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A METHODOLOGY TO EVALUATE THE IMPACT OF MAJOR GEOLOGICAL FAULTS ON THE STABILITY OF BACKS IN OPEN STOPES USING DEM

TESIS PARA OPTAR AL GRADO DE MAGISTER EN MINERÍA

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Since the introduction of the original Stability Graph Method by Mathews et al. in 1981, its use has been widely extended in underground mining of Sublevel Open Stoping (SLS). However, the methodology is restrictive because objectively represents an applicable tool only for similar conditions to the case histories (data) where it was calibrated (site-specific conditions). In addition, and not less important, the stability graph methodology does not contemplate in its analysis the major geological structures but assesses the incidence of minor discontinuities, leaving aside one of the main reasons associated to the unplanned dilution due to overbreak, the faults. Therefore, is important to determine the influence of major geological faults in the design of open stopes and their direct impact in the information for mining design and planning.

The main objective of this thesis is to calibrate and verify a criterion that allows an evaluation of the impact of major geological faults in the stability on backs of open stopes, through the numerical modelling and analysis of overbreak of three case studies. To numerically model the response of the rock mass and underground excavations, a linear-elastic constitutive model using the distinct element method (DEM, software *3DEC*[1] *v5.2*) was built.

The numerical methodology is divided in three specific stages for each selected case, resulting in nine models in total. Then, based on the results of the numerical modelling, the first and third stage were compared, analyzing if there was an impact caused by the major geological faults in the overbreak on backs of the open stopes.

The results show that $\sigma_3 \leq 1Mpa$ remains the criterion that best recreates the overbreak in the cases. Based on the criterion and numerical models, a fault factor for the backs (F_b) of the open stopes was introduced to improve the empirical stability result of the case studies.

[1] Property of Itasca

RESUMEN DE LA TESIS PARA OPTAR AL GRADO DE: Magíster en Minería
POR: Mireya Patricia Móvil Castro
FECHA: Septiembre 2021
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UNA METODOLOGÍA PARA EVALUAR EL IMPACTO DE FALLAS GEOLOGICAS MAYORES EN LA ESTABILIDAD DE TECHOS DE CASERONES USANDO DEM

Actualmente, y desde su presentación, el gráfico de estabilidad original de Mathews *et al*. (1981), ha sido ampliamente utilizado en minería subterránea de Sublevel Stoping (SLS). Sin embargo, la metodología presenta varias limitantes, una de ella es el ser representativa sólo para condiciones similares a las cuales fue calibrada (condiciones específicas de sitio). A su vez, el gráfico no contempla en su análisis a las estructuras geológicas mayores, sino que evalúa la incidencia de las diaclasas, dejando de lado una de las principales razones a las cuales se encuentra asociada la dilución no planificada por sobre excavación, las fallas. Por tal motivo, resulta importante determinar la influencia de las fallas mayores en el diseño de caserones y su repercusión directa en la confiabilidad de la información para planificación minera.

El objetivo principal de esta tesis es calibrar y verificar un criterio que permita evaluar el impacto de las fallas geológicas mayores en la estabilidad del techo de caserones mediante modelamiento numérico, a partir de la selección de dos casos de estudio en dónde se ha presentado sobre excavación en el techo de caserones (minería chilena de Sublevel Stoping) y uno adicional de verificación considerado cómo estable. Es importante destacar que, para modelar numéricamente la respuesta del macizo rocoso, se utilizaron modelos constitutivos de carácter lineal-elástico en el software de elementos distintos *3DEC*[1] *v.5.2*. El análisis se concentró en los techos de caserones, debido a que no se aprecian grandes sobre excavaciones en las paredes verticales.

La metodología numérica se dividió en tres etapas aplicables a cada caso seleccionado, dando como resultado nueve modelos en total. En una primera etapa para evaluar la sobre excavación, se seleccionó cómo criterio la magnitud del esfuerzo principal menor (σ_3). Luego, se replicó en el software: el diseño del caserón, las fallas mayores asociadas, las propiedades del macizo rocoso, y las condiciones de esfuerzos. Utilizando el CMS (Cavity Monitoring System) se calibraron los modelos utilizando el criterio seleccionado previamente. La segunda etapa, correspondió a una verificación del criterio calibrado. Luego, una tercera etapa, consistió en replicar la primera, pero esta vez sin incorporar las fallas, evaluando nuevamente la sobre excavación. Así, basado en el modelamiento numérico se compararon los resultados de la primera etapa con la tercera, logrando determinar el impacto debido a las fallas en la sobre excavación del techo de caserones, y mostrando que el criterio que mejor recrea dicha sobre excavación es $\sigma_3 \ \leq 1 \ MPa.$

Finalmente, utilizando los resultados numéricos, un factor de falla para los techos (F_b) que describe la influencia de las estructuras geológicas mayores en su estabilidad fue introducido. Mediante el factor de falla, es posible modificar y mejorar la clasificación de estabilidad empírica de cada techo de los caserones estudiados, modificando la metodología clásica de Mathews, e incorporando el impacto de fallas geológicas mayores a la metodología. Sin embargo, se recomienda tener cuidado con el uso de resultados y parámetros presentados, ya que su uso está restringido a cada condición de sitio en particular y solo debe usarse en términos referenciales para el modelamiento numérico.

[1] Propiedad de Itasca

To Guillermo and Maryi.

"Honor your father and your mother, as the Lord your God has commanded you, so that you may live long and that it may go well with you in the land the Lord your God is giving you"

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In addition, I wish to acknowledge the infinite and unconditional support that my parents and siblings have given to me over the past few years. They are an example of love and unconditionality. All my triumphs are yours. I love you so much.

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Introduction

1.1 Presentation of the thesis

The stability of an excavation depends on several factors, mainly including: stress conditions, rock mass properties, geometry and geological characteristics. In Chile, at medium level mining the most used underground mining method is the Sublevel Open Stoping (SLS) (Zablocki, 2013). Sublevel Stoping is a selective, self-supported, mechanized and with high levels of productivity method. SLS is used in regular ore deposits of large thickness and tabular to massive habit with vertical to sub-vertical dip. In this context, for open stope design the stability graph method developed by Mathews et al. (1981) and later modified mainly by Potvin (1988), Nickson (1992), Clark and Pakalnis (1997), Suorineni (1998); Suorineni et al. (2001, 1999), Trueman et al. (2000) and Mawdesley (2002) is widely used. The graphical stability method relates the hydraulic radius (HR) and the stability number (N) defined as shown in the equation 1:

$$N = A \cdot B \cdot C \cdot Q' \quad (1)$$

Where, A is the rock stress factor, B is the joint orientation adjustment factor, C is the gravity adjustment factor, and Q' is the modified tunnelling quality index from the one introduced by Barton et al. (1974). The modified tunnelling quality index (Q') sets the stress reduction factor (SRF) and the joint water condition factor (J_w) to 1. Therefore, this modified index only accounts for rock mass strength and its structures. Note that the stress condition is accounted by the A factor.

Since this empirical methodology should only be implemented in similar conditions (site-specific conditions) to those it was calibrated, several authors (Bewick and Kaiser, 2009; Nickson, 1992; Papaioanou and Suorineni, 2016; Stewart and Trueman, 2004, 2001; Vallejos et al., 2018, 2016; Vallejos and Díaz, 2020; among others) have made critical reviews and modifications in order to achieve a greater representativeness. This has been reflected in a significant increase in the database, from 55 initial case histories (Mathews et al., 1981) to more than 400 case histories (Mawdesley et al., 2001). However, note that it is recommended to use the stability curves from literature when there is not enough information for the analysis (site-specific case histories) in the early stages of engineering (new mines).

Furthermore, the methodology does not consider the presence of major geological faults within its analysis. The first advances related to the subject were introduced by Suorineni (1998), who presented an additional adjustment factor which he called F_w. The factor determines the effect of faults on the walls of the stope, considering values from 0.01 (maximum potential impact) to 1 (no adverse effect). Thus, the application of this factor achieves a significant decrease in the value of N up to two orders of magnitude depending on the potential influence that the faults may have on the stability of the walls (Azorín, 2018). Although it is theoretically highly applicable to walls (with some issues in the mining practice), it shows a restriction by not considering the presence of major geological faults directly related to the stability of backs in open stopes.

To face this problem, this thesis aims to evaluate, calibrate and verify an overbreak criterion that allows explaining the impact of major geological structures on backs of open stopes. As a part of the study, numerical models of representative case histories of SLS Chilean mining in the software *3DEC v5.2* were implemented.

1.2 Hypothesis

The hypothesis of this research considers that the presence of major geological faults on the backs of open stopes influence and lead the overbreak, and their impact can be evaluated through numerical modelling. Previous authors, such as Mathews et al. (1981), Clark & Pakalnis (1997), Nickson (1992), Potvin (1988), Suorineni (1998, 1999, 2001), Mawdesley (2002), Castro (2015), Vallejos & Díaz (2020), among others, have concentrated all their efforts on the global analysis of overbreak in the walls of the open stopes, not being specifically focused on their backs.

1.3 General Background

According to Brady & Brown (1993), underground mining methods can be classified by the type of support they have. The type of support can be natural, artificial or none, as indicated in Figure 1.1. This study will focus on Sublevel Open Stoping mining, which is mostly naturally supported.

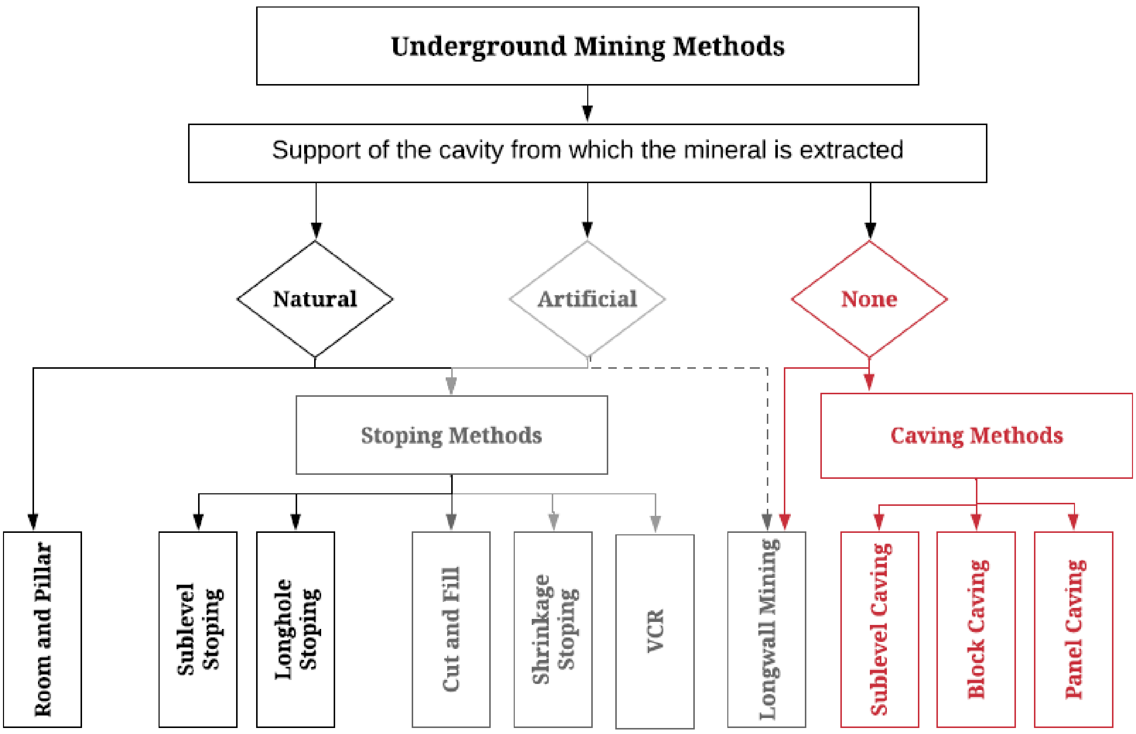

Figure 1.1 Classification of underground mining methods (Brady & Brown, 1993).

