Ore-Skin Design to Control Sloughage in Underground Open Stope Mining

by

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A thesis submitted in conformity with the requirements for the degree of Master of Applied Science
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2015

Abstract

In open stope underground mines, stope hangingwall (HW) sloughage is typically the main source of ore dilution. In cases where the immediate stope HW consists of a weak rock mass, the HW sloughage can be substantial and can result in increased dilution rates which have significant economic impact on the mine operation.

This thesis presents an alternative design method which requires leaving a certain thickness of ore unmined (i.e., an ore-skin) to support the weak adjacent/overlying hangingwall. The ore-skin design approach is presented in the context of a hypothetical open stope mine to determine the feasibility of implementing this alternative design method to control HW sloughage. Both the economics and the geomechanical aspects of the ore-skin design are investigated. This study aims to determine the importance of considering alternative design approaches to optimize the open stope mining process.
Acknowledgments

Firstly, I would like to thank my supervisor Professor Kamran Esmaeili for his trust and for the opportunity he gave me to work with him on this project. I also wish to express my sincere gratitude for his support, guidance and patience throughout the entire time of my study.

I would also like to thank my colleagues on the 4th floor for the ideas, the conversation, but also the coffees and the laughter. A special thanks to Gambino.

Finally, I wish to thank my family for the unconditional support, the continuous encouragement, and the precious advices.

So much can’t be express in so few words. Thanks to all who helped me with this.

À Anne-Lynne et Bruno
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CHAPTER 1
INTRODUCTION

The competitive global economy and the increasing demand for raw materials has placed significant pressure on the mining industry to increase its production while lowering the cost of mineral extraction. This has driven the efforts for the development of cost-effective approaches and the improvement of mining techniques.

This thesis focuses on the optimization of stope design in underground open stope mines. In cases where the immediate stope hangingwall (HW) consists of weak rock, the HW sloughage can be substantial, resulting in very high dilution rates which can significantly increase the direct and indirect costs related to its mining. To address this problem, an alternative design method is proposed: the ore-skin design. The ore-skin design requires leaving in place a certain thickness of ore unmined along the HW or the footwall (FW) to control stope wall instability and consequently control ore dilution. This alternative design approach becomes economically feasible when the immediate HW or FW consists of weak rock and the ore-body consists of a relatively competent material.

1.1 Motivation

The overall value of any mining project must be positive and also tend to the highest possible value in order to be feasible and competitive in the present economy. The reduction of the cost of dilution has always been an essential challenge in underground mine projects. Hangingwall sloughage is the main source of ore dilution in open stope underground mines (Ran, 2002; Henning and Mitri, 2007); therefore, adopting an appropriate and optimized HW sloughage
control strategy would be required to increase the value of the stope extraction. The appropriate and optimized strategy is generally defined as the one which offers the best economic outcome.

Several HW sloughage control strategies are available and can be categorized as either using reinforcing techniques or using designing and planning approaches. In the presence of a weak HW material and a competent ore-body, it is believed that the ore-skin design strategy could offer the optimal economic outcome when compared to other approaches. Nonetheless, the advantage of implementing one strategy over another must be assessed using a cost-benefit analysis.

Despite the considerable number of studies investigating the causes of HW sloughage and ore dilution in open stope underground mines, there has been no attempt to evaluate the effect of implementing alternative design approaches, such as the effect of ore-skin design on the stability of the stope HW and on the resulting ore dilution. This work focuses on the economic and geomechanical aspects of implementing the ore-skin design as the feasibility of this design approach is a trade-off between both aspects. Overall, this study results in the development of an efficient decision-making tool regarding the viability of implementing the ore-skin design.

1.2 Research Objectives

This thesis will contribute to our understanding of the viability of the ore-skin design approach as an alternative technique for HW sloughage control in underground mines. This primary objective is divided into two secondary objectives:

(1) Propose an assessment method which determines the break-even ore-skin thickness that can remain unmined while ensuring the economic viability of the stope design.

(2) Conduct empirical, analytical and numerical methods to assess the minimum ore-skin thickness that ensures the structural integrity of the stope design.
1.3 Methodology

The methodology employed to meet the objectives is summarized by the following procedures. Firstly, a literature review is conducted. The mechanism of hangingwall sloughage in open stope underground mines was reviewed and the common tools that are used in the industry to predict and control the hangingwall sloughage were identified. This is followed by the economic assessment of the ore-skin design. Using the cost-benefit approach, an economic assessment tool was developed to determine the maximum ore-skin thickness that can be left unmined. The next step consists of the geomechanical assessment of the ore-skin design. Different empirical, analytical and numerical methods were conducted to assess the effect of the ore-skin thickness on the stability of open stopes and to determine the minimum geomechanically stable ore-skin thickness. The economic viability and geomechanical analysis of the ore-skin design approach was investigated for a hypothetical Canadian underground gold mine throughout the entire thesis as a didactic example.

1.4 Thesis Outline

This thesis is divided into a total of five chapters and this section provides an overview of their content.

- The current chapter introduces the scope of the study, the objectives, the methodology, and the structure of this thesis.
- Chapter 2 presents the necessary information on the open stope mining method, the importance of controlling HW sloughage in open stopes and the ore-skin design technique. The mechanisms of HW sloughage are also discussed, and the HW stability assessment methods are reviewed. Finally, the different techniques that are used to control HW sloughage are explained.
• Chapter 3 presents the methodology that aims determining the feasibility of the ore-skin design technique from an economic point of view.

• Chapter 4 illustrates the application of empirical, analytical and numerical approaches which aims to determine the minimal allowable ore-skin thickness to be left unmined to retain the stability of the adjacent/overlying HW.

• Chapter 5 provides a summary and conclusion of the research and includes recommendations for future research.
CHAPTER 2
HANGINGWALL STABILITY ANALYSIS IN OPEN STOPE UNDERGROUND MINES

2.1 Background

2.1.1 Open stope mining method

The open stope mining technique is a bulk underground mining method typically employed to mine tabular ore-bodies. The mining method consists of the extraction of large blocks of ore using mechanized equipment. These large blocks of extracted ore are called stopes and they are sequenced division of the ore-body.

The open stope mining approach is a “non-entry method”, making it one of the safest underground mining methods. Each stope requires a minimum of two developments: a top sill and a bottom sill (Figure 1). The large block of ore that is to be mined is mucked using drilling and blasting techniques that are operated from the top sill development, advancing downward to the bottom sill. After the stope has been blasted, the broken ore is extracted from the bottom sill. The stope remains generally open and unsupported during the entire extraction process. Once the broken ore is completely extracted, the stope is generally backfilled with waste rock material to provide support to the surrounding stope walls. Figure 1 illustrates the basic geometry of an underground open stope.
Figure 1. Basic geometry of an underground open stope.

The open stope mining method has grown in popularity since the 1970s and at the end of the 20th century, it was reported that more than 50% of all Canada underground metal mines ore production was extracted from underground open stoping mines (Pakalnis et al., 1995). The popularity of the open stope method is due to the high production rates that it can achieve. Nonetheless, the operation of a large unsupported underground opening is often accompanied by instability issues that can lead to significant dilution rates. This can be problematic because high dilution rates can results in the suspension of a mine operation or to its closure (Tintor, 1988; Scoble and Moss, 1994; Revey, 1998; Stewart et al., 2005).
2.1.2 Dilution and its cost

In the context of open stope mining, dilution is defined by the amount of waste material that is drawn from the stope and enters the ore processing stream, and which lowers the average grade of ore, known as the head grade. Waste material refers to the material whose metal content is below the acceptable limit (i.e., cut-off grade). Based on a survey of Canadian mines, Pakalnis (1986) identified that a total of ten different definitions of dilution are used in the mining industry. In 1995, the use of one of these definitions was recommended by the same author as a common measure for being more sensitive to the amount of waste material coming from outside the mining limits (i.e., sloughage or overbreak) (Pakalnis et al., 1995). It defines the dilution as follows:

\[
\text{Dilution} = \frac{\text{Tons waste mined}}{\text{Tons ore mined}}
\]  

(1)

This is the definition which will be considered throughout the thesis. This measure of dilution does not account for the origin of the waste material; but can refer to either a measure of planned dilution or unplanned dilution. The planned dilution is due to the waste material that is included in the designed mining limits and that will be extracted with the ore (Figure 2). Normally, this measure is evaluated and integrated in an economic analysis by which the profitability of the exploitation is confirmed. Planned dilution is considered unavoidable as it arises from the complexity of the ore body shape and the precision of the mining techniques.
Alternatively, unplanned dilution refers to the waste material that is located beyond the stope boundary, in the HW or FW, and which falls into the stope and enters the ore processing stream. The term overbreak is synonymous with unplanned dilution and can be used interchangeably. Scoble and Moss (1994) stated that there are three possible sources of unplanned dilution including: blast-induced overbreak, sloughing of an unstable stope wall, and backfill material from an adjacent stope that could inadvertently be excavated from the stope.

The amount of waste material and hence the level of dilution to be expected in the mine is controlled by a series of different factors. These factors derive directly from the sequence of exploration, mining, and processing operations. This topic is well discussed in a paper presented by Scoble and Moss (1994). More recently, de la Vergne (2014) successfully developed a fishbone chart which summarizes the main factors influencing dilution (Figure 3).
Figure 3. Fishbone chart of the dilution inducing factors (after de la Vergne, 2014).

The effect of dilution on the economics of an open stope mining operation is well understood. In general, its impact on the mine economics is due to the direct and indirect costs associated with the excavation and handling of the waste material as well as its processing. The direct costs of dilution include the cost of drilling, blasting, mucking, hauling, crushing, hoisting, milling, and treatment of the waste material and the indirect costs include delay in stope cycle, increased volume of tailings, additional backfill material for the stope, etc. These costs can become significant when there is excessive ore dilution and can result in an uneconomic mine.

Hangingwall sloughage has been identified as the dominant source of unplanned dilution in open stope underground mines (Ran, 2002; Henning, 2007; Saeedi et al., 2010). Thus, reducing the volume of waste material due to HW sloughage is essential for improving the profitability of the open stope design. However, to reduce the volume of sloughage from the HW, it is important to understand the mechanism of HW sloughage.
2.2 Hangingwall Sloughage

Hangingwall sloughage is related to the detachment of unstable material located beyond the designed stope boundaries, in the overhanging zone of the designed limit (Figure 2), which falls into the stope and enters the mining process. It is thus directly related to the stability of the material located in the HW.

When excavating a volume of rock, such as a stope, the self-supporting effect of the extracted volume is removed and the stress is redistributed in the adjacent rock. Potvin (1988) stated that the induced stresses can result in a zone of relaxation or a zone of compression around the stope. A conceptual stress redistribution response to an excavation is resented by Hoek and Brown (1980) (Figure 4). The stress streamlines bypass the opening just as water would when flowing around an obstacle. The zone where the streamlines separate to bypass the opening is analogous to the zone of relaxation and the zone where they crowd is analogous to the zone of compression (i.e., stress concentration).

Figure 4. Conceptual stress flow around an excavation (after Hoek and Brown, 1980).
The way a rock mass will respond to the induced stress state is very complex due the variable nature of the medium and also because the response of the rock mass is a result of the interaction between the medium, the opening geometry and the stress state. After studying HW sloughage in several underground open stope mines, Capes (2009) identified three typical types of sloughing failure mode. These include:

1. Crushing or Buckling of intact rock followed by a gravity failure
2. Shear failure along joints due to a stress change leading to the detachment of a block from the rock mass
3. Separation of joints due to gravity and a lack of confining stress leading to detachment of block from the rock mass

As previously mentioned, this study focuses on the HW sloughage control when the HW consists of a weak rock mass and the ore-body is relatively competent. At the scale of a stope excavation, it is reasonable to assume that the medium will always be jointed or highly jointed. In such medium, discontinuities are generally the weakest component of the rock mass and therefore, represent an important stability-controlling parameter. Potvin (1988) stated that unless the average block size of the rock mass is of several cubic meters (i.e., the discontinuities are widely spaced), the failure at the walls of an opening in the compressed or relaxed zone will occur by the unravelling of the rock blocks. Thus, if a weak HW is present, failure by unraveling is expected, referring to the third mode of failure observed by Capes (2009). Finally, both Potvin (1988) and Capes (2006a; 2009) observed that in such a medium (i.e., weak or highly fractured HW), the HW failure will occur either until the blocks interlock and form a stable arched shape or until a higher quality rock mass is met (Figure 5).
Additionally, at the opening of a stope, the exposed surfaces (i.e., free surfaces) allow the release of stress, lowering the minor principal stress to zero (i.e., unconfined stress state) or less (i.e., tensile stress state). Consequently, the stresses tangential to the stope wall are reduced (i.e., relaxed) in comparison to the initial in situ conditions, creating the relaxation zone which can be considered as an unconfined environment that develops around the opening.

This relaxed stress state allows the movement of the blocks located within the zone. Effectively, the material located within the relaxation zone is more sensitive to the influence of...
gravity as there is no confining stress (i.e., clamping stress). Hence, the gravitational forces are more important for the stope stability than the load bearing capacity of the medium, causing the structural geometry of the rock mass and the shear strength of the discontinuities to be the major controlling parameters for HW stability.

The instability zone generated around an excavation is therefore closely related to the relaxation zone induced by the opening. This is why the material contained within this zone is commonly considered to be the unstable material or the potential sloughage zone (Clark, 1998; Wang, 2004; Henning, 2007; Wang et al., 2007; Hughes et al., 2010; Saeedi et al., 2010).

