite iron formation.</td>
<td>Boyum, 1961</td>
</tr>
<tr>
<td>Copper Mountain (B.C., Canada)</td>
<td>Contact Block</td>
<td>1937-1949</td>
<td>350</td>
<td>79-90</td>
<td>69-74</td>
<td>65</td>
<td>Good (cross-section with scale bar).</td>
<td>Nelson and Fahrni, 1950</td>
</tr>
<tr>
<td></td>
<td>122-East Block</td>
<td>1941-1949</td>
<td>210</td>
<td>82-90</td>
<td>67-74</td>
<td>-</td>
<td>Good (cross-section with scale bar). Lower angle in range aligns with dipping fault.</td>
<td></td>
</tr>
<tr>
<td>Copper Queen East Orebody (Arizona, USA)</td>
<td>600 &amp; 650 Lift 725 Lift 850 Lift 950 Lift 1100 Lift</td>
<td>1925-1927 1927-1928 1928-1930 1930-1931 1931-1933</td>
<td>155 180 215 240 290</td>
<td>68-98 64-88 57-75 56-75 54-70</td>
<td>58-95 56-72 54-72 52-72 45-67</td>
<td>-</td>
<td>Good (multiple cross-sections without scale bars but with mine levels and depths; assumed to be drawn to scale). Lower angles correspond with weak hangingwall rock, relative to higher angles in stronger footwall rock.</td>
<td>Kantner, 1934</td>
</tr>
<tr>
<td>Copper Queen – Queen Hill (Arizona, USA)</td>
<td>Queen Hill Block</td>
<td>1913-1933</td>
<td>100</td>
<td>78</td>
<td>78</td>
<td>78</td>
<td>Marginal (cross-section without scale bar but with mine levels and depths; assumed to be drawn to scale). Caved block is bound on all sides by faults, along which the block drops and across which subsidence is limited. Marginal (cross-section without scale bar; scale estimated from other data provided; assumed to be roughly drawn to scale). Undercut inclined at 38°, extending from surface to same level at depth of neighbouring section mined by block caving.</td>
<td>Flores and Kazulovic, 2002</td>
</tr>
<tr>
<td>Corbin (B.C, Canada)</td>
<td>No. 6 Mine - East</td>
<td>1917-1934</td>
<td>80</td>
<td>40-52</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section without scale bar; scale estimated from other data provided; assumed to be roughly drawn to scale). Undercut inclined at 38°, extending from surface to same level at depth of neighbouring section mined by block caving.</td>
<td>Warburton, 1936</td>
</tr>
<tr>
<td>Mine (location)</td>
<td>Orebody/Lift</td>
<td>Mining Period Reported</td>
<td>Undercut Depth (m)</td>
<td>Caving Angle</td>
<td>Fracture Initiation Angle</td>
<td>Angle of Subsidence</td>
<td>Data Confidence/Comments</td>
<td>Source</td>
</tr>
<tr>
<td>----------------</td>
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<td>--------</td>
</tr>
<tr>
<td>Gath’s (Rhodesia/Zimbabwe)</td>
<td>99 Level</td>
<td>1971-1976</td>
<td>60</td>
<td>50-75</td>
<td>50-75</td>
<td>-</td>
<td>Good (cross-section with caving angles reported). Lower angles correspond with dip of orebody; steeper angles correspond to caving in dipping hangingwall.</td>
<td>Brown and Ferguson, 1979</td>
</tr>
<tr>
<td></td>
<td>158 Level</td>
<td></td>
<td>120</td>
<td>50-65</td>
<td>50-65</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>183 Level</td>
<td></td>
<td>145</td>
<td>50-56</td>
<td>50-56</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grangesberg (Sweden)</td>
<td>140 Level</td>
<td>?-1921</td>
<td>140</td>
<td>62-80</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with caving angles and sublevel depths). Caving angle on footwall side coincides with dip of inclined orebody at 62°.</td>
<td>Hoek, 1974</td>
</tr>
<tr>
<td></td>
<td>180 Level</td>
<td>1921-1933</td>
<td>180</td>
<td>62-88</td>
<td>-</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>190 Level</td>
<td>1933-1936</td>
<td>190</td>
<td>62-80</td>
<td>-</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>210 Level</td>
<td>1936-1939</td>
<td>210</td>
<td>62-80</td>
<td>-</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>240 Level</td>
<td>1939-1943</td>
<td>240</td>
<td>62-64</td>
<td>-</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>300 Level</td>
<td>1943-1961</td>
<td>300</td>
<td>60-62</td>
<td>-</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grangesberg (Sweden)</td>
<td>410 Level</td>
<td>1960-1974</td>
<td>410</td>
<td>64-83</td>
<td>-</td>
<td>-</td>
<td>Marginal (cross-section without scale bar but with sublevel depths; assumed to be roughly drawn to scale). Caving angle on footwall side coincides with dip of inclined orebody at 64°.</td>
<td>Sisselman, 1974</td>
</tr>
<tr>
<td>Havelock (Swaziland)</td>
<td>Level 1</td>
<td>1952-1972</td>
<td>135</td>
<td>52-82</td>
<td>52-78</td>
<td>-</td>
<td>Good (cross-section with scale bar). Lower angles in range controlled by dip of bedding in footwall. Deformation in hangingwall develops through flexural toppling and shearing along bedding.</td>
<td>Heslop, 1974</td>
</tr>
<tr>
<td></td>
<td>Level 2</td>
<td>1963-1972</td>
<td>180</td>
<td>52-90</td>
<td>52-64</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Level 3</td>
<td>1966-1972</td>
<td>225</td>
<td>52-90</td>
<td>52-60</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Kiirunavaara/Kiruna (Sweden)</td>
<td>700 Level</td>
<td>1965-1995</td>
<td>465</td>
<td>60-94</td>
<td>53-74</td>
<td>40-60</td>
<td>Marginal (subsidence map but without indication of the sublevel depth; sublevel depth estimated from other sources). Caving angle on footwall side coincides with dip of orebody. Higher caving angle points to overhanging nature of dipping hangingwall.</td>
<td>Henry and Dahnér-Lindqvist, 2000 Henry et al, 2004</td>
</tr>
<tr>
<td></td>
<td>785 Level</td>
<td>1965-2000</td>
<td>500</td>
<td>50-82</td>
<td>50-60</td>
<td>40-50</td>
<td>Marginal (subsidence map; angles calculated based on projection of lowest sublevel undercut and depth at time of</td>
<td>Henry et al, 2004</td>
</tr>
<tr>
<td>Mine (location)</td>
<td>Orebody/Lift</td>
<td>Mining Period Reported</td>
<td>Undercut Depth (m)</td>
<td>Caving Angle</td>
<td>Fracture Initiation Angle</td>
<td>Angle of Subsidence</td>
<td>Data Confidence/Comments</td>
<td>Source</td>
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<td>----------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------</td>
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</tr>
<tr>
<td>Kiirunavaara/Kiruna (Sweden)</td>
<td>400 Level</td>
<td>1965-1971</td>
<td>165</td>
<td>-</td>
<td>60-74</td>
<td>-</td>
<td>Good (cross-section without scale bar but with sublevel depths; assumed to be drawn to scale). Only fracture initiation angle on hangingwall side provided; no indication of caving angles for the same periods. Fracture initiation angle on footwall side reported as coinciding with dip of orebody (60°).</td>
<td>Villegas, 2008</td>
</tr>
<tr>
<td></td>
<td>415 Level</td>
<td>1971-1974</td>
<td>180</td>
<td>-</td>
<td>60-66</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>425 Level</td>
<td>1974-1977</td>
<td>190</td>
<td>-</td>
<td>60-61</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>465 Level</td>
<td>1977-1981</td>
<td>230</td>
<td>-</td>
<td>60-63</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>530 Level</td>
<td>1981-1985</td>
<td>285</td>
<td>-</td>
<td>60-63</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>570 Level</td>
<td>1985-1989</td>
<td>325</td>
<td>-</td>
<td>60-61</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>750 Level</td>
<td>1989-1995</td>
<td>505</td>
<td>-</td>
<td>60-66</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>865 Level</td>
<td>1995-2005</td>
<td>615</td>
<td>-</td>
<td>60-73</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Lake Superior (Michigan, USA)</td>
<td>Gogebic range (Case B; Type C)</td>
<td>?</td>
<td>300</td>
<td>86-105</td>
<td>-</td>
<td>-</td>
<td>Poor (cross-section without scale bar or mining depth, but with caving angles; assumed to be drawn to scale). Caving occurs primarily along steeply dipping footwall slates.</td>
<td>Crane, 1929</td>
</tr>
<tr>
<td></td>
<td>Malmberget (Sweden)</td>
<td>Pillar Recovery</td>
<td>1970-1974</td>
<td>300</td>
<td>78-92</td>
<td>-</td>
<td>Good (cross-section with scale bar). Lower angle coincides with footwall, whereas higher angle points to overhanging hangingwall. Good (cross-section with scale bar). Mostly mined by top slicing and sublevel caving. Caving limits controlled by vertical boundary drifts. Lower angles are sub-parallel to foliation of schist. Marginal (cross-section without scale bar but with sublevel depths; assumed to be drawn to scale). Lower angles coincide with dip of footwall (70°).</td>
<td>Hannweg and Van Hout, 2001</td>
</tr>
<tr>
<td></td>
<td>300 Level</td>
<td>Main Orebody</td>
<td>1910-1925</td>
<td>180</td>
<td>60-84</td>
<td>60-68</td>
<td></td>
<td>MacLennan, 1929</td>
</tr>
<tr>
<td>Mine (location)</td>
<td>Orebody/Lift</td>
<td>Mining Period Reported</td>
<td>Undercut Depth (m)</td>
<td>Caving Angle</td>
<td>Fracture Initiation Angle</td>
<td>Angle of Subsidence</td>
<td>Data Confidence/Comments</td>
<td>Source</td>
</tr>
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</tr>
<tr>
<td>Perseverance (Australia)</td>
<td>10130 Level</td>
<td>1989-1995</td>
<td>390</td>
<td>84-90</td>
<td>-</td>
<td>-</td>
<td>Marginal (subsidence map without scale bar; depths determined from secondary information and used to calculate angles; assumed to be drawn to scale). Sublevel caving beneath large open pit. Lower caving and fracture initiation angles extend beyond pit limits on hangingwall side of orebody.</td>
<td>Tyler et al, 2004</td>
</tr>
<tr>
<td></td>
<td>10100 Level</td>
<td>1995-1996</td>
<td>420</td>
<td>66-90</td>
<td>-</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>10030 Level</td>
<td>1996-1997</td>
<td>490</td>
<td>66-87</td>
<td>63-90</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>9920 Level</td>
<td>1997-1998</td>
<td>600</td>
<td>73-81</td>
<td>63-81</td>
<td>-</td>
<td></td>
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<tr>
<td></td>
<td>9870 Level</td>
<td>1998-1999</td>
<td>650</td>
<td>74-80</td>
<td>63-80</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
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<td>9860 Level</td>
<td>1999-2000</td>
<td>660</td>
<td>70-80</td>
<td>62-80</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>9850 Level</td>
<td>2000-2001</td>
<td>670</td>
<td>70-83</td>
<td>62-83</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>9815 Level</td>
<td>2001-2002</td>
<td>705</td>
<td>73-85</td>
<td>65-85</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>9760 Level</td>
<td>2002-2003</td>
<td>760</td>
<td>72-84</td>
<td>66-83</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rajpura Dariba (India)</td>
<td>South 465 Level</td>
<td>?</td>
<td>185</td>
<td>70-90</td>
<td>55-70</td>
<td>-</td>
<td>Poor (no data provided; angles cited in text). 70° angle coincides with dip of footwall. Marginal (subsidence map and cross-section without scale bar but with mining levels and depths; assumed to be drawn to scale).</td>
<td>Singh, 1993</td>
</tr>
<tr>
<td>San Giovanni (Italy)</td>
<td>Contatto Ovest</td>
<td>1985-1990</td>
<td>100</td>
<td>75-92</td>
<td>-</td>
<td>-</td>
<td></td>
<td>Balia et al, 1990</td>
</tr>
</tbody>
</table>
### A.4 Subsidence data for block cave operations caving to surface into an open pit.