1.3.1 Sublevel Stoping (SLS)

Sublevel Stoping (and Sublevel Open Stoping) is an underground mining method applied to sub-vertical (and vertical) ore bodies and widely used in Chilean medium level mining. The mineral is extracted by sublevels using blasting and gravitational flow of material towards the extraction points located at the base of the open stopes. There are two possible variants of the method: longitudinal and transverse. Longitudinal Sublevel Stoping is used in narrow veins, which do not exceed 15 m in thickness and consider stopes parallel to the strike of the vein. On the other hand, for massive ore bodies, the stopes are oriented perpendicular (transverse) to the strike and pillars are left between cavities. For these cases, the recovery of the pillars requires some type of consolidated fill (Brady et al., 2005). Figure 1.2 shows a scheme of Sublevel Stoping (SLS) development, whereas Figure 1.3 illustrates an example of a regular open stope in SLS.

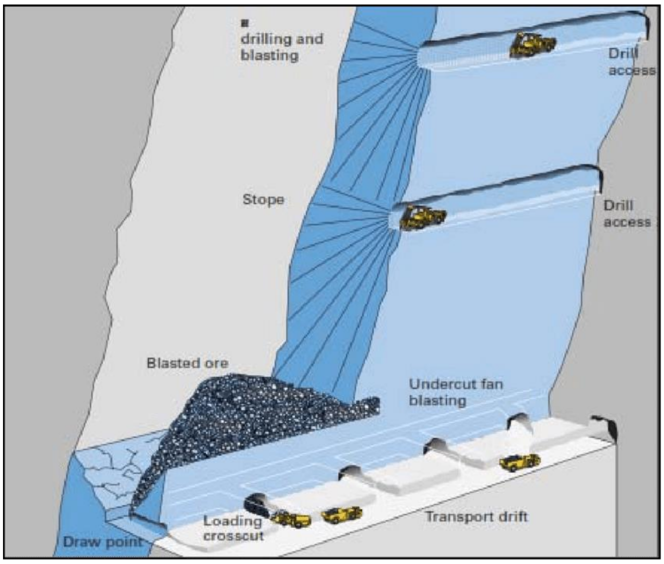

Figure 1.2 Scheme of Sublevel Stoping development (Hamrin et al., 2001).

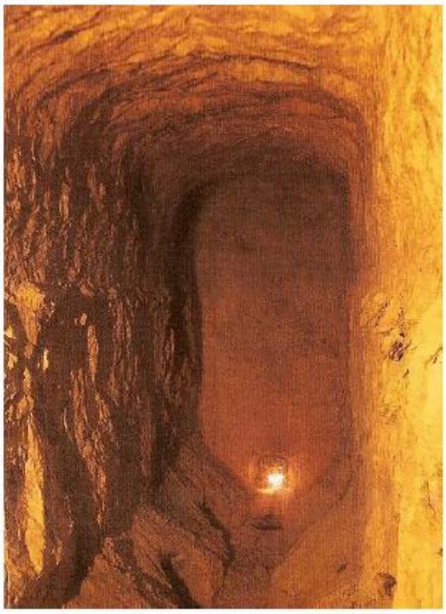

Figure 1.3 Example of a regular open stope in SLS (Castro et al., 2007).

The stability graph method developed by Mathews et al. in 1981 as an engineering project for Golder Associates, has been widely used for the design of open stopes in SLS (originally conceived for depths below 1000m). Through the development of two factors: the stability number (N) and the shape factor (S) (or hydraulic radius, RH), defined in detail in the following chapters, the method qualifies the stability of the walls in open stopes. To date, the stability graph method (original and modified) present, among others, the following limitations:

- **i.** Complex stopes geometries are often oversimplified as they were mostly intended for regular, non-veined stopes.
- **ii.** The effects of bad blasting are usually ignored.
- **iii.** Subjectivity in defining the areas of the stability graph.
- **iv.** The effects of major faults are not considered.
- **v.** Quantification of overbreak for the backs and hanging walls.
- **vi.** Fills are not considered.
- **vii.** The sequence of extraction of multiple stopes is not considered.

In the past 30 years, researchers have focused on minimizing the effects of these limitations. The most relevant events related to the empirical stability graph method are shown in Table 1.1. These changes have resulted in different values of the stability number for the same surface of the open stope and different transition zones in the stability graph.

Table 1.1 Based on the stability graph modifications[2].

Period	Developments
1980-1985	Introduction of stability graph – 26 case histories (Mathews et al., 1981).
1985-1990	Calibration of stability graph factors and zones – 175 cases (Potvin, 1988).
1990-1995	1. Tentative cablebolt support line (Potvin and Milne, 1992). 2. Re-definition of unstable/cave (supportable transition boundary – cablebolt support line) (Nickson, 1992). 3. 1st partial statistical definition of stable/unstable zone (Nickson, 1992) 4. Proposed dilution lines added to stability graph (Scoble and Moss, 1994).
1995-2000	1. Re-definition of the transition zones (Stewart and Forsyth, 1995). 2. Modified gravity factor for sliding failure (Hadjigeorgiou et al., 1995). 3. Second partial statistical definition of stable/unstable zones (Hadjigeorgiou et al., 1995). 4. Introduction of radius factor RF (Milne et al., 1996). 5. Calibration of proposed dilution lines ELOS (Clark and Pakalnis, 1997) . 6. Modified gravity factor for footwalls with shallow dips <70 (Clark and Pakalnis, 1997) . 7. Proposed volumetric index (Germain and Hadjigeorgiou, 1998). 8. First complete statistical analysis of stability graph using Baysian likelihood statistic (Suorineni, 1998). 9. Introduction of fault factor (Suorineni, 1998; Suorineni et al. 1999). 10. Modified stress factor to include tension and stress-dependent transition zones (Diederichs and Kaiser, 1999).
2000-2005	1. Expanded database to about 400 cases and modified stability graph zones from Australian database (Mawdesley et al., 2001; Trueman et al., 2000). 2. Second complete statistical analysis using logistic regression – 483 case histories (Trueman and Mawdesley, 2003). 3. Time-dependent stability graph (Suorineni et al., 2001a).
2005-2010	1. Incorporation of a fault factor into the stability graph method: Kidd mine case studies (Suorineni et al., 2001b). 2. Numerical modeling to validate the B-factor (Bewick and Kaiser, 2009).
2010-2015	1. Development of an integrated platform for stability analysis and design in sublevel stoping mines—MineRoc (Vallejos et al., 2015). 2. Statistical analysis of the stability number adjustment factors and implication for underground mine design (Vallejos et al., 2016). 3. Numerical modeling of overbreak in underground mining of Sublevel Stoping (Castro, 2015).
2015-2020	1. Development of a generalised dilution-based stability graph for open stope design (Papaioanou and Suorineni, 2016). 2. Stability graph using major geological structures (Vallejos et al., 2018a). 3. Three-dimensional effect of stresses in open stope mine design (Vallejos et al., 2018b). 4. Assisted geotechnical design for sublevel open stoping using MineRoc® software (Vallejos et al., 2020). 5. A new criterion for numerical modelling of hangingwall overbreak in open stopes (Vallejos and Díaz, 2020).

1.3.2 Dilution / Overbreak

When open stopes are designed, stability in both walls and backs is essential for mine planning process. Therefore, minimize the dilution and maximize the recovery of ore are the main objectives. However, the concept of dilution is quantified by the extra amount of waste material extracted in the process, mainly related to walls. According to Scoble & Moss (1994), in relation to the degree of reliability on the origin of the dilution, it is possible to distinguish two types of dilution:

- ***Planned dilution*****:** Refers to waste material that is outside the economic definition of ore and is incorporated as part of the mining design.
- ***Unplanned dilution****:* Corresponds to the additional material extracted that does not correspond to ore, either rock or fill from the outskirts of the margins of the stope.

Figure 1.4 illustrates the types of dilution in the design of a stope.

[2] Modified from Suorineni (2012), A critical review of the stability graph method for Open Stope Design.

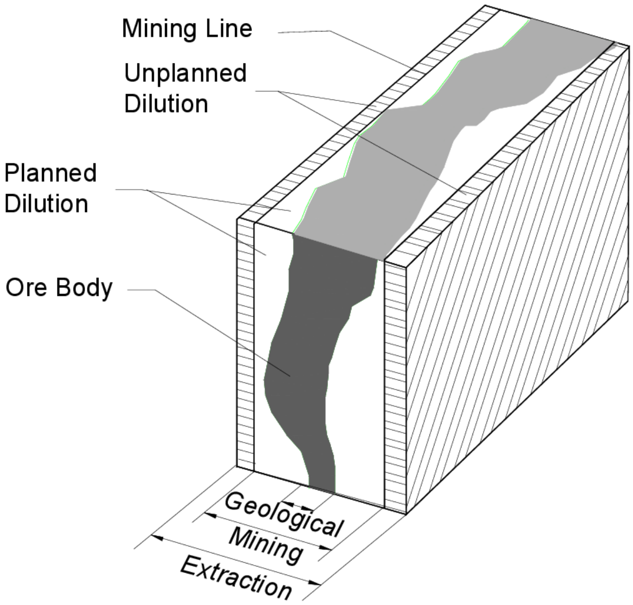

Figure 1.4 Types of dilution in the design of a stope (Scoble & Moss, 1994).

The estimation of unplanned dilution allows a design of stopes with a higher level of reliability in relation to their stability. In addition, this allows to define the extra costs associated for the mining method implemented. While there are some different definitions for the concept of dilution, the focus of this study is the unplanned dilution for the backs of open stopes related to major geological faults.

It should be taken into account that the concept of dilution is mainly associated with mine planning. Whereas in geotechnics the preferable concept is overbreak, which does not distinguish between ore and waste.

There are two main mechanisms to estimate and represent the overbreak (dilution): The cavity monitoring system (CMS) and the equivalent linear overbreak/Slough (ELOS).

- **Cavity Monitoring System (CMS)**

The cavity monitoring system (CMS) is one of the main tools used for the representation and measurement of overbreak (dilution) in an underground excavation. Basically, it is composed by a scanning laser capable of delineates remotely the shape of an underground cavity through three-dimensional profiles. In most cases, CMS is the only medium that determines the extent and volume of an otherwise inaccessible underground cavity. (Jarosz & Shepherd,2002).

The information obtained by the CMS is then converted into a three-dimensional mesh for analysis by design programs such as AutoCAD. The measuring instrument consists of three main components: a laser scanner unit, a laptop, and a support package. The CMS is used to monitor cavity stability and often to evaluate stope performance for dilution determination. The technology was designed by Noranda Technology Center and Optech Systems to delineate the size and shape of underground excavations.