Thus, sloughage can be seen as the result of a stress relaxation that goes below a critical stress state, resulting in the unravelling and falling of rock blocks. Consequently, the volume of HW sloughage is controlled by both the characteristics of the rock mass and the stress conditions and as a result, the major failure mechanism is the structurally-controlled gravity-driven failure mode. The next section will discuss the main factors influencing the extent of HW sloughage.

Other factors which could be categorized as operational factors, such as drilling and blasting and extraction time, also have an influence on the HW sloughage. However, these operational factors are not discussed, as their influence on the volume of HW sloughage is independent of the control strategy employed for the open stope HW sloughage.

2.2.1 Factors influencing stope hangingwall sloughage

When a stope is excavated, the in situ stress state is perturbed and a new induced stress state develops around the opening. A conceptual stress redistribution response to an excavation is presented by Hoek and Brown (1980) (Figure 4). The magnitude and orientation of the component of this new induced stress field is dependent of three factors: the in situ stress field, the opening geometry and the constitutive behaviour of the material which affects the
redistribution of the stresses around the opening. These factors control the relaxation zone that forms around the stope walls and thereby they influence the extent of HW sloughage. The following sections summarize the interactive influence of these factors on the volume of HW sloughage.

2.2.1.1 In situ stress environment

If the in-situ stresses tensor is known, the induced stress at the surrounding of the opening can be evaluated either by analytical solution or by using numerical models. In all cases, the magnitude and orientation of the in situ stresses has been shown to greatly influence the relaxation zone (Clark, 1998). For example, for similar geometries, the greater the stress regime is, the deeper the relaxation zone extends into the surrounding rock mass (Clark, 1998; Wang et al., 2007; Saeedi et al., 2010).

The orientation of the principal stresses around an opening is also an important factor. Potvin (1988) studied the influence of the orientation of the stresses on the stope and showed that if the maximal horizontal stress is perpendicular to the stope strike, the HW and FW will be in a relaxed state while the endwalls will be in a compressive state. These observations were also experimentally confirmed by Kaiser et al. (2001); after the excavation of a stope, the stresses perpendicular to the FW and HW were relaxed while the stress at the back of the stope had become the major principal stress, putting the endwalls in a compressive stress state (Figure 6).
In the Canadian Shield, the minimum principal stress is nearly vertical and the two other principal stresses are sub-horizontal (Arjang and Herget, 1997). In addition, during the mining progression along the strike of the ore, the maximum principal stress tends to rotate perpendicular to the ore-body strike. Therefore, it is reasonable to consider that the maximum horizontal pre-mining stress will generally be perpendicular to the stope strike (Potvin, 1988; Wang et al., 2007) (Figure 6).

Thus, in general, the HW and FW are in a relaxed stress state in underground mines in the Canadian Shield, which may explain the importance of HW sloughage as a significant source of dilution in open stope mines.

### 2.2.1.2 Stope geometry

The stope geometry has an important influence on the relaxation zone that forms around the stope walls and in turn, on the extent of HW sloughage and consequently the dilution. Potvin (1988) stated that the relative shape of the stope surface is the most influential parameters on the development of a relaxation zone. Using analytical, numerical, and observational methods, it has been shown that the geometry of a stope can greatly influence the average depth of HW relaxation zone (Clark, 1998; Wang, 2004; Henning, 2007; Wang et al., 2007; Hughes et al., 2010).
The primary geometric components influencing the extent of HW sloughage are the stope height, strike length, dip angle and shadow zones (i.e., zones where the stress state is reduced).

The greater the exposed surface is, the further the relaxation zone will extend into the rock mass. The surface of a stope HW depends on the strike length of the stope and the stope height along the HW. One measure that is commonly used in the industry to express the exposed surface of the stope HW is the hydraulic radius (HR). The HR is defined as the area of the HW surface divided by its perimeter:

\[
HR = \frac{A}{P}
\]

where \(A\) is the cross sectional area of surface analysed (i.e. stope HW); and \(P\) is the perimeter of surface analysed.

Evidence from underground open stope mines indicates that increasing the HR of the stope HW will increase the extent of HW sloughage (Clark, 1998).

The stope inclination (i.e., dip) also influences the stress redistribution and consequently, the depth of HW sloughage. The shallower the dip angle, the more HW overbreak can occur. Other geometric factors which influence the shape and depth of the relaxation zone within the surrounding rock mass include the stress shadows generated by undercut or similar geometry.

Remembering the example of the obstacle bypassed by the stress streamlines, the undercut is seen as a second obstacle that the stresses have to interact with. An undercut creates a shield for the first obstacle encountered by stress streamlines, creating zones of reduced stress which are referred to as stress shadows. In terms of geomechanics, undercuts introduce free faces that reduce the confinement and remove the HW abutments. Undercuts thereby increase the zone of stress relaxation (Figure 7).
The designing approaches which are employed to reduce HW sloughage use these geometric factors which have effects on the extent of relaxation zone and aim reducing their influence on the HW stability.

### 2.2.1.3 Rock mass characteristics

The complexity of rock mechanics comes from working with a medium with significant variability. Rock masses are comprised of intact rock and discontinuities, such as joints. The mechanical response of a rock mass to loading or unloading is not only dependent on the constitutive behaviour of the intact rock, but is also dependent on the interaction of the intact rock blocks with each other (i.e., the interaction between the discontinuities).

As previously discussed, the major failure mechanism that controls HW sloughage in underground open stopes is the gravity-driven structurally-controlled failure mode. This will
cause the rock material to fall into the stope by gravity due to the relaxation process. Therefore, the characterisation of the discontinuities is of more interest for the assessment of the HW stability than the characterisation of the intact rock at the exception of the intact rock unit weight.

When a fractured rock mass is subjected to a relaxed state stress, the rock blocks delineated by discontinuities and the opening walls can unravel. The unravelling process includes the opening of the discontinuities, followed by the free falling or sliding of rock blocks along discontinuities. While the opening of discontinuities is generally controlled by the cohesive characteristic of them, the sliding of rock blocks is mostly influenced by the roughness of the discontinuities. Roughness is often integrated into stability analysis by the friction angle of the discontinuities. Laboratory tests can be performed on discontinuities to determine these parameters. However, it is difficult to test samples of adequate and representative sizes which could serve to better predict the response of the rock mass due to stress state changes. Overall, the lower the value of the intrinsic parameters characterising the discontinuities, the greater the possibility that unravelling will occur.

2.3 Stability Assessment of Open Stope Hangingwall

The previous section showed that HW sloughage is typically the result of a relaxation process, which leads the rock blocks in the de-stressed zone to unravel. The stope HW failure mode is mainly gravity-driven and thereby structurally controlled. It was also discussed that the stress conditions, the stope geometry and the rock mass properties are the major factors contributing to the extent of the sloughage zone. This section aims to present different methods that can be used to assess HW stability in open stope underground mines. These methods can be categorized as follows: analytical approaches, empirical approaches, and numerical approaches.
2.3.1 Analytical solutions

Two common analytical solutions that are used to assess the HW stability are: the kinematic wedge failure analysis and the Voussoir beam analysis. This section presents each of them.

2.3.1.1 Kinematic wedge failure analysis

Rock wedges are polyhedra formed by the intersections of joints and the free face of the opening (i.e., stope). Wedges can present different geometries depending on the spatial distribution of the joints. The instability of a wedge will occur when the forces under which it is subjected pushes the wedge to fall into the opening. The instability is associated with the rotational or translational movement of the block.

The forces are categorized either as the driving forces or the resisting forces. The driving forces are those acting to move the wedge (i.e., gravity). The resisting forces are those acting against the block movement (i.e., friction). If no reinforcement tools are used, the resisting forces include the frictional forces generated by the weight of the wedge and the cohesive strength of the joints. The kinematic analysis consists in summing those forces to evaluate the stability of each individual wedge. A block will be considered stable if the resisting forces are greater than the driving forces.

In general, the approach, which originates from the key-block theory (Goodman and Shi, 1985), consists of identifying the key-blocks (i.e., the removable blocks) and then determining their individual stability using the limit equilibrium approach previously described. The main difficulty of this approach resides in the identification of the key-blocks. Effectively, such assessment requires an accurate knowledge of the joints in three-dimensions which is difficult and sometimes impossible to obtain.
Menéndez-Diaz et al. (2009) group the different methods which can be employed to identify the key-blocks into three categories as follows: the specific approaches, the ubiquitous approaches, and the probabilistic approaches. The specific approaches require the knowledge of the exact position in space of each discontinuity and of the opening, making these approaches obsolete for HW stability assessment.

The ubiquitous approaches consider the sets of discontinuities that can occur anywhere in space and on any surface of the opening. Therefore, all possible combination of joints is considered and the key-blocks are determined based on the largest possible blocks forming from these combinations. A common tool used in the industry which belongs to this category is the computer program Unwedge, developed by Rocscience. The program was specifically developed to identify the potential wedges in underground hard rock mines and is often used for deterministic analysis. It uses the average dips and dip directions of the major joint sets determined through the data collection campaign and calculate the locations and dimensions of the key-blocks. The joints are assumed to be fully persistent. The major shortcoming of this approach is that the structural complexity of the rock mass is not adequately captured.

Alternatively, the probabilistic approaches, overcome this shortcoming since they take into account the variability of the discontinuity properties such as orientation, persistence, spacing and dispersion, which are all recognized factors of underground opening stability (Tollenaar, 2008). Probabilistic approaches were shown to provide a more realistic estimate of the wedges that can form around an opening (Lee and Park, 2000; Grenon and Hadjigeorgiou, 2003; Liu et al., 2004).

Fracture system models, also referred to as Discrete Fracture Networks (DFNs), apply a probabilistic approach and is currently referred to as state-of-the-art. The concept requires different joint sets (including random joint sets) to be stochastically generated with respect to the
probability density function of the various properties (e.g., spacing). The resulting numerical rock mass defined by these generated joints is called a Discrete Fracture Network (DFN). For the same input data, different DFN models can be generated with an equal probability. This approach allows a probabilistic assessment of the different type of wedges which can form around the opening, providing frequency distribution of block sizes and their stability. There are few examples of the use of DFN models for an underground opening stability assessment in the scientific literature, which include studies by Stacey et al. (2005), Hadjigeorgiou and Grenon (2005) and more recently Elmouttie et al. (2010).

2.3.1.2 Voussoir beam analogy

The Voussoir beam analogue can be used for the assessment of the stability of an opening when the rock mass is laminated. Parallel lamination is a common geological structure observed in many underground mines. It can be the result of sedimentary layering, tensile jointing, metamorphism or even igneous flow processes. When an underground opening is excavated in a laminated geological environment, the lamination can become a dominant influencing factor with regards to the stability of the opening. When the lamination is parallel or nearly parallel to the stope HW, it can create beams of rock parallel to the stope HW (Figure 8). This rock beam, consisting of a single bed with nearly perpendicular joints, can be referred to as a Voussoir beam.
The first analytical solution developed for Voussoir beam and applied to rock mechanics was presented by Evans (1941). Since then, the Voussoir beam solution has improved over the years (Beer and Meek, 1982; Hutchinson and Diederichs, 1996; Sofianos, 1996; Diederichs and Kaiser, 1999). The version presented by Diederichs and Kaiser (1999) is considered in this analysis. Their version consists of an iterative process during which the initial load distribution and the line of action in the system are continuously changed. The main assumptions of the model are summarized below:

- As the beam deflects, a parabolic compression arch develops within the beam (Figure 8b).
- The Voussoir beam does not sustain tensile stresses.
- The beam is assumed to be initially stress-free.

Figure 8. Voussoir beam adjacent to stope opening: a) the jointed rock beam; b) the Voussoir beam analogy.
• Deflection of the beam occurs before slippage at the abutment.
• The abutment are stiff and do not deform under arching stress.
• The cross cutting structures (joints) are angled from the wall’s normal significantly less than the minimum angle of friction of the joints.

Three failure modes are analysed and include the following: crushing, snap-thru, and sliding of the unsupported beam under self-weight. It is important to note that the solution is highly sensitive to the rock mass elastic modulus ($E_{rm}$). The necessary input parameters are listed in Table 1:

Table 1. Parameters for Voussoir beam analysis.

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Parameters</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>$E_{rm}$</td>
<td>Rock mass modulus</td>
<td>GPa</td>
</tr>
<tr>
<td>UCS ($\sigma_c$)</td>
<td>uniaxial compressive strength of intact rock</td>
<td>MPa</td>
</tr>
<tr>
<td>S.G.</td>
<td>Specific gravity</td>
<td></td>
</tr>
<tr>
<td>$T$</td>
<td>Thickness of the Voussoir beam</td>
<td>MPa</td>
</tr>
<tr>
<td>$S$</td>
<td>Span of the studied face</td>
<td>MPa</td>
</tr>
<tr>
<td>$\alpha$</td>
<td>Dip angle of the surface</td>
<td>°</td>
</tr>
<tr>
<td>$\varphi$</td>
<td>Joint surface friction angle</td>
<td>°</td>
</tr>
</tbody>
</table>

The method was first developed for a laminated back, but was generalized for any stope face with an inclination angle ‘$\alpha$’ and hence, applied to stope HWs. The way of integrating the inclination within the solution consists in calculating an effective specific gravity:

$$S.G_{eff} = S.G. \times \cos (\alpha)$$ (2)

Examples of its application within design procedures for HW stability can be found in the literature (Daehnke et al., 2000; Ran, 2002; Capes, 2009). The main assumption allowing this method to be applied to inclined surface is that pressure does not build up parallel to the beam. Only the normal to the opening self-weight pressure component is considered.
Design charts were also developed which use the generalised form of the solution. A buckling limit (BL) of 35% is considered for the generalised form. Both a design chart for long stope surface and one for a square surface were proposed (Figure 9 and 10). Using the long stope surface design chart, where the width S (Figure 9) is significantly smaller than the length, provides a conservative solution. Using the square surface solution (Figure 10), where the width and length are equal to S, provides less conservative results as the four abutments are considered to contribute to the confinement and the stability of the Vousoir beam. It is recommended to use both design charts when analysing a rectangular face to bound the possible solution. Only the crushing and snap-thru failure modes are included in the design charts.