<table>
<thead>
<tr>
<th>Mine (location)</th>
<th>Orebody/Lift</th>
<th>Mining Period Reported</th>
<th>Undercut Depth (m)</th>
<th>Caving Angle</th>
<th>Fracture Initiation Angle</th>
<th>Angle of Subsidence</th>
<th>Data Confidence/Comments</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Finsch (South Africa)</td>
<td>Block 4</td>
<td>2004-2006</td>
<td>700</td>
<td>74-82</td>
<td>-</td>
<td>-</td>
<td>Good (cross-section with scale bar). Block caving into existing workings mined by open stoping, and earlier by large open pit. Caving angles coincide with walls of already existing crater.</td>
<td>Preece and Liebenberg, 2007</td>
</tr>
<tr>
<td>Jagersfontein (South Africa)</td>
<td>1870 level</td>
<td>1947-1962</td>
<td>550</td>
<td>75-82</td>
<td>-</td>
<td>-</td>
<td>Good (cross-section with scale bar). Block caving into existing underground workings and open pit. Caving angles incorporate open pit and sloughing of wall rock.</td>
<td>Stucke, 1965</td>
</tr>
<tr>
<td>Koffiefontein (South Africa)</td>
<td>49 Level</td>
<td>1987-2001</td>
<td>480</td>
<td>90</td>
<td>-</td>
<td>-</td>
<td>Poor (cross-section without scale bar but with mine levels; depths determined from other sources; assumed to be drawn to scale). Poor (cross-section without scale bar but with mining level; depths determined from other sources). Lower fracture initiation angle coincides with back scarp of large rockslide that developed in deep open pit.</td>
<td>Hannweg and Van Hout, 2001</td>
</tr>
<tr>
<td>Palabora (South Africa)</td>
<td>Lift 1</td>
<td>2001-2007</td>
<td>1200</td>
<td>84-86</td>
<td>60-84</td>
<td>-</td>
<td>Poor (cross-section without scale bar but with mining level; depths determined from other sources). Lower fracture initiation angle coincides with back scarp of large rockslide that developed in deep open pit.</td>
<td>Pretorius, 2007</td>
</tr>
</tbody>
</table>
Appendix B : Detailed Background Descriptions of Subsidence Observations Compiled in the Empirical Database
Alaska-Juneau (Alaska, USA) – 1929

Source: Bradley (1929)

Caving Period Reported: 1916-1929

Summary: Block cave mining methods applied to the Alaska-Juneau gold mine are reported. In describing the system of caving, a schematic cross-section is included that depicts the caving zone and surface fractures that develop over a single block undercut from the 4 Level at 200 m depth. It is assumed that the section is based on visual indicators; no indication is given that subsidence measurements were made. The sections are to scale and show the depth of the undercut workings and original surface. Blasting was used to aid the caving process, and the presence of a dipping fault plays a controlling role in the extent of caving and subsidence on the footwall side of the orebody.
Alaska-Juneau (Alaska, USA) – 1944

Source: Petrillo & Hilbelink (1999)

Caving Period Reported: 1923-1944

Summary: The history of the Alaska-Juneau gold mine is reported, encompassing the North, South and Perseverance orebodies, which were mined using a combination of block caving and shrinkage stoping. Minimal data is provided regarding the mining operations and no specific data regarding subsidence measurements are reported. However, a scaled cross-section is provided, which shows the caving zone and undercuts from which the caving angles can be estimated.
Andina-Rio Blanco (Chile) – Panel I

*Source:* Torres et al. (1981)

*Caving Period Reported:* 1970-1980

*Summary:* The block caving operations for Panel I at Andina’s Rio Blanco mine are described. This includes a cross-section showing the surface profile, undercut level, mining of the first block in Panel I, and the corresponding caved area at surface. No direct data in the form of subsidence measurements is given, but it is assumed that the altered surface profile is based on visual observations. It is also assumed the cross-section is to scale; no scale bar is provided, but the different mine levels are shown with their respective elevations from which the depth of mining can be approximated. The undercut is located under the steep slope of a mountain. However, the early stage of cave development and production depicted is not yet shown to be influenced by topography.
Andina-Rio Blanco (Chile) – Panel I & II

*Source:* Flores & Karzulovic (2002)

*Caving Period Reported:*

**Summary:** Data is reported from the ICS II benchmarking study for the Andina mine (formerly the Rio Blanco mine). Focus is placed on the planned third lift (Panel III), which will be mined by panel caving. However, a schematic cross-section is provided showing the state of caving above Panels I and II, mined by block caving. It is assumed the cross-section is to scale; no scale bar is provided, but the undercut levels are indicated from which the depth can be approximated based on the reported elevations of the Panel II and III undercuts. The caving zone is located under the steep slope of a mountain, for which the lower angle of caving corresponds with the uphill side of sloping surface. The report also provides a range of break angles, defined as the mean inclination of the caving crater walls at various depths for different ranges of MRMR values. However, because the angles do not appear to be measured with respect to the undercut level, they are not reported in the above Tables. The ranges cited are 52-90° for MRMR 41 to 50, 58-90° for MRMR 51 to 60, and 70-90° for MRMR 61 to 70.

*Photograph 9.9:* Typical subsidence crater of a block caving mine, II Panel, Andina Mine, Chile.
Athens (Michigan, USA) – 1932

**Source:** Allen (1934)

**Caving Period Reported:** 1918-1932

**Summary:** Data is reported for the Athens mine in the Marquette iron range. This is one of the unnamed regions reported by Crane (1929); see Lake Superior District below. The caving period reported is from the start of mining to the completion of Blocks 1 and 2 at 630 m depth. Mining was carried out using a combined block caving and top slicing approach progressing upward in successive blocks to the east. A scaled cross-section is provided based on surface observations of caving features, which shows the extent of caving on surface, depth of mining and caving angles. The cross-section also shows that a thick blanket of glacial till covers the bedrock, partly obscuring the subsidence zones at surface. Upward propagation of the cave is shown to be bounded and controlled in part by two vertical diorite dykes.

![Cross-section of the Athens mine](image)

**FIG. 1.—MINING SYSTEM AND RESULTING SUBSIDENCE, ATHENS MINE.**
Athens (Michigan, USA) – 1950

Source: Boyum (1961)

Caving Period Reported: 1918-1950

Summary: This report updates the subsidence observations of Allen (1934), providing data on the extension of the subsidence zone for the mining carried out up to the suspension of the operations in 1951. Repeated are the data for the mining of Blocks 1 and 2, supplemented with data for the mining of the neighbouring blocks (3 and 4) and extension of Blocks 1 and 2 to 670 m depth (from 630 m). As before, mining was carried out using a combined block caving and top slicing approach. A scaled cross-section is provided showing the extent of caving at surface relative to the underground workings. Also included are surface fractures observed despite the thick blanket of glacial till that covers the bedrock. The section is based on surveys of subsidence pins laid out on a grid over the area. Present are vertical diorite dykes that partly bound and control the upward propagation of the cave.
Bagdad (Arizona, USA)

Source: Hardwick (1959)

Caving Period Reported: 1937-1944

Summary: Mining methods applied at the Bagdad mine are reported, including the block caving of the West orebody. A scaled surface subsidence map is provided showing the boundary of the subsidence area, several collapse features, and the outline of the undercut (note that only the northwest half of the undercut was caved). Based on the reported depth of the undercut (2990 Level, 265 m below surface), it is possible to estimate the caving and fracture initiation angles. No indication is given as to how the subsidence area was measured, although it is assumed that it is not at a resolution that depicts the limits of continuous subsidence but more likely fracture initiation. The source notes that the subsidence area coincides with an area where the top of the ore block is close to surface. Subsequent to this, block caving for the remaining West orebody was gradually phased out. Boundary level drifts were used to control the lateral extent of caving.
**Cambria Jackson (Michigan, USA)**

*Source:* Boyum (1961), Crane (1929)

*Caving Period Reported:* 1941-1945

*Summary:* Data is reported for a sublevel caving operation in hematite iron ore in the Marquette Iron Range of the Negaunee district. This is one of the unnamed regions reported by Crane (1929); see Lake Superior District below. The lowermost level at the time of reporting was at 350 m depth (260 level), with 50 m of mined ore above. The mining method was initially top slicing and then chanced to sublevel caving. A scaled cross-section is provided showing the original topography and the zones of caving and fracture initiation. No direct indication is given as to how the subsidence was measured, but it can assumed that surface surveys were carried out. Cave propagation occurs primarily through a thick, weak hematite iron formation but also through a competent diorite sill near surface.
Catavi (Bolivia)

*Source:* Weisz (1958)

*Caving Period Reported:* 1948-1957

*Summary:* The block caving operations for the Catavi tin mine are described. This includes a cross-section showing the original surface profile, undercut levels, and mining of Block 2 with the corresponding caved area at surface. No direct data in the form of subsidence measurements is given, but the cross section is drawn to scale. Caving is depicted as extending from the 160 level (115 m depth). The extended limits of the subsidence boundary are not included, but the source text reports that surface subsidence of approximately 60° occurred above the undercut level in the first year of block caving.

**THREE COMPONENTS** make up blocks: high grade vein reserves; old low grade fills (shaded); and former waste.
Climax – 1945 (Colorado, USA)

Source: Vanderwilt (1949)

Caving Period Reported: 1940-1945

Summary: Ground movements are reported for the Climax molybdenum mine for block caving above the Phillipson Level. The average undercut depth for this level is 145 m, with caving extending to surface. A representative cross-section is provided that reports initial (1943) and updated caving angles (1945). The cross-section is based on surface observations and mapping and shows the original topography together with a clear zone of caving/collapse from which an angle of fracture initiation can be inferred. Notably, the angle of fracture initiation appears to be affected by the sloping surface topography with subsidence extending up slope through retrogressive slumping of the upper rock scarp.
Climax – 1980 (Colorado, USA)

Source: Vera (1981)

Caving Period Reported: 1945-1980

Summary: Caving operations at Climax are described, reporting the changeover to continuous retreat panel caving from block caving of the Phillipson level as the mine moves to deeper levels. No direct data is provided in the form of subsidence measurements, but a schematic cross-section is included which shows the increase in the caving zone with the progression of mining across three levels. No scale bar is provided, but the elevations of the levels are provided from which the scale can be inferred. The panel caves are located under a sloping surface resulting in variable depths to the lowermost undercut (600 Level). With 90 m spacings between levels, the 600 Level is 180 m below the Phillipson and has an average depth of 325 m. The caving angle information provided in the cross section is significantly less detailed than the earlier data provided by the Vanderwilt, as the paper is more focussed on the general operations.

Figure 32. General Mine Layout-Schematic
Copper Mountain (B.C., Canada)

Source: Nelson & Fahrni (1950)

Caving Period Reported: 1937-1949

Summary: Subsidence data is reported for a combination of shrinkage stoping and sublevel caving of two copper porphyry ore bodies, named Contact Block and 122-East Block. In both cases, the caved ground extends to surface. The maximum undercut depths are 350 m (Contact Block) and 210 m (122-East Block) with sub-levels above. Scaled cross-sections are provided for both blocks showing the original topography, extent of subsidence, surface scars and zone of caving/collapse. From these, the angle of subsidence is reported and angles of fracture initiation and caving can be inferred. For the 122-East Block, the angle of subsidence measured from the cross-section varies with the lower angle reported being parallel to a dipping fault. The subsidence features reported in these cross-sections were based on surface surveys, mapping and aerial photographs. These were subsequently used to calculate ratios of total subsidence to ore extraction for the two blocks.
Copper Queen Branch – East Orebody (Arizona, USA)

Source: Kantner (1934)

Caving Period Reported: 1925-1933

Summary: Data is reported for block caving of several lifts of a copper porphyry deposit (East Orebody), which due to their short heights (9-36 m) and inclined nature, is more representative of a sublevel caving operation (and is classified as such in the above Tables). The initial and final undercut levels in the source paper are 140 and 285 m below the surface, respectively. Caving at surface appears in the form of measured subsidence (elevation surveys), tension cracks and several small glory holes, the latter likely reflecting the sublevel nature of the caving. A scaled cross-section is provided reporting the angle of break and angle of subsidence for each lift. However, the terms used by the author differ from the terminology used here by Van As et al. (2003). In the Discussion that follows the paper, reference is made to subsidence being the steeper angle of “marked” subsidence, with the break angle being the limit of visible cracking in the same section. These are interpreted here as referring to the caving and fracture initiation angles, respectively. Rock mass conditions influence the resulting angles, with lower angles developing in the weaker hangingwall relative to those that develop in the stronger footwall. The presence of a major fault to the north and northwest may also play a controlling role as no cracking beyond this fault was observed for a period of time.

![Diagram 1](image1.png)

![Diagram 2](image2.png)
Copper Queen Branch – Queen Hill Block (Arizona, USA)

*Source:* Trischka (1934)

*Caving Period Reported:* 1913-1933

*Summary:* Data is provided for the Queen Hill Block of the Copper Queen mining area, near but separate from the East Orebody block described above. The top slicing mining method was utilized with two main levels being caved (200 and 300 Levels) along a sub-horizontal, tabular orebody. The mining levels cross under a steep hill with depths ranging from 30 to 210 m. A cross-section is provided showing the pre-mining and “present” topography (note that no indication is given as to when mining was completed; top slicing was initiated in 1913 replacing a square set method). Reported is the fracture initiation angle, although this also corresponds to the caving angle and subsidence angle as the caved block is described as being bounded on all four sides by faults, along which the block drops and across which the subsidence disturbance is limited.

