GEOMECHANICAL EVALUATION OF A MECHANICAL UNDERCUTTING SYSTEM IN BLOCK CAVING

by

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ABSTRACT

Nowadays, Block Caving is the underground mining method with the highest productivity in the world. Many of the components of block caving have changed in the last few years, as new techniques and equipment have become available, increasing the efficiency of the method. Such components include automatic LHD (load, haul and dump) trucks and trains, electronic detonators and mechanical equipment for the development of drifts and shafts.

Nevertheless, the first link of the chain of value in Block Caving is the undercutting process, which continues, in essence, being performed in the same way it was when the method was invented, utilizing drill and blast. This thesis proposes to use a mechanical system to perform the undercutting, similar to the longwall method used in coal mines. In order to evaluate the technical feasibility of this option, a geomechanical evaluation about the feasibility of the implementation of this alternative system in block caving was developed.

Starting with the geomechanical study, the first step was exploring the possible geomechanical hazards that could arise if this alternative method is implemented in block caving, taking into consideration that the undercutting work should be performed from inside of caving volume. An undercutting mine design was proposed and interest zones were defined, also a risk analysis was carried out. The result of the risk analysis was the identification of three possible hazards, losses of confinement in the cutting front, wedges in the roof of the work area, and collapse of the pillar between panels.

In order to assess the geomechanical hazards, numerical models and kinematic analyses were conducted. For these, numerical models were built in FLAC$^{3D}$ to study the geomechanical behavior of the cutting front, work area, and the pillar between panels. The models further studied the relation between each of these elements and the undercutting advance and caving propagation. A mine design and sequence has been proposed for these analyses. The mine design was made based on the macro block method and the sequence of undercutting advance was made based on the longwall method.

The undercutting front advance was modeled in two stages, pre-caving and continuous caving, in order to take into consideration the undercut advance and the caving progress. The results of the analyses showed that the stress quickly raised with the increase of the undercut area until it reached the hydraulic radius of caving. In order to estimate the initiation and depth of
fracturing in the cutting front and to evaluate the risk of losses of confinement in the center of the cutting front, two methodologies were used: UCS_{lab} and critical strain \( \varepsilon_c \). These methodologies showed that on the cutting front fracturing started when the undercut area reached 20 to 30% of the panel area. Thus, the conditions where the equipment will work change constantly with the advance of the undercutting front. Therefore, the machine should cut and remove the rock in the cutting front. In addition, three different cutting front widths were analyzed: 120, 150, and 180 meters. For this range of widths, the difference in stress behavior is low. On the other hand, estimates of vertical displacements in the work area increase with the advance of the undercutting front, but near the cutting front the vertical displacements will be low; therefore, stability in the work area could probably be controlled structurally by wedges formed in the roof.

Numerical models show that during continuous caving, the length of the beam does not have greater influence on variations in the stress condition. Regarding the pillar between panels, it should accomplish the function of giving continuity to the method and helping it to provide stability of the work area. The pillar design is robust, with a width-to-height ratio of five; therefore, it has a high strength. The numerical models confirm this approach by showing that although there is damage in the sides of the pillar, the damage never reaches the center of the pillar.

Estimation of the possible undercutting advance rate using a mechanical undercutting system in block caving shows that it is similar to or higher than the traditional drill and blast rate.

The research developed in this thesis presents an opportunity to improve the quality of undercutting, ensuring the cut of the block and eliminating the risk of a remnant pillar that would affect the production level. In addition, it delivers the option of a possible increase in the performance of the undercutting, and an exploration of a possible automation of the process, both of which would contribute to the modernization of block caving exploitation. The main conclusion of this work is that from a geomechanical point of view, it is feasible to use a mechanical undercutting system in block caving.
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1.1 Background

The largest mining companies in the world have begun the construction phase of their biggest projects in block caving mining history. In the coming decades we will see that underground production rates increase more dramatically than ever before (Brown, 2007b, Chitombo, 2010). Block caving is an efficient method of underground mining with a high production rate; it is the method currently used in the biggest underground mines around the world. It has evolved through time from a manual to a mechanized production operation, adapting to different rock qualities and stress conditions. During the application of block caving, the first activity in the process to generate the caving is the undercut, which uses drilling and blasting to cut the base of the blocks (Hustrulid, 2001).

Modern block caving operations such as Palabora, DOZ, and El Teniente have incorporated changes in the sequences of mining construction. Most of these changes have been related to the undercutting advance, because this process causes an important increase in the abutment stress, which in turn damages the infrastructure near the undercutting front (Chitombo, 2010). In order to mitigate this effect, new undercutting variants have been incorporated which are pre-undercutting and advanced undercutting (Bartlett, 2000). These changes have led to an increase in the available area for the production and a decrease in the repair rates for the production levels (Araneda, 2007). However, along with these positives come negative effects in the undercut level, because these variants need around twice as much development in the undercut level, compared with the post undercut method, to compensate for the swelling factor and achieve an adequate undercutting blasting. This causes a decrease in the width of the pillars in the undercut level, which leads to reduction in strength of the pillars, causing collapses in undercut and production levels (Villegas, 2008).

Nowadays, considering the deepening of mining (and high stresses due to the height of the rock column), the pre-undercutting and advanced undercutting variants would be the most adequate for a high stress condition. Nevertheless, the stability problem in the undercut level (specifically the pillar collapses due to the thinness of the rock pillars) has not yet been solved,
and it is necessary to make mitigation efforts. Such efforts may include pre-charge of explosives or use of extra support in the undercut level (Bartlett, 2000).

Due to a shortage of new mineral deposits close to the surface, the trend of the mining industry is toward underground exploitations, and block caving is the method most used and which provides the highest production (Brown, 2007).

One of the main concerns during mining by block caving is to avoid leaving areas with a deficient undercut, which could provoke intense damage in the levels below and generate collapses that result in loss of productive areas. In contrast, the longwall caving method that is used in underground coal mining has overcome this problem. The longwall method uses a machine to cut the rock, a series of hydraulic shields to maintain ground stability in the work area, advancing and cutting the base of the rock column. The method provides absolute certainty that the cut of the panel is total, without the possibility of a remnant pillar (Peng, 2006).

1.2 Justification for the research conducted

Codelco is in the first phase of construction of the two biggest underground projects in the world: the Chuquicamata Underground Project and the New Mine Level, name given to the new production sector in El Teniente mine. They will be exploited by using the block caving method, with a production of 140,000 ton per day. The topic developed in this thesis is part of the aim of modernization of future operations through changes in the process. In this case, changes to the undercutting process are examined in order to ensure the cut quality and research of possible management opportunities from changing a discontinuous undercutting process such as drill and blast to a continuous process such as mechanical undercutting, eliminating or mitigating the negative geomechanical effects. Based on the aforesaid, this thesis proposes the possibility to use longwall method as a mechanical undercutting system. This is based on the experience of the longwall method in coal mining

1.3 Objective of the research undertaken

The main objective of this thesis is to evaluate, based on a geomechanical point of view, the technical feasibility of using a mechanical excavation system for undercutting in underground block caving.
This thesis presents a geomechanical evaluation of a proposed alternative method for the classical undercutting system. Advantages of the proposed undercutting system include ensuring the undercutting and decreasing the quantity of development in the undercut level. This in turn would improve excavation stability and reduce long-term costs. The use of a mechanical system would also offer the ability to increase the undercutting rate and offer the possibility to automate the system.

In the first part of this work the possible geomechanical aspects of the use of a mechanical undercutting system in block caving are identified and evaluated. The objective of this new approach may be applied in future block caving operations. During this first part, it was necessary to explore all possible stability problems in different stages of the caving, and identify the places more susceptible to damage. Every condition is explained and illustrated for a better understanding. With this information, a risk evaluation was made in order to identify the most critical problems for the geomechanical analyses. This work should also be useful for possible industrial testing.

The geomechanical evaluations of the main risks were made through analytical and numerical models, to identify parameters and conditions that control the stability problems and the alternative solutions. The numerical tool used for the geomechanical analyses was FLAC$^3$D. To develop this thesis, the data of the Chuquicamata Underground Project was used.

The numerical models were used in order to estimate the stress behavior in the undercutting front when the undercut front advance and the caving propagation grow, all within a scheme of advance undercutting similar to the longwall method.

A resulting consideration is the evaluation of the technical feasibility of controlling possible stability problems in the area of the operation of the cutting system, and in the pillar between undercut panels, in order to permit a safe operation for equipment and people.

Another objective is to present a rough estimation of possible productivity of a mechanical undercutting system compared to the classical drill and blast.

1.4 Research conducted and methodology

The research was conducted based on a geomechanical point of view, an evaluation of the possible application of a mechanical undercutting system in block caving. The bibliographic review was focused on geomechanical issues occurred during the undercutting process with
different undercutting methods, this due to that there are not experiences in applying mechanical undercutting in block caving. Because of this, it was necessary to propose a type of undercutting method with a mine design for mechanical undercutting. Based on the review and the Codelco Chile experience in block caving operations, an identification and evaluation of the possible risks in the implementation of a mechanical undercutting system were undertaken to recognize and prevent the most relevant hazards. Mining design and the rock mass properties used are derived from the Chuquicamata Underground Project.

With the information previously mentioned, numerical models were completed by using different caving stages and different widths of the undercutting front. The results of the numerical models showed the stress and deformation, and the analyses resulted in obtaining a behavior hypothesis, which has made possible an estimation of the potential performance of the system’s productivity and the exercise of a comparison with the drill and blast classical system.

1.5 Scope of the research

This work evaluates the technical feasibility of incorporating the mechanical undercutting in the block caving process, from a geomechanical approach, without considering the mechanical design of the equipment. Currently, the equipment—such as the shearer and hydraulic support—are developed for soft rock, but the technology associated with cutting tools and roof support capacities have grown significantly in recent years. This study is offered as a starting point toward the adaptation and development of this technology in block caving. Furthermore, the mechanical design of the equipment is beyond the scope of this work.

The information used in the analyses and numerical models is from the Chuquicamata Underground Project. This project’s characteristics, in terms of rock properties and stress conditions, correspond to average block caving operations elsewhere in the world. Also, this project sponsored the author of this thesis during the time of this research.

1.6 Original contribution

Nowadays, there is little research about undercutting system in block caving, and the latest innovations associated with the undercutting, such as pre and advance undercut, have increased the complexity of the system. In addition, the use of drill and blast system to initiate the caving has never been questioned.
This thesis is the first work to consider and evaluate, from a geomechanical point of view, the application of a mechanical undercutting system in block caving.

In this work, there is a mine design and an undercutting sequence which would allow the possible implementation of a project for industrial testing. In addition, this thesis presents an idea about of the stress behavior with the undercut, which could be used by mining equipment manufacturers to design or adjust the technology necessary for this application in block caving.

Another original contribution is a description of the possible geomechanical risk in the eventual uses of a mechanical system in block caving. This alternative, use a mechanical system to undercutting, opens the door for automation of the undercut system in block caving.

This work is an innovation work, never before in the metallic mining industry has been evaluate the option of use a mechanical system to do the undercutting in block caving, therefore an objective of this thesis will be an original contribution to the mining engineering.

1.7 Thesis organization

This thesis is divided into seven chapters.

Chapter 1 is an introduction that explains the motivation of the study and the problem statement, current geomechanical challenges in block caving, the objectives of the study and the scope of the thesis and how it is organized. Also, original contributions of this study are discussed and suggestions for further research are provided.

Chapter 2 provides a literature review, which present the different block caving undercut alternatives, the longwall method in coal mines and an overview of the study sector which was used as the basis for the studies and analyses developed in this work.

Chapter 3 in this chapter there is a description of mining method proposed for the analyses developed in this study, the mine design and performance for both methodologies are discussed, drill and blast method and mechanical undercutting, from the perspective of the implementation, and also gives a practical breakdown of the differences between a mechanical system and the classical system of drilling and blasting.

Chapter 4 identifies and evaluates the possible geomechanical aspects in the use of a mechanical undercutting system in block caving, and reveals the places most susceptible to suffer damage. Based on this a risk analysis was done.
Chapter 5 describes the organization of numerical models used for the studies and analyses of the mechanical undercutting system, and also the criteria for the evaluations used.

Chapter 6 discusses and analyzes the results of the numerical models for the different steps of the caving process in order to understand the behavior of the rock mass with regard to the results of the undercutting process.

Chapter 7 provides a summary of the work undertaken and the conclusions drawn from this study.
2.1 Block caving method

Block caving is a method of mining normally used in massive orebodies, in which a block of rock mass is defined, this block is fully undercut to initiate the caving. The undercutting process is performed through drill and blasting, the broken material is drawn to create a void to initiate the caving. The broken material is removed through the production level from the drawbells.

Block caving is a low cost production method with high productivity, but requires considerable investment in mine preparation. Today, there are large mining operations that use mechanized block caving and some other operations have trials for automated mode Figure 2.1.

![Figure 2.1 Typical scheme of block caving with LHD (Arce, 2002)](image)

2.1.1 Conceptual model of caving

Duplancic and Brady developed a conceptual caving model (Brown, 2007) Figure 2.2. The model includes four zones each has different rock mass properties due to their different degrees of fracturing. As caving progresses, the zones change sequentially in a process of rock mass degradation associated with the propagation of the caving. The zones are defined as follows.
o **Pseudo-continuous domain**: The virgin rock mass around the caving, without the caving effect. The behavior of the rock mass in general is elastic.

o **Seismogenic zone**: In this zone, the geological discontinuities start to open and new fractures are created. This generates seismic activity associated with damage to the rock mass.

o **Zone of loosening**: The rock mass in this zone has lost its cohesive strength. The rock mass is intensely fractured, and only has residual strength.

o **Caving zone**: This is fractured material that is to be removed through the production draw. It does not have cohesion properties.

o **Air gap**: The air gap is a zone that does not contain any material, which allows the caving to progress.

![Figure 2.2 Conceptual model of caving (Brown, 2007)](image)

Based on this conceptual model of caving, the behavior of stress and strain in the rock around the undercutting area is affected by the caving process. Therefore, the mine design is affected by the caving geometry (which changes with the caving propagation and the undercutting advance). Also, the size and the properties of the different zones defined in Figure 2.2 change, through a degradation process caused by continuous redistribution of stress. To estimate the behavior of the
stress/strain for any alternative or sequence of block caving using numerical models, it is necessary to consider the conceptual model of caving.

### 2.2 Undercut methods

Caving is initiated with the undercutting process in which the base of the block or panel is cut until the hydraulic radius of caving is reached (hydraulic radius is calculated dividing the area by the perimeter of undercut zone). The material inside the caving zone is removed through the drawpoints on the production level and the rock above this area collapses and the breaking process progresses until reaching the surface.

The undercutting activity is critical for the success of block caving (Laubscher, 2000), so it is necessary to have a plan and a design for the undercut. If the cut is of low quality, it is possible to lose production area, due to the damage or collapse of the production level, so it may be necessary to re-install ground support or rehabilitate some zones, which delays the production schedule.

The traditional undercut system is post-undercutting. Recently developed undercutting systems are the pre-undercut, advanced undercut, and macro block methods.

#### 2.2.1 Post undercut

Post-undercutting is the classical undercut method, (Figure 2.3), all levels (undercut, production ventilation, transference, hammer, transport, and other) are developed before the undercut process begins (drilling and blasting). Therefore, in this methodology it is necessary to make investments in advance in order to build the necessary infrastructure. The advantage is that when the undercut is done the sector is incorporated immediately into the production area. In this method the pillars in the undercut level have more strength than that achieved with the pre-undercut and advance undercut methods. The drifts in the undercut level have the same distance between them as the drifts in the production level, which gives greater pillar strength in the undercut level. In addition, the undercut level is used only for undercut drilling and blasting, and it is not necessary to handle muck on this level.

With post- undercutting, between one and three “drawbell lines” are blasted before the undercutting front passes above the draw points on the production level, to absorb part of the rock volume from the undercut blasting. Another advantage of post-undercutting is the relative
ease to handle the muck, which is extracted from the production levels (Figure 2.3). The disadvantage of the post-undercut is that abutment stress caused by the undercutting advance, which provokes damage in the production level, reduces the effective life of the drawpoints and production drifts (Brannon, 2011). Accordingly, Butcher (2000) suggests that this method is recommended only at a depth of about 500 meters. At greater depths, the resulting high stress conditions indicate this method is not recommended.

![Figure 2.3 Post-undercut (Brown, 2007)](image)

### 2.2.2 Pre undercut

In the pre-undercut method, the undercutting is done before the development and construction of the production level, (Figure 2.4). The undercut front advances ahead of any development on the production level. The blasting of the drawbell, and the development on the production level is made in a de-stressed environment. This method has the advantage of requiring only minor support in the production level, and the effective life of the drawpoints is higher than the effective life with the post-undercut method. The disadvantages of pre-undercutting are the complexities of coordinating development, construction, and the advance of the undercut front. In addition, pre-undercutting requires twice the amount of development in the undercut level than required with post-undercutting to absorb the rock removed from undercutting
blasting. This has an important effect on the stability of the undercut level, which requires more support than other methods such as pre-undercut and advance undercut. However, from an economic point of view, this alternative has the advantage that investment in production level infrastructure is made shortly before the area is incorporated into production.

![Figure 2.4 Pre-undercut (Brown, 2007)](image)

**2.2.3 Advance undercut**

In the advance undercut method, the undercut front advances ahead the blasting of drawbells and part of the development and construction of the production level is done before the undercutting front passes, Figure 2.5. This method has the advantage that it decreases the time necessary to complete the development and construction of the production level. Also, like the pre-undercut method, it requires less support on the production level. Nonetheless, the infrastructure developed before the undercut level needs strong support, and after the undercut front has passed it is sometimes necessary to perform some reparation. This method increases the effective life of the drawpoint, compared with post-undercutting. The disadvantage of this method is the complexity of coordinating development, construction, and the advance of the undercut front simultaneously. As with pre-undercutting, the advance undercut method requires
twice the amount of development in the undercut level, to absorb the rock from the undercut blasting.

Figure 2.5 Advanced undercut (Brannon, 2011)

2.2.4 “Crinkle cut” layout and blasting design in advance undercutting

The narrow inclined undercut, or crinkle cut, corresponds to the advance undercutting most used in block caving (Workshop undercutting 2008). It considers two concepts: the stability of the undercut level and production level, and the clearance of muck generated from the blast (clearance of muck creates the void necessary to contain the material for the next undercutting blast). To improve the stability of the production level, the height of the apex of the pillar between the drawbells is increased, and the drawbell is formed by the inclined blasting in the undercut level (Butcher, 2000). In addition, the inclination helps to remove the muck because the material flows by gravity due to its own weight. Figure 2.6 illustrates the inclined blasting and is possible to see the drawbell, drilling design, blasting design and also the caving and production levels all in 3D.

This design presents some problems inherent to the method (Casten, 2002). For example, it creates the perfect condition to generate a coarse fragmentation in the caving initiation, due to
the effect of the chevron geometry in the undercut roof. Butcher (2000) pointed out several other potential problem areas that must be taken into consideration when this method is used (Figure 2.7). These include: the possibility of leaving a remnant pillar in the apex of the crown pillar due to poor drilling and blasting, muckpile stacking that affects the next blasting, and the requirement of re-drilling in some blasting holes. Another difficult part of this method is the need for strict control of the process: the crinkle cut requires strong discipline in the operation (Bartlett, 2000).

Moreover, the geometry of the undercutting advance (plan view) is irregular, with high stress concentrations in the sharp parts, and the risk of losses of drills or abandonment of areas. Some solutions for this problem have been the use of pre-charges of explosives and remote control equipment to clear the area, in order to avoid the presence of people (Casten, 2002).

The undercutting method is highly demanding in the quantity of meters of the drift development.

In this method, the criterion is to open the drawbells behind the undercutting front, in a distress zone with a distance that normally depends on the capacity to complete the infrastructure of the production level (Araneda, 2007) (e.g., draw points construction, drilling of pilot shafts, and drilling of drawbells). The minimum horizontal distance between the undercutting front and the development and constructions, in the production level, is frequently the vertical distance between the undercut level and the production level. This is called “the 45 degree rule” (Brown, 2007). For example, in the case of the Esmeralda sector of El Teniente Mine—despite the fact that the distance between levels is 18 meters—the minimum distance used was 22.5 meters (Jofre, 2000).

### 2.2.5 Cavability

Brown (2007) defined cavability as “a measure (often non-quantitative) of the ability of an orebody to cave under particular circumstances” Cavability is associated with a minimum area (area of caving) and a minimum span, which is the minimum width of the undercutting front. This area of caving (hydraulic radius of caving, hydraulic radius is calculated dividing the area by the perimeter of undercut zone) has significant importance in the stability of in the undercutting front, because when the undercutting area achieves this value, the caving process starts and the roof of the cavity collapses— which changes the abutment stress around the caving area.
Figure 2.6 Illustration of crinkle cut in Palabora mine (Brown, 2007)

Figure 2.7 Crinkle cut undercut potential problem areas (Butcher, 2000)

2.2.6 Undercutting rate

There is not an optimal value for the undercutting rate, as it depends on the characteristics of the rock mass, the capacity of the development, the construction, and the production requirements (Flores, 2002). From the geomechanical point of view, the undercutting rate has influence on the stability of the abutment stress zone (Karzulovic, 2001). If the rate is very low or the undercutting front stops, the magnitude of the stress begins to open structures and create
new fractures (Butcher, 1999). This decreases the strength of the rock mass, provoking damage in the mining infrastructure. On the other hand, if the undercutting rate is too fast, it can originate high seismicity with the possibility of rock burst (Karzulovic, 2006). An average rate of undercutting is around 2,250 m$^2$/month (Figure 2.8) (Flores, 2002).

![Figure 2.8 Relative frequency of different undercut rates in caving mines (Flores, 2002)](image)

### 2.2.7 Geometry of undercutting front

The geometry of the undercutting front has an effect on the extent of damage ahead of the undercut front, basically because it can generate the stress of confinement, in the case of concave geometry, or traction, in the case of convex geometry. A convex geometry should be avoided because it generates more extensive damage (Figure 2.9) (Laubscher, 2000).

Based on Laubscher’s work (1994), Butcher (1999) developed five rules to avoid damage in block caving operations. One of the most important is the geometry of the undercutting front. Butcher points out that the problems associated with the geometry of the undercut are the large irregularities in the horizontal geometry of the undercut front (cave line), which cause an increase in stress concentration that may result in serious damage, (Figure 2.10). Therefore, the horizontal irregularity in the undercut front panel should be avoided (Brown, 2007).
2.2.8 Undercutting method in high stress

Mining in later years has demonstrated that caving methods can be applied successfully in strong rock mass conditions at greater depths, which increases geotechnical challenges (Flores, 2008). Together with the deepening of mining operations, there has been an increase in stress conditions, (Figure 2.11). The use of post-undercutting in a condition of high in-situ stress
(Flores, 2002) causes significant redistributions of stress (abutment stress) in the caving front, which causes damage in the production levels that in turn affects the available production area and increases the number of repairs. Figure 2.12 shows the increase in stress caused by the undercutting advance effect in the roof of a production drift (Bartlett, 1992).

![Figure 2.11](image1.png)

Figure 2.11 Variation of in-situ stress with depth (Karzulovic, 2004)

![Figure 2.12](image2.png)

Figure 2.12 Stress conditions in the roof of production drift by effect of the undercutting front advance (Trueman, 2002)
Some changes in the sequence of development and construction of mining infrastructure have been applied to solve these problems in order to manage the stress. In high-stress conditions, postponing all or part of the development and construction in the production levels until the undercut front has passed can prevent premature damage. This causes a decrease in the width of the pillars in the undercut level, which leads to a reduction in strength of the pillars, difficulty the undercutting blasting due to the damage in the blast hole, thus causing partial blasting and therefore the formation of remnant pillars (Villegas, 2008) (Figure 2.13, Figure 2.14 and Figure 2.15) or even collapses in undercut and production levels. For these reasons, some mines are considering returning to the post-undercut method and will assume the costs of reparation, but will gain flexibility and stability for the undercut level (Araneda, 2007).

![Diagram](image)

**Figure 2.13 Example of advanced undercutting front with small pillars (Workshop undercutting, 2008)**

### 2.2.9 Generation of dynamic stress

In all undercutting methods used in block caving, the undercut is performed by drilling and blasting to cut the base of the block. This is a discontinue process that increases stress due to the explosive detonation and the instantaneous redistribution of stress caused by the volume removed from the base of the block by the undercutting blast (Aguirre, 1994).
In order to control these effects, every mine carefully plans its undercutting program and avoids blasting large areas or using significant amounts of explosives (Flores, 2002). Mine project staff normally monitors the vibrations and seismic activity to adjust the blasting design. The effect of dynamic stress increases if the in-situ stress condition of high stress, because it causes a major transfer of energy around the blasting area (Dunlop, 1999). For this reason, the blasting of drawbells can cause minor damage in a pre- or advance undercut method where the stress condition is relaxed after the passage of the undercut front (Potvin, 2010).

