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INVESTIGATING THE PARAMETERS GOVERNING THE RATE OF LEACHING OF PRIMARY SULFIDES COPPER DEPOSITS IN IN-MINE RECOVERY

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ABSTRACT

It is expected that there will be very few viable approaches for mining mineral deposits in the future. This is because profit is constantly reduced as higher costs are associated with the required technologies and techniques to extract minerals from lower grades deposits in challenging geotechnical conditions. It is also expected to have additional constrains in terms of environment requirements in the mines of future.

One of the mining techniques that might be useful for such conditions is In-Mine Recovery, IMR. Although very few studies have been done on this specific subject, it seems that IMR is environmentally friendly and can well fulfil the economic restrictions for several mining with challenging conditions. Additionally, by implementing IMR, the required mining developments can be significantly reduced. Material handling and comminution costs may also noticeably be decreased. There also would be much less pollution of the ground surface because of processing less materials on the surface.

Therefore, main objective of this study is to investigate the effective parameters controlling the rate of leaching in IMR. For example, enhanced fragmentation can significantly improve the metal recovery of the leaching and improve the profitability of the mining method. Other relevant factors, such as mineralization, temperature, pH, aeration, leaching solution, time of leaching and height of the load of material, are also considered in this report to develop a robust and reliable numerical model(s) to estimate the recovery of the copper deposits for IMR.

A case study of underground copper operation in Chile referred as “Mine A”, with a primary sulfide ore, a unique environment with unique geological and geo-mechanical conditions is considered in this research. The results show that the developed model is suitable and accurate enough for basic studies like, for the pre-feasibility study. It is noted that the prediction approach is highly dependent on various factors, such as shape factor of the fragmented rock particles as well as the mineralogical and geological characteristics of rocks. Performing detailed site inspections and evaluations are needed to properly calibrate the model and to estimate such factors. This can be achieved through performing small scale leaching tests as well as pilot and bed tests in sites.

According to the sensitivity analyses that was done for the “Mine A” case, temperature is among the most influential factors affecting the leaching recovery. Pressure and Time are also the other important parameters that must be considered for implementation of IMR in primary sulfide ores.

Keywords: Bornite, Chalcopyrite, Fragmentation, In-place mining, Leaching, Sulfides.

RESUMEN

Se espera que existan pocos métodos viables de extraer minerales en el futuro. Esto se debe a que el beneficio económico se está reduciendo por el aumento de costos necesarios en la implementación de distintas tecnologías y técnicas para la extracción del mineral en depósitos de baja ley con condiciones geomecánicas y geológicamente complicadas. Se espera también que existan mayores restricciones medioambientales para desarrollar minería en el futuro.

Uno de los métodos para procesar minerales que puede ser útil para satisfacer las crecientes restricciones para desarrollar minería es In-Mine Recovery, IMR. A pesar de que se han realizado muy pocos estudios en el área, IMR aparece como una técnica amigable con el medio ambiente y que puede cumplir también con los estándares necesarios para desarrollar minería en ambientes subterráneos con características complejas. Adicionalmente, implementando IPM los desarrollos necesarios para minería pueden verse significantemente reducidos, como también el movimiento de material y los costos de conminución. También es posible reducir la contaminación en la superficie debido a la reducción de material transportado y tratado en ésta.

Por lo tanto, el principal objetivo de este estudio es investigar el efecto de los parámetros que controlan la recuperación del metal en lixiviación. Por ejemplo, una fragmentación mejorada del material puede incrementar significativamente la recuperación metalúrgica y así mejorar el beneficio económico. Existen otros factores que gobiernan la recuperación del mineral tales como mineralización, temperatura pH, aeración, solución lixiviante, tiempo de lixiviación y altura de la carga de material, los cuales son considerados en este reporte para construir un modelo numérico para estimar la recuperación de cobre para IMR.

El caso de estudio considerado en esta investigación corresponde a la operación de una mina subterránea de cobre citada como ‘’Mine A’’ con sulfuros primarios y condiciones ambientales, geológicas y geomecánicas específicas. Los resultados mostrados para el caso de estudio son solo útiles para la estimación de un rango para la recuperación de cobre, siendo altamente dependientes de parámetros desconocidos, como el factor de forma y algunas consideraciones mineralógicas y geológicas. Según el análisis de sensibilidad desarrollado para la mina estudiada, la temperatura es el factor que tiene mayor efecto en la recuperación. Presión y tiempo son también factores altamente influyentes que deben ser analizados para un plan de implementación de IMR en sulfuros primarios.

Palabras claves: Bornita, Calcopirita, Fragmentación, In-place mining, Lixiviación, Sufuros.

Nothing in the world replaces constancy. TALENT does not replace it, because nothing is as common as the frustrated smart.

THE GENIUS either, since the case of ignored geniuses turns out to be typical.

Not even EDUCATION replaces constancy, because the world is full of failed well educated.

Only CONSTANCE AND DECISION achieve everything.

Anonymous

A mis tías Ani y Dalia por criarme.

Mis padres Jaquelín y Hermenegildo por guiarme.

A mi hermanito Panchito, por darme la fortaleza.

y mi familia en general por apoyarme.

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A todos MUCHAS GRACIAS.

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ABBREVIATIONS

A: Rock mass factor in Kuz-Ram model

B: Burden (m)

b: Undulation coefficient in Swebrec function

Bn: Bornite

Cpy: Chalcopyrite

CSC: Circular stope

CSIRO: Commonwealth Scientific and Industrial Research Organization

CSRD: Centrals Swedish Ring Draft

DRD: Diamond ring draft

GE: Grade engineering

ILM: In-Line Mining

IMR: In-Mine Recovery

IPM: In-Place Mining

M2M: Mine to mill

Ø: Shape factor

$Ø_{h}$: Blasthole diameter (mm)

PLS: Pregnant leach solution

PSD: Particle Size Distribution

Py: Pyrite

S: Spacing (m)

St: Stemming (m)

UGA: Geo-environmental unit

VCR: Vertical crater retreat

VoD: Velocity of Detonation (m/s)

1. INTRODUCTION

1.1 Basis

In-place mining, IPM, a novel concept developed by Mining3, is considered a reliable and economic mining method that allows recover metals such as copper, gold, uranium, potash and others, with less energy utilisation and allows extracting minerals from deeper deposits that may not be feasible with conventional mining methods. IPM method is also more environmentally friendly, with much less footprints on the surface than other conventional mining methods.

IPM combines methods such as conventional mining, In-Situ Recovery, ISR, In-Mine Recovery, IMR and In-Line Mining, ILM, to develop a cheaper material handling, consequently, a better hauling cost. It also helps to reduce the costs associated with material loading and hydrometallurgy infrastructure and assist to fulfil the environmental standards.

In this study we will focus on the In-Mine Recovery method (also named stope leaching), which integrates the conventional underground mining methods, where a stope is prepared with previous blasting techniques, and then a lixiviant solution is irrigated by injection wells or sprinkling within the stopes and extracted from the bottom of the stope. By implementing IMR stopes will always be full of broken rock particles and this helps to avoid subsidence and surface disturbance[1].

The process of extraction, concentration and refinement of minerals requires energy. This energy consumption is expected to be higher if the required fragmentation is finer. Depending on the implemented size reduction approach, this energy consumption, can be very expensive.

In traditional mining, when the cost of wear parts (50 000 million SEK[1]/year) and other costs are added, Outcherlony [2] estimated cost breakdown to be 1 % for explosives fracturing, 2 % for coarse (primary) crushing, 20 % for fine (secondary) crushing and 77 % for grinding. This shows that a large proportion of the energy used for comminution goes into the grinding. In fact, changes in the properties of the blasted rock fragments that carry through to the grinding stage could result in large economical and labour savings. However, in In-Mine Recovery only 20% to 30% of the materials will be processed conventionally. This means that the processing costs can significantly be reduced since less comminution is needed to obtain the valuable metal (only blasting and 20-30 % fragmented conventionally).

There are also several other factors that can be important in terms of hydrometallurgical processing.

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[1] 1 SEK=0.1 USD (01/03/2020)

Parameters such as Particle Size Distribution, PSD, resulting from fragmentation methods, also different particle shapes, the size and shape of the broken amount of material, the permeability of rock particles, temperature, oxygenation, and dewatering, are all relevant factors in the recovery of leaching in heap, dump and in-place mining methods.

Previous studies have shown that in general a finer fragmentation of the ore improves the leaching performance [2].

Additionally, by implementing a good fragmentation design and combining it with the concepts of Mine to Mill, M2M, and Grade Engineering, GE, the benefit from proper fragmentation design can even be increased. For example, separating and sorting the target fragmented particles according to the concepts of GE can also help to enhance the leaching yield. M2M is a concept that involves the effects of fragmentation on the downstream processes. It can be defined by PSD and metal process relation.

In terms of IPM, fragmentation shall be modified by optimization in blasting for all the material inside of a stope and sensing, sorting and crushing of the extracted material from the stope, to achieve the best recovery from the leaching of ore in the underground silos/stopes. If IPM involves concepts such as ILM the rock fragmentation shall be augmented not only to reduce the costs of the downstream crushing and milling processes but also to achieve a good perfomace of recovery in leaching. ILM refers to the processing of extracted materials (e.g., by applying concepts of M2M or GE) and then leaching of the processed materials, underground.

Bearing in mind the rate of recovery in leaching of copper, the effects of constrains in the relevant processes must be well understood in order to carry out fragmentation and mine design analyses. The change in parameters such as temperature, pH, aeration, PSD, and mineralization, can impact on the recovery and leaching costs. These should, therefore, be considered in the design process of IPM.

1.2 Objectives

The main objective of this research is to identify and investigate the relevant parameters that can affect the rate of recovery of leaching in IPM. The outcomes of this research will be used for the pre-feasibility study of a real case study of underground copper mine in Chile. The study will, therefore, mainly be performed based on a comprehensive literature study of the cases with similar geological, geo-metallurgical and geo-mechanical conditions. For example, the role of fragmentation of ore in in-situ silos on the recovery of IPM will be investigated. The main subjects that will be investigated in this report can be summarised as:

- To identify all the parameters that can affect the recovery of IPM.
- To develop a model to predict the rate of recovery of leaching based on the parameters in leaching. Such a model will then be used for the pre-feasibility study of IPM in a real case study of copper mine in Chile.
- To perform a sensitivity analysis to investigate the effects of variations in the different parameters on the recovery of copper in the considered case study.
- To find a proper fragmentation design for the case study according to the geological and geotechnical condition of the mining deposits and to maximise the performance of leaching under economical constrains.
- To recommend some considerations for further studies.

1.3 Hypothesis

- The drill and blasting techniques used in conventional mining can be adopted for the fragmentation of ore in silos/stopes for IMR.
- Physical-chemical concepts of in situ recovery, heap and dump leaching can be considered for the leaching of copper in in-situ stopes.
- IMR is performed in underground silos full of broken materials. The technical problems related to leaching in such an environment would be out of the scope of this study.
- Copper ore mainly consists of sulfide and contain chalcopyrite, pyrite and bornite. with anhydrite and gypsum presence. Based on the geological studies of the case study it is expected that chalcopyrite is the dominant mineral in the area of the study and is therefore considered as the main core of this study.
- Underground stopes/silos are stable, and no stability assessment is considered in this study. This has been the topic of another bachelor thesis that was conducted parallel to this study.
- The stable sizes of stopes are assumed to be 40×40×240 m^3.
- It is expected that 20% to 30% of broken materials in stopes/silos should be withdrawn to ensure enough free surface and space for the efficient fragmentation in stopes (to allow for bulking effect).
- Extracted materials are sent to an existent processing plant that belongs to an adjacent open pit operation. Therefore, no capital investment is needed to prepare a processing plant.

1.4 Research questions

As discussed earlier, the main objective of this research study is to develop a model that can be used for predicting the recovery of copper in IPM. The model may then be used for the pre-feasibility study of a real case study of underground copper in Chile. Due to the confidentiality of the project we may call this case study “Mine A” here. The research aims at finding proper answer for the following questions:

A: What are the effective factors impacting on the recovery of ore in underground silos/stopes of IPM?

B: What are the main constrains in leaching of sulfide ores, like chalcopyrite and bornite?

C: How does fragmentation affect the recovery of in-situ leaching and what is the optimum fragmentation for IMR?

D: Is it possible to develop a simple but accurate model to account for the effect of the governing factors in the rate of recovery of IMR?

E: Can we implement this model to predict the recovery of copper in IMR for the considered case study?

1.5 Methodology

1.5.1 Background of this research

The main strategy used in this thesis was a comprehensive desktop study of the relevant literature. Data from different sources have been extracted and processed to investigate the effects of different parameters on the recovery of copper by leaching in IMR. The case study of this research, “Mine A”, in Chile, is then introduced as a typical copper mine that is struggling with conventional mining method and is looking for alternative mining approaches, like In-Place Mining (IMR+ILM). Therefore, the author mainly focused on the literature that have geological, geo-metallurgical and geo-mechanical conditions similar to Mine A.

Data from relevant literature are processed and used to and propose analytical/empirical models to take into account the effects of different parameters on the leaching of ore in IPM. Such data and information are extracted from previous works and relevant literature, including books, journal and conference papers and reports such as [1, 3].

The author followed the methodology defined below:

- The first step of the study was to investigate the geological, geotechnical and geomechanical condition of the case study.
- The second step was to perform a relatively comprehensive literature study in order to extract the useful data about leaching and mining from relevant literature with similar geological and geo-mechanical conditions.
- At the third step of this study, the author identified the effective parameters in leaching of copper in IMR.
- Then at the fourth stage of the research the useful data were extracted, refined, processed and integrated to investigate the effects of different factors on the recovery of leaching in IMR.
- The author then developed a simple model that can be implemented for the prediction of recovery of leaching based on the main effective parameters and mining design.
- A sensitivity analysis then was performed using the “Excel” and “R” tools to investigate the effects of different parameters on the recovery of copper based on the proposed model.

It is also noted that Mining 3 and CSIRO have constantly been looking for improvements of In-Place Mining Technologies. The author, during his vacation study in CSIRO, has been involved with a research project related to the prefeasibility study of IPM in an underground copper mine in Chile, known as “Mine A” in this research. The author has therefore decided to develop a simple but realistic model that can be used for the prediction of recovery in IMR of copper.

The author also attended the ALTA Metallurgical, Mineral and Chemical Processing conference in Perth, Australia. This helped the author to highly enhance his background in in-situ recovery and leaching. The author also attended an interesting short course about heap leaching in conjunction with the conference. This course contained several fundamental approaches for in-situ leaching and provided the author with a decent understanding of the technique that can be implemented for stope leaching in IPM. Additionally, on the same research journey, the author performed a technical visit of the CSIRO leaching team and their labs, in Perth. The author inspected the leaching, and mineralogy laboratory, as well as several metallurgical facilities.

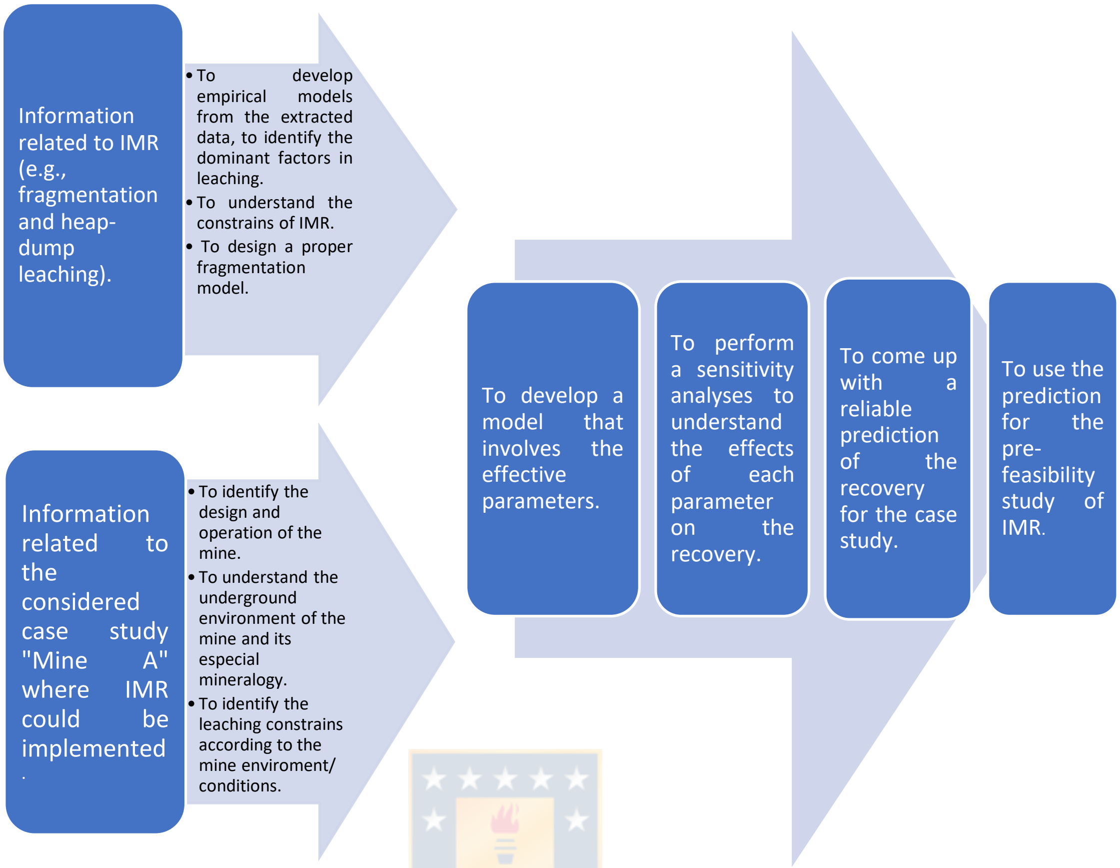


Figure 1 Work plan for the thesis

1.5.2 Research Design

It is noted that IPM is a relatively new approach. Therefore, one of the main constrains in this research is the lack of information about real case studies of IPM. It is however expected that some of the theories and approaches developed for dump-heap-in situ leaching, and for underground fragmentation in stope mining, that exist in the literature can be used here to extract the required materials for the assessment of leaching for underground IMR. It is noteworthy that the author has a unique opportunity to work with several experts and the developers of different In-Place Mining Techniques in CSIRO and Mining3, in Australia. The majority of the IPM terms were, therefore, collected from Mining3 and CSIRO current and previous works.

1.6 Scopes

The scopes of this project are:

- To extract the data from the relevant literature and to refine and process them in order to develop a simple but reliable model that can be used for predicting the recovery of copper in IMR.
- To determine the effective parameters controlling the leaching of sulfide copper minerals based on a comprehensive literature study of mines with similar geological, geo-mechanical and geo-metallurgical conditions. These include but are not limited to mineralization, geotechnical conditions, yield and environmental constrains in IMR.
- To design blasting arrangements to achieve a proper fragmentation for stopes that can help to improve the rate of recovery of leaching in IMR.
- To identify leaching constrains that can negatively affect the recovery and to find the ways that can be used to remove/remedy such issues.

2. BACKGROUND

In this chapter conceptual information about the context of mining in Chile, the novel concept of IPM and its corresponding methods; ISR, IMR and ILM, GE, Leaching and analysed ore deposits is given.

2.1 Actual and future Chilean Mining

Chile is currently, the world's dominant producer of copper ore but this position should not be taken for granted. Rising costs, declining productivity, lower grades and rising social and regulatory pressure in the areas of community engagement (social license to operate) and environmental sustainability are hampering the sector's profitability and ability to remain globally competitive [4].