- **Equivalent Linear Overbreak / Slough (ELOS)**

The term Equivalent Linear Overbreak / Slough, *ELOS* (Clark & Pakalnis, 1997; Dunne & Pakalnis, 1996), is an indirect quantitative measurement of the dilution (overbreak) produced by instability within a stope (Papaioanou & Suorineni, 2016).

Fundamentally, it extends (standardizes) the total volume of overbreak of a wall throughout the entire extension of its area. The basis for the *ELOS* stability graph has been developed from narrow vein mine.

Physically the term *ELOS* represents an average depth of overbreak for a wall, and has the advantage of providing information independently of the stope's size, as opposed to the percentage overbreak term. When dilution or overbreak is expressed in percentage terms (%), the stope width generates large distortions in the analysis given its weight in the total volume of the excavation. This distortion is apreciable in the comparison between narrow veins' stopes and stopes of considerable width (Castro, 2015).

1.3.3 Influence of major faults on the stability of open stopes

Major faults have been cited as the main cause of overbreak in underground excavations in the last 30 years (Dunne & Pakalnis, 1996; Mikula, 2020; Suorineni, 1998, 1999, 2001). This overbreak impacts not only the costs of the process (mine planning) but also the reliability on the stability graph method.

The factors that influence the overbreak due to the presence of major faults include (Suorineni, 1998):

- The geometric relationship between a major fault and the wall of an open stope (unfavorable location and orientation of a fault with respect to the excavation; the size, shape and orientation of the excavation).
- The properties of the major faults (low cohesion and friction).
- In-situ stresses (relationship and value among stresses).

To manage the influence of major faults on open stopes, Suorineni (2001) proposed the fault factor F_w as an adjustment of the stability number (N) when a wall of an open stope is intersected by a major fault. This factor is considered in the calculation of the modified stability number (N'_f) as shown in equation 2.

$$N_f' = Q' \cdot A \cdot B \cdot C \cdot F_w \quad (2)$$

The fault factor F_w is directly related with the amount of overbreak due to the presence of a major fault and it is defined by two values of $ELOS$. The equation 3, illustrates the definition of the fault factor F_w.

$$F_w = \frac{\mathrm{N}_\xi'}{\mathrm{N}_0'} \quad (3)$$

Where:

- N'_{ξ} is the stability number corresponding to the $ELOS$ value influenced by the major fault ($ELOS_f$).
- N'_0 is the stability number corresponding to the $ELOS$ value of 0.5m.

The surfaces of an open stope with ELOS <0.5 are considered stable and then the ELOS value of 0.5m is a reference point when the fault factor is calculated. The procedure to obtain the fault factor F_w is described below:

- Determine the hidraulic radius (HR) of the studied surface (wall of an open stope).
- Determine N'_0 as the intersection of the HR with the stability curve of ELOS=0.5m (Figure 1.5).
- Considering the same HR, determine N'_{ξ} as the intersection of the HR value with the stability curve of ELOS value influenced by the major fault ($ELOS_f$). In this stage the use of interpolations maybe necessary (Figure 1.5).
- The fault factor F_w is the ralation between N'_{ξ} and N'_0 (equation 3).

The factor determines the effect of faults on the walls of the stope, considering values from 0.01 (maximum potential impact) to 1 (no adverse effect). Thus, the application of this factor achieves a significant decrease in the value of N up to two orders of magnitude depending on the potential influence that the faults may have on the stability of the walls (Azorín, 2018).

Although it is highly applicable to walls, it shows a restriction by not considering the presence of major geological faults directly related to the stability of backs in open stopes. In this sense, these analyses can be made using numerical modelling.

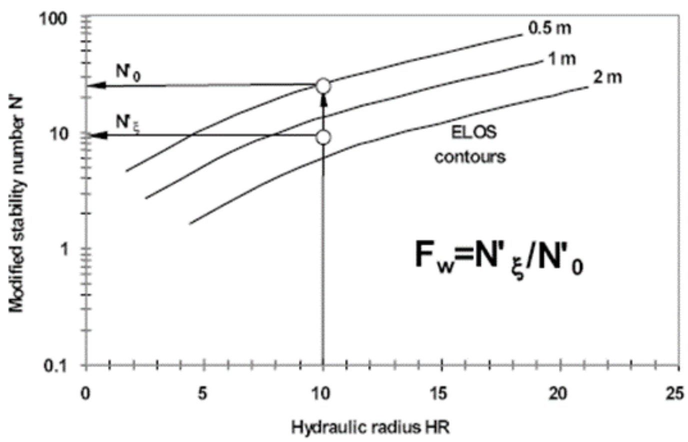

Figure 1.5 Determination of F_w through the ELOS stability graph *(Suorineni, 2001).*

More literature on sublevel stoping mining and the stability graph method for open stopes is available in chapter 2.

1.3.4 Numerical Modelling

Numerical modelling (methods) is used in geotechnics (rock mechanics) because it allows to find new aspects, comprehensions and solutions of complex problems, and also it optimizes empirical methods. These features improve the engineering decisions used in the design stages.

Among the main characteristics of numerical modelling:

- Applicability to any geometry
- Does not assume mechanisms / failure surfaces
- It is necessary to specify input data such as: geometry, rock mass parameters, initial and boundary conditions, stresses, constraints, mesh density - size of elements / zones, constitutive model.

There are a great variety of numerical modelling softwares based on various numerical methods, but they all start from the premise that no single software can capture all the characteristics of the behaviour of an actual system. Therefore, the most relevant aspects of the problem (actual system) and the appropriate software must be determined prior to numerical modelling.

Additionally, it is important to be clear about what to expect from each numerical method in order to better interpret the results and select a helpful tool to adress the problem. Even in relatively simple cases, numerical modelling only complements the expertise of the user (Bobet, 2010; Carter et al., 2000; Jing & Hudson, 2002).

Figure 1.6 indicates a comparison among different types of mehtods in numerical modelling according to the excavation scale.

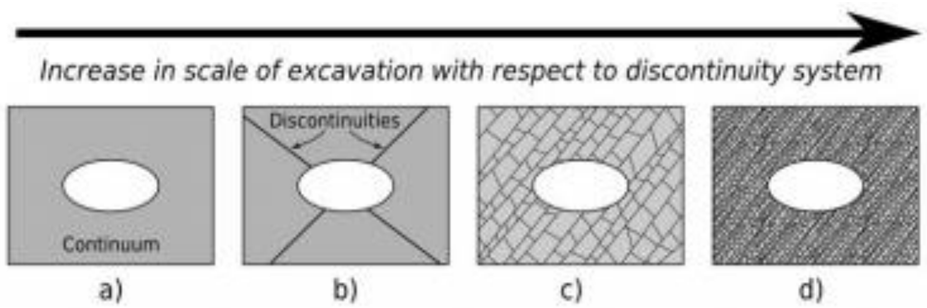

Figure 1.6 Methods in numerical modelling with respect to excavation scale: a) continuos ; b) discontinuous with discontinuity elements or discrete method; c) discrete method; d) equivalent continuous method. Modified from Jing (2003).

1.3.5 Continuos Methods in Numerical Modelling

Some of the oldest and most widely used numerical methods assume that materials behave as a continuum element. This assumption results in the restriction that the materials cannot be divided. This means that all material points remain in the same neighborhood of points throughout the deformation process. The most common types of continuous methods of modelling are: boundary element method (BEM), finite element method (FEM), and finite difference method (FDM).

- **Boundary Element Method (BEM)**

BEM is a numerical method for modelling used generally for stability analysis where the medium to be modeled is defined as a continuum. In this case only the boundaries of the excavation and the external boundaries of the model are discretized to obtain responses, such as the stress and displacement distribution. Due to the simplification of the problem its response is fast and highly efficient. It is ideal for simple constitutive models, and specifically in rock mechanics it assumes the type of rock CHILE (continuous, homogeneous, isotropic, linear and elastic). Therefore, the stress calculation is independent of the elastic parameters of the rock.

- **Finite Element Method (FEM)**

Finite element method is perhaps the most applied numerical method in engineering. Since its introduction in the early 1960s, much of FEM's development work has been specifically geared toward rock mechanics problems (Jing & Hudson, 2002). This is because it was the first numerical method with sufficient flexibility to deal with material heterogeneity, nonlinear deformability (mainly plasticity), complex boundary conditions, in situ stresses, and gravity. The medium is defined as a continuum and unlike the BEM method, the finite element method requires the discretization of both the boundaries and the medium. This discretization considers the division of the medium generating a grid (mesh) in which different areas coexist joined together by nodes (vertices of polygons), thus generating the interaction of the complete domain.

- **Finite Difference Method (FDM).**

The finite difference method was the first numerical method used to approximate the solutions to complex partial differential equations (Jing, 2003). The medium is defined as a continuum, however for the representation of the medium the discretization of the complete domain to be modeled is required. Then the creation of a mesh in which there are areas (polygons / polyhedrons) joined together by nodes at their vertices is necessary. These nodes are in charge of representing the interaction among all the zones, transmitting the interaction forces of the problem.

Unlike the FEM method, the finite difference method provides a solution through the balance of forces and nodal displacements. This makes an explicit node-by-node calculation of the equations of motion present in the problem and not inverts a global matrix. In addition, by incorporating a damping rate to absorb energy an adequate convergence rate of the solution is guaranteed, which will depend on the unbalanced forces present in the calculation of the problem. (Castro, 2015).

1.3.6 Discontinuous methods in numerical modelling

A discontinuous medium is distinguished from a continuous one by the existence of contacts or interfaces among the discrete elements that compose the system. Discontinuous methods can be classified both by the way they represent contacts and by the way they represent discrete elements (bodies) in the numerical formulation (Jing & Hudson, 2002).

Discontinuous modelling techniques treat the material directly as a collection of separate blocks or particles. According to the original definition proposed by Cundall & Hart (1992), discontinuous methods refer to any modelling technique that allows finite displacements and rotations of discrete elements (including complete spalling), and automatically recognizes new contacts as the simulation progresses. Based on the adopted solution algorithm, DEM implementations are broadly divided into time-explicit and implicit methods.
In these discontinuous models, the use of discrete fracture networks **(DFN)** generally complements the analysis through realistic representations of rock mass fracturing.

- **Distinct element method (DEM)[3].**

The distinct element method (DEM) is a numerical solution used to describe the mechanical behavior of discontinuous bodies. Introduced by Cundall & Strack (1979), the DEM was developed for the analysis of rock mechanics problems using the scheme of discrete elements in polygonal deformable particular blocks. DEM uses an explicit solution in the time domain of the original equations (not the transformed modal equations).
Many finite element, boundary element, and Lagrangian finite difference programs have interface elements or "slip lines" that allow you to model a discontinuous material to some extent. However, its formulation is generally restricted in one or more of the following ways:

I. The logic may break down when many intersecting interfaces are used
II. There may not be an automatic scheme for recognizing new contacts
III. The formulation may be limited to small displacements and/or rotation (such programs usually are adapted from existing continuum programs)

The choice of continuous or discontinuous methods depends on many specific factors, mainly the scale of the problem and the geometry of the fracture system. The continuous approach can be used if there are only a few fractures and if the opening of the fracture and the detachment of entire blocks are not significant factors. The discrete approach is best suited for moderately fractured rock masses where the number of fractures is too large for the fractured continuous element approach, or where large-scale displacements of individual blocks are possible. There are no absolute advantages of one method over another.