Figure 2.14 Remnant pillar in undercut level (Karzulovic, 2005)

Figure 2.15 Damage caused by collapse in production level (Karzulovic, 2003)
2.2.10 Seismicity associated to undercutting process

The relationship between the undercutting advance with drill and blast and seismicity has been studied in El Teniente mine (Potvin, 2010). The conclusion reached was that there is a direct cause/effect relationship between the two. To illustrate this concept, Figure 2.16 presents the relationship between the undercut area and the seismic events in the sector RENO of El Teniente Mine (historically known for the rockburst). The red line represents the cumulative number of significant seismic events (seismic events that the people in the El Teniente mine could feel) in the timeframe measured. The green line shows the cumulative undercutting area by drill and blast. Clearly, there is a direct relationship between the frequency of the undercutting blasts and the increase in seismicity.

In addition, it is interesting to note that one of the strategies used to control the rockburst in block caving is to decrease the area of undercut in each blast along with the frequency between the blasts, to decrease the number and magnitude of the seismic events.

![Figure 2.16 Cumulative undercut area blasted (green dots) compared to the cumulative number of significant events (red line) showing a very strong correlation (Potvin, 2010)](image)

2.2.11 Numerical models of undercutting in block caving

The most common use of numerical models in block caving is associated to the caving propagation relationship with the extraction rate, namely beyond hydraulic radius, and parameter studies such as:
- Geometry of the cutting front during the font advance, in order to study the stability of undercut and production levels. Also, these analyses allow selecting the suitable, undercutting method (Brown, 2007).
- The caving rate, in this case the focus is on trying to predict when and how the fracturing arises in the surface. This is important information in the transition from open pit to underground method (Brown, 2007).
- The dilution prediction, this is a key economical factor, because it is related to the mineral recover and it is an input for planning the needs of reposition of the production area (Board, 2009).
- Subsidence estimation, this prediction is important for the location of infrastructure, shaft, crushers, and other that can be affected by the caving process. In addition, this parameter may be related to with the dilution entry (Board, 2006).

Usually, the numerical models in block caving are three dimensional, because the caving process affects the whole rock mass around the caving volume.

There is a significant uncertainty on the transition from intact rock to caving material (Sainsbury, 2010), the propagation rate of caving and the relation between the rock mass quality and the stress condition. In general, the numerical model of caving uses strain softening as constitutive property, which through the evaluation of “critical plastic strain” adjusts the rock properties and simulates the rock mass degradation around caving volume.

### 2.3 Longwall mining method

Longwall mining is a highly mechanized underground cave mining technique that is usually used to extract a seam from a bedded sedimentary deposit such as coal or trona. Seam heights are in the range of 2 to 5 m, which is relatively thin compared to block caving column heights.

Within non-metallic underground methods, longwall method has the highest productivity (Haycocks, 1997). The drawback is the high capital cost of equipment and installation (Energy Information Administration, 1994). The main components in longwall are hydraulic roof support, a cutting machine, and an armored face conveyor (Figure 2.17) (Mitchell, 2012).

The longwall system advances along a full face wall of rock with a cutting machine, and loads the coal mineral into a conveyor system parallel to the cutting face. A shield of hydraulic
support that advances together with the undercutting front advance protects the whole equipment system. Behind the shield support, the roof collapses, filling the cavity (Mitchell, 2012).

This method is used mainly in soft rock such as coal or potash. A disadvantage of longwall mining is the high level of skill and training required for personnel operating the system (Energy Information Administration, 1994). Moreover, if some of the components have problems, it stops the entire production.

![Diagram of longwall mining](image)

**Figure 2.17 Classic schematic of longwall, in a coal mine (Energy Information Administration, 1994)**

### 2.3.1 Mine design

A typical layout for a longwall coal mine is given in Figure 2.18. A series of roadways called main entries service a section of the mine. The roadways are developed using continuous mining machines (Peng, 2006). These entries are used for the main mine ventilation system, and provide access for people, machinery, electrical supply, communication systems, water pump out lines, compressed air lines and gas drainage lines.

The longwall design is related to the geometry of the coal seam such as height, dip, depth etc. and the production requirement (Peng, 2006). The geological and geotechnical investigations are very important to design suitable longwall application and choose the equipment properly.
Parameters such as water table, aquifers, faults and lithology of the rock columns, may define the success or failure of the method implementation. During the longwall designs is necessary to estimate the longwall production and define the width and length of the panel (Energy Information Administration, 1995), the number of entries, size of pillars, advance rate, advance direction, ventilation requirements and others.

A panel of coal to be extracted is blocked out using entries along the sides of the panel and connected together across the end of the panel. The pillars formed between adjacent entries are called chain pillars. In a developed longwall panel, the headgate is the entry where the belt conveyor system, the stage loader and crushing system used to transport the extracted coal are located. The tailgate entries are on the opposite side of the panel and are used to complete the ventilation circuit to remove methane and dust from the longwall cutting face. A barrier pillar is often left at the end of the panel to protect the main entries. Panel dimensions range from 150 to 300 meters wide, and 1000 to 3500 meters long.

Other typical longwall drift are bleeder entries (allow the ventilation to eliminate the methane concentration) and the main entries that allow the connection with other panel and access to the surface (Energy Information Administration, 1994). Longwall could be advancing or retreat type. In the advancing type the gate roads are formed as the coal face advances (Wikipedia, 2014). In retreat longwall the panel is formed by driving the headgate, tailgate and development before the cutting face advances. Figure 2.18 shows a classical longwall retreat design.

### 2.3.2 Production system

Currently in longwall all the production systems are mechanized. The roadways are developed using continuous mining machines (Peng, 2006). The production is obtained cutting the coal seam or other mineral to full face using a cutting machine called shearer, which consists in a drum with cutting picks (between 40 to 60) which cuts the rock in a continuous way. The rock falls over a conveyor system (AFC armored face conveyor) and is extracted by means of a belt system to the surface. The production operation is in the face, to protect the people and equipment a shield is used, which consists in hydraulic jacks that support the roof and advances coordinately; Figure 2.19 shows the production system (University of Wollongong, 2014).

The shearer haulage system is attached to the AFC, through a mechanical interface, the system pushes the drum to the face and this rotates cutting the coal, also the AFC is connected
with the hydraulic shield and everything is an interconnected system (University of Wollongong, 2014).

Figure 2.18 A typical longwall coal mine layout showing the panels, entries, and chain pillars, (Source: Ch13.8, SME Handbook, 2011).

Hydraulic shield is the system used to support the roof, this allows the work of the cutting machine and protects the people and production equipment (Barczak, 1992) (Peng, 2006):

The relationship between the support load, generated by a shield support and the stiffness of the support in terms of roof convergence, prior to generating the yield load is critical to control the roof above the work area (Mitchell, 2012). The purpose of hydraulic shield support is to prevent or delay the failure of the roof during the mining operation (Barczak, 1992).
The AFC, is formed for short sections, called pans, which allows to move the material from the cutting face. The section is 1.5 meters long average, the transport velocity is around 1 meter/second. The AFC is installed in front of the hydraulic support and behind shear system. All the system advances together.

In longwall, a crusher is used to reduce the coal fragment to the appropriate size for the belt design and then it is loaded onto the first conveyor belt by the BSL (Beam Stage Loader) (Peng, 2006).

The components of the longwall machine are highly integrated and controlled by a computerized control system. When the shearer passes, individual shields are advanced forward, and the entire longwall system advances thereby together in a systematic fashion.

![Figure 2.19 Overview of longwall system, main equipment (University of Wollongong website, 2014)](image)

### 2.3.3 Numerical model of longwall

The numerical models are widely used in the longwall studies. The main analyses are related to the stability of gates (Shabanimashcool, 2012), in order to ensure the safety of the workers and infrastructure stability, also the numerical models are used to evaluate the mine design, panel orientation and effect of undercut advance (Chen, 1998) (Qing-Sheng, 2013). Regarding design of pillars, chain pillars and yielding pillar, numerical model are used to estimate the strength of pillars (Badr, 2002).
The hydraulic support and shield support, are inherent aspects to the longwall method, in which is necessary to estimate the load variation during the undercut front advance (Barczak, 1992), namely the stress and deformations, for these analyses numerical model helps to predict possible behavior in different rock mass conditions. In order to study the behavior of these parameters, the numerical model must incorporate the strength of rock mass beyond the peak strength, incorporating the residual strength parameters. With these concepts most of the numerical models have used strain softening as constitutive property (Ozbay, 2003).

On the other hand, the longwall method is a caving method, the mineral is extracted and the cavity is filled by surrounding material, fracturing advances in vertical direction (Peng, 2006). Velocity and magnitude of vertical movement depend on the depth of exploitation and the rock mass properties, causing subsidence as a consequence (Keilich, 2006). In cases when there is infrastructure in the surface models are used to study the possible effects (Zhang, 2013).

Figure 2.20, shows an example of numerical model in longwall to study the abutment stress around a panel (Wei, 2011).

![Three dimensional model for computation.](image-url)

Figure 2.20 Three dimensional model (Wei, 2011)

The numerical models in longwall method and block caving method during the modeling of undercut process are similar, only with the difference in the damage caused by the blasting used
in the undercutting process in block caving, which is normally confined, this changes the rock properties estimation around of the undercutting area, Hoek 1998 presents a correction factor “D” to represent this effect in the rock mass properties (Cai, 2007). Another important difference is that in the block caving the mineral column is extracted and the draw process must be modeled.

2.4 Chuquicamata underground project

This work considers using the stress condition and the rock properties of a real case, where the mechanical undercutting could be applied.

In this case the Chuquicamata underground project information is considered to be appropriate, because this project is in the construction phase and is possible to evaluate the option of using a mechanical system to do the undercut.

2.4.1 Chuquicamata mine

Chuquicamata mine is an open pit mine located near Calama, II Region of Antofagasta, 1,650 kilometers north of Chile’s capital, Santiago. It is approximately 2,900 meters above sea level (masl) (Figure 2.21). Chuquicamata started its operations in 1915, although the high quality of the ore has been well known for centuries, dating back to the prehispanic cultures that inhabited the region. The production is 443,381 metric tons of fine copper electrorefined, electrowon cathodes, and copper concentrates. In 2011 Chuquicamata had 6,527 employees (CODELCO, 2011). Today Chuquicamata is a big open pit 4.9 kilometers long, 2.9 kilometers wide and 1.0 kilometers deep Figure 2.22 (Allende, 2009).

Chuquicamata plans to change its mining method from open pit to underground, because the open pit is approaching to the end of its economic mine life. The Chuquicamata Underground Project considers recovering around 1,700 million tons of ore with an average grade of 0.71% Cu, 499 ppm of Mo and 460 ppm of As, over an operational period of approximately 45 years. The studies to date suggest that the earliest start of production for the underground mine would be towards the end of 2018 at a rate of 4,000 tpd (metric tons per day) of ore from tunnels development. The production will increase in 2019 by supplementing with ore from the beginning of caving activities, followed by a production ramp-up to achieve the design capacity of 140 ktpd (Fuentes, 2009).
The project has considered four exploitation levels, starting in the central area and then advancing north and south. Two blocks were planned to test new technologies. These blocks will begin together with the regular exploitation blocks. This thesis uses the information from one of those blocks, block N 42 (Figure 2.23), for the different studies.
2.4.2 Geology

The mineralized area of the Chuquicamata deposit is found in a tertiary intrusive body in the immediate footwall of the West Fault. This intrusive unit corresponds to a porphyric stockwork of granodiorite composition, known as the Chuquicamata Porphyry. This unit has various textural and hydrothermal alteration differences that allow it to be subdivided into various units, including sericitic, potassic, and chloritic domains (Figure 2.24). The mineable portions of the ore body are found in the sericite-altered rock mass within the immediate footwall of the West Fault. The highest ore grades (>1% Cu) are found directly adjacent to the fault in the highly altered and weak RQS, Q<S (Quartz less than Sericite) unit, with decreasing grades in the less altered RQS (Q=S and Q>S) and porphyry (Porphyry East Sericitic, Porphyry East Potassic (K), and Porphyryc East Choritic) units. The immediate hanging wall of the West Fault is a heavily sheared, poor-quality Fortuna Granodiorite that increases in rock quality with distance away from the West Fault (Board, 2006).
2.4.3 Mining method

The mine exploitation method being used is the block caving method, in four levels (Figure 2.25). Two levels will be simultaneously in operation during the production (Fuentes, 2009). When the underground mine starts, the open pit operation should be stopped. There is not consideration of a transition time with both operations working.

In this project, the block caving method has been modified to the variant called “macro block” (Figure 2.26). This mining method offers advantages in cost and production capacity (Fuentes, 2009). The characteristic of this method is to separate and group the activities of developments and construction in one sector without interferences from other activities.

The advantages of macro block method with respect to panel caving are:

- It gives more flexibility in production planning, developments of mining infrastructure, and other operations.
- It is suitable for the geometry of the Chuquicamata’s ore body (long and thin).
- The modular design allows the incorporation of technological changes more easily.
- Major collapses in the production areas could be isolated easily.
- The activities of development, construction, and undercutting are separated from the production operation with the possibility of achieving a better index of productivity.

Figure 2.25 Position of production levels in Chuquicamata Underground Project (Fuentes, 2009)

Figure 2.26 Macro block method (Fuentes, 2009)
The macro block method was developed during the Chuquicamata Underground Project as a way of managing mining preparation and production administration. It is a variant of block caving, but with larger block size and with a pillar to separate the macro blocks. Its configuration was developed by the Chuquicamata Underground Project to confront the important need of preparing and replacing the depletion area in order to keep production over 140,000 tons/day (Fuentes, 2009). With this configuration, it is possible to isolate zones that have stability problems such as collapses, air blast risk, and mud rushes without stopping new areas from being incorporated into the production. The macro block method is a variant of block caving that offers great management opportunities to organize the activities of construction and production. Each macro block could be considered an independent unit with its own mine design and organization. Figure 2.27 shows the macro block configuration for the Chuquicamata Underground Project (Villegas, 2008). The method gives flexibility to incorporate new technologies or adjust designs. Each macro block should have enough area to initiate a new caving. In addition, each new macro block should have a special draw system, to control the dilution from neighboring blocks (Fuentes, 2009).

Figure 2.27 Macro block configuration in first and second levels of Chuquicamata Underground Project (Allende, 2009)
CHAPTER 3
UNDERCUTTING WITH A MECHANICAL SYSTEM

3.1 Block caving mine designs with mechanical undercutting

This work presents an alternative to the use of the drill and blasting method for the undercutting process; in this case, using a mechanical system similar to the one used in a longwall panel in underground coal mining. For this idea to be feasible, the undercutting process should be compatible with the production process. Therefore, it is necessary to have a mine design that incorporates both activities.

In the longwall panel method, the undercut front advances as the coal is cut and removed and behind it, the overlying rockmass collapses and is left behind, this area is called the Gob. In the block caving case, the Gob area is above the production level, where the caved rock is removed through drawpoints.

One important aspect to consider for the implementation of this mechanical undercutting system in block caving is the undercutting process, because when undercutting in block caving is developed, in any of its forms, the undercutting activity is located ahead of the cutting front, but in a mechanical system such as longwall is located behind the cutting front—namely, inside of the caving, Figure 3.1.

![Diagram of Drill and Blasting Undercutting vs Mechanical Undercutting System](image)

Figure 3.1 Differences in location of the work area in classical undercutting and mechanical undercutting system
When applying a longwall like mechanical undercutting system in block caving, the cutting process is accomplished from the inside of the undercut area. Therefore, when using a block caving exploitation system, it is necessary to separate the undercutting zone from the production zone or caving volume (where caving grows by the extraction process) Figure 3.2. This is possible using pre or advance undercut. In this configuration, the cutting system advances along with the undercut, and the drawbells are opened in the back to incorporate the area into the caving. For this reason, it is important to understand what might be the effect of the caving advance in the undercutting process, which is studied in Chapter 5.

![Scheme of mechanical undercutting system in block caving (modified from Brown, 2007)](image)

**Figure 3.2** Scheme of mechanical undercutting system in block caving (modified from Brown, 2007)

### 3.1.1 Undercutting mine design

The layout of the mine design for a mechanical undercutting system should consider the following requirements.

**Access to the cutting front is needed for:**
- The assembly and disassembly of the undercutting machine,
- Access to the work face for operation and maintenance of equipment,
- Ventilation of the front face, and
- Handling of muck.
Separation of undercut panels is needed for:

- Allowing access to the next panel to be undercut
- Use of all the potential associated with the block caving method by macro blocks.

**Dividing a panel to reduce the width of the undercutting front provides:**

- Control of the stability in the work area, and
- The ability to increase the rate of the undercutting front advance.

The design proposed as shown in Figure 3.3, considers four drifts: two as access to the undercutting front and two as connections between accesses. This configuration allows for ventilation during the undercutting process. One drift would be developed to initiate the undercutting work; the other would serve as a connection between accesses. In addition, this configuration needs other extra drifts to locate the electrical power system, electrical cabling, hydraulic systems, pumps, etc. This might be considered in the undercut planning, using the development of the next panel.

To give continuity to the mechanical undercutting method, it is necessary to leave a separation pillar between panels in order to ensure access and initiation of the undercutting process in the next panel. These pillars are located between the access drifts to the work areas (Figure 3.3, Figure 3.4 and Figure 3.5). The objectives of the pillars are to provide ground support for the access drifts to the cutting front, separate the panels, and divide the exploitation area to reduce the undercut width.

A section view of the proposed system is given in Figure 3.4, which shows the pillars between both panels and the distances between levels. In addition, with this method it is necessary to move all the material from the cutting front and put part of this material behind the shield support using a conveyor belt without exposing the worker to haul other material to an ore pass.

One of the main advantages of this design is that it significantly reduces the number of drifts compared with the post undercut, pre undercut and advance undercut which improves the stability of undercut level.

To facilitate the analysis of possible risks and hazards in the application of the system proposed, it is necessary to identify and define the zones considered in this study Figure 3.6
shows some definitions used. The sequence considered that when the equipment finishes the undercut in one panel, then the next panel starts.

Figure 3.3 Layout undercut level for a mechanical undercutting system

Figure 3.4 Section layout undercut and production level in a mechanical undercutting system

In order to identify the possible geomechanical risks during the use of a mechanical undercutting system in block caving, the area considered was divided into five zones.

These are the:

(1) Cutting front: The wall where the machine will cut the rock

(2) Work area: The zone where the machine and the people are transiting, which is protected by a roof support system

(3) Access drifts to panel: Undercutting drifts to access the cutting front
(4) **Pillar of separation between panels**: The rock volume between panels
(5) **Cave volume**: The volume of rockmass where the caving process occurs

![Plan view of undercut mine design for mechanical undercutting system in block caving](image)

**Figure 3.5** Plan view of undercut mine design for mechanical undercutting system in block caving

![Illustration of mechanical undercutting system](image)

**Figure 3.6** Illustration of mechanical undercutting system
3.1.2 Relation between undercut design and production level design

The use of a mechanical system for the undercutting process in block caving is feasible with the pre-undercut or advance undercut methods. This is because it is extremely risky or impossible to cut the rock in the undercut level with a longwall system and connect the drawbell at the same time (as in the post-undercut case). To advance with the undercutting process and production process simultaneously there are two parameters that need to be considered in the mine design: (1) The distance between the cutting front and the opened drawbell, (2) The undercut drift access position required to ensure good caving connection between panels.

The designs of the undercut and production level are related and construction of the production level must be scheduled according to the undercutting advance rate.

The design would also take into consideration the need of eliminating the pillar between the panels Figure 3.7.

![Mine design for mechanical undercutting system](image)

Figure 3.7 Mine design for mechanical undercutting system in block caving

3.2 Undercutting sequence

For this evaluation, the undercutting sequence in a mechanical undercutting method has the following considerations:

- The open drawbells (production initiation) start after the hydraulic radius of caving is achieved.
- The panel exploitation is sequential when the first panel finishes the undercutting and the caving propagation reaches the surface, the undercutting in the second panel starts.
- The minimum undercutting beam is one drawbell line (20 meters).
- Simultaneous undercutting is not considered in adjacent panels.

3.3 Performance of mechanical undercutting system.

In this section the possible performance and sequence for a traditional drill and blast and an eventual mechanical system are estimated, in order to provide a basis for comparing the two alternatives. The parameters used for the estimate correspond to normal parameters used in block caving, and the geometry of advance of the undercutting front is independent of the estimate.

3.3.1 Productivity in advance undercut with drilling and blasting

To estimate the productivity of this method, it is necessary to make some assumptions. For the advanced undercut with crinkle cut, the assumptions are as follows.

- The blasting activity will be six days per week.
- The undercutting blasting could occur one or two times per day.
- The simultaneous blasting of neighboring blocks is not allowed.
- Every blasting shot includes two rings.
- A maximum of 8 meters of leads or lags between drifts is recommended during the advance of the undercutting front.
- The swelling factor of the material that must be removed for the next blast is 40%.
- The area per blasting is 60 m².
- For every blast it is necessary to remove the swell material before the next shot.
- There is one ore pass in the undercut level.
- No remote LHD equipment is considered to remove the muck.

An estimate of the undercutting productivity for different undercutting front width was made based on the above assumptions (Table 3.1).
Table 3.1 Productivity estimation for an advance undercut with drill and blast.

<table>
<thead>
<tr>
<th>Undercutting data</th>
<th>Unit</th>
<th>Panel width panel (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>120 m</td>
</tr>
<tr>
<td>Length undercut</td>
<td>m</td>
<td>220</td>
</tr>
<tr>
<td>Width of panel</td>
<td>m</td>
<td>120</td>
</tr>
<tr>
<td>Area to undercutting</td>
<td>m²</td>
<td>26,400</td>
</tr>
<tr>
<td>Day/week</td>
<td>un</td>
<td>6</td>
</tr>
<tr>
<td>Day/month</td>
<td>un</td>
<td>24</td>
</tr>
<tr>
<td>Blast per day</td>
<td>un</td>
<td>1.5</td>
</tr>
<tr>
<td>Advance per blast</td>
<td>m²</td>
<td>60</td>
</tr>
<tr>
<td>Advance per month</td>
<td>m²</td>
<td>2,160</td>
</tr>
<tr>
<td>Time undercutting a panel</td>
<td>months</td>
<td>12</td>
</tr>
</tbody>
</table>

There is a linear relationship between the time necessary to undercut a panel and the panel width. One year is required to complete the undercut of a panel with a width of 120 meters (Fuentes, 2009). Table 3.2 presents an estimate of the time needed for this work.

Table 3.2 Muck to be removed in advance undercut.