Figure 9. Vousoir beam design chart for long stope surface (after Diederichs and Kaiser, 1999).
2.3.2 Empirical methods

Empirical tools are commonly used in underground mine design, including the design of open stopes. The statistical analysis of underground observations has allowed the development of such empirical design tools which make it possible to capture the variability of the medium and conditions in which underground stopes are excavated. Nevertheless, it is important to keep in mind that these tools must be used in the appropriate conditions (i.e. the case being studied should present similar condition as the cases upon which the empirical model was developed).

This section presents two empirical tools that can be used to assess the HW stability: the Modified Stability Graph and the Equivalent Linear Overbreak/Sloughage (ELOS)/Dilution Graph.
2.3.2.1 The Modified Stability Graph

The Stability Graph was first developed by Mathews et al. in 1981 and it was introduced as an empirical qualitative approach to predict the stability of a deep (below 1000 m) underground open stope. The method was then modified by Potvin (1988), who introduced the Modified Stability Number (N') and renamed the empirical method to the Modified Stability Graph which is commonly used in the industry to assess HW stability. It is also presented as an accepted method for open stope design in many of the mining handbooks including *Rock Mechanics for underground mining* by Brady and Brown (2006) or more recently *Guide Pratique du Soutènement Minier* by Hadjigeorgiou and Charette (2009) and few examples of its application can be found in the literature (Henning and S. Mitri, 2008; Capes, 2009; Whipple et al., 2009).

The Modified Stability Graph as presented by Potvin (1988) is shown in Figure 11. One of the main characteristics of this approach consists in analysing each face of the stope individually. For each face, a stability analysis is performed based on two criteria: the N' and the hydraulic radius of the face.
Figure 11. Modified Stability Graph (after Potvin, 1988).

The $N'$ value represents the geotechnical stability of the studied face in regards of the ground condition. A large modified stability number corresponds to a good quality or favourable ground conditions, while a smaller modified stability number corresponds to a weak quality or unfavourable ground conditions (Figure 11). The $N'$ value is the result of the major factors influencing stope stability and it is computed using expression:

$$N' = Q' \times A \times B \times C$$  \hspace{1cm} (3)

Where $Q'$ is a modified version of the Tunnelling Quality Index ($Q$) (Barton et al., 1974), $A$ is the stress factor, $B$ is the rock defect factor and $C$ is the gravity factor.

$Q'$ differs from the typical $Q$ by equating the stress reduction factor (SRF) to one. The effect of stress is accounted for in the $N'$ by the factor $A$. The water reduction factor $J_w$, is often set to one and consequently $Q'$ is written as follow:

$$Q' = \frac{RQD}{J_u} \times \frac{J_w}{J_u}.$$  \hspace{1cm} (4)
Where RQD is a measure of the degree of rock mass fracturing (Deere and Deere, 1988) which is equal to the sum of the rock pieces whose length are greater than 10 cm over the length of the entire sample, $J_r$ is the joint roughness number, $J_a$ is the joint alteration number and $J_n$ is the joint set number as defined by Barton et al. (1974).

The stress factor $A$ is determined using the unconfined compressive strength (UCS) of the intact rock and the induced compressive stress ($\sigma_1$) acting parallel to the studied face, and it can be computed using the following relations (Table 2):

Table 2. Relations to determine Factor A.

<table>
<thead>
<tr>
<th>Condition</th>
<th>A</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS/ $\sigma_1 &lt; 2$</td>
<td>0.1</td>
</tr>
<tr>
<td>$2 &lt;$ UCS/ $\sigma_1 &lt; 10$</td>
<td>0.1125 (UCS/ $\sigma_1$)-0.125</td>
</tr>
<tr>
<td>UCS/ $\sigma_1 &gt; 10$</td>
<td>1.0</td>
</tr>
</tbody>
</table>

The effect of joint orientation on the stability of the studied stope face is presented by Factor B. This factor depends on the difference of orientation with respect to the studied face. The value of B can be determined using the following chart (Potvin, 1988).
The effect of the stope surface inclination on the stability is defined by the factor $C$. It aims to quantify the effect the gravity has on the stope stability. This factor is determined independently either for falls and slabbing failure mode or for sliding failure mode. Figure 13 presents the chart used to determine factor $C$ for sliding failure mode, and the following formula can be used to determine the $C$ value for gravity falls or slabbing.

$$C = 8 - (6 \times \cos \left( HW dipangle \right))$$  \hspace{1cm} (5)

For HW sloughage, it is more appropriate to use Equation 5 to determine the $C$ value than the chart (Clark, 1998).
Plotting the modified stability number \( (N') \) against the HR allows the user to assess the stability of the stope face using the graph in Figure 11, where the stability can be characterized as either stable or unstable. If the \( N' \) value for the studied face lies in the ‘Caved’ or ‘Transition’ zone shown by the blue zone in Figure 11, then the stope is deemed unstable and significant unplanned dilution can be expected. Inversely, if it lies within the ‘Stable’ zone, a low dilution can be expected. Thus, as mentioned earlier, the HW stability assessment can be performed using the Modified Stability Graph method, but the outcome is a qualitative index of the face stability.

**2.3.2.2 The ELOS/Dilution Graph**

The dilution graph was introduced by Clark (1998) in an attempt to provide an empirical quantitative design tool based on the desired level of dilution instead of being based on a qualitative assessment of stability (i.e., the Modified Stability Graph). One of the key
components of this method is the introduction of a new dilution parameter called equivalent linear overbreak/sloughage (ELOS).

The ELOS is an unplanned dilution parameter that aims to quantify overbreak independently of the width of the stope which permits a simple comparison with other mining operations which present different ore widths. The equivalent linear overbreak/sloughage represents a conversion of the true volumetric measurement of the sloughage or overbreak to an average depth of failure. A schematic defining the ELOS was presented by Clark (1998) and is shown in Figure 14.

Figure 14. ELOS definition (after Clark, 1998).

The equivalent linear overbreak/sloughage can be calculated using the following equation:

$$\text{ELOS} = \frac{\text{Volume of slough from stope surface}}{\text{Stope height} \times \text{Wall strike length}}$$

(6)

The original Dilution Graph was developed by plotting the location of 88 ELOS measurements on the Modified Stability Graph as presented by different authors (Potvin, 1988; Nickson, 1992; Scoble and Moss, 1994). With these values of ELOS plotted on the
Modified Stability Graph, and using statistical analysis as well as engineering judgement, ELOS lines were drawn. The first database was composed of 88 ELOS measurements which was increased by Capes (2009) to a total of 255 ELOS measurements, including both HW and FW overbreak. Thus, Capes (2009) revised the ELOS lines based on this comprehensive database and the resulting revised Dilution Graph is reproduced in Figure 15.

![Revised Dilution Graph](image)

Figure 15. Revised Dilution Graph (after Capes, 2009).

Considering sloughage as the result of HW instability, one can use the Dilution Graph and the ELOS lines to assess the stability of the HW by plotting the studied design HR as a function of N'. However, as noted by Clark (1998) and Capes (2009), it is important to consider the database used to develop this design method whenever it is being used for a stope stability assessment. Like any other empirical tool, this method is only applicable for the design of stopes similar to those which comprise the database.

Nonetheless, the ELOS lines can be adjusted to a particular mine site using on-site ELOS measurements and a rigorous methodology. If this is done, it can become a useful tool for stope stability assessment as noted by Whipple et al. (2009).
2.3.3 Numerical methods

Numerical methods are commonly used in rock mechanics. Their efficiency in terms of computation makes them very useful when it comes to solving complex problems such as assessing open stope stability. Different numerical methods have been developed throughout the years including Boundary Element Method (BEM), Finite Element Method (FEM) and Distinct Element Method (DEM). Jing (2003) presented an interesting review of these different numerical methods and their application to rock mechanics problems.

Many application of using numerical methods can be found in the scientific literature. For example, Harris and Li (1995) used the BEM approach to model the stresses around a stope and consequently the stability of the stope by assessing the major principal stress and the maximum extension strain obtained. Milne et al. (2004) also used BEM to study the deformations occurring at the surface of a stope HW. More recently, Wang et al. (2007) used BEM to study the influence of stope geometry parameters on the extent of the relaxation zone to assess the HW sloughage. Additionally, assuming that the relaxation zone defines the overbreak zone to be expected, Hughes et al. (2010) used FEM to study unplanned dilution of stopes with different strike length.

The DEM is the most appropriate method to properly incorporate the effect of the discontinuities in the sloughing process as, unlike the other methods, it assesses the rock mass as an assembly of blocks interacting with each other through deformable joints. Jing and Stephansson (2007) demonstrated the ability of DEM for solving complex rock mechanics problems. Examples specific to HW stability assessment using DEM include Board et al. (2000) or Andrieux et al. (2009).
2.4 Techniques to Control Hangingwall Sloughage in Underground Open Stope Mines

In order to reduce the amount of unplanned dilution due to HW sloughage in open stope mining, different strategies can be implemented. These strategies can be categorized as either using reinforcing techniques or using designing and planning approaches.

The first category includes the strategies that apply techniques and reinforcement tools which improve the properties of the rock mass around the stope, enhancing the self-supporting capacity of the rock mass. The second category, the designing and planning approaches, refers to the strategies that requires the sizing of the stopes and the sequencing of their extraction.

The choice of one strategy over another is generally driven by the feasibility of the approach. There is an important incentive in the mining industry to increase the production at a lower cost. Thus, the implementation of one of these strategies must be studied in terms of profit; the cost related to their implementation must be weighted against the cost related to their effectiveness in terms of HW sloughage control or dilution rates. For this reason, it is important to perform a cost-benefit assessment of implementing alternative approaches.

2.4.1 Reinforcement techniques

The application of reinforcement tools serves to improve the properties of the rock mass around the open stope, making it self-supporting. The common reinforcement tools that are used to control unravelling of large-span excavations (i.e., open stope hangingwalls) are cable bolts which provide localized reinforcement between unravelling blocks.

The cable bolts are generally grouted in the boreholes and can be tensioned, but are generally left untensioned as they are typically installed prior to the opening of the stope or sequentially during the stoping operation. As the cable bolt becomes stressed, it tends to pull out
the grout; the cable wires try to get off their grout imprint, causing radial displacement. Consequently a confining stress response is induced and a resisting shear stress is then created to keep the cable from sliding, increasing the confining stresses in the rock mass (Hoek et al., 1993), thereby limiting joint dilation.

Different bolting patterns can be designed for underground open stopes. The most popular include the drift fan, the even anchor support pattern, and the point anchor support pattern (Fuller, 1983) (Figure 16).

Figure 16. Typical HW cable support pattern (after Fuller, 1983).

The design pattern to be chosen depends on several factors such as the rock mass characteristic. Nickson (1992) noted that several design approaches can be used to determine the bolting pattern including: the dead weight design method, the rock mass classification method, the beam theory method, the past experience method, the Mathews method and bolt factor method, the rock mass stiffness based method and the Potvin method.
In any case, all of these methods agree on the fact that the more jointed the stope HW is and the poorer the quality of the rock material which constitutes it, the greater the required support will be (Figure 17). Potvin and Hadjigeorgiou (2001) developed the cable bolt density curves reproduced in Figure 17 which shows that the smaller the relative block size ratio (i.e. RQD/Jn) divided by the HR is, the denser the cable bolt pattern should be to provide sufficient support and increase stability. Thus, as more cable bolts are installed, the greater the support that will be provided.

![Cable bolts density design chart](after Potvin and Hadjigeorgiou, 2001)

In addition, referring to Figure 17, it is shown that for a small block size (<0.6-0.7), the data set analysis showed that no bolt density was sufficient to provide the necessary stability, making the reinforcing strategy trivial. Thus, based on previous mining experience, if a very weak immediate HW is in place, cable bolting may not prove to be an adequate strategy to adopt.

In cases where reinforcement is used, the cost related to cable bolting includes the cost of the reinforcing tools themselves (including installation equipment), the opening of access drifts
(if required) and the indirect cost related to the time delay to allow mining and bolting should be estimated. These costs can become very significant and must be weighed against the cost related to the savings made in terms of dilution cost.

2.4.2 Designing and planning approaches

The second category of strategies to control hangingwall sloughage in open stope underground mines, refers to the strategies that involves the sizing of the stopes and the sequencing of their extraction. As discussed in section 2.2.1, the size of the stope and the pre-mining stresses have a significant influence on the relaxation zone that form around the stope HW and thereby on the expected volume of sloughage and rate of dilution. The mine sequencing is a regular approach used in mining as it aims to control the stress redistribution in a fashion which helps improve stability. While sequencing is generally adopted regardless of the size of the stope, this section focuses more on the stope sizing. This includes altering the initial stope design to reduce the influence that the stope size has on the HW stability.

One of the most popular tools used in Canadian underground mines when adopting the sizing approach is the empirical modified stability graph method introduced by Potvin (1988) or an alternative version of it (Nickson, 1992; Hadjigeorgiou et al., 1995). In all cases, the weaker the rock mass quality in the HW, the smaller the ideal HW span in order to ensure the HW stability (where N’ is the indicator of the rock mass quality and HR is the indicator of the span size). As the rock mass quality is determined by nature, the controllable factors include the stope height and the stope strike length. By reducing one or the other, thereby reducing the HR, the stability of the designed stope face will be enhanced.