For this thesis, the distinct element method (DEM) was the selected since it is the ideal to represent faults on a larger scale as the case studies illustrated in this research. In this case, the election of the software ***3DEC v5.2***. (Itasca, 2016) represents a three-dimensional numerical tool for advanced geomechanical and geotechnical analysis of rock, soil, groundwater, structural support, and masonry. The software simulates the response of discontinuous media that are subject to static or dynamic loading. Discontinuous material is represented as a set of discrete blocks. Additionally, it contains Itasca's integrated programming language called ***FISH***, with which it is possible to write own scripts for users who wish to add functionality for personalized analysis.

[3] https://www.itasca.cl/en/software/distinct-element-method

Table 1.2 illustrates a comparison among commercial softwares available with continuous and discontinuous methods.

Table 1.2 Comparison of commercially available continuous and discontinuous software.

Continuous Methods		
Software	***Formulation***	***Dimension***
FLAC	Finite Difference Method (FDM)	Two
FLAC3D		Three
PLAXIS	Finite Element Method (FEM)	Two and three
ADINA		
ABAQUS		
RS2		Two
MAP3D	Boundary Element Method (BEM)	Three
EXAMINE3D		
Discontinuous Methods		
UDEC	Distinct Element Method (DEM)	Two
3DEC		Three

1.4 Objetives of the Thesis

In this section, the objectives related to the work to be carried out through this research are presented.

1.4.1 General Objective

To define a methodology to evaluate the influence of major geological faults on the stability of backs of open stopes using 3D numerical modelling.

1.4.2 Specific objectives

- To define case studies from Chilean Sublevel Stoping mining associated with overbreak possibly influenced by major geological faults.

- To determine and characterize major geological faults associated with each of the proposed case studies.

- To verify the calibrated criteria through numerical modelling, replicating the actual cavity (*CMS*) using major geological faults and geotechnical information of the case studies.

- To introduce a fault index (F_b) in the stability graph method that describes the influence of major geological structures on the stability of backs of open stopes.

1.5 Scope of the research

The work developed through this thesis is focused on the evaluation and influence of major geological faults on the stability of backs in open stopes through discontinuous numerical modelling.

- The numerical models defined correspond to three representative case studies of a Chilean Sublevel Open Stoping mine. As the mine and its information is classified, the names and some references related to the mine used in this thesis will be ficticious (not the database).

- To simplify the data analysis and due to the lack of information regarding residual parameters, the numerical models will be linear - elastic in the distinct elements software *3DEC v5.2*.

1.6 General methodology

The research is divided into three structured stages, which seek to fully meet each of the proposed objectives in an optimal and systematic way.

- ***Literature Review:*** This stage includes the state of the art of empirical stability methodologies; open stope design; fundamentals and analysis of numerical modelling in rock mechanics; overbreak criteria and description of geological faults.

- ***Numerical modelling in 3DEC***: This stage considers the three steps summarized in Figure 1.7.

- ***Summary and conclusions:*** In this stage is expected that the numerical modelling analysis allows to establish a representative criterion of overbreak for the three case studies. This criterion will be able to explain the impact of the major geological faults on backs of open stopes in the Chilean mining of Sublevel Stoping. Finally, a summary and discussion of the results is carried out with the aim of contributing new findings through this research.

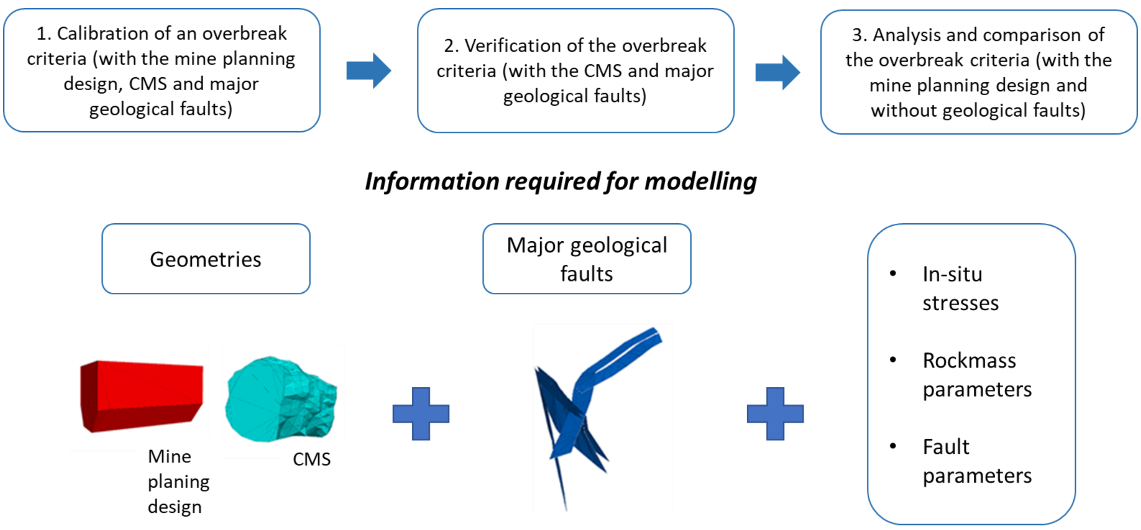

Figure 1.7 Steps considered in the numerical modelling.

1.7 Thesis contents

This thesis is based on papers and is structured in three chapters as shown below:

Chapter 1: The reader is introduced to the subject and the problems to be developed through the thesis. This chapter shows a summary of the state of the art in addition to the proposed objectives and general research methodology.

Chapter 2: This chapter presents an article which establishes a fault index (F_b) that describes the influence of major geological structures on the stability of backs in open stopes. The title of the article is "*A methodology to evaluate the impact of major geological faults on the stability of backs in open stopes using DEM*" and is in preparation for the International Journal of Mining, Reclamation and Environment.

Chapter 3: It describes the most relevant findings of the research, as well as recommendations for the implementation of the methodology and analysis of results for future work related to the subject.

Chapter 2

A methodology to evaluate the impact of major geological faults on the stability of backs in open stopes using DEM

Abstract

Since the introduction of the original Stability Graph Method by Mathews et al. in 1981, its use has been widely extended in underground mining of Sublevel Open Stoping (SLS). However, the methodology is restrictive because objectively represents an applicable tool only for similar conditions to the case histories (data) where it was calibrated (site-specific conditions). In addition, and not less important, the stability graph methodology does not contemplate in its analysis the major geological structures but assesses the incidence of minor discontinuities, leaving aside one of the main reasons associated to the unplanned dilution due to overbreak, the faults. Therefore, is important to determine the influence of major geological faults in the design of open stopes and their direct impact in the information for mining design and planning. The main objective of this paper is to calibrate and verify a criterion that allows an evaluation of the impact of major geological faults in the stability on backs of open stopes, through the numerical modelling and analysis of overbreak of three case studies. To numerically model the response of the rock mass and underground excavations, a linear-elastic constitutive model using the distinct element method (DEM, software *3DEC v5.2*) was built. The numerical methodology is divided in three specific stages for each selected case, resulting in nine models in total. Then, based on the results of the numerical modelling, the first and third stage were compared, analyzing if there was an impact caused by the major geological faults in the overbreak on backs of the open stopes. The results show that $\sigma_3 \leq 1Mpa$ remains the criterion that best recreates the overbreak in the cases. Based on the criterion and numerical models, a fault factor for the backs of the open stopes was introduced to improve the empirical stability result of the case studies.

Keywords: *Sublevel Stoping, Faults, Overbreak, Numerical Modelling, Distinct Element Method*.

2.1 Introduction

The stability of an excavation depends on several factors, mainly including: stress conditions, rock mass properties, geometry and geological characteristics. In Chile, at medium level mining the most used underground mining method is the Sublevel Open Stoping (SLS) (Zablocki, 2013). Sublevel Stoping is a selective, self-supported, mechanized and with high levels of productivity method, used in regular ore deposits of large thickness and tabular to massive habit with vertical to sub-vertical dip. In this context, for open stope design, the stability graph method developed by Mathews et al. (1981) and later modified mainly by Potvin (1988), Nickson (1992), Clark and Pakalnis (1997), Suorineni (1998); Suorineni et al. (2001, 1999), Trueman et al. (2000) and Mawdesley (2002) is widely used.

The graphical stability method relates the hydraulic radius (HR) and the stability number (N) as illustrated in equation (1), (2) and (3). The adjustment factors of the Mathews' methodology are shown in the Figure 2.1.

$$N = Q' \times A \times B \times C \tag{1}$$

$$HR = \frac{Area\ of\ the\ wall\ (m^2)}{Perimeter\ of\ the\ wall\ (m)} \tag{2}$$

$$Q' = (RQD\ /\ Jn)\ (Jr\ /\ Ja) \tag{3}$$

Where:

- $\boldsymbol{Q'}$ is the modified tunnelling quality index (Barton et al., 1974).
- $\boldsymbol{A}$ is the Mathews' rock stress factor obtained from Figure 2.1 (a).
- $\boldsymbol{B}$ is the Mahtews' joint orientation adjustment obtained from Figure 2.1 (b).
- $\boldsymbol{C}$ is the Mathews' gravity adjusment factor obtained from Figure 2.1 (c).
- ***RQD*** is the rock quality designation (Deere et al., 1969).
- ***Jn*** is the joint set number (Barton et al., 1974).
- ***Jr*** is the joint roughness number (Barton et al., 1974).
- ***Ja*** the joint alteration number (Barton et al., 1974).
- ***Area*** is refered to the wall surface of the open stope studied (the hanging wall, footwall or back of the stope).
- ***Perimeter*** is refered to the distance around the ***Area*** of the wall surface of the open stope studied.

Since this empirical methodology should only be implemented in similar conditions (site-specific conditions) to those it was calibrated, several authors (Bewick and Kaiser, 2009; Nickson, 1992; Papaioanou and Suorineni, 2016; Stewart and Trueman, 2004, 2001; Vallejos et al., 2018, 2016; Vallejos and Díaz, 2020; among others) have made critical reviews and modifications in order to achieve greater representativeness. This has been reflected in a significant increase in the database, from 55 initial (Mathews et al., 1981) cases to more than 400 (Mawdesley et al., 2001). However, note that it is recommended to use the stability curves from literature when there is not enough information for the analysis (site-specific case histories) in the early stages of engineering (new mines).

2.2 Influence of faults on open stopes stability

An important limitation found in the classical empirical methodology is that the role of major geological faults is not considered in the analysis. The first advances in relation to the subject were provided by Suorineni (1998, 1999, 2001), who introduced an additional adjustment factor (F_w) which determines the effects of geological faults on walls of open stopes. The factor introduced by Suorineni et al. (2001) is located between values of 0.01 (maximum potential impact) and 1 (no adverse effect). Thus, achieving a reduction in the value of N up to two orders of magnitude, accounting for the potential influence that a geological fault may have in the stability graph (Azorin et al., 2018). Although this factor is of great applicability on walls of open stopes, it is not possible to consider the presence of major geological faults directly related to the stability on the backs of open stopes.