<table>
<thead>
<tr>
<th>Data Mucking</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average distance to ore pass</td>
<td>m</td>
<td>500</td>
</tr>
<tr>
<td>Time cycle</td>
<td>min</td>
<td>4</td>
</tr>
<tr>
<td>Numbers of LHD</td>
<td>un</td>
<td>1</td>
</tr>
<tr>
<td>Capacity LHD</td>
<td>Ton</td>
<td>6</td>
</tr>
<tr>
<td>Rock cut per blasting</td>
<td>Ton</td>
<td>629</td>
</tr>
<tr>
<td>Rock cut per day</td>
<td>Ton</td>
<td>1,258</td>
</tr>
<tr>
<td>Muck to remove per day 40%</td>
<td>Ton</td>
<td>503</td>
</tr>
<tr>
<td>Number of buckets per blasting</td>
<td>un</td>
<td>84</td>
</tr>
<tr>
<td>Time mucking per day</td>
<td>hours</td>
<td>5.6</td>
</tr>
</tbody>
</table>

3.3.2 Advance undercut with a mechanical undercutting system

The undercutting front advance with drill and blast is discontinuous. In contrast, a mechanical undercutting system similar to a “Longwall Shearer System”, gives continuous cutting of the rock. The coal excavation rate depends on the penetration capacity of the cutting element and the cutting velocity.
To estimate the productivity for the mechanical system, it was necessary to assume some design aspect. It was assumed that the cutting modality is performed by a drum miner system with mini discs of 5 inches, as is shown in Figure 3.8. The cutting parameters are presented in Table 3.3, and operational parameters were the same as used in the drill and blast estimate. For calculating the cutting performance, the methodology developed for Tunnel Boring Machine was used (Rostami, 2008). Forces in the cutting front are estimate by:

\[
F_t = CTR\phi \sqrt{\frac{\sigma_c}{\sigma_t} S} \sqrt{RT}
\]  

(3.1)

Where:

- \(F_t\) = Total forces (N or pounds)
- \(C\) = Constant equal to 2.12
- \(T\) = Cutter tip width (millimeters or inches)
- \(R\) = Cutter radius
- \(\sigma_c\) = Uniaxial compressive strength of rock (UCS) (Mpa or psi)
- \(\sigma_t\) = Uniaxial traction strength of rock (UCS) (Mpa or psi)
- \(S\) = Cutter spacing (millimeters or inches)
- \(\phi\) = Angle of the contact area estimated as:

\[
\phi = \cos^{-1}\left(\frac{R - p}{R}\right)
\]  

(3.2)

- \(p\) = Cutter penetration (millimeters or inches)

Individual cutting forces (normal force \(F_N\) and rolling force \(F_R\)) can be estimated by:

\[
F_N = Ft \cos (\beta)
\]  

(3.3)

\[
F_R = Ft \sin (\beta)
\]  

(3.4)

Where \(\beta = \phi/2\) and the cutting/rolling “RC” is the coefficient, which is the ratio of the rolling to normal force.
Table 3.3 Parameters for estimating the productivity of a mechanical system with different widths of the undercutting front.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>120 m</th>
<th>150 m</th>
<th>180 m</th>
</tr>
</thead>
<tbody>
<tr>
<td>Width of the drum</td>
<td>m</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
</tr>
<tr>
<td>Diameter of the drum</td>
<td>m</td>
<td>2.00</td>
<td>2.00</td>
<td>2.00</td>
</tr>
<tr>
<td>Diameter of the disc cutters</td>
<td>m</td>
<td>0.13</td>
<td>0.13</td>
<td>0.13</td>
</tr>
<tr>
<td>Spacing</td>
<td>m</td>
<td>0.05</td>
<td>0.05</td>
<td>0.05</td>
</tr>
<tr>
<td>Number of dresses</td>
<td></td>
<td>2.00</td>
<td>2.00</td>
<td>2.00</td>
</tr>
<tr>
<td>Maximum cutting speed</td>
<td>m/s</td>
<td>0.82</td>
<td>0.82</td>
<td>0.82</td>
</tr>
<tr>
<td>Tip Width</td>
<td>m</td>
<td>0.01</td>
<td>0.01</td>
<td>0.01</td>
</tr>
<tr>
<td>Uniaxial compressive strength</td>
<td>MPa</td>
<td>133.20</td>
<td>133.20</td>
<td>133.20</td>
</tr>
<tr>
<td>Brazilian tensile strength</td>
<td>MPa</td>
<td>8.88</td>
<td>8.88</td>
<td>8.88</td>
</tr>
<tr>
<td>Width of panel</td>
<td>m</td>
<td>120.00</td>
<td>150.00</td>
<td>180.00</td>
</tr>
<tr>
<td>Length of panel to undercutting</td>
<td>m</td>
<td>220.00</td>
<td>220.00</td>
<td>220.00</td>
</tr>
<tr>
<td>Number of cutters</td>
<td></td>
<td>40.00</td>
<td>40.00</td>
<td>40.00</td>
</tr>
<tr>
<td>Maximum rotation speed of drum</td>
<td>1/min</td>
<td>7.86</td>
<td>7.86</td>
<td>7.86</td>
</tr>
<tr>
<td>Penetration rate</td>
<td>mm</td>
<td>10.00</td>
<td>10.00</td>
<td>10.00</td>
</tr>
<tr>
<td>Contact angle</td>
<td>rad</td>
<td>0.57</td>
<td>0.57</td>
<td>0.57</td>
</tr>
<tr>
<td>Average Contact Pressure</td>
<td>MPa</td>
<td>173.65</td>
<td>173.65</td>
<td>173.65</td>
</tr>
<tr>
<td>Normal Force</td>
<td>kN</td>
<td>60.21</td>
<td>60.21</td>
<td>60.21</td>
</tr>
<tr>
<td>Rolling Force</td>
<td>kN</td>
<td>17.60</td>
<td>17.60</td>
<td>17.60</td>
</tr>
<tr>
<td>Total thrust</td>
<td>kN</td>
<td>1,204.11</td>
<td>1,204.11</td>
<td>1,204.11</td>
</tr>
<tr>
<td>Total torque</td>
<td>kNm</td>
<td>352.02</td>
<td>352.02</td>
<td>352.02</td>
</tr>
<tr>
<td>Required cutterhead power</td>
<td>kW</td>
<td>289.77</td>
<td>289.77</td>
<td>289.77</td>
</tr>
<tr>
<td>Maximum traverse speed</td>
<td>m/s</td>
<td>0.0026</td>
<td>0.0026</td>
<td>0.0026</td>
</tr>
<tr>
<td>Production rate</td>
<td>m³/h</td>
<td>18.87</td>
<td>18.87</td>
<td>18.87</td>
</tr>
<tr>
<td>Time to traverse</td>
<td>hours</td>
<td>12.72</td>
<td>15.90</td>
<td>19.08</td>
</tr>
<tr>
<td>Total time to cut all of the panel</td>
<td>hours</td>
<td>2,798.81</td>
<td>3,498.51</td>
<td>4,198.21</td>
</tr>
<tr>
<td>Volum to cut</td>
<td>m³</td>
<td>52,800.00</td>
<td>66,000.00</td>
<td>79,200.00</td>
</tr>
</tbody>
</table>

Rostami (2008) points out that “This formula is based on the statistical analysis of a database of cutting forces developed by full scale linear cutting test of disc cutters in various rock types. The database included cutters with diameter ranging from 125 to 480 mm and rocks with UCS ranging from 50 to over 250 Mpa.” This range of rock strength (UCS) is within the typical ranges for rocks where block caving is used.

The estimate shows: an average undercutting front advance of 1.04 meters/day; a muck generation, as a result of the cutting, of 49 tons/hour; a velocity of undercutting of 18.87
m³/hour, and an equipment utilization of 13.31 hours/day (these estimates are for an undercutting front of 120 meters). These results could be considered extremely low for a longwall system in a coal mine, but for block caving, where undercutting is considered, these results are reasonable.

In this estimate, when the width of the undercutting front increases, the time required to move the muck increases, as do break down times. For this reason, the performance (m/day) decreases as the width of the undercutting front increases Table 3.4.

![Figure 3.8 Concept of drum miner (Frenzel, 2011)](image)

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>120 m</th>
<th>150 m</th>
<th>180 m</th>
</tr>
</thead>
<tbody>
<tr>
<td>Re-set shearer</td>
<td>hours</td>
<td>0.02</td>
<td>0.02</td>
<td>0.02</td>
</tr>
<tr>
<td>Availability</td>
<td>%</td>
<td>0.90</td>
<td>0.90</td>
<td>0.90</td>
</tr>
<tr>
<td>Break downs</td>
<td>hours</td>
<td>1.27</td>
<td>1.59</td>
<td>1.91</td>
</tr>
<tr>
<td>Shift changes</td>
<td>hours</td>
<td>2.00</td>
<td>2.00</td>
<td>2.00</td>
</tr>
<tr>
<td>Maintenance time</td>
<td>hours</td>
<td>3.00</td>
<td>3.00</td>
<td>3.00</td>
</tr>
<tr>
<td>Change of dics</td>
<td>hours</td>
<td>2.00</td>
<td>2.00</td>
<td>2.00</td>
</tr>
<tr>
<td>Number of travels</td>
<td></td>
<td>220.00</td>
<td>220.00</td>
<td>220.00</td>
</tr>
<tr>
<td>Hour day</td>
<td>hours</td>
<td>13.29</td>
<td>12.97</td>
<td>12.65</td>
</tr>
<tr>
<td>Days to cut panel</td>
<td>days</td>
<td>210.63</td>
<td>269.74</td>
<td>331.83</td>
</tr>
<tr>
<td>Number of months</td>
<td>moths</td>
<td>8.78</td>
<td>11.24</td>
<td>13.83</td>
</tr>
<tr>
<td>Performance per month</td>
<td>m²/month</td>
<td>3,008.12</td>
<td>2,936.12</td>
<td>2,864.12</td>
</tr>
<tr>
<td>Muck pre hour</td>
<td>ton</td>
<td>49</td>
<td>49</td>
<td>49</td>
</tr>
<tr>
<td>Muck per day</td>
<td>ton</td>
<td>657</td>
<td>641</td>
<td>625</td>
</tr>
<tr>
<td>Advance per day</td>
<td>m/day</td>
<td>1.04</td>
<td>0.82</td>
<td>0.66</td>
</tr>
</tbody>
</table>

Table 3.4 Estimate of productivities for different widths of undercut fronts.
Based on this analysis is possible to conclude that a mechanical undercutting system has possibility to give productivity highest than the classical undercutting system, because the estimation was done with conservative parameters.

3.4 Possible differences between the use of a mechanical undercutting system in block caving and longwall

Some differences between the longwall mining of coal and the proposed mechanical undercutting system for block caving include:

(1) The mechanical undercutting system in block caving is not a production method. Therefore, the material removed is only the material necessary to continue with the undercut. The simple estimate of muck is present in Table 3.4 or a panel width of 120 meters is 49 tons/hour, compared with the capacity of an armored face conveyor (AFC) in longwall, with current capacities between 4,000 and 7,000 tons/hour (Peng, 2006).

(2) An eventual mechanical undercutting system in metallic mines does not have the problem of methane concentration; therefore, it does not need the strict control and monitoring of the gas concentration that is normally very important in coal mines. Requirements for ventilation should be different.

(3) The convergence between the roof and floor by effect of the undercut should be different in block caving, where the rock is hard and the natural fracturing is sub-vertical. In this case, the instability could have a structural control.

(4) In the case of mechanical undercutting in block caving, it would probably be convenient to put part of the material (muck) from cutting front in the beam, behind the support shield in order to control the risk of a possible air blast.

(5) In the case of an undercut method typo advance undercut with a mechanical undercutting system, the incorporation of the drawbells and the length of the beam change the caving geometry. Therefore, this would affect the displacement in the work area and the load over the support system, which means that the plan and sequence of the incorporation of the drawbells should be controlled and evaluated constantly. This condition does not exist in longwall, nor does its possible effect in the cutting front.
CHAPTER 4

GEOMECHANICAL HAZARDS IN THE USE OF A MECHANICAL UNDERCUTTING SYSTEM

4.1 Methodology used for risk analysis

In this work, the evaluation of the geomechanical risks during a potential application of a mechanical undercutting system was based on Chapter 11, “Risk Assessment for Block Caving” in Brown’s Block Caving Geomechanics (2007). This methodology considers the likelihood that an event will occur and the consequence of the event. Three tables were developed to evaluate the risk: probability of occurrence / likelihood (Table 4.1), possible impact / consequence (Table 4.2), and a matrix of identification of significant risks (Table 4.3).

The author of this thesis, with the support of his adviser, described which could be the possible hazards and identified the geomechanics stability problems that could occur as a result of those problems. They then considered the consequences of those events and estimated, according to their opinion, what could be the value for each parameter.

<table>
<thead>
<tr>
<th>Level</th>
<th>Measurement</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Unlikely</td>
</tr>
<tr>
<td>2</td>
<td>Possible</td>
</tr>
<tr>
<td>3</td>
<td>Probable</td>
</tr>
<tr>
<td>4</td>
<td>Very Probable</td>
</tr>
</tbody>
</table>

Table 4.1 Matrix of evaluation of probability of occurrence (likelihood)

<table>
<thead>
<tr>
<th>Level</th>
<th>Measurement</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Low</td>
</tr>
<tr>
<td>2</td>
<td>Medium</td>
</tr>
<tr>
<td>3</td>
<td>High</td>
</tr>
<tr>
<td>4</td>
<td>Very High</td>
</tr>
</tbody>
</table>

Table 4.2 Matrix of evaluation of impact (consequence)
### Table 4.3  Matrix to identify the significant risks of the project

<table>
<thead>
<tr>
<th>Probability of occurrence (Likelihood)</th>
<th>Very Low</th>
<th>Low</th>
<th>Moderate</th>
<th>High</th>
</tr>
</thead>
<tbody>
<tr>
<td>Very Unlikely</td>
<td>1</td>
<td>2</td>
<td>3</td>
<td>4</td>
</tr>
<tr>
<td>Unlikely</td>
<td>2</td>
<td>4</td>
<td>6</td>
<td>8</td>
</tr>
<tr>
<td>Probable</td>
<td>3</td>
<td>6</td>
<td>9</td>
<td>12</td>
</tr>
<tr>
<td>Highly Likely</td>
<td>4</td>
<td>8</td>
<td>12</td>
<td>16</td>
</tr>
</tbody>
</table>

This matrix is used to identify the possible significant geomechanical risks during the use of a mechanical undercutting system (Table 4.3). The green color means a low risk, yellow and orange indicate a medium risk, and red indicates a high risk.

### 4.2 Description of hazard

In order to better understand the undercutting process with a mechanical undercutting system and the geomechanical problems (stability problems) that could occur during the advance of the undercutting front, a list of possible geomechanical hazards was proposed for each zone and the possible operational problems and consequences associated with those hazards were described. In addition, Table 4.4 was drawn for every hazard, in order to evaluate the likelihood of the hazard arising and the consequences if it did.

#### 4.2.1 Cutting front

- **Wedges in face** (Problem A, Table 4.4)

  A combination of stress condition and geological structural domain may cause wedges. Depending on their size, wedges should be reduced or removed from the undercutting front.

  **Likelihood:**
  - Free blocks in the front
  - Irregularity in the cutting face

  **Consequences:**
  - Stoppage of the undercutting operation
- Delay in the undercutting schedule
- Damage to equipment
- Need of extra support
- Need of moving blocks
- Possible necessity of drilling and blasting blocks

- A geological structure parallel to the undercutting front (Problem B, Table 4.4)

This condition could create a pillar between the undercutting front and the geological structure, which would be the thinnest at the advance of the undercutting front, creating the possibility that it could accumulate high stress and cause stability problems.

Likelihood:
- Collapse of cutting front
- Rock burst in the cutting front
- Irregularity in the cutting face

Consequences:
- Stoppage of the undercutting operation
- Delay in the undercutting schedule
- Injury of people or damage to equipment
- Need of extra support
- Need of moving excessive muck
- Difficulty restarting the cutting process

- High stress (Problem C, Table 4.4)

With hard rock that has high stiffness and low fracture frequency and contains structures or weaknesses, high stress could provoke a sudden release of energy that could result in damage to the mining infrastructure.

Likelihood:
- Overbreak in the cutting front
- Rock burst in the cutting front
Irregularity in the cutting face

Consequences:
- Stoppage of the undercutting operation
- Delay in the undercutting schedule
- Injury of people or damage to equipment
- Need of extra support
- Need of moving excessive muck
- Difficulty in restarting the cutting process

- Loss of confinement in the center of the cutting front (Problem D, Table 4.4)

For a straight and long undercutting front (over 200 meters), experience (Araneda, 2007) indicates that—depending on the stress condition—there is a high probability of generating traction forces in the center of the undercutting front, provoking damage in the center of the cutting front and affecting the cutting advance.

Likelihood:
- Geometric curve in the cutting face, difficult for machine operation
- Irregularity in the cutting face

Consequences:
- Problems with the operation of the machine
- Delay in the undercutting schedule
- Need of extra support
- Need of moving excessive muck

4.2.2 Work area

The work area will be the most important sector (Figure 3.6 zone f) for the performance of the undercutting activities, because workers will be located in this sector and the cutting machine, muck system, and roof support will work here.
-  *Cavities* (Problem E, Table 4.4)

A weak zone (geological zone with a high fracturing), such as a dyke, vein, old shaft, or ore pass, could form a cavity over the roof support.

Likelihood:
- Lack of contact surface for hydraulic roof support
- Irregular roofs

Consequences:
- Stoppage of the cutting front advance
- Operational problems to roof support
- Delay in the undercutting schedule
- Need of filling cavities
- Need of moving excessive muck
- Injury of people or damage equipment

-  *Wedge in roof* (Problem F, Table 4.4)

There are two different conditions in which wedges in the roof could be generated: during pre-caving and/or continuous caving. Each condition has different caving back geometry and different possibilities of wedge mobilization.

Likelihood:
- Wedges falling from the roof trapping the roof support
- Irregular roofs

Consequences:
- Stoppage of the cutting front advance
- Operational problems to roof support
- Delay in the undercutting schedule
- Need of extra support
- Need of moving excessive muck
- Need of drill and blast to liberate hydraulic support
- Injury of people or damage to equipment

4.2.3 Drifts access to panel

The mechanical undercutting system will need a few accesses for entrance to the machine, ventilation, and handling of muck. For this reason, access is an important part of the undercutting mine design.

- **Collapse** (Problem G, Table 4.4)

  It is defined as the loss of functionality of the drift (e.g., it is not possible to use the drift because the reduction of the drift section does not allow equipment to be moved or makes it risky to move people).

  Likelihood:
  - Excessive deformation
  - Important damage for rockburst
  - Fall of wedges in intersection of drifts

  Consequences:
  - Stoppage of the undercutting front advance
  - Loss of area due to abandonment
  - Delay in the undercutting schedule
  - Need of extra support
  - Loss of equipment
  - Need of extra development to restart the process

- **Overbreak** (Problem H, Table 4.4)

  A particular stress condition (normally induced stress in the contour of the drift, of a magnitude which is close to the rock mass strength) that could provoke an overbreak in the drifts that the original support design could not control.

  Likelihood:
o Excessive deformation
o Fall down of rock

Consequences:
o Stoppage of cutting front advance
o Delay in the undercutting schedule
o Need of extra support

4.2.4 Pillars of separation between panels

- Collapse of pillar (Problem I, Table 4.4)
The failure of the pillar limits access to the undercutting front.

Likelihood:
o Collapse of the pillar
o Transference of load to bottom levels

Consequences:
o Stoppage of the undercutting front advance
o Delay in the undercutting schedule
o Need of extra support
o Possible decrease in production levels
o Problems in the caving propagation for the next macro block
o Change in the macro sequence
o Change in the mine plan
o Change in the global geometry of the caving

- High damage or excessive deformation (Problem J, Table 4.4)
High damage and excessive deformation are defined as the loss of the original geometry of the drifts.

Likelihood:
- Collapse of the pillar
- Transference of load to bottom levels

Consequences:
- Stoppage of the undercutting front advance
- Delay in the undercutting schedule
- Need of extra support
- Decreased pillar strength

- Remnant pillar for poor connection to caving (Problem K, Table 4.4)

Because it is necessary to avoid the possible transfer of load from the undercut level to the production level, it is necessary to eliminate the pillar between macro blocks from the production level, and incorporate this volume into the mine production. This activity should be accomplished with drill and blast through the drawbells in the production level.

Likelihood:
- Major collapse in production level
- Seismic activity

Consequences:
- Loss of production area
- Ventilation problems
- Loss of production
- Changes in the production schedule

4.2.5 Caving volume

The geometry and the velocity of the growth of the caving back have an effect on the abutment stress around the undercutting area and may generate some geomechanical stability problems.

Initiation or re-initiation of caving process (Problem L, Table 4.4)
When the undercutting starts, the undercutting area is not enough to initiate the caving, and the roof of the cavity is stable (pre-caving). As the area continues to increase, the roof will eventually fall and generate air movement and redistribution of stress around the caving area. During the caving the growth process could be stopped and create a stable arch and re-start and generate the same problems (Brown, 2007).

Likelihood:
- Air blast
- Rock bursts

Consequences:
- Stoppage of the undercutting front advance
- Delay in the undercutting schedule
- Need of extra support
- Injury of people or damage to equipment

*Seismicity of high energy* (Problem M, Table 4.4)

The caving process generates seismicity, associated with caving growth. This may generate rockbursts or cause wedges to fall down the cutting front.

Likelihood:
- Rock bursts
- Fall down of wedges

Consequences:
- Stoppage of the undercutting front advance
- Delay in the undercutting schedule
- Need of extra support
- Injury of people or damage to equipment

*Movement of big wedges* (Problem N, Table 4.4)
Because the mechanical undercutting system should cause less damage than drill and blast, this could affect the initial caving fragmentation, given the possibility of a coarse fragment of rock that could affect the flow of mineral through of the drawbells and therefore affect the production level.

Likelihood:
- Collapses
- Decreased production level

Consequences:
- Stoppage of the undercutting front advance
- Delay in the undercutting schedule
- Need of rehabilitation
- Production losses

Based on the descriptions detailed above, each problem was illustrated to facilitate understanding of the condition inherent to each problem and its evaluation (Table 4.4).

Table 4.4 Identification of geomechanical hazards during the undercutting process using a mechanical undercutting system

<table>
<thead>
<tr>
<th>Locate</th>
<th>Problem</th>
<th>Type</th>
<th>Sketch</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cutting front</td>
<td>Wedge in face</td>
<td>Toppling</td>
<td>![Advance Direction](Section View)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Slide</td>
<td>![Advance Direction](Section View)</td>
</tr>
</tbody>
</table>
Table 4.4 Continued

<table>
<thead>
<tr>
<th>Locate</th>
<th>Problem</th>
<th>Type</th>
<th>Sketch</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cutting front</td>
<td>A geological structure parallel to the undercutting front</td>
<td>Fault, dyke, or lithological contact</td>
<td>![Sketch of geological structure and undercutting front]</td>
</tr>
<tr>
<td>High stress</td>
<td>Overbreak</td>
<td></td>
<td>![Sketch of overbreak]</td>
</tr>
<tr>
<td></td>
<td>Spalling or popping</td>
<td></td>
<td>![Sketch of spalling or popping]</td>
</tr>
<tr>
<td></td>
<td>Rockburst</td>
<td></td>
<td>![Sketch of rockburst]</td>
</tr>
<tr>
<td>Locate</td>
<td>Problem</td>
<td>Type</td>
<td>Sketch</td>
</tr>
<tr>
<td>--------</td>
<td>---------</td>
<td>---------------</td>
<td>--------</td>
</tr>
<tr>
<td></td>
<td>Loss of confinement in the center of cutting front</td>
<td>Overbreak</td>
<td><img src="image" alt="Diagram of Overbreak" /></td>
</tr>
<tr>
<td></td>
<td>Cavities in the cutting work area</td>
<td>Overbreak</td>
<td><img src="image" alt="Diagram of Overbreak" /></td>
</tr>
<tr>
<td>Cutting work area</td>
<td></td>
<td>Pre-caving</td>
<td><img src="image" alt="Diagram of Pre-caving" /></td>
</tr>
<tr>
<td></td>
<td>Wedge in roof</td>
<td>Pre-caving</td>
<td><img src="image" alt="Diagram of Pre-caving" /></td>
</tr>
<tr>
<td>Drifts access to cutting front</td>
<td>Collapse</td>
<td>Pillar collapse</td>
<td><img src="image" alt="Diagram of Pillar collapse" /></td>
</tr>
<tr>
<td>Locate</td>
<td>Problem</td>
<td>Type</td>
<td>Sketch</td>
</tr>
<tr>
<td>-----------------------------</td>
<td>----------------------------------------------</td>
<td>---------------</td>
<td>---------------------------------------------</td>
</tr>
<tr>
<td>Overbreak</td>
<td>High stress</td>
<td></td>
<td><img src="image" alt="Sketch of Overbreak" /></td>
</tr>
<tr>
<td>Collapse of pillar</td>
<td>Over stress</td>
<td></td>
<td><img src="image" alt="Sketch of Collapse" /></td>
</tr>
<tr>
<td>Pillar between cutting panels</td>
<td>Damage of pillar</td>
<td>Stress condition</td>
<td><img src="image" alt="Sketch of Damage" /></td>
</tr>
<tr>
<td>Remnant pillar for poor connection to caving</td>
<td>Difficult to blast from production level</td>
<td></td>
<td><img src="image" alt="Sketch of Remnant" /></td>
</tr>
<tr>
<td>Caving volume</td>
<td>Initiation or re-initiation of caving process</td>
<td>Air blast</td>
<td><img src="image" alt="Sketch of Caving" /></td>
</tr>
</tbody>
</table>
4.3 Evaluation of risks

Table 4.5 summarizes the risk assessment (likelihood multiplied by consequence) of each of the risks described above. The last column indicates if the risk is new in block caving.

\[ Risk = \text{Likelihood} \times \text{Consequence} \]

(Brown, 2007)