![Cross-section diagram](image-url)
Corbin

**Source:** Warburton (1936)

**Caving Period Reported:** 1917-1934

**Summary:** Data is provided for the mining of a thick coal seam (No. 6 mine) split into two sections, West and East, separated by a rock instruction (i.e. wedge of waste rock). The West block was mined by block caving using a number of lifts at 20 m intervals, the lowest of which being at 80 m at the time of reporting. The neighbouring East block was mined by top slicing, the undercut for which is inclined, dipping at 38° and extending from surface to 2 Level at 95 m depth. A schematic cross-section is provided from which the caving angles and undercut depths can be estimated. No scale is provided, but associated information is provided that allows the scale to be approximated. The lower caving angle for the East block aligns with the dip of the undercut.

![Schematic cross-section diagram](image-url)
Creighton (Ontario, Canada) – 1955

Source: Brock et al. (1956)

Caving Period Reported: 1951-1955

Summary: Data is reported for panel caving at the Creighton mine, subsequent to earlier mining by shrinkage stoping. Blasting was used to induce caving beyond limits of previously mined stopes. A schematic cross-section is provided that shows the caved stopes mined by shrinkage stoping beneath a 60 m deep open pit, and the neighbouring cave mined by panel caving along the strike of the orebody. From this cross-section, a caving angle can be estimated at the time of break through at surface, but little additional information is provided. Draw in the panel cave is limited by a cut-off grade given the dilution arising from the cap rock above hangingwall.

Fig. 7.—Diagrammatic section, first stage of panel caving.
**Creighton (Ontario, Canada) – 1963**

*Source:* Dickhout (1963)

*Caving Period Reported:* 1951-1963

**Summary:** Mining operations and ground control issues for the Creighton mine are reported. The increased time interval from that reported by Brock et al. represents a more fully developed cave, which although not specified, appears to include the pillar between the earlier shrinkage stoping operation and subsequent panel caving. No direct subsidence measurements are reported, however a general cross-section is provided that shows the development of caving relative to the different levels, from which the depth of mining can be estimated (420 m). Furthermore, in the Discussion that follows the paper, it is explained that the outline of the surface cave closely follows the outline of the completed undercut, with the exception of the hangingwall where the cave extends beyond the footprint of the undercut. A plan view map showing the outline of the limit of fracturing is provided and the angles of caving and fracture initiation relative to the undercut are specified. Strain gauge measurements are also reported with respect to specifying the angle of subsidence.
Crestmore (California, USA)

*Source:* Long and Obert (1958)

*Caving Period Reported:* 1930-1954

*Summary:* Mining operations are reported for the block caving of a dipping limestone bed. No direct data is provided in the form of subsidence measurements, but a schematic cross-section is included illustrating the mining of Block 1A. Included are the caved ground at surface and a rough outline of the fracture initiation from which an estimate of its angle is possible. The depth of the undercut is 60 meters. Vertical cutoff stopes were excavated on all four sides of the block to limit the caving angle. However, the fracture initiation angle on the footwall side of block extended beyond the cutoff stope to align with a shallower dipping fault.

![Diagram](image-url)

**FIGURE 10.** - Section A-A' Through Block 1A After Vertical Pillars Z and W Had Been Broken by Blasting.

**FIGURE 11.** - Section A-A' Through Block 1A, Showing the Initiation of Caving Action After Remnants of the Undercut Pillars Had Been Removed by Blasting.
El Teniente (Chile) – South 1 & North 4

Source: Ovalle (1981), Kvapil et al. (1989)

Caving Period Reported: 1940-1980

Summary: Mining operations at the El Teniente mine are reported for the panel caving of the South and North blocks from the Teniente 1 and 4 levels, respectively. No direct data is provided with respect to subsidence measurements, however two schematic cross-sections are included illustrating the mining of the North and South blocks. Shown are those caves already exhausted and those currently in production, together with the original and caved surface profiles. A similar cross-section for the North block is produced by Kvapil et al. but is less detailed. No scale bar is provided, but the different mine levels are shown from which the undercut depths can be estimated. Because the mine is positioned below a steep slope, with the South block being downslope of the North block, the depths to the respective undercut levels, Teniente 1 and 4, are approximately the same (510 and 540 m, respectively). The lower caving angles occur on the uphill side.
El Teniente (Chile) – Regimiento 4


*Caving Period Reported:* 1982-1998

*Summary:* A review of break angles is reported for the caved zone above El Teniente’s 4 Level, Regimiento Sector. Curves are provided for estimating break angles measured as a function of depth along the crater walls. These show that angles near the undercut are sub-vertical, gradually flattening towards surface. Minimal details are given with respect to the data the curves are based on. Similar curves for other sectors at El Teniente are reported to be based on observations of fracturing in galleries at different levels. In this case, reference is made to the use of numerical models calibrated against observations. No depth is given, but based on other sources can be estimated to be approximately 250 m deep (averaging for steep topography). From this, the caving angle at surface can be estimated from the respective curves.
El Teniente (Chile) – Esmeralda

*Source:* Rojas et al. (2001)

*Caving Period Reported:* 1997-2001

*Summary:* Panel caving operations for the Esmeralda sector are reviewed. Included is a design chart of break angles as a function of height above the undercut, based on numerical models calibrated against crater geometry data and observations of fracturing in galleries at different elevations. Separate curves are provided for the uphill and downhill sides of the cave. A cross-section is also provided that depicts the caving angle and angle of fracture initiation referred to as the “influence level”. The undercut is located under steep slope at an average depth of approximately 800 m.
**Finsch (South Africa)**

*Source:* Preece & Liebenberg (2007)

*Caving Period Reported:* 2004-2006

*Summary:* Cave management operations are reported for the block caving of Block 4 at the Finsch diamond mine. Caving of the kimberlite pipe above the undercut at 700 m depth occurs over a block height of 150 m that then opens up into the bottom of a 550 m deep open pit that had been subsequently deepened by the mining of previous blocks using open stoping techniques. A cross-section is provided, which shows the limits of the caving zone, which coincides with the angles of the already existing crater.

![Diagram of Finsch Mine](image1)

*Fig. 11.* GEMCOM generated section showing ALS profiles with outline geology and drift positions on 53 and 63 Levels showing

![Diagram of Finsch Mine](image2)

*Fig. 2* Section through Finsch Mine showing mining blocks
Gath’s (Zimbabwe)

*Source:* Brown and Ferguson (1979)

*Caving Period Reported:* 1971-1976

*Summary:* Data is provided for a sub-level shrinkage stoping operation for three different sublevels (99, 158 and 183 Levels; block caving is planned for subsequent deeper levels). Caving occurs in the hangingwall and extends to surface above the 40-50° dipping orebody. The depth of each level is variable as the topography above is steep across what appears to be a 100 m deep open pit slope. A cross-section is provided, which shows the different angles of break and surface tension cracks for each sub-level. Caving angles on the footwall side of the orebody coincide with the dip of the orebody and footwall parallel jointing.
Grace (Pennsylvania, USA)

Source: Sainsbury (2010)

Caving Period Reported: 1958-1977

Summary: Subsidence observed after the closure of the Grace iron mine is reported following panel caving of the deposit from 1958 to 1977. Reported are surface observations together with survey data based on levelling measurements of subsidence pins. A subsidence map is provided showing the limits of surface cracking and the outlines of a lake that formed in the caving zone and the relative position of the undercut. Based on the approximate depth of the undercut, caving and fracture initiation angles can be estimated. It should be noted that cave breakthrough only occurred above one half of the undercut, facilitated by a steeply dipping fault.

Figure 7  Limit of large-scale surface cracking
Grangesberg (Sweden) - 1961

**Source:** Hoek (1974)

**Caving Period Reported:** 1921-1961

**Summary:** Hangingwall failures induced by sub-level caving are reported for six different sublevels (from 140 to 300 m depth) during the mining of an iron ore deposit. The orebody is approximately 54 m thick dipping at 64°. Data is provided in the form of a simplified cross-section showing the caving angle for each sublevel. Few additional details are provided, with references pointing to a Swedish report as the original source. It was observed that the deeper the caving, the lower the caving angle on the hangingwall side. The angle of caving on the footwall side is shown to be constant (with depth), coincident with the contact between the orebody and host rock.

![Diagram of hanging-wall failure induced by caving at Grangesborg](image)

**Fig. 8** Hanging-wall failure induced by caving at Grangesborg
Grangesberg (Sweden) - 1974

Source: Sisselman (1974)

Caving Period Reported: 1960-1975

Summary: Mining operations at the Grangesberg iron ore mine are described, reporting the use of both sublevel and block caving methods depending on the ore thickness. Approximately 70% of the mining is by block caving. A cross-section is provided showing the extent of the caving zone relative to the undercut level at 410 m depth. The orebody is approximately 54 m thick dipping at 64°. No details are provided with respect to the measurement of surface subsidence. The angle of caving on the footwall side is shown to be coincident with the contact between the orebody and host rock.

(Left) Cross-section of continuous block caving. New method will greatly reduce risk of accident when ore is loaded, transported and when handling rock hang-ups.
Grasberg (Indonesia) – IOZ

Source: Hubert et al. (2000), Barber et al. (2001)

Caving Period Reported: 1980-2000

Summary: Block caving of P.T. Freeport’s Ertsberg East Skarn System is reported, including the operations for the Gunung Bijih Timur (GBT), Intermediate Ore Zone (IOZ) and Deep Ore Zone (DOZ) caving sectors. Focus is given to the IOZ. No direct data is provided in the form of subsidence measurements in either source, but scaled cross-sections are included showing the development of caving above the IOZ. Included are the boundaries of the caving zone relative to the undercut level at 650 m depth. The cross-section in Hubert et al. also includes the limits of fracture initiation referred to as the “subsidence zone”. The IOZ undercut is located under a steep slope, for which the lower caving angle corresponds with the uphill side.

Fig 2 - Schematic long section of GBT, IOZ and DOZ.
Havelock (Swaziland)

Source: Heslop (1974)

Caving Period Reported: 1952-1966

Summary: Data is reported for three levels of a shrinkage stoping and sublevel caving asbestos operation. The lowermost undercut is at approximately 225 m depth (no direct information for depth is given; estimates can be made based on a scaled cross-section). A cross-section is provided, which shows the different angles of caving and extent of flexural toppling and surface fracturing above the hangingwall for different periods of time corresponding to the development of the different levels. Caving angles on the hangingwall side are seen to remain constant, while the zone of fracture initiation increases. Caving angles on the footwall footwall side are likewise shown as being constant and aligned parallel to the dip of the foliation and bedding.
Henderson (Colorado, USA) – 8100

**Source:** Brumleve & Maier (1981), Stewart (1984)

**Caving Period Reported:** 1976-1983

**Summary:** Subsidence at Henderson is reported by Stewart for panel caving of the 8100 Level along Panel 1. The cave zone is reported to have appeared on surface four years after caving was initiated, with cave growth and subsidence being measured using aerial photography, surface surveys and TDR. Data is provided in the form of a block diagram and subsidence maps showing the outline of the caving zone on surface relative to the undercut at 1050 m depth. A cross-section showing the caving angles extended from the undercut is provided by Brumleve & Maier in their description of the rock mass response to panel caving. These were used to estimate the angles of caving. Vertically spaced boundary cutoff drifts together with steeply dipping faults contributed to the vertical nature of the cave that developed. The caving zone was observed to not change in its direction despite the advance of the caveline, likely due to the controlling influence of topography and faulting.
**Henderson (Colorado, USA) – 7700 Level**

*Source:* Rech et al. (2000)

*Caving Period Reported:* 1976-2000

**Summary:** An update on the panel caving operations at Henerson is reported, describing the mining of the 7700 Level following the depletion of the 8100 Level. The 7700 Level is 100 m below the 8100 Level at approximately 1150 m depth. No direct data is provided in the form of subsidence measurements, but a schematic cross-section is included showing the boundaries of the caving zone above the 7700 Level undercut. No scale bar is provided, but the different levels are shown from which the scale can be calculated. It is assumed the section is roughly drawn to scale. Vertically spaced boundary cutoff drifts together with steeply dipping faults contribute to the vertical nature of the cave that developed.

![Diagram of the Henderson Mine and Mill](image)

**Fig 1 - General cross-section of the Henderson Mine and Mill.**
Inspiration (Arizona, USA)

*Source:* Hardwick (1963)

*Caving Period Reported:* 1954

*Summary:* The history of mining operations at Inspiration is reviewed; the Inspiration mine is adjacent to the Miami block cave mine. The report includes details on the transition from block caving to open pit mining in 1954. Specifically, the block caving of a transfer block is reported from an undercut 70 m below the pit bottom. Data is provided in the form of a scaled cross-section showing the outlines of the caving zone over time, from which the caving angles can be calculated.