In the last years, several authors (Bruneau et al., 2003; Hao and Azzam, 2005; Liu et al., 2016; Schubert et al., 2006; H. Wang et al., 2016; Zhang et al., 2017; Zhong et al., 2020; among others) have been studying the influence of geological faults on underground excavations using in-situ analyses and numerical modelling. However, the quantification of the impact of these faults generally is determined for the site-specific conditions and specific case studies (mainly tunnels). Therefore, an analysis of the influence of geological faults focused on the backs of open stopes is necessary, in order to improve the classical empirical tools widely used in Sublevel Open Stoping mining.

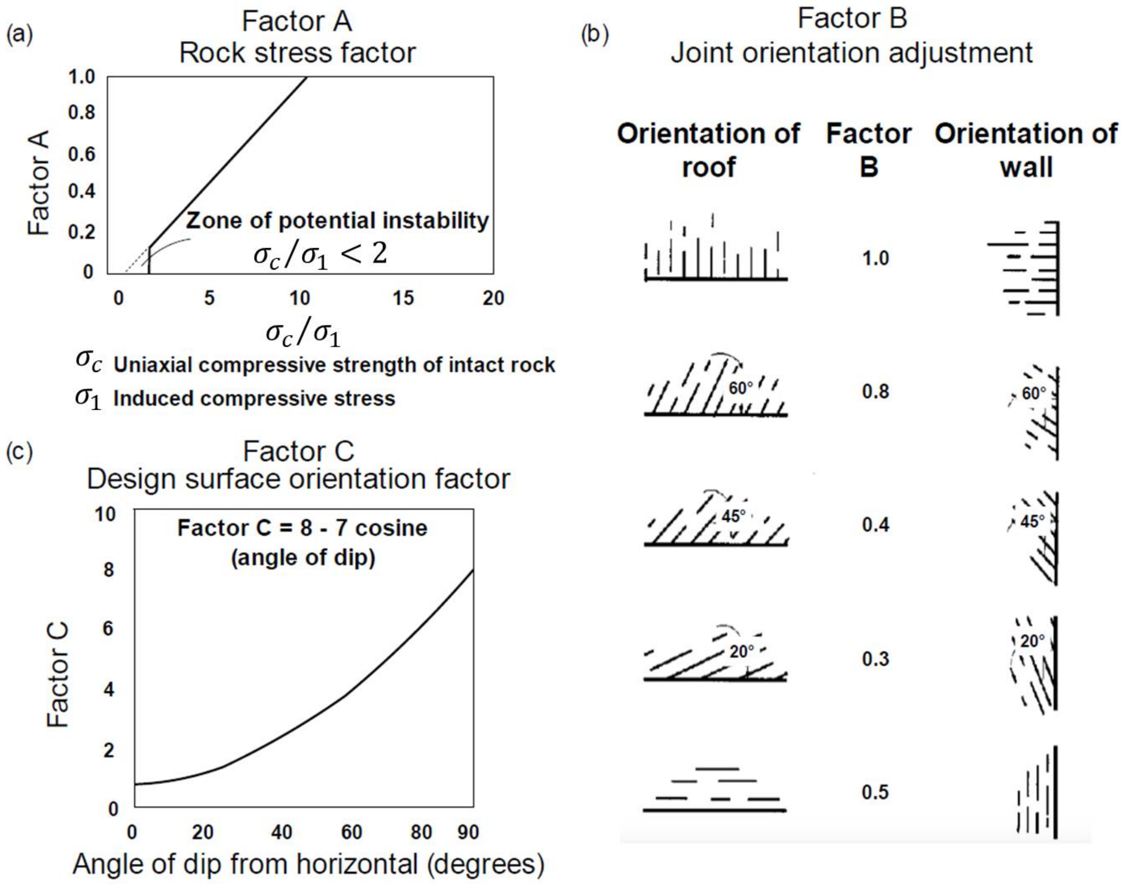

Figure 2.1 Case studies (open Mathews' adjustment factors; (a) Rock Stress Factor A; (b) Joint Orientation Factor B; (c) Design Surface Gravity Factor C.

2.3 Methodology

2.3.1 Data collection

From the compilation of 45 case histories from a Chilean Sublevel Stoping mine, it was possible to have a complete database in which representative cases were taken in order to illustrate the problem previously described.

For this research, three case studies of open stopes with overbreak on their backs were defined (obtained from their CMS, Cavity Monitoring System). The common characteristic among them is the presence of major geological faults that could be associated with the overbreak on their backs.

Then, it is necessary as an input parameter the selection of faults that are intimately related to the overbreak volume produced case by case, categorizing the faults between major

faults (1 to 10 cm of thickness) and main faults (> 10 cm of thickness). Note that most of the faults are interpreted and come from the geological and geotechnical mapping of the mine.
Figure 2.2 illustrates the studied open stopes (case studies), the definition of their walls, and one of the structures that intercept them (triangulated fault). The complete detail of all geological structures is shown in Figure 2.4.

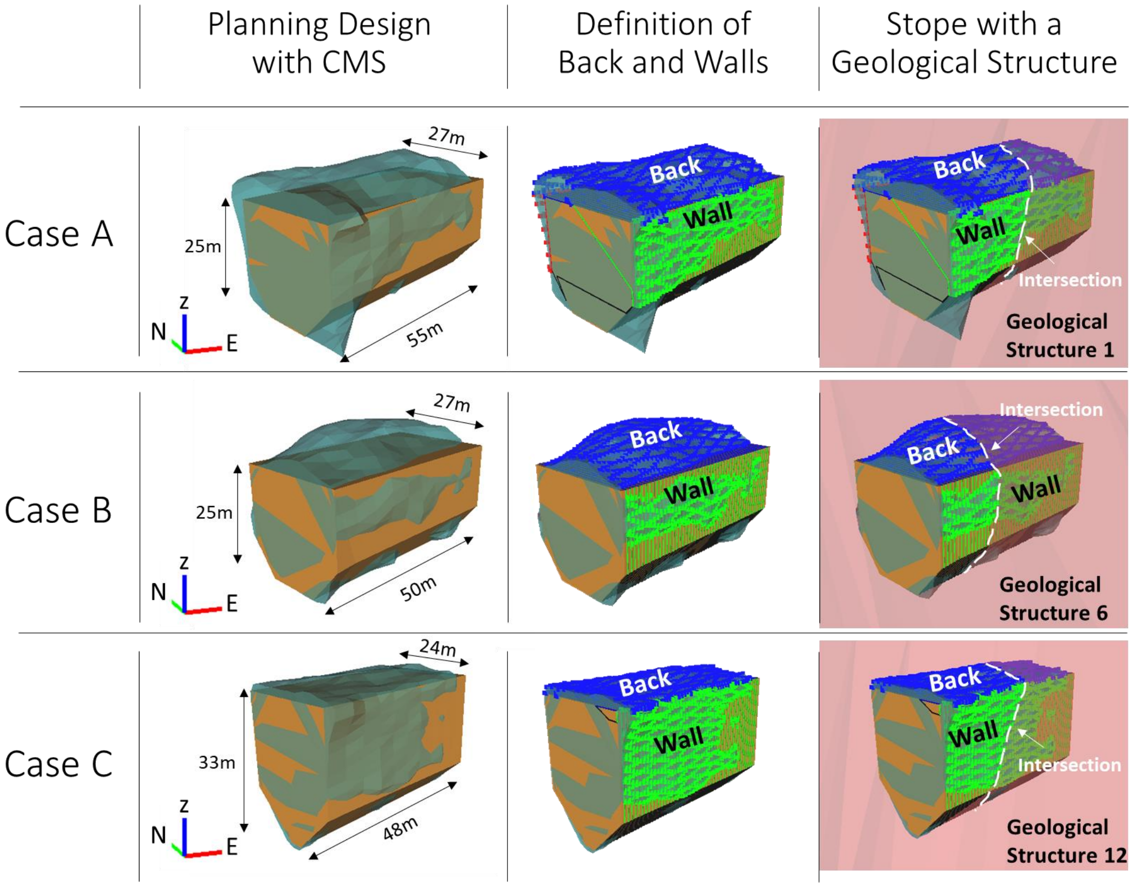

Figure 2.2 Case studies (open stopes) with the definition of their walls and one of the geological structures that intercept them.

2.3.2 Numerical Methodology

The numerical modelling was performed through the distinct elements method (DEM) in a discontinuous medium. Thus, the software 3DEC v5.2 (3-Dimensional Distinct Element Code, Itasca 2016) was used to model and solve the analysed case studies.
DEM is one of the types of discrete elements that separates the medium into a discontinuous one, presenting an explicit and time-dependent solution scheme to directly solve all the governing equations of the model. In the last years, DEM has been widely used in mining to model the behaviour of geological structures and their interaction with the continous medium (among structures) and excavations.
In Sublevel Open Stoping mining the DEM is mainly used to study the overbreak (Huang et al., 2017; Karim et al., 2013; Lavoie et al., 2019; Sainsbury et al., 2015; Turner et al., 2020; Urli and Esmaeili, 2017; Wang et al., 2016; among others), whereas in Caving mining the DEM is mainly used to study the caving propagation, subsidence or rockburst phenomenon (Cumming-Potvin et al., 2018; Pérez-Rey et al., 2017; Suzuki Morales and Suorineni, 2017; Wang et al., 2020; among others). Another underground mining

applications of DEM include analysis of intact rock (testing); ground support; pillar strength; blasting; debris flow; among others (Bahrani and Hadjigeorgiou, 2017; Bouzeran et al., 2017; Karampinos et al., 2018; Vagnon et al., 2017; Walls and Mpunzi, 2017; among others).

Therefore, for the proposed overbreak analysis using DEM a procedure was implemented to create the models. In general terms, the analysis is subdivided into three stages, distributed as shown in Figure 2.3.

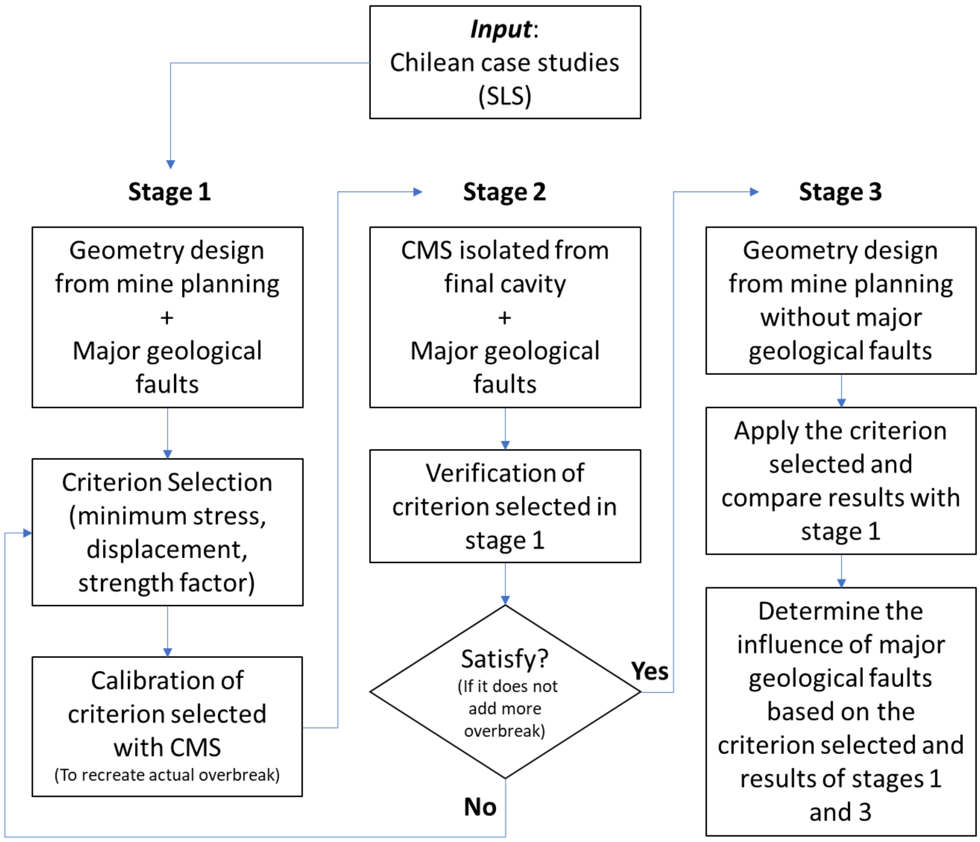

Figure 2.3 Flowchart of the numerical methodology.