Relevant research can identify the opportunities for step change innovation within the changing global landscape to forge a new upward trajectory for economic growth for both the industry and the nation. Hence, mining has the potential to act as a platform for change in Chile. Through advances in scientific research and development and technological innovation it can address the challenges that not only hinder the productivity of the industry but also the country, such as rising energy costs. This may drive the growth of a new mining services sector that helps to diversify the economy. Innovative mining can also help the country to take a leadership position on issues such as environmental and social sustainability. Changes such as these will be vital to ensure that Chile remains a key player within the global mining value chain over the coming decades [4].

Therefore, IPM appears as a proper alternative to conventional mining in order to carry out mining operations due to is environmentally friendly characteristics, and less water consumption. On the other hand, it can provide a better economic profit since less labour work and less material movement is needed. Additionally, much less equipment would be involved in such mining and in general a smaller crew is required to carry out the mining operation. In all mining project, it is always needed to have a good understanding of the environment where the mine is intended to be sited. Considering this, IMR is a process which facilitates the mineral extraction in underground environments, but it is restricted by the onsite constrains. The leaching efficiency is affected by increasing or decreasing yield of the process, and factors such as fragmentation, mineralogy, leaching time, temperature, leaching solution, pH, height of the pile of broken rock, irrigation rate and aeration.

2.2 In-place Mining

Figure 2 shows a schematic view of In-place mining. The In-place mining concept involves three different techniques for the extraction of minerals from underground mines and deep deposits. These include In-Line Recovery, In-Mine Recovery and In-Situ Recovery. The main advantage of these new ways of mining is the reduction of material movement, which allows better economic profits, reduces the environmental effects and may allow to go for ore in deeper deposits in challenging geotechnical conditions.

Moving and processing vast amounts of rock to extract a small percentage of valuable metal is not only environmentally disruptive, but also representing over 80% of the total production costs in mining. The industry normally sets their cut-off grades according to this cost structure and are forced into looking for economies of scale. Therefore, In Place Mining, IPM, takes advantage of saving costs, because of less material movement, being consequently economically more profitable due to less usage of equipment, energy and human resources.

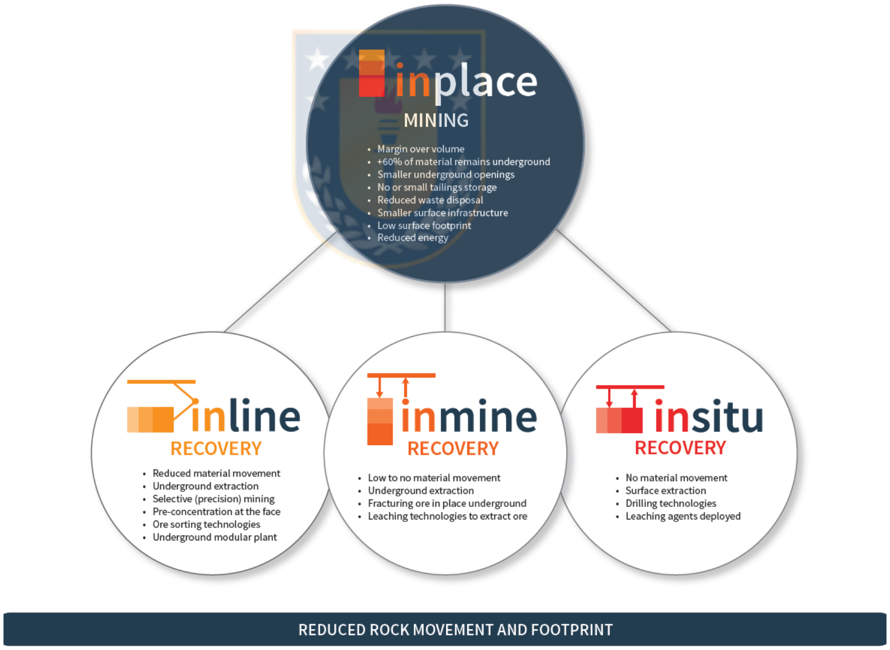


Figure 2 In-place mining scheme.

Courtesy of Mining 3 (2019)

2.2.1 In-Situ Recovery

For a better understanding of In-Place Mining procedures, in situ leaching/recovery appears as a first conceptual base to develop IMR. This method has been used in deposits of minerals such as uranium and copper deposits. The method basically involves injecting a solution to underground by pipes or created channels. The solution that contain a certain concentration of a lixiviant extracts the ore from the deposits by chemical reactions with the host rocks. The final solution, known as pregnant solution, is then pumped and extracted from the surface. Figure 3 shows the fundamental of In-Situ Recovery, ISR.

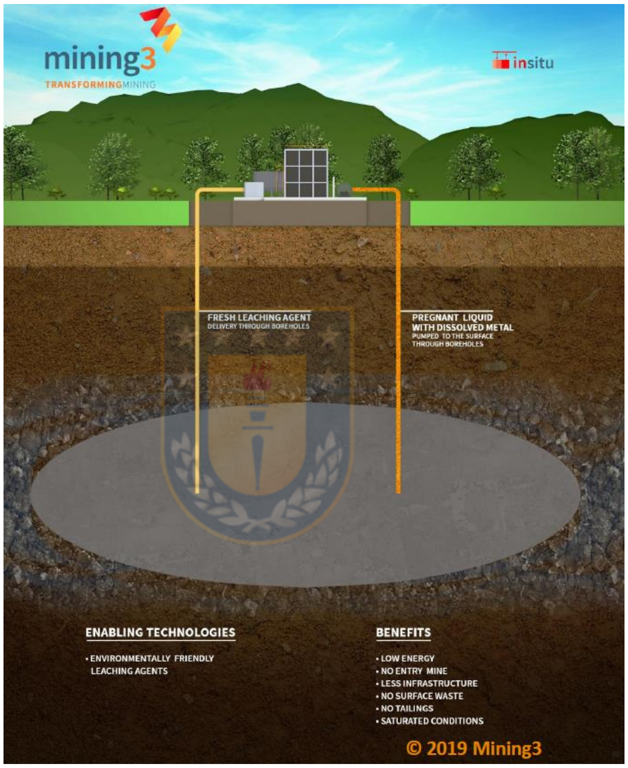


Figure 3 In-situ recovery scheme.

Courtesy of Mining 3 (2019)

2.2.2 In-Mine Recovery

In Mine Recovery, IMR, or stope leaching is a process used to recover minerals such as copper and uranium through boreholes drilled into a previously blasted stope, where the minerals are leached by chemical/biological solutions and then are extracted in a pregnant leach solution, PLS. In this case of mining, fragmentation of the material is a critical parameter in the leaching kinetics of the ore. Enhancing the permeability by a proper fragmentation can help to facilitate the flow of the lixiviant to easily react with particles surface (see Figure 4).

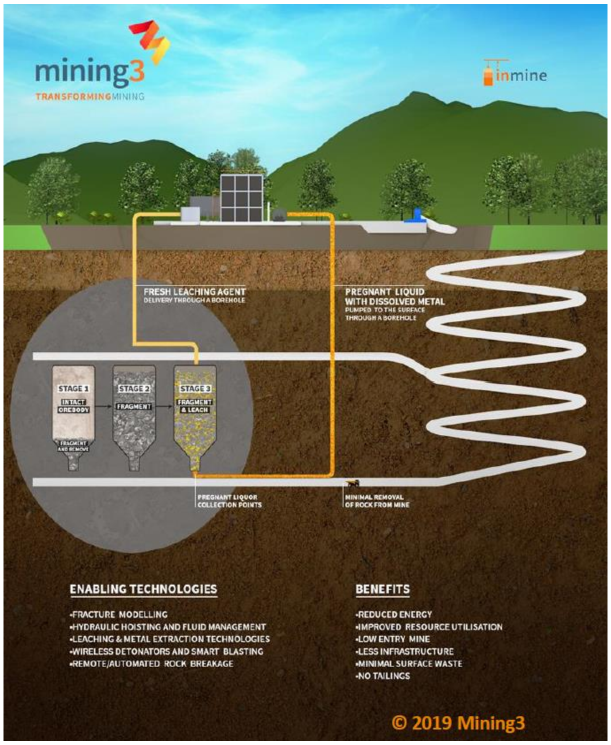


Figure 4 In-mine recovery scheme.

Courtesy of Mining 3 (2019)

This method is more environmentally friendly than conventional stope mining and can be implemented in different ways, in terms of mine design, which is an advantage of the method as the operation can be selected according to the geological and economical constrains. The cut-off grade of the method is expected to be much lower than the corresponding value in the conventional methods.

The concepts of stope and fragmentation are needed to be understood in this thesis. They are therefore as follow:

Stope: The openings made in the process of extracting ore are called stopes or rooms. Stopes are a defined area within an underground orebody where the material is fragmented and, in conventional mining, extracted and transported to surface for further processing.

Fragmentation: Rock fragmentation can be represented as the particle size distribution of rock material. Fragmentation is used in the mining industry as an index to estimate the effect of stope blasting. It is well known that the rock fragmentation in a stope is affected by blasting parameters such as specific charge, spacing and burden, involving rock heterogeneity, dynamic fracture phenomena, etc.

2.2.3 In-Line Recovery

In In-Line Recovery, ILM, the process to size and sort rock to achieve a better yield in further refinement processes of mineral is mainly developed underground. In some cases, building smaller processing plants on site (underground) and avoiding significant material movement to the surface can be considered as a suitable approach for ILR. Sorting of material in the primary stages, selective crushing, and then, processing the classified minerals, can help to implement the concept of Grade Engineering [5] (see Figure 5).

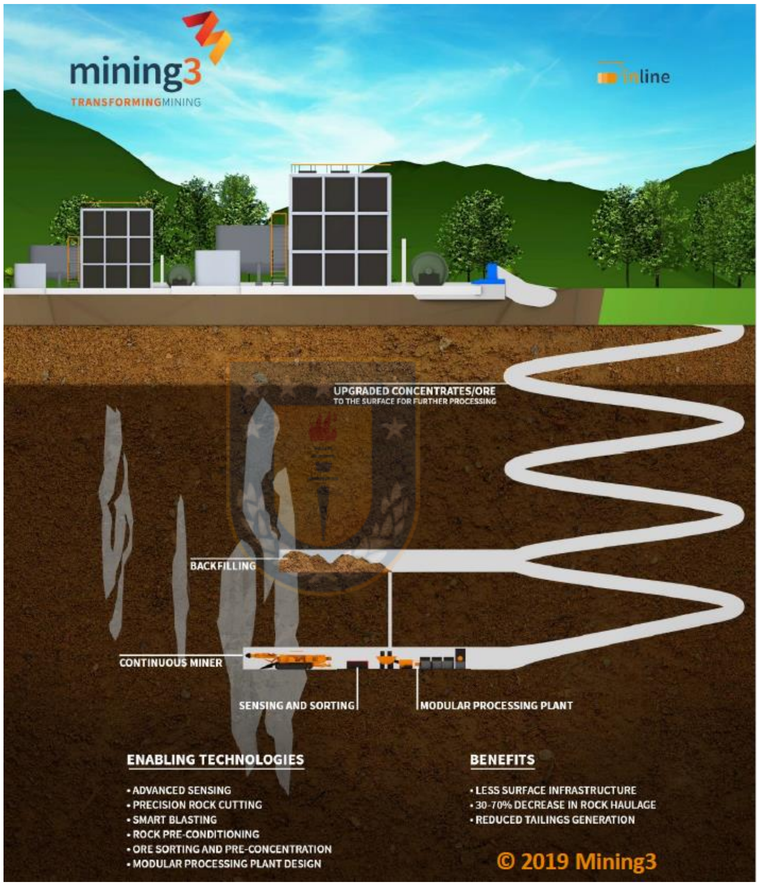


Figure 5 In-line mining scheme.

Courtesy of Mining 3 (2019)

2.3 Leaching

Leaching according to the definition of Ruiz [3], is a selective dissolution of a target mineral of the ore or concentrate, obtaining as a result a liquid solution called Pregnant Leach Solution (PLS) which contains valuable metal and an insoluble waste without value. Following the leaching stage, generally one or more stages of liquid-solid separation are included and at the end of these processes the valuable metal is precipitated from the clear solution.

Leaching is fundamentally a solid-liquid mass transferring. The leaching process can be carried out in different environment conditions as high temperatures and/or pressure. The conditions of the process depend on the chemical reactions that are occurring. The objective in all the cases is to obtain metallic or complex ions which can be selectively extracted from solution.

For this study the definitions of heap leaching and dump leaching are also required, because the governing parameters of the process and its respectively modelling for stope leaching are made according to previous studies and characteristics of those technologies.

Heap Leaching: Heap leaching is frequently used to extract copper from medium-grade minerals. These grades allow to incorporate a dewatering system to extract the PLS from the base of the heap, also having pipes for forced aeration into the heap, and agglomeration when permeability must be enhanced. Heap leaching has a recovery between 60-70 % in periods of 250-350 days for secondary copper sulphide minerals. However, two criteria must be satisfied to assure this recovery: (1) heaps must be uniformly permeable to solution flow and (2) solution must be uniformly distributed over the surface of the heap.

Dump Leaching: Dump leaching is used in low-grade minerals that may be considered waste in terms of processing in the plant. At the surface of the dump a leaching solution is applied. This solution percolates by gravity flow through the mineral. Dump Leaching is characterised by a recovery between 20-50 %, in large cycles (years) having lower capital and operation costs compared with heap leaching.

2.3.1 Leaching flowsheet for IPM

The IPM leaching flowsheet is presented in Figure 6. The process is explained as:

- Firstly, the stope rocks are fragmented by blasting. A proportion of the broken materials (20 to 30%) will be extracted from the stope in order to create the needed space and permeability to effectively circulate the leaching solutions,
- The stope is irrigated with a leaching solution,
- Pregnant Leach Solution, PLS, is then extracted from the bottom of the stopes,
- The valuable minerals are separated by solvent extraction process. Then the remaining useful part of this liquid is forwarded to be recirculated into the stope, performing the same process once again.
- After extracting the valuable metals, this metal is sent to the electrowinning process, to obtain the final product.

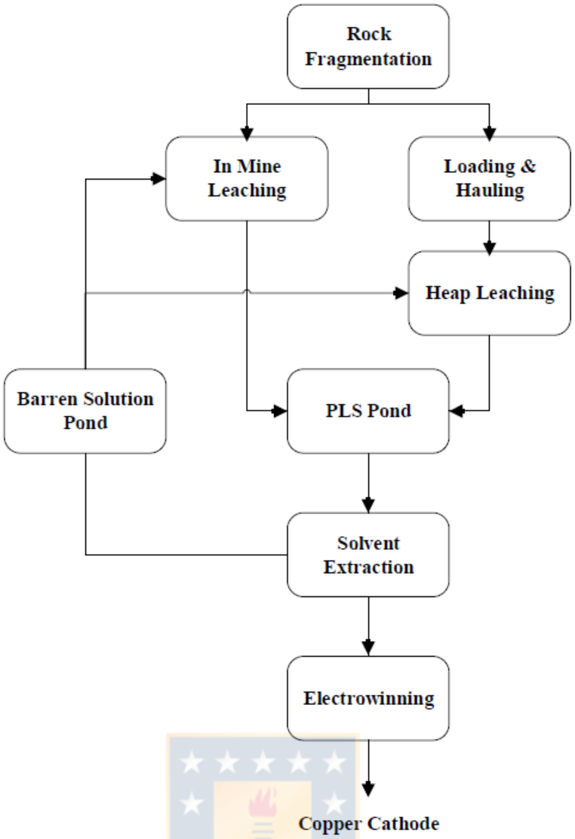


Figure 6 Flowsheet of In-Mine Recovery Leaching.

(Source: Development and Evaluation of an Economic Model of the In-mine recovery Concept. Rossien (2018))

2.4 Grade engineering

Carrasco and Keeney [6] explain that the grade of particles is dependent on the sizes of the particles. Therefore, it might be possible to obtain a better recovery by sorting the broken material based on their sizes and rejecting low grade material early in the extraction process. Screening and sorting for Grade Engineering can also help to reduce the costs in crushing and milling stages. In other words, the preferential grade by size is the main concept behind a few of the Grade Engineering levers.

The preferential grade by size is explained based on the difference of the elastic limit breakage of particles of ore and the gangue materials. When a particle is finer, it is highly probable that this particle was much more fragmented and has been weaker in terms of strength. Many valuable minerals are weaker than the surrounding water rock mass.

2.5 Primary copper sulfides

Copper sulfide ores are typically among the most profitable types of copper ores to be mined because they have high copper content. It is also relatively easily to separate such minerals from the undesirable accompanying waste. A copper ore that primary consists of sulfide minerals is expected to be located deeper than oxides and secondary sulphides in an underground deposit of mineral. Figure 7 shows a scheme of the distribution of the mineralization in a common copper deposit. Chalcopyrite is the most abundant copper mineral in primary sulfides, however, is the most refractory in terms of leachability

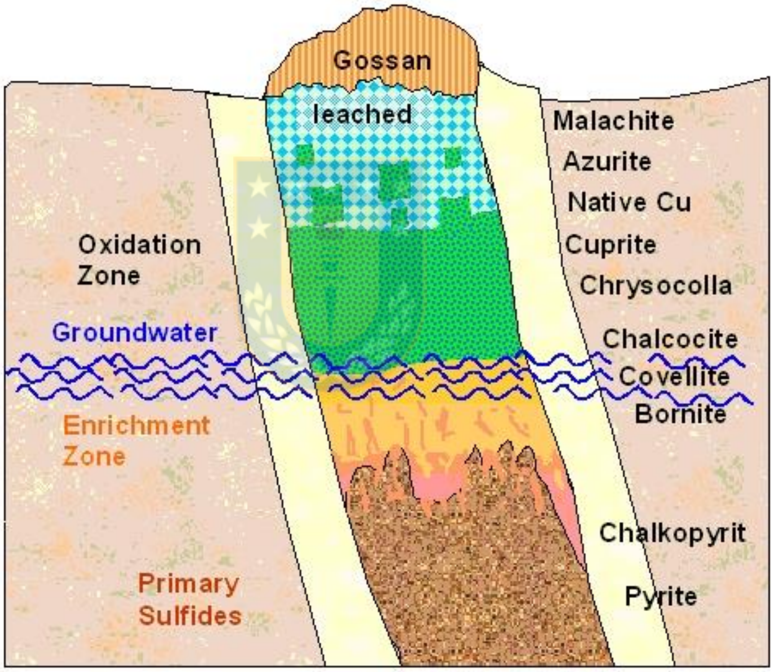


Figure 7 Spatial distribution of copper ores.

(Source: Pernicka/ Wweisgerber)

3. GOVERNING FACTORS IN LEACHING OF SULFIDE COPPER

In-Place Mining, IPM, considers leaching to separate value metals from the entire extracted or fragmented material in an underground environment, either through IMR where the orebody is fragmented and leached in-situ in some underground stopes, silos, or the conventional leaching techniques where the material is extracted and leached in heaps/dumps at the surface or in smaller scale in underground.

In this study, the author has mainly focused on the leaching of copper for IMR. However, to develop a prediction model for the rate of recovery, the characteristics, behaviours and theories of leaching of a sulfide ore in heap, and dump leaching and the concepts of ISR are used. IMR involves conditions from geology, mining and processing at the same time. This is different than the conventional method where these three processes are ordered in a lineal sequence, on this case the processes are related to each other in a circular way, it means that all the conditions belonging each stage should be considered at the same moment for the design of the operation.

The effective parameters that are analysed in this study are the fragmentation of the ore (Particle Size Distribution), mineralogy, leaching time, temperature, aeration, leaching solution, (lixiviant, oxidant, etc.), the height of the pile of broken rock, irrigation rate and the hydrogeology of the stope including issues such as segregation, channelling and ponding.