![Section Through Transfer Block](image-url)
Jagersfontein (South Africa)

*Source:* Stucke (1965)

*Caving Period Reported:* 1947-1962

**Summary:** Plans to block cave a new lift at the Jagersfontein diamond mine are reported. The description includes a schematic cross-section showing the current block caving undercut level at 550 m depth and earlier workings, including an older open pit operation. The cross section shows that the limits of the caving zone roughly coincide with the boundaries of the kimberlite pipe and already existing crater. It is reported that approximately one million tonnes of waste rock from the crater walls slough into the crater each year.

![Diagram of Jagersfontein Mine](image-url)
**Jeni (California, USA)**

**Source:** Obert & Long (1962)

**Caving Period Reported:** 1952-1957

**Summary:** Data is reported for a single block cave experiment in a thick, sub-horizontal borate deposit, where previous mining was by room and pillar. The undercut level is at a depth of 160 m with a block height of 70 m. Caving of the orebody and overlying cap rocks extended to surface, although longhole blasting at intermediate depths was required to aid caving of the ore. A scaled cross-section is provided showing a clear zone of caving/collapse, and a plan view map is provided showing subsidence contours (with 10' contour intervals). Both show that the caved ground propagated slightly to the southeast of the undercut area. One of the unique features of the caving zone was that its periphery was marked by a single, continuous, steep-wall face. Little to no change in the peripheral outline of the subsided area occurred over the 4.5 year period following the initial subsidence.
King (Zimbabwe)


*Caving Period Reported:* None given (<1983 – 1988)

*Summary:* Block caving of the West Flank (W11-14 blocks) of the King asbestos mine is described. Caving of the steeply dipping orebody was initiated below the foot of a steep hill and extended towards its 250 m high peak; the corresponding average depth of the undercut is 275 m. Schematic cross-sections are provided in both sources showing the extent of caving on surface relative to the undercut on the 276 Level. Wilson’s section shows more detail, including lines connecting the undercut to surface fractures (fracture initiation angle) but for an earlier stage of cave development. Neither cross-section includes a scale bar. Caving angles on the footwall side of the orebody are shown to coincide with the dip of the orebody.

![Diagram 1](image1.png)

![Diagram 2](image2.png)

*Fig. 3* North-south section through west flank showing cave, production levels 192, 241 and 276 and haulage level 296. (Line of section shown in Fig. 1)
Kiirunavaara/Kiruna (Sweden) - 1995


Caving Period Reported: 1965-1995

Summary: Analysis of the progressive failure of the hangingwall and footwall at the Kiirunavaara iron ore sublevel cave mine is reported. The source papers briefly describe the history of hangingwall failures above the sublevel caving, and the more recent failure of the footwall (previous caving angles had coincided with the footwall contact of the orebody dipping at 60°). Lupo provides an air photo outlining the zones of caving, surface cracking and subsidence. The exact depth of the sublevel undercut for these zones is not reported but can be estimated as approximately 560 m depth. Henry & Dahnér-Lindqvist provide a scaled cross section for the same footwall failure event from which the undercut level and caving angle can be estimated. The caving angle on the footwall side coincides with the dip of the orebody, whereas the higher caving angle results from the overhanging nature of the dipping hangingwall. Values for the caving, fracture initiation and subsidence angles from these sources is also reviewed.
Kiirunavaara/Kiruna (Sweden) - 2000

Source: Henry et al. (2004)

Caving Period Reported: 1965-2000

Summary: The application of InSAR monitoring of mining-induced deformations is reported for the Kiirunavaara iron ore sublevel cave mine. The source largely focuses on InSAR principles, but includes a subsidence map of the caving, fracture initiation and subsidence zones. These are based on surface geodetic and benchmark surveys. The outline of the lowermost sublevel undercut for the measurement period (500 m depth) is not provided on the map, but can be approximated, from which the respective angles can be calculated. Caving, fracture initiation and subsidence angles on the footwall side are shown to coincide with one another at approximately 50°.
Kiirunavaara/Kiruna (Sweden) - 2005

*Source*: Villegas (2008)

*Caving Period Reported*: 1965-2005

*Summary*: A thesis study is reported involving the numerical analysis of the hangingwall at the Kiirunavaara iron ore sublevel cave mine. A detailed description of the modelling input is provided (rock mass characteristics, properties and in situ stresses), together with the modelling results. A simplified cross-section is also provided as a form of model constraint showing the extent of the farthest surface crack observed on the hangingwall side of the orebody for several different time periods. These are interpreted as representing the limits of the fracture initiation zone. Although exact undercut depths for each sublevel are not provided, they can be approximated from the cross-section. No indication is given as to the caving angles for the same period, or the caving and fracture initiation angles on the footwall side.

![Figure 2: Cross section Y1500 (view from the north)]](image)
Koffiefontein (South Africa)

Source: Hannweg (2001)

Caving Period Reported: 1987-2001

Summary: Caving operations at the Koffiefontein diamond mine are reported, describing the use of a front caving method that combines aspects of block and sub-level caving. The undercut occurs at 480 m depth and caves into previous underground workings and the bottom of a deep open pit. This is shown in a schematic cross-section, from which the caving angles can be estimated. These are shown as being sub-vertical and confined within the limits of the pit bottom.
Lake Superior District (Michigan, USA)

Source: Crane (1929)

Caving Period Reported: none

Summary: Data is presented for several cases involving copper and iron mines, but without specific reference to the mine location, mining period or mining method. Based on the “type cases” reported, most appear to involve open stoping where failure (caving) of the hangingwall has occurred, although indication is given that a small number of sublevel caving cases are also included. Angles of break are reported, but these are defined as the angles observed underground with reference to the backs of hangingwall failures and not necessarily the angles representing the extension of caving to surface. Cross-sections are provided for two cases, showing examples of caving and subsidence at surface, which appear to be sublevel or block caving mines. Both are without scale bars. The first is described as Case B (also Type C), involving an iron-bearing formation lying on highly inclined slate. The angle of caving on the footwall side aligns with the dip of the slates. No depth is given for the mining depth. Reference is made to 300-400 m as a minimum, although references are also made to multiple depths and sub-levels. The second case (Case C, Types D,E,F), appears to refer to the block caving of lenticular iron ore deposits. A cross section is provided showing the caving angle extending from the undercut, but again, no scale is provided. Other information provided suggests a depth of 450 m, including a thick blanket of glacial till that partly obscures the caving and subsidence zones at surface.
Malmberget (Sweden)

*Source:* Haglund & Heberg (1975)

*Caving Period Reported:* 1970-1974

**Summary:** Recovery of a 160 m high pillar is described using a modified block caving method referred to as ‘slotblocking’. However, the dipping nature of the orebody and multiple use of sublevels, more closely resembles a sublevel caving approach and is classifies as such in the Tables above. Previous and subsequent mining of the iron ore deposits involved a combination of shrinkage and sublevel stoping and sublevel and block caving. No direct data is provided in the form of subsidence measurements, but a scaled cross-section is included showing the boundaries of the caving zone on surface relative to several undercut levels below. The most relevant is the block caving of the 300 Level (300 m depth), with lower levels acting more like a sublevel caving operation. The cross-section shows that the caving zone on the footwall side aligns with the orebody contact, whereas the overhanging hangingwall collapses into the cave.
Miami – 1928 (Arizona, USA)

Source: Maclennan (1929)

Caving Period Reported: 1926-1928

Summary: The source reports data from the block caving of two ore bodies, referred to as Main and Low Grade, together with a smaller block named Stope 11. Earlier mining was by shrinkage stoping and top slicing. At the time of reporting, mining of the Low Grade orebody was still in progress. The depths of the three undercuts vary from 180-195 m for ore blocks 65 to 120 m high. Boundary caving drifts were driven at suitable vertical intervals to limit the amount of caving beyond them. The south boundary of the Low Grade block caves into the already caved Main block. A plan view subsidence map and several cross-sections are provided showing the extent of measured subsidence on surface relative to the undercut level, including limiting scarps from which the angle of fracture initiation can be estimated. Caving angles and angles for fracture initiation are reported in the paper, but are measured from the top of the mined block (i.e. ore column); angles reported here, in the Tables above, have been corrected to be measured from the extraction level. Lower caving/fracture initiation angles for the Main and Low Grade blocks were observed to be sub-parallel to foliation of schist.
Scale: 1 in. = 118 ft.

Fig. 2.—Sections through orebody showing stoped areas and resulting surface subsidence.

Scale: 1 in. = 118 ft.

Fig. 3.—Sections through orebody showing stoped areas and resulting surface subsidence.
Miami – 1958 (Arizona, USA)

**Source:** Fletcher (1960)

**Caving Period Reported:** 1910-1958

**Summary:** This report updates the subsidence observations of MacLennan (1929) with an intermediate set of observations from 1939 and current measurements as of 1958. The data for 1939 involves a plan view map showing the limits of caving and fracture initiation relative to the caved ore bodies. This period continues the mining of the low grade orebody, as reported in 1929, from the same mining depth (approximately 195 m). As before, boundary caving drifts were driven at different vertical intervals to limit the amount of caving beyond them. Fletcher (1960) reports that the caving angles have flattened considerably since the 1929 set of measurements. The data reported for the mining period up to 1959 incorporates an extension of the High Grade orebody to greater depths through lifts at the 700 and 1000 (foot) Levels, with a bottom undercut at 300 m depth. Unlike previous blocks, those undercut at the 1000 Level were done so without the vertical boundary cutoff drifts. Scaled cross-sections are provided for the deeper lifts, showing the original topography, extent of subsidence, and surface scarps. It should be noted that although a large fault (the Miami fault) cuts across the deposit, separating the schist-hosted orebody from the conglomerate cap, it is not seen to have any influence on the caving or fracture initiation limits.
FIG. 8 E 800 section, No. 2 coordinates, showing stope areas and subsidence, Feb. 14, 1958.
Mt. Lyell – Cave Horn (Tasmania, Australia)

Source: North & Callaghan (1980)

Caving Period Reported: 1972-1980

Summary: Subsidence related to the mining of several different orebodies is reported at Mt. Lyell, including the Cape Horn orebody mined by a combination of sublevel caving and stoping. A schematic cross-section is provided that shows the extent of caving above the #5 undercut at 160 m depth, together with the angle of fracture initiation above the hangingwall. The latter is described as a concentric pattern of surface cracking. Caving on the footwall side is shown to coincide with the dip of the orebody (70°).

Fig. 4 - Cross section of the Cape Horn mine
Northparkes (Australia) – E26 Lift 1

Source: Duffield (2000)

Caving Period Reported: 1993-2000

Summary: The design of the second lift for the Northparkes’ E26 mine is reported. Included in this description is a geological cross-section showing the outline of the mined out Lift 1 block cave. No direct data is provided in the form of subsidence measurements, but caving angles for Lift 1 can be approximated from the cross-section supplemented by subsidence-related information in the source paper. Caving of Lift 1 involved the collapse of the crown pillar into an air gap beneath the cave back, owing in part to a change in the geology related to a gypsum leached zone. As a result, the cave angles are near vertical. The lift also caved into the bottom of a small open pit; the caving zone measured does not include ground disturbed due solely to open pit mining.
**Palabora (South Africa)**

*Source:* Pretorius (2007)

*Caving Period Reported:* 2001-2007

*Summary:* The effects of dilution resulting from a 130 million ton pit wall failure above an active block cave are reported. The block cave undercut is approximately 400 m below the bottom of the 800 m deep pit at a depth of 1200 m. The source reports the caving angles originally projected to open up into the floor of the pit, and the unexpected caving-induced triggering of a large pit wall failure. Physical and numerical modeling predicted a loss of around 30% of the original ore reserve. The source paper reports the caving angle at 86-88 degrees. The fracture initiation angle is projected with respect to the location of the back scarp of the rockslide behind the crest on the north wall of the pit.