For example, a case study by Hughes et al. (2010) on Lapa mine in north-western Quebec (Canada) showed that by reducing the stope strike length by 4 m, a reduction of 37% of dilution
volume coming from the HW could be achieved. These values were determined using numerical analysis, but the case was also examined using Nickson’s (1992) modified stability graph and similar results were obtained. In summary, for a $N'$ value of 2.6, a stope height of 30 m and by reducing the stope strike length from 12 m to 8 m (i.e., reducing the HR from 4.3 m to 3.2 m), the design was reclassified from the ‘Transition Zone’ to the ‘Stable Zone’.

![Graph](image)

Figure 18. Potvin’s method using Nickson’s modified stability graph (1992) for HW stability assessment of resized stope (after Hugues et al., 2010).

However, reducing the HR of the stope faces to better control the HW sloughage and the rate of dilution results in a smaller stope size for an underground mine. This will increase the cost of mine developments for a project, as more development will be required for the extraction of the same delineated ore-body. These costs are not negligible in terms of the overall economic value of a mining project.
Considering the Lapa mine case and an average typical cost of development of $5000 /m. The cost of development of the unstable stope that has a 30 m height and 12 m strike length is compared to the stable stope that has a 30 m height and 8 m strike length. Transversal mining is assumed as well as top sills and bottom sills of 15 m long. For simplification, the cost is studied over 24 m strike length of mining.

Table 3. Comparative analysis of stope size on the cost of development.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Values</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stop height</td>
<td>m</td>
<td>30</td>
</tr>
<tr>
<td>Stop height</td>
<td>m</td>
<td>30</td>
</tr>
<tr>
<td>Stope strike length</td>
<td>m</td>
<td>12</td>
</tr>
<tr>
<td>Stope strike length</td>
<td>m</td>
<td>12</td>
</tr>
<tr>
<td>HR (HW)</td>
<td>m</td>
<td>4.3</td>
</tr>
<tr>
<td>Total strike length</td>
<td>m</td>
<td>24</td>
</tr>
<tr>
<td>Total strike length</td>
<td>m</td>
<td>24</td>
</tr>
<tr>
<td>Number of stope</td>
<td></td>
<td>2</td>
</tr>
<tr>
<td>Number of development</td>
<td></td>
<td>4</td>
</tr>
<tr>
<td>Cost of development</td>
<td>($/m)</td>
<td>5000</td>
</tr>
<tr>
<td>Cost of development</td>
<td>($/m)</td>
<td>5000</td>
</tr>
<tr>
<td>Total length of development</td>
<td>m</td>
<td>60</td>
</tr>
<tr>
<td>Total cost of development</td>
<td>$</td>
<td>300000</td>
</tr>
<tr>
<td>Total cost of development</td>
<td>$</td>
<td>450000</td>
</tr>
</tbody>
</table>

Table 3 shows that the initial cost of development for the larger stope is of $300,000 while it increases to $450,000 for the resized design. Thus, over a distance of 24 m along the ore-body, the cost difference of designing a smaller stope is $150,000. These numbers clearly show that there is an incentive for increased stope size. It is also important to note that these numbers do not account for the indirect costs of resizing for smaller stopes include; such as the time required for pre-production development or the drilling-blasting design change.

An alternative design approach which allows the achievement of relatively large stopes when facing the challenge of a weak HW consist of adopting the ore-skin design. The ore-skin design consists of leaving a certain thickness of ore unmined along the HW to enhance the structural integrity of the stope, preventing HW sloughage. This approach, illustrated in Figure 19, avoids the costs related to designing for smaller stopes described previously.
The ore-skin design has already been implemented in different mining operations where the HW consisted of a weak rock and the ore-body of a competent rock. The method has shown to be a successful approach in achieving reduced dilution rate in open stope underground mines (Harris and Li, 1995; Mubita, 2005; Capes, 2009). However, leaving a certain thickness of ore unmined results in reduced ore recovery. In this thesis, the ore-skin design approach is further investigated as an alternative design approach to control ore dilution in open stope underground mines.

Figure 19. Conceptual model of the ore-skin design method: a) open stope without ore-skin; b) open stope with ore-skin.

2.5 Summary

This chapter summarized the main topics related to the understanding of HW sloughage as well as its importance in terms of a mine economics. Effectively, it was discussed that HW sloughage is the main source of ore dilution in open stope mines and that dilution can greatly affect the economic viability of a mine and even cause the closure of a mine.
Therefore, there is a need to implement an optimal strategy for HW sloughage control. In countries like Canada where the horizontal in situ stresses are higher than the vertical in situ stresses, the main mode of failure causing HW sloughage in open stopes is structurally-controlled gravity-driven and is caused by the relaxation process. The main factors influencing the volume of sloughage to be expected include the stope geometry parameters (e.g., the stope hydraulic radius), the stress regime, and the rock mass quality, which is function of the spatial distribution of the joints and their mechanical properties.

Accordingly, the analytical, empirical and numerical approaches that can be used to assess the HW stability were also presented. The choice of a possible solution depends on the nature of the problem and its scale.

The different methods which can be used to control HW sloughage were categorized as either using reinforcement techniques or designing and planning approaches. In presence of a weak immediate HW material, experience shows that the use of reinforcement such as cable bolts will not provide the desired stability. Under the same conditions, it was shown that reducing the HR of the stope can significantly increase the overall costs of extraction due to the increase of required development. Finally, the ore-skin design was introduced as an alternative design method which can achieve the extraction of large block of ore, while maintaining the structural integrity of the stope.
CHAPTER 3
ECONOMIC ORE-SKIN STOPE DESIGN

Like most resource-based industries, there is a great effort in mining put towards the optimization of the production at a low cost. In other words, one of the main objectives in mining is to develop innovative approaches that allow more efficient extraction of valuable materials at a lower cost. As discussed earlier, ore dilution in underground open stope mines can greatly affect the cost of mining. Experience from underground mine operations around the world have shown that ore-skin design can be used as an effective dilution control strategy for the particular geological settings where the immediate HW consists of a weak material and the ore-body of a competent material. As for other designing approaches, it is necessary to conduct a cost-benefit assessment in order to evaluate the economic viability of implementing the ore-skin design.

This chapter presents a methodology that addresses the economic viability of the ore-skin design technique. The methodology consists of determining the maximum (break-even) ore-skin thickness that can be left in place by comparing the economic benefit of this design method to a stope design which uses no other HW sloughage control technique.

3.1 Break-Even Ore-Skin Thickness

Different economic parameters are considered to investigate the maximum ore-skin thickness that can be left unmined while keeping the stope economically viable. These parameters influence the overall costs and revenues of a stope extraction and include:

- the in situ grade of the planned stope and the unplanned overbreak,
- the price of mineral commodity,
- the specific gravity or density of the ore and the host rock (i.e., the weak HW and the Host rock),
• the mining operating cost, and
• the mill recovery.

Other parameters that are included in this assessment relate to the geometry of the stope and are as follows:

• stope width (SW),
• stope height (SH),
• stope strike length (SL),
• weak HW thickness (WHWT),
• HW dip angle, and
• ore-skin thickness (OST) (Figure 20).

Figure 20. Stope geometry and governing parameters.
3.1.1 The assumptions

The following assumptions were made for this study:

- The volume of the material comprising the stope is assumed to have an average in situ grade value that is constant and homogeneous. These grades, considering the volume contribution of each rock type respectively, represent the potential profit in terms of metal tonnage content of the designed stope. The in situ grade of both the planned stope and overbreak material can be obtained from the simulated ore body models or from sampling and assaying of drills.

- An average operating cost is assumed. The total operating cost includes expenses related to the development of underground opening, exploitation and processing of a stope, including drilling, blasting, excavation, hauling, crushing, hoisting, milling and treatment. Operating cost is expressed in terms of cost per tonnage mined. The indirect costs of dilution are difficult to obtain and were ignored.

- The mill recovery could either be considered as fixed or it can vary as a function of ore head grade. Experimental studies indicate that a higher ore head grade will increase ore recovery during processing. In the current study, a fixed mill recovery is assumed.

Using these parameters, the economic profitability of a stope designed with the ore-skin approach can be compared to a typical open stope design (i.e., without ore-skin) of the same stope size. This comparison allows one to determine the economic viability of the proposed design approach.

3.1.2 Determination of the input parameters

The profit equation of a single open stope can be determined using the following procedures. First, the stope volume (SV) (Equation 7) and stope tonnage (ST) (Equation 8) are calculated as follows:

\[
SV = (SW - OST) \times SH \times SL
\]  

(7)
Where $SW$ is the stope width (m), $SH$ is the stope height (m), $SL$ is the stope strike length (m), $SG_1$ is the density of material within the planned stope (tonne/m$^3$), and $OST$ is the ore-skin thickness (m).

The profitability of a stope is a function of several parameters, but with the exception of hangingwall overbreak volume ($HW_{OV}$), the other parameters are not dependant on the use of the ore-skin technique. Estimating $HW_{OV}$ is difficult due to the complex shape of the HW overbreak. This volume is generally expressed as the ELOS, as described earlier in section 2.3.2.2, which simplifies the geometry of the overbreak zone (Clark and Pakalnis, 1997; Clark, 1998), but field observations indicate that the hangingwall overbreak has an arched shape (Ran, 2002; Henning, 2007; Capes, 2009). In this methodology, it is assumed that the sloughage extends into the weak immediate HW until it reaches the higher quality rock mass and that it forms a trapezoidal shape (Figure 21). This is in accordance with the observations made by different researchers and reported in the literature (Potvin, 1988; Capes et al., 2006a; Capes, 2009). Using this assumption, the volume of HW overbreak ($HW_{OV}$) can be determined using Equations 9 and 10 as follows:

$$HW_{OV} = AOV \times SL$$  \hspace{1cm} (9)

$$AOV = SH \times MOT - \frac{MOT^2}{\tan (AHWO)}$$  \hspace{1cm} (10)

Where $AOV$ is the average area of overbreak along the HW face (m$^2$), $MOT$ is the weak HW width (i.e., maximum overbreak thickness) (m), and $AHWO$ is the angle of HW overbreak in regards of the stope HW (°).
Figure 21. Trapezoidal shaped sloughage assumed for the volume of HW sloughage (i.e., AOV).

The total overbreak volume considered should also include sloughage material from the footwall and the back of the stope and the volumes of overbreak from these sections should also be calculated. It is considered that the volume of overbreak coming from the footwall \((FOV)\) and coming from the back \((BOV)\) can be estimated as a lower percentage of the total volume of overbreak coming from the HW (Equation 11 and 12).

\[
FOV = FBWOV_{-1} \times HWOV
\]  
\[
BOV = FBWOV_{-2} \times HWOV
\]

Where \(FBWOV_{-1 or -2}\) is the percentage of the hangingwall overbreak volume considered for the footwall and back respectively.
Thus, the total volume of overbreak \((TOV)\) to be considered is the sum of \(FOV\), \(BOV\) and \(HWOV\) (Equation 13).

\[
TOV = HWOV + FOV + BOV
\]  
(13)

To determine the profitability of the stope, the economic and geological parameters listed in Table 4 must be incorporated.

Table 4. Economic and geological parameters.

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Parameters</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>M</td>
<td>Mill recovery</td>
<td>%</td>
</tr>
<tr>
<td>P</td>
<td>Price of metal</td>
<td>$/g</td>
</tr>
<tr>
<td>X</td>
<td>Operating cost</td>
<td>$/tonne mined</td>
</tr>
<tr>
<td>SG1</td>
<td>Density of planned stope material</td>
<td>tonnes/m(^3)</td>
</tr>
<tr>
<td>SG2</td>
<td>Density of unplanned overbreak material</td>
<td>tonnes/m(^3)</td>
</tr>
<tr>
<td>G1</td>
<td>Average in situ grade of planned stope</td>
<td>g/tonne</td>
</tr>
<tr>
<td>G2</td>
<td>Average in situ grade of unplanned overbreak</td>
<td>g/tonne</td>
</tr>
</tbody>
</table>

Using the parameters in Table 4, the total tonnage of overbreak can be determined as follows:

\[
TOT = (HWOV + FOV) \times SG_2 + BOV \times SG_1,
\]  
(14)

where the metal content within the stope is:

\[
TMS = ST \times G_1,
\]  
(15)

and the metal content coming from the overbreak is

\[
TMO = ((HWOV + FOV) \times SG_2 \times G_2) + (BOV \times SG_1 \times G_1),
\]  
(16)

then the total metal content extracted is

\[
TMC = TMS + TMO,
\]  
(17)
and the total tonnage mined is:

\[ TTM = ST + TOT \quad (18) \]

The stope head grade (unit metal/tonne mined) can then be determined by dividing Equation 17 by Equation 18 as follows:

\[ SHG = \frac{TMC}{TTM} \quad (19) \]

The revenue \((R)\) is function of the total metal content extracted, the market price of the metal and the mill recovery (Equation 20). The price of mineral commodity is an important parameter and it generally fluctuates over time. Due to its direct influence on the profitability of exploitation and its versatility, particular attention must be given to it when designing a stope.

\[ R = TMC \times M \times P \quad (20) \]

The cost \((C)\) is related to the exploitation of the stope (Equation 21) and it is the product of the operating cost and the total tonnage mined.

\[ C = TTM \times X \quad (21) \]

The profit \((P)\) is defined by the subtraction of the cost from the revenue generated by the stope (Equation 22).

\[ P = R - C \quad (22) \]

The difference between the application of the ore-skin technique and a conventional open stope design lies in the stope head grade \((SHG)\) value. By leaving an ore-skin, the mining recovery will decrease and thus the total metal content \((TMC)\) will be reduced. At the same time,
due to the ore-skin along the HW, dilution will be controlled, and therefore the total tonnage mined \((TTM)\) will be reduced. In terms of profit, the ore-skin will reduce the total metal content \((TMC)\) and total tonnage mined \((TTM)\), and thus will reduce both the revenue and cost.