3.1 Fragmentation

One of the main parameters involved in IPM, which must be considered in relevant analyses is the fragmentation of ore materials. Fragmentation is a key factor in the recovery of leaching as it increases the exposed surfaces of the particles and subsequently increases the potential of leaching recovery. This is because many sulfide deposits have very low permeabilities. Indeed, a proper rock fragmentation increases the permeability of the rock and allows the lixiviants to flow through the ore body much more easily and efficiently. This can well help to leach a larger area of the ore materials. Generally, the rock’s particle sizes, and particle shapes are the factors that determine the surface area of particles. Hence, they should be considered in investigating the rate of recovery in leaching.

In terms of IPM method, fragmentation can affect the recovery of extraction in both ILM and IMR. In ILM the material is sorted by size crushers and sent to different streams to process and extract the valuable metal, either by flotation or leaching. In IMR the objective of fragmentation of the rock mass is to liberate and expose the mineral allowing the lixiviant to go through the particles and get the valuable metal from this blasted-rock material. Therefore, in case of IMR the objective is to create a suitable permeability in rock masses whose permeability is too low, creating gaps and channels to allow an efficient circulation of the lixiviant through the stope. The same strategy is also developed for heap leaching where a certain PSD is desired to get the maximum amount of the valuable mineral from the materials leached in the pads. Such a favourable PSD can be reached by adjusting the blasting and/or by adding a few crushers in the operations. However, blasting is generally the most effective and cost-effective comminution. Therefore, the objective would be to improve fragmentation to optimise the recovery of leaching without going to energy and time-consuming mechanical crushing [7].

Fragmentation analysis is, therefore, needed to investigate the relationship between percentage of exposed valuable mineral with respect to the particle size for a given ore type. In other words, to predict the extend of recovery in heap/dump leaching or in stope leaching processes it is necessary to know the specific PSD**.**

A finer fragmentation allows the lixiviant to go through the particles and to achieve a better recovery because intense fragmentation can increase the surface exposure of the precious metal. There are, however, a few constrains that must be considered. For example, in some cases the generation of very fine particles may produce flow patterns, also named as channelling. This can affect the kinetics of reaction, and/or block of the flow, retain the pregnant solution inside the heap/stope (ponding) and reduce the efficiency of the leaching processes. The PSD of rocks in IMR might presents larger particles (ROM) than a Heap leaching (crushed material).

Ghorbani and Franzidis [8] reported on the sizes of fragments of rock material for some leaching operations. Table 1 shows the range of p_{80} sizes for each kind of leaching.

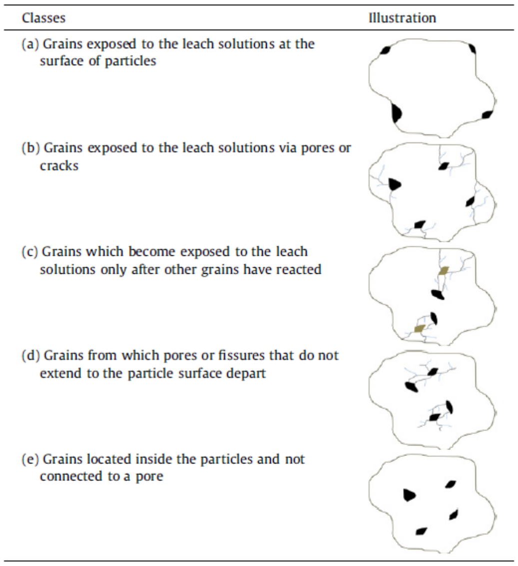
In the table the p_{80} size for ISL is over 1000 mm. Finer sizes for PSD are expected to be observed for the ROM materials sent for dump leaching in mining,(to have an enhanced dump leaching yield, the crushing specific range is typically 30 to 1000 mm) which can fulfil the fragmented material conditions induced with blasting.

Similar kinetics rules from dump and in-situ leaching can, therefore, be used to explain the leaching processes in IMR. Such a particle size range should, then, be aimed for blasting operations to achieve a useful PSD in IMR. However, stope stability constrains, and the costs associated with blast fragmentation should also be considered in the fragmentation design. Although a proper PSD should be defined for efficient IMR, the PSD alone does not provide enough information for a decent understanding of the leaching behaviour. Care should also be given to the mineralogical and environmental conditions such as temperature, clays presence, lixiviant characteristics, and so on.

Table 1 Percolation leaching types and typical criteria (Source: Ghorbani, Franzidis [8])

Type	Particle size (P80 mm)	Crushed	Agglomeration	Irrigation rates (Lm-2h-1)	Lift height (m)	Leach Time (Years)	Recovery (Typical %)
In Situ Recovery	>1,000 mm	Can be in situ blasted	n.a.	Wide and varied	n.a.	Cu: >5 and U: 1–3	5-50
Dump Leaching	1,000–30 mm	No	No	2–15	8–75 m	Cu: >10 and Au: 2–6	20-85%
Heap Leaching	100-5 mm	Yes	Mostly	2-15	2-10 m	Cu 1-4, Ni:1-5 U: 1-3, Au: 0.1-2	40-97
Vat Leach	10-0.5 mm	Yes	Maybe	10-50	1-5 m	4-30 days	80-97
Agglomerated Fines Heap Leach	1-0.25 mm	Yes and/or milled	Yes	2-15	1-5 m	As per heap leach	70-97

In terms of exposure and degree of freedom of the valuable metal, the micro-structure of the rock is important. The micro-structure shows the grains and the bonding between the grains. The rocks grains may be defined based on different distributions of their components. In other words, the minerals grains can be classified by their accessibility to leach solutions (see Figure 8).



Classes	Illustration
(a) Grains exposed to the leach solutions at the surface of particles	
(b) Grains exposed to the leach solutions via pores or cracks	
(c) Grains which become exposed to the leach solutions only after other grains have reacted	
(d) Grains from which pores or fissures that do not extend to the particle surface depart	
(e) Grains located inside the particles and not connected to a pore	

Figure 8 Classification of grains according to ore particles distribution (Source Ghorbani, Becker [9])

Figure 8 shows the different kinds of situations that can be found in the rock grain inside a stope designed for IMR. Types (d) and (e) are not the favourable grains in terms of recovery, at least in early stages of leaching. However, as the time elapses and chemical reactions are happening, the encapsulated mineral can be exposed or liberated by fissures into the surrounding gangue materials. For cases (a), (b) and (c), the time to achieve a suitable recovery is expected to be faster. This is because the lixiviant can be in contact with the valuable metals inside the grains. It should be noted that the largest particles size can control the kinetic of the process which is diffusion-controlled or mixed (diffusion and surface-chemical reaction controlled).

A study [10] was also performed using experimental procedures where 7-tons of low-tenor copper sulfide ores 7-ton were leached for 500 days. The outcomes of this study showed that when particles with a -1/2 inches or <12.7 mm were removed, the mixed kinetic model was appropriate to estimate the leaching recovery. The recovery could rise as high as 56.5% at the 460th day. On the other hand, particles of ROM (including fines) showed a lower recovery, around 47% of valuable metal content. The outcomes of this study indicated that copper was leached more readily from coarse material where the fines were removed from the samples than when the fines were not removed. It was concluded that fines indirectly affect the recovery because they produce channelling and ponding despite increasing the exposed surface of the particle.

In order to assess the effective particle size for IMR leaching purpose there are three main limitations that should be addressed:

- At infinite separation (i.e. very dilute slurry leaching), where the leach particle dimension is the actual blasted particle size.
- At stope voidage equal to the limit moisture capacity (plus the occluded and advection voids), the leach particle needs space to allow its volume (solid and liquid). External strains can affect negatively the circulation of the solution in the particles surface.
- As stopes are underground and additionally as blasting is a risky operation, it may result in stability issues on pillars or posterior leakage due to over- breaking and damaging of the surrounding stopes. Hence, the blast operation must be executed in a responsible way.

In this study, the costs associated with the leaching of minerals in-place are not analysed and not included as a very decisive factor. However, it shall be noted that the induced changes in fragmentation to generate a better PSD for in-situ leaching in stope can be reached using an adequate or intense blasting. This also helps to reduce the stages of comminution needed for the conventional processing of broken rocks. Therefore, by employing a proper blast design in IMR, the reduction of comminution cost can be significant. Generally speaking, in conventional methods of mineral processing, there are three to four stages of crushing, milling and grinding which are needed to produce the proper particle sizes. Additionally, if IMR is designed properly and combined with the theories of ILM the broken particles can then be sorted according to their sizes and be processed or leached based on the concepts of Grade Engineering.

If the concept of IMR is exclusively used, the materials in stopes are not entirely extracted, except the percentage that is needed to accommodate the swelling/bulking effects of the broken materials. This part (20 to 30%) of the materials will be removed from the stope and the rest will remain inside the stope to be leached in-place. This proportion is also sufficient to ensure that the stopes remains stable.

3.2 Mineralogy

Mineralogy is a governing factor that should carefully be considered to understand the mechanisms of leaching and to predict the leaching rates of minerals for IPM. Different minerals in a deposit may represent significantly different behaviours during leaching.

Generally, different minerals have different rates of leaching because they contain diverse chemical and physical characteristics, crystal structure, density, and so on.

Although leaching of copper has been investigated for many years and numerous theories were developed to understand this process, much remains unknown concerning the reaction and the chemistry of various sulfide minerals, especially Cu-Fe sulfides which are among the dominant minerals in mining deposits.

A summary of the time required for leaching for copper minerals is shown in the following Table *2*. A longer leaching time means that the mineral is harder to leach due to its kinetic characteristics.

Table 2 Summary of common rates of leaching for different copper minerals. (Source: Watling [11])

Mineral	Leaching and oxidation Reactions	Duration
Atacamite	$Cu_2Cl\ (OH)_3+3H^+ \rightarrow 2Cu^{2+}+Cl^-+3H_2O$	Hours to days
Chrysocolla	$CuSiO_3 \cdot 2H_2O+2H^+ \rightarrow Cu^{2+}+SiO_2 \cdot 3H_2O$	
Neotocite	$(Cu, Mn)2H_2Si_2O_5(OH)_4 \cdot nH_2O+4H^+ \rightarrow Cu^{2+}+Mn^{2+}+4SiO_2+6+nH2O$	
Tenorite	$CuO+H_2SO_4 \rightarrow CuSO_4+H_2O$	
Malachite	$Cu_2(CO_3)(OH)_2+2H_2SO_4 \rightarrow 2CuSO_4+CO_2+3H_2O$	
Azurite	$Cu_3(CO_3)_2(OH)_2+3H2SO_4 \rightarrow 3CuSO_4+2CO_2+4H_2O$	
Brochantite	$Cu_4(SO_4)(OH)_6+6H^+ \rightarrow CuSO_4+3Cu^{2+}+6H_2O$	
Native copper	$Cu+1/2O_2+ H_2SO_4 \rightarrow CuSO_4+H_2O$	Days to months
Cuprite	$Cu_2O+1/2O_2+2H_2SO_4 \rightarrow 2CuSO_4+2H_2O$	
Chalcocite	$Cu_2S+1/2O_2+HH_2SO_4 \rightarrow CuS^*+CuSO_4+H_2O$	
	$Cu_2S+ Fe_2(SO_4)_3 \rightarrow CuS^*+CuSO_4+2FeSO_4$	

	*Cu is a reaction product rather than the mineral covellite	
Bornite	$Cu_5FeS_4+2Fe_2(SO_4)_3\rightarrow 2CuS+CuFeS_2+2CuSO_4+4FeSO_4$	Months to years
Covellite	$CuS+2O_2\rightarrow CuSO_4$	
	$CuS+Fe_2(SO_4)_3\rightarrow CuSO_4+2FeSO_4+S^0$	
Enargite	$Cu_3AsS_4+41/2Fe_2(SO_4)_3+2H_2O\rightarrow 3CuSO_4+9FeSO_4+4S^0+HAsO_2+11/2H_2SO_4$	Years
Chalcopyrite	$CuFeS_2+O_2+2H_2SO_4\rightarrow CuSO_4+FeSO_4+2S^0+2H_2O$	

For the analyses of copper mineralogy, tests have normally been done on sulfides minerals, whose leaching times are longer than oxides and having a high refractory behaviour due to the formation of passivating layers. In this chapter, the recovery of sulphide minerals containing chalcopyrite, bornite, gypsum and anhydrite was investigated, which are present in the studied case of “Mine A”.

Chalcopyrite leaching kinetics have been found to be very slow. Consequently, the rate of recovery is expected to be a low value. Indeed, refractory of chalcopyrite requires the addition of an oxidant for dissolution [12]. On the other hand, bornite presents better kinetics than chalcopyrite, being easier and faster to dissolve. However, bornite can also need chemical reactions to be transformed in another mineral with faster leaching rates such as chalcopyrite, chalcocite or covellite. The existence of minerals like anhydrite ($CaSO_4$) and gypsum ($CaSO_42H_2O$) should also be considered in the prediction of recovery as they affect the acid consumption and reduce the kinetics of leaching due to interference of the gangue at the layer of ore particles. This interference is called “passivation”. In some cases, passivation layers may restrict lixiviant access to the reaction surface.

Temperature is also a key factor in leaching and plays a dominant role in the dissolution kinetics of chalcopyrite. The dissolution rate of chalcopyrite is generally increasing with a higher oxidant concentration and/or higher temperature. The chalcopyrite leaching behaviour is also different according to the media in which it is leached. In most of the cases the chalcopyrite is dissolved in sulfate media and chloride media [13]. Time of leaching, particle sizes, acid concentration, and type and concentration of oxidant should also be considered in analysing the sulfide’s behaviour in leaching. For example; the extraction rate of chalcopyrite is expected to strongly be dependent on the solution pH values [14]

Secondary enrichment can occur during the leaching of individual sulfides leaching. For example, bornite (Cu_5Fe_4) can be altered to chalcopyrite ($CuFeSe_2$) and chalcocite (Cu_2S) or Covellite (CuS) (see equations 1 and 2). This conversion is amenable to further treatment.

$$Cu_5Fe_4 \rightarrow CuFeSe_2+2Cu_2S \tag{1}$$

$$CuFeSe_2 \rightarrow CuS+FeS \tag{2}$$

Chetty [15] stated that acid consumption from gangue interactions varies according to different mineralogy, temperature and size of fragments, containing a variable behaviour for different conditions. For example, the formation of gypsum is not always detrimental. Under specific conditions gypsum may act as a binding agent, agglomerating particles and enhancing percolation. However, when there are fines particles within a stope, the binding of the zone may produce retention of moisture, ponding and channelling. The interaction between mineral and solution can be affected by gypsum presence, encapsulating rock-forming minerals. In a sulfide containing rock, this may also affect the oxidation rate of the associated sulfides by modifying the composition and ionic strength of solution containing Cl^- and SO_4^{-2} inhibiting pyrite oxidation. Hence, the dissolution of gypsum in sedimentary rock containing pyrite would impede oxidation.

Anhydrite has dissolution conditions for higher temperatures, being detrimental for the kinetics generating passivation. Additionally, Anhydrite is also an acid consumer, increasing the quantity of acid needed for a certain quantity of material, and subsequently increasing the pH of the solution.

3.3 Leaching Time

Leaching time is a factor which is analysed in this chapter due to its important effect on yield, production and economics of IPM.

The time that is needed to achieve a proper recovery always depends on the mineralogy of the ore, lixiviant concentration, temperature, size of the pile of material in the stope, or in heap or dump, and the PSD of the broken rock. In most of the cases when the time of leaching is longer a higher recovery is expected to be achieved. This is because there is enough time for the lixiviant to diffuse inside the particles, which can be explained based on the mixed kinetic model. However, there is a limit where the time is no longer effective. This condition is seen in the curve time vs recovery, the curve slope tends to 0. Therefore, the time to reach this maximum recovery is the optimum leaching time.

Although, the leaching time can improve the leaching rate, in some cases, such as in sulfate media, the rate of recovery may negatively be affected because of passivating behaviour of sulfates developed through time.

A column test is generally needed to obtain the recovery curve behaviour versus time. An example of such curves is shown in Figure 9. The leaching time for chalcopyrite at different temperatures is shown in this figure. At 65 °C the curve becomes parallel to the “x” axis between 80 and 100 min of reaction, which is the optimum time range for leaching recovery for this specific type of ore. However, this does not mean that for all different testing conditions or different product the recovery versus time is similar as the other leaching parameters such as the quantity of materials, porosity, mineralogy, might be different.

It has also been previously shown that the extraction of copper from actual waste heaps and dumps is roughly proportional to the square root of time [16]. This finding indicates three possible rate controlling mechanisms: (1) decreasing reaction surface area for individual waste particles, (2)

Increases in the diffusion rates of oxygen and ionic species through growing leached rims in the particles containing waste materials, and (3) diffusion limitations by product layers which are formed in individual sulfide grains. (without waste).

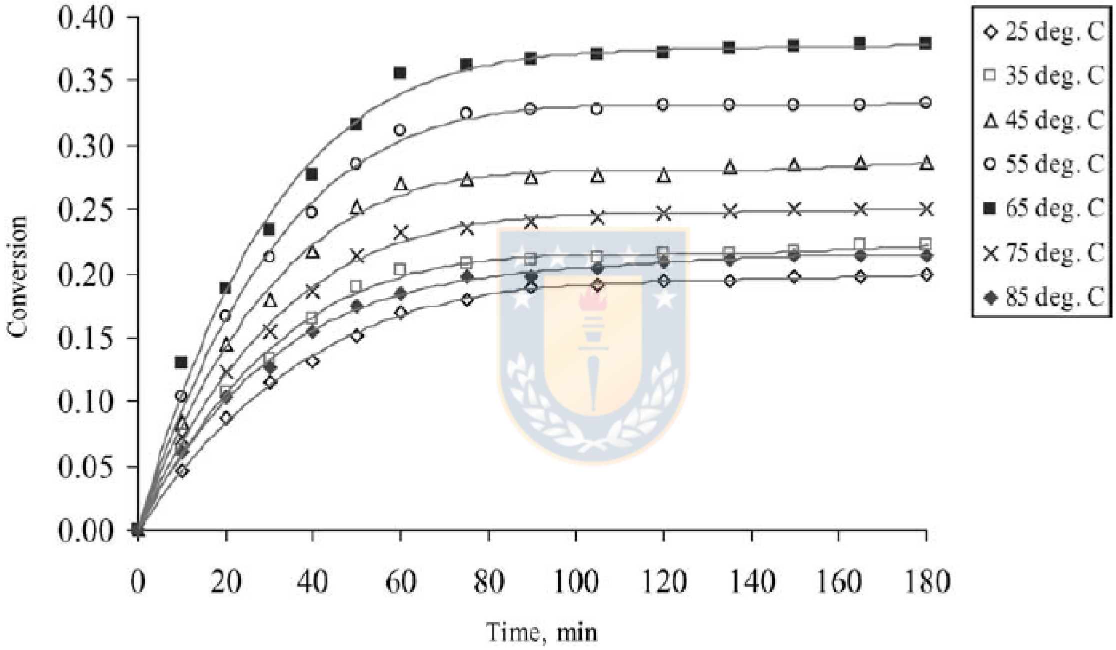


Figure 9 Plot of recovery vs time at various temperatures (0.1 M HCl, stoichiometric amounts).(Source: Dreisinger and Abed [13])

3.4 Temperature

As previously mentioned, temperature is a fundamental factor in leaching. Each lixiviant has a specific range of temperature where it works efficiently over a specific type of ore material. However, the rate of chemical and biological reactions can negatively be affected by the dissolution of gangue at certain temperatures. A cold temperature means lower recovery and throughput. This is especially observed with sulfides minerals. Therefore, in order to increase the temperature to accomplish a better recovery, one may suggest heating the feed solution and keeping the environment of the stope at a targeted and desirable temperature with ventilation.