*Figure 8: Ore Loss effects on the Palabora reserve from external dilution:*

1 – Dilution entry, 2 – Replacement of ore by waste
Perseverance (WA, Australia) - 2000

Source: Jarosz et al. (2007)

Caving Period Reported: 1994-2000

Summary: A report is provided on the use of InSAR to measure mining-induced deformations above a sublevel caving operation. Included is a cross-section that shows the caving profile above the sublevel undercut at 640 m depth (9920 Level). These underground operations are located below a large open pit with caving on the hangingwall side extending beyond the pit. InSAR data is also presented, however, no indication is given as to how the monitored displacements relate spatially to the undercut sublevels.
Perseverance (Australia) - 2004

*Source:* Tyler et al. (2004)

*Caving Period Reported:* 1994-2004

*Summary:* Subsidence above the Perseverance sub-level caving nickel mine is reported. Subsidence maps are provided for several different years and mining levels, showing the limits of caving and fracture initiation based on the interpretation of air photographs, walk-over surveys and GPS/prism data. Caving and fracture initiation angles calculated based on stated mining depths and outlines of the caving levels relative to the outlines of caving on surface. Caving occurs beneath a large open pit with caving on the hangingwall side extending beyond the pit limits.
**Questa (New Mexico, USA)**

*Source:* Gilbride (2005)

*Caving Period Reported:* 1979-2005

**Summary:** Subsidence at the Questa mine related to historic block caving (Goathill orebody) and block caving of a new orebody (“D”) is reported. Data is provided for the measured historic subsidence over the Goathill orebody in the form of an air photo outlining the caving and fracture initiation zones, and limits of ground deformation (i.e. continuous subsidence zone). The respective angles are reported with reference to the undercut level at 300 m depth. The limits depicted are based on field measurements and air photo analysis. Incorporated in the subsidence zone is a shallow-seated slide undercut at its toe by the caving zone. A subsidence contour map is provided for the ground deformations over the D orebody, Panel 1 undercut (600 m depth). Deformations were measured by surface surveys across a grid. At the time of reporting, subsidence over Panel 1 was not developed enough to allow the measurement of the caving or fracture initiation angles. The subsidence magnitudes, however, were large enough to initiate large-scale sliding of the hillside above. Subsidence above the D Orebody was first detected in April 2003, 30 months after caving was initiated. Caving propagated to surface through 550 m of overburden at an average rate of 0.21 m per day.
Rajpura Dariba (India)

*Source:* Singh (1993)

*Caving Period Reported:* ?

*Summary:* Numerical modelling of progressive hangingwall failure is reported, including a brief description of a case history of the Rajpura Dariba sublevel caving mine. No direct data is reported but the caving and fracture initiation angles on the hangingwall side of the orebody are reported together with the mining depth. The respective angles on the footwall side are assumed to be aligned with the 70° dip of the footwall.
Salvador (Chile)

Source: Escobar & Tapia (2000)

Caving Period Reported: 1995-1999

Summary: The investigation into the 1999 air blast event at the Salvador panel cave operation is reported. This involved the sudden collapse of the cave back in the Inca West area after a stable arch had formed resulting in the development of a large void. No direct data is presented in the form of subsidence measurements, but a scaled cross-section showing the development of the collapse is provided which includes the outline of the caving zone and original topography relative to the undercut level (at 700 m depth).
San Giovanni (Italy)

*Source:* Balia et al. (1990)

*Caving Period Reported:* 1985-1990

*Summary:* Analysis of progressive hangingwall failure is reported for the San Giovanni lead-zinc mine. The mining methods employed include cut and fill stoping in the lower levels, and sub-level caving and shrinkage stoping in the upper levels. The undercut depth for the lowermost sublevel caving level is at 100 m depth. Data is provided in the form of a longitudinal cross-section and subsidence map outlining the caving zone. The caving angle calculated for the footwall side of the orebody approximately coincides with the dip of the footwall at 75-80°.
San Manuel (Arizona, USA)

Source: Buchanan & Buchella (1960); Johnson & Soulé (1963)

Caving Period Reported: 1956-1960

Summary: Data is reported for the block caving of the South orebody, Lift 1 (1450 Level undercut) from two different sources. The South ore body is the largest of the three ore bodies at San Manuel. The data from Johnson & Soulé (1963) provides more detail at a higher resolution and is the primary source used here. The undercut for Lift 1 is at a depth of 420 m with a block height of 180 m. At the time of reporting, only the central third of the ore zone has been caved. Surface subsidence data is provided in the form of a cross-section showing the subsidence profile for different intermediate stages of caving, several cross-sections showing the caving angle and angle of subsidence for different profiles above the undercut, and a subsidence contour map (with 25’ contour intervals) showing the limits of scarp development and surface cracking. Note that the definitions from Kantner (1934) are used with angle of subsidence referring to the limits of caving on surface and break angle being applied to the limit of visible cracking in the same section. These are interpreted here as referring to the caving and fracture initiation angles, respectively. A general recommendation was provided to assume a setback distance of 230 m of lateral distance on surface for each 300 m of depth mined in order to protect structures and ensure safety.
San Manuel (Arizona, USA)

Source: Thomas (1971)

Caving Period Reported: 1956-1970

Summary: Data is provided summarizing that previously reported for the South orebody, Lift 1 (Johnson & Soule 1963), subsequent data for its completion in mid-1962, data for the mining of Lift 2 (2015 Level), and data for the mining of the North orebody, Lift 1, which is comprised of two blocks, West and East, separated by a 200 m pillar. The undercut for Lift 1 (1450 Level) was at 420 m depth, whereas the undercut for Lift 2 (2015 Level) was at 605 m depth. Lift 2 for the South orebody was mined by block caving, although the method was gradually modified to follow a panel caving type sequencing. A similar panel caving approach was applied to the North orebody, although the small size of the West and East undercuts effectively resulted in block being caved. Surface subsidence data is provided in the form of several subsidence contour maps (with 50’ contour intervals) showing the extension of the caving zone with time, together with scaled cross-sections showing the subsidence profiles. Active subsidence for the South orebody, Lift 2, is primarily contained within well established boundaries for Lift 2, with only minor activity outside this periphery. The increased depth of Lift 2 therefore results in a steepening of the caving angles. The caving zones for the North orebody, West and East, remained separate.
Shabani (Zimbabwe)


Caving Period Reported: - 1999

Summary: Block caving and ground support practices at the Shabani asbestos mine are reported. Caving involves an inclined undercut dipping at an angle of approximately 30° from horizontal, with blocks being developed to target discrete, elongated pods of ore. No direct data is provided in the form of subsidence measurements, but a scaled cross-section is included that outlines the caving zone above two mined blocks (52 and 58) relative to the undercut level. An average depth of 630 m is shown for the dipping undercut.

Fig 2 - Generalised cross-section through Shabanie mine.
Urad (Colorado, USA)

Source: Kendrik (1970)

Caving Period Reported: 1967-1969

Summary: Data is reported for panel caving from the 1100 Level of the Urad deposit. The height of the ore varies from 60 to 210 m and the overall depth varies from 120 to 300 m due to its position under a steep slope. Considerable pre-splitting and induction blasting was required to aid the caving process. A schematic drawing is provided from which the caving angle can be estimated assuming the sketch is to scale. Vertical boundary cut-off stopes were mined (shrinkage stoping) to limit the extent of the caving zone.
Appendix C: Depth-Dependent Material Properties used for Benchmark Testing of Numerical Methods for Modelling the Influence of Undercut Depth on Caving-Induced Subsidence

The following tables provide in detail the input properties used in the benchmark study, varied as a function of depth. Values used for the FLAC3D modelling were similar except for the cohesion, friction and tension values which were scaled to 75% of those used in the 2-D modelling.

Table 1. Mohr-Coulomb rock mass properties derived for the 500 m deep undercut.

<table>
<thead>
<tr>
<th>Lithology</th>
<th>$\sigma_{3\text{max}}$ (MPa)</th>
<th>$E_{\text{rm}}$ (GPa)</th>
<th>$c_{\text{rm}}$ (MPa)</th>
<th>$\phi_{\text{rm}}$ (°)</th>
<th>$T_{\text{rm}}$ (MPa)</th>
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<tbody>
<tr>
<td>Rhyolite</td>
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<td>south of South fault</td>
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<td>9.1</td>
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Table 2. Mohr-Coulomb rock mass properties derived for the 1000 m deep undercut.

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<th>$c_{\text{rm}}$ (MPa)</th>
<th>$\phi_{\text{rm}}$ (°)</th>
<th>$T_{\text{rm}}$ (MPa)</th>
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Table 3. Mohr-Coulomb rock mass properties derived for the 1500 m deep undercut.

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Table 4. Mohr-Coulomb rock mass properties derived for the 2000 m deep undercut.

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<td>Biotite Granodiorite north of North fault south of South fault</td>
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<td>7.7</td>
<td>8.2</td>
<td>33</td>
<td>2.3</td>
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</table>

The following tables provide in detail the input properties used in the benchmark study, including their variation with depth to account for increased strength with increasing confinement.
Appendix D: Representative Input Source Codes Used for Numerical Modelling Benchmark Study
D.1 ELFEN Benchmark: 500m Deep Undercut

# Material database  = cemi
# Material selected  = Sandstone and Siltstone-south
Material_data { 4
  Material_name {
    "Sandstone and Siltstone-south"
  }
  Elastic_material_flags { NFGELA { 4 }
    0 1 0 0
  }
  Elastic_properties { NMPRP { 15 }
    1.02e+010 0 0 0.25 0 0 0 0 0 0 0 0 2700 0 0
  }
  Number_state_variables {
    12
  }
}

# Material database  = cemi
# Material selected  = Quartz-Monsodiorite-Undercut
Material_data { 5
  Material_name {
    "Quartz-Monsodiorite-Undercut"
  }
  Elastic_material_flags { NFGELA { 4 }
    0 1 0 0
  }
  Elastic_properties { NMPRP { 15 }
    7e+009 0 0 0.25 0 0 0 0 0 0 0 0 2700 0 0
  }
  Number_state_variables {
    12
  }
}

# Material database  = cemi
# Material selected  = Biotite Granodiorite-north
Material_data { 6
  Material_name {
    "Biotite Granodiorite-north"
  }
  Elastic_material_flags { NFGELA { 4 }

0 1 0 0
}
Elastic_properties { NMPRP { 15 }
7.7e+009 0 0 0.25 0 0 0 0 0 0 2700 0 0
}
Number_state_variables {
  12
}
#
# Material database = cemi
# Material selected = Rhyolite
Material_data { 7
  Material_name {
    "Rhyolite"
  }
  Elastic_material_flags { NFGELA { 4 }
    0 1 0 0
  }
  Elastic_properties { NMPRP { 15 }
    1.26e+010 0 0 0.25 0 0 0 0 0 0 0 0 2700 0 0
  }
  Number_state_variables {
    12
  }
}
#
# Material database = cemi
# Material selected = Quartz-Monsodiorite-south
# Non linear Criterion = mc_rc
# Non linear prop 1=cohesion
# Non linear prop 2=friction_angle
# Non linear prop 3=dilation
# Non linear prop 4=tensile
# Non linear prop 5=gf
Material_data { 8
  Material_name {
    "Quartz-Monsodiorite-south"
  }
  Elastic_material_flags { NFGELA { 4 }
    0 1 0 0
  }
  Elastic_properties { NMPRP { 15 }
    7.0e+009 0 0 0.25 0 0 0 0 0 0 0 0 2700 0 0
  }
}
Material database = cemi
Material selected = Biotite Granodiorite-north
Non linear Criterion = mc_rc
# Non linear prop 1=cohesion
# Non linear prop 2=friction_angle
# Non linear prop 3=dilation
# Non linear prop 4=tensile
# Non linear prop 5=gf
Material_data { 9
  Material_name {
    "Biotite Granodiorite-north"
  }
  Elastic_material_flags { NFGELA { 4 }
    0 1 0 0
  }
  Elastic_properties { NMPRP { 15 }
    7.7e+009 0 0 0.25 0 0 0 0 0 0 0 0 0 2700 0 0
  }
  Plastic_material_flags { NMFPLS { 2 }
    0 19
  }
  Plastic_properties { NPRPLS { 5 }
    1.9e6 49 5 0.35e6 63
  }
  Failure_material_flags { 3
    0 2 1
  }
  Failure_properties { 1
    0.002
  }
  Fracturing_material_flags { 2
    0 1
  }
  Fracturing_properties { 1
    0
  }
  Number_state_variables {
    12
  }
}
0 19
}
Plastic_properties { NPRPLS { 5 }
3.2e6 44 5 0.7e6 63
}
Failure_material_flags { 3
0 2 1
}
Failure_properties { 1
0.002
}
Fracturing_material_flags { 2
0 1
}
Fracturing_properties { 1
0
}
Number_state_variables {
12
}
}
# Material database  = cemi
# Material selected  = Rhyolite
# Non linear Criterion = mc_rc
#  Non linear prop 1=cohesion
#  Non linear prop 2=friction_angle
#  Non linear prop 3=dilation
#  Non linear prop 4=tensile
#  Non linear prop 5=gf
Material_data { 10
Material_name {
"Rhyolite"
}
Elastic_material_flags { NFGELA { 4 }
0 1 0 0
}
Elastic_properties { NMPRP { 15 }
12.6e+009 0 0 0.25 0 0 0 0 0 0 0 0 0 2700 0 0
}
Plastic_material_flags { NMFPLS { 2 }
0 19
}
Plastic_properties { NPRPLS { 5 } 
3.3e6 52 5 0.55e6 63 
} 
Failure_material_flags { 3 
0 2 1 
} 
Failure_properties { 1 
0.002 
} 
Fracturing_material_flags { 2 
0 1 
} 
Fracturing_properties { 1 
0 
} 
Number_state_variables { 
12 
} 
# Material database = cemi 
# Material selected = Sandstone and Siltstone-undercut 
# Non linear Criterion = mc_rc 
# Non linear prop 1=cohesion 
# Non linear prop 2=friction_angle 
# Non linear prop 3=dilation 
# Non linear prop 4=tensile 
# Non linear prop 5=gf 
Material_data { 2 
Material_name { 
"Sandstone and Siltstone-undercut" 
} 
Elastic_material_flags { NFGELA { 4 } 
0 1 0 0 
} 
Elastic_properties { NMPRP { 15 } 
10.2e+009 0 0 0.25 0 0 0 0 0 0 0 0 2700 0 0 
} 
Plastic_material_flags { NMFPLS { 2 } 
0 19 
} 
Plastic_properties { NPRPLS { 5 } 
9.1e6 32 5 2.5e6 63 
}
Failure_material_flags { 3
    0 2 1
}
Failure_properties { 1
    0.002
}
Fracturing_material_flags { 2
    0 1
}
Fracturing_properties { 1
    0
}
Number_state_variables {
    12
}