2.3.3 Constitutive Model

A linear elastic constitutive model was defined to the analysis of the problem. Therefore, the incorporation of the deformation modulus (E), the compressibility modulus (K) and the shear modulus (G) in the medium modeled in 3DEC were considered.

2.3.4 Rock Mass Features

To simulate the rock mass, a medium of dimensions established in relation to the requirements of each case study was designed. In Figure 2.4, the geometry for the medium and the detail of all geological structures considered for the case studies are illustrated.

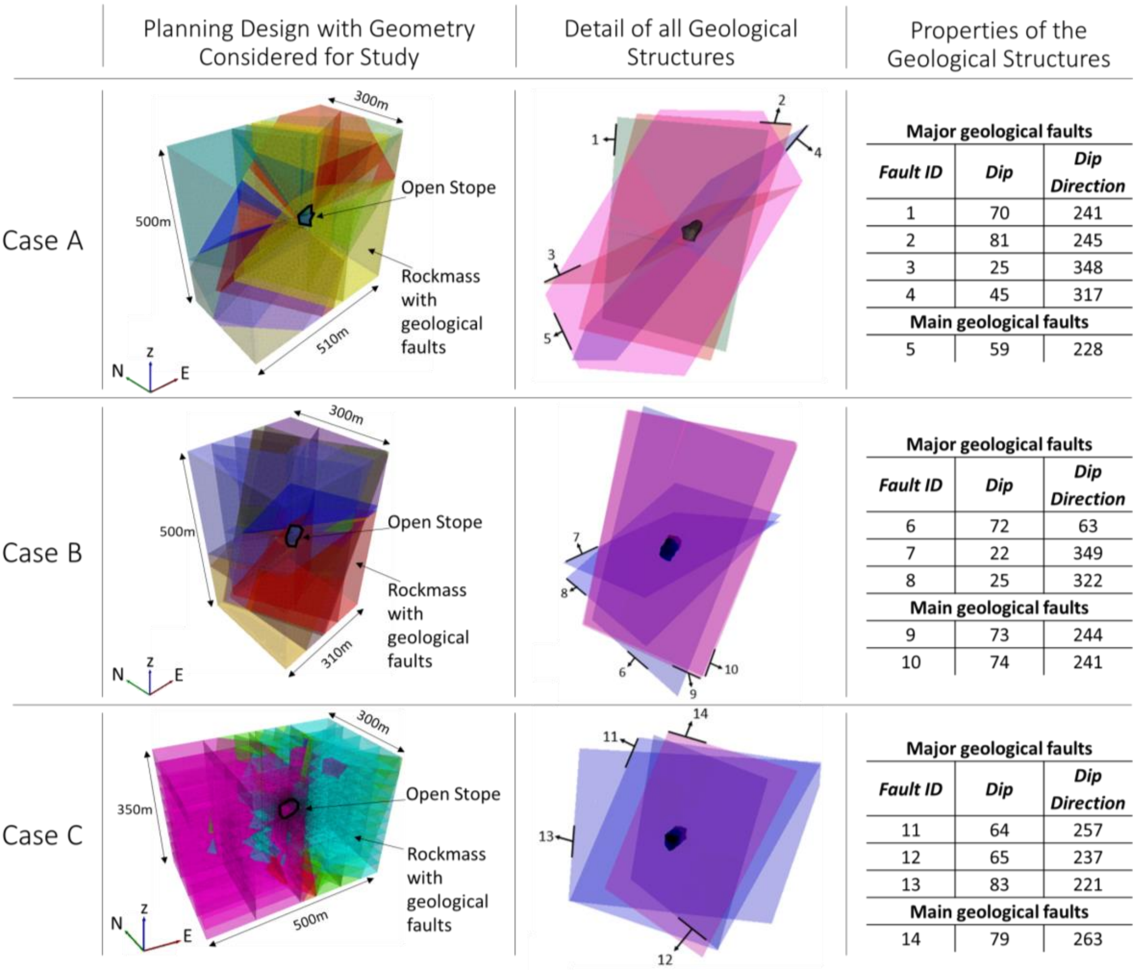

Figure 2.4 Models built in 3DEC with their dimensions and detail of all geological structures analyzed.

Additionally, in Table 2.1, the intact rock parameters of the study area are indicated, which are incorporated as input parameters for the models.

Table 2.1 Intact rock parameters

Rock density [ton/m^3]	Young's Modulus [GPa]	Poisson's ratio
2.7	93.9	0.2

In order to scale the parameters from intact rock to the rock mass, the Generalized Hoek-Brown Criterion is used (Hoek et al., 2002) as illustrated in equation (4).

$$\sigma_1{}' = \sigma_3{}' + \sigma_{ci}\,(m_b \frac{\sigma_3'}{\sigma_{ci}} + s)^a \tag{4}$$

Where:

- $\boldsymbol{\sigma_1{}'}$ is the major effective stress at failure condition.
- $\boldsymbol{\sigma_3{}'}$ is the minor effective stress at failure condition.
- $\boldsymbol{m_b}$ is a reduced value of the material constant m_i for intact rock (equation (5)).
- $\boldsymbol{s}$ and $\boldsymbol{a}$ are material constants that depend on the characteristics of the rock mass (equation (6) and (7), respectively).

- $\boldsymbol{\sigma_{ci}}$ is the uniaxial compressive strength of the intact rock.
- ***GSI*** is a Geological Strenght Index (Hoek, 1994; Hoek et al., 1995) used to estimate the reduction in strenght of the rock mass for different geological conditions.
- ***D*** is a factor which depends upon the degree of disturbance to which the rock mass has been subjected by blast damage and stress relaxation. It varies from 0 for undisturbed in situ rock masses to 1 for very disturbed rock masses (Hoek et al., 2002).

$$m_b = m_i \exp\left(\frac{GSI-100}{28-14D}\right) \tag{5}$$

$$s = \exp\left(\frac{GSI-100}{9-3D}\right) \tag{6}$$

$$a = \frac{1}{2} + \frac{1}{6}\left(e^{-GSI/15} - e^{-20/3}\right) \tag{7}$$

Thereby, based on the site-specific information from the mine, the parameters indicated in Table 2.2 were those used for this research. Note that the disturbance factor D was selected as 0.2 due to operational conditions at the mine. This parameter affects all the simulated rock mass.

Table 2.2 Input $\boldsymbol{\sigma_{ci}}$, GSI, mi and D to calculate the Generalized Hoek-Brown failure envelope parameters mb, s and a

$\boldsymbol{\sigma_{ci}}$ (UCS) [Mpa]	GSI	mi	D
211	70	15	0.2

In addition to the model, the scaled rock mass elastic modulus (Hoek and Diederichs, 2006) was incorporated (equation (8)). This was determined with the input data indicated previously.

$$E_{rm} = E_i \cdot \left(0.02 + \frac{1-\frac{D}{2}}{1+e^{\frac{60+GSI+15\cdot D}{11}}}\right) \tag{8}$$

Where, E_{rm} is the elastic rock mass modulus and E_i is the elastic intact rock modulus.

2.3.5 Faults Properties

Goodman et al. (1968) introduced the terms normal stiffness (K_n) and shear stiffness (K_S) to describe the rate of change of normal stress (σ_n) with respect to normal displacements (d_n) and shear stress (τ) with respect to cutting displacements (d_c), respectively. Equation (9) and equation (10) show the relationship among these variables.

$$K_n = \frac{\sigma_n}{d_n} \tag{9}$$

$$K_S = \frac{\tau}{d_c} \tag{10}$$

Karzulovic (2001) presents a guideline for the choice of K_n and K_S. According to his work, the K_n values can vary between 0.001 and 2000 Gpa/m, but in general it presents values lower than 10 Gpa/m in the case of structures with soft fillings. Values of 10 to 50

Gpa/m are admissible in the case of structures with host rock of fair quality; and values of 100 to 200 Gpa/m in the case of structures with host rock of competent quality. Whereas, the K_S values indicate that its secant in peak condition can vary between 0.01 and 50 Gpa/m, but in general it has values less than 1 Gpa/m in the case of structures with soft fillings. Values not greater than 10 Gpa/m in the case of structures with host rock of fair quality; and values up to 50 Gpa/m only in the case of structures with host rock of competent quality.

The guideline given by Karzulovic (2001) suggests a range that could be used in the numerical modelling of each case study.

2.3.6 In-situ stresses and boundary conditions

In order to initialize the model, it was necessary to determine the Cartesian stresses case by case, distributed by the stress tensor. To account the overburden model, a vertical downward stress (σ_z) is applied to the top surface of the simulated rock mass portion. Whereas lateral stresses (σ_{NS} and σ_{EW}) are applied to the lateral sides of the simulated rock mass portion. While the stresses are applied, the lateral sides and the bottom of the simulated rock mass model are fixed in place by assigning a boundary condition of zero displacement and zero velocity. Table 2.3 illustrates the in-situ stresses applied to the case studies.

Table 2.3 In-situ stresses (cartesian) used in the models

Cases	In-situ Stress [MPa]		
	$\boldsymbol{\sigma_{NS}}$	$\boldsymbol{\sigma_{EW}}$	$\boldsymbol{\sigma_z}$
A	18.2	12.5	10.8
B	19.2	13.2	11.4
C	19.4	13.4	11.6

2.4 Results and discussion

2.4.1 Parametric analysis of the normal and shear stiffness of geological faults

The lack of consensus on the values of geological fault properties available in the literature (Bandis et al., 1983; Goodman et al., 1968; Karzulovic, 2001) leads to uncertainty in determining a correct parameter of K_n and K_s that really represents the site-specific conditions. Then, the choice of their values is an important part of the calibration (stage 1).

Therefore, a parametric analysis was made to evaluate the maximum displacement on the stope back of the case studies A and B. In each case study, the K_n value was set to 1 GPa/m while several values of K_S were evaluated. Next, the same procedure was applied to the K_S value, setting its value to 1 and evaluating several K_n values. The results of the parametric analysis conducted are shown in Figure 2.5.