The transport mechanisms of leaching solutions into the material must be considered as convection and diffusion. A recirculation of solution from ponds after solvent extraction should be implemented to keep the temperature constant and to reduce the energy consumption. As the lixiviant flows through the stope, the temperature is decreasing according to the heat transfer capacity of material in contact with the solution. In some cases, leaching of sulfides is assisted by heat-input from bio-leach or chemical leaching activities which can be significant for ores with reactive sulfides such as massive sulfides with high pyrite content.

Temperature also affects the chemical reactions in oxygen presence environment. Oxygen uptakes into solution across a gas-liquid interface which is a temperature dependent mass transfer step.

In general, underground environments present a raise in the temperatures as depth increases, therefore, the effects of depth on temperature should also be considered in analysing the performance of in-situ leaching. Normally, the geothermal gradient is 25 °C per km depth into the ground. This shall therefore be considered in the design of leaching stopes for In-Mine Recovery.

In general, the leaching rates of copper minerals, are affected by temperature. The activation energies (Ea) of chalcopyrite and bornite dissolution obtain and use considerable heat in leaching reactions. The activation energy is the least amount of energy required to activate atoms or molecules to a state in which they can undergo a chemical reaction [17].

There are also several controversies about bioleaching. For example, a study [11] showed that the bioleaching of chalcopyrite occurs faster when the temperature increases, especially above 55 °C. On the other hand, some micro-organisms, like T. Ferroxidants, show the best recovery in an optimum temperature range, depending upon the specific leaching, which can range from 25 to 45 °C, declining their activity markedly about 45°C[16]. Additionally, the passivation of chalcopyrite during bacterial leaching could be countered by controlling the temperature (45°-65° C) and electrochemical potential (0.45-0.65 V SCE) conditions. Consequently, the behaviour of the microorganisms in a specific environment must be studied on site before bioleaching.

For chemical controlled processes, the recovery can increase significantly by rising the temperature even just a few degrees. If the role of temperature is not important, it can be inferred that the kinetic of leaching is controlled by diffusion rather than by chemical reactions. Dreisinger and Abed [13] in chalcopyrite leaching have shown that an increment of temperature above 65 °C in a sulfate media produces interference due to passivation of mineral particle layer by reactions of sulfates, which have a dissolution behaviour when temperature rises. In other words, such gangue materials can encapsulate ore particles. This is well shown in Figure 9. In another test, a rough increase of reaction kinetics in acid chloride solutions was observed by rising the reaction temperature from 60° to 70°. However, raising temperature from 70° to 90° did not affect the leaching and the leaching rate remained stable [13]. It was additionally noted that the passive layers formed at 25 °C strongly inhibit ferric ion reduction on polarized chalcopyrite surfaces [11].

In an Underground mine environment, raising the temperature may have some negative impacts and adversely affect the working condition for the mining crew. This factor must be considered in the design of the IPM. The accepted standard of working conditions may differ in different countries. For example, in Chilean mines the maximum temperature in a working site; shall not exceed 32°C in a light operation [18]. In other words, this is the maximum temperature that could be accepted in an underground mine with low personnel involved in the operation. The main aim would therefore be to select a temperature which is optimum for leaching and can fulfil the working standard for the crew members. A higher temperatures environment can significantly affect the performance of the personnel and equipment and can impose serious risks. To achieve a desirable and safe working environment in the mine, having an effective and efficient ventilation system is highly required.

When there is an increase in the solution temperature, it may cause the evaporation of the solution, a condition that can negatively affect the yield of the solution and water consumption.

3.5 Leaching Solution

3.5.1 Generalities

There are different types of dissolution reactions available for the leaching of minerals. These reactions can generally be classified as physical, chemical, electrochemical, and electrolytic types. In this chapter, chemical reactions and their components are briefly studied and discussed.

The leaching solutions generally contain water, acidified water or acid ferric sulfate, depending on the type of lixiviant considered for different applications. The leaching solution can be recirculated to take advantage of the resources used for leaching. In order to recirculate the solution, it is necessary to extract the valuable metal and to pass the leachate through an oxidation basin, in which the lixiviant, bacteria, ferric iron, among other solutions are regenerated.

The extracted solution which contains the valuable metal and elements produced by reactions into stopes is called Pregnant Leaching Solution (PLS).

For stope leaching, it is important to make a good selection of leaching agents. This is because if the leaching solution contains hazardous materials (e.g., corrosive materials and gas emanation), then the process of in-situ leaching could be dangerous and may negatively affect the performance of the crew members as well as the equipment and subsequently the production.

Many lixiviants can be used for the leaching of copper, such as sulfates, chlorides, ammonia and nitrates. However, ferric sulfate and sulfuric acid are among the most common types because they are cost effective, easily accessible and their application is also simple. In most cases the selection of lixiviant can be a real challenge, especially for a heterogeneous mineralisation. This is because chemical reactions in different chemical compositions are not identical. Some typical examples of leaching reactants are presented in Table 3

Table 3 Typical leaching reactants commonly used in mining

(Source: Ruiz [3])

Classification	Reactant
Acids	• Sulfuric acid • Hydrochloric acid • Nitric acid
Bases	• Lime • Sodium hydroxide • Carbonate hydroxide
Complexing agents	• Ammonia • Cyanuric salts • Chloride salts • Carbonate salts
Oxidant agents	• Oxygen • Ferric salts • Hydrogen or sodium peroxides • Permanganate • Magnesium peroxide
Reductant agents	• Gas CO • Gas SO2 • Gas H_2

The concentration of lixiviant is also important in predicting the recovery of leaching processes, allowing a better recovery when the concentration is higher. By controlling the concentration of lixiviant, one can overcome the acid consumption which is affected by wastes or even one may reduce the time of leaching depending on the mineralisation of the ore. Figure 10 shows an example of the lixiviant concentration effect on the recovery of a leached chalcopyrite at 80° C.

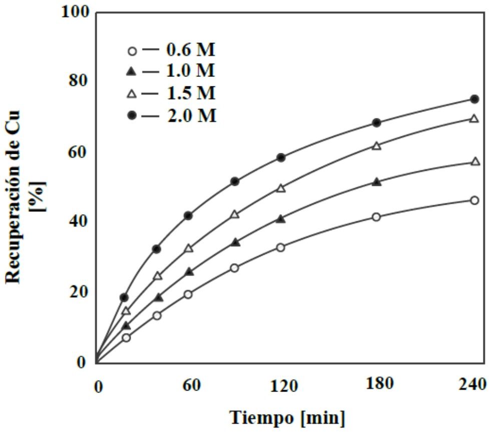


Figure 10 Copper recovery at different sulfuric acid concentrations (H_2SO_4) (Source: Bahamondez [1])

The concentration of lixiviant which is selected for irrigation into the underground stopes is a critical parameter directly affecting the operation costs. In other words, when a solution with a higher concentration of lixiviant is used, the costs associated with the utilisation of acid increase. Likewise, if a solution with less concentration will be used, lower operation costs may be expected. Therefore, selecting a suitable lixiviant/oxidant system can help to enhance the recovery rates and to reduce the lixiviant consumption. For example, curing with a strong lixiviant solution might be included even when agglomeration is not required, but curing may reduce impurity levels, such as silica, in some acid copper leaching systems, while in others it may result in higher lixiviant consumption. The other important factor that must be considered is the possible damage that can result from the consumption of acid on the equipment and the surrounding rock mass around the stopes. Such issues may impose operational problems and increase the cost for efficient leaching in stopes.

Pre-wetting of the ore with water, lixiviant and/or bacteria is also commonly used for crushed ore to minimise the migration of fines in the heaps, to initiate early leaching, to avoid dry areas, to promote bioactivity, and to avoid an adverse lixiviant concentration profile when leaching of a new lift started[19].

The oxidation or reduction of sulfides can be reached by chemical or biological processes. Principal oxidants are ferric and cupric ions, oxygen and bacteria. Some other typical examples of oxidants and lixiviants are shown in Table 4.

Table 4 Some oxidant/lixiviant system and their molar ratio dissolving Chalcopyrite(Source: Hidalgo, Beinlich [20])

Oxidant	Molar ratio CuFeS2	Lixiviant
FeCl3	16	HCl
Fe2(SO4)3	8	H2SO4
CuCl2	15	HCl
CuSO4	2	H2SO4
K2Cr2O7	2.8	HCl
H2O2	1%	Glycine

3.5.2 Typical Leaching media used in Chile

Some typical media used in Chile for copper leaching are Sulfate media and Chlorite media, they are described as following:

3.5.2.1 Sulfate media

Processes in sulfate media include acid pressure leaching, ferric sulfate leaching, bacteria leaching. Leaching reactions historically involve the presence of sulphuric acid, and mixed oxide-sulfide ores which have been leached since the early decades of the last century using sulphuric acid-ferric sulfate mixtures. Lixiviant is dilute sulphuric acid for oxide ores, and ferric sulfate/sulphuric acid for mixed and secondary sulfides ores. The presence of ferric sulfate may lead to side reactions that can be acid generating and may buffer the leach system by creating basic ferric sulfate precipitates.

3.5.2.2 Chlorite media

Another common lixiviant for sulphide ores is ferric chloride, which has the highest recovery rate compared with ferric sulfate types whose rate of dissolution decreases after a certain leaching time. Processes in chloride media include ferric chloride leaching, cupric chloride leaching, chlorine and hydrochloric acid leaching and so on. Catalytic formation of ferric with chloride ions has been developed as an alternative to bio-oxidation for mixed and sulfide ores. In general, it has been shown that introducing chloride can improve the rate of leaching [14].

3.5.3 Some interesting options for In-Mine Recovery.

3.5.3.1 Bioleaching

Bioleaching, bacteria leaching, or micro-organism leaching is a technology which is widely used for metal extraction from low-grade-ores and mineral concentrates. This technology is based on injecting bacteria into the rock or obtaining it naturally by the growing-up of indigenous bacteria under certain conditions. Metal recovery from sulfide minerals works using the activity of chemolithotrophic bacteria, mainly Thiobacillus ferroxidans and thiooxidants, converting insoluble copper sulfides into soluble metal sulfates. Bacteria leaching has best results where the environment is suitable to grow up micro-organisms. To prepare the suitable conditions nutrients, oxygen, $CO_{2,}$ pH, temperature and minerals are needed. Bacterial Leaching activities can be classified into direct or indirect actions.

In *direct bacterial leaching*, a physical interaction between bacterial cell and mineral sulfide surface occurs. In this process, pyrite is oxidized to ferric sulfate according to the following reactions:

$$4FeS_2+14O_2+4H_2O+\text{bacteria}\rightarrow 4FeSO_4+4H_2SO_4 \qquad (3)$$

$$4FeSO_4+O_2+2H_2SO_4+\text{bacteria}\rightarrow 2Fe_2(SO_4)_3+2H_2O \qquad (4)$$

The next reaction may be:

$$4FeSO_4+15O_2+2H_2O+\text{bacteria}\rightarrow 2Fe_2(SO_4)_3+2H_2SO_4 \qquad (5)$$

Therefore, direct bacterial leaching may also be described as the following equation:

$$MeS+2O_2+\text{bacteria}\rightarrow MeSO_4 \qquad (6)$$

Where MeS is metal sulfide.

In *indirect bacterial leaching* of sulfide minerals, the bacteria generate a lixiviant which chemically oxidizes the sulfide mineral, in cases that bacteria are not necessarily in direct contact with minerals. In this case, ferric iron is the key oxidising agent and soluble iron species are the main influential factors of redox potential. The oxidation type or chemical reaction depends on the mineral which is aimed to be leached. This reaction can be written as follow:

$$MeS+Fe_2(SO_4)_3\rightarrow MeSO_4+2FeSO_4+S^0 \qquad (7)$$

For example, in chalcopyrite leaching; the reaction is:

$$CuFeS_2+4Fe_3+\rightarrow 5Fe_{2+}+Cu_{2+}+2S^0 \qquad (8)$$

3.5.3.2 Hydrogen Peroxide

Hydrogen peroxide is powerful oxidant. The chalcopyrite dissolution for hydrogen peroxide is represented by the following reactions [21]:

$$CuFeS_2\rightarrow Cu^{2+}+Fe^{2+}+2S^{2-} \qquad (9)$$

$$2S^{2-}\rightarrow 4e^-+2S^0 \qquad (10)$$

$$2H_2O_2+4H+4e^-\rightarrow 4H_2O \qquad (11)$$

Hydrogen peroxide has a decomposition behaviour strongly dependent of the temperature, in fact, a study has been done to understand the rate of decomposition [21]. The results of the study showed that the hydrogen peroxide decomposition slowed down with the addition of ethylene glycol.

In summary, a higher recovery can be reached using hydrogen peroxide during leaching for longer periods of time.

3.5.3.3 Methane Sulfonic Acid (MSA).

Methane sulfonic acid, MSA, (CH_3SO_3H) appears as a good lixiviant option, because this lixiviant is stable at lower temperatures, unlike bioleaching and sulfuric acid, whose behaviours at low temperatures are inefficient in terms of recovery. MSA is also a good choice being an option to avoid hazardous issues, because this lixiviant does not have a corrosive performance and does not produces toxic gases being biodegradable. This organic acid has commonly been used in solution with ferric chloride.

3.5.3.4 Acidic potassium dichromate solution.

The dichromate ion ($Cr_2O_7^{2-}$) has been used recently for chalcopyrite leaching in acidic medium because of its higher oxidation potential compared with other oxidising leaching reagents.[22]

The dissolution of chalcopyrite by dichromate ion in sulfuric acid media is represented by the following reactions:

$$6CuFeS_2+5Cr_2O_7^{2-}+10H^+ \leftrightarrow 6Cu^{2+}+6Fe^{3+}+12S^0+10Cr^{3+}+35H_2O\text{, and} \qquad (12)$$

$$6CuFeS_2 + 17Cr_2O_7+142H^+ \leftrightarrow 6Cu^{2+}+6Fe^{3+}+12SO_4^{2-}+34Cr^{3+}+71H_2O \qquad (13)$$

However, further studies are needed to understand the behaviour of this reagent in large scale. Nevertheless, from reactor tests, it was noted that the rate of leaching increases when temperature and/or concentration of acid and potassium chromate increase. The rate of leaching was also negatively affected when the particle size was coarser.

3.5.3.5 Ammonia leaching

Ammonia leaching appears to be a good solution in cases where there is a high presence of carbonate rocks and iron minerals because of its high selectivity. Selectivity is defined as the solubilisation of desired metals and precipitation of undesirable iron in one unit operation [23]. However, ammonia can present problems in the case of ammonia transferring during the subsequent extraction process. For example, in solvent extraction, ammonia is strongly dependent on the type of diluent. This method has a higher cost due to higher number of stages for the process to extract the final valuable pure metals.

3.6 Hydrogen potential (pH).

The hydrogen potential, pH, has a noticeable effect in leaching kinetics. This is because pH affects the yield of chemical reactions, producing different mineral formations according to its value range.

Bacterial Leaching needs acidic conditions, the acidity is often increased by the chemical reactions of pyrite oxidation. The bacterial hydrophobicity increases as pH decreases.

The most common lixiviants used for chalcopyrite dissolution are ferric chloride and ferric sulfate, with a pH range of 1.0-3.0. When bioleaching is present, the pH must be at least below 5.0 to keep bacteria alive and produce the corresponding reactions

The ability of associated non-sulfide minerals to control the pH of the solution is the most important factor affecting the rate of oxidation of sulfides in a rock [9]. A higher acid presence may be neutralized in various acid consuming reactions of minerals such as carbonates or silicates. Natural neutralization of acid in the media by oxidation of associated non-sulfide present in the rock, can produce coating of the reaction surface of sulfides, by the formation of secondary minerals.

A basic pH is used in ammonia leaching. This is, because the reaction does not involve chemical interaction with carbonates, therefore, does not produce acid consumption. The pH for ammonia leaching is higher than 7, but, in next stages of the process, such as washing or extraction of the valuable metal, a pH below 3 is needed to optimise the yield of the process.

3.7 Height of the Pile of Broken Rock

Studies [24, 25], have shown that the height of dump, heap, or on this specific case study, stope, affects the rate of recovery, having an inverse relation between height of the pile and rate of recovery. The problem is explained by the increase of material into the mineral load, which cause numerous physical interactions between particles inside of the pile, and longer times of solution and/or air diffusions between the higher and lower part of the stopes. In fact, to avoid solution diffusion issues along the stope, much more leaching solution must be irrigated through the stope. One alternative is creating technologies such as injection pipes installed into the material load (broken muck pile) to ensure that the leaching solutions reach all the mass of material inside the stopes. An example of the copper extraction behaviour vs heap height in heap leaching is shown in Figure 11, where the copper extraction is negatively affected by the height of the heap.

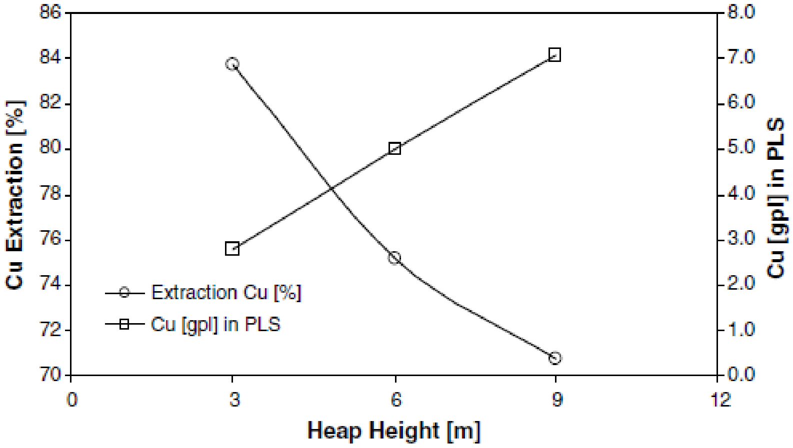


Figure 11 Recovery from leaching stage and copper concentration in the PLS, as function of heap height

(Source: Lizama, Harlamovs [25])

3.8 Irrigation Rate

Recovery of the metal elements in leaching is directly proportional with the irrigation rate [25]. When the rate of irrigation is higher or more frequent, the solution reaches better the entire stope. The only constrain is the presence of channels or ponding areas produced by the action of fines or waste. The ponding of solution is produced when an excess of solution is injected, producing a saturated environment, which can produce a decrease of gas transference. Therefore, less leaching microorganism activity is generated, reducing the rate of leaching. Figure 12 shows an example of different irrigation rates in a leaching test. The figure shows a reduction of copper recovery when the material was excessively irrigated.

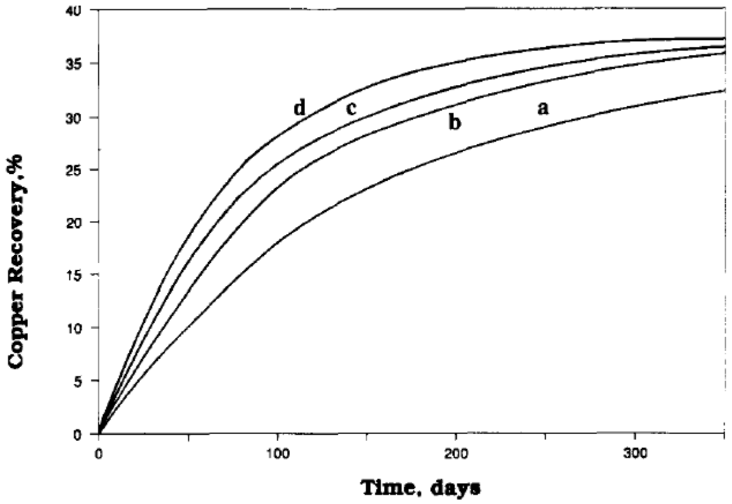


Figure 12 Effect of the irrigation flowrate (Ls) on the predicted copper recovery versus leaching time. Ls (1 m^-2 h^-1): a:69.00, B:24.00, c:13.88, d:3.47 (Source: Neuburg, Castillo [26].)