Table 4.5 Matrix of risk assessment

<table>
<thead>
<tr>
<th>Locate</th>
<th>Problem</th>
<th>Type</th>
<th>Likelihood</th>
<th>Consequence</th>
<th>Risk</th>
<th>New</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cutting front</td>
<td>Wedge in face</td>
<td>Toppling</td>
<td>2</td>
<td>1</td>
<td>2</td>
<td>Yes</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Slide</td>
<td>3</td>
<td>1</td>
<td>3</td>
<td>Yes</td>
</tr>
<tr>
<td></td>
<td>A structure parallel to the</td>
<td>Fault, dyke or lithological contact.</td>
<td>2</td>
<td>3</td>
<td>6</td>
<td>Yes</td>
</tr>
<tr>
<td></td>
<td>undercutting front</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>High stress</td>
<td>Overbreak</td>
<td>2</td>
<td>2</td>
<td>4</td>
<td>Yes</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Spalling or popping</td>
<td>3</td>
<td>2</td>
<td>6</td>
<td>Yes</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Rockburst</td>
<td>2</td>
<td>3</td>
<td>6</td>
<td>Yes</td>
</tr>
</tbody>
</table>

Seismicity of high energy

Moving of big wedges

Table 4.4 Continued

<table>
<thead>
<tr>
<th>Locate</th>
<th>Problem</th>
<th>Type</th>
<th>Sketch</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Seismicity of high energy</td>
<td>Rockburst</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Moving of big wedges</td>
<td>Collapses of tunnels</td>
<td></td>
</tr>
<tr>
<td>Location</td>
<td>Problem</td>
<td>Type</td>
<td>Likelihood</td>
</tr>
<tr>
<td>---------------------------</td>
<td>------------------------------------------------------------------------</td>
<td>--------------------</td>
<td>------------</td>
</tr>
<tr>
<td>Cutting front</td>
<td>Loss of confinement in the center of the cutting front</td>
<td>Overbreak</td>
<td>3</td>
</tr>
<tr>
<td>Work area</td>
<td>Wedge in roof</td>
<td>Structural</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Cavities in the work area</td>
<td>Dyke, vein, old shaft, or ore pass</td>
<td>2</td>
</tr>
<tr>
<td>Tunnel access to cutting front</td>
<td>Collapse</td>
<td>Pillar collapse</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Overbreak</td>
<td>High stress</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Collapse of pillar</td>
<td>Over stress</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Damage of pillar</td>
<td>Stress condition</td>
<td>3</td>
</tr>
<tr>
<td>Pillar between panels</td>
<td>Remnant pillar for poor connection to caving</td>
<td>Difficult to blast from production level</td>
<td>3</td>
</tr>
<tr>
<td></td>
<td>Remnant pillar for poor connection to caving</td>
<td>Difficult to blast from production level</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Difficult to start over undercutting process</td>
<td>Loss of initial tunnel</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Pillars burst</td>
<td>Rockburst</td>
<td>2</td>
</tr>
<tr>
<td>Caving back</td>
<td>Initiation or re-initiation of caving process</td>
<td>Air blast</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>Seismic of high energy</td>
<td>Rockburst</td>
<td>2</td>
</tr>
</tbody>
</table>
The result of the risk assessment shows that three risk problems during the undercutting process could be:

(a) Loss of confinement in the center of the undercutting front
(b) Wedge in roof
(c) Collapse of pillar between panels

The first two problems occurred in the work area and the cutting front, and the pillar that collapsed was typically the pillar between panels. These three problems have a relationship with mean (a) stress analysis of the work area (include the cutting front), (b) kinematic structural analysis of the work area and (c) pillar strength. Therefore, the analyses were focused on three main zones (the cutting front, the work area and the pillar between panels), by way of stress/strain analysis and kinematic estimations.
CHAPTER 5
NUMERICAL MODELS

5.1 Framework of numerical models

The risk analysis in Chapter 4 identified three zones of high risk: (1) the undercutting front, (2) the work area, and (3) the pillar between panels. These zones are related to *losses of confinement in the center of the panel, creation of wedges in the roof of the work area, and collapse of the pillar between panels*. These risks are associated with damage to the rock mass that were investigated through stress/strain analyses through numerical models focusing on the zones above mentioned. The stress/strain analyses were divided into two stages based on the zones defined by Laubscher (Laubscher, 1994): (1) pre-caving and (2) continuous caving. A total of 14 numerical models were built (Table 5.4 and Table 5.5) with one and two panels.

These analyses will give an idea about of the possible behavior of the rock mass for effect of undercutting process. The literature review and the block caving model indicate that is necessary a tridimensional study. The validation of numerical models has been done comparing the result with information obtained from literature review.

The information about the rock properties (Russo, 2010) and stress condition (Board, 2009) corresponds to the data from the Chuquicamata Underground Project, specifically the sectors of macro blocks called N42 and N52 (Chapter 2).

5.1.1 Numerical models objectives

The numerical models were built with the following objectives:

(a) Evaluate the possible behavior of block caving mine design proposed for a mechanical undercutting system (Chapter 3).

(b) Evaluate the possible behavior of the risks zones, identified in Chapter 4.

(c) Estimate the variation of stress and strains in the undercutting front due to of undercutting advance.

(d) Estimate the initiation and depth of the damage in the undercutting front.

(e) Estimate the deformation in the roof of the work area.

(f) Evaluate the possible effect of the undercutting front width in the stress distribution.

(g) Evaluate the possible effect of the undercutting front beam in the stress distribution.
Evaluate the strength of the pillar between panels.

5.1.2 Numerical model software

Nowadays, there are different rock mechanics software that allows the modeling of the undercut advance and the caving progress, these models are only an approach to reality, which allow to have an idea about the geotechnical behavior and mining options. The most used three dimension software in the mining industry include the Itasca Consulting Group pack (FLAC, FLAC\(^3\)D, PFC\(^2\)D, PFC\(^3\)D, UDEC and, 3DEC), Abaqus FEA and MAP 3D.

Due to the experience in the numerical model of caving that was gained in the “Caving Study” project conducted from 1997 to 2007 (Brown, 2007), as well as the availability of software and the capacity of computers, this project used FLAC\(^3\)D. The main reason for using this software was because the professionals that took part on the project had extensive experience in this software, which allowed validating the hypothesis of caving propagation (Sainsbury, 2010). In addition, caving and longwall studies have used FLAC\(^3\)D.

5.1.3 Caving condition

The first big issue in the life of a block caving is when the undercutting area reached the hydraulic radius of caving (hydraulic radius is calculated dividing the area by the perimeter of undercut zone). The hydraulic radius of caving is the hydraulic radius required to ensure propagation of the caving when the rock is extracted through the production level, without stopping the caving progress (Laubscher, 2003). An empirical estimation was defined by Laubscher and take into account the ratio between MRMR (Mining Rock Mass Rating) and hydraulic radius (HR), where MRMR is the mining evaluation of the rock mass quality (Laubscher, 2001) and defined two main conditions. The *pre-caving*, which corresponds to the initial step of the undercutting, where the roof of the undercutting area is flat and the production have not yet begun. The *Continuous caving* is the stage in which the caving is propagated as a result of the production draw (Figure 5.1). Simulating the initiation of the caving process (hydraulic radius of caving) through numerical models is a complex task.
5.2 Modeling with FLAC$^{3D}$

In this thesis, the numerical models were made with FLAC$^{3D}$. This software is widely used in civil and mining engineering with significant studies in longwall and block caving. The following are some features available in FLAC$^{3D}$ that are important for modeling the undercutting advance in block caving (Board, 2013).

- The capability to represent three-dimensional mining geometries.
- The ability to model sequential mining with geometry changes.
- The possibility to model yield zones and the degradation of the rock mass associated with stress redistribution and deformation. It is important to study the possible damage in the undercutting front of the work and investigate the behavior of the rock mass by the undercut advance.
- The ability to model rock as a material with a peak and residual strength (strain softening).
- The ability to represent the effect of gravity as well as in situ stress.
FLAC\textsuperscript{3D} is “an explicit finite difference program to study, numerically, the mechanical behavior of a continuous three-dimensional medium as it reaches equilibrium or steady plastic flow” (Itasca, 2012). This geomechanics software solves equations of solid mechanics, using the finite differences that allow simulating elastic-plastic behavior in soil and rock to study geomechanical problems beyond rupture limit in the plastic zone, allowing big displacements. The basic cycle of work is showed in Figure 5.2 and each cycle is a temporary solution. FLAC\textsuperscript{3D} incorporates an important aspect in the numerical model, the full dynamic equations of motion even for quasi-static problems. FLAC\textsuperscript{3D} is especially useful to study problems over the peak strength, with a plastic behavior such as caving process and pillars collapse. The software uses a programming language called FISH which allows incorporating particular analyses and constitutive laws.

![Basic Explicit Calculation Cycle](image)

Figure 5.2 The basic calculation cycle of FLAC\textsuperscript{3D} (Itasca, 2012)

In FLAC\textsuperscript{3D} (taken from FLAC\textsuperscript{3D} version 5 manual 2012) the state of stress at a given point of the medium is characterized by the symmetric stress tensor $\sigma_{ij}$. The traction vector $[t]$ on a face with unit normal $[n]$ is given by Cauchy’s formulae (tension positive):

$$t_i = \sigma_{ij}n_j$$  \hspace{1cm} (5.1)
Let the particles of the medium move with velocity \([\mathbf{v}]\). In an infinitesimal time, \(dt\), the medium experiences an infinitesimal strain determined by the translations \(\mathbf{v}_i dt\), and the corresponding components of the strain-rate tensor may be written as:

\[
\varepsilon_{ij} = \frac{1}{2} (\mathbf{v}_{i,j} + \mathbf{v}_{j,i}) \tag{5.2}
\]

where partial derivatives are taken with respect to components of the current position vector \([x]\).

Aside from the rate of deformation characterized by the tensor \(\xi_{ij}\), a volume element experiences an instantaneous rigid-body displacement, determined by the translation velocity \([\mathbf{v}]\), and a rotation with angular velocity \((\Omega)\).

\[
\Omega_l = -\varepsilon_{ijk} \omega_{jk} \tag{5.3}
\]

where \(\varepsilon_{ijk}\) is the permutation symbol, and \(\omega\) is the rate of rotation tensor whose components are defined as

\[
\omega_{ij} = \frac{1}{2} (\mathbf{v}_{i,j} - \mathbf{v}_{j,i}) \tag{5.4}
\]

Application of the continuum form of the momentum principle yields Cauchy’s equations of motion:

\[
\sigma_{ij,j} + \mathbf{b}_i = \rho \frac{d\mathbf{v}_i}{dt} \tag{5.5}
\]

where \(\rho\) is the mass-per-unit volume of the medium, \(\mathbf{b}_i\) is the body force per unit mass, and \(\frac{d\mathbf{v}_i}{dt}\) is the material derivative of the velocity. These laws govern, in the mathematical model, the motion of an elementary volume of the medium from the forces applied to it. Note that, in the case of static equilibrium of the medium, the acceleration \(d\mathbf{v}/dt\) is zero, and reduces to the partial differential equations of equilibrium:
\[ \sigma_{ij,j} + \rho b_i = 0 \] 

(5.6)

In the formulation of the finite difference method, each derivative in the previous equations (motion and stress-strain) is replaced by an algebraic expression relating variables at specific locations in the grid (Itasca 2012, FLAC\textsuperscript{3D} Tutoring).

In addition to the law of motion, a continuous material must obey a constitutive relation; this is the relation between stress and strains.

In general it is:

\[ [\tilde{\sigma}_{ij}] = H(\sigma_{ij}, \varepsilon_{ij}, k) \] 

(5.7)

Where

\[ [\tilde{\sigma}_{ij}] = \text{Corotational stress rate tensor} \]

\( H = \text{It is a given function} \)

\( \varepsilon_{ij} = \text{strain rate tensor} \)

\( k = \text{parameter that takes into account the history of loading} \)

The algebraic expressions are fully explicit; all quantities on the right-hand side of the expressions are known. Consequently each element (zone or gridpoint) in a FLAC\textsuperscript{3D} appears to be physically isolated from its neighbors during one calculation timestep, solving their own equations.

To model in FLAC\textsuperscript{3D} is recommended to use the following sequence in Figure 5.3.

### 5.3 Modeling construction

Once the objectives and scope of the numerical modeling have been defined, the geometries to be modeled were also defined. In this case two types of geometry were used, one with one panel and another with two panels. Based on the conceptual methodology of numerical model process presented in the Figure 5.4, a process map was constructed for this work, where the first step of Pre-modeling construction is where the conceptual model is created, in the Pre-excavation are defined all the parameters of numerical models and the details to modeling to answer the question defined in the objectives of the numerical models. This is a recursive
methodology where the model tests in term of stress and the convergence of mathematical solution, logical and coherent result when the equilibrium is reached and the models converges, after this is possible to continue with the next step which is **Excavation sequence** where the zones to excavate in the planned sequence are defined. In this step the results must be checked regarding their logic and coherence with the conceptual model. During this work FISH functions were incorporate in order to estimate the depth of the fracturing in the undercutting front and understand the relation with the undercutting advance.

### 5.3.1 Numerical model geometry

As mentioned above two basic geometries of numerical models were built, one and two panel, for the first geometry the undercutting width and the length of the beam were tested, in continuous caving. On the other hand, for two panels the focus was in the pillar behavior. The geometries were based on the mechanical undercutting mine design (Chapter 3).

The first configuration designed was one panel limited for two drifts and two crosscuts (Figure 5.5). These models were used to study in detail the variation of the stress and deformations in the undercutting front and work area during the pre-caving and continuous caving stages, with the undercutting front advance. In the undercutting zone a fine mesh was constructed.

Three different undercutting widths were chosen: 120, 150 and 180 meters. Dimensions greater than 200 meters were not considered because the macro block concept is to have the undercutting front small and the production areas between 20,000 to 40,000 m² for easy management of preparation and production (Araneda, 2007). The increment of 30 meters in the widths was selected because the distance between the production drifts frequently is 30 meters, and this distance provide the possibility of incorporating a new production drift into the production system. In Figure 5.5 is showed a plan view of the numerical model mesh and Figure 5.6 an isometric of the numerical model.

Regarding the second numerical configuration, it was focused on the study of the pillar between panels (20 meters) to evaluate the stress behavior of the pillar during the undercutting front advance in the first panel (A), the second panel (B), and the pillar reduction. For the two-panel model, the same configuration as the one panel model was used. The only difference was that the finer mesh of finite differences was used on the pillar in order to give more precision to
the stability estimation. Figure 5.7 show the simple geometry modeled, in Figure 5.8 is showed the configuration of numerical models and in Figure 5.9 it is possible to see a section with the numerical model mesh.

![Diagram of numerical model process](image)

Figure 5.3 Numerical model process (Itasca, 2012)
Figure 5.4 Process map of numerical model in block caving undercutting (Modify from Itasca 2012).
Figure 5.5 Plan view of numerical model mesh for one panel

Figure 5.6 Configuration of model for one-panel FLAC$^{3D}$
Figure 5.7 Geometry of numerical models two panels

Figure 5.8 Configuration of model for two-panel FLAC$^{3D}$
5.3.2 Boundary conditions

The borders of the models were fixed to a distance beyond of three times the width of the undercutting front, far enough not to affect the stress behavior during the advance of the undercutting and the caving propagation. In the bottom face the displacements were fixed in all directions. North, south, east, and west walls were fixed in the x and y directions, but the z direction was left free. The upper face was left free in all directions. This allow that the gravity work (vertical direction) and limited the number of calculations, because if the model were infinite the solution of model could not converge (Itasca, 2012). Figure 5.10 is presents the border conditions in the numerical models.

5.3.3 Stress condition

The initial stress condition used in the numerical models was obtained from stress measurements in Chuquicamata Mine (Board, 2009) (see Table 5.1). This stress estimation considers the stress induced by the open pit and the effect of tectonic stress (induced by the subduction plate). Normally in Chile, pre-mining horizontal stress component is higher than the vertical stress for this effect. This estimation was calibrated and compared with the stress measured in the field (Board, 2009) in the “exploration tunnels” in the position of where the future panels would be located.
Table 5.1 Stress condition of macro block N-42 (see Chapter 2) level 1841

<table>
<thead>
<tr>
<th>( \sigma_{xx} ) (Mpa)</th>
<th>( \sigma_{yy} ) (Mpa)</th>
<th>( \sigma_{zz} ) (Mpa)</th>
<th>( K_{E-V} )</th>
<th>( K_{N-V} )</th>
</tr>
</thead>
<tbody>
<tr>
<td>33.91</td>
<td>23.2</td>
<td>18.24</td>
<td>1.86</td>
<td>1.27</td>
</tr>
</tbody>
</table>

Where

\( \sigma_{xx} = \) Horizontal stress north-south

\( \sigma_{yy} = \) Horizontal stress east-west

\( \sigma_{zz} = \) Vertical stress

\( K_{E-V} = \) Ratio stress east-west/vertical

\( K_{N-V} = \) Ratio stress north-south/vertical

Figure 5.10 Border conditions in the numerical models

Gravity was incorporated in the models and run to equilibrium (group of model pre-mining, Table 5.4) before the undercutting excavation was initiated.
5.3.4 Constitutive model

To consider the progressive failure of the rock mass, a strain softening constitutive model has been used because it represents in a more realistic way the degradation and failure of the rock mass regarding the post-peak strength (Figure 5.11). This type of constitutive model is commonly used in numerical models of longwall and block caving, because stress condition changes constantly with the advance of the undercutting and the caving progress, thereby causing constant degradation to the rock around the caving volume. This overall process of strain softening considers that the rock mass is loaded until a peak strength, followed by post-peak reduction in the strength to some residual level with the increasing strain. FLAC3D incorporates this effect by means of parameter reduction tables (this is the way that FLAC3D used to incorporate the geotechnical parameter in strain/softening (Itasca, 2012) such as cohesion, internal friction angle, tension strength, and dilatancy. This reduction is associated with a strain value, which is affected by the mesh size (Brown, 2007). According to FLAC3D manual version 5.0 “Hardening and softening behaviors for the cohesion, friction and dilation in terms of the shear parameter are provided by the user in the form of tables. Each table contains pairs of values: one for the parameter, and one for the corresponding property value. It is assumed that the property varies linearly between two consecutive parameter entries in the table. Softening of the tensile strength is described in a similar manner, using the parameter Δε_{pt}.” For this study, the values of these tables were the estimated through the relationship with the GSI parameter (Geological Strength Index, Hoek et al, 1998), which is incorporated in the software for the caving models.

5.3.5 Rock mass properties

In the numerical model the type of rock considered was PEK (Porphyry East Potassic) which corresponds to the rock type in the study area (Chapter 2). The set of properties correspond to the input of Chuquicamata underground project, Table 5.2 and Table 5.3, which were obtained from the laboratory tests (RocLab, 2012) in the case of intact rock and for the rock mass they were estimated basically from GSI.
Figure 5.11 Theoretical model of strain-softening (Cai, 2007)

Table 5.2 Properties of intact rock (Laboratory tests, Russo 2010)

<table>
<thead>
<tr>
<th>Geomechanics Parameters</th>
<th>PEK, Balmaceda</th>
</tr>
</thead>
<tbody>
<tr>
<td>Specific weight, ( \gamma ) (Ton/m(^3))</td>
<td>2.62</td>
</tr>
<tr>
<td>Uniaxial compression strength, UCS (( \sigma_{ci} )) (Mpa)</td>
<td>133.6</td>
</tr>
<tr>
<td>Tensile Strength, ( \tau_{ti} ) (Mpa)</td>
<td>-3.3</td>
</tr>
<tr>
<td>Parameter ( m_i ) (Hoek &amp; Brown, failure criterion)</td>
<td>14.8</td>
</tr>
<tr>
<td>Modulus of deformation, ( E_i ) (Gpa)</td>
<td>52.7</td>
</tr>
<tr>
<td>Poisson’s ratio of, ( \nu_i )</td>
<td>0.25</td>
</tr>
<tr>
<td>Bulk Modulus, ( K_i ) (Gpa)</td>
<td>35.13</td>
</tr>
<tr>
<td>Shear Modulus ( G_i ) (Gpa)</td>
<td>21.08</td>
</tr>
</tbody>
</table>
where; (Jeremic, 1987)

\[
k = \frac{E_i}{3(1-2\nu_i)} \quad (5.9)
\]

\[
G = \frac{E_i}{2(1+2\nu_i)} \quad (5.10)
\]

The properties of the rock mass were estimated through of the relation between the intact rock and the GSI (Geological Strength Index, Hoek (1998))

<table>
<thead>
<tr>
<th>Geomechanics Parameters</th>
<th>PEK, Balmaceda</th>
</tr>
</thead>
<tbody>
<tr>
<td>Geological Strength Index, GSI</td>
<td>59</td>
</tr>
<tr>
<td>Uniaxial compression strength rock mass, (\sigma_{\text{rm}}) (Mpa)</td>
<td>34.1</td>
</tr>
<tr>
<td>Modulus of deformation of rock mass, (E_{\text{rm}}) (Gpa)</td>
<td>27.5</td>
</tr>
<tr>
<td>Poisson’s ratio of rock mass, (\nu_{\text{rm}})</td>
<td>0.2</td>
</tr>
<tr>
<td>Parameter s (constant of Hoek &amp; Brown criterion)</td>
<td>0.01051</td>
</tr>
<tr>
<td>Value Hoek-Brown constant m for the rock mass, (m_{\text{b peak}})</td>
<td>3.42240</td>
</tr>
<tr>
<td>Tensile Strength rock mass, (\tau_{\text{rm}}) (Mpa)</td>
<td>-0.41</td>
</tr>
<tr>
<td>Bulk Modulus, (K_{\text{rm}}) (Gpa)</td>
<td>26.72</td>
</tr>
<tr>
<td>Shear Modulus, (G_{\text{rm}}) (Gpa)</td>
<td>16.032</td>
</tr>
</tbody>
</table>

### 5.3.6 Meshing and element size

The numerical models developed in this work were limited by the number of elements possible to consider during the run of the models. The size of the elements of the mesh of finite differences in the undercutting front, in the work area and in the pillar between panels were of 50 centimeters with a ratio of size x, y, and z of 1:1:1 that correspond to one quarter of the height of undercutting (2 meters). This mesh size was used in the study zone, out of this zone the mesh increased (from 50 cm to 64 meters). The expansion the mesh size was amplified twice, so that the connections of the nodes coincide (red circle, Figure 5.12).
5.4 Excavation sequence

The first step in the numerical models is to create the stress condition where the excavations will be done, once the in-situ stress models are run and checked, and coherent and logical results have been obtained, the excavation process starts.

5.4.1 One Panel

As mentioned above, the first group of numerical models is the undercutting of a panel, in stages, pre-caving and continuous caving, and the excavation begins with the access tunnels and crosscut tunnels. After this the undercutting advance started.

For the sequence of the undercutting front advance, the advance rate during the pre-caving was set up in steps of 20 meters until achieving 120 meters of span (where the hydraulic radius of caving is 30 meters) (see Figure 5.13). To choose the advance rate, different lengths of steps were tested, considering vertical displacement using isoSurface (a new tool of FLAC 3D version 5.0). These showed that a 20 meter advance provided enough data for the analyses, because only the first advance resulted in a significant jump in the vertical displacement. Nevertheless, the following step had a gradual increment. For this reason, an extra step of 10 meters was incorporated. This sequence results in a reasonable simulation of the behavior of the stress/strain in the undercutting front and the work area as a result of the undercutting front advance Figure 5.13 and Figure 5.14.

Figure 5.12 Increase in the mesh size.
In the numerical model, once the undercutting advance had reached the hydraulic radius of caving, it was necessary to take into consideration the caving progress, in particular the continuous caving stage. The progress of the caving was modeled gradually with a rate of caving growth of 20 meters. This meant simulating the extraction of 20 meters of material and incorporating that material into the caving volume, then running the model until equilibrium was reached Figure 5.15.

Figure 5.13 Model of undercutting sequence for one panel

Figure 5.14 Isosurface of vertical displacement to find the steps of undercut for the numerical model.
In this way, it was possible to obtain the stress and strain path, and also consider the degradation of the rock mass around the caving—which could affect the stress and deformation in the cutting front, work area, and pillar of separation. In addition, the rock incorporated into the caving volume was replaced by broken material without cohesion and with a low internal friction angle (Brown, 2007). Inside the model, this material only has the effect of its own weight and its confinement within the caving walls.

During the continuous caving, three different beam lengths (20, 40, and 60 meters) were modeled to estimate the effects of the distance between the cutting front and the caving, and to give an antecedent useful for designing a strategy of implementation for a mechanical undercutting system. For each distance, new numerical models were done. In addition, the largest height of the caving (into the model) was 143 meters. Limiting to this height ensured that the deformation reached the surface. The connection of the caving to the surface could have an effect on the stress condition in the pillar, because the geometry of the caving generated a caving subsidence that could modify the stress condition.

5.4.2 Two Panels

The geometry and the excavation sequence of each panel were the same. The objective was to evaluate the stability of the pillar, considering that it should support the entire undercutting front advance, the growth of the caving, and the connection to the surface for both panels Figure 5.16. In the sequence of the advance of the second panel, it was taken into consideration that the pillar was cut from the production level. The pillar cut began when the
undercutting front advance, in the second panel, a distance bigger than two drawbell lines (40 meters).