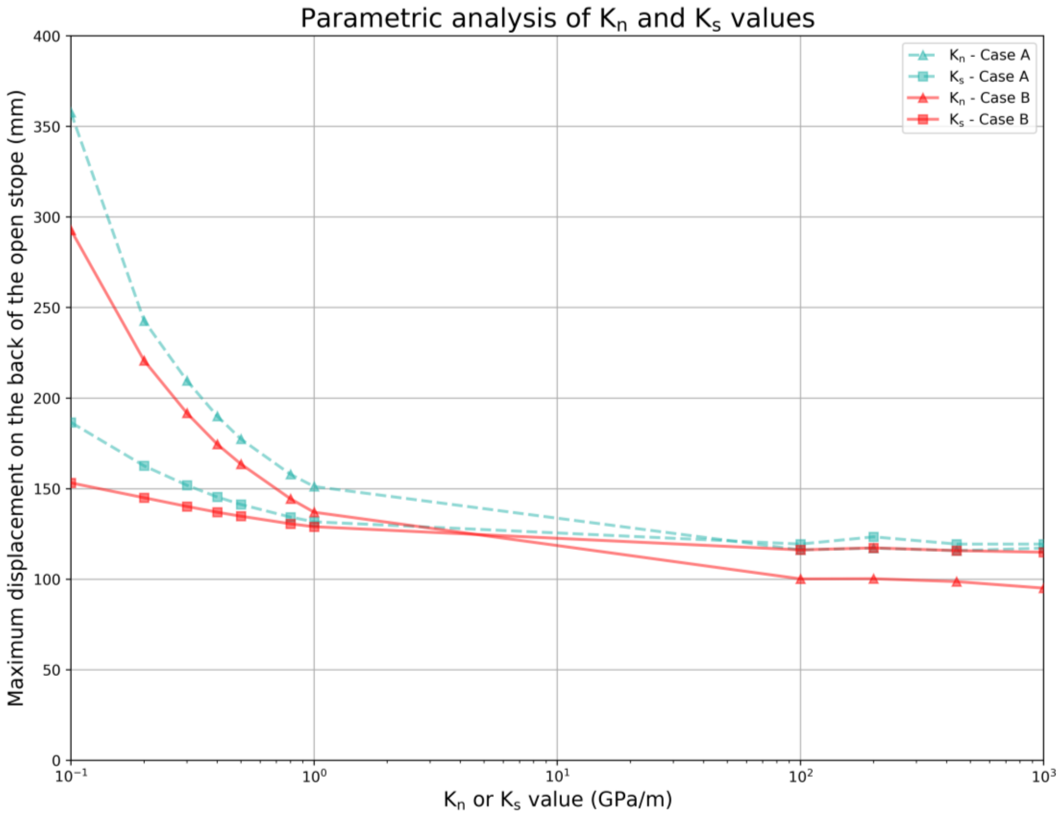

Figure 2.5 Maximum displacement on the back of the open stopes (Case studies A and B)

The trend of the graph illustrated in Figure 2.5 indicates that the maximum displacement on stope backs of case studies A and B decrease as the value of both K_n and K_S increase and shows a constant behavior when the values are greater than or equal to 100 Gpa/m. The maximum displacements varying the K_S value have a similar behaviour to the analysis of K_n. However, the K_S value has a minor impact in the displacement compared to the K_n value.

Therefore, the final values for the normal stiffness (K_n) and shear stiffness (K_s) chosen for the models were 1 GPa/m and 0.5 GPa/m, respectively. The choose of the values were based on the numerical results exposed previously and the guidelines stated by Karzulovic (2001).

Note that case study C was not considered in this parametric analysis (case of verification), then the results obtained from case study A and case study B were used for its modelling.

2.4.2 Numerical analysis of Case Studies

Applying the methodology indicated in section 2.3.2, nine models were made in total that join the three case studies shown in this research to assess the impact of major geological faults in the overbreak on backs of open stopes. The results suggest that the faults, using the minor principal stress (σ_3) as an index of stability, have a direct influence in the overbreak. These findings were made through a three-dimensional analysis in the software 3DEC v5.2.

The election of the minor principal stress (σ_3) as an index of stability was based on the guidelines stated by some authors (Alcott et al., 1999; Bewick and Kaiser, 2009; Cai et al., 2000; Henning and Mitri, 2007; Martin et al., 1999; among others), where σ_3 is used as an index of stability for underground excavations. Furthermore, this choice is related with the ease of access to in-situ and laboratory data.

In the first case study analyzed (Case study A), in each of the three stages proposed (mine planning design with major geological faults, CMS with major geological faults and mine planning design without major geological faults), it was determined that the resulting overbreak on the back of the open stopes is recreated under the criterion $\sigma_3 \leq 1MPa$ (See Figure 2.6).

Evaluating the first case study (Case study A), in the stage 1 (Figure 2.6a) can be seen that the zone colored (red to blue) corresponds to the area of damage on the back of the original design of the open stope from mine planning. Therefore, when the criterion $\sigma_3 \leq 1MPa$ is applied in stage 2 (Figure 2.6b), i.e., evaluating the mined cavity (CMS), it is possible to visualize how the area of damage on the back corresponds to the cavity, verifying that the criterion selected is calibrated and the excavation is now stable.

Finally, stage 3 (Figure 2.6c) contemplates an analysis of the minor principal stress (σ_3) on the original open stope design from mine planning without the presence of faults, giving as a result that there is a minor overbreak associated on the back. In this case, the elements on the back were not in tension. Thus, comparing stage 1 and stage 3, it is determined that through the established numerical analysis, the major geological faults have a direct influence in the overbreak on the back of the open stope.

In the second case study (Case study B), the results were very similar to the first case study (A), then the overbreak is also recreated using the criterion $\sigma_3 \leq 1MPa$ (See Figure 2.6d, e, f). As the previous case study (A), Figure 2.6f illustrates that the elements on the back of the open stope are not in tension, having a minor overbreak and indicating again that the presence of the major geological faults have an influence in the stability results.

To verify the selected criterion ($\sigma_3 \leq 1$ MPa) used in the previous case studies (A and B), a third case study (Case study C) was analyzed through the empirical (Mathews' methodology) and numerical methodology in relation to the two previous cases. Case study C empirically corresponded to a stable open stope (as illustrated later in Figure 2.7) considering the major geological faults. On the other hand, the results of the case study C through the numerical modelling (Figure 2.6g, h, i) show that the major geological faults are not directly related to the overbreak. These findings reaffirm that the excavation is stable, and the criterion selected works with the site-specific conditions analyzed.

Accordingly, through the analyses performed and illustrated previously, a simple criterion was established which allows evaluating the overbreak on the backs of open stopes based on the minor principal stress (σ_3). Thus, it was determined that $\sigma_3 \leq 1\ MPa$ is the most representative criterion for the three case studies.

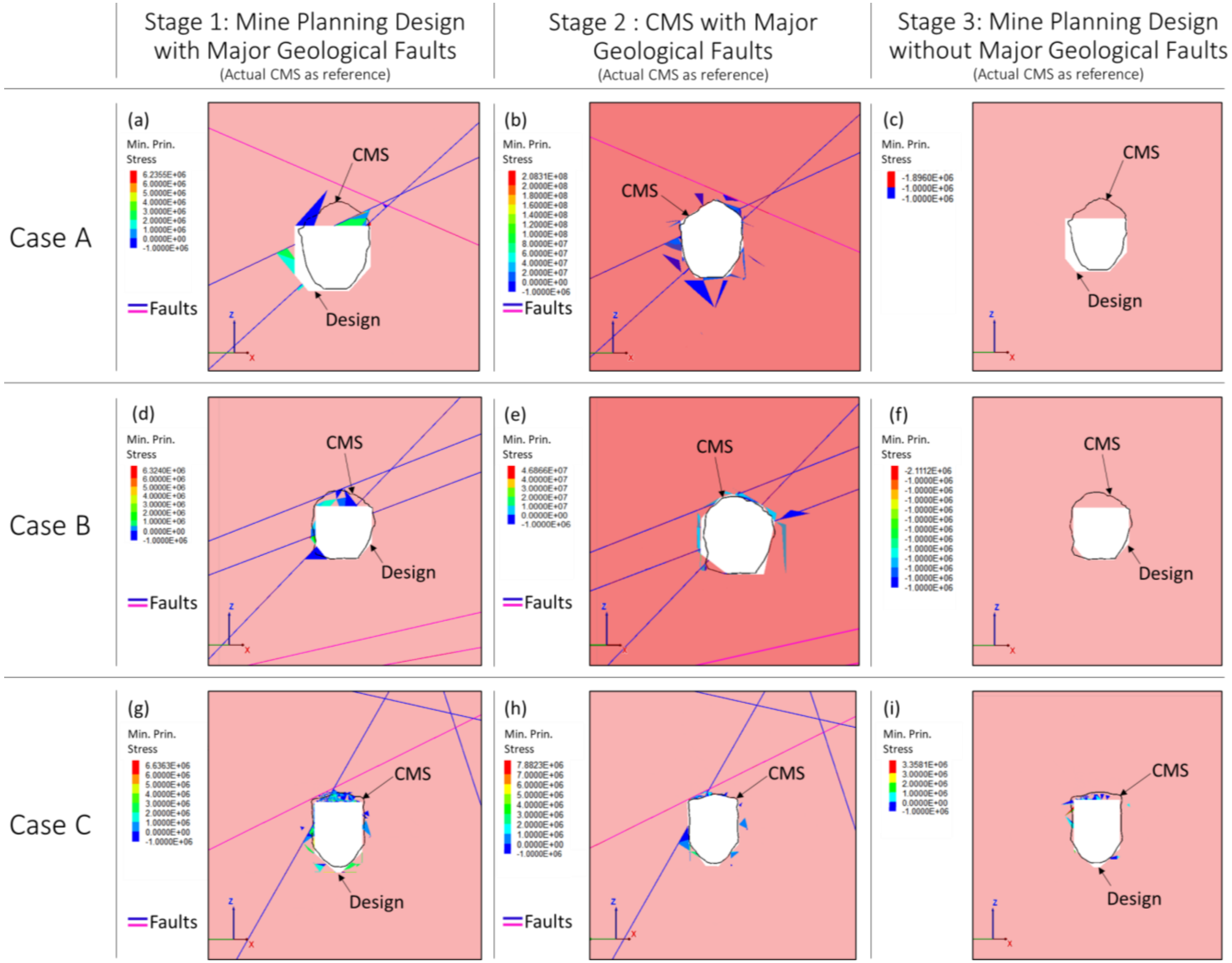

Figure 2.6 Cross section in the middle of case studies A, B and C showing the $\sigma_3 \leq 1MPa$ envelope, respectively. (a, d, g) Mine planning design with geological faults: calibration. (b, e, h) Verification of the criterion with the CMS. (c, f, i) Overbreak analysis of the mine planning design without geological faults. Note that negative values mean compression; whereas the pink and white areas are zones with $\sigma_3 > 1MPa$ and excavated, respectively.

2.4.3 Application of the numerical results to the classical Mathews' empirical model

To complete the analysis, and as an example of application, the criterion selected and developed through the numerical model was transformed into a fault index (F_b) to evaluate the stability of the backs of the case studies through the Mathews' stability chart. Then, the overbreak volume that envelopes the criterion $\sigma_3 \leq$ 1MPa on the backs of the case studies was considered, i.e. considering the mine planning design with geological faults (stage 1) and the mine planning design without faults (stage 3). Therefore, a fault index (F_b) was developed as illustrated in equation (11).

$$F_b = \frac{VOB^{d}_{\sigma_3 \leq 1MPa}}{VOB^{d-fault}_{\sigma_3 \leq 1MPa}} \quad (11)$$

Where:

- $\boldsymbol{VOB^{d}_{\sigma_3 \leq 1MPa}}$ is the volume of overbreak on the back that envelopes the criterion $\sigma_3 \leq$ 1MPa in the model that considers the mine planing design without geological faults (stage 3).
- $\boldsymbol{VOB^{d-fault}_{\sigma_3 \leq 1MPa}}$ is the volume of overbreak on the back that envelopes the criterion $\sigma_3 \leq$ 1MPa in the model that considers the mine planing design with geological faults (stage 1).