Table 5 shows the effects and limiting conditions of the ratio between the irrigation rate L and the height of muck pile, h, in a bioleaching media.

Table 5 Influence of L/h for bioleaching in columns or heaps.

Source: Lizama, Harlamovs [25]

	Effect	Limiting conditions
Increase L/h	• Faster bioleach rate • Faster mineral colonization • Faster pH breakthrough • More sulphur oxidation/acid generation • Decreased selectivity in sphalerite vs pyrite bioleaching • Faster net ferric generation	• Flooding • Decomposition of agglomerate • Migration/washout of fines
Decrease L/h	• Increased selectivity of sphalerite vs pyrite bioleaching • Less sulphur oxidation/acid generation • Slower net ferric generation • Slower ph breakthrough • Slower mineral colonization • Slower bioleach rate	• Acid limitation • Excessive ferric precipitation

It might be possible to irrigate a 240 m height stope by sprinkling; it may also be possible to irrigate the stope through injecting the solution through pipes installed along the mass of the broken material. The spacing between pipes affects positively the reaching of the mass of material when the pipes are closer from each other. Pipes will be grouped as injection and extraction pipes; These leaching pipes are distributed in geometrical patterns that are suitable for arranging a reliable distribution of leaching solution.

3.9 Aeration

The aeration may allow the oxygen to go through the broken material producing the needed oxidation in particles of ore. Indeed, a rock configuration for good distribution of the solution and an availability of oxygen are required to carry out the leaching reactions. This is because sulfides must be oxidized to water-soluble sulfates. In case of sulfide minerals especially chalcopyrite, which is the most abundant mineral in the case study of the research; an oxidant is needed. In most of the cases this oxidant is oxygen, which in dumps environments is obtained from the convection of air. In stope leaching, where there is a lack of oxygen, it is essential to provide enough air to the mass of broken material by blowers/fans (forced aeration, also called advection). Therefore, an optimized ventilation system should be implemented with the purpose of carrying oxygen for the chemical reaction of sulfides within the material and to extract the possible gases formed in the processes. Oxygen is a key reactant in bioleaching, as the microbes oxidize ferrous iron and reduce sulphur species. A lack of oxygen causes recovery loses, situation that might change the selection of the oxidant to other option, such as cupric and ferric ions, among others.

3.10 Hydrogeology

3.10.1 Permeability

Fluid flow is a critical factor controlling the metal extraction efficiency in leaching. The movement of fluid carries reactants and products from ore particle surfaces to the exterior. For this reason, it is very important to consider the permeability of the target materials which has been considered for leaching. A material has permeability when a certain quantity of fluid can go/pass through it. Otherwise a material is not permeable when the quantity of fluid that go through the material is very limited. Higher permeability enhances the rate of recovery. When the material has lower permeability, fragmentation of the material must be applied to allow the solutions to penetrate the rock in order to achieve a better recovery and to reduce the time of leaching and to obtain a good leaching yield. If the chemical reaction at the surface of material is faster, the concentration gradient is steeper, and reagent is consumed before it penetrates the particle. The porosity is a particle or material property, defined as empty spaces inside a mass of rock or soil. A solid/liquid contact area for rocks with low porosity is mainly dependent on the particle size. In the presence of low porosity, the internal surface of the material cannot be involved in the leaching reactions, and the particle size is the dominant factor controlling the recovery in low porosity rocks.

The formula that represent permeability is:

$$k = \frac{\epsilon}{\o\, A_s^2\, \iota} \tag{1}$$

Where k is permeability, ϵ is porosity, which is defined as the empty space in a particle, $\o$ is shape factor, A_s is Internal surface and ι is tortuosity.

Where A_s can be expressed as a function of hydraulic ratio:

$$A_s = 1/R_h \tag{2}$$

Hidalgo [27], mentioned that it is possible to enhance permeability using staged leaching [28], by microcracking; a chemical action over particles. Furthermore, chemical reactions can change the texture of the product phase. The formed phase produced due to the chemical interaction is porous and allows the fluid to circulate beyond the original phase. In sulfide deposits environments, Chalcocite that is formed after acidic leaching of bornite, showed surface cracks and porosity in the rim.

The hydraulic conductivity of the fluid affects the solution transport through the material. This factor affects the leaching time and kinetics of the processes. The hydraulic conductivity K_{fr} is defined by Corey [29] as:

$$K_{fr} = \frac{\rho g a^2}{12\mu} \tag{3}$$

where ρ is fluid density (kg/m^3), g is the acceleration due to gravity (m/s^2), a is the aperture opening (m), and μ is the dynamic viscosity (kg/s·m)

3.10.2 Segregation and compaction

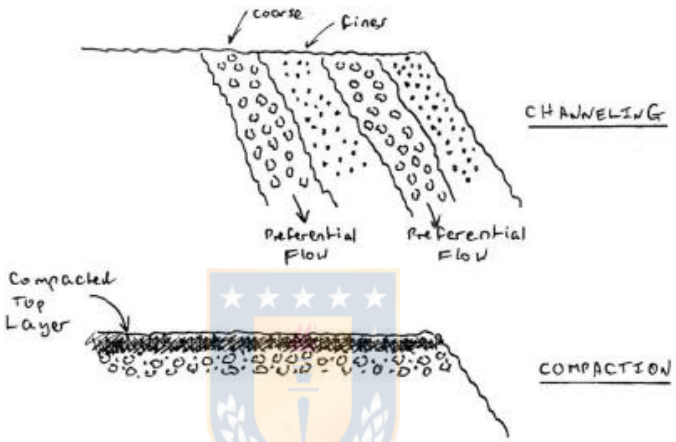


Figure 13 Leaching issues: Channelling and compaction. (Source: ALTA conference 2019)

The segregation of fine material in heaps or stopes leads to preferential flow of solutions through the coarse areas. Such a heterogeneity of particle sizes in the stope may adversely affect the leaching efficiency and the PLS concentration.

The compaction of the heap/stope at the surface or bottom can result in reduction in the achievable leach solution irrigation rate, uneven flow through the material, and, if severe, completely plug portions of the heaps/stopes. The effect is generally worse for ores that contain clay minerals. Compaction of fines around coarser particles interferes with circulation around the ore particles and affect the rate of leaching.

Effective pressure on material is higher when the material over the target areas contains some liquid (weight is higher).

A study [30] shows that the saturation of the material within the heap or stope is directly related to the dominant particle sizes of the rock materials. Therefore, it can be concluded that the leaching is controlled directly by the PSD.

4. FRAGMENTATION ARRANGEMENTS

The fragmentation of the rock material inside a stope can be obtained using blasting operation. The methodology used for the blasting design to achieve a desirable PSD distribution, for enhancing the rate of recovery in leaching, is discussed in this chapter of this Thesis.

4.1 Blasting

One of the main techniques which for long time has been used to fragment a rock mass is Rock Blasting. Rock Blasting is one of the most important operation used for the excavation of hard rock, especially in mining and tunnelling. To date, the design of rock blasting has mainly been based on experience. This shows that blast design could be challenging and sensitive, especially due to the existence of numerous different cases of implementation. The main goal of rock blasting in IMR is the fragmentation of rock masses to particles of suitable shapes and sizes in order to enhance the leaching of ore materials. If the blast design cannot be properly done, it may result in a poor blast outcome. Technically and economically speaking, such a poor fragmentation might be due to inefficient use of the explosives energy and can cause, safety, issues in the mine [31]. Indeed, rock fragmentation using blasting affects the efficiency, productivity and the cost of the downstream operations including loading, crushing and grinding [7]. Poor drilling and blasting practices may result in excessive over-break, ore dilution, oversize fragmentation, restricted access and increased local support and reinforcement requirements. It can also lead to increased mining cycle times and costs and may have several negative effects on the efficiency of mining activities [32] .

Additionally, to understand and improve the blasting induced rock damage, it is necessary to know how the zones around the blasthole are affected by blast induced loads. There are three main zones with different levels of fragmentation around a blasthole, defined by Persson et al. [33].These zones are: crushed zone, fractured zone and elastic zone in which the fragmentation is mainly controlled by the natural discontinuities in the rock mass.

The annular rock region immediately around a blasthole is known to experience a significant shock wave, where the rock is intensely crushed and shattered [34]. This area is where most of the fines are generated in rock blasting. The formation of fines (crushing) and fracture creation are important factors in rock blasting and consume a considerable proportion of the explosive energy. In terms of leaching, the presence of fines causes solution channelling and deficiency of oxygen which inhibits the rate of recovery. Therefore, a blasting design with horizontal drill holes may produce preferential flow or ponding due to the distribution of the different fragment sizes in the area. Layers of fine and coarse fragmentation may be generated (see Figure 14). A proper design of the blastholes can improve the flow of solutions due to the heterogeneity of the particles size position and the gravity of the flow; this controls the equivalent permeability of the rock (flow through layers).

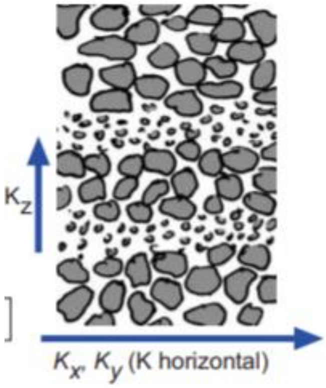


Figure 14 Scheme of layers generated with coarse and fines position. This could be produced by horizontal blasting

Dowding and Aimone [35] and Kanchibotla and Valery [36] defined the crushed zone as the region between the circumference of the initial borehole and the circumference of the post blast borehole, Cunningham [37] defined this as borehole expansion zone. In controlled blasting techniques, the size of crushed zone surrounding the blasthole is related to the decoupling ratio and decreases with increasing the de-coupling ratio [38]. In the fractured zone, the rock breakage is generated based on interaction between the blast induced tensile waves and the in-situ block size distribution defined by natural discontinuities. A coarser fragmentation is expected to be observed in this fragmentation region [36]. Figure 14, schematically shows the three zones of fragmentation.

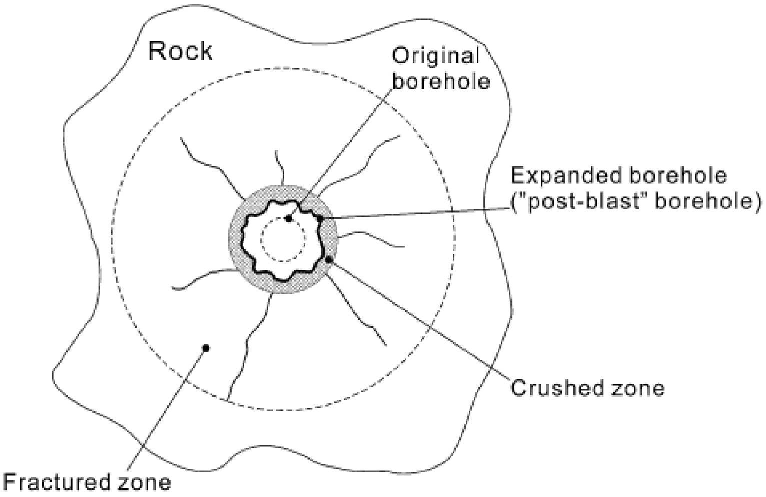


Figure 15 Definition of damage zones surrounding the borehole (Source: Chi, Zhang [38])

Free surfaces are required for an efficient blast. They play a dominant role in the results of blasting, such as the fragmentation and the distribution of the post-blast muck pile [39]. Indeed, to achieve an efficient fragmentation a free face must be created before the blast operation.

There are a few parameters which are considered in blast design in practice. Typical examples of these parameters are powder factor, diameter of blast hole and confinement provided by the elastic properties of the surrounding mass, and the rock mass dynamic parameters (e.g., dynamic compressive and tensile strength). For a given formulation, a larger diameter of blast hole and greater confinement generally result in less energy losses from the reaction zone, which can produce a higher velocity of detonation (VoD). On the other hand, a higher powder factor means that the explosive contains more energy (larger mass) which would produce a stronger detonation.

Table 6 shows the factors that are considered for the design of an efficient rock blast to achieve a desired fragmentation.

Table 6 Modelling stages.

Physics	Deliverables/Influences
• Ideal detonation • Non-ideal detonation • Wave propagation • Stresses/Strains • Gas Flow • Fracture/fragmentation • Cooperative motion	• Pressure/VoD • Energy release • Seismicity • Cracks • Rock fragments-size distributions • Vibration and Damage • Airblast • Muckpile-shape and swell • Ore delineation • Flow and draw • Flyrock • Fume and dust

Generally, in a blast design model, three sequence of stages is considered:

1) Selection of detonation to make sure that the explosive is efficient for usage in the environment.

2) Assessing the energy transfer from the explosive to the rock and identifying how the rock responds, to estimate a certain PSD.

3) Analysing the bulk movement of the fractured mass to consider the volume of material to be leached at the stopes.

4.2 Underground Blasting Design

Underground ring design can be done following the geotechnical conditions of the mine. The remaining rock mass such as pillars, should always be considered because they are directly or indirectly affected by the blasting operation.

To calculate the PSD generated by blasting and to apply the technical constrains, a clear understanding of the blasting design into a stope is needed. To do so, the author and his supervisor developed four underground blasting design models to study the effects of the differences between them. The four blasting designs are:

- Central Swedish Ring Drift (CSRD): five galleries are made into a stope, four for production and one for blasting of the drawbell zone , the blastholes are drilled in a ring shape around the galleries [40].

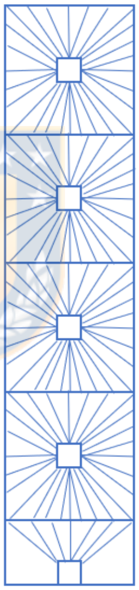


Figure 16 Central Swedish Ring Draft (CSRD)

- Cylindrical Stope (CS): A free face is created to permit the drilling of rings around a circular shape shaft in the center of a stope. Such a shaft can be built by using a raise borer machine [41].

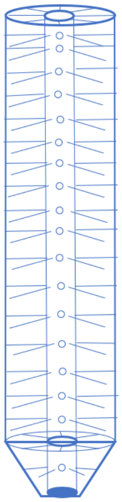


Figure 17 Cylindrical Stope (CS)

- Diamond Ring Draft (DRD): This method considers the creation of galleries, presenting two galleries per sublevel and drilling blastholes in a diamond shape in order to improve the blasting accuracy [42].

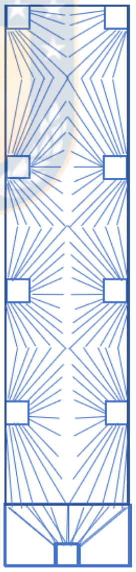
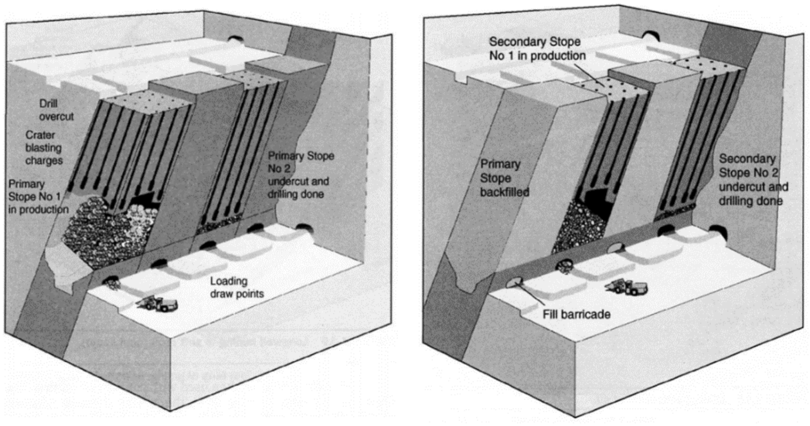


Figure 18 Diamond Ring Draft (DRD).

Vertical Crater Retreat Draft (VCR): In this method, large blast holes are drilled vertically either from the bottom or over the top of the stope, including three sublevels per stope. For longer heights of stopes, it is needed to consider an extra support below the bottom of the stope due to the stability problems. Vertical Crater Retreat (VCR) is implemented in steeply dipping orebodies in competent host rocks. As it can be seen in Figure 19, the holes are drilled nearly vertical downward through to the undercut. Crater blasting technique is used in VCR method, which uses powerful explosive charges in large diameter holes where holes are drilled vertically where possible [43].



(a) (b)

Figure 19 Vertical Crater Retreat Draft (VCR) a) Primary stopes b) secondary stopes. (Source Hamrin [43])

The four different methods mentioned before have pros and cons that are considered for the leaching operation, fragmentation yield, costs of the operation and the stability of the stopes. These characteristics are shown on Table 7.

Table 7 Comparison of methods selected for the study

Method	Advantages	Disadvantages
Central Swedish Drift	• A better blasting yield is achieved well-delimiting the stope, getting finer PSD. • CSRD eases the leaching solution injection due to the shape of stope.	• More number of developments are needed to start the production of blasting operations (less than diamond ring design). Therefore, is a more expensive method. However, the drilling and blasting cost is cheaper than other methods.
Cylindrical Stope	• Low drilling and blasting costs. • Does not use sublevels.	• Requires a high investment for the raise bore machine. • A portion of the material remains at the corners of the stope (compared with square stope), reducing the mine recovery. • It can have operational loses due to the damage induced by blasting and the leaching flow through the height of stope (240 m). This is worse due to possible layers formations.
Diamond Ring Drift	• A better blasting yield is achieved well-delimiting the stope and getting finer PSD. • DRD eases the leaching solution injection due to the shape of stope.	• Higher number of developments are needed to start the production of blasting operations. Therefore, DRD is a more expensive method than CSRD.
Vertical Crater Retreat Draft	• Can produce a good distribution of the position of the fragments due to the results of a vertical blasting assessment, where the lixiviant can flow easily. • It would assure the swell support. • It needs a smaller number of sublevels than central Swedish draft. Therefore, it has lower costs. Solutions can be injected from the bottom of the stope.	• Poor stability, therefore, it is needed to combine with other mining methods as room and pillar to bring support at the base of the stope.

4.2.1 Effects of Rock Mass Mechanical Parameters on Its Blastability

A proper characterization of the rock mass is always needed to have an estimation of the rock mass geological and geo-mechanical parameters of rock masses on blast results. Perhaps the rock strength, the rock density and the rock structures are the most dominant rock mass characteristics affecting the outcomes of blast fragmentation. The rock structures control the in-situ block size distribution in the rock mass. The rock strength shows the resistance of the intact rock blocks against the blast loads and the rock density indicates the amount of forces needed to move the broken rock fragments. The effects of these parameters are briefly explained in this section.

4.2.1.1 Block Size Distribution

It is always difficult to map the rock characteristics in underground environments. There exist a few empirical approaches that especially developed to estimate the particle size distribution of in-situ blocks. The implementation of such empirical models may provide a good description of the rock mass integrity. The in-situ block size distribution may also be estimated based on digital image processing of drill cores or image analyses of the surfaces of underground excavations. In this study, the following equation, developed by Hardy, Ryan [44], will be used to calculate the in-situ block size distribution in a rock mass:

$$P(x) = 100[1 - f_f^3 \exp\left(-f_f x\right)\left(\frac{x^3}{6} + \frac{x^2}{2f_f} + \frac{x}{{f_f}^2} + \frac{1}{{f_f}^3}\right)] \tag{4}$$

where f_f is fracture frequency, *x* is the size of the particle in meters and *P(x)* is the passing pecentage at that size (%).