### 3.1.3 Formulation of the break-even ore-skin thickness

The proposed method aims to determine the maximum ore-skin thickness that can be left in place unmined to control the HW sloughage while being more economic (i.e., the generated profit is increased) in comparison to the conventional stope design.

The maximum ore-skin thickness can be determined by isolating the ore-skin thickness \((OST)\) in a profit equality equation. The design that does not include the ore-skin will be referred to as Scenario A while the design that includes the ore-skin will be referred to as Scenario B. Equation 23 is the equality equation on which this methodology is based.

\[
\text{Profit from scenario A} = \text{Profit from scenario B} \tag{23}
\]

Solving this equality results in finding the OST value for which the profits generated by the use or the non-use of the ore-skin technique are equal. This will determine the break-even ore-skin thickness. If geomechanical studies prove that an OST less than the break-even point is able to retain the stability of the HW, then it is more profitable to design the stope with an ore-skin.

Assuming that the volume of sloughage from the footwall and the back of the stope are equal for both design scenarios:

\[
\text{FOV}_A = \text{FOV}_B \quad \text{and} \quad \text{BOV}_A = \text{BOV}_B \tag{24}
\]

then the maximum OST that is also economically viable can be determined as follows:
where HWOV\textsubscript{B} is equal to zero as it is assumed that the ore-skin design will not permit any overbreak.

The advantage of this method is that it makes it possible for mine design engineers to achieve a better understanding of the suitability of the ore-skin design approach for underground open stope design. It should be noted that the estimation of the maximum ore-skin thickness (OST) is only based on the economic and geometric design parameters that can be easily estimated and measured. For instance, one could decide to use the mine recorded ELOS values multiplied by the strike length to estimate the HWOV\textsubscript{A}. Thus, Equation 25 provides a quick insight into the economic viability of the ore-skin approach. The calculated break-even ore-skin thickness should be further investigated to determine whether the beam thickness can geomechanically retain the stability of the overlying weak HW. Methods to determine the latter are presented in chapter 4.

### 3.2 Example of Application

In order to present how the proposed method can be applied to an open stope mine design, a conceptual underground gold mine was considered. For this example, typical values for gold mines in Canada are used. Table 5 summarizes the input parameters used for this conceptual example. It is assumed that the ore-skin design will not permit any HW sloughage \((HWOV\textsubscript{B} = 0 \text{ m}^3)\).
Table 5. Parameters used for the ore-skin design for a hypothetical open stope gold mine.

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Parameters</th>
<th>Unit</th>
<th>Values</th>
</tr>
</thead>
<tbody>
<tr>
<td>FBWOV -1 and 2</td>
<td>Percentage of the hangingwall overbreak volume considered for the footwall and back respectively</td>
<td>%</td>
<td>5</td>
</tr>
<tr>
<td>SG1</td>
<td>Density of planned stope material</td>
<td>tonne/m³</td>
<td>2.9</td>
</tr>
<tr>
<td>SG2</td>
<td>Density of unplanned overbreak material</td>
<td>tonne/m³</td>
<td>2.7</td>
</tr>
<tr>
<td>G1</td>
<td>Avg. In Situ grade of gold in planned stope</td>
<td>g/tonne</td>
<td>3</td>
</tr>
<tr>
<td>G2</td>
<td>Avg. In Situ grade of gold from the unplanned overbreak</td>
<td>g/tonne</td>
<td>0.2</td>
</tr>
<tr>
<td>MOT</td>
<td>Weak hangingwall width (maximum overbreak thickness)</td>
<td>m</td>
<td>4</td>
</tr>
<tr>
<td>FBWOV</td>
<td>Total footwall &amp; back-wall overbreak volume (% of the hangingwall overbreak volume)</td>
<td>%</td>
<td>0.05</td>
</tr>
<tr>
<td>AHWO</td>
<td>Angle of hangingwall overbreak</td>
<td>degrees</td>
<td>45</td>
</tr>
<tr>
<td>M</td>
<td>Mill recovery</td>
<td>%</td>
<td>85</td>
</tr>
<tr>
<td>P</td>
<td>Price of metal (gold)</td>
<td>$/g</td>
<td>42*</td>
</tr>
<tr>
<td>X</td>
<td>Operating cost</td>
<td>$/tonne mined</td>
<td>50</td>
</tr>
</tbody>
</table>

* which is equal to 1260 $/t.oz

Typical values for transverse stope dimensions in Canadian underground mines are used.

Figure 22 defines the stope dimensions as 20 m wide, 30 m high and 15 m long along strike. The hydraulic radius of the HW is thereby of 5 m and the HW is dipping 70° to West.

![Figure 22. Typical dimensions of a transverse stope.](image-url)
Table 6 compares the results obtained from applying the proposed method for the ore-skin design approach (Scenario B) with the typical stope design (Scenario A) for the transverse stope. The results indicate that the break-even ore-skin thickness in this case is of 2.4 m. Leaving this thickness of ore-skin to retain the stability of the immediate hangingwall and consequently controlling the ore dilution will provide the same economic profit as using a typical stope design.

Table 6. Comparison of the ore-skin design approach with a typical stope design.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Typical Design (Scenario A)</th>
<th>Ore-skin Design (Scenario B)</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stope Width</td>
<td>20</td>
<td>17.6</td>
<td>m</td>
</tr>
<tr>
<td>OST</td>
<td>0</td>
<td>2.4</td>
<td>m</td>
</tr>
<tr>
<td>Stope Volume</td>
<td>9,000</td>
<td>7,910</td>
<td>m³</td>
</tr>
<tr>
<td>Stope Tonnage</td>
<td>26,100</td>
<td>22,938</td>
<td>tonnes</td>
</tr>
<tr>
<td>Hangingwall Overbreak Volume</td>
<td>1560</td>
<td>0</td>
<td>m³</td>
</tr>
<tr>
<td>Footwall Overbreak Volume</td>
<td>78</td>
<td>78</td>
<td>m³</td>
</tr>
<tr>
<td>Back Overbreak Volume</td>
<td>78</td>
<td>78</td>
<td>m³</td>
</tr>
<tr>
<td>Total Overbreak Volume</td>
<td>1,716</td>
<td>156</td>
<td>m³</td>
</tr>
<tr>
<td>Total Overbreak Tonnage</td>
<td>4,649</td>
<td>437</td>
<td>tonnes</td>
</tr>
<tr>
<td>Tonnage of Metal In Stope</td>
<td>78,300</td>
<td>68,815</td>
<td>g</td>
</tr>
<tr>
<td>Tonnage of Metal In Overbreak</td>
<td>1,563</td>
<td>721</td>
<td>g</td>
</tr>
<tr>
<td>Total Metal Content</td>
<td>79,863</td>
<td>69,536</td>
<td>g</td>
</tr>
<tr>
<td>Total Tonnage Mined (Stope + Overbreak)</td>
<td>30,749</td>
<td>23,375</td>
<td>tonnes</td>
</tr>
<tr>
<td>Stope Head Grade</td>
<td>2.6</td>
<td>2.97</td>
<td>g/tonne</td>
</tr>
<tr>
<td>Revenue</td>
<td>2.85E+06</td>
<td>2.48E+06</td>
<td>$</td>
</tr>
<tr>
<td>Cost</td>
<td>1537440</td>
<td>1168760</td>
<td>$</td>
</tr>
<tr>
<td>Profit</td>
<td>1.31E+06</td>
<td>1.31E+06</td>
<td>$</td>
</tr>
</tbody>
</table>

In the next step, a geotechnical analysis should be undertaken to determine whether or not this ore-skin thickness or a smaller thickness would be sufficient to ensure the stability of the stope HW. If the geotechnical analysis indicates that a narrower ore-skin can maintain the stability of the HW, then the ore-skin design approach is economically favourable compared to the typical stope design approach.
3.2.1 Sensitivity analysis

In the previous example, all of the input parameters were considered constant. However, some of these parameters, such as metal prices, the in situ ore grade, the mine operating cost, etc., can vary over time. This section provides a brief overview of the influence of the variation of the price of metal, operating costs and average in situ ore grade on the break-even OST.

Figure 23 shows that an increase in the price of metal results in a decrease of the maximum ore-skin thickness which is economically viable. This means that as the metal price increases, aiming for a higher mining recovery rate may be economically more viable than attempting to obtain a reduced ore dilution. The same can be observed for the influence of operating cost. As the operating cost decreases, the economically viable ore-skin thickness reduces. In other words, the lower the mining cost related to the treatment of waste material, the lower the impact that dilution has on the economics of the stope.

![Figure 23. The variation of the break-even ore-skin thickness based on the gold price and the mining cost.]
The effect of in situ ore grade variation for both the planned and unplanned material on the break-even ore-skin thickness was also investigated. For this analysis, all of the other parameters were held constant (Figure 24).

Figure 24. The variation of the break-even ore-skin thickness based on the average in situ grade of the planned stope and overbreak.

Figure 24 demonstrates that as the in situ grade of overbreak material decreases, the break-even OST increases. The same trend can be observed for the average in situ grade of the planned stope; however, the relationship is not linear. For instance, as $G_1$ decreases beyond the average grade of 3 g/tonne (ppm) and lower, the break-even OST tends to increase considerably.

The results of sensitivity analysis indicate that the input values of particular parameters may significantly change the break-even ore-skin thickness. The design engineer should be aware of the parameter variability when implementing this method.
3.3 Summary

A simple and easy-to-use method was presented to estimate the maximum economic ore-skin thickness that can remain unmined in open stope design. Only the geometric and economic parameters are required in order to perform this analysis. The proposed approach of determining the maximum ore-skin thickness must be complemented by a geomechanical analysis before it is implemented in the underground mining operation.

The advantage to this approach is that it can be used as a quick and simple decision-making tool. With the sensitivity analysis, it was also shown that the break-even OST can be significantly influenced by the variable parameters such as commodity price and mine operating cost. In every case, it is the responsibility of the mine design engineer to select the most appropriate value for the variable parameters of a project.

The next chapter presents different tools that can be used to evaluate the geomechanical stability of an ore-skin.
CHAPTER 4
THE GEOMECHANICALLY STABLE ORE-SKIN THICKNESS

In the previous chapter, it was shown that for an ore-skin thickness smaller than break-even thickness, the ore-skin design is more profitable. This profit margin increases as the difference between the geomechanically stable ore-skin thickness, which prohibits HW sloughage, and the break-even OST increases. For optimization of the stope design, the minimal ore-skin thickness that can be left in place while maintain the structural integrity of the stope must be determined. This is purely a geomechanical problem and as such, it can be treated independently of the economics of the project. This chapter outlines the different approaches that one could use to assess the stability of the ore-skin thickness.

In this chapter, the different approaches which can be used to determine the stability of the minimal OST are presented. This includes analytical, empirical, and numerical methods. To clearly illustrate how these different approaches can be used, the same hypothetical gold mine presented in section 3.2 is studied. Furthermore, the geomechanical characteristics which were missing from section 3.2 are introduced.

4.1 Geomechanical Properties

It is believed that the ore-skin design would be an optimal design for open stoping exploitation when the immediate HW consists of a weak quality material and the ore-body consists of a good quality rock mass. Thus three main rock units defining the geotechnical model are assumed as: the Ore-body, the Host rock, and the weak HW. The properties of the geological
units are based on some real case studies. Table 7 summarizes the intact rock properties for each rock unit.

Table 7. Intact rock properties.

<table>
<thead>
<tr>
<th>Zone</th>
<th>UCS (MPa)</th>
<th>Poisson’s Ratio</th>
<th>Density (tonne/m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Host rock</td>
<td>160</td>
<td>0.2</td>
<td>2.7</td>
</tr>
<tr>
<td>Ore-body</td>
<td>160</td>
<td>0.2</td>
<td>2.9</td>
</tr>
<tr>
<td>Weak HW</td>
<td>60</td>
<td>0.3</td>
<td>2.7</td>
</tr>
</tbody>
</table>

For the sake of simplicity, the rock mass properties of the Host rock and the Ore-body are assumed to be similar (Table 8). The weak HW consists of a weak rock mass quality \( (0.5 < Q' < 2) \) while the Host-rock and Ore-body units consist of a good rock mass quality \( (11 < Q' < 33) \).

Table 8. Rock mass properties.

<table>
<thead>
<tr>
<th>Zone</th>
<th>Erm (GPa)</th>
<th>RQD (%)</th>
<th>( Q' )</th>
<th>Rock Mass Quality</th>
</tr>
</thead>
<tbody>
<tr>
<td>Host rock/Ore-body</td>
<td>40</td>
<td>90</td>
<td>11-33</td>
<td>Good</td>
</tr>
<tr>
<td>Weak HW</td>
<td>5</td>
<td>40</td>
<td>0.5-2</td>
<td>Poor</td>
</tr>
</tbody>
</table>

A total of three joint sets are considered for both the Ore-body and Host rock as well as the weak HW. The three joint sets are characterized as a schistosity set (S1) sub parallel to the stope HW, and two inclined joint sets (S2 and S3) (Table 9).
Table 9. Geometry of discontinuity sets.