The same concepts used for modeling the one-panel caving propagation and rock mass degradation were used for the two-panel numerical model.

To evaluate the stability of the pillar between panels, two panels were built in the model (Panel A and Panel B). In these models the fine mesh was put in the pillar in order to show more detail in this zone (mesh was 50 centimeter-size, similar to the mesh in models used to study the undercutting front). The stress and strain in the pillar change with the advance of the mining activity for this model. The steps considered were:

1. Pre-caving Panel A,
2. Continuous caving Panel A and connection to surface,
3. Pre-caving Panel B with pillar reduction, and
4. Continuous caving panel B with pillar reduction and connection of Panel B to surface.

In this model, the width and the beam were fixed at 120 and 40 meters respectively.

In Table 5.4 and Table 5.5 summarized the numerical models, the codes are in Appendix A.
### Table 5.4 Consideration of numerical models

<table>
<thead>
<tr>
<th>Risk zones</th>
<th>Type of model</th>
<th>Caving stage</th>
<th>Width undercutting front (m)</th>
<th>Length beam (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cutting front and work area</td>
<td>One panel</td>
<td>Pre-caving</td>
<td>120</td>
<td>N/A</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>150</td>
<td>N/A</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>180</td>
<td>N/A</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Continuous caving</td>
<td>120</td>
<td>20</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>40</td>
</tr>
<tr>
<td></td>
<td>Two panels</td>
<td>Pre-caving Panel A</td>
<td>120</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Continuous caving Panel A</td>
<td>120</td>
<td>40</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Pre-caving Panel B</td>
<td>120</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Continuous caving Panel B</td>
<td>120</td>
<td>40</td>
</tr>
</tbody>
</table>

### Table 5.5 Detail of numerical models built

<table>
<thead>
<tr>
<th>Type of model</th>
<th>Caving stage</th>
<th>Pre-mining (file)</th>
<th>Advance undercutting (file)</th>
<th>Width undercutting front (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>One panel</td>
<td>Pre-caving</td>
<td>In situ_stress_Generic.dat</td>
<td>Advanc_precaving.dat</td>
<td>Avance_pre_120.dat</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Avance_pre_150.dat</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Avance_pre_180.dat</td>
</tr>
<tr>
<td></td>
<td>Continuous caving</td>
<td>In situ_stress.dat</td>
<td>Caving_20_120.dat</td>
<td>N/A</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Caving_40_120.dat</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Caving_60_120.dat</td>
<td></td>
</tr>
<tr>
<td>Two panels</td>
<td>Pre-caving Panel A</td>
<td></td>
<td>Advanc_precaving_panel_Pillar.dat</td>
<td>N/A</td>
</tr>
<tr>
<td></td>
<td>Continuous caving Panel A</td>
<td></td>
<td>Caving_40_120_Pillar.dat</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Pre-caving Panel B</td>
<td></td>
<td>Advanc_precaving_PanelB.dat</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Continuous caving Panel B</td>
<td></td>
<td>Caving_40_120_PillarB.dat</td>
<td></td>
</tr>
</tbody>
</table>
5.5 Criteria to estimate the initiation and depth of fracturing in the cutting front

As noted previously, one objective of the geomechanical evaluation was to understand the risk of loss of confinement in the center of the cutting front due to the advance of the undercutting front advance. To evaluate this risk, a couple of criteria related to initiation and extension (depth) of fracturing were used.

The criteria were chosen with consideration for the brittle rock. For this reason, a plotting of triaxial tests was checked in order to determine if the rock mass considered in the numerical models would have brittle behavior. Figure 5.17, shows a plot of all of the laboratory tests (blue points) over the Mogi line (the criterion used to classify the rock behavior, represented by the green line) (Roclab version 1.0, 2012).

Based on studies for tunnels (Diederichs, 2003 and Stacey, 1981) two criteria were chosen: one related to the uniaxial compression strength of intact rock (UCS$_{lab}$) and the other related to critical strain ($\varepsilon_c$). Both are described below.

![Figure 5.17 Plot of laboratory tests in PEK (Board, 2006)](image)

5.5.1 Estimation of initiation and depth of fracturing in the cutting front based on the uniaxial compression strength of intact rock (UCS$_{lab}$)

In some block caving mines, such as El Teniente, an empirical criterion between the UCS$_{lab}$ and the major principal stress induced by the mining is used to predict the damage caused by stress in drifts and shafts. The criterion shows that if the value of the major principal stress is
between 40 and 60% of the value of the UCS_{lab}, then damage could occur. (Cuello, 2010), in “Reservas Norte” of El Teniente mine, corroborated this finding through the numerical model in Map^{3D}. Cuello defined the damage condition for zero confinement in the undercut level as between 41 and 56% of the strength of the intact rock. Martin (1994) studied the laboratory tests and the field behavior of Lac du Bonnet granite and found that the fracture or failure process began with an applied stress of 0.3 to 0.4 times the uniaxial compressive strength of the intact rock. With the same type of studies, Martin et al. (1999) showed that for massive and moderately jointed granitic rock, the initiation of damage can be calculated by Equation 5.11 (see also Figure 5.18)

\[ 0.35 \leq \frac{\sigma_1 - \sigma_3}{UCS_{Lab}} \leq 0.45 \quad \text{(5.11)} \]

where \( \sigma_1, \sigma_3 \) are the principal stresses induced.

![Figure 5.18 Empirical thresholds for in-situ damage in moderately jointed hard rock mass (Diederichs, 2003)](image)

This equation was incorporated in a Fish function (language of FLAC^{3D}), with which it is possible to estimate the deviatoric strength (difference between principal stress, \( \sigma_1 - \sigma_3 \), (Brady, 1999)) of initiation of fracturing in each node of the finite differences mesh and to obtain an approximation of the depth of fracturing.

In connection with the Chuquicamata Underground Project, (Board and Poek, 2009) made an estimation of the strength of the rock mass using the technique of Synthetic Rock Mass (SRM). They generated a Discrete Fracture Network (DFN). DFNs are “stochastic models of fracture
architecture that incorporate statistical scaling rules derived from analysis of fracture length, height, spacing, orientation, and aperture,” according to Guohai (2007). The DFN is supported by the mapping of veinlets and fractures from the exploration of tunnels, and was superimposed on the matrix PFC\textsuperscript{3D} model to obtain a synthetic rock with a matrix and discontinuities in three dimensions. To calibrate their model, they tested samples of rock of different sizes: 1,000, 750, 500 and 250 millimeters. The result of the extrapolation to the rock mass properties for a block of 1 meter x 1 meter x 1 meter was strength of 40 to 45% of the uniaxial laboratory test for intact rock. In conclusion, the data that support this criterion suggest it is an appropriate approach for estimating the initiation and depth of damage in the cutting front.

5.5.2 Estimation of depth fracturing in base of critical strain

Stacey (1981), in search of a simple criterion to estimate the initiation of fracture in brittle rock, found that a good prediction of both orientation and extent of fracturing was achieved using the limit of strain criterion. According to Stacey (1981), “Fracture of brittle rock will initiate when the total extension strain in the rock exceeds a critical value which is characteristic of that rock type”. This relation can be expressed as:

\[ e > e_s \]  \hspace{1cm} (5.12)

In which \( e_s \) is the critical strain when the critical strain is exceeded, fracturing begins. To estimate the strain, Equation 5.13 is used. It is interesting to note that this equation takes into consideration the three principal stresses.

\[ e_3 = \frac{1}{E} \left[ \sigma_3 - \nu (\sigma_1 + \sigma_2) \right] \]  \hspace{1cm} (5.13)

where \( \sigma_1, \sigma_2, \) and \( \sigma_3 \) are the principal stresses; \( E \) is the elasticity modulus; and \( \nu \) is the Poisson’s ratio. Stacey found, through uniaxial tests in brittle rock and back analysis of tunnels, that a value of 0.0002 is a good critical strain limit to estimate the initial fracture.

Equation 5.13 was incorporated in a Fish function (language of FLAC\textsuperscript{3D}), with which it is possible to estimate the critical strain (\( e_s \)) leading to the initiation of fracturing in each node of the finite differences mesh and to obtain an approximation of the depth of fracturing.
This criterion is illustrated in Figure 5.19, where it is possible to observe the fracturing and the geometry around a square tunnel, and the slabbing in the wall which should be similar to the potential behavior of the undercutting front due to the effect of the stress variation when the undercut advances.

Figure 5.19 Concept of initiation of fracture and slabbing (Stacey, 1981)
CHAPTER 6
GEOMECHANICAL ANALYSES

6.1 Framework of geomechanical analyses

This chapter summarizes the results of analyses of the stress variation in the undercutting front, the vertical deformation in the work area, and the relationship of these with the undercutting front advance in pre-caving and continuous caving stages. The results of the numerical models for different undercutting front width are hereby presented and the effect of the undercutting beam length is analyzed. The strength of the pillar is estimated for numerical and analytic methods. To assess depth of fracturing, due to stress variation, two methodologies were used. One methodology evaluated the relationship with the uniaxial compression stress, and the other evaluated the relation with the critical deformation (Chapter 5).

The kinematic analyses are normally used in the wedge analyses in underground mines. In this work, a kinematic analysis was used to evaluate the formation of possible falls of wedges in the work area (Chapter 3) and estimate if this could be a fatal condition. In addition, the kinematic analysis provided an initial estimate about necessary roof support. The block caving fragmentation curve for the macro blocks of testing was used to estimate the distribution and probability of the occurrence of wedges. In addition, the software UNWEDGE was used for a tridimensional visualization of maximum possible wedges and to estimate wedge shape and weight. This chapter finishes with a brief discussion of the possibility of the rotation of big blocks in the border of the caving.

6.2 Geomechanical behavior of cutting front and work areas during pre-caving

In order to analyze the effect of the undercutting advance in the cutting front, stress was analyzed in three orthogonal directions: vertical stress ($\sigma_v$), horizontal stress along the face ($\sigma_h$), and normal stress to the cutting face ($\sigma_n$). This is presented in Figure 6.1, where the volume represents the work area.

In Figure 6.2, the vertical stress ($\sigma_v$) distribution for a different undercutting front advance was graphed, advance length is understood as the horizontal distance that the cutting system advances cutting the base of the block, this incorporates a new rock volume to the caving process. Below this graphic, there is an illustration of the work area and the cutting front in order
to represent the location of the stress ahead of the cutting face. The stress rapidly grows as the undercutting front advances; for example, between 20 and 40 meters, the stress shows an increment of around 65%. In the next advance, from 40 to 60 meters, the increase of stress is only 20%. With the subsequent advances, the stress distribution tends to stabilize as the undercutting area approaches the hydraulic radius of caving. To 1 meter, measured from the cutting front, the vertical stress is at its maximum and after this, decrease. Up to 10 meters of advance, the stress tends to return to its initial value of stress (pre-mining stress).

Figure 6.1 Scheme of analysis of stress in the cutting front

Figure 6.3 shows a graph of the behavior of normal stress, $\sigma_n$, for different undercutting front advances. Below this graph, there is an illustration of the work area and the cutting face coincident with a graph that shows stress distribution ahead of the rock mass. In the graph, it is possible to see that the stress of confinement grows significantly to a few centimeters ahead of the cutting face, and it increases with the undercutting front advance until it reaches the in-situ, pre-mining value at around 10 meters from the cutting front. Figure 6.4 presents the behavior of
the horizontal stress, \( \sigma_h \), which has a similar behavior to vertical stress, with the difference that the magnitude of the stress is lower than that of the vertical stress.

Figure 6.2 Behaviour of vertical stress with advance of the undercutting front

Figure 6.3 Behaviour of normal stress with the advance of the undercutting front
Figure 6.4 Behaviour of horizontal stress with the advance of the undercutting front

To complement the analysis of the work area, the vertical displacement in the roof was also studied, considering a zone of 5 meters from the cutting front during different undercutting front advances. Figure 6.5 shows the distributions of vertical displacement for different distances of the undercutting front advance. It is possible to observe that close to the cutting front, the displacements are low, but they rapidly grow up to caving volume (where the drawbells are open and production exists). For example, in the first 3 meters the vertical displacement never exceeds the 40 millimeters that would be below the capacity of the support system (such as hydraulic roof support). However, the magnitudes of displacements grow with the advance of the undercutting front. Therefore, the load over the roof support increases with the undercutting advance.

The fracturing in the cutting front was estimated using Equations 5.11 and 5.13 (the criteria of initiation and depth of fracturing described above in Chapter 5) through the run-on simple routines made in FLAC\textsuperscript{3D} called FISH functions.
Initiation and propagation of fracture in the models of pre-caving were estimated. A step of 20 meters of undercutting advance was used in a range of 20 to 120 meters of the undercutting advance. Results indicated that the critical strain criterion is more sensitive to stress increases than in the UCS_{lab} criterion, and showed that a fracture in the cutting front started to be generated before the initial 20 meters of the undercutting front advance. Meanwhile, the UCS_{lab} criterion showed that the fracture started before an initial span of 40 meters in the undercutting front advance. In addition, the two criteria showed differences in the depth of the fracture. However, despite the fact that the critical strain predicted an initiation of the fracture before the UCS_{lab}, the difference in the depth of the fracture is minor. In Figure 6.6, the depth of the fracture for both methodologies is plotted for different distances of the undercutting front advance. Also in this graphic, it is possible to see that the depth of the fracture increases gradually to a certain level. In addition, in the UCS_{lab} criterion, the depth of fracturing was 100 centimeters, whereas in the strain criterion it was 60 centimeters. Figure 6.7 shows the fracture prediction with both methods. Both methodologies agree that the fracturing start from the lower part of the cutting front, this would cause that the front tends to have a concave geometry (red - yellow for UCS lab...
methodology in the upper figure; and yellow – orange for critical deformation methodology in the lower figure).

![Initiation and depth of fracturing in the cutting front](image)

**Figure 6.6** Variation of depth of fracture in the cutting front with the advance of the undercutting front

### 6.3 Geomechanical behavior of cutting front and work areas in continuous caving

An analysis similar to pre-caving was made for the continuous caving. It considered three lengths of beam and an advance rate of 20 meters. The results showed that the components of stress ($\sigma_v$, $\sigma_h$, and $\sigma_n$) do not change with the undercutting front advance. This means that after the undercutting area achieves the hydraulic radius of caving, the stress variation in the cutting front and in the work area does not change significantly. Figure 6.8 shows the vertical stress ($\sigma_v$) distribution for the undercutting front advance of 180, 200, and 220 meters using a beam of 40 meters. In this figure, it is possible to see that the three distributions are similar.

In addition, a comparison of stress magnitude for different lengths of the beam does not show significant differences in the stress condition. Figure 6.9 shows vertical stress, $\sigma_v$, plotted for beams of 20, 40, and 60 meters. The three curves present same behavior.

In general, the numerical model shows that close to the cutting face the displacements are low and increase as they move away from the cutting face. Figure 6.10 shows vertical displacements above and below 50 centimeters, for a beam of 20 meters and an undercutting...
advance of 180 meters. In this figure, it is possible to appreciate that in the work area, vertical displacements are less than 50 cm. This effect is the same for beams of 40 and 60 meters.

Figure 6.7 Prediction of fracturing in the cutting front with 120 meters of undercutting front advance with UCS\textsubscript{lab} and critical strain criteria beams, which is reasonable considering the effect of the proximity of the caving volume. Figure 6.11 shows vertical displacements in the work area for different sizes of beams, from the cutting face to the caving volume. It is possible to observe the variation in the displacements with the undercutting front advance for different beams, but the variations are not large when compared to the variations in displacements during pre-caving. Figure 6.11, below show vertical displacement in the work area and illustrates the cutting face and the undercutting advance.
Figure 6.8 Distributions of tress in the cutting front of different undercutting advance operations in continuous caving with a 40 meters beam

Figure 6.9 Distributions of vertical stress in the cutting front for different lengths of beams with an undercutting advance of 180 meters

Regarding the vertical displacements in the work area, the analysis shows that they decrease in relationship to the beam size. Short beams have more displacement in the roof than long
The initiation and depth of fracturing was also analyzed for continuous caving, and this analysis shows that the condition of fracturing in the cutting front does not change significantly with the advance of the undercutting front and the length of the beam.

When gathering the information from both numerical models, the major principal stress ($\sigma_1$) in the cutting front was graphed in Figure 6.12. It shows that at the beginning of undercutting, the stress grew until the area reached the hydraulic radius of caving and it continued constantly, with low variation, considering that the geomechanical conditions did not change.

Based on the fracturing level in the cutting front area, Figure 6.12 shows a plot of the principal major stress distribution ($\sigma_1$) in the undercut front due to the undercutting front advance this value is normalized by the principal pre-mining major stress. This plot shows that stress increases until becomes constant.

This distribution has identified zones obtained from fracturing depth analysis (Figure 6.6) for the UCS$_{lab}$ methodology where fracturing starts between 20 and 40 m, then 40 to 80 fracturing continuous and above 80 m fracturing stays constant.

### 6.4 Effect of the width of the panel in the cutting front

In order to evaluate the possible effect of the increases of width of the undercutting front, three numerical models were built for 120, 150, and 180 meters of undercutting front widths.
Figure 6.13 shows the major principal stress distributions for the three widths to distance of undercutting front advance.

The stress increases with the undercutting advance until reaching the caving area, and the maximum stresses are nearly the same in the three cases.

6.5 Stability of the pillar between panels

The sequence of the numerical models for the pillar is the same as for one panel in precaving and continuous caving, with the difference that it is necessary to incorporate the elimination of the pillar between panels in the sequence. This reduction of the pillar begins when the undercutting front of Panel B advances more than the length of the beam. There is a lag between the cutting front and the border of the pillar equal to the beam length. When the undercutting advances, the elimination of the pillar advances as well. The idea is to recover the mineral between the panels and to connect the caving of Panel A with the caving of Panel B. For...
this method, it is necessary to implement a procedure of extraction to avoid early ingress of dilution. Figure 6.14 presents the sequence of advance of undercutting front for Panel B and shows how the pillar is cut.

Figure 6.12 Variation of principal stress with the undercutting front advance and change of the cutting front condition

Figure 6.13 Variation of stress in the undercutting front for different widths of the undercutting front
The pillar between panels should support four different conditions of loads. First, it supports the advance of the undercutting front and the initiation of caving in Panel A. In this step, the function of the pillar is to permit safe access to Panel B. Second, it should support the increase in the caving of Panel A. During this time, the pillar will suffer changes in stress and will lose confinement. Thirdly, it must support the initiation of the undercutting advance of Panel B. Fourthly, the pillar is cut from the production level.

A monitoring point was set in the center of the pillar and in the middle of the panel aimed at capturing the data of the vertical stress during the sequences of undercutting in Panels A and B (Figure 6.15). From monitoring of the pillar, it is possible to identify the first three of the four steps. During the advance of Panel A, the stress over the pillar increases until the undercutting front is beyond 40 meters. In the second stage, the caving progresses in Panel A, and the stress decreases. In the last stage, part of the pillar is cut, and the pillar beam begins to support the load. Figure 6.16 show the results of monitoring FLAC$^{3D}$ in the center of the pillar.
Figure 6.15 Mining steps in the life of the pillar between panels

Figure 6.16 Monitoring of vertical stress in the middle of the pillar

Complementary with the analysis of the load over the pillar, the strength of the pillar was analytically estimated using the Stacey and Page (1986) equation developed for hard rock and big cavities.
\[ \sigma_p = DRMS \left( \frac{W_{eff}^{0.7}}{h^{0.5}} \right) \] (6.1)

Where:

- \( \sigma_p \) = Average strength of the pillar
- DRMS = Design Rock Mass Strength
- \( W_{eff} \) = Effective width of the pillar
- \( h \) = Height of the pillar

\[ DRMS = RMS \times C_w \times C_j \times C_m \times C_b \times C_{w/i} \] (6.2)

Where:

- RMS = Rock Mass Strength
- RMS = \((A-\frac{B}{70})\times RBS\)
- RBS = Rock Block Strength
- \( A \) = RMR value
- \( B \) = RBA rating (Laubscher, 2001)
- RBS = 0.8 * IRS (homogeneous rock blocks)
- IRS = Intact Rock Strength
- \( C_w \) = Adjustment for weathering (1 – 0.3)
- \( C_j \) = Adjustment for joint orientation (0.95 – 0.7)
- \( C_m \) = Adjustment for mining induced stress (1.2 – 0.6)
- \( C_b \) = Adjustment for blasting (1.0 – 0.8)
- \( C_{w/i} \) = Adjustment for water/ice (1.2 – 0.7)

In this case:
- IRS = 133.6 Mpa
- \( A = 58 \)
- \( B = 22 \)
Therefore,
- RMS = 54.97 Mpa.
Where
$C_w = 1.0$ (dry rock mass)  
$C_j = 0.8$ (some families of structures parallel to the pillar)  
$C_m = 0.75$ (the stress deterioration of the pillar)  
$C_b = 1.0$ (without damage from blasting)  
$C_{w/h} = 1.0$ (the mine is in the desert)  
$W_{eff} = 20$ meters  
$h = 4.0$ meters  
Therefore,  
$DRMS = 32.98$ Mpa  
Thus,  
The Average Pillar Strength is $\sigma_p = 56$ Mpa

Numerical models of the pillar were performed and the average load inside of the pillar were estimated and compared with the estimation of the pillar strength. Figure 6.17 shows a plotting of the vertical stress (load in the pillar) at 62, 80, and 98 meters (these values represent locations in the pillar; 62 meters is close to the border, 80 meters is near the middle of the pillar, and 98 meters is only 2 meters from the cutting front). Note that at the border of the pillar (62 meters, as indicated by the blue line), the average stress is below the strength estimated. However, in the rest of the pillar, the load is more than the estimated strength. This estimate was tested for the other undercutting advance, with the same result. In conclusion, the pillar fulfills its function of protecting the work area and allows the advance of the undercutting front in Panel B.

Figure 6.18 presents the profiles of the average vertical stress for different undercutting front advances in the border of the cut pillar (2 meters from the cutting face). The plots show there is not a big difference between the results.

Together with this analysis, an estimation of fracturing was made using $UCS_{lab}$ criterion (the estimate is conservative, because the $UCS$ criterion gives a greater depth of fracture than the strain criterion). The $UCS_{lab}$ analysis shows the same behavior as the analytical estimation, i.e., there is high fracturing in the border of the pillar, but in the work area the pillar is stable. Figure 6.19 (plot of the numerical model of fracturing) shows that near the border (at 82 meters), the fracture crosses the whole pillar (yellow); but moving away from the border, the fracturing decreases and the center of the pillar is stable. In the middle of the pillar beam there are no
fractures, there is only some fracturing in the walls. Close to the cutting face (118 meters), the pillar is not damaged. In Figure 6.19, there is a sketch of the profile location.

![Diagram](image)

Figure 6.17 Profile of vertical stress in the pillar to 100 meters of undercutting advance

6.6 Kinematic analyses of the work area

In block caving, three types of fragmentation are identified (Brown, 2007): in-situ fragmentation, primary fragmentation, and secondary fragmentation. The in-situ fragmentation is created by the faults, joints, diaclases, or other natural weaknesses, and it is inherent to the rock mass. Primary fragmentation represents the blocks created by a combination of natural weakness and induced fractures from the effect of shear or tension stress. Secondary fragmentation is caused by the movement of fragments inside the cavity by the production draw.

During pre-caving and continuous caving, when utilizing an undercut method of advanced undercutting, the blocks that could fall down in the work area would correspond to in-situ or
primary fragmentation. To estimate the size and frequency of these blocks or wedges, fragmentation in the Chuquicamata Underground Project (Board, 2006) was estimated with the software BCF (Block Caving Fragmentation) (Figure 6.20). The structural information for evaluating the distribution of blocks underground in the Chuquicamata mine is presented in Table 6.1.

![Graph of vertical stress in the border of the pillar 2.0 meters from the caving area](image1)

![Graph of vertical stress in the position of the undercutting front](image2)

Figure 6.18 Profile of vertical stress in the edge of the pillar to the position of the undercutting front

In this case, considering the fragmentation curve (Figure 6.22) for the blocks over 10 m$^3$, there is only a 2% probability that they would appear in the work area.

With the information of the structural set, a simple stability analysis for maximum wedge was undertaken combining the structural sets (Table 6.1) with the software UNWEDGE (Unwedge, 2011). Figure 6.21 shows that structural sets 2, 3, and 5 form the maximum wedges with a weight of 133.54 tons. For this estimate, a work area of 6 meters was used (this is two times the work area used in the numerical model analyses, in order to generate a big wedge).
Assuming a perfect distribution of load over the hydraulic roof support (hydraulic support used in longwall, Chapter 3), the load distributed would be 8.16 ton/m², and the capacity of the hydraulic support with legs of 380 millimeter internal diameter is around 120 ton/m² (Mitchell, 2012). Therefore, the load is within the design capacity (Figure 6.22).