4.2.1.2 Blastability Index

The approach proposed by Scott and Onederra [45] is also used to estimate the rock mass blastability which will be shown by Kuz-Ram rock factor, *A*.

$$A = 11.5 \times (\text{Strength Factor} \times \text{Structure Factor} \times \text{Density Factor})\text{-}1 \tag{5}$$

The strength factor, f_s, density factor, f_d, and rock structure factor, f_{st}, can be estimated as:

$$f_s = 0.0549 \times \sigma_c^{0.5315} \tag{6}$$

$$f_d = 0.0371 \times \rho_r^2 - 0.0512\rho_r + 0.9172 \tag{7}$$

$$f_{st} = -0.064Ln(f_f) + 0.9819 \tag{8}$$

where σ_c (UCS) is the rock strength (MPa), ρ_r is the density of rock in ton/m^3.

4.2.2 Ring Design

The main geometrical parameters for each kind of blasting design method are presented in Table 8:

Table 8 Geometric parameters of stope for underground blasting design.

Parameter	Symbol	Unit	Value
Central Ring Draft			
Stope Height	H_s	m	240
Stope Width	D_s	m	40
Stope Length	R_s	m	40
Drift Height	D_{sh}	m	5
Drift Width	R_{sh}	m	5
Cylindrical Stopes			
Stope Height	Hs	m	240
Stope Diameter	Ds	m	40
Stope Radius	Rs	m	20
Shaft Diameter	Dsh	m	5
Shaft Radius	Rsh	m	2.5
Diamond Ring Design			
Stope Height	Hs	m	240
Stope Width	Ds	m	40
Stope Length	Rs	m	40
Drift Height	Dsh	m	5
Drift Width	Rsh	m	5
Vertical Crater Retreat Stopes			
Stope Height	Hs	m	240
Stope Width	Ds	m	40
Stope Length	Rs	m	40
Top Sill Height	Dsh	m	5
Bottom Sill Height	Rsh	m	5

The following parameters were used for the explosives that are charged into the drillholes:

Table 9 Charge parameters.

Parameter	Symbol	Unit	Value
Charge Type	-	-	Emulsion
Density of Charge	ρ_e	(kg/m^3)	1000
Velocity of Detonation	VOD	(m/s)	5700
Relative Weight Strength	S_{anfo}	(%)	107

4.2.2.1 Central Ring Draft and Diamond Ring Draft

For the considered blast designs, it is assumed that the separation between sublevels is 60 m, and the length of up holes is 20 m, and down holes are around 40 m in length. The length of the buffer zone to avoid extra damage over stope walls is around 2 m, with a blasthole of 102 mm diameter. The explosive is fully coupled with the walls of the blastholes.

The recommended Burden (m) by Rustan [46] is estimated as:

$$B = 8.5 * \left(\frac{\emptyset_h}{1000}\right)^{0.525} \tag{9}$$

Additionally, the values of Spacing, *S* (m), Stemming, *St* (m), can be calculated as:

$$S = 1.15B \tag{10}$$

The recommended stemming (m) by [47, 48]

$$St = 50 * (\emptyset_h/1000) \tag{11}$$

And, for Diamond and Central Drift Designs, the powder factor, q (kg/m^3), is calculated based on the equation proposed by Gustafsson [40]:

$$q = q_0 + 0.03 * (greatest\ drilling\ length) + \frac{0.4}{width\ of\ round} \tag{12}$$

The q_0 value is selected according to Jimeno, Jimeno [42], in Table 10.

Table 10 Base Powder Factor

Description	Value (kg/m3)
Fissured and hard rock	0.6
With joints	0.55
Fractured	0.5
Relatively homogeneous	0.45
Homogeneous and hard	0.4
Soft and homogeneous	0.35
q_0 Considered for this study	**0.4**

The summary table with the ring design parameters for CSRD and DRD is as following:

Table 11 CSRD and DRD parameters

Parameter	Symbol	Unit	Value
Buffer Zone Length	L_{bz}	m	2
Distance of Levels	Dl	m	60
Length of Up Holes	Luh	m	20
Length of Down Holes	Ldh	m	40
Blasthole Diameter	$Ø_h$	mm	102
Blasthole Radius	R_b	mm	51
Charge Radius	R_c	mm	51
Spacing/Burden Ratio	S/B	(-)	1.15
Burden1 – Recommended	B	m	2.6
Spacing – Recommended	S	m	2.9
Stemming -Recommended	St	m	5.1
Minimum Up Hole Length	$L_{min\text{-}up}$	m	15.5
Maximum Up Hole Length	L_{max_up}	m	22.7
Average Length of Up Holes	$L_{av\text{-}up}$	m	19.4
Minimum Down Hole Length	$L_{min\text{-}d}$	m	15.5
Maximum Down Hole Length	L_{max_d}	m	39.2
Average Down Hole Length	$L_{av\text{-}d}$	m	31.6
Primary Powder Factor Estimation	q_p	(kg/m^3)	1.6
Volume of the Stope Between Levels	Vs	(m^3)	96000
Total Mass of Charge per Level	$M_{ch\text{-}p}$	kg	152215.2
Number of Rows	N_r	(-)	14
Mass of Charge per Row	$M_{ch\text{-}r}$	kg	10872.5
Mass of Charge per Up Holes	$M_{ch\text{-}up}$	kg	3624.2
Mass of Charge per Down Holes	$M_{ch\text{-}d}$	kg	7248.3
Average Up Hole Charge Length	$L_{c\_uh\_a}$	kg	14.3
Average Mass of Charge per Single Up Hole	$M_{ch\text{-}up}$	kg	117.0
Number of Holes in an Upper Ring	$N_{h\text{-}ur}$	(-)	30
Average Down Hole Charge Length	$L_{c\_dh\_a}$	m	26.5
Average Mass of Charge per Single Down Hole	$Q_{ch\text{-}dh}$	kg	216.2
Number of Holes in a Lower Ring	$N_{h\text{-}lr}$	(-)	33.0
Total Number of Holes Per Ring	$N_{h\text{-}tr}$	(-)	63.0

4.2.2.2 Cylindrical Stope Draft

The blastholes in CS can be wider than CSDR and DRD. This is because the explosive charge is loaded horizontally, whose orientation warrant the stability of the charges. The diameter of the blasting hole in this case could be considered as 112 mm.

The recommended burden (B) by Konya is:

$$B = 30 * (Ø_h/1000) \tag{13}$$

And the spacing (m) is:

$$S = 1.15B \tag{14}$$

The recommended stemming (m) on this case is:

$$St = 20 * (Ø_h/1000) \tag{15}$$

And the powder factor is calculated by:

$$q = \frac{M_{ch}}{V_{bz}} \tag{16}$$

The summary of the recommended blast design for the CS is schematized in Table 12 and shown in Figure 20

Table 12 Ring design for CS

Parameter	**Symbol**	**Unit**	**Value**
Buffer Zone Length	L_{bz}	M	2
Hole Length	L_h	M	15.5
Blasthole Diameter	$Ø_h$	mm	112
Blasthole Radius	R_b	mm	56
Charge Radius	R_c	mm	56
Burden - Recommended	B	M	3.4
Spacing - Recommended	S	M	4.4
Stemming -Recommended	St	M	2.2
Charge Length	L_c	M	13.3
Mass of Charge per Hole	Q_{ch}	kg/m^3	130.6
Shaft Area	A_{sh}	m^2	19.6
Stope Area	A_s	m^2	1256.6
Charged Area	A_{ch}	m^2	998.2
Blast Area	A_b	m^2	1237.0
Number of Blastholes	N_h	(-)	15
Alternative Hole Number	AN_h	(-)	16
Differential Angle between Holes	Df	(Deg)	22.5
Blast Zone Area	A_{bz}	m^2	77.3
Blast Zone Volume	V_{bz}	m^3	259.8
Powder Factor	q	kg/m^3	0.50

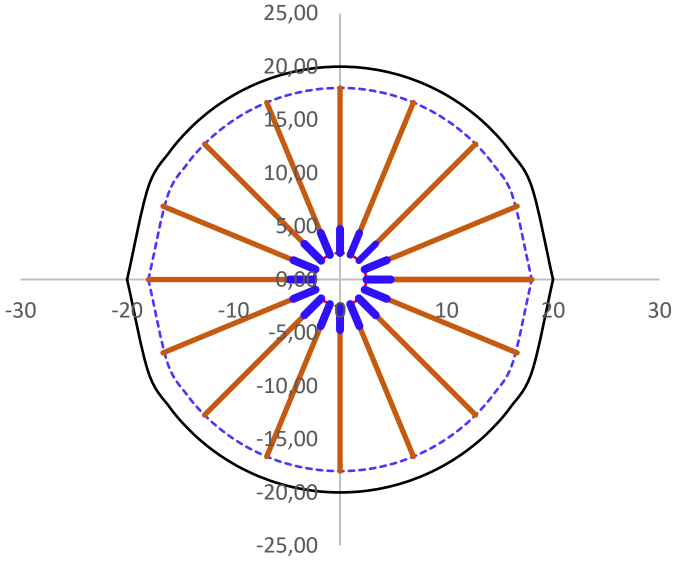


Figure 20 Scheme of the CS design from a plant view. Blue: Stemming; Red: charge; Pointed blue: buffering; Black circumference; bounds of the stope.

4.2.2.3 VCR Draft

The VCR method basically design by drilling blastholes vertically into the stope, having a blasting pattern design with a rectangular shape. The blastholes are drilled in a vertical position with the stemming at the upper part of the hole. In this method, it is possible to charge huge quantities of explosive, avoiding problems with the support of the charge. This advantage is due to the explosive support which is given by the surrounding material around blastholes.

Burden (m), here, is calculated by the following equation proposed by Jimeno, Jimeno [42]

$$B = B_{max} - 2 * \left(\frac{Ø_h}{1000}\right) - 0.02 * L_h \tag{17}$$

And the maximum burden, B_{max} (m), is estimated by the formula proposed by Kou and Rustan [49]:

$$B_{max} = 0.958 * (Ø_h/1000) * \sqrt[2]{((\rho e * S_{anfo}/100)/(MRC * BIF * S/B))} \tag{18}$$

Then, stemming (m) is calculated as following:

$$St = 0.7B \tag{19}$$

And MRC and BIF are selected from Table 13 and Table 14:

Table 13 Langefors rock constant

Langefors Rock Constant ©, LRC	**Value (kg/m3)**
Brittle crystalline	0.2
Most of rock types	0.35
Hard Rocks, like Swedish granite	0.4
Selected for this study	0.35
Modified Rock Constant (MRC)	1.1

Table 14 Blasthole inclination factor

Blasthole Inclination Factor, BIF	**Value (kg/m3)**
Vertical Blastholes	1
Inclined Blastholes 3:1	0.9
Inclined Blastholes 2: 1	0.85
Selected for this study	0.9

where:

$$MCR = LRC + 0.75 \tag{20}$$

The powder factor is calculated as following:

$$q = \frac{Q}{B * S * Lh} \tag{21}$$

Finally, the summary of the ring design for VCR is presented in Table 15

Table 15 VCR Ring Design

Parameter	**Symbol**	**Unit**	**Value**
Buffer Zone Length	L_{bz}	m	2
Hole Length	L_h	m	80
Blasthole Diameter	$Ø_h$	mm	165
Blasthole Radius	R_b	mm	82.5
Charge Radius	R_c	mm	82.5
Spacing/Burden Ratio	S/B	(-)	1.15
Burden – Recommended	B	m	2.9
Maximum Burden	Bmax	m	4.8
Spacing – Recommended	S	m	3.4
Stemming -Recommended	St	m	2.0
Maximum Blast Length to Satisfy VCR	Lvcr	m	1.0
Length of Charge in Holes	L_c	m	78.0
Mass of Charge per Hole	Q_{ch}	kg/m^3	1667
Stope Area	A_{sh}	m^2	1600
Charged Area	A_b	m^2	1296
Blast Zone Area	A_{bz}	m^2	9.8
Number of Blastholes	N_h	(-)	132
Alternative Hole Number	AN_h	(-)	120
Overall Blast Zone Volume	V_{bz}	m^3	782.2
Powder Factor	q	kg/m^3	2.13

4.2.3 PSD Assessment

To be able to estimate the recovery of leaching, it is needed to predict the Particle Size Distribution of the blasted muckpile. There are several different approaches can be used for predicting the PSD of the blasted muckpile. One of the most accurate approaches for predicting the PSD of a muckpile is the Swebrec model. This approach is explained in this section.

4.2.3.1 Swebrec Function

It is necessary to quantify the fragmentation and to identify the particle sizes according to the blasting design. There are a few equations that represent the PSD as a useful input parameter.

Studies on the sieved size distribution of blasted rock have shown that Swebrec function will reproduce the PSD quite well in the size range 1–500 mm. Swebrec function describes very well ~~describes~~ many sieved size distributions of fragmented rocks. The Swebrec function has three parameters: the average (median) fragment size x_{50}, the maximum fragment size x_{max} and the curve undulation parameter “b” [50].

The equation required for the calculation of PSD; known as Swebrec function; Ouchterlony [51] is shown below:

$$P(x) = \frac{1}{\left\{1 + \left[\frac{\ln\left(\frac{x_{max}}{x}\right)}{\ln\left(\frac{x_{max}}{x_{50}}\right)}\right]^{b}\right\}} \tag{22}$$

The parameter b is calculated according to equation developed by Ouchterlony [50]:

$$b = 0.5 * x_{50}^{0.25} * \ln\left(\frac{x_{max}}{x_{50}}\right) \tag{23}$$

where:

X_{max}= min (in situ block size, S or B)

X_{50}= median or 50% size of passing (mm)

$$x_{50} = AQ^{\frac{1}{6}}\left(\frac{115}{S_{ANFO}}\right)^{\frac{19}{30}}/q^{0.8} \tag{24}$$

Q= charge weight per hole, kg; q= powder factor or specific charge, kg/m^3; and S_{ANFO}=Weight strength relative to ANFO. In this case a modified mean size, mx_{50} is used instead of x_{50}.

The rock friability factor and delay time factor are needed to calculate the modified Kuz-Ram mean size, the Table 16 and Table 17 show the selected values for each one.

Table 16 Rock Friability Factor (RFF)

Description	Value
Very friable massive sulfide, mainly pyrite	0.5
Moderate friable massive sulphide	0.7
Semi massive sulphide	0.9
Granite or limestone	1
Selected RFF	0.9

Table 17 Delay Timing Factor (DTF)

Description	Value
Fast initiation: < 25 ms/hole, < 100 ms/ring, > 4 rings/blast	0.9
Medium firing speed: 50-100 ms/hole, 100-200 ms/ring, 2-4 rings/blast	0.95
Slow initiation: > 100 ms/hole, > 200 ms/ring, 1 ring/blast	1
Selected DTF	0.95

Finally, mx_{50} is calculated with the following equation:

$$mx_{50} = RFF^2 * DTF * x_{50} \tag{25}$$

4.2.4 Cost estimation

The fragmentation costs per 240x40x40 m stope of each kind of blasting method are summarized in the following table:

Table 18 Fragmentation cost per stope for the four different blasting designs

Total Fragmentation Cost	Value ($/Stope)
CSRD	3,526,307
CS	1,966,772
DRD	3,910,307
VCR	4,105,573

*The details for the cost estimation are in appendix B 11.2.

4.2.5 Blasting Design Selection

As mentioned earlier, some fragmentation measuring parameters, such as PSDs, are needed to calculate the rate of recovery for leaching. Therefore, a proper underground blasting design is selected in this chapter considering the assumption that smaller particles can result in a higher recovery as they produce more exposed areas. The comparison of the four different designs was executed according to the given recommended parameters, which were previously developed by Mining3. Although a modification of parameters can be done to improve fragmentation, there are some operational problems that would be presented with this action. For example, the majority of the parameters are depending on burden and diameter of blasthole. It is understood that a reduction of burden and spacing can results in the reduction of the size of the fragmented material, but an excessive damage of the surrounding zone would adversely be produced, causing the probability of leakage in stopes. Another example of operational issues is where the blasthole diameter is increased, in this case the risk is the lack of stability and support of the emulsion or explosive inside the blasthole due to very long vertical blastholes containing a high charge of explosive.

Note that it is not needed that all the blasting design options may have same burden or powder factor, therefore, they follow the design parameters shown on chapter *4.2.2 Ring Design.* It is also assumed that the fragmentation methods adopted from conventional mining can be used for IPM with simple modifications.

The four designs mentioned explained earlier will be tested in rock masses with similar mechanical and geological parameters. The only difference is the design patterns parameters (e.g., burden, spacing). Rock conditions on which the blasting designs were tested are the following:

-UCS: 180 Mpa

-FF: 2 FF/m

-Density of Rock: 2.6 tonne/m^3

Obtained P_{80}s were:

Table 19 x_{80} results of different blasting methods

Blasting design	**P_{80} (mm)**
CSRD	241.64
CS	495.93
DRD	241.64
VCR	269.87

As can be seen in Table 19, the CSRD and DRD approach can generate the same p_{80} values. Therefore, DRD is discarded as an option for the blasting design, because a higher number of developments are needed to perform the method and achieve the same desired fragmentation in comparison with CSRD which is associated with a higher cost. CS is also discarded due to its very large p_{80}, as it could be quite hard to achieve a good leaching rate with such a p_{80} size. An important factor that implies discard the option of this blasting method is the loss of material on the corners of the stope, reducing the recovery of the mine. Hence, CSRD and VCR appear to be reliable options to produce the required fragmentation in the stopes of IMR. It is acknowledged that there could be challenging technical issues with the implementation of these approaches in site which needs further engineering research. But such changes and issues are beyond the scope of this research. Both methods represent relatively similar fragmentation. However, VCR presents a high risk for stability issues, needing a mixed design i.e. a combination with other methods as room and pillar, increasing the costs of the implementation. In fact, Central Swedish Ring Design is selected as the viable method for IPM, although it has a higher cost of developments, assuring a proper fragmentation. It seems that this approach can results in a proper leaching recovery. Safety and efficiency of the implementation are also acceptable for this approach. The particle size distribution graph generated by implementing this approach is shown in Figure 21

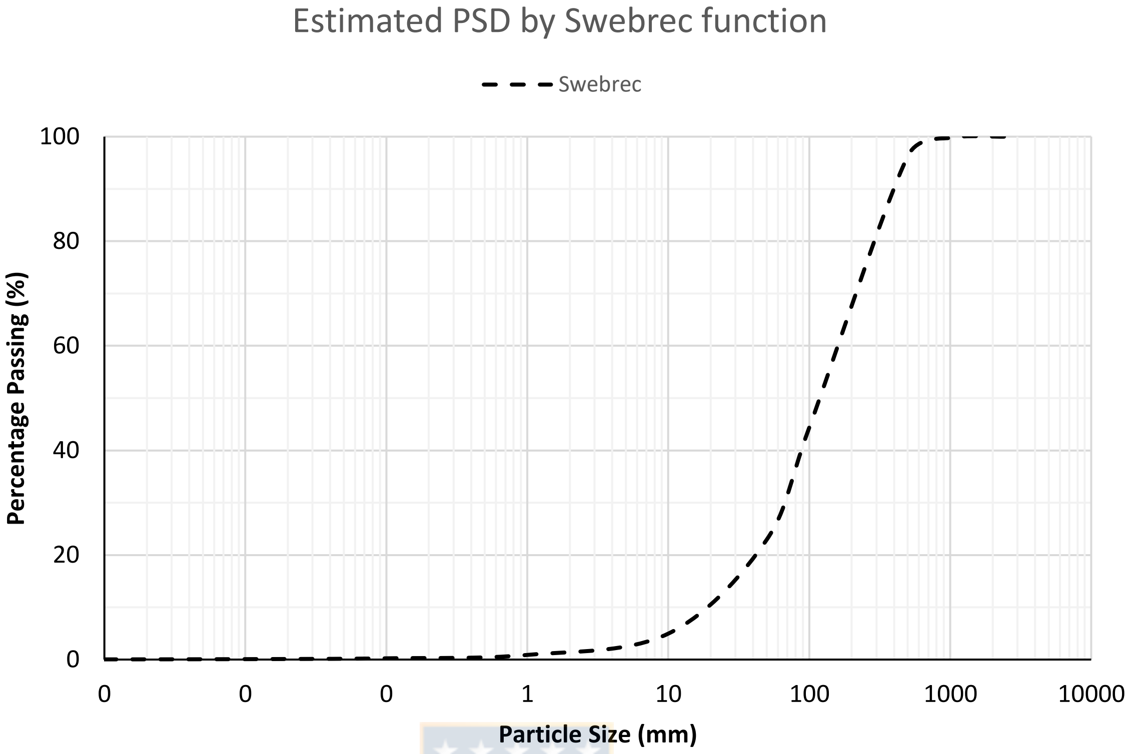


Figure 21 Particle size distribution of the tested rock for CSRD

4.2.6 Grade Engineering

Grade Engineering is one of the approaches developed by CRCORE for the efficient mining in heterogeneous and low-grade ores. Grade Engineering is based on five main pillars from which the preferential grade by size is an important lever.