<table>
<thead>
<tr>
<th>Joint Set</th>
<th>Description</th>
<th>Dip (◦)</th>
<th>Dip Dir. (◦)</th>
<th>Normal joint spacing</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Weak HW (m)</td>
</tr>
<tr>
<td>S1</td>
<td>Schistosity</td>
<td>80</td>
<td>270</td>
<td>0.4</td>
</tr>
<tr>
<td>S2</td>
<td>Major Joint Set</td>
<td>45</td>
<td>30</td>
<td>0.3</td>
</tr>
<tr>
<td>S3</td>
<td>Major Joint Set</td>
<td>45</td>
<td>150</td>
<td>0.3</td>
</tr>
</tbody>
</table>

The same joint surface characteristics were assumed for all the joint sets, but differ depending on the rock unit they are found in (Table 10). A lower friction angle was assumed for the joints formed in the weak HW zone compared to the ones formed in the Host rock/Ore-body zone.

Table 10. Joints surface characteristics.

<table>
<thead>
<tr>
<th>Rock unit</th>
<th>Kn (GPa/m)</th>
<th>Ks (GPa/m)</th>
<th>Tensile strength (MPa)</th>
<th>Cohesion (MPa)</th>
<th>Friction angle (◦)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Host rock/Ore-body</td>
<td>500</td>
<td>100</td>
<td>0</td>
<td>0</td>
<td>30</td>
</tr>
<tr>
<td>Weak HW</td>
<td>500</td>
<td>100</td>
<td>0</td>
<td>0</td>
<td>25</td>
</tr>
</tbody>
</table>

The weak HW is defined as having a smaller fracture spacing than the Host rock and Ore-body, thus it has a higher fracture intensity. The quality of a rock mass depends upon the rock properties and the fracture intensity, thus the information in Table 7, 8, 9 and 10 assumes that the immediate weak HW zone is a weaker rock unit than the Host-rock and Ore-body.

In the next sections these geomechanical parameters together with the stope geometric parameters (Figure 22) will be used to assess the minimum ore-skin thickness that should be left unmined to ensure the structural integrity of the stope.
4.2 Analytical Solution

4.2.1 Voussoir beam analogy

Considering the presence of a schistosity set (S1) oriented semi-parallel to the stope, it is possible to apply the Voussoir beam analogy. Therefore, the low dip angle of 10° for S1 with respect to the HW is neglected, and the schistosity is assumed to be parallel to the HW. Thus, leaving a particular thickness of ore-skin is similar to leaving a Voussoir beam of ore as the immediate HW. Based on Table 9, the width of this beam in average is equal to 0.9 m which is the spacing of the S1 joint set.

The stability of the proposed ore-skin was studied for the snap-thru and the crushing failure modes using the Voussoir beam design charts presented by Diederichs and Kaiser (1999) and introduced in Section 2.3.1.2. First, the value of UCS to be used for the crushing failure mode analysis was determined. Diederichs and Kaiser (1999) recommend using a UCS value of one-third to one-half of the value obtained from the laboratory tests, which represents, using the values given in Table 7,

\[
\frac{1}{3} \times UCS = \frac{1}{3} \times 160 = 53 \, (MPa), \quad \text{and} \quad (26)
\]

\[
\frac{1}{2} \times UCS = \frac{1}{2} \times 160 = 80 \, (MPa). \quad (27)
\]

An average value, UCS = 65 MPa, was subjectively considered. This UCS value and the elastic modulus of rock mass (E_{rm}) were then normalized using the effective specific gravity. The E_{rm} (Table 8) is equal to 40 GPa and the effective specific gravity is defined as follow:

\[
S.G_{\text{eff}} = S.G \times \cos(\alpha) = 2.9 \times \cos(70°) = 1 \quad (28)
\]
where $\alpha$ = HW dip angle ($^\circ$). The following equations show the normalized $E_{rm}$ and UCS values of 40 GPa and 65 MPa, respectively (Equation 29 and 30).

$$E_{rm\_norm\ (GPa\ )} = \frac{E_{rm\ (GPa\ )}}{S.G.\_{eff}} = \frac{40}{1} = 40 \ (GPa)$$  \hspace{1cm} (29)

$$UCS\_{norm\ (MPa\ )} = \frac{UCS\ (MPa\ )}{S.G.\_{eff}} = \frac{65}{1} = 65 \ (MPa)$$  \hspace{1cm} (30)

As indicated in section 2.3.1.2, both the long stope surface and square stope surface design graphs were used to allow a better approximation of the solution. The final step involved plotting the necessary input values previously determined in the two design graphs against the span length of 15 m (Figure 25).

The crushing failure mode could not be analysed with the UCS$_{\text{norm}}$ of 65 MPa because it is greater than the range of the design charts, suggesting it is unlikely to be the failure mode in this case.

For snap-thru failure mode, using an infinitely long HW, a minimal lamination thickness of approximately 0.25 m was obtained (Figure 25a) while using a square HW, a value of 0.2 m was computed (Figure 25b). Thus, the ore-skin should be stable for a minimal thickness ranging between 0.20 m to 0.25 m. As previously discussed, the average beam thickness in this case is of 0.9 m, which is almost four times greater than the minimum required thickness, suggesting that the proposed ore-skin should therefore remain stable.
Figure 25. Estimation of the minimal Vousoir beam thickness using the design chart for the conceptual case-study (after Diederichs and Kaiser, 1999).

Considering the potential damage from blasting, the pressure exerted by the weak HW on the beam, and the effect of cross jointing and wall irregularities that are not considered in the
Voussoir beam analysis, this approach underestimates the minimum ore-skin thickness required for the stability of stope HW.

Capes (2009) indicated that the Voussoir beam analysis could support the general failure mode that was observed at the George Fisher mine in Australia. However, he highlighted that it is an insufficient design approach. It is thus suggested that this approach be considered as a quick way to estimate the feasibility of the ore-skin design, but a complimentary and more rigorous analysis should be undertaken.

4.3 The ELOS Dilution Graph

As elaborated in section 2.3.2.2, the Dilution Graph helps to assess the ELOS for a stope design based on the stope HR and N' values. This section demonstrates how the Dilution Graph and the ELOS curves can be used together as a tool to determine the minimum ore-skin thickness that can be left unmined to maintain the structural integrity of the stope.

As per its definition, ELOS represents the depth of failure to be expected in regards of the HR of the studied face and the N' of the rock mass in place. Considering the Dilution Graph and the expected depth of failure, one could argue that the minimum ore-skin thickness should be at least the thickness of the ELOS value, so that only the ore-skin material, thus the ore, would fall into the stope.

The objective of leaving a particular thickness of ore along the HW is to improve the quality of the rock mass which comprises the immediate HW, consequently enhancing the stope HW stability. For the hypothetical case study, the rock mass quality \(Q'\), indicated in Table 8 were used to calculate the N' values. The N' of the rock units were determined using the four input parameters: \(Q'\), \(A\), \(B\) and \(C\), as introduced in section 2.3.2.1. Table 11 presents the resulting values of N' determined for the different rock units. The A factor (i.e., the stress factor) of the
modified stability number $N'$ is set equal to 1 for a state of relaxation. The B factor (i.e., the effect of joint orientation on the stability) was determined using the graph presented earlier (Figure 12). Considering the joint orientation presented in Table 9 and the stope configuration, a B value of 0.3 was determined. The C factor, on the other hand, accounts for the stope surface inclination effect on the stability and by Equation 5, a C factor of 5.9 was determined.

Table 11. Modified stability number values.

<table>
<thead>
<tr>
<th>Rock units</th>
<th>Q'</th>
<th>A</th>
<th>B</th>
<th>C</th>
<th>$N'$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Host rock/Ore-body</td>
<td>11-33</td>
<td>1</td>
<td>0.3</td>
<td>5.9</td>
<td>19-58</td>
</tr>
<tr>
<td>Weak HW</td>
<td>0.5-2</td>
<td>1</td>
<td>0.3</td>
<td>5.9</td>
<td>0.8-3</td>
</tr>
</tbody>
</table>

These $N'$ values for the stope design which includes an ore-skin as well as the design which omits an ore-skin (i.e., the typical stope design) were plotted on the Dilution graph for a HW hydraulic radius of 5 m (Figure 26). The blue line represents the design without an ore-skin and the yellow line represents the design with an ore-skin.

![Figure 26. Minimum ore-skin thickness determination using the ELOS lines/ Dilution Graph (after Capes, 2009).](image-url)
It is shown that for the stope design without an ore-skin, a major failure is expected to occur with the ELOS between 1 to 4 m. While for the design including ore-skin, which increases the quality of the HW rock mass, an ELOS less than 0.5 m is expected. This means that the ore-skin would only slough by blasting damage. Therefore, an ore-skin thickness larger than 0.5 m should be sufficient to keep the weak HW material from entering the mining process, diluting the ore grade.

Due to the uncertainty on the interaction of the weak HW zone and the ore-skin as well as the blast-induced damage, it will be more appropriate to apply a safety factor greater than one. In this particular case, if one decided to design the ore-skin with a safety factor of two, the resulting ore-skin thickness would be 1 m.

4.4 Numerical Solution

This section presents the results of two different numerical approaches employed for the assessment of the geomechanically stable minimum OST applied to the hypothetical underground gold mine. The two approaches were selected based on their ability to capture the importance of the joint network, which is the main governing factor for sloughage since the main HW failure mode was determined to be structurally-controlled gravity-driven.

4.4.1 Key-blocks analysis using Discrete Fracture Networks (DFNs)

The Discrete Fracture Network (DFN) modelling is a numerical tool that attempts to simulate fractured rock masses by stochastically generating three dimensional synthetic fractures. It pertains to the probabilistic approaches as described in Section 2.3.1.1 which can be used to identify the key-blocks which form around an opening. The DFN models created for this study are generated using FracMan (Golder Associates, 2014).
4.4.1.1 Background

This section presents the principles of the DFN modelling. The DFN modelling is based on stochastic method that aims to generate equiprobable 3D discontinuity models. The models are based on probability density functions of the fracture parameters including joint intensity (spacing), orientation, size and termination obtained from a limited amount of field data and are characteristic of the studied rock mass.

To generate and validate a DFN model, appropriate steps must be taken. The first step requires the collection of field data. The second step consists of statistically analysing these data and determining the probabilistic distribution functions (PDF) which fits the data. The third step involves generating the DFN model based on the input values and the final step requires the validation of the DFN model by comparing the results of the model with those obtained from the field data. Figure 27 presents the steps that are used to develop a DFN model. The validated three-dimensional and probabilistic DFN models provide more realistic estimate of the in situ fragmentation (section 2.3.1.1) and subsequently, they can be used for key-block analyses.
Figure 27. The methodology to generate a DFN model (after Grenon, 2000).

It should be noted that the DFN models should be developed based on reliable input data collected from the field. Using appropriate sampling methods and applying a bias correction, it is usually feasible to statistically characterize the structural complexity of a rock mass (Dershowitz and Einstein, 1988).
To conduct key-blocks analyses, the FracMan kinematic rock wedge built-in analysis can be used. The latter is based on the key-block theory developed by Goodman and Shi (1985). The theory uses the geometry of the fracture network to identify the blocks which form at the face of an opening and then it assesses the potential stability of each individual rock blocks. As a result, blocks are divided into two major categories; either removable or non-removable.

By modelling the stope within a DFN model, the built-in kinematic analysis will provide information on the mode of failure and the factor of safety related to the falling of each of the blocks identified in the stope HW face. The applied forces that are considered to determine the factor of safety are the normal and frictional forces. The effects of stresses are not accounted for in this limit equilibrium analysis. Based on the geometry of the block and the factor of safety obtained, the mode of failure of each block will be characterized as one of the following:

- Free falling: No sliding, \( \text{FoS} < 1 \);
- Sliding on one plane, \( \text{FoS} \leq 1 \);
- Sliding on two planes, \( \text{FoS} \leq 1 \);
- Stable, \( \text{FoS} > 1 \).

The blocks which have a computed factor of safety less than one are considered unstable. As previously defined, the total volume of these unstable blocks is equal to the expected volume of instantaneous HW sloughage. Thereby, one could design the minimal stable ore-skin thickness by examining the expected volume of sloughage.

### 4.4.1.2 DFN model generation

This section presents the steps necessary to generate the DFN models for the hypothetical case study. As previously defined, the input parameters used for the DFN models were not based
on field data, however the values used are reasonable estimations. Thus, the discussion in this section does not include steps 1 and 2 (Figure 27).

It is important to understand the stochastic nature of the DFN models and the underlying assumptions that are made. The main assumptions include:

- The use of four-sided polygonal fractures.
- The fractures are randomly distributed in space using a Poisson distribution which applies an Enhanced Baecher algorithm.
- The lognormal distribution was selected for the fracture size probabilistic density function.
- The volumetric fracture intensity (P32) was selected for the measure of fracture intensity.

The density or intensity is a measure of spatial distribution of fractures. There are four options in FracMan to assign an intensity: P10, P32, P33 and fracture count. A P10 value is the number of fractures per unit length of a linear feature such as scan-line. A P32 value is the total area of fractures per unit volume of rock mass. While the P33 value is the volume of fractures per unit volume of rock mass. As DFN models allow access to the three-dimensional volume of rock intersected by the fractures, more information can be determined compared to the usual scan-line (1D) or scan-plane (2D) fracture mapping.