# Material database    = cemi
# Material selected    = Quartz-Monsodiorite-south
Material_data { 3
    Material_name {
        "Quartz-Monsodiorite-south"
    }
    Elastic_material_flags { NFGELA { 4 }
        0 1 0 0
    }
    Elastic_properties { NMPRP { 15 }
        7e+009 0 0 0.25 0 0 0 0 0 0 0 0 2700 0 0
    }
    Number_state_variables {
        12
    }
}
new

ro 0.5
set edge 5
; config cell 2100 840

;; commands for build model geometry
;-------------------------------------------------------------
block 0 -700 0 3500 10500 3500 10500 -700
; cal readprop.fis
;---------------------------------------------------

;; define regions for generate joints, generate density and assign properties,
;;-------------------------------------------------------------
jreg id 1 0,3500 4485,3500 4338.386,2750.876 0,2750.876;granodiorite
jreg id 11 0,2750.876 4338.386,2750.876 4172.096,1901.222 0,1901.222
jreg id 12 0,1901.222 4172.096,1901.222 3800,-700 0,-700
jreg id 2 4485,3500 6180,3500 6180,2345 4338.386,2750.876;rhyolite between faults
jreg id 3 4338.386,2750.876 6180,2345 6180,1500 4172.096,1901.222;qtz monzodiorite
jreg id 4 4172.096,1901.222 6180,1500 6180,-700 3800,-700;sanstone/siltstone
jreg id 5 6180,3500 10500,3500 10500,2635 6180,2635;qtz monzodiorite right of fault
jreg id 6 6180,2635 10500,2635 10500,2030 6180,2030;sandstone/siltstone
jreg id 7 6180,2030 10500,2030 10500,-700 6180,-700;granodiorite
;------------------------------------------------------

; faults
crack 4485 3500 3800 -700
crack 6180 3500 6180 -700

; jointing
jset 0,0 5000,0 0,0 100,0 range jreg 12
jset 90,0 5000,0 0,0 100,0 range jreg 12
jset 0,0 5000,0 0,0 100,0 range jreg 4
jset 90,0 5000,0 0,0 100,0 range jreg 4
jset 0,0 5000,0 0,0 100,0 range jreg 7
jset 90,0 5000,0 0,0 100,0 range jreg 7
jset 0,0 2000,0 0,0 50,0 range jreg 11
jset 90,0 2000,0 0,0 50,0 range jreg 11
jset 0,0 2000,0 0,0 50,0 range jreg 3
jset 90,0 2000,0 0,0 50,0 range jreg 3
jset 0,0 2000,0 0,0 50,0 range jreg 6
jset 90,0 2000,0 0,0 50,0 range jreg 6
jset 0,0 2000,0 0,0 50,0 range jreg 1
jset 90,0 2000,0 0,0 50,0 range jreg 1
jset 0,0 2000,0 0,0 50,0 range jreg 2
jset 90,0 2000,0 0,0 50,0 range jreg 2
jset 0,0 2000,0 0,0 50,0 range jreg 5
jset 90,0 2000,0 0,0 50,0 range jreg 5

;elastic subsidence zone
jreg id 17 2500,3300 4450,3300 4450,3000 2500,3000
jset 0,0 5000,0 0,0 25,0 range jreg 17
jset 90,0 1000,0 0,0 25,0 range jreg 17
jreg id 18 6050,3300 6180,3300 6180,3000 6050,3000
jset 0,0 5000,0 0,0 25,0 range jreg 18
jset 90,0 1000,0 0,0 25,0 range jreg 18
jreg id 19 6180,3300 8000,3300 8000,3000 6180,3000
jset 0,0 5000,0 0,0 25,0 range jreg 19
jset 90,0 1000,0 0,0 25,0 range jreg 19
jreg id 20 0,3300 2500,3300 2500,3500 0,3500
jset 0,0 5000,0 0,0 25,0 range jreg 20
jset 90,0 1000,0 0,0 25,0 range jreg 20
jreg id 21 8000,3300 10500,3300 10500,3500 8000,3500
jset 0,0 5000,0 0,0 25,0 range jreg 21
jset 90,0 1000,0 0,0 25,0 range jreg 21

; plastic subsidence zone
jreg id 14 2500,3300 4435,3300 4485,3500 2500,3500
jset 0,0 5000,0 0,0 12.5,0 range jreg 14
jset 90,0 1000,0 0,0 12.5,0 range jreg 14
jreg id 15 4435,3300 6180,3300 6180,3500 4485,3500
jset 0,0 5000,0 0,0 12.5,0 range jreg 15
jset 90,0 1000,0 0,0 12.5,0 range jreg 15
jreg id 16 6180,3300 8000,3300 8000,3500 6180,3500
jset 0,0 5000,0 0,0 12.5,0 range jreg 16
jset 90,0 1000,0 0,0 12.5,0 range jreg 16

; ore column
jreg id 13 4450,2900 6050,2900 6050,3300 4450,3300
jset 0,0 2000,0 0,0 12.5,0 range jreg 13
jset 90,0 500,0 0,0 12.5,0 range jreg 13
del area 2
jdelete
save orth-500-geom.sav
rest orth-500-geom.sav

;---------------------------------; change rigid block into deformable
gen edge 50 range jreg 17
gen edge 50 range jreg 18
gen edge 50 range jreg 19
gen edge 50 range jreg 20
gen edge 50 range jreg 21
gen edge 50 range jreg 13
gen edge 50 range jreg 14
gen edge 50 range jreg 15
gen edge 50 range jreg 16

gen edge 100

save orth-500-mesh.sav
rest orth-500-mesh.sav

;----------------------------------

;assign properties

; assign rock properties to different regions

change mat 4 cons 1 range jreg 1
change mat 4 cons 1 range jreg 11
change mat 4 cons 1 range jreg 12
change mat 4 cons 1 range reg 2500,3300 4435,3300 4485,3500 2500,3500 ; plastic
change mat 4 cons 1 range reg 2500,3300 4450,3300 4450,3000 2500,3000

change mat 1 cons 1 range jreg 2
change mat 1 cons 3 range reg 4435,3300 6180,3300 6180,3500 4485,3500 ; plastic
change mat 1 cons 3 range reg 6050,3300 6180,3300 6180,3000 6050,3000

change mat 2 cons 1 range jreg 3
change mat 3 cons 1 range jreg 4

change mat 5 cons 1 range jreg 5
change mat 5 cons 1 range reg 6180,3300 8500,3300 8500,3500 6180,3500 ; plastic
change mat 6 cons 1 range jreg 6
change mat 7 cons 1 range jreg 7
change mat 8 cons 1 range xr 5500 6000 yr 3000 3200 ;ore column - Panel A
change mat 9 cons 1 range xr 5000 5500 yr 3000 3200 ;ore column - Panel B
change mat 10 cons 1 range xr 4500 5000 yr 3000 3200 ;ore column - Panel C

; assign properties for horizontal and vertical joint sets
prop jmat 1 jkn=1e10 jks=1e9 jfric=30.0 jcoh=0.1 ;fault, joints
change jmat 1 jcons 2

;------------------boundary condition
boun yvel=0 range 0 10500 -701 -699 ;bottom
boun xvel=0 range -1 1 -700 3500 ;left
boun xvel=0 range 10499 10501 -700 3500 ;right

;----------------------------------------------
;in situ stress
;(k=1.925 for y , k=1.222 for z)
insitu stress -181.9e6 0 -94.5e6 szz -115.4e6 &
ygrad 5.2e4, 0, 2.7e4 zgrad 0,3.3e4
set grav 0 -10
step 5000
sav ini-500-orth.sav
rest ini-500-orth.sav
reset displ jdispl vel rot hist
set ovtol 1
hist unbal  ; hist 1
hist ydis  500,3500 ;hist 2
hist ydis  10000,3500 ;hist 3
hist ydis  1000,3500 ;hist 4
hist ydis  9500,3500 ;hist 5
hist ydis  1500,3500 ;hist 6
hist ydis  9000,3500 ;hist 7
hist ydis  2000,3500 ;hist 8
hist ydis  8500,3500 ;hist 9
hist ydis  2500,3500 ;hist 10
hist ydis  8000,3500 ;hist 11
hist ydis  3000,3500 ;hist 12
hist ydis  7500,3500 ;hist 13
hist ydis  3500,3500 ;hist 14
hist ydis  7000,3500 ;hist 15
hist ydis  3750,3500 ;hist 16
hist ydis  7250,3500 ;hist 17
hist ydis  4000,3500 ;hist 18
hist ydis  6500,3500 ;hist 19
hist ydis 4250,3500 ;hist 20
hist ydis 6250,3500 ;hist 21
hist ydis 4500,3500 ;hist 22
hist ydis 6000,3500 ;hist 23

; change to plastic
change mat 4 cons 3 range jreg 1
change mat 4 cons 1 range jreg 11
change mat 4 cons 1 range jreg 12
change mat 1 cons 3 range jreg 2
change mat 1 cons 3 range jreg 15; plastic
change mat 2 cons 1 range jreg 3
change mat 3 cons 1 range jreg 4
change mat 5 cons 3 range jreg 5
change mat 6 cons 1 range jreg 6
change mat 7 cons 1 range jreg 7

caved rock
prop mat 20 dens=2300 bulk=0.1e9 shear=0.05e9
change mat 20 cons 1 range 5500 6000 3000 3020
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3020
prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-A-20-orth.sav
change mat 20 cons 1 range 5500 6000 3000 3040
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3040
prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-A-40-orth.sav
change mat 20 cons 1 range 5500 6000 3000 3060
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000  3000,3060

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-A-60-orth.sav

change mat 20  cons 1 range  5500 6000  3000 3080
change mat 20  cons 1 range  5000 5500  3000 3020

insitu stress  \(-23.7e6,0, -71.3e6\) szz \(-23.7e6\) &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000  3000,3080
insitu stress  \(-23.7e6,0, -71.3e6\) szz \(-23.7e6\) &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500  3000,3020

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-B-20-orth.sav

change mat 20  cons 1 range  5500 6000  3000 3100
change mat 20  cons 1 range  5000 5500  3000 3040

insitu stress  \(-23.7e6,0, -71.3e6\) szz \(-23.7e6\) &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000  3000,3100
insitu stress  \(-23.7e6,0, -71.3e6\) szz \(-23.7e6\) &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500  3000,3040

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-B-40-orth.sav

change mat 20  cons 1 range  5500 6000  3000 3100
change mat 20  cons 1 range  5000 5500  3000 3040

insitu stress  \(-23.7e6,0, -71.3e6\) szz \(-23.7e6\) &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000  3000,3100
insitu stress  \(-23.7e6,0, -71.3e6\) szz \(-23.7e6\) &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500  3000,3040

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-B-40-orth.sav

change mat 20 cons 1 range 5500 6000 3000 3120
change mat 20 cons 1 range 5000 5500 3000 3060

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3120
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3060

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-B-60-orth.sav

change mat 20 cons 1 range 5500 6000 3000 3140
change mat 20 cons 1 range 5000 5500 3000 3080
change mat 20 cons 1 range 4500 5000 3000 3020

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3140
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3080
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000 3000,3020

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-20-orth.sav

change mat 20 cons 1 range 5500 6000 3000 3160
change mat 20 cons 1 range 5000 5500 3000 3100
change mat 20 cons 1 range 4500 5000 3000 3040

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3160
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3100
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 4500,5000  3000,3040

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-40-orth.sav

change mat 20 cons 1 range  5500 6000  3000 3180
change mat 20 cons 1 range  5000 5500  3000 3120
change mat 20 cons 1 range  4500 5000  3000 3060

insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
    ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000  3000,3180
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
    ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500  3000,3120
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
    ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000  3000,3060

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-60-orth.sav

change mat 20 cons 1 range  5500 6000  3000 3200
change mat 20 cons 1 range  5000 5500  3000 3140
change mat 20 cons 1 range  4500 5000  3000 3080

insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
    ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000  3000,3200
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
    ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500  3000,3140
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
    ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000  3000,3080