The following relation was proposed by Carrasco and Keeney [6] to compute the metal upgrade generated based on preferential grade by size fragmentation of minerals based on the particle size distribution of the broken ore:

$$Upg = \frac{RR}{1 + W * (RR - 1)} \tag{26}$$

Figure 22 shows the diagram of Metal Upgrade, Upg, versus Cumulative Weight, CW, for different Ranking Responses (RR =2, 3, and 4). The particle size distribution of the muckpile (See Figure 21) is needed to identify the value of the metal upgrade for each screen size, based on the cumulative passing weight. This procedure has schematically been shown in Figure 22.

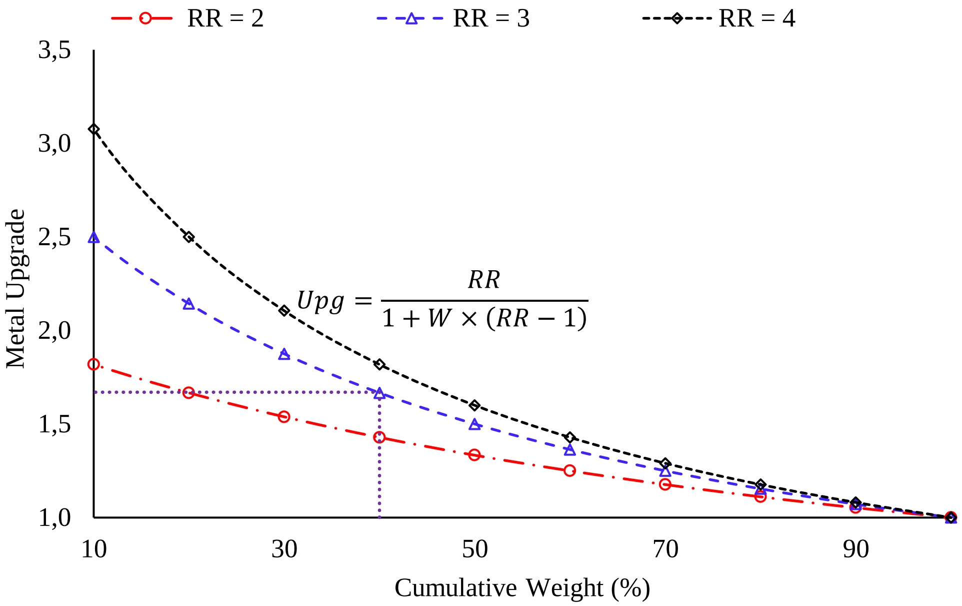


Figure 22 Preferential grade by size deportment response

In the case study of this study with relatively low to moderate tendency of preferential grade by size for copper, the ranking response is assumed to be around 2.

5. PREDICTING THE RATE OF RECOVERY IN LEACHING BASED ON NUMERICAL MODELING

5.1 Introduction

Mining engineering is as old as human being. The procedure of extracting metals from minerals has therefore been gradually developed during years. This procedure includes the knowledge related to the leaching behaviour of the different materials and quantification of the measurement processes for investigating the governing parameters, as well as identifying the effects of the leaching environment and the intrinsic characteristics of the material. The measurements and identification of the relevant parameters now allow us to quantify the recovery in a dump/ heap, or in situ leaching based on numerical formulations which have been developed based on small and large scale tests, such equations can be used to investigate the effects of different parameters on the performance of leaching.

Often the critical leaching result, to be predicted by a model, is the quantity of extracted metal over time, at realistic mine conditions. It is important to mention that a model is not precisely an exact estimation of the behaviour of leaching. It, however, could be a good approximation to some aspect of the process of leaching, and it can be used for pre-feasibility studies.

A major problem in model verification has been obtaining accurate and clear experimental results for comparison. As leaching depends strongly on the environment and the associated parameters, replicating the procedure of leaching in a laboratory scale is always difficult. Sometimes the parameters present in a laboratory environment and in scale up, either pilot or commercial tests show significant differences. Among these parameters are the effects of large rock particles, rock size segregation, solution channelling during percolation, and high loading of material. For example, in the case study considered in this thesis, a stope with a height of 240 m is intended to be irrigated. In practice it might be very hard for solutions to achieve the core volume inside of the stope.

Only a few large-scale experiments for dump leaching have been done in the past. These experiments can be used to extract some of the parameters that are needed for the prediction of the rate of recovery in this study.

Numerical simulation of the process of leaching may be divided into two classes of models including empirical and deterministic approaches.

Empirical models are based on historical data of operations, using rules of thumb or expert systems combined.

Deterministic models are made using physicochemical parameters that have a predominant effect on the target solution. However, to use these kinds of models it is always necessary to have several accurate input parameters to run the model and achieve reasonable outcomes. This might be difficult, when we are dealing with cases that are suffering of the lack of information. For example, it is easy to measure rock porosity from a sample in 1 kg laboratory test, but it becomes progressively more difficult to obtain the representative porosity for the rock in an experiment with a scale of 10000 tons or more. The necessity to know the exact quantity of material accessible for leaching in large-scale, is also a significant problem. For example, the only way to know how much valuable material is inside a stope is using block modelling with drillholes information.

Due to the lack of information, it is usually implicitly assumed that all the material is accessible to the dissolution penetrating through pores. However, this assumption may not be valid for all different type of rock masses containing minerals.

To estimate the rate of recovery a deterministic and mathematical modelling extracted from literature review is used in this study.

It is noted that every kinetic model works in a specific range of parameters, such as pH, PSD, temperature, etc. The performance of the models therefore strongly depends on the in-put parameters. The limitation of the models shall, therefore, be considered prior to the analyses. It is relevant to select an accurate kinetic model according to the characteristics of the material and environment which it has been designed for.

Table 20 shows some examples for oxidation of chalcopyrite with some parameters whose variation strongly affect the rate of leaching.

Table 20 Chalcopyrite leaching rate and mechanisms. Note: x is the oxidized fraction.

(Source: Li et. al [14])

T (°C)	Leach media	Rate dependence	Rate constant (K)	Ea (kJ mol-1)	Mechanism
10 - 40	HClO + HCl	Particle size	1-17x10^{-4} min^{-1}	19.88	Diffusion through product layer
50 - 97	$K_2Cr_2O_7$ + H_2SO_4		0.30-0.93x10^{-3} min^{-1}	24	Diffusion through a porous product layer by shrinking core model
45 - 100	$FeCl_3$ + H_2SO_4	$[Fe^{3+}]^{0.12}$		42	Diffusion
30 - 90	$FeCl_3$ + HCl	Particle size	2.63-6.93x10^{-6} s^{-1} 2.3-15.9x10^{-7} s^{-1}	15-28	Mixed control model (chemical reaction control; diffusion of ions through
70 - 90	$NaNO_3$ + H_2SO_4		Kt^n=-ln(1-X)	83	Mixed control model (surface reaction control; lixiviant diffusion through sulfur layer)
40 - 95	Fe^{3+} + HCl (H_2SO_4)	$[H^+]^{0.8}[Fe3+]^{0.42}$		48±10	Surface reaction control
23 - 40	$FeCl_3$ + HCl	$[Cu^-][Fe^{3+}][Cl^-]^2/[H^+]$		86.4	
25 - 50	H_2O_2 + H_2SO_4	$[H+]^{0.3}$	Kt= $1-(1-X)^{1/3}$	60	Surface reaction control by shrinking core model
30 - 80	$K_2Cr_2O_7$ + H_2SO_4	$[H+]^{0.8-0.92}$		48-54	Chemical reaction control
70 - 100	$FeCl_3$ + HCl	$[Fe^{3+}]^{0.38}$		93	Surface chemical reaction
25 - 70	HCl at pH 3			32±5	Surface reaction control
25 - 85	Fe + HCl	$[H^+]^1$	0.386-3.321x10^{-3} min^{-1} $1-3(1-\mathrm{Xb})^{\frac{2}{3}}+2\,(1-\mathrm{Xb})=\frac{k_0^{"}}{r^2}\exp\,(-\frac{22{,}423}{RT})$	22.4	Diffusion of hydrogen ions through a product layer by shrinking core model
125 - 150	O_2 + H_2SO_4	$[Po_2]^1$	Kt=$1-(1-(1-0.45X)^{1/3}$	93.5	Surface chemical reaction by shrinking core model
50 - 92	Fe2(SO4)3 + H_2SO_4		Kt=$1-(1-X)^{1/3}$	79.5	Surface reaction control by shrinking core model
25 - 75	H_2O_2 + H_2SO_4	$[H_2O_2]^1$	Kt=$1-(1-X)^{1/3}$	30	Surface reaction control by shrinking core model
25 - 75	Cu^{2+} + HCl			72	Surface chemical reaction and electrochemical control
55 - 85	H_2SO_4 Eh 750 mV SHE, pH 1	1. Fe^{3+} 2. Fe^{3+} 3. Without Fe^{3+} 4. Without Fe^{3+}	1. Kt=$1-(1-X)^{2/3}$ kt=$1-(1-X)^{1/3}$ 2. Kt=$1-(1-X)^{1/3}$ 3. Kt=$1-(1-X)^{1/3}$ 4. Kt=$1-(1-X)^{1/3}$	1. E_{Cu}<10h=21±5 2. E_{Cu}>10h=83±10 3. E_{Fe}=76±10 4. E_{Cu}=80±10 5. E_{Fe}=84±10	1. Mixed control model by shrinking core model (diffusion or transport control; chemical reaction control) 2. Chemical reaction control 3. Chemical reaction control 4. Chemical reaction control
30 - 80	H_2O_2+ H_2SO_4	$[H_2O_2]^{1.45}$ $[H^+]^{0.77}$	Kt=$1-(1-X)^{1/3}$	39	Surface chemical reaction by shrinking core model
50 - 90	[bmim*] HSO_4		Kt=$1-(1-X)^{1/3}$	69.4	Electrochemical surface control

Mixed kinetics occur when slow reaction minerals such as chalcopyrite in large particle size are presented within the broken rock. In such cases the mineral dissolution rate must be considered along with diffusion.

Braun, Lewis [52] proposed a mixed kinetic model for leaching which was developed based on tests in a large scale operation for large particle sizes [10]. The outcomes of this model are well consistent with the real tests data.

5.2 Kinetic Model

A mixed kinetic model was selected by the author of this thesis after performing a comprehensive literature review. The considered model was developed by Braun and Lewis [52]. The model was selected due to its high accuracy for predicting the leaching rate in materials with large particle sizes coinciding with the characteristics of the typical ROM material obtained after blasting operations in a stope. The model simulates an acidic leaching pressurized with oxygen, condition that could be present in stope Leaching. Although this kind of leaching is not precisely accurate for sulfides in sulfate media [12], it still can provide an approximation of the rates of recovery for large scale operations in real conditions which is believed to be suitable for pre-feasibility study. The model also was the only one tested in large scale operation in the work developed by Madsen [10]. This kinetic model also considers several parameters including: temperature, mineralogy, shape of the particle, time and indirectly aeration, and associate this parameter with the pressure of oxygen. The model does not however consider height of load of the material, assuming that all the material is completely irrigated by the leaching solution. It was also assumed that particles have a moving zone of reaction as the time goes since the reaction occurs. This reaction zone separates an unreacted portion of the particle on the core from an already reacted zone at the exterior surface. The model assumes that the circulation of leach solution around the particle is enough to maintain the oxidant concentration. Hence, that bulk solution transport is not rate controlling. Therefore, the kinetics are limited by only reactions occurring inside the particle.

5.2.1 Reaction Zone Model

To be able to use the model developed by Braun and Lewis [52], it is assumed that rock particles have spherical shapes. Deviations from a spherical shape are then corrected by the shape factor parameter ($ø_{i0}$), including boundary roughness and sphericity changes.

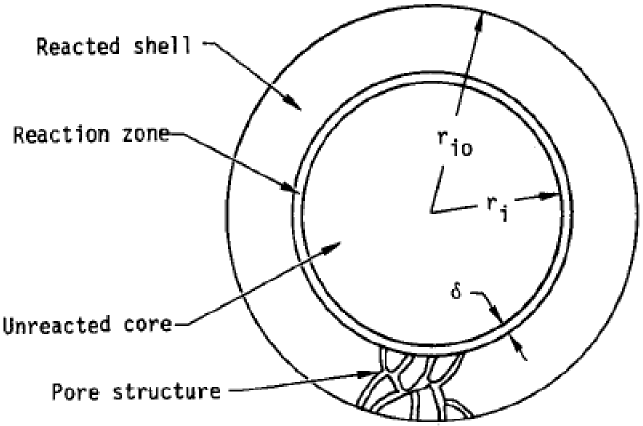


Figure 23 Ore particle of radius r_{I0}, showing reaction zone. Braun and Lewis [52]

Figure 23 shows an idealized ore particle including the reaction zone, with a thickness of δ. The thickness δ can be calculated by the equation developed by Cathles and Apps [53] as follow:

$$\delta = \frac{\mathrm{Df}}{\mathrm{K} * a_{sulf}^{R}} \tag{27}$$

where *D* is the effective coefficient of diffusion for the reactant (cm^2/s), the factor *f* is related to porosity (ϵ) and tortuosity (ι) $f=\epsilon/\iota$, *k* (cm/s) is the reaction rate constant and a_{sulf}^{R} is the surface area of sulphide mineralization per unit of volume of material, considering this ≈ 80 cm^{-1} for chalcopyrite.

The porosity of the mineral is given by the following equation:

$$\epsilon = \left(\frac{\mathrm{G}\rho_r}{3\rho_p}\right)^{2/3} \tag{28}$$

where ρ_p is the density of the copper sulphide, ρ_r is the bulk rock density, and *G* is the grade of the particle.

The initial average radius of rock particles is represented by r_{i0} and the radius at time t is r_i. As the reaction zone moves to smaller values of r_i, mineral particles are included in the core. The concentration of minerals within the reaction zone is essentially considered constant.

The rate of reaction within the leaching zone for a given particle is expressed as:

$$(\frac{dn}{dt})_i = (\frac{4\pi r_i^2}{\emptyset_{io}})\delta n_p A_p C_i k \tag{29}$$

where n_p is the number of sulphide particles per unit volume of rock, A_p is the average area per sulfide particle. C_i is the concentration of the reactant in the reaction zone (moles/cc) (e.g. oxygen), *k* is the rate constant, and *n* is the number of moles of unreacted chalcopyrite in an ore fragment.

The rate of transport to the reaction zone is expected to be equal the rate within the reaction zone. Diffusion through pores to the reaction zone may be expressed as:

$$(\frac{dn}{dt})_i = -\left(\frac{4\pi r^2}{\emptyset_{io}}\right)\left(\frac{Df}{\sigma}\right)\left(\frac{dC}{dr}\right) = J_i \tag{30}$$

where σ is the stoichiometry number, and in this case shows the molar acid consumption (moles of acid/moles of copper extracted),

For a given value of r_i, the equation must be integrated for steady-state conditions (J_i = constant) for all *r* values between r_i and r_{i0} which combined with equation gives an expression for the concentration of reactant *Ci* in the reaction zone in terms of bulk solution concentration C_b, or

$$C_i = C_b[1/\left(1 + \left(\frac{\delta n_p A_p \sigma k}{Df}\right)\left(\frac{r_i}{r_{i0}}\right)(r_{io} - r_i)\right)] \tag{31}$$

Substituting in Equation 28, gives the generalized expression:

$$(\frac{dn}{dt})_i - \frac{4\pi r_i^2}{ø_{io}} C_b[\frac{1}{\left(\frac{1}{\delta n_p A_p \sigma k}\right) + \left(\frac{\sigma}{Df}\right) + \left(\frac{r_i}{r_{i0}}\right)(r_{io} - r_i)}] \tag{32}$$

where the Grade G (weight fraction copper sulphide mineral) is calculated by:

$$G = \frac{n_p A_p r_p \rho_p}{3\rho_r} \tag{33}$$

where r_p is the average radius of the spherical copper sulphide particles. Combining equations 32 and 33 results in:

$$(\frac{dn}{dt})_i - \frac{4\pi r_i^2}{\o_{io}} C_b [\frac{1}{\left(\frac{1}{G\beta}\right) + \left(\frac{\sigma}{Df}\right) + \left(\frac{r_i}{r_{i0}}\right)(r_{io} - r_i)}] \tag{34}$$

where β is initially:

$$\beta = \frac{3\rho_r \delta k}{r_p \rho_p} \tag{35}$$

The model presented above is applicable for leaching calculation at 90°C because the model was developed in experimental environment with specific characteristics. Unfortunately, due to the lack of suitable data the performance and validity of the model for predicting the leaching at other temperatures has not been explored. It is however expected that temperature can have significant effect on the chemical and physical behaviours of materials during leaching. A fixed pH may be adjusted to normalize physical properties of the system. However, the effects of temperature on coefficients such as C_b, β and Df/σ shall well be analysed. For example, C_b changes according to the solubility of oxygen in water at different temperatures. This can be summarized by the following equation known as Henry constant (moles*cm^3/psi)

$$\mathrm{k_h} = 1.068 * 10^{-7} - 1.168 * 10^{-9}\mathrm{T} + 6.109 * \ 10^{-12}\,\mathrm{T}^2 \tag{36}$$

With T in °C, and the solubility of the oxygen in water is given by:

$$C_b = k_h * pO_2 \tag{37}$$

where pO_2 indicates the pressure of the oxygen in the system in psi.