It should be noted that the fault index (F_b) is very similar to the introduced by Suorineni (1998) for the walls of open stopes, and also is similar to the treatment exposed by Bewick and Kaiser (2009) for the analysis of the Mathews' joint orientation factor (B).

Figure 2.7 shows the original empirical result of the backs for the three case studies compared with the site-specific stability curves of the mine (considering the original 45 case histories) at the time of the study (upper points). Note that the stability curves were built considering the ELOS (Clark and Pakalnis, 1997) as stability index.

Analysing the empirical result and comparing the actual ELOS value of the backs for the three case studies, it is clear that the stability numbers (N) of the case studies A and B are not well correlated (compared with the stability curves).

Therefore, including the fault index as a multiplier for the original stability number (N), a new stability number considering the effects of the geological faults on the back (N_{fb}) is developed, as shown in equation (12). Then, the original empirical result change to that illustrated also in Figure 2.7.
In this new empirical result (lower points), it is appreciated an improvement in the correlation between the mine stability curves and the actual ELOS values of the case studies.

$$N_{fb} = F_b \times N \quad (12)$$

Table 2.4 shows the actual ELOS values and the ELOS values obtained from the numerical models used for the development of the fault index (F_b).

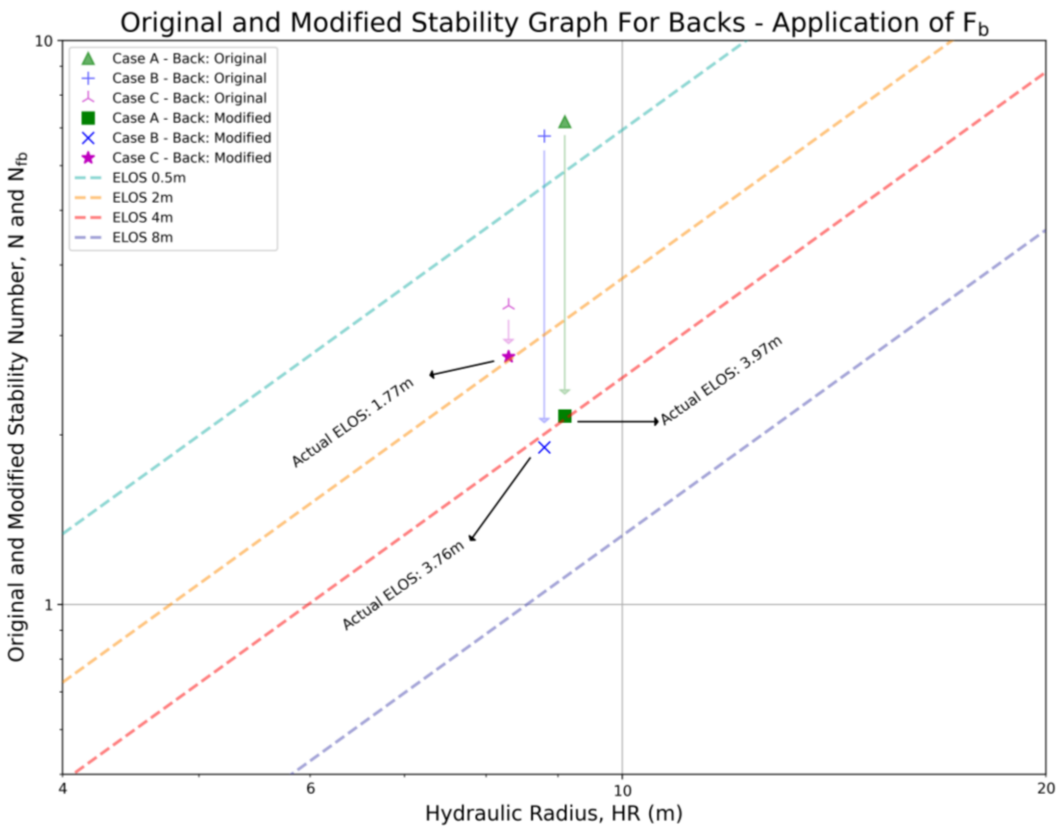

Figure 2.7 Original and modified stability graph compared with the site-specific stability curves for backs showing the three case studies.

Table 2.4 Summary of the ELOS values and the fault factor of the backs of each case study

Case studies	*Actual ELOS (m) from CMS*	*Numerical models*		*Fault factor of the backs (F_b)*
		ELOS (m) with geological faults (Stage 1)	*ELOS (m) without geological faults (Stage 3)*	
Case A	3.97	3.23	0.97	0.30
Case B	3.76	2.93	0.82	0.28
Case C	1.77	1.68	1.36	0.81

This result demonstrates how the methodology presented through the manuscript can be applied as a practical tool that improves the empirical stability analysis on open stopes not well correlated with the site-specific curves.

Finally, the next step for a mining application should be the introduction of the methodology for all open stopes empirically not well classified and redefine the site-specific stability curves. In addition, it is recommended to correlate the results of all numerical models (much more than those presented) in a tool (graph, table, etc.) that directly determines the influence of major geological faults, avoiding the numerical modelling in the future.

2.5 Summary and conclusions

The evaluation of the effect of major geological faults is essential for the open stope design. Using 3DEC software, three case studies corresponding to a mine in Chile have been evaluated.
A distinct element method (3DEC software) was used to study the effects of major geological structures (faults) on underground excavations, since it allows to design them easily and incorporate their specific properties. Additionally, it is essential for its use to have the largest amount of information from the site-specific conditions (mine) that allows to characterize the geological faults.

From a database of 45 case studies, three of them were selected to evaluate the influence of major geological faults on the stability of their backs. Nine numerical simulations of the three case studies were built and analyzed. Thus, the properties of the faults with greatest influence in the behavior of the overbreak on the backs of the open stopes were determined.

The K_n and K_S values were evaluated in relation to the maximum displacement on the open stope backs, indicating that the K_n value has a greater impact on the maximum displacement at the open stope back. Whereas K_S value has a reduced impact on the displacement.

The proposed methodology: calibration, verification and overbreak analysis (without faults) for each of the cases, showed that the overbreak on the backs of open stopes is recreated in the software under the criterion $\sigma_3 \leq 1MPa$. In this sense, it was also possible to verify that the geological faults that most contribute to the overbreak are sub-parallel to the backs as can be seen in case studies A and B (Figure 2.6). Whereas the case study C shown a lower impact of the geological faults due to their greater slope regarding its back (Figure 2.6). These findings are consistent with empirical studies available in the literature (Bewick and Kaiser, 2009; Clark and Pakalnis, 1997; Mathews et al., 1981; Mawdesley et al., 2001; Potvin, 1988; Suorineni, 1998; among others).

Using the selected criterion and the analysis carried out through the numerical modelling, a fault factor (F_b) that describes the influence of major geological structures on the stability of backs of the open stopes was introduced. Therefore, the factor can be used to modify the empirical stability classification of the backs based on the Mathews' methodology.
In this sense, in a next step of the study, it is recommended to correlate the results of all numerical models (much more than those presented) in a tool (graph, table, etc.) that directly determines the influence of major geological faults, avoiding the numerical modelling in the future.

The results shown an improvement in the empirical classification of the backs (of known case histories) through the incorporation of the factor that takes into account the impact of major geological faults to the Mathews' methodology. However, it is recommended to be careful with the use of the results and parameters that describe the behavior of the materials and elements involved, as their use is restricted to the site-specific conditions of the case studies and should only be used as a guide in terms of numerical modelling. Therefore, each mine should be able to conduct a site-specific study that reflects their own conditions, using as a reference the methodology described in this paper.

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Chapter 3

Conclusions

The main conclusions of the thesis carried out are shown below:

- The evaluation of the effect of major geological faults is essential for the open stope design. Using 3DEC software, three case studies corresponding to a mine in Chile have been evaluated. Through the study, the methodology introduced to evaluate the impact of major geological structures on the stability of backs of open stopes was validated, also verifying the hypothesis of the thesis.
- A distinct element method (3DEC software) was used to study the effects of major geological structures (faults) on underground excavations, since it allows to design them easily and incorporate their specific properties. Additionally, it is essential for its use to have the largest amount of information from the site-specific conditions (mine) that allows to characterize the geological faults.
  However, building the mesh (grid) through the tools available in the software could take a considerable time (several days even in computers with a processor intel i7 and 16GB of RAM) depending on the complex of the geometry and the building planes (fault planes) considered. This could lead to problems in the meshing process such as the impossibility of joining tetrahedral elements or having coarse elements around excavations, which make the numerical solution difficult. It is recommended for future users to considerer this issue and evaluate meshing alternatives when they use the software.
- From a database of 45 case studies (belonging to a Chilean Sublevel Open Stoping Mine), three of them were selected to evaluate the influence of major geological faults on the stability of their backs. Nine numerical simulations of the three case studies were built and analyzed. Thus, the properties of the faults with greatest influence in the behavior of the overbreak on the backs of the open stopes were determined.
- The K_n and K_S values were evaluated in relation to the maximum displacement on the open stope backs, indicating that the K_n value has a greater impact on the maximum displacement at the open stope back. Whereas K_S value has a reduced impact on the displacement.
- The proposed methodology: calibration, verification and overbreak analysis (without faults) for each of the cases, showed that the overbreak on the backs of open stopes is recreated in the software under the criterion $\sigma_3 \leq 1MPa$. In this sense, it was also possible to verify that the geological faults that most contribute to the overbreak are sub-parallel to the backs as can be seen in case studies A and B (Figure 2.6). Whereas the case study C shown a lower impact of the geological faults due to their greater slope regarding its back (Figure 2.6). These findings are consistent with empirical studies available in the literature (Bewick and Kaiser, 2009; Clark and Pakalnis, 1997; Mathews et al., 1981; Mawdesley et al., 2001; Potvin, 1988; Suorineni, 1998; among others).

- Using the selected criterion and the analysis carried out through the numerical modelling, a fault factor (F_b) that describes the influence of major geological structures on the stability of backs of the open stopes was introduced. Therefore, the factor can be used to modify the empirical stability classification of the backs based on the Mathews' methodology.
  In this sense, in a next step of the study, it is recommended to correlate the results of all numerical models (much more than those presented) in a tool (graph, table, etc.) that directly determines the influence of major geological faults, avoiding the numerical modelling in the future.
- The results shown an improvement in the empirical classification of the backs (of known case histories) through the incorporation of the factor that takes into account the impact of major geological faults to the Mathews' methodology. However, it is recommended to be careful with the use of the results and parameters that describe the behavior of the materials and elements involved, as their use is restricted to the site-specific conditions of the case studies and should only be used as a guide in terms of numerical modelling. Therefore, each mine should be able to conduct a site-specific study that reflects their own conditions, using as a reference the methodology described in this paper.

Chapter 4

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