Dershowitz and Herda (1992) reported that the P32 is the most appropriate indication of the state of fracturing of the rock mass. The P32 cannot be directly measured on the field, however, it can be related to P10 which is a measure that can be directly acquired through ground mapping, (Dershowitz and Herda, 1992). For the current hypothetical study, the P32 was chosen as the intensity input parameter.

To generate a DFN model which would represent the rock units described earlier: the weak immediate HW and the Host rock/Ore-body units, three key components were modelled as follow (Figure 28):
• a larger box model of 70 m x 40 m x 60 m which represents all the rock masses was developed,
• a tabular parallelepiped dipping 70° to the west that is 4 m wide and approximately 100 m high along the dipping face was generated in the middle of the box. This part represents the weak HW zone which overlies the ore body, and
• a stope of 20 m wide, 15 m long and 30 m high which also dips 70° west, was generated parallel to the weak HW.

Figure 28. The model geometry of the conceptual study elaborated using FracMan.

Table 12 presents the input values used for the generation of the DFN model. It should be noted that a Fisher distribution was assumed for the major joint sets with the exception of joint set S1 (i.e., the schistosity). A Fisher distribution was selected for this study as it is one of the most commonly used distribution for joint orientation (Tollenaar, 2008). The K value in the table describes the dispersion of the orientation cluster. A high K value represents a tight cluster and a
low K value a dispersed cluster. For the schistosity set (S1) the objective is to ensure a constant orientation, thus a constant distribution was applied. This is the distribution generally observed for schistosity joints. The joint sets S1 was considered fully persistent and a very high joint size (i.e., equivalent radius) was assigned to this joint set. The two other sets S2, and S3, were assigned smaller joint sizes.

Table 12. Input data used for DFN model generation.

<table>
<thead>
<tr>
<th>Zone</th>
<th>Joint Set</th>
<th>Description</th>
<th>Dip (◦)</th>
<th>Dip Dir. (◦)</th>
<th>Distribution</th>
<th>K</th>
<th>Equivalent Radius (m)</th>
<th>P32</th>
</tr>
</thead>
<tbody>
<tr>
<td>Box model</td>
<td>S1</td>
<td>Schistosity</td>
<td>80</td>
<td>270</td>
<td>Constant</td>
<td>-</td>
<td>100</td>
<td>0</td>
</tr>
<tr>
<td></td>
<td>S2</td>
<td>Major Joint Set</td>
<td>45</td>
<td>30</td>
<td>Fisher</td>
<td>130</td>
<td>2</td>
<td>1</td>
</tr>
<tr>
<td></td>
<td>S3</td>
<td>Major Joint Set</td>
<td>45</td>
<td>150</td>
<td>Fisher</td>
<td>130</td>
<td>1.5</td>
<td>0.5</td>
</tr>
<tr>
<td>Weak HW</td>
<td>S1'</td>
<td>Schistosity</td>
<td>80</td>
<td>270</td>
<td>Constant</td>
<td>-</td>
<td>100</td>
<td>0</td>
</tr>
<tr>
<td></td>
<td>S2'</td>
<td>Major Joint Set</td>
<td>45</td>
<td>30</td>
<td>Fisher</td>
<td>200</td>
<td>8</td>
<td>2</td>
</tr>
<tr>
<td></td>
<td>S3'</td>
<td>Major Joint Set</td>
<td>45</td>
<td>150</td>
<td>Fisher</td>
<td>200</td>
<td>6</td>
<td>1</td>
</tr>
</tbody>
</table>

Firstly, the three joint sets were generated in the entire box model, including the weak HW. The box model was designed to be larger than the actual studied zone in order to limit boundary problems. Secondly, to create more fractures in the weak HW zone, an additional generation of the three joint sets was implemented in the parallelepiped only. This additional generation uses a revised definition which includes higher fracture intensity (P32) values. That sums to the fracture intensity previously generated. These additional fractures are immediately clipped from outside the weak HW zone, producing the desired clean separation between the weak HW and the Host rock/Ore-body units. A total of five different DFN realisations were generated for this study. Figure 29 illustrates one of these realisations.
Based on the total of five realisations, an average P32 value of 3.18 m$^{-1}$ in the Ore-body and 8.30 m$^{-1}$ in the Weak HW were obtained. Ranges of the normal spacing of joint sets for each rock unit were also determined and are summarized in Table 13. The characteristic normal joint spacing shown in Table 9 falls within these ranges, validating the model.

**Table 13. The joint sets normal spacing per rock unit.**

<table>
<thead>
<tr>
<th>Joint Set</th>
<th>Description</th>
<th>Normal Set Spacing</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Weak HW</td>
<td>Host rock/Ore-body</td>
</tr>
<tr>
<td></td>
<td>m</td>
<td>m</td>
</tr>
<tr>
<td>S1</td>
<td>Schistosity</td>
<td>0.2-0.6</td>
</tr>
<tr>
<td>S2</td>
<td>Major Set</td>
<td>0.2-0.4</td>
</tr>
<tr>
<td>S3</td>
<td>Major Set</td>
<td>0.2-0.4</td>
</tr>
</tbody>
</table>

The in situ fragmentation analyses were also performed to characterise the generated models using a FracMan built-in function. Figure 30 and Figure 31 present the block volume distribution obtained for the Host rock and Ore-body unit as well as the weak HW, respectively. An average passing curve was also determined for the different realisations. An average D$_{50}$
value (i.e., the median block size) of about 40 m$^3$ could be determined for the Host rock and Ore-body unit and a $D_{50}$ value of about 0.2 m$^3$ was computed for the weak HW unit.

Figure 30. In situ blocks volume distribution of the Host rock/Ore-body unit for 5 DFN realisations.

Figure 31. In situ blocks volume distribution of the Weak HW unit for 5 DFN realisations.
4.4.1.3 Block stability analysis using DFN model

A total of six stopes with an OST varying between 0 m to 5 m were modelled within each of the five different DFN realisations. For each one of these stopes, the kinematic rock wedge analysis was performed on the stope HW face. For each identified rock block or wedge, a factor of safety was determined using the Mohr-Coulomb failure criterion. The joint surface properties presented in Table 10 and the rock density of the rock units presented in Table 7 were considered for this kinematic analysis. For each realisation and for the different OSTs, an analysis was performed to determine the volume of unstable blocks forming at the HW face. Figure 32 shows an example of the results obtained from the analysis for one of the five DFN realisations.

Wedges forming at the HW face are identified and analysed in terms of stability. The stable and unstable wedges are shown in green and red respectively. The analysis shows that the total volume of wedges forming when there is an OST equal to zero is significantly greater than the amount of wedges which form when the OST is greater or equal to 1 m (Figure 32).

Figure 32. Rock wedge analysis performed for different OST for one of the DFN realisation (green: stable; red: unstable).

Figure 33 shows the average volume of unstable material determined for the different ore-skin thicknesses. Bars bounding the minimal and maximal values obtained from the different DFN realisations are also shown.
The typical stope which does not include an OST shows an average volume of unstable (i.e., falling) material of 189 m$^3$ (an equivalent ELOS of 0.42 m). This value drastically decreases for the case with an ore-skin of 1 m. For an OST equal to 1 m, an average volume of unstable blocks of 14 m$^3$ is observed (an equivalent ELOS of 0.03 m). This value then remains in a similar range for all cases with an OST greater than 1 m.

Figure 34 and Figure 35 show the volume distribution of the unstable blocks for both the design without an ore-skin and the design with an ore-skin of 1 m. The volume distributions show that the unstable blocks in the weak HW have a D$_{50}$ value equal to 0.2 m$^3$ for the case without an ore-skin. This value is less than a third of the D$_{50}$ value of 0.9 m$^3$ obtained from the unstable blocks which formed in the case using a 1 m ore-skin thickness.
Figure 34. Fragmentation analysis of unstable material without an ore-skin.

Figure 35. Fragmentation analysis of unstable material with an OST equal to 1 m.
This analysis is purely kinematic and the use of DFN models made it possible to consider the spatial distribution of the three dimensional joint sets and to study a more appropriate range of the wedge volume distribution in comparison to a more deterministic approach. However, there are limitations and simplifications which make the use of the kinematic wedge failure analysis unsatisfactory for the geomechanical assessment of a stable minimum OST. These include:

- the limited representation of the rock mass complexity,
- the stresses acting on the opening are not taken into account,
- the fractures are assumed to be planar,
- the wedges are assumed to be rigid blocks and cannot deform,
- the displacement and interaction of the block which modifies the force equilibrium is not considered.

Effectively, as one block falls, the other blocks located behind, which were previously stable (i.e., restrained of movement), are now exposed and the force equilibrium is changed. In DFN kinematic analysis, the time dependency of the sloughage induced by unravelling of the blocks is not taken into account. Therefore, the volume of sloughage estimated with the DFN kinematic analysis method is underestimated. In addition, using DFN kinematic analysis the stability of the blocks is assessed without considering the effect of the confining stresses which generally enhance the stability of the wedges. The next section presents an approach that aims overcoming some of the limitations highlighted here by integrating the DFN models into a stress analysis numerical model.
4.4.2 Stress-structural analysis (DFN-3DEC)

To investigate the minimum OST which is geomechanically stable for the hypothetical case study, a stress-structural analysis was performed. The stress-structural analysis was carried out as a hybrid DFN-3DEC model, which embeds the DFN model into the 3DEC (3-dimensional Discrete Element Code) software developed by Itasca.

The interest of the DFN-3DEC combination approach is to overcome the limitations highlighted in the previous section by linking the comprehensive fracture network (the DFN model) to a stress analysis package.

4.4.2.1 Background

A general discussion on the use of 3DEC as a DEM is provided in this section. The algorithm of the DEM is based on both a force-displacement law which determines the blocks interaction and a law of motion which determines the displacements induced by the acting forces. The DEM makes it possible to evaluate the different forces acting on independent blocks, where the blocks are defined by the intersection of discontinuities. The distinguishing features of 3DEC as formulated by Itasca (2014) are as follows:

- ‘The rock mass is modeled as a 3D assemblage of rigid or deformable blocks.
- Discontinuities are regarded as distinct boundary interactions between these blocks; joint behavior is prescribed for these interactions.
- Continuous and discontinuous joint patterns can be generated on a statistical basis. A joint structure can be built into the model directly from the geological mapping.
- 3DEC employs an explicit in-time solution algorithm that accommodates both large displacement and rotation, and permits time-domain calculations.’

The DFN-3DEC approach combines the advantages of the DFN (i.e., the three-dimensional representation of the fracture network) and the 3DEC features which allow
capturing the rock mass unraveling process. The unravelling process as expected to occur in stope HW sloughage can be simulated by the changes in force distribution that accompany the progressive displacement of the rock blocks. Additionally, 3DEC considers the effect of in situ stresses and computes in-time stress redistribution, incorporating the confining stress which was not previously considered in the kinematic stability analysis using DFN.

4.4.2.2 Model generation

A large 3DEC model of 60 m wide, 60 m long, and 90 m high was developed to reduce the boundary effect related to computing stresses. The rule of thumb suggests using a radius three times the size of the object of interest (i.e., the stope) to determine the dimensions of the model. For this purpose, the dimensions of the stope, as illustrated in Figure 22, were used.

The three rock units (weak HW, Ore-body and Host rock) were generated throughout the 3DEC model as illustrated in Figure 36; the weak HW is shaded yellow, the Ore-body is shaded red and the Host rock is shaded blue. The weak HW is 4 m thick and dipping 70° west similar to the Ore-body. The intact rock properties summarized in Table 7 were assigned to all the blocks pertaining to their respective modelled zone. Boundaries with a 1 m thickness were modelled around the model, shaded green, and are assigned the Host rock properties. All of the blocks are modelled as rigid (i.e., non-deformable).
A DFN model was imported into the 3DEC model directly from FracMan. Although a total of five different DFN realisations were generated in the previous section, only one DFN-3DEC model was generated. Thus, one of the FracMan model was arbitrarily selected to be integrated in the 3DEC model. For simplification purposes and computational time reduction, the DFN was only used to discretize the model in the zone of interest (Figure 37). This zone of interest was defined as a volume which extends 3 m into the competent HW, 3 m into the back, extends 7 m into the Ore-body and covers the entire thickness of the weak HW.
Figure 37. Illustration of the DFN model integrated into the 3DEC model along with the stope location and dimensions.

Beyond this volume of interest, orthogonal joints were generated and to keep the discrete blocks formed by the orthogonal joints from displacing, a very high strength was assigned to these ‘fictitious’ joints, restraining them from yielding (Table 14).

Table 14. Properties assigned to fictitious joints.

<table>
<thead>
<tr>
<th>‘Fictitious’ joints</th>
<th>Kn (GPa/m)</th>
<th>Ks (GPa/m)</th>
<th>Tension (GPa)</th>
<th>Cohesion (GPa)</th>
<th>Friction angle (◦)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>500</td>
<td>100</td>
<td>1000</td>
<td>1000</td>
<td>89</td>
</tr>
</tbody>
</table>

The joints properties summarized in Table 10 were assigned to the discontinuities located in the zone of interest. The Coulomb-slip constitutive model was used to assess the stability
analysis of rock mass in the stope HW zone. Additionally, in situ stresses were implemented in the model by placing the center of the 3DEC model at a depth of 500 m below surface. The in situ stress field used for the analyses is summarized in Table 15.

Table 15. In situ stress field used for the analyses.