Figure 6.19 Fracturing zones in the pillar using UCS_

Table 6.1 Structural sets in macro block N 42 (Zaro, 2009).

<table>
<thead>
<tr>
<th>Structural Domain</th>
<th>Faults (VIF)</th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Dip (degrees)</td>
<td>Dip Direction (degrees)</td>
<td>Trace (meters)</td>
<td>Spacing (meters)</td>
</tr>
<tr>
<td>Balmaceda</td>
<td>80 ± 09</td>
<td>218 ± 15</td>
<td>606 ± 264</td>
<td>73 ± 26</td>
</tr>
<tr>
<td></td>
<td>80 ± 09</td>
<td>040 ± 08</td>
<td>606 ± 264</td>
<td>73 ± 26</td>
</tr>
<tr>
<td></td>
<td>80 ± 09</td>
<td>176 ± 23</td>
<td>876 ± 274</td>
<td>126 ± 55</td>
</tr>
<tr>
<td></td>
<td>83 ± 06</td>
<td>359 ± 21</td>
<td>876 ± 274</td>
<td>126 ± 55</td>
</tr>
<tr>
<td></td>
<td>79 ± 11</td>
<td>270 ± 13</td>
<td>313 ± 135</td>
<td>236 ± 143</td>
</tr>
</tbody>
</table>
VIF = Very Important Fault (definition given in Chuquicamata to fault with traces over 200 meters)

Figure 6.20 Estimated fragmentation of macro blocks for Chuquicamata Underground Project, using BCF (Board, 2006)

Figure 6.21 Three-dimensional view of maximum wedge in work area, and one illustrative example (Brown, 2007)
There is another possible wedge geometry (Brown, 2007), but it is only described in one case history involving post-undercut (Brown, 2007). It is a big block formed by two sub-vertical structures with dips in opposite directions that intersect the cutting front and have a free face in the cavity of the caving volume. The back face could be the caving face, the bottom face the beam and the other face could be formed by geological structures. An illustration is shown in Figure 6.23 for two lengths of beam of 20 and 60 meters. If the beam is bigger, the wedge increases too. In this case, the long beam forms a wedge bigger than the short beam. In this situation, the wedge could be predicted with good geological mapping and a geotechnical analysis of key blocks. The length of the beam could also be controlled in order to handle this condition.
6.7 Numerical model validation

Due to there are not previous experience in longwall as undercutting method in block caving and the block caving in Chuquicamata will start in 2019, the validation of numerical models has been comparing the result with information obtained from literature review (Chapter 2). In addition, considering the Chuquicamata underground project compared the damage condition around a panel access with the exploration tunnels condition.

- Comparison regarding fracturing depth in the cutting front

In the work of Qing-Sheng (2013) “Numerical modeling on brittle failure of coal wall in longwall face, a case study” they presents a back analysis of fracture in the cutting face analyzed through the shear failure. This example can be compared to the numerical modeling used in this thesis; despite it is a different rock (coal) because it also considers a brittle failure behavior and a constitutive stress-softening model are used. The work concludes that the ground data and the numerical model give a fracture depth between 0.70 and 1.20 meters (Figure 6.24) which is similar to result obtained in this thesis using UCS lab and critical deformation fracturing between 0.8 to 1.0 meters Figure 6.25.

Figure 6.24 Brittle failure in longwall face (Qing- Sheng, 2013)
Comparison regarding to the abutment stress in the undercutting front.

In the same work, of Qing-Sheng (2013) “Numerical modeling on brittle failure of coal wall in longwall face, a case study” they present the stress distribution in the undercutting front validated with ground data Figure 6.26, which is coincident with the stress distribution obtained in this thesis numerical model Figure 6.27.
• Comparison regarding to the displacements in the work area.

In the work of González, C. Menéndez, A. Álvarez, A.E. M.I. Álvarez, M.I. (2008) “Analysis of support by hydraulic props in a longwall working” International Journal of Coal Geology 74 (2008) page 67–92, they present and validate through numerical 2D models the damage in the roof of work area Figure 6.28, which is similar to the estimation developed in this thesis Figure 6.29.

Other exercise was performed to compare the prediction of the numerical models with the current condition of an exploration tunnel in the zone of study. Information obtained from the field shows that the fracturing around the tunnel is null or minimal. For the UCS\textsubscript{lab} criterion, two values of percentage of UCS were tested: 30 and 40% (Chapter 5). The criterion of 30% shows a little initiation of the fracture in the middle of the roof. Otherwise, the 30% criterion shows zero fracture around the tunnel in UCS\textsubscript{lab}. The strain criterion was tested with the limit value of 0.0002 as recommended by Stacey (1981), which showed an incipient fracture in the middle of the roof, Figure 6.30 (This is consistent with the UCS\textsubscript{lab} criterion of 30%). In general the possible location of fracture around the tunnel coincides with the exploration tunnel, there are small fracturing in the roof of the tunnel (with mesh support) and in the tunnel walls without damage Figure 6.30.

![Figure 6.28 Collapse of the hanging wall of the working for a sub-critical support pressure (Gonzalez, 2008)](image-url)
Figure 6.29 Plasticity zones for 20 and 40 meters undercutting advances, showing the instability zones in the roof.

Figure 6.30 Numerical model fracturing initiation and exploration tunnel condition.
CHAPTER 7
CONCLUSION AND FUTURE WORK

7.1 Conclusions

In the last ten years, undercut mine design has changed considerably. Since the introduction of the first advance undercut in Palabora Mine (South Africa) and the first pre-undercut in El Teniente Mine (Chile), undercut design has evolved to improve the stability of the undercut level and the quality of the undercut. This has increased the costs, and introducing new problems such as management of mine construction. The most common current design in pre-undercut and advance undercut is the crinkle cut, which—despite its advantages—is not fully accepted by the mining industry.

Based on the studies presented, this thesis concludes that from a geomechanical point of view, it is feasible to use a mechanical system to undertake the undercut in block caving, using the same longwall method used in coal mining. Below are the specific conclusions of this research.

1. The risk analysis in Chapter 4 identified three high risk zones: (1) the cutting front, (2) the work area, and (3) the pillar between panels. The risks were associated with losses of confinement in the center of the panel, creation of a wedge in the roof of the cutting front, and collapse of a pillar between panels, respectively. This is logical, given that the cutting front is where the machine cuts the rock, the work area is the place where the machine and the support system should move, and the pillar is the support allowing the continuity of the method in new sectors.

2. The results of this thesis indicate that the mechanical undercutting system offers the option of increasing the undercutting rate in block caving.

3. This alternative method allows modifications of the work system in block caving mines. Due to this, it is not necessary to organize the shifts around undercutting blasting.

4. The numerical models show that in the first 2 meters ahead of the cutting front, the stress increases and decreases suddenly, and that is coincident with the height of the cutting.
5. During pre-caving, the stress increases with the advance of the undercutting front. Decrease in rate is higher in the first meters of advance, and decreases when the area approaches the hydraulic radius of caving.

6. Regarding deformation in the roof of the work area, in pre-caving the vertical displacement changes with the undercutting advance at a rate similar to the stress. There is low vertical deformation close to the cutting front, which increases when approaching the caving area. In general, the magnitudes of the deformations estimated are inside the capacity of the support element.

7. In continuous caving, the magnitude of stress does not change significantly with the advance of the undercut front.

8. The length of the undercutting beam in continuous caving does not greatly affect the magnitude of principal stress in the cutting front, for a beam shorter than 60 meters. This length is not long enough to generate a new caving area independently of the caving volume.

9. Regarding vertical displacements, the length of the undercutting beam in continuous caving affects the magnitude of vertical displacement in the work area. When the cutting front is closer to the caving (short beam), the displacements are higher than when it is further away. Nevertheless, the variation in vertical displacement is less than during pre-caving.

10. The abutment stress, before the undercutting reaches the hydraulic radius of caving, increases with the advance of the undercutting front.

11. Based on the fracturing estimate prepared using the UCS_{lab} and critical strain criteria, it is possible to conclude that during the undercutting process, the condition of the cutting front changes with the undercutting advance from a smooth cutting face (in the initiation of the undercut) to a fractured cutting front (in the regimen condition).

12. A simple analysis of wedges, from the structural sets, shows that is possible support with the hydraulic roof support system, the maximum wedge possible in the work area.

13. The estimate of the performance of a mechanical system shows that the undercutting advance in hard rock reaches values similar to the normal advances in block caving with drill and blast.

14. The concept, design, and sequence of a mechanical system in the undercut process in block caving and the longwall method are similar. However, there is a great difference between both, which is that caving application is not a production method. Another key difference is
that some of the equipment used in longwall should be re-designed for be used in block caving.

15. The undercutting mine design proposed in this thesis corresponds to a design for an operationally feasible advance undercut. This thesis serves as a guideline to evaluate the feasibility of using a mechanical undercutting system in any mining operation or project.

7.2 Future work

Taking into account the important advances in cutter for TBMs in hard rock technology, there is a good opportunity to incorporate this advance in shearing machines (the machines used in longwall) and re-design the equipment in order to use this technology in undercutting in block caving method. To future work further risk assessments is required utilizing other type of analysis. This analysis should include a panel of block caving and longwall experts to joints identify the potential risk.

In addition, the idea of simplifying the undercut design could be an interesting alternative that could make in feasible alternative in deep caving without the presence of people.
REFERENCES CITED

Aguirre, E. (1994) “A study of the rock mass damage associated with the panel caving method” Third JKMRC Student conference, University of Queensland, Australia.


Allende, V. (2009) Chuquicamata Underground Mine Pre-Feasibility Study API N07DM43. Presentación, Vice-presidency of project (VP) Codelco Chile. (Internal presentation)


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Chuquicamata Underground Project in Chile.” *Second International Symposium on Block and Sublevel Caving*. Perth.


APPENDIX A
CODIES FLAC\textsuperscript{3D} OF NUMERICAL MODELS

For modelling the caving and its effects in the cutting front and in the work area, a group of numerical models were run in FLAC\textsuperscript{3D} (Geomechanics software), the details of these models are in Table 5.5. In this Appendix there are source code in FISH (lenguaje FLAC\textsuperscript{3D}) and commentaries about how were done the numerical models.

1A.1 Continuous Caving

1A.1.1 Caving\_20\_120

*Caving\_20\_120.dat (Archive)

```plaintext
new
set fish safe off
config cppudm ; Configuration for cavehoek contitutive model
rest 8\_FIRST\_EXCAV.sav
;********************************************************************************
;Model continuous caving with 20 m of beam
;********************************************************************************
;Definition fo parameters
;********************************************************************************
def mod_prop_20
cave_angle = 50. ; cave front angle
exc_inc = 20. ; thickness of excavation increments
first_excav = 160. ; (length of excavation before introducing cave)
beam = 20. ;
panel_width = 120. ; width of the panel
panel_length = 220. ; panel_length
undercut_height = 2. ; undercut height
;ROCK MASS Properties CRrockfrag_aspect=0.5 CRgsi=59
CRdensity= 2620 CRyoung_intact = 52.7e9 CRm_intact= 14.83 CRhb_sci=133.62e6
end @mod_prop_20

;********************************************************************************
;Define the model boundary conditions
;********************************************************************************
def bound command
free x
free y
free z
```

121
ini xvel 0 yvel 0 zvel 0

fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix x range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 90 ori @max_x 0 0 dis @y_pladis

fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
end_command
end

;*******************************************************************
;grows the cave until it reaches (first_excav - beam). Change the properties inside of caving to ;residual 1, ;volumetric stress increment (vsi) 0.67 ;fixed all mesh except the CAVE material, initial stress zero and ;solve.
; Free the mesh and restore the original border conditions. Save each increment of the triangle.
;*******************************************************************

def cgrowth _
x= 0
num_inc=(first_excav-beam)/exc_inc ; Number of steps pre-caving
cc_=0
loop while cc_ < num_inc cc_=cc_+1
file_name=string(cc_)+"PRE_CAVE.sav" _x=_x+exc_inc ; Increase of span
(command

group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 & plane below dip 90 dd 0 origin 0 panel_width 0 z 0 5000 model mech cavehoek range group CAVE prop rockfrag_aspect CRrockfrag_aspect range group CAVE prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE prop start_at_residual 1 vsi 0.67 range group CAVE ini vel 0,0,0 fix x range group CAVE not ; fixed all mesh except CAVE volum fix y range group CAVE not

fix z range group CAVE not ini stress 0 range group CAVE; All stress 0 step 100

solve @bound
step 100 solve sav file_name end_command
end_loop
end @cgrowth

;********************************************************************
;CGROWTH2 :grows the cave and undercut until end of panel reached; Advace with undercutting area an caving area; with 20 m of increment excape 20 m and save this new advance. Change the properties of new volume incorporate to caving.;fix the mesh except the CAVE group. Stress to zero and run. Continueo with the same process above.
;********************************************************************
def cgrowth2 num_inc=(panel_length-first_excav)/exc_inc cc_=0

loop while cc_ < num_inc cc_=cc_+1
file_name=string(cc_)+"_CAVE_and_UNDERCUT.sav"  x=x+exc_inc
dist = x + beam f_name=string(cc_)+"_EXCAV.sav" command
   group EXCAVATE range z 0 undercut_height y 0 panel_width x 0 dist group CAVE not
   model mech null range group EXCAVATE solve
   sav f_name

   group zone CAVE range plane below dip cave_angle dd 90 origin x 0 0 plane above dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 plane below dip 90 dd 0 origin 0
   panel_width 0 z 0 5000 model mech cavehoek range group CAVE
   prop rockfrag_aspect CRrockfrag_aspect range group CAVE
   prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
   prop start_at_residual 1 vsi 0.67 range group CAVE ini vel 0,0,0
   fix x range group CAVE not fix y range group CAVE not fix z range group CAVE not ini stress
   0 range group CAVE step 100
   solve @bound step 100 solve
   sav file_name end_command
end_loop end @cgrowth2

ret
1A.1.2 Caving_40_120

*Caving_40_120.dat (Archive)

new

set fish safe off

config cppudm ; Configuration for cavehoek contitative model

rest 8_FIRST_EXCAV.sav

;*******************************************************************
;Model continuous caving with 40 m of beam
;*******************************************************************
;defines the model boundary conditions
;*******************************************************************
def bound command
free x
free y
free z

ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
end_command
end

;*******************************************************************
grows the cave until it reaches (first_excav - beam). Change the properties inside of caving to residual 1,;volumetric stress increment (vsi) 0.67 ;fixed all mesh except the CAVE material, initial stress zero and solve.
; Free the mesh and restore the original border conditions. Save each incremet of the triangle.
;*******************************************************************
def cgrowth
x= 0
num Inc=(first_excav-beam)/exc_inc ; Number of steps pre-caving cc_=0
loop while cc_ < num_inc cc_=cc_+1
file_name=string(cc_)+"PRE_CAVE.sav" _x=_x+exc_inc ; Increase of span
command
group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd 90 origin 0 0 plane above dip 90 dd 0 origin 0 0 & plane below dip 90 dd 0 origin 0 panel_width 0 z 0 5000
model mech cavehoek range group CAVE
prop rockfrag_aspect CRrockfrag_aspect range group CAVE
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not ; fixed all mesh except CAVE volum
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE; All stress 0
step 100
solve
@bound
step 100 solve
sav file_name end_command
end_loop
end @cgrowth
;********************************************************************
;CGROWTH2 :grows the cave and undercut until end of panel reached; Advace with undercutting area an caving area; with 20 m of increment excave 20 m and save this new advance. Change the properties of new volume incorporate to caving, ;fix the mesh except the CAVE group. Stress to zero and run. Continue with the same process above. ;********************************************************************
def cgrowth2 num_inc=(panel_length-first_excav)/exc_inc cc_=0
loop while cc_ < num_inc cc_=cc_+1
file_name=string(cc_)+"_CAVE_and_UNDERCUT.sav" _x=_x+exc_inc
dist = _x + beam f_name=string(cc_)+"_EXCAV.sav" command
group EXCAVATE range z 0 undercut_height y 0 panel_width x 0 dist group CAVE not
model mech null range group EXCAVATE
solve
sav f_name

group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd 90 origin 0 0 plane above dip 90 dd 0 origin 0 0 plane below dip 90 dd 0 origin 0 panel_width 0 z 0 5000
model mech cavehoek range group CAVE
prop rockfrag_aspect CRrockfrag_aspect range group CAVE
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE
step 100
solve @bound step 100 solve
sav file_name end_command
end_loop
end @cgrowth2

;********************************************************************
;grows cave to surface: In the end of the undercut advance the cave increase to 90 degree(vertical subsidence; the same process above, change the properties, fix the mes except CAVE group velociti to zero run. Free the mesh and restore the border conditions.
;********************************************************************
group zone CAVE range plane below dip 90 dd 90 origin panel_length 0 0 plane above dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 plane below dip 90 dd 0 origin 0 panel_width 0 z 0 5000
model mech cavehoek range group CAVE

prop rockfrag_aspect CRrockfrag_aspect range group CAVE

prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE
step 100
solve @bound
step 100 solve
sav CAVE_TO_SURFACE.sav
ret
1A.1.3 Caving_60_120

*Caving_60_120.dat (Archive)

new

set fish safe off

config cppudm ; Configuration for cavehoek constitutive model

rest 8_FIRST_EXCAV.sav

;*****************************************************************************
***
; Model continuous caving with 60 m of beam
;*****************************************************************************
; Definition of parameters
;*****************************************************************************
def mod_prop_20
cave_angle = 50. ; cave front angle

exc_inc = 20. ; thickness of excavation increments
first_excav = 160. ; (length of excavation before introducing cave)
beam = 60. ;
panel_width = 120. ; width of the panel
length = 260. ; panel_length undercut
height = 2. ; undercut height

; ROCK MASS Properties
CRrockfrag_aspect=0.5 CRgsi=59 CRdensity= 2620
CRyoung_intact = 52.7e9
CRm_intact= 14.83 CRhb_sci=133.62e6
end @mod_prop_20

;*****************************************************************************
; defines the model boundary conditions
;*****************************************************************************
def bound command
free x
free y
free z
ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
def cgrowth_
    x= 0
    num_inc=(first_excav-beam)/exc_inc ; Number of steps pre-caving cc_=0
    loop while cc_ < num_inc cc_=cc_+1
        file_name=string(cc_)"PRE_CAVE.sav" _x=_x+exc_inc ; Increase of span

        command
        group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd
        90 origin 0 0 plane above dip 90 dd 0 origin 0 0 & plane below dip 90 dd 0 origin 0
        panel_width 0 z 0 5000
        model mech cavehoek range group CAVE
        prop rockfrag_aspect CRrockfrag_aspect range group CAVE
        prop gsi CRgsi density CRdensity young_intact CRyoung_intact

        m_intact CRm_intact hb_sci CRhb_sci range group CAVE
        prop start_at_residual 1 vsi 0.67 range group CAVE
        ini vel 0,0,0
        fix x range group CAVE not ;
        fixed all mesh except CAVE volum
        fix y range group CAVE not

        fix z range group CAVE not
        ini stress 0 range group CAVE; All stress 0 step 100

        solve
        @bound
        step 100 solve
        sav file_name end_command

    end_loop
end @cgrowth
;*********************************************************************
;CGROWTH2 : grows the cave and undercut until end of panel reached; Advance with 
undercutting area an caving area; with 20 m of increment excave 20 m and save this new 
advance. Change the properties of new volume incorporate to caving; fix the mesh except the 
CAVE group. Stress to zero and run. Continueo with the same process above.
;*********************************************************************
def cgrowth2
num_inc=(panel_length-first_excav)/exc_inc
cc_=0
loop while cc_ < num_inc cc_=cc_+1
file_name=string(cc_)+"_CAVE_and_UNDERCUT.sav" _x=_x+exc_inc
dist = _x + beam f_name=string(cc_)+"_EXCAV.sav"
command
group EXCAVATE range z 0 undercut_height y 0 panel_width x 0 dist group CAVE not
model mech null range group EXCAVATE
solve
sav f_name

group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 plane below dip 90 dd 0 origin 0
panel_width 0 z 0 5000

model mech cavehoek range group CAVE

prop rockfrag_aspect CRRockfrag_aspect range group CAVE
prop gsi CGsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE
step 100
solve @bound step 100
solve
sav file_name end_command
end_loop
end @cgrowth2
ret
1A.2 Different Width of the Undercutting Front

1A.2.1 Avance_pre_120

*Avance_pre-120.dat (Archive)

new
set fish safe off config cppudm

;*******************************************************************
rest Insitu_stress_Generic.sav ; restore initial stres condition
;*******************************************************************
;Advance with an undercutting front width of 120 m
;*******************************************************************
;Displacement to zero
;********************************************************************
ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0
ini state 0
;*********************************************************************
;defines the model boundary co
;********************************************************************
def bound
command free x
free y
free z

ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
end_command
end
;*****************************************************************
;change model properties here
;assumes bottom left corner of the panel in plan view is at coordinate 0,0
;*****************************************************************
; Definition for parameters
;***********************************************************************

def mod_prop

cave_angle = 50. ;cave front angle
exc_inc = 20. ;thickness of excavation increments
first_excav = 160. ;(length of excavation before introducing cave)
beam = 40. ;
panel_width = 120. ;width of the panel panel_
length = 220. ;panel_length undercut_
height = 2. ;undercut height
;*********************************************************************

;ROCK MASS Properties
CRrockfrag_aspect=0.5 CRgsi=59
CRdensity= 2620
CRyoung_intact = 52.7e9 CRm_intact= 14.83 CRhb_sci=133.62e6
end @mod_prop

;*********************************************************************

;excavate the first increment (up to first_excav)
def exc_first
dist=0.
inc = 1.
loop while dist < first_excav
dist = exc_inc * inc f_name=string(inc)+"_FIRST_EXCAV.sav" command

model mech null range group EXCAVATE

solve
sav f_name list inc end_command inc=inc+1

end_loop

end @exc_first

ret
\textbf{1A.2.2 Avance\_pre\_150}

*Avance\_pre\_150.dat (Archive)*

new

set fish safe off config cppudm

;**************************************************************
; Width 150 meters
;**************************************************************
rest Insitu\_stress\_Generic.sav ; restore initial stress condition
;**************************************************************
;Advance with an undercutting front width of 150 m
;**************************************************************
;Displacement to zero
;**************************************************************
ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0
ini state 0
;**************************************************************
;defines the model boundary conditions
;**************************************************************
def bound

command free x
free y
free z
ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min\_x 0 0 dis @x\_pladis
fix x range plane dip 90 dd 90 ori @max\_x 0 0 dis @x\_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min\_z dis @z\_pladis
fix x range plane dip 0 dd 0 ori 0 0 @max\_z dis @z\_pladis
fix y range plane dip 90 dd 0 ori 0 @min\_y 0 dis @y\_pladis
fix y range plane dip 90 dd 0 ori 0 @max\_y 0 dis @y\_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min\_z dis @z\_pladis
fix y range plane dip 0 dd 0 ori 0 0 @max\_z dis @z\_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min\_z dis @z\_pladis
fix z range plane dip 0 dd 0 ori 0 0 @max\_z dis @z\_pladis
end\_command

end
;**************************************************************
;change model properties here
;assumes bottom left corner of the panel in plan view is at coordinate 0,0,0
;**************************************************************
;Definiton fo parameters
;**************************************************************
def mod\_prop
cave_angle = 50. ; cave front angle
exc_inc = 20. ; thickness of excavation increments
first_excav = 160. ; (length of excavation before introducing cave)
beam = 40. ;
panel_width = 150. ; width of the panel
panel_length = 220. ; panel_length
undercut_height = 2. ; undercut height

;*******************************************************************
; ROCK MASS Properties
CRrockfrag_aspect=0.5 CRgsi=59
CRdensity= 2620
CRyoung_intact = 52.7e9
CRm_intact= 14.83 CRhb_sci=133.62e6
end @mod_prop

;*******************************************************************
; excavate the first increment (up to first_excav)
;********************************************************************
def exc_first
    dist=0.
    inc = 1.
    loop while dist < first_excav-40
        dist = exc_inc * inc
        f_name=string(inc)+"_FIRST_EXCAV.sav" command
group zone EXCAVATE range x 0 dist y 0 panel_width z 0 undercut_height model mech null
range group EXCAVATE
solve
sav f_name list inc end_command inc=inc+1
    end_loop
end @exc_first
ret
1A.2.3 Avance pre_180