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-80-orth.sav

return
D.3 UDEC Benchmark: 500m Deep Undercut with Orthogonal Jointing at 45 and 135 Degrees

new

ro 0.1
;config cell 2100 840

;; commands for build model geometry
;-------------------------------------------------------------
block 0 -700 0 3500 10500 3500 10500 -700
;cal readprop.fis
;-------------------------------------------------------------

;; define regions for generate joints, generate density and assign properties,
;jreg id 1 0,3500 4485,3500 4338.386,2750.876 0,2750.876 ;granodiorite
;jreg id 11 0,2750.876 4338.386,2750.876 4172.096,1901.222 0,1901.222
;jreg id 12 0,1901.222 4172.096,1901.222 3800,-700 0,-700

;jreg id 2 4485,3500 6180,3500 6180,2345 4338.386,2750.876 ;rhyolite between faults
;jreg id 3 4338.386,2750.876 6180,2345 6180,1500 4172.096,1901.222 ;qtz monzodiorite
;jreg id 4 4172.096,1901.222 6180,1500 6180,-700 3800,-700 ;sanstone/siltstone

;jreg id 5 6180,3500 10500,3500 10500,2635 6180,2635 ;qtz monzodiorite right of fault
;jreg id 6 6180,2635 10500,2635 10500,2030 6180,2030 ;sandstone/siltstone
;jreg id 7 6180,2030 10500,2030 10500,-700 6180,-700 ;granodiorite
;-------------------------------------------------------------

; faults
crack 4485 3500 3800 -700
crack 6180 3500 6180 -700

;jointing
jset 45,0 5000,0 0,0 200,0 range jreg 12
jset 135,0 5000,0 0,0 200,0 range jreg 12
jset 45,0 5000,0 0,0 200,0 range jreg 4
jset 135,0 5000,0 0,0 200,0 range jreg 4
jset 45,0 5000,0 0,0 200,0 range jreg 7
jset 135,0 5000,0 0,0 200,0 range jreg 7
jset 45,0 2000,0 0,0 50,0 range jreg 11
jset 135,0 2000,0 0,0 50,0 range jreg 11
jset 45,0 2000,0 0,0 50,0 range jreg 3
jset 135,0 2000,0 0,0 50,0 range jreg 3
jset 45,0 2000,0 0,0 50,0 range jreg 6
jset 135,0 2000,0 0,0 50,0 range jreg 6
jset 45,0 2000,0 0,0 50,0 range jreg 1
jset 135,0 2000,0 0,0 50,0 range jreg 1
jset 45,0 2000,0 0,0 50,0 range jreg 2
jset 135,0 2000,0 0,0 50,0 range jreg 2
jset 45,0 2000,0 0,0 50,0 range jreg 5
jset 135,0 2000,0 0,0 50,0 range jreg 5

; elastic subsidence zone
jreg id 17 2500,3300 4450,3300 4450,3000 2500,3000
jset 0,0 5000,0 0,0 20,0 range jreg 17
jset 90,0 1000,0 0,0 20,0 range jreg 17

jreg id 18 6050,3300 6180,3300 6180,3000 6050,3000
jset 0,0 5000,0 0,0 20,0 range jreg 18
jset 90,0 1000,0 0,0 20,0 range jreg 18

jreg id 19 6180,3300 8000,3300 8000,3000 6180,3000
jset 0,0 5000,0 0,0 20,0 range jreg 19
jset 90,0 1000,0 0,0 20,0 range jreg 19

jreg id 20 0,3300 2500,3300 2500,3500 0,3500
jset 0,0 5000,0 0,0 20,0 range jreg 20
jset 90,0 1000,0 0,0 20,0 range jreg 20

jreg id 21 8000,3300 10500,3300 10500,3500 8000,3500
jset 0,0 5000,0 0,0 20,0 range jreg 21
jset 90,0 1000,0 0,0 20,0 range jreg 21

; plastic subsidence zone
jreg id 14 2500,3300 4435,3300 4485,3500 2500,3500
jset 0,0 5000,0 0,0 10,0 range jreg 14
jset 90,0 1000,0 0,0 10,0 range jreg 14

jreg id 15 4435,3300 6180,3300 6180,3500 4485,3500
jset 0,0 5000,0 0,0 10,0 range jreg 15
jset 90,0 1000,0 0,0 10,0 range jreg 15

jreg id 16 6180,3300 8000,3300 8000,3500 6180,3500
jset 0,0 5000,0 0,0 10,0 range jreg 16
jset 90,0 1000,0 0,0 10,0 range jreg 16

; ore column
jreg id 13 4450,2900 6050,2900 6050,3300 4450,3300
jset 0,0 2000,0 0,0 10,0 range jreg 13
jset 90,0 500,0 0,0 10,0 range jreg 13

del area 5
deldelete

save ini-500-geom.sav

rest ini-500-geom.sav

;-----------------------------
; change rigid block into deformable
gen edge 50 range jreg 17
gen edge 50 range jreg 18
gen edge 50 range jreg 19
gen edge 50 range jreg 20
gen edge 50 range jreg 21
gen edge 20 range jreg 13
gen edge 20 range jreg 14
gen edge 20 range jreg 15
gen edge 20 range jreg 16
gen edge 100
save ini-500-mesh.sav
rest ini-500-mesh.sav

;-----------------------------
;assign properties

; define different properties of rocks
prop mat 1 dens=2700 bulk=8.4e9 shear=5.1e9 coh 3.3e6 & fric 52 ten 0.6e6 dil 5 ;Rhyolite
prop mat 2 dens=2700 bulk=4.7e9 shear=2.8e9 coh 6.5e6 & fric 34 ten 1.8e6 dil 5 ;Quartz-Monzonite(Undercut)
prop mat 3 dens=2700 bulk=6.8e9 shear=4.1e9 coh 9.1e6 & fric 32 ten 2.5e6 dil 5 ;Sand & Silt(Undercut)
prop mat 4 dens=2700 bulk=5.1e9 shear=3.1e9 coh 3.2e6 & fric 44 ten 0.7e6 dil 5 ;Biotite(N of Fault)
prop mat 5 dens=2700 bulk=4.7e9 shear=2.8e9 coh 1.9e6 & fric 49 ten 0.4e6 dil 5 ;Quartz-Monzonite(S of Fault)
prop mat 6 dens=2700 bulk=6.8e9 shear=4.1e9 coh 6.9e6 & fric 35 ten 1.8e6 dil 5 ;Sand & Silt(S of Fault)
prop mat 7 dens=2700 bulk=5.1e9 shear=3.1e9 coh 8.2e6 & fric 33 ten 2.3e6 dil 5 ;Biotite(S of Fault)
prop mat 8 dens=2700 bulk=8.4e9 shear=5.1e9 coh 3.3e6 & fric 52 ten 0.6e6 dil 5 ;Panel A ore column Rhyolite
prop mat 9 dens=2700 bulk=8.4e9 shear=5.1e9 coh 3.3e6 & fric 52 ten 0.6e6 dil 5 ;Panel B ore column Rhyolite
prop mat 10 dens=2700 bulk=8.4e9 shear=5.1e9 coh 3.3e6 & fric 52 ten 0.6e6 dil 5 ;Panel C ore column Rhyolite

; assign rock properties to different regions

change mat 4 cons 1 range jreg 1
change mat 4 cons 1 range jreg 11
change mat 4 cons 1 range jreg 12
change mat 4 cons 1 range reg 2500,3300 4435,3300 4485,3500 2500,3500 ; plastic
change mat 4 cons 1 range reg 2500,3300 4450,3300 4450,3000 2500,3000
change mat 1 cons 1 range jreg 2
change mat 1 cons 3 range reg 4435,3300 6180,3300 6180,3500 4485,3500 ; plastic
change mat 1 cons 3 range reg 6050,3300 6180,3300 6180,3000 6050,3000
change mat 2 cons 1 range jreg 3
change mat 3 cons 1 range jreg 4
change mat 5 cons 1 range jreg 5
change mat 5 cons 1 range jreg 6
change mat 6 cons 1 range jreg 6
change mat 7 cons 1 range jreg 7
change mat 8 cons 1 range xr 5500 6000 yr 3000 3200 ; ore column - Panel A
change mat 9 cons 1 range xr 5500 5500 yr 3000 3200 ; ore column - Panel B
change mat 10 cons 1 range xr 4500 5000 yr 3000 3200 ;ore column - Panel C

; assign properties for horizontal and vertical joint sets
prop jmat 1 jkn=1e10 jks=1e9 jfric=30.0 jcoh=0.1 ;fault, joints
change jmat 1 jcons 2

;------------boundary condition
boun yvel=0 range 0 10500 -701 -699 ;bottom
boun xvel=0 range -1 1 -700 3500 ; left
boun xvel=0 range 10499 10501 -700 3500 ;right
;
;in situ stress
; (k=1.925 for y, k=1.222 for z)
insitu stress -181.9e6 0 -94.5e6 szz -115.4e6 &
ygrad 5.2e4, 0, 2.7e4 zgrad 0,3.3e4

set grav 0 -10
step 5000
sav ini-500.sav
rest ini-500.sav
reset displ jdispl vel rot hist
set ovtol 1

hist unbal ; hist 1
hist ydis 500,3500 ;hist 2
hist ydis 10000,3500 ;hist 3
hist ydis 1000,3500 ;hist 4
hist ydis 9500,3500 ;hist 5
hist ydis 1500,3500 ;hist 6
hist ydis 9000,3500 ;hist 7
hist ydis 2000,3500 ;hist 8
hist ydis 8500,3500 ;hist 9
hist ydis 2500,3500 ;hist 10
hist ydis 6000,3500 ;hist 11
hist ydis 3000,3500 ;hist 12
hist ydis 7500,3500 ;hist 13
hist ydis 3500,3500 ;hist 14
hist ydis 7000,3500 ;hist 15
hist ydis 3750,3500 ;hist 16
hist ydis 7250,3500 ;hist 17
hist ydis 4000,3500 ;hist 18
hist ydis 6500,3500 ;hist 19
hist ydis 4250,3500 ;hist 20
hist ydis 6250,3500 ;hist 21
hist ydis 4500,3500 ;hist 22
hist ydis 6000,3500 ;hist 23
; change to plastic
change mat 4 cons 3 range jreg 1
change mat 4 cons 1 range jreg 11
change mat 4 cons 1 range jreg 12
change mat 1 cons 3 range jreg 2
change mat 1 cons 3 range jreg 15; plastic
change mat 2 cons 1 range jreg 3
change mat 3 cons 1 range jreg 4
change mat 5 cons 3 range jreg 5
change mat 6 cons 1 range jreg 6
change mat 7 cons 1 range jreg 7

;caved rock
prop mat 20 dens=2300 bulk=0.1e9 shear=0.05e9
change mat 20 cons 1 range 5500 6000 3000 3020
insitu stress -23.7e6,0,-71.3e6 szz -23.7e6 &
ygrad 7.67e3,0,2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3020
prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-A-20.sav
change mat 20 cons 1 range 5500 6000 3000 3040
insitu stress -23.7e6,0,-71.3e6 szz -23.7e6 &
ygrad 7.67e3,0,2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3040
prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-A-40.sav
change mat 20 cons 1 range 5500 6000 3000 3060
insitu stress -23.7e6,0,-71.3e6 szz -23.7e6 &
ygrad 7.67e3,0,2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3060
prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-A-60.sav

change mat 20 cons 1 range 5500 6000 3000 3080
change mat 20 cons 1 range 5000 5500 3000 3020
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3080
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3020

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-B-20.sav

change mat 20 cons 1 range 5500 6000 3000 3100
change mat 20 cons 1 range 5000 5500 3000 3040
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3100
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3040

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-B-40.sav

change mat 20 cons 1 range 5500 6000 3000 3100
change mat 20 cons 1 range 5000 5500 3000 3040
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3100
insitu stress  -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3040

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-B-40.sav

change mat 20 cons 1 range 5500 6000 3000 3120
change mat 20 cons 1 range 5000 5500 3000 3060

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3120
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3060

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-B-60.sav

change mat 20 cons 1 range 5500 6000 3000 3140
change mat 20 cons 1 range 5000 5500 3000 3080
change mat 20 cons 1 range 4500 5000 3000 3020

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3140
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3080
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000 3000,3020

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-20.sav

change mat 20 cons 1 range 5500 6000 3000 3160
change mat 20 cons 1 range 5000 5500 3000 3100
change mat 20 cons 1 range 4500 5000 3000 3040

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3160
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3100
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000 3000,3040

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-C-40.sav

change mat 20 cons 1 range 5500 6000 3000 3180
change mat 20 cons 1 range 5000 5500 3000 3120
change mat 20 cons 1 range 4500 5000 3000 3060