<table>
<thead>
<tr>
<th>Principal stress component</th>
<th>Orientation</th>
<th>Gradient (MPa/m of depth)</th>
<th>Value at 500 m depth (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Major</td>
<td>Horizontal (East-West)</td>
<td>0.0477, k1=1.8</td>
<td>24</td>
</tr>
<tr>
<td>Intermediate</td>
<td>Horizontal (North-South)</td>
<td>0.0318, k2=1.2</td>
<td>19</td>
</tr>
<tr>
<td>Minor</td>
<td>Vertical</td>
<td>0.0265</td>
<td>13</td>
</tr>
</tbody>
</table>

Once the in situ stresses were implemented into the model, the stope dimension was extracted from the 3DEC model. A total of three different 3DEC models were generated in order to simulate stope with OST varying from 0 to 2 m (Figure 38).

![Figure 38. The three 3DEC models with OST equal to 0 m, 1 m and 2 m.](image)

4.4.2.3 The Hangingwall stability analysis

The three models were run for a similar period of time of 25 seconds (problem time). For each 3DEC model, approximately 500 hours (three weeks) were required to reach this amount of time. Due to the calculations schemes and the complexity of the model geometry, the
computational time was too significant. Consequently, the models were not cycled to equilibrium and the long-term stability was not captured. As unravelling is a long-term deformation process, it could therefore not be analysed. Instead, it can be argued that the results obtained after 25 seconds for the different ore-skin design represent the instantaneous stability of the stope designs (Figure 39).

![Figure 39. 3DEC model results for computed instantaneous sloughage.](image)

Figure 39 shows that the typical stope design (i.e., without ore-skin) experiences more HW sloughage. A total volume of 15 m³ of instantaneously sloughed material was measured for the typical stope design compared to a total volume of about 5 m³ for the design with an OST equal to 1 m and 2 m³ for the design with an OST of 2 m (Figure 40). The results indicate that the instantaneous stability of the HW is enhanced by the presence of an ore-skin.
Figure 40. Volume of material computed as instantaneous sloughage.

To analyse the long-term stability of the stope HW (i.e., capture the unravelling process) without running the models for more computational cycles, the displacement contours were investigated. A similar approach was used by Katsaga and Andrieux (2014) for the stability assessment of an underground opening in a mine in Yukon, Canada. The displacement of a rock block is an indicator of kinematic freedom and therefore of its stability. Figure 41 shows the total displacement contours computed after 25 seconds. From the three different 3DEC models, arched displacement contour patterns can be observed in the HW as well as a reduced extent of the high displacement values when an ore-skin is designed. As the thickness of the ore-skin augments, the extent of the high displacement values contours reduces. Similar observations can be drawn from looking at the displacement contours in the back. This leads to interpret that the ore-skin improves both the stability of the HW and of the back of the stope.
Figure 41. Displacement plot- side view at mid span, looking North.
Two displacement thresholds were considered for the long-term stability analysis and consequently two displacement groups were formed: blocks which displace by more than 8 mm are grouped together and shaded red while the blocks which displace by more than 9 mm are grouped together and shaded dark grey.

Figure 42 shows the displacement plots for each stope profile. The weak HW zone is outlined by the dotted red lines. The displacement groups for each profile are identified by a dotted black lines. It can be seen that for an OST of 0 m, blocks which displace more than 0.9 mm reach the back of the weak HW, while for a case where OST is 1 m the sloughage extent is within the weak HW with a maximum depth of approximately 3 m. For OST of 2 m, the displacement group reaches a maximum depth of approximately 2 m and remains in the ore. For the displacement group of more than 0.8 mm, the extent of sloughage zone become larger and deeper in all the three stope design models.

Figure 43 shows the sloughage area on the HW face of the different stope models using the two displacement groups as threshold. The footprint of the sloughage zone on the HW face is more important for the case without an OST than for the models where the OST is of 1 m or of 2 m.
Figure 42. Displacement plot – looking at the stope profiles; a) displacement group ≥ 0.9 mm b) displacement group ≥ 0.8 mm.
Figure 43. Displacement plot – looking at the HW face; a) displacement group ≥ 0.9 mm b) displacement group ≥ 0.8 mm.

Overall, Figure 42 and Figure 43 show that the displacement groups tend to form an ellipsoidal shape, meaning that the expected long-term sloughage profile would present a similar shape. The sloughage seem to develop from bottom to top which can be explained by the low angle that the schistosity joint set (S1) makes with the HW. However, the models do not account for the presence of the blasted ore which remains in the open stope until the end of the extraction (i.e., the ore rill) and which contribute to the stability of the HW (Capes et al., 2006b).

Nonetheless, it is interesting to note that the interpreted long-term unravelling shape is analogous to the arched shape observed by different authors and previously reported in chapter 2 and 3 (Ran, 2002; Henning, 2007; Capes, 2009) and that its extent stops when it reaches the
higher quality HW as also reported in the scientific literature (Potvin, 1988; Capes et al., 2006a; Capes, 2009).

The total volume of unstable blocks as defined by the displacement groups for the different stope models is summarized in Table 16 and their translated equivalent linear overbreak/sloughage (ELOS) values are also given.

Table 16. Volume of unstable blocks measured based on displacement groups.

<table>
<thead>
<tr>
<th>OST (m)</th>
<th>Volume (m$^3$)</th>
<th>Displacement ≥ 0.9 mm</th>
<th>Displacement ≥ 0.8 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Weak HW</td>
<td>Ore-Body</td>
</tr>
<tr>
<td>0</td>
<td></td>
<td>269</td>
<td>0</td>
</tr>
<tr>
<td>1</td>
<td></td>
<td>86</td>
<td>165</td>
</tr>
<tr>
<td>2</td>
<td></td>
<td>0</td>
<td>141</td>
</tr>
</tbody>
</table>

The total volume of unstable blocks recorded for the typical stope design (i.e., without ore-skin) and the design with an OST of 1 m is not significantly different for each displacement group. The difference exists in the type of material that is sloughing. For the typical stope design, the sloughing volume is comprised entirely of the uneconomic weak HW material. While for the design with an OST of 1 m, the sloughing volume is composed of both economic (i.e., ore) and uneconomic material. For the design using an OST of 2 m, the total volume of expected long-term sloughage reduces considerably compared to the two other models. Additionally, almost the entire sloughing volume is composed of ore as the two displacement groups were almost entirely located within the ore-skin. Considering the displacement group ‘≥ 8 mm’, the long-term expected volume of uneconomic material sloughing reduces from 493 m$^3$ for the design with no ore-skin to 230 m$^3$ for the design with an OST of 1 m and to 23 m$^3$ for the design with an OST of 2 m. These results concur with the qualitative assessment of the stability formulated earlier; the HW stability increases with the increase of the OST.
Although these displacements are time-dependant, this long-term stability analysis ignores the amount of time required for failure; and time is of major interest when implementing the ore-skin design. The effect of time on failure represents an important factor as ultimately, the objective of designing an ore-skin is to keep the uneconomic material from falling into the stope and entering the mining process. Therefore, the ore-skin design does not need to stand-up longer than the time that is required to extract the ore out of the stope (i.e., the stope life time). Even if the ore-skin fails at the end of the stope extraction process, part of the ore which was sterilized by the ore-skin design could be recovered. The optimal ore-skin thickness to be design is therefore the thickness which stands at the yielding point.

The results of the DFN-3DEC approach showed that the OST of 1 m and of 2 m considerably reduce the amount of instantaneous sloughed material. The long-term HW stability analyses showed that for an OST of 1 m and 2 m, the expected volume of uneconomic material to fall into the stope was significantly reduced. As the time required for the expected long-term failure is unknown, either the OST of 1 m or the OST of 2 m could be considered as being the geomechanically minimum stable OST, depending on the stope life time.

Finally, the significant amount of computational time required to perform the 3DEC-DFN analysis is identified as being the major disadvantage of the approach. This led to providing an estimate for the expected sloughage volume based on computed displacement. Although this is believed to be a good estimate, it ignores the notion of time which is of major interest when optimizing both the stope economics and stability.
4.5 Summary

A total of four different approaches were employed to investigate the geomechanically stable minimum OST. The different approaches which belong to either the analytical, empirical or numerical methods, all showed that leaving a certain thickness of ore along the weak HW increases the stope stability, consequently reducing the volume of HW sloughage to be expected.

The analytical solution, the Vousoir beam analogy, is the approach that requires the most simplifications and it underestimates the required thickness of the ore-skin. The Dilution Graph approach is a qualitative approach developed based on past experiences, however, if developed as site specific, the degree of confidence can increase and it can be used as a good design tool. The kinematic analysis using DFN provides a realistic distribution of rock blocks that could form on the stope HW, however, it fails to capture the total volume of HW slough due to the unravelling process. Finally, the 3DEC-DFN approach allowed to incorporate the effect of in situ stresses on the structural stability of the stope HW. Using this approach it was possible to quantify the volume of slough which fell into the stope immediately after its excavation. Moreover, the approach enabled investigation of the long-term profile of the HW sloughage for different stope design based on displacement contours. However, due to the significant computational time, it was not possible to predict the exact time for the evolution of the stope HW sloughage profile. The use of more than one approach when designing an ore-skin is recommended as it would provide the mine design engineer with a reasonable estimate.
CHAPTER 5
CONCLUSIONS

There is a clear interest in the resource industry to improve mining practices. This thesis focused on the optimization of stope design in underground open stope mines. More specifically, it focused on a particular alternative design called the ore-skin design. Although the ore-skin approach has been used in different underground mine operations around the world, it was found that no previous attempt had been made to evaluate the effect of implementing this alternative design approach on a mine operation. The objective of this thesis was to contribute to our understanding of the viability of the ore-skin design approach as an alternative technique for HW sloughage control.

5.1 Summary

A two-stage approach which serves determining the viability of implement the ore-skin design was proposed and a hypothetical typical Canadian underground gold mine was investigated throughout the entire thesis using this alternative design approach. The first stage of the approach consists in evaluating the economic viability of the design by determining the break-even ore-skin thickness. The second stage consists in determining the minimum geomechanically stable ore-skin thickness which will keep the uneconomical material from falling within the stope and entering the processing stream.

In Chapter 3, a method that aims to determine the maximum ore-skin thickness that can be left unmined while ensuring the economic viability of the stope design was proposed. This ore-skin thickness represents the thickness for which the profit generated by the typical stope design (i.e., with no ore-skin) and the one generated by the ore-skin design are equal. Only the
economic and geometric parameters of the stope are necessary for this investigation. A break-even OST of 2.4 m was determined for the hypothetical Canadian underground gold mine.

In chapter 4, the minimum geomechanically stable OST was investigated using a total of four different approaches. Using the analytical Vousoir beam analogy, a minimal stable Vousoir beam of 0.2-0.25 m was obtained. Considering the limitations and assumptions of the approach, it was found to be underestimating the minimum ore-skin thickness required to maintain the stability of the HW.

The second approach consisted in using the Dilution Graph. The assessment showed that an OST of 1 m should be sufficient to keep the weak HW from sloughing. It was highlighted that the Dilution Graph approach becomes an even greater tool if it is developed as a site-specific tool.

The third approach, the key-block theory, was performed using DFN models. The added value of the DFN models was well explained in section 4.4.1. The assessment showed that the amount of expected sloughage was greatly reduced for an OST of 1 m. However, the approach does not capture the rock mass unravelling and does not account for the effect of the confining stresses on the stability of the wedges that could form in the stope HW. It was therefore concluded that this approach underestimates the volume of sloughage to be expected.

The fourth approach consisted of a 3DEC-DFN analysis. Performing the latter, the instantaneous HW stability was computed and showed that an OST of 1 m and of 2 m significantly increased the HW stability. Using the computed total displacement of rock blocks, the long-term stability was also investigated and it was concluded that an OST of 1 m or 2 m both could be the minimum geomechanically stable OST, depending on the stope life time. Effectively, time was highlighted as being an important factor for the ore-skin design as it is a
controlling factor for both the stability and the economics of the stope. Additionally, it should be noted that none of the above approaches aimed capturing the effect of time into account.

However, the 3DEC-DFN analysis concluded that both designs (i.e., with an OST equal to 1 m and with an OST equal to 2 m) are feasible. For the conceptual underground gold mine case study, the stope design with a 1 m and 2 m OST generate a total profit of $1,419,700 and $1,345,200, respectively. These profits, in comparison to the typical stope design (i.e., without ore-skin), result in a positive profit margin of approximately $106,000 and $31,500, respectively.

The results of the two-stage approach showed the viability of implementing the ore-skin design in the studied hypothetical Canadian underground gold mine. However, the results also indicated that the stope life time is an important factor as it influences the ultimate stable stope shape and also the minimum geomechanically stable ore-skin thickness. Acceleration of stope production rate could limit time-dependent stope hangingwall fall off. The optimal ore-skin thickness design is then the one that stands at the yielding point.

5.2 Major Achievements

The major achievements of this thesis include the following:

- Introduction of the ore-skin design as a viable alternative open stope design option and establishment of its interest for the overall mine economics;

- Development of a break-even formula which aims comparing the economic viability of implementing the alternative design versus the typical stope design;

- Identification of the analytical, empirical and numerical methods which can serve assessing the minimum geomechanically stable ore-skin thickness and identification of their advantages and limitations;
• Discussion on the performance of the proposed ore-skin design method through a didactical example (i.e., investigation of the hypothetical Canadian underground gold mine).

5.3 Future Work

This thesis focused on developing an approach which one could use to evaluate the effect of implementing the ore-skin design in an open stope mine as an alternative to control HW sloughage. For future work, it is recommended to further investigate the followings:

• The effect of time on hangingwall stability.
• The role played by the ore rill on the stability of the design.
• Finally, the effect installing cable bolts together with leaving an ore-skin has on the ore-skin design stability. Potentially, the mix-design could show greater results on the overall economics of the mine, leading the way to continuous improvement of the mining practices.
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