*Avance_pre-180.dat (Archive)

new

set fish safe off config cppudm

;*******************************************************************
rest Insitu_stress_Generic.sav ; restore initial stres condition
;*******************************************************************
;Advance with an undercutting front width of 180 m
;*******************************************************************
;Displace ment to zero
;********************************************************************
ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0
ini state 0
;********************************************************************
;defines the model boundary conditions
;********************************************************************
def bound
command free x
free y
free z
ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix x range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix x range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
end_command
end
;**********************************************************************
;change model properties here
;assumes bottom left corner of the panel in plan view is at coordinate 0,0
;**********************************************************************
;Definiton fo parameters
;***********************************************************************
def mod_prop

cave_angle = 50. ;cave front angle
exc_inc = 20.; thickness of excavation increments
first_excav = 160.; (length of excavation before introducing cave)
beam = 40.;
panel_width = 180.; width of the panel
panel_length = 220.; panel_length
undercut_height = 2.; undercut height
;***************************************************************************
;***************************************************************************

; ROCK MASS Properties
CRrockfrag_aspect=0.5 CRgsi=59
CRdensity= 2620
CRyoung_intact = 52.7e9 CRm_intact= 14.83 CRhb_sci= 133.62e6
end @mod_prop

;***************************************************************************
;excavate the first increment (up to first_excav)
;***************************************************************************
def exc_first

dist=0.
inc = 1.
loop while dist < first_excav-40
  dist = exc_inc * inc
  f_name=string(inc)+"_FIRST_EXCAV.sav" command
  group zone EXCAVATE range x 0 dist y 0 panel_width z 0 undercut_height
  model mech null range group EXCAVATE
  solve
  sav f_name list inc end_command inc=inc+1
end_loop
end @exc_first
ret
1A.3 Insitu Stress

1A.3.1 Insitu_stress

*Insitu_stress.dat (Archive)

new

set fish safe off config cppudm

;******************************************************************
;Insitu stress for one panel
;******************************************************************

def mod_prop

;MODEL GEOMETRY

cave_angle = 50.;cave front angle
texc_inc = 20.;thickness of excavation increments
first_excav = 160.;(length of excavation before introducing cave)
beam = 40.;distance between cave and underctuing front
panel_width = 120.;width of the panel
panel_length = 220.;panel_length
undercut_height = 2.;undercut height

;ROCK MASS Properties
CRrockfrag_aspect=0.5;Rock Fragment Aspect Ratio
CRgsi=59;GSI
CRdensity=2620;Rock mass Density CRyoung_intact = 52.7e9;Young Intact
CRm_intact=14.83;Mb
CRhb_sci=133.62e6;UCS
D = 0.;Hoek-Brown D_
pois=0.20;Poissons ratio

;INITIAL STRESS CONDITIONS

x_kfac=1.859;ratio of horizontal to vertical stress in x-direction
y_kfac=1.267;ratio of horizontal to vertical stress in y-direction
gravity_=-9.81;value for gravity (down is negative)
zcoord_of_gs=696.;thickness of rock above undercut in meters
Density=2620.
end
@mod_prop
def _input

; Outer layer 64 m
xmin = -616
xmax = 536
ymin = -640
ymax = 768
zmin = -264
zmax = 696

dim_ = 64

n1 = int((xmax-xmin)/dim_)
n2 = int((ymax-ymin)/dim_)
n3 = int((zmax-zmin)/dim_)

; Coarse zoning of model, all 32 meter zones
xmin_2 = -296
xmax_2 = 536
ymin_2 = -320
ymax_2 = 448
zmin_2 = -264
zmax_2 = 376
dim_2 = 32
n1_2 = int((xmax_2-xmin)/dim_2)
n2_2 = int((ymax_2-ymin)/dim_2)
n3_2 = int((zmax_2-zmin)/dim_2)

; Mid-size nucleus, 16 meter zoning
dim_nuc=16
xnl = -136
xnu = 408
ynl = -160
ynu = 288
znl = -104
znu = 216

n zx = int((xnu-xnl)/dim_nuc)
nzy = int((ynu-ynl)/dim_nuc)
nzz = int((znu-znl)/dim_nuc)
;mid-size nucleus, 8 meter
dim_nuc_2= 8
xnl_2 = -72
xnu_2 = 296
ynl_2 = -80
ynu_2 = 208
znl_2 = -72
znu_2 = 136
nzx_2 = int((xnu_2-xnl_2)/dim_nuc_2)
nzy_2 = int((ynu_2-ynl_2)/dim_nuc_2)
nzz_2 = int((znu_2-znl_2)/dim_nuc_2)

;Fine Zoning 4
dim_fine = 4
xfl = -40
xfu = 256
yfl = -40
yfu = 168
zfl = -48
zfu = 96
nzx2 = int((xfu-xfl)/dim_fine)
nzy2 = int((yfu-yfl)/dim_fine)
nzz2 = int((zfu-zfl)/dim_fine)

;Fine zoning 2
dim_fine1 = 2
xfl1 = -28
xfu1 = 236
yfl1 = -24
yfu1 = 148
zfl1 = -24
zfu1 = 36
nzx21 = int((xfu1-xfl1)/dim_fine1)
nzy21 = int((yfu1-yfl1)/dim_fine1)
nzz21 = int((zfu1-zfl1)/dim_fine1)

;Fine zoning 1
dim_fine2 = 1
xfl2 = -18
xfu2 = 230
yfl2 = -12
yfu2 = 132
zfl2 = 8
zfu2 = 14
nzx22 = int((xfu2-xfl2)/dim_fine2)
nzy22 = int((yfu2-yfl2)/dim_fine2)

nzz22 = int((zfu2-zfl2)/dim_fine2)

; Fine zoning 0.5
dim_fine3 = 0.5

xf13 = 140

xfu3 = 226

yf13 = -6

yfu3 = 126

zfl3 = -6

zfu3 = 8

nzt23 = int((xfu3-xf13)/dim_fine3)

nzy23 = int((yfu3-yf13)/dim_fine3)

nzz23 = int((zfu3-zfl3)/dim_fine3)

end

@_input

; Generate main grid, 64m zones gen zone brick &

p0 @xmin @ymin @zmin &
p1 @xmax @ymin @zmin &
p2 @xmin @ymax @zmin &
p3 @xmin @ymin @zmax &

size @n1 @n2 @n3 group OUTER ;PAUSE

; Generate main grid, 32 m zones
delete zone range x @xmin_2 @xmax_2 y @ymin_2 @ymax_2 z @zmin_2 @zmax_2 gen zone brick &
p0 @xmin_2 @ymin_2 @zmin_2 & p1 @xmax_2 @ymin_2 @zmin_2 & p2 @xmin_2 @ymax_2 @zmin_2 & p3 @xmin_2 @ymin_2 @zmax_2 &

size @n1_2 @n2_2 @n3_2 group COARSE

; Generate the nucleus, 16 meter zones
delete zone range x @xnl @xnu y @ynl @ynu z @znl @znu
gen zone brick &
p0 @xnl @ynl @znl &
p1 @xnu @ynl @znl &
p2 @xnl @ynu @znl &
p3 @xnl @ynl @znu &

size @nxz @nyz @nzz group MED1
; Generate the nucleus, 8 meter zones
delete zone range x @xnl_2 @xnu_2 y @ynl_2 @ynu_2 z @znl_2 @znu_2
gen zone brick &
p0 @xnl_2 @ynl_2 @znl_2 & p1 @xnu_2 @ynl_2 @znl_2 & p2 @xnl_2 @ynu_2 @znl_2 &
p3 @xnL_2 @ynl_2 @znu_2 &

size @nzx_2 @nzy_2 @nzz_2 group MED2

; Generate fine zone region, 4 meter zones
Delete zone range x @xfl @xfu y @yfl @yfu z @zfl @zfu

gen zone brick &
p0 @xfl @yfl @zfl & p1 @xfu @yfl @zfl & p2 @xfl @yfu @zfl &
p3 @xfl @yfl @zfu &

size @nzx2 @nzy2 @nzz2 group SMALL4

; Generate fine zoned region, 2 meter
Delete zone range x @xfl1 @xfu1 y @yfl1 @yfu1 z @zfl1 @zfu1

gen zone brick &
p0 @xfl1 @yfl1 @zfl1 & p1 @xfu1 @yfl1 @zfl1 &
p2 @xfl1 @yfu1 @zfl1 &
p3 @xfl1 @yfl1 @zfu1 &

size @nzx21 @nzy21 @nzz21 group SMALL3

; Generate fine zoned region 1 meter

delete zone range x @xfl2 @xfu2 y @yfl2 @yfu2 z @zfl2 @zfu2

gen zone brick &
p0 @xfl2 @yfl2 @zfl2 &
p1 @xfu2 @yfl2 @zfl2 &
p2 @xfl2 @yfu2 @zfl2 &
p3 @xfl2 @yfl2 @zfu2 &

size @nzx22 @nzy22 @nzz22 group SMALL2

; pause

; Generate fine zoned region 0.5 meter

delete zone range x @xfl3 @xfu3 y @yfl3 @yfu3 z @zfl3 @zfu3
gen zone brick &

p0 @xf31 @yf31 @zf31 &

p1 @xf3u @yf31 @zf31 &

p2 @xf31 @yf3u @zf31 &

p3 @xf31 @yf31 @zf3u &

size @nxf23 @nyf23 @nzf23 group SMALL1

gen merge 0.1

attach face

;*****************************************************************
; DETERMINE LIMITS OF MODEL
;*****************************************************************
def get_limits min_x=1e20 max_x=-1e20 min_y=1e20 max_y=-1e20 min_z=1e20 max_z=-1e20

p_gp=gp_head

loop while p_gp # null

min_x=min(min_x,gp_xpos(p_gp)) max_x=max(max_x,gp_xpos(p_gp))

min_y=min(min_y,gp_ypos(p_gp)) max_y=max(max_y,gp_ypos(p_gp))

min_z=min(min_z,gp_zpos(p_gp)) max_z=max(max_z,gp_zpos(p_gp))

p_gp=gp_next(p_gp)

end_loop

orig_x=(min_x+max_x)/2.

orig_y=(min_y+max_y)/2. orig_z=(min_z+max_z)/2.

x_pladis=0.1 y_pladis=0.1 z_pladis=0.1

mid_x=((max_x-min_x)/2.)+min_x

mid_y=((max_y-min_y)/2.)+min_y

end @get_limits

;*****************************************************************
; Properties of rock mass
;*****************************************************************
def _pprop

_young=27.5e9; del macizo rocoso _pois=0.20

_bulk=_young/(3*(1-2*_pois)) _shear=_young/(2*(1+_pois))

end @_pprop

;*****************************************************************
; ELASTIC PROPERTIES AND EQUILIBRIUM
;*****************************************************************

model mech elas ini density 2620.

prop bulk _bulk shear _shear

;*****************************************************************

;INITIAL STRESS CONDITIONS
def ini_stresses rep_density =Density
szz_orig=gravity_*rep_density*zcoord_of_gs
sxx_orig=x_kfac*szz_orig
syy_orig=y_kfac*szz_orig
szz_grad=-1*gravity_*rep_density
sxx_grad=x_kfac*szz_grad
syy_grad=y_kfac*szz_grad
command
set gravity 0 0 @gravity_
initial szz @szz_orig grad 0 0 @szz_grad initial sxx @sxx_orig grad 0 0 @sxx_grad initial syy @syy_orig grad 0 0 @syy_grad
end_command end

@end_stresses ;initialize stresses

;***********************************************************************
;BOUNDARY CONDITIONS
;***********************************************************************
i ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @max_z dis @z_pladis
solve rat 5e-5
save ELASTIC_INI.sav
pause
ini xdis 0 ydis 0 zdis 0 ini xvel 0 yvel 0 zvel 0

;***********************************************************************
;Equilibrium plastic
;***********************************************************************
model mech cavehoek

prop rockfrag_aspect CRrockfrag_aspect
trop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci
;
solve rat 5e-5
save Insitu_stress.sav
ret
1A.3.2 Insitu_stress_generic

new

set fish safe off config cppudm

;*********************************************************************
; Insitu stress for different width of the panel
;*********************************************************************
;CHANGE MODEL PROPERTIES HERE
;**********************************************************************

def mod_prop ;MODEL GEOMETRY

cave_angle = 50. ;cave front angle

exc_inc = 20. ;thickness of excavation increments
first_excav = 160. ;(length of excavation before introducing cave)

;*********************************************************************
;ROCK MASS Properties
CRrockfrag_aspect=0.5 ;Rock Fragment Aspect Ratio
CRgsi=59 ;GSI
CRdensity= 2620 ;Rock mass Density CRyoung_intact = 52.7e9 ;Young Intact
CRm_intact= 4.83 ;Mb CRhb_sci=133.62e6 ;UCS
_D = 0. ;Hoek-Brown D _
pois=0.20 ; Poissons ratio

;INITIAL STRESS CONDITIONS
;*********************************************************************

x_kfac = 1.859;ratio of horizontal to vertical stress in x-direction
y_kfac = 1.267;ratio of horizontal to vertical stress in y-direction
gravity_ = -9.81:value for gravity (down is negative)
zcoord_of_gs = 696.;thickness of rock above undercut in meters
_Density= 2620.

end

@mod_prop

;*********************************************************************

;INITIAL STRESS CONDITIONS

;*********************************************************************

def _input

; Outer layer 64 m
xmin = -616
xmax = 856
ymin = -640
ymax = 768
zmin = -264
zmax = 696

dim_ = 64

n1 = int((xmax-xmin)/dim_)
n2 = int((ymax-ymin)/dim_)
n3 = int ((zmax-zmin)/dim_)

;Coarse zoning of model, all 32 meter zones
xmin_2 = -296
xmax_2 = 536
ymin_2 = -320
ymax_2 = 576
zmin_2 = -264
zmax_2 = 376
dim_2 = 32

n1_2 = int((xmax_2-xmin_2)/dim_2)
n2_2 = int((ymax_2-ymin_2)/dim_2)
n3_2 = int ((zmax_2-zmin_2)/dim_2)

;mid-size nucleus, 16 meter zoning

dim_nuc=16
xnl = -136
xnu = 408
ynl = -160
ynu = 384
znl = -104
znu = 216

nzx = int((xnu-xnl)/dim_nuc)
nzy = int((ynu-ynl)/dim_nuc)
nzz = int((znu-znl)/dim_nuc)

;mid-size nucleus, 8 meter dim_nuc_2= 8
xnl_2 = -72
xnu_2 = 296
ynl_2 = -80
ynu_2 = 288
znl_2 = -72
znu_2 = 136
nzx_2 = int((xnu_2-xnl_2)/dim_nuc_2)
nyz_2 = int((ynu_2-ynl_2)/dim_nuc_2)
nzz_2 = int((znu_2-znl_2)/dim_nuc_2)

; Fine Zoning 4
dim_fine = 4
xfl = -40
xfu = 256
yfl = -40
yfu = 208
zfl = -48
zfu = 96

nzx2 = int((xfu-xfl)/dim_fine)
nyz2 = int((yfu-yfl)/dim_fine)
nzz2 = int((zfu-zfl)/dim_fine)

; Fine zoning 2
dim_fine1 = 2
xfl1 = -28
xfu1 = 236
yfl1 = -24
yfu1 = 200
zfl1 = -24
zfu1 = 36

nzx21 = int((xfu1-xfl1)/dim_fine1)
nyz21 = int((yfu1-yfl1)/dim_fine1)
nzz21 = int((zfu1-zfl1)/dim_fine1)

; Fine zoning 1
dim_fine2 = 1
xfl2 = -18
xfu2 = 230
yfl2 = -12
yfu2 = 192
zfl2 = -8
zfu2 = 14

nzx22 = int((xfu2-xfl2)/dim_fine2)
nyz22 = int((yfu2-yfl2)/dim_fine2)
nzz22 = int((zfu2-zfl2)/dim_fine2)

; Fine zoning 0.5
dim_fine3 = 0.5
xfl3 = -4
xfu3 = 130
yfl3 = -6
yfu3 = 186
zfl3 = -6
zfu3 = 8
nzx23 = int((xfu3-xfl3)/dim_fine3)
nzy23 = int((yfu3-yfl3)/dim_fine3)
nzz23 = int((zfu3-zfl3)/dim_fine3)

end @_input

; Generate main grid, 64m zones gen zone brick &

p0 @xmin @ymin @zmin &
p1 @xmax @ymin @zmin &
p2 @xmin @ymax @zmin &
p3 @xmin @ymin @zmax &
size @n1 @n2 @n3 group OUTER ; PAUSE ; Generate main grid, 32 m zones
delete zone range x @xmin_2 @xmax_2 y @ymin_2 @ymax_2 z @zmin_2 @zmax_2 gen zone brick &
p0 @xmin_2 @ymin_2 @zmin_2 & p1 @xmax_2 @ymin_2 @zmin_2 & p2 @xmin_2 @ymax_2 @zmin_2 & p3 @xmin_2 @ymin_2 @zmax_2 &
size @n1_2 @n2_2 @n3_2 group COARSE

; Generate the nucleus, 16 meter zones
delete zone range x @xnl @xnu y @ynl @ynu z @znl @znu
gen zone brick &
p0 @xnl @ynl @znl &
p1 @xnu @ynl @znl &
p2 @xnl @ynu @znl &
p3 @xnl @ynl @znu &
size @nzx @nzy @nzz group MED1

; Generate the nucleus, 8 meter zones
delete zone range x @xnl_2 @xnu_2 y @ynl_2 @ynu_2 z @znl_2 @znu_2 gen zone brick &
p0 @xnl_2 @ynl_2 @znl_2 & p1 @xnu_2 @ynl_2 @znl_2 & p2 @xnl_2 @ynu_2 @znl_2 & p3 @xnL_2 @ynl_2 @znu_2 &
size @nzx_2 @nzy_2 @nzz_2 group MED2

; Generate fine zone region, 4 meter zones
Delete zone range x @xfl @xfu y @yfl @yfu z @zfl @zfu

gen zone brick & p0 @xfl @yfl @zfl & p1 @xfu @yfl @zfl & p2 @xfl @yfu @zfl & p3 @xfl @yfl @zfu &
size @nzx2 @nzy2 @nzz2 group SMALL4

; Generate fine zoned region, 2 meter
Delete zone range x @xfl1 @xfu1 y @yfl1 @yfu1 z @zfl1 @zfu1

gen zone brick &
p0 @xfl1 @yfl1 @zfl1 &
p1 @xfu1 @yfl1 @zfl1 &
p2 @xfl1 @yfu1 @zfl1 &
p3 @xfl1 @yfl1 @zfu1 &
size @nzx21 @nzy21 @nzz21 group SMALL3

; Generate fine zoned region 1 meter
delete zone range x @xfl2 @xfu2 y @yfl2 @yfu2 z @zfl2 @zfu2

gen zone brick &
p0 @xfl2 @yfl2 @zfl2 &
p1 @xfu2 @yfl2 @zfl2 &
p2 @xfl2 @yfu2 @zfl2 &
p3 @xfl2 @yfl2 @zfu2 &
size @nzx22 @nzy22 @nzz22 group SMALL2

; pause

; Generate fine zoned region 0.5 meter
delete zone range x @xfl3 @xfu3 y @yfl3 @yfu3 z @zfl3 @zfu3

gen zone brick &
p0 @xfl3 @yfl3 @zfl3 &
p1 @xfu3 @yfl3 @zfl3 &
p2 @xfl3 @yfu3 @zfl3 &
p3 @xfl3 @yfl3 @zfu3 &  
size @nzx23 @ny23 @nzz23 group SMALL1

gen merge 0.1  
attach face  

;----------------------------------------------------------------------------------------------------------  
;DETERMINES LIMITS OF MODEL  
;----------------------------------------------------------------------------------------------------------  

def get_limits min_x=1e20 max_x=-1e20 min_y=1e20 max_y=-1e20 min_z=1e20 max_z=-1e20  
p_gp=gp_head  
loop while p_gp # null min_x=min(min_x,gp_xpos(p_gp)) max_x=max(max_x,gp_xpos(p_gp))  
min_y=min(min_y,gp_ypos(p_gp)) max_y=max(max_y,gp_ypos(p_gp))  
min_z=min(min_z,gp_zpos(p_gp)) max_z=max(max_z,gp_zpos(p_gp))  
p_gp=gp_next(p_gp)  
end_loop  
orig_x=(min_x+max_x)/2. orig_y=(min_y+max_y)/2. orig_z=(min_z+max_z)/2.  
x_pladis=0.1 y_pladis=0.1 z_pladis=0.1  
mid_x=(max_x - min_x)/2.+ min_x mid_y=((max_y - min_y)/2.)+ min_y  
end @get_limits  

;----------------------------------------------------------------------------------------------------------  
; Properties of rock mass  
;----------------------------------------------------------------------------------------------------------  

def _pprop  
_young=27.5e9; del macizo rocoso _  
pois=0.20 _  
bulk=_young/(3*(1-2*_pois)) _shear=_young/(2*(1+_pois))  
end @_pprop  

;----------------------------------------------------------------------------------------------------------  
;ELASTIC PROPERTIES AND EQUILIBRIUM  
;----------------------------------------------------------------------------------------------------------  
model mech elas ini density 2620.  
prop bulk _bulk shear _shear  

;----------------------------------------------------------------------------------------------------------  
;INITIAL STRESS CONDITIONS  
;----------------------------------------------------------------------------------------------------------  

def ini_stresses  
rep_density = _Density  
szz_orig=gravity_*rep_density*zcoord_of_gs sxx_orig=x_kfac*szz_orig  
syy_orig=y_kfac*szz_orig szz_grad=-1*gravity_*rep_density sxx_grad=x_kfac*szz_grad
syy_grad = y_kfac * szz_grad

set gravity 0 0 @gravity_
initial szz @szz_orig grad 0 0 @szz_grad initial sxx @sxx_orig grad 0 0 @sxx_grad initial syy @syy_orig grad 0 0 @syy_grad
end_command
end

@ini_stresses ; initialize stresses

;******************************************************
; BOUNDARY CONDITIONS
;******************************************************
ini xvel 0 yvel 0 zvel 0

fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 90 dd 0 ori 0 0 @min_z dis @z_pladis

solve rat 5e-5
save ELASTIC_INI.sav

pause

ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0

;**********************************
; Equilibrium plastic
;**********************************
model mech cavehoek

prop rockfrag_aspect CRrockfrag_aspect

prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci

solve rat 5e-5
save Insitu_stress_Generic.sav

ret
1A.3.3 Insitu_stress_pillar

* Insitu_stress_pillar.dat (Archive)

new
set fish safe off config ccppudm

;******************************************************************************
;Model for pillar evaluation for pillar of 20 m
;******************************************************************************
;CHANGE MODEL PROPERTIES HERE
;******************************************************************************
def mod_prop ;MODEL GEOMETRY

cave_angle = 50. ;cave front angle
exc_inc = 20. ;thickness of excavation increments
first_excav = 160. ;length of excavation before introducing cave
beam = 40. ;distance between cave and underctuing front
panel_width = 120. ;width of the panel
panel_length = 220. ;panel_length
undercut_height = 2. ;undercut height
;ROCK MASS Properties
CRrockfrag_aspect=0.5 ;Rock Fragment Aspect Ratio
CRgsi=59 ;GSI
CRdensity= 2620 ;Rock mass Density CRyoung_intact = 52.7e9 ;Young Intact
CRm_intact= 14.83 ;Mb
CRhb_sci=133.62e6 ;UCS
_D = 0. ;Hoek-Brown D _pois=0.20 ; Poissons ratio

;******************************************************************************
;INITIAL STRESS CONDITIONS
;******************************************************************************

x_kfac = 1.859;ratio of horizontal to vertical stress in x-direction
y_kfac = 1.267;ratio of horizontal to vertical stress in y-direction
gravity_ = -9.81;value for gravity (down is negative)
zcoord_of_gs= 696.;thickness of rock above undercut in meters
_Density= 2620.
end
@mod_prop

; OUTER zones of model, all 64 meter zones

;******************************************************************************
def _input
; Outer layer 64 m
xmin = -616
xmax = 856
ymin = -804
ymax = 1116
zmin = -264
zmax = 696
dim_ = 64
n1 = int((xmax-xmin)/dim_)
n2 = int((ymax-ymin)/dim_)
n3 = int ((zmax-zmin)/dim_)

;Coarse zoning of model, all 32 meter zones
xmin_2 = -296
xmax_2 = 536
ymin_2 = -356
ymax_2 = 668
zmin_2 = -264
zmax_2 = 376
dim_2 = 32
n1_2 = int((xmax_2-xmin_2)/dim_2)
n2_2 = int((ymax_2-ymin_2)/dim_2)
n3_2 = int ((zmax_2-zmin_2)/dim_2)