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3180
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3120
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000 3000,3060

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-C-60.sav

change mat 20 cons 1 range 5500 6000 3000 3200
change mat 20 cons 1 range 5000 5500 3000 3140
change mat 20 cons 1 range 4500 5000 3000 3080

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3200
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3140
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000 3000,3080

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010
step 5000
save 500-C-80.sav

change mat 20 cons 1 range 5500 6000 3000 3200
change mat 20 cons 1 range 5000 5500 3000 3160
change mat 20 cons 1 range 4500 5000 3000 3100

insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5500,6000 3000,3200
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 5000,5500 3000,3160
insitu stress -23.7e6,0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0,7.67e3 range 4500,5000 3000,3100
prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-100.sav

change mat 20 cons 1 range 5500 6000 3000 3200
cchange mat 20 cons 1 range 5000 5500 3000 3180
cchange mat 20 cons 1 range 4500 5000 3000 3120

insitu stress -23.7e6, 0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 5500, 6000 3000, 3200
insitu stress -23.7e6, 0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 5000, 5500 3000, 3180
insitu stress -23.7e6, 0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 4500, 5000 3000, 3120

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-120.sav

change mat 20 cons 1 range 5500 6000 3000 3200
cchange mat 20 cons 1 range 5000 5500 3000 3200
cchange mat 20 cons 1 range 4500 5000 3000 3140

insitu stress -23.7e6, 0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 5500, 6000 3000, 3200
insitu stress -23.7e6, 0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 5000, 5500 3000, 3200
insitu stress -23.7e6, 0, -71.3e6 szz -23.7e6 &
ygrad 7.67e3, 0, 2.3e4 zgrad 0, 7.67e3 range 4500, 5000 3000, 3140

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone
change jmat 1 jcons 2

prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; block height
change jmat 2 jcons 2 range xr 4500 6000 yr 3010 3300

prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr 0 10000 yr -700 3010

step 5000
save 500-C-140.sav

change mat 20 cons 1 range 5500 6000 3000 3200
cchange mat 20 cons 1 range 5000 5500 3000 3200
cchange mat 20 cons 1 range 4500 5000 3000 3160
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $5500,6000$ $3000,3200$
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $5000,5500$ $3000,3200$
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $4500,5000$ $3000,3160$

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0;block height change jmat 2 jcons 2 range xr $4500$ $6000$ yr $3010$ $3300$
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr $0$ $10000$ yr $-700$ $3010$
step 5000
save 500-C-160.sav
change mat 20 cons 1 range $5500$ $6000$ $3000$ $3200$
change mat 20 cons 1 range $5000$ $5500$ $3000$ $3200$
change mat 20 cons 1 range $4500$ $5000$ $3000$ $3180$

insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $5500,6000$ $3000,3200$
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $5000,5500$ $3000,3200$
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $4500,5000$ $3000,3180$

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0;block height change jmat 2 jcons 2 range xr $4500$ $6000$ yr $3010$ $3300$
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr $0$ $10000$ yr $-700$ $3010$
step 5000
save 500-C-180.sav
change mat 20 cons 1 range $5500$ $6000$ $3000$ $3200$
change mat 20 cons 1 range $5000$ $5500$ $3000$ $3200$
change mat 20 cons 1 range $4500$ $5000$ $3000$ $3200$

insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $5500,6000$ $3000,3200$
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $5000,5500$ $3000,3200$
insitu stress $-23.7e6,0,-71.3e6$ szz $-23.7e6$ & ygrad $7.67e3,0,2.3e4$ zgrad $0,7.67e3$ range $4500,5000$ $3000,3200$

prop jmat 1 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0; subsidence zone change jmat 1 jcons 2
prop jmat 2 jkn=1e9 jks=1e8 jfric=30.0 jcoh=0;block height change jmat 2 jcons 2 range xr $4500$ $6000$ yr $3010$ $3300$
prop jmat 3 jkn=1e9 jks=1e8 jfric=40.0 jcoh=0
change jmat 3 jcons 2 range xr $0$ $10000$ yr $-700$ $3010$
step 5000
256
D.4 FLAC3D Benchmark: 500m Deep Undercut

new
rest mesh.sav

model mohr
prop density 2700 bulk 5.1e9 shear 3.1e9 coh 2.4e6 fric 33 ten 0.5e6 dil 5
range group Biotite-granodiorite-NoFF
prop density 2700 bulk 5.1e9 shear 3.1e9 coh 6.2e6 fric 25 ten 1.7e6 dil 5
range group Biotite-granodiorite-Soff
prop density 2700 bulk 8.4e9 shear 5.1e9 coh 2.5e6 fric 39 ten 0.5e6 dil 5
range group Rhyolite
prop density 2700 bulk 4.7e9 shear 2.8e9 coh 1.4e6 fric 37 ten 0.3e6 dil 5
range group Quartz-Montonite-Soff
prop density 2700 bulk 4.7e9 shear 2.8e9 coh 4.9e6 fric 26 ten 1.4e6 dil 5
range group Quartz-Montonite-undercut
prop density 2700 bulk 4.7e9 shear 2.8e9 coh 4.9e6 fric 26 ten 1.4e6 dil 5
range group Quartz-Montonite-Woff
prop density 2700 bulk 6.8e9 shear 4.1e9 coh 5.2e6 fric 26 ten 1.4e6 dil 5
range group Sandstone-Siltstone-Soff
prop density 2700 bulk 6.8e9 shear 4.1e9 coh 6.8e6 fric 24 ten 2e6 dil 5
range group Sandstone-Siltstone-undercut

inter 1 prop kn 1e9 ks 1e8 fric 30 coh 0.1
inter 2 prop kn 1e9 ks 1e8 fric 30 coh 0.1
inter 3 prop kn 1e9 ks 1e8 fric 30 coh 0.1

ini szz -81e6 grad 0 0 2.7e4
ini syy -155.92e6 grad 0 0 5.2e4 ;1.925
ini sxx -98.98e6 grad 0 0 3.3e4 ;1.222

fix x range x -2010.1 -1900.1
fix x range x 7899.9 8100.1
fix y range y -10.1 100.1
fix y range y 7999.9 8000.1
fix z range z -10.1 100.1

set gravity 0 0 -10
hist unbal
solve

save mesh-ini.sav
cal cave.dat
new
rest mesh-ini.sav
set large
ini xdis=0 ydis=0 zdis=0

model elas range x 3500 4500 y 3250 3750 z 2500 2550
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3250 3750 z 2500 2550
ini szz -58.65e6 grad 0 0 2.3e4 range x 3500 4500 y 3250 3750 z 2500 2550
ini syy -19.55e6 grad 0 0 7.67e3 range x 3500 4500 y 3250 3750 z 2500 2550
ini sxx -19.55e6 grad 0 0 7.67e3 range x 3500 4500 y 3250 3750 z 2500 2550

solve

save 500-step1.sav

;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;
model elas range x 3500 4500 y 3250 3750 z 2500 2600
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3250 3750 z 2500 2600
ini szz  -59.8e6 grad 0 0 2.3e4 range 3500 4500 y 3250 3750 z 2500 2600
ini syy  -19.9e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2600
ini sxx  -19.9e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2600

model elas range x 3500 4500 y 3750 4250 z 2500 2550
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3750 4250 z 2500 2550
ini szz  -58.65e6 grad 0 0 2.3e4 range 3500 4500 y 3750 4250 z 2500 2550
ini syy  -19.55e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2550
ini sxx  -19.55e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2550

solve
save 500-step2.sav

;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;
model elas range x 3500 4500 y 3250 3750 z 2500 2650
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3250 3750 z 2500 2650
ini szz  -60.95e6 grad 0 0 2.3e4 range 3500 4500 y 3250 3750 z 2500 2650
ini syy  -20.32e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2650
ini sxx  -29.32e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2650

model elas range x 3500 4500 y 3750 4250 z 2500 2650
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3750 4250 z 2500 2650
ini szz  -59.8e6 grad 0 0 2.3e4 range 3500 4500 y 3750 4250 z 2500 2650
ini syy  -19.9e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2650
ini sxx  -19.9e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2650

solve
save 500-step3.sav

;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;
model elas range x 3500 4500 y 3250 3750 z 2500 2700
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3250 3750 z 2500 2700
ini szz  -62.1e6 grad 0 0 2.3e4 range 3500 4500 y 3250 3750 z 2500 2700
ini syy  -20.7e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2700
ini sxx  -20.7e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2700

model elas range x 3500 4500 y 3750 4250 z 2500 2650
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3750 4250 z 2500 2650
ini szz  -60.95e6 grad 0 0 2.3e4 range 3500 4500 y 3750 4250 z 2500 2650
ini syy  -19.95e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2650
ini sxx  -19.95e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2650

solve
save 500-step3.sav

;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;
model elas range x 3500 4500 y 3250 3750 z 2500 2700
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3250 3750 z 2500 2700
ini szz  -62.1e6 grad 0 0 2.3e4 range 3500 4500 y 3250 3750 z 2500 2700
ini syy  -20.7e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2700
ini sxx  -20.7e6 grad 0 0 7.67e3 range 3500 4500 y 3250 3750 z 2500 2700

model elas range x 3500 4500 y 3750 4250 z 2500 2650
prop density 2300 bulk 2e8 shear 1e8 range x 3500 4500 y 3750 4250 z 2500 2650
ini szz  -60.95e6 grad 0 0 2.3e4 range 3500 4500 y 3750 4250 z 2500 2650
ini syy  -19.95e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2650
ini sxx  -19.95e6 grad 0 0 7.67e3 range 3500 4500 y 3750 4250 z 2500 2650

solve
save 500-step3.sav

;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;;
solve
save 500-step4.sav
Appendix E: Major Lithological Units at Palabora and Summary of their Assessed Rock Mass Characteristics, Used to Develop the FLAC3D Palabora Model

<table>
<thead>
<tr>
<th>Unit</th>
<th>Geological Description</th>
<th>Rock Mass Characteristics</th>
</tr>
</thead>
<tbody>
<tr>
<td>Carbonatite</td>
<td>Igneous rock composed predominantly of carbonate minerals. Represented by two mineralogically similar units: a fine-grained Transgressive Carbonatite, which lacks significant foliation, and a medium- to coarse-grained Banded Carbonatite with vertical to steeply dipping foliation.</td>
<td>Both units are relatively strong, with average UCS values of 120 MPa. They are described as being moderately fractured to massive (GSI 50 to 75) with mostly vertical jointing. Both are treated as one unit within the numerical model given their complex boundary contact and similar rock mass properties.</td>
</tr>
<tr>
<td>Foskorite</td>
<td>Coarse-grained ultra-basic igneous rock composed of olivine, magnetite, apatite and phlogopite. It occurs on all walls as a broad zone between the Pyroxenite and the Carbonatite.</td>
<td>Typically strong and competent with an average UCS of 90 MPa. This unit is described as being moderately to highly fractured (GSI 45 to 75) with mostly vertical jointing.</td>
</tr>
<tr>
<td>Pyroxenite</td>
<td>Ultramafic igneous rock consisting essentially of minerals of the pyroxene group. Represented by two units: Feldspathic and Micaceous Pyroxenite. Feldspathic Pyroxenite occurs in limited quantities, primarily on the north and west sides of the pit. Micaceous Pyroxenite occupies a large portion of the north, west and east walls as well as a narrow band within the south wall.</td>
<td>The UCS of Feldspathic pyroxenite is typically higher than the Micaceous, with average values of 100 and 85 MPa, respectively. These units are described as being highly fractured to massive (GSI 45 to 65) with mostly vertical jointing. Both are treated within the numerical model as one geomechanical unit.</td>
</tr>
<tr>
<td>Glimmerite</td>
<td>An ultrabasic igneous rock, consisting almost wholly of essential dark mica, either phlogopite or biotite. Occurs in limited quantities on the western side and southwest corner of the pit.</td>
<td>Highly variable strength, with an average UCS of 40 MPa. This unit is described as being highly fractured and sheared (GSI 35 to 50) with mostly vertical jointing. Glimmerite represents the weakest rockmass material present.</td>
</tr>
<tr>
<td>Fenite</td>
<td>Na- and K-rich silicate rock developed through alteration of the Archean granite contact. Fenite occupies a large portion of the South Wall and a narrow band behind the western and northwestern pit crest.</td>
<td>A hard, strong rock with an average UCS of 200 MPa. This unit is described as being moderately fractured (GSI 50 to 75) with mostly both vertical and horizontal jointing.</td>
</tr>
<tr>
<td>Granite</td>
<td>Medium to coarse grained intrusive, felsic, igneous rock. The Archean granite surrounds the complex and occupies a small section of the upper southwest pit wall.</td>
<td>A hard, strong rock with an average UCS of 200 MPa. This unit was described as being relatively massive (GSI 70 to 80).</td>
</tr>
</tbody>
</table>