;mid-size nucleus, 16 meter zoning
dim_nuc=16
xnl = -136
xnu = 408
ynl = -164
ynu = 508
znl = -104
znu = 216
nrx = int((xnu-xnl)/dim_nuc)
nry = int((ynu-ynl)/dim_nuc)
nrz = int((znu-znl)/dim_nuc)

; mid-size nucleus, 8 meter
dim_nuc_2= 8
xnl_2 = -72
xnu_2 = 296
ynl_2 = -84
ynu_2 = 428
znl_2 = -72
znu_2 = 136
nzx_2 = int((xnu_2-xnl_2)/dim_nuc_2)
nzy_2 = int((ynu_2-ynl_2)/dim_nuc_2)
nzz_2 = int((znu_2-znl_2)/dim_nuc_2)

; Fine Zoning 4
dim_fine = 4
xfl = -40
xfu = 256
yfl = -44
yfu = 364
zfl = -48
zfu = 96
nzx2 = int((xfu-xfl)/dim_fine)
nzy2 = int((yfu-yfl)/dim_fine)
nzz2 = int((zfu-zfl)/dim_fine)

; Fine zoning 2
dim_fine1 = 2
xfl1 = -28
xfu1 = 236
yfl1 = -20
yfu1 = 300
zfl1 = -24
zfu1 = 36
nzx21 = int((xfu1-xfl1)/dim_fine1)
nzy21 = int((yfu1-yfl1)/dim_fine1)
nzz21 = int((zfu1-zfl1)/dim_fine1)

; Fine zoning 1
dim_fine2 = 1
xfl2 = -18
xfu2 = 230
yfl2 = 114
yfu2 = 152
zfl2 = -8
zfu2 = 14
nzx22 = int((xfu2-xfl2)/dim_fine2)
nzy22 = int((yfu2-yfl2)/dim_fine2)
nzz22 = int((zfu2-zfl2)/dim_fine2)

; Fine zoning 0.5
; dim_fine3 = 0.5
; xfl3 = 140
; xfu3 = 226
; yfl3 = -6
; yfu3 = 126
; zfl3 = -6
; zfu3 = 8
; nzx23 = int((xfu3-xfl3)/dim_fine3)
; nzy23 = int((yfu3-yfl3)/dim_fine3)
; nzz23 = int((zfu3-zfl3)/dim_fine3)

end @_input

; Generate main grid, 64m zones gen zone brick &

p0 @xmin @ymin @zmin &
p1 @xmax @ymin @zmin &
p2 @xmin @ymax @zmin &
p3 @xmin @ymin @zmax &
size @n1 @n2 @n3 group OUTER ; PAUSE
; Generate main grid, 32m zones
delete zone range x @xmin_2 @xmax_2 y @ymin_2 @ymax_2 z @zmin_2 @zmax_2
gen zone brick &
p0 @xmin_2 @ymin_2 @zmin_2 &
p1 @xmax_2 @ymin_2 @zmin_2 &
p2 @xmin_2 @ymax_2 @zmin_2 &
p3 @xmin_2 @ymin_2 @zmax_2 &
size @n1_2 @n2_2 @n3_2 group COARSE

; Generate the nucleus, 16 meter zones
delete zone range x @xnl @xnu y @ynl @ynu z @znl @znu
gen zone brick &
p0 @xnl @ynl @znl &
p1 @xnu @ynl @znl &
p2 @xnl @ynu @znl &
p3 @xnl @ynl @znu &
size @nzx @nzy @nzz group MED1

; Generate the nucleus, 8 meter zones
delete zone range x @xnl_2 @xnu_2 y @ynl_2 @ynu_2 z @znl_2 @znu_2
gen zone brick &
p0 @xnl_2 @ynl_2 @znl_2 &
p1 @xnu_2 @ynl_2 @znl_2 &
p2 @xnl_2 @ynu_2 @znl_2 &
p3 @xnl_2 @ynl_2 @znu_2 &
size @nx2 @ny2 @nz2 group MED2

; Generate fine zone region, 4 meter zones
Delete zone range x @xfl @xfu y @yfl @yfu z @zfl @zfu

gen zone brick & p0 @xfl @yfl @zfl & p1 @xfu @yfl @zfl & p2 @xfl @yfu @zfl & p3 @xfl @yfl @zfu &
size @nx2 @ny2 @nz2 group SMALL4

; Generate fine zoned region, 2 meter
Delete zone range x @xfl1 @xfu1 y @yfl1 @yfu1 z @zfl1 @zfu1

gen zone brick &
p0 @xfl1 @yfl1 @zfl1 &
p1 @xfu1 @yfl1 @zfl1 &
p2 @xfl1 @yfu1 @zfl1 &
p3 @xfl1 @yfl1 @zfu1 &
size @nx21 @ny21 @nz21 group SMALL3

; Generate fine zoned region 1 meter
delete zone range x @xfl2 @xfu2 y @yfl2 @yfu2 z @zfl2 @zfu2

gen zone brick &
p0 @xfl2 @yfl2 @zfl2 &
p1 @xfu2 @yfl2 @zfl2 &
p2 @xfl2 @yfu2 @zfl2 &
p3 @xfl2 @yfl2 @zfu2 &
size @nx22 @ny22 @nz22 group SMALL2

; pause

; Generate fine zoned region 0.5 meter
; delete zone range x @xfl3 @xfu3 y @yfl3 @yfu3 z @zfl3 @zfu3

; gen zone brick &

;p0 @xfl3 @yfl3 @zfl3 &
p1 @xfu3 @yfl3 @zfl3 &
;p2 @xf13 @yfu3 @zf13 &
;p3 @xf13 @yfl3 @zf13 &
:size @nx23 @ny23 @nzz23 group SMALL1

gen merge 0.1

attach face

;********************************************************************
;DETERMINES LIMITS OF MODEL
;********************************************************************
def get_limits min_x=1e20 max_x=-1e20 min_y=1e20 max_y=-1e20 min_z=1e20 max_z=-1e20
p_gp=gp_head
loop while p_gp # null min_x=min(min_x,gp_xpos(p_gp)) max_x=max(max_x,gp_xpos(p_gp))
min_y=min(min_y,gp_ypos(p_gp)) max_y=max(max_y,gp_ypos(p_gp))
min_z=min(min_z,gp_zpos(p_gp)) max_z=max(max_z,gp_zpos(p_gp)) p_gp=gp_next(p_gp)
end_loop orig_x=(min_x+max_x)/2. orig_y=(min_y+max_y)/2. orig_z=(min_z+max_z)/2.
x_pladis=0.1 y_pladis=0.1 z_pladis=0.1
mid_x=((max_x-min_x)/2.)+ min_x mid_y=((max_y-min_y)/2.)+ min_y
end @get_limits

;********************************************************************
;Properties of rock mass
;********************************************************************
def _pprop
_young=27.5e9; del macizo rocoso
_pois=0.20
_bulk=_young/(3*(1-2*_pois)) _shear=_young/(2*(1+_pois))
end @_pprop

;********************************************************************
;ELASTIC PROPERTIES AND EQUILIBRIUM
;********************************************************************
model mech elas ini density 2620.
prop bulk _bulk shear _shear

;********************************************************************
;INITIAL STRESS CONDITIONS
;********************************************************************
def ini_stresses
rep_density =Density
szz_orig=gravity_*rep_density*zcoord_of_gs
sxx_orig=x_kfac*szz_orig
syy_orig=y_kfac*szz_orig
szz_grad=-1*gravity_*rep_density
sxx_grad=x_kfac*szz_grad
syy_grad=y_kfac*szz_grad

command

set gravity 0 0 @gravity_
initial szz @szz_orig grad 0 0 @szz_grad
initial sxx @sxx_orig grad 0 0 @sxx_grad
initial syy @syy_orig grad 0 0 @syy_grad
end_command end

@ini_stresses ;initialize stresses

;*****************************************************************************************************************************
;BOUNDARY CONDITIONS
;*****************************************************************************************************************************
ini xvel 0 yvel 0 zvel 0

fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix x range plane dip 0 dd 0 ori 0 0 @max_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis

solve rat 5e-5
save ELASTIC_INI.sav
pause
ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0

;*****************************************************************************************************************************
;Equilibrium plastic
;*****************************************************************************************************************************
model mech cavehoek
prop rockfrag_aspect CRockfrag_aspect
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRM_intact hb_sci CRhb_sci

solve rat 5e-5

sav Insitu_stress_pillar.sav
ret
1A.4 Pillar Between Panels

1A.4.1 Advanc_precaving_panel_B

*Advanc_precaving_Panel_B.dat (Archive)

new
set fish safe off config cppudm

;*******************************************************************
rest CAVE_TO_SURFACE.sav
;*******************************************************************
:Advance Panel B, molde with pillar
;*******************************************************************
:Definition of parameters

;***********************************************************
;Now with advance of 20 m
;***********************************************************

def mod_prop

cave_angle = 50. ;cave front angle
ext_inc = 20. ;thickness of excavation increments
first_excav = 160. ;(length of excavation before introducing cave)
beam = 40. ;
panel_width = 120. ;width of the panel
panel_length = 220. ;panel_length
undercut_height = 2. ;undercut height

;ROCK MASS Properties
CRrockfrag_aspect = 0.5
CRgsi = 59
CRdensity = 2620
CRyoung_intact = 52.7e9 CRm_intact = 14.83 CRhb_sci = 133.62e6
end @mod_prop

;******************************************
;excavate the first increment (up to first_excav)
;******************************************
def exc_first

dist = 0.
inc = 1.
cont = 1

dist_pillar = 20
loop while dist < 100; first_excav
dist = exc_inc * inc
f_name=string(inc)+"_FIRST_EXCAV.sav"
command
group zone EXCAVATE range x 4 dist y 148 268 z 0 undercut_height
model mech null range group EXCAVATE
:step 5 solve

end_command
:list inc

if dist>40
;cont=1
;dist_pillar=20 command
group zone PILLAR range x 4 dist_pillar y 124 144 z 0 undercut_height
model mech null range group PILLAR
:step 5
solve

end_command
cont=cont+1 dist_pillar=dist_pillar+20
end_if

command sav f_name end_command

inc=inc+1
end_loop
end @exc_first

ret
1A.4.2 Advanc_precaving_pillar

*Advanc_precaving_pillar.dat (Archive)

new
set fish safe off config cppudm

;*******************************************************************
rest Insitu_stress_pillar.sav; restore initial stress condition
;*******************************************************************
;Advance Panel A, model with pillar
;*******************************************************************
;Displacement to zero
;*******************************************************************
ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0
ini state 0
;*******************************************************************
;defines the model boundary conditions
;*******************************************************************
def bound
command free x free y free z

ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix x range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_z 0 dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @max_z dis @z_pladis
end_command
end
;*******************************************************************
;change model properties here
;assumes bottom left corner of the panel in plan view is at coordinate 0,0,0
;*******************************************************************
;Definition for parameters
;*******************************************************************
;Now with advance of 20 m
;*******************************************************************
def mod_prop

cave_angle = 50. ;cave front angle
exc_inc = 20. ;thickness of excavation increments
first_excav = 160. ;(length of excavation before introducing cave)
beam = 40.;
panel_width = 120.; \text{width of the panel}
panel_length = 220.; \text{panel_length}
undercut_height = 2.; \text{undercut height}

; ROCK MASS Properties
CRrockfrag_aspect=0.5
CRgsi=59
CRdensity= 2620
CRyoung_intact = 52.7e9 CRm_intact=14.83 CRhb_sci=133.62e6
end @mod_prop

def Exc Long_tunn_xs=268 Long_tunn_ys=220 Pillar_Width=20
end

@Exc

;************************************************************************************************************
/* Excavation of tunnels */
;************************************************************************************************************
group TUNNEL_1 RANGE X 0 4 y 0 Long_tunn_xs z 0 4
group TUNNEL_2 RANGE X 220 224 y 0 Long_tunn_xs z 0 4
group TUNNEL_3 RANGE X 4 220 y 120 124 z 0 4
group TUNNEL_4 RANGE X 4 220 y 144 148 z 0 4
model mech null range group TUNNEL_1
model mech null range group TUNNEL_2
model mech null range group TUNNEL_3
model mech null range group TUNNEL_4

solve
save Initial_Tunnels.sav

;************************************************************************************************************
/* Histories */
;************************************************************************************************************
hist id 2 gp zdisp 5 134 1 nstep 50;
hist id 3 gp zdisp 15 134 1 nstep 50;
hist id 4 gp zdisp 25 134 1 nstep 50;
hist id 5 gp zdisp 50 134 1 nstep 50;
hist id 6 gp zdisp 100 134 1 nstep 50;
hist id 7 gp zdisp 150 134 1 nstep 50;
hist id 8 gp zdisp 200 134 1 nstep 50;

hist id 9 zone szz 5 134 1 nstep 50;
hist id 10 zone szz 15 134 1 nstep 50;
hist id 11 zone szz 25 134 1 nstep 50;
hist id 12 zone szz 50 134 1 nstep 50;
hist id 13 zone szz 100 134 1 nstep 50;
hist id 14 zone szz 150 134 1 nstep 50;
hist id 15 zone szz 200 134 1 nstep 50;

;*******************************************************************************
; excavate the first increment (up to first_excav)
;*******************************************************************************
def exc_first

dist=0.
inc = 1.
loop while dist < first_excav
dist = exc_inc * inc
f_name=string(inc)+"_FIRST_EXCAV.sav" command
group zone EXCAVATE range x 0 dist y 0 panel_width z 0 undercut_height
model mech null range group EXCAVATE

solve
sav f_name list inc end_command inc=inc+1

end_loop

end
@exc_first

ret
1A.4.3 Caving_40_120_pillar

*Caving_40_120_Pillar.dat (Archive)

new

set fish safe off

cfg config cppudm ; Configuration for cavehoek constitutive model

rest 8_FIRST_EXCAV.sav

;**********************************************************************
;Growth caving Panel A
;**********************************************************************
;defines the model boundary conditions
;**********************************************************************
def bound command

free x
free y
free z

ini xvel 0 yvel 0 zvel 0

fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix x range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix x range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis

end_command

end

;**********************************************************************
;grows the cave until it reaches (first_excav - beam). Change the properties inside of caving to
residual 1.;volumetric stress increment (vsi) 0.67 ;fixed all mesh except the CAVE material,
initial stress zero and solve.; Free the mesh and restore the original border conditions. Save each
increment of the triangle.
;**********************************************************************
def cgrowth
_x= 0
num_inc=(first_excav-beam)/exc_inc ; Number of steps pre-caving cc_=0
loop while cc_ < num_inc cc_=cc_+1
file_name=string(cc_)+"PRE_CAVE.sav" _x=_x+exc_inc ; Increase of span

command
group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd
90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 &
plane below dip 90 dd 0 origin 0 123 0 z 0 5000
model mech cavehoek range group CAVE
prop rockfrag_aspect CRrockfrag_aspect range group CAVE
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci
CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not ; fixed all mesh except CAVE volum
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE; All stress 0
step 50 ;step 10 solve

@end

loop while cc_ < num_inc-1 cc_=cc_+1
file_name=string(cc_)+_CAVE_and_UNDERCUT.sav" _x=_x+exc_inc
dist = _x + beam f_name=string(cc_)+"_EXCAV.sav" command

@bound
step 50 ;step 10 solve
sav file_name end_command

end_loop
end @cgrowth

;********************************************************************
;CGROWTH2 :grows the cave and undercut until end of panel reached; Advace with
undercutting area an ;caving area;with 20 m of increment excave 20 m and save this new
advance. Change the properties of ;new volume incorporate to caving, ;fix the mesh except the
CAVE group. Stress to zero and run. Continuo with the same process above.
;********************************************************************
def cgrowth2 num_inc=(panel_length-first_excav)/exc_inc cc_=0

loop while cc_ < num_inc-1 cc_=cc_+1
file_name=string(cc_)+_CAVE_and_UNDERCUT.sav" _x=_x+exc_inc
dist = _x + beam f_name=string(cc_)+"_EXCAV.sav" command
group EXCAVATE range z 0 undercut_height y 0 panel_width x 0 dist group CAVE not
model mech null range group EXCAVATE
solve
;step 10 sav f_name
group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 plane below dip 90 dd 0 origin 0 123 0 z 0 5000

model mech cavehoek range group CAVE

prop rockfrag_aspect CRrockfrag_aspect range group CAVE
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE step 50
;step 10 solve @bound step 50
;step 10 solve
sav file_name end_command
end_loop end @cgrowth2

;*******************************************************************
;grows cave to surface: In the end of the undercut advance the cave increase to 90 degree(vertical 
;subsidence the same process above, change the properties, fix the mes except CAVE group 
;velociti to zero run. Free ;the mesh and restore the border conditions. 
;*******************************************************************

group zone CAVE range plane below dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 0 0 plane below dip 90 dd 0 origin 0 123 0 z 0 5000

model mech cavehoek range group CAVE

prop rockfrag_aspect CRrockfrag_aspect range group CAVE
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not
fix y range group CAVE not
fix z range group CAVE not
ini stress 0 range group CAVE step 50
;step 10 solve @bound step 50
;step 10 solve
sav CAVE_TO_SURFACE.sav ret
1A.4.4 Caving_40_120_pillar_B

*Caving_40_120_Pillar_B.dat (Archive)

new

set fish safe off

config cppudm ; Configuration for cavehoek contitutive model

rest 5_FIRST_EXCAV.sav

;****************************************************************************
; Advance panel B with caving propagation
;****************************************************************************
;defines the model boundary conditions
;*****************************************************************************
def bound command
free x
free y
free z

ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 0 @max_y 0 dis @y_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis

end_command

end

;*****************************************************************************
;grows the cave until it reaches (first_excav - beam). Change the properties inside of caving to
residual 1, volumetric stress increment (vsi) 0.67 ;fixed all mesh except the CAVE material,
initial stress zero and solve. Free the mesh and restore the original border conditions. Save each
increment of the triangle.
;*****************************************************************************
def cgrowth
_x= 0

num_inc=(first_excav-beam)/exc_inc ; Number of steps pre-caving
cc_=0

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loop while cc_ < 3 ; pongo el nuevo borde 100 en vez de 120 num_inc
cc_=cc_+1 file_name=string(cc_)+"PRE_CAVE.sav" _x=_x+exc_inc ; Increase of span
command
group zone CAVE range plane below dip cave_angle dd 90 origin _x 0 0 plane above dip 90 dd 90 origin _x 0 0 plane above dip 90 dd 0 origin 0 123 0 &
plane below dip 90 dd 0 origin 0 268 0 z 0 5000 ; lo ajusto al panel B partiendo a un metro de la
cara del panel A

model mech cavehoek range group CAVE
prop rockfrag_aspect CRrockfrag_aspect range group CAVE
prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci
CRhb_sci range group CAVE
prop start_at_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not ;
fixed all mesh except CAVE volum
fix y range group CAVE not

fix z range group CAVE not
ini stress 0 range group CAVE; All stress 0
step 400
;step 5 ;solve
@bound
step 400 ;step 5
;solve
sav file_name end_command

end_loop
end @cgrowth ;pause

;*******************************************************************************
;CGROWTH2 :grows the cave and undercut until end of panel reached; Advace with
undercutting area an caving area; with 20 m of increment excave 20 m and save this new
advance. Change the properties of new volume incorporate to caving, ;fix the mesh except the
CAVE group. Stress to zero and run. Continueo with the same process above.
;*******************************************************************************
def cgrowth2

cc_=0
dist=100 dist_pillar=80
loop while cc_ < 5 ;num_inc-1
cc_=cc_+1
file_name=string(cc_)+"_CAVE_and_UNDERCUT.sav" _x=_x+exc_inc

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\_xx=\_x+20
dist = dist + exc\_inc f\_name=string(cc\_)+"\_EXCAV.sav"

command
group EXCAVATE range z 0 undercut\_height y 148 268 x \_xx dist group CAVE not; excava
panel B de y 148 a 268
model mech null range group EXCAVATE

;solve
step 400

end\_command

;dist\_pillar=80
ini\_pillar=dist\_pillar-20

command
group zone PILLAR range x ini\_pillar dist\_pillar y 124 144 z 0 undercut\_height
model mech null range group PILLAR
step 400
;solve
sav f\_name
end\_command broken_=dist\_pillar

dist\_pillar=dist\_pillar+20

;step 10 ;sav f\_name

command
group zone CAVE range plane below dip cave\_angle dd 90 origin broken\_ 0 0 plane above dip
90 dd 90 origin 0 0 0 plane above dip 90 dd 0 origin 0 123 0 &
plane below dip 90 dd 0 origin 0 268 0 z 0 5000 ; lo ajusto al panel B partiendo a un metro de la
cara del panel A

model mech cavehoek range group CAVE
prop rockfrag\_aspect CRrockfrag\_aspect range group CAVE
prop gsi CRgsi density CRdensity young\_intact CRyoung\_intact m\_intact CRm\_intact hb\_sci
CRhb\_sci range group CAVE
prop start\_at\_residual 1 vsi 0.67 range group CAVE
ini vel 0,0,0
fix x range group CAVE not
; fixed all mesh except CAVE volum
fix y range group CAVE not

fix z range group CAVE not
ini stress 0 range group CAVE; All stress 0
;step 50
step 300 ;solve
;end_comand @bound
;step 50 step 400 ;solve
sav file_name end_command

;sav file_name ;end_command
end_loop
end @cgrowth2 ret ;pause
;*************************************************************************
;grows cave to surface: In the end of the undercut advance the cave increase to 90 degree-vertical
subsidence; the same process above, change the properties, fix the mes except CAVE group
velociti to zero run. Free the mesh and restore the border conditions.
;**************************************************************************
;group zone CAVE range plane below dip 90 dd 90 origin 200 0 0 plane above dip 90 dd 90
origin 0 0 0 plane above dip 90 dd 0 origin 0 00 plane below dip 90 dd 0 origin 0 123 0 z 0 5000
;model mech cavehoek range group CAVE

;prop rockfrag_aspect CRockfrag_aspect range group CAVE
;prop gsi CRgsi density CRdensity young_intact CRyoung_intact m_intact CRm_intact hb_sci
CRhb_sci range group CAVE
;prop start_at_residual 1 vsi 0.67 range group CAVE
;ini vel 0,0,0
;fix x range group CAVE not
;fix y range group CAVE not
;fix z range group CAVE not
;ini stress 0 range group CAVE
;step 50
;step 5
;solve ;@bound ;step 50
;step 10 ;solve
;sav CAVE_TO_SURFACE.sav
;ret
1A.5 Pre Caving

1A.5.1 Advanc_precaving

*Advanc_precaving.dat (Archive)

new
set fish safe off config cppudm

;*******************************************************************
rest Insitu_stress.sav; restore initial stress condition
;*******************************************************************
;Advance Panel A
;*******************************************************************
;Displacement to zero
;*******************************************************************
ini xdis 0 ydis 0 zdis 0
ini xvel 0 yvel 0 zvel 0 ini state 0
;*******************************************************************
;defines the model boundary conditions
;*******************************************************************
def bound
cmd
free x
free y
free z
ini xvel 0 yvel 0 zvel 0
fix x range plane dip 90 dd 90 ori @min_x 0 0 dis @x_pladis
fix x range plane dip 90 dd 90 ori @max_x 0 0 dis @x_pladis
fix x range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix y range plane dip 90 dd 0 ori 0 @min_y 0 dis @y_pladis
fix y range plane dip 90 dd 0 ori 0 @max_y 0 dis @y_pladis
fix y range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
fix z range plane dip 0 dd 0 ori 0 0 @min_z dis @z_pladis
end_cmd
end
;*******************************************************************
;change model properties here

; assumes bottom left corner of the panel in plan view is at coordinate 0,0,0
;*******************************************************************
; Definiton fo parameters
;*******************************************************************
def mod_prop
cave_angle = 50. ; cave front angle

cave_angle = 50. ; cave front angle

exc_inc = 20. ; thickness of excavation increments
first_excav = 160. ; (length of excavation before introducing cave) beam = 40. ;
panel_width = 120. ; width of the panel panel_length = 220. ; panel_length undercut_height = 2. ; undercut height

; ROCK MASS Properties
CRRockfrag_aspect=0.5
CRgsi=59
CRdensity= 2620
CRyoung_intact = 52.7e9 CRm_intact= 14.83 CRhb_sci=133.62e6
end @mod_prop

;********************************************************************
; excavate the first increment (up to first_excav)
;********************************************************************
def exc_first
dist=0.
inc = 1.
loop while dist < first_excav
dist = exc_inc * inc
f_name=string(inc)+"_FIRST_EXCAV.sav" command
group zone EXCAVATE range x 0 dist y 0 panel_width z 0 undercut_height
model mech null range group EXCAVATE

solve
sav f_name
list inc end_command inc=inc+1

end_loop

end
@exc_first

ret