It is possible to assess the temperature dependency of the parameter β determined experimentally. This parameter β shows the chemical and physical reaction rate and it is highly required for the modelling. The reaction rate constant k (cm/s) is also related to the specific rate constant k' based on the following equation:

$$\frac{k\rho_p}{M} = Sk_0k' \tag{38}$$

where S is the fraction of the surface of $CuFeS_2$ (most abundant mineral) that is chemically active, M is the molecular weight of $CuFeS_2$, and k_0 is the total number of moles of surface sites per cm^2 on the surface of $CuFeS_2$. The value of k_0 (moles/cm^2) may be approximated by:

$$k_0 = \left(\frac{\rho_p}{M}\right)^{\frac{2}{3}} \left(\frac{1}{N}\right)^{1/3} = 9.55\ x\ 10^{-10} \tag{39}$$

where N is Avogadro's Number. The specific rate constant *k'* is also given by:

$$k' = \frac{k_b T e^{-\frac{\Delta H}{RT}} e^{\frac{\Delta S}{R}}}{h} \tag{40}$$

where k_b is the Boltzmann constant, *h* is the Planck constant, *ΔH* is the enthalpy of activation and *ΔS* is the entropy of activation. For considered model, equation 40 may be expressed in terms of the experimental activation energy *ΔE* as:

$$k' = \frac{e k_b T e^{-\frac{\Delta E}{RT}} e^{\frac{\Delta S}{R}}}{h} \tag{41}$$

Additionally, in the proposed model, the entropy of activation is expected to be small, containing only vibrational degrees of freedom. Considering ΔS=0, equation 35 becomes

$$\beta = \frac{3\rho_r \delta M}{r_p {\rho_p}^2} S k_0 \frac{e k_b T e^{-\frac{\Delta E}{RT}}}{h} \tag{42}$$

According to equation 35 the ratio $\delta S/d_p$ may now be evaluated as:

$$\frac{\delta S}{d_p} = 1.25\ x\ 10^{-12} e^{-\frac{\Delta E}{RT}} \tag{43}$$

where d_p= copper sulfide particle diameter, ρ_p=4.2 for chalcopyrite and β=4 x 10^{-6}. It is needed to estimate reasonable limits for the value of $\delta S/d_p$. Examination of partially leached specimen indicates a sharp reaction interface such that δ/d_pcannot be much greater than 10. The quantity S is the fraction of the cathodic surface of the $CuFeS_2$ which results in the reduction of oxygen. If the anodic and cathodic surface are equal in area, *S* would be 0.5. It seems unlikely that it would be less than 0.1 using the limits $1 < \frac{\delta}{d_p} < 10\ and\ 0.1 < S < 1$ the range of values for $\delta S/d_p$ would be $0.1 < \frac{\delta S}{d_p} < 10$. It gives a range of 18.0 to 21.4 kcal/ mole for the value of ΔE. This agrees with values previously reported (e.g Cathles and Apps [53]) and lends strong support to the validity of the kinetic model.

Finally, it must be considered the quantity Df/σ, so the molecular diffusivity of oxygen in water, D, is 2.5 x 10^{-5} cm^2/s at 25 °C. Although D is proportional to the ratio of absolute temperature to viscosity in dilute solution at 1 atm, the actual value of *D* has not been measured. However, this parameter can be calculated for the conditions of interest. Nevertheless, the overall temperature dependence of *D* is indeed small compared with that of the chemical reaction rate constant *β*, and this dependency can be neglected. Therefore, the principal temperature dependent parameters have now been included in the model.

5.2.2 Analytical integration of leaching rate

Equation 34 is useful only for a given particle size (r_{io}). Therefore, it may be integrated for a range of particle size either from a laboratory study or commercial test. It is also useful to normalize the rate in terms of fraction reacted (α), since α for a single particle or an assemblage of particles of size r_{io} is the same. For a given particle:

$$\alpha_i = 1 - \frac{r_i^3}{r_{io}^3} \tag{44}$$

For a sample of broad particle size distribution of i sizes,

$$\alpha = \sum_i \alpha_i W_i \tag{45}$$

where a_i is the fraction reacted for size r_{io} and w_i is the weight fraction of that size (%).

The number of moles of unreacted mineral in the core of the particle is given by

$$n = \frac{\frac{4}{3}\pi r_i^3 \rho_r G}{M} \tag{46}$$

where M is the molecular weight of the copper sulfide mineral (gr/mol). The rate of movement of the reaction interface may be determined from equations 34 and 46

$$\left(\frac{dr_i}{dt}\right) = -\frac{M}{\varnothing_{io}\,\rho_r G} C_b [\frac{1}{\left(\frac{1}{G\beta}\right) + \left(\frac{\sigma}{Df}\right) + \left(\frac{r_i}{r_{i0}}\right)(r_{io} - r_i)}] \tag{47}$$

The fractional reaction rate (dα/dt) may be determined from 43 and 46 such that:

$$\frac{d\alpha}{dt} = \frac{3MC_b}{\varnothing_{io}\,\rho_r G r_{io}} \left[\frac{(1-\alpha)^{2/3}}{\left(\frac{1}{G\beta}\right) + \left(\frac{\sigma r_{io}}{Df}\right)(1-\alpha)^{1/3}[1-(1-\alpha)^{1/3}}\right] \tag{48}$$

In an integrated form, equation 48 becomes

$$\mathbf{1} - \frac{\mathbf{2}}{\mathbf{3}}\boldsymbol{\alpha} - (\mathbf{1} - \boldsymbol{\alpha})^{\frac{2}{3}} + \frac{\boldsymbol{\beta'}}{\boldsymbol{Gr_{io}}}\left[\mathbf{1} - (\mathbf{1} - \boldsymbol{\alpha})^{\frac{1}{3}}\right] = \frac{\Upsilon}{\boldsymbol{\varnothing_{io}\,Gr_{io}^2}}\boldsymbol{t} \tag{49}$$

where t is time in seconds, and:

$$\beta' = \frac{2Dfr_p\rho_p}{3\sigma\rho_r\delta k} = \frac{2Df}{\sigma\beta} \tag{50}$$

And:

$$\Upsilon = \frac{2MDfC_b}{\rho_r \sigma} \tag{51}$$

In summary, copper extraction curves can now be calculated by employing equation 49 when the ore particle size range is narrow enough to be described adequately by one average size.

5.2.3 PSD Applied to the Model:

Generally, the fragment sizes inside a stope are ranging from around $x_{max} =$1500 mm to 0.1 mm. This range of sizes is not narrow enough to be adequately described by a single average particle size. Therefore, the model was extended to include the actual size distribution and the grade of each sized fraction. In other word, the model will be used for N ordered pairs [x_i,P(x_i)] for the entire distribution for x_i = 0 to x_i = x_{max}.

This set of N ordered pairs (xi, p(x)) is used to separate the material into a set of m pairs (r_j, W_j) where W_j is the weight fraction of ore having an average radius r_j, by the following relationships:

$$r_j = \frac{x_j + x_{j+1}}{4} \tag{52}$$

$$W_j = P(x_{j+1}) - P(x_j) \tag{53}$$

To explain it further, for example when only one range of material is used the weight fraction becomes:

$$W_j' = \frac{W_j'}{1 - P(x_s)} \tag{54}$$

where P(x_s), is the accumulative weight passing the smallest theoretical diameter of rock fragments.

6. CASE OF STUDY MINE ‘A’

A real case of study copper operation was considered in this study. The copper mine that will operate in Chile is used to estimate the potential recovery of leaching in an underground mine that is supposed to be extracted using IMR. In this study, the name of the mine is protected due to possible exposure of confidential information, therefore, it is called as “Mine A”.

The aim in the mine operation is to reduce the material movement from underground to the surface as well as to reduce the required developments, such as the ventilation tunnels and backfill shafts. These are needed to reduce the overall mining costs and to reduce the environmental footprint of mining in such a critical area. These objectives are important because the mine is sited in the middle of an area with the presence of nature reserves and glaciers. Therefore, the objectives of the project are:

- The project must be economically viable, compatible with the economic requirements.
- The project will use the existing infrastructure as defined in the current “Mine A” during the Life of Mine Plan, so no additional processing plant, tailings disposal or waste dump capacities are required.
- IMR has no negative surface impact on the surrounding Nature Sanctuary.
- In-mine recovery case no negative impact on the close by glaciers.
- Mining is expected to have no negative impact in quantity or quality of Chilean society water basins.

6.1 Mine, deposit and environment characteristics

6.1.1 Mine Design

The mine contains three levels (levels 3800,3560 and 3320) with a prefeasibility study already available for the conventional method of long hole open stopping. A fourth level (level 3080) would be done according to cut off grades of the Mine Plan when the IPM methodology may be included. The current extraction method presents basic units of stopes inside of production modules of 7x5 stopes defined as pillars, backfilled stopes and active stopes. A simple scheme of the mine is presented in Figure 24:

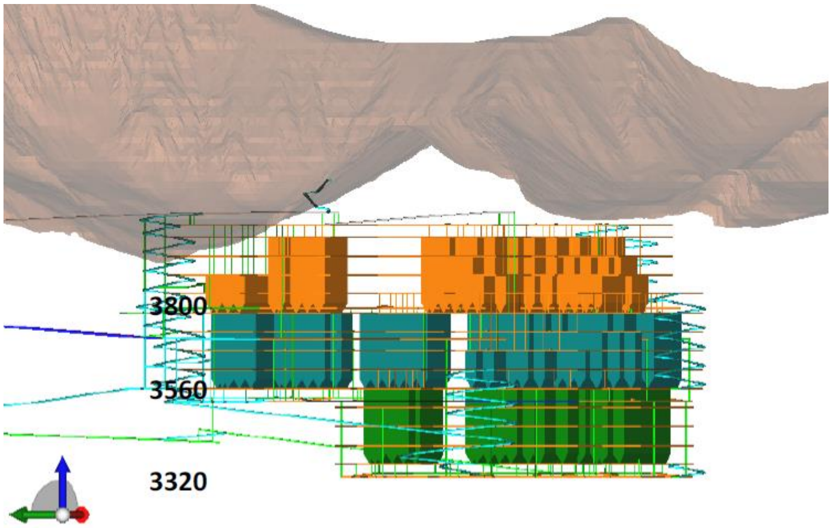


Figure 24 Scheme of 'Mine A'

The number of stopes per level are 724, 1063, 1238 and 725 at the levels 3800, 3560, 3320 and 3080 respectively.

6.1.2 Mineralogy

The orebody contains mainly copper in presence of chalcopyrite, pyrite and bornite (Cpy, Py and Bn) combined with some presence of anhydrite and gypsum.

The average grade of copper of the mine is 1.18% and 0.018% of molybdenum contained in a 200 Mt deposit. Figure *25* shows the spatial distribution of the mineralization.

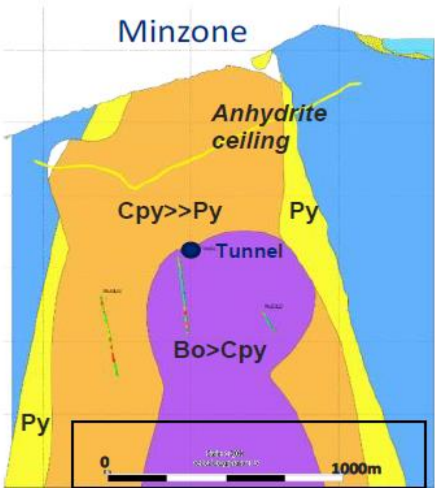


Figure 25 Mineralization zone of 'Mine A'

6.1.3 Alteration

In ‘Mine A’ area the typical hydrothermal events of a copper porphyry system are identified. The Porphyry (PA) Granodioritic in Mine ‘A’’ shows a strong potassic alteration characterized by biotite and potassic feldspar, associated with intense A type veinlets (bornite – chalcopyrite). Intense biotitization is also observed in the andesites located at the contact with the PA, in an approximately 50 (m) strip.

A sericitic (sericite – chlorite) alteration event is super-imposed to the potassic centre, apparently controlled by high permeability given by the pre-existing intense A type veinlets. The sericitic alteration shows as bands of halos of D type veinlets (chalcopyrite – pyrite) and DL (chalcopyrite–pyrite–specularite). In areas where the sericitic super–imposition is pervasive and more intense, the bornite – chalcopyrite mineralization gives place to chalcopyrite–pyrite, with decreasing Cu grades.

6.1.4 Temperature

Regarding the temperature, due to the geothermal gradient of the rock, measurements made from the drillholes can be summarised as follow:

- At 100 m from the surface, below the glaciers the rocks are at 0 ° C.
- At 600 m from the surface, the rocks are at 11 ° C
- At 1000 m below the surface, the rocks are at 22 ° C

6.1.5 Lithology

In summary, the mineral deposit has Breccias and Andesites lithology there is also presence of Quartz monzonite and porphyry. Only by the geological and physicochemical characteristics of the rocks and their copper ores it is possible to observe that they may not be suitable to be treated by in situ leaching process.

On the other hand, because of the sensitive environmental characteristics of the area, where the authorities want the least presence of acid drainage, the feasibility of obtaining a permit for an operation of this type could also be complex. This means that the mining operation and, consequently the leaching operation must fulfil the required environmental conditions.

6.1.6 Leaching test

The levels of pH, Eh, EC, T were measured with humidity cells to evaluate the kinetic of acid dewatering starting from oxidation of present sulfides detected in samples corresponding to Geo-environmental units (UGAs) to extract from underground area at the mine.

Figure 26 shows the results of a test performed with humidity cells installed at the “Mine A” at the end of 20 weeks. Humidity cells are based on 7 days cycles, where weekly measured parameters such as: pH, redox potential, electric conductivity, calcium, copper, iron and carbonates. However, a longer test should be done at the end of 40 weeks, to understand the material behaviour in leaching due to the fact that the reactivity of copper porphyries is slower than other type of copper minerals, and it is very difficult to observe the oxidation of sulfides within 20 cycles.[54, 55]. Some results are shown in Figure 25.

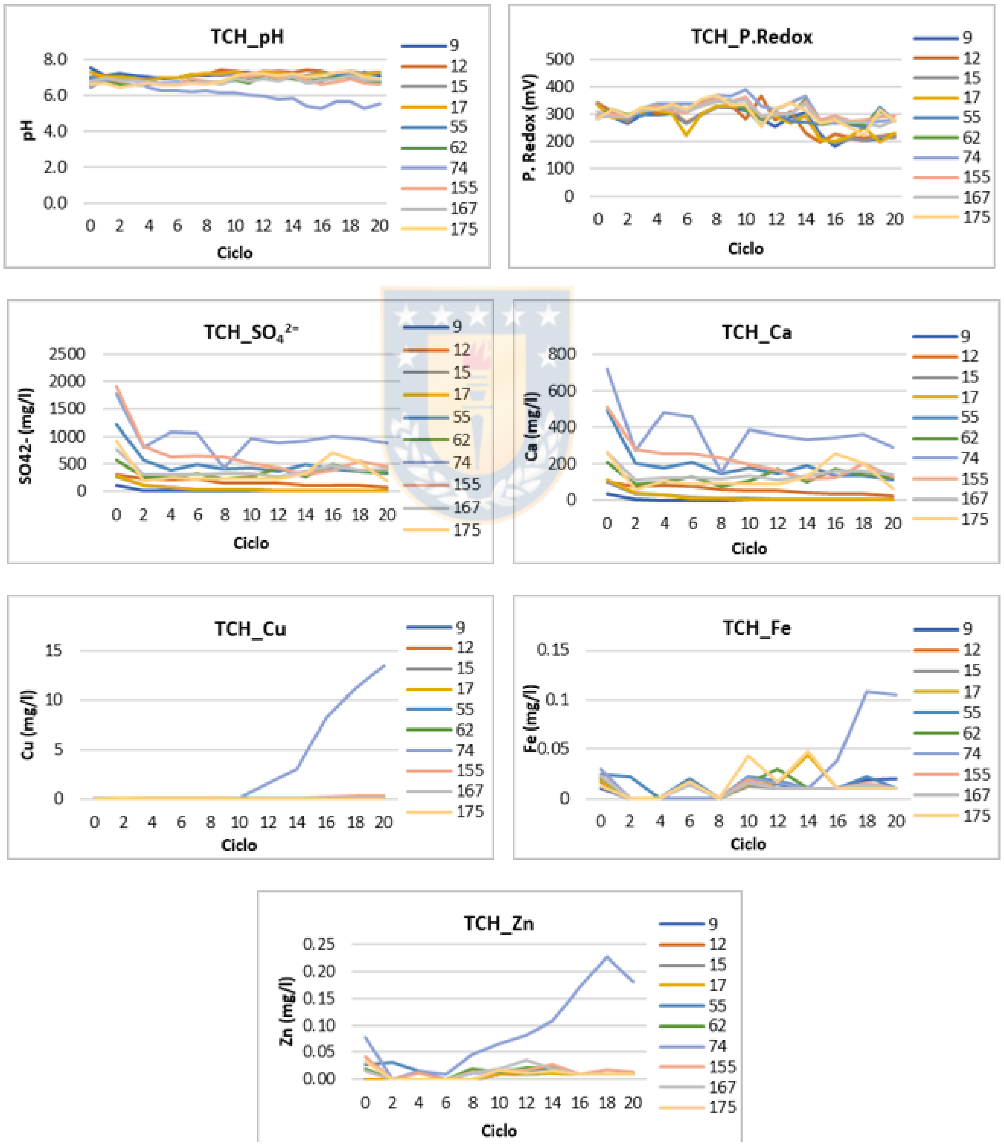


Figure 26 Humidity cells test

According to Figure 26 from the plots it can be concluded that 9 of the 10 samples do not show generation of acid dewatering until the '20' cycle reported, as the pH has been stable around ~~to~~ neutral values (close to 7). Hence, the same liberated sulfate obeys the dissolution of other mineral phases such as gypsum $CaSO_4\cdot 2H_2O$ and/or anhydrite $CaSO_4$ and does not strongly depend on sulfides oxidation (chalcopyrite and bornite). This is appropriate with the calcium plot that shows the same pattern as sulfate and electric conductivity for these 9 samples. On the other hand, the oxidation of sulfides should liberate Fe, Cu and other traces metals, which is not evidenced either on the plots for the 9 samples.

Only the sample 74 shows a beginning of sulfides oxidation, as the pH had decreased gradually, showing from 15th week a pH around 5.5.Particularly, this sample corresponds to andesite (AND) with a high chalcopyrite content (Cp>5%) higher than the rest of the samples and within thick veins, proportion and frequency that have beneficiated the sulfide mineral exposure to the oxidant agents forced at humidity cell.

According to the results, when 20 weeks have passed of cells operation, for case that the oxygenation and interaction with water occurred, it can be concluded that:

- Waste samples (N°9,12,15 and duplicate 17) on 20 weeks of exposure, do not show acid generation, although the material has potential due to pyrite (FeS_2) presence. This is because the oxidation processes of the sulfides are slow and conditioned by the quantity and degree of freedom of themselves.
- Mineral samples (N°55, 62, 74, 155, 167 y 165) only the N° 74 has shown acid generation, it should be noted that all the samples have potential to generate acid due to Cu sulfides content (chalcopyrite and Bornite) but only the N° 74 had been shown on the indicated sample, after 15 weeks of exposure.
- As the pH does not decrease in most of the cells, not generating acid, it can be concluded that it is needed to have an additional incorporation of sulfuric acid in the leaching solution to achieve a good recovery of leaching. This is because of the low capacity of the system to produce a self-buffered reaction, either by bioleaching or oxygen oxidation. This characteristic is also attributed to the presence of gangue in the rock.

The other important point is the high presence of clays and fines in blasted rocks, which can negatively affect the rate of leaching, consuming acid. Consequently, it is crucial to use strong acid for leaching of such cases or to adjust the blasting to not to generate lots of fines.

Interpreting the leaching data, and considering that the testing time was not long-enough to see the effect of the reactions in approximately 2 years, the data from the work developed by Braun and Lewis [52] and the chemical reactions occurring during that leaching operation were considered in this study. It is assumed that the material used in Braun and Lewis's tests are almost identical to those present in "Mine A". An increase in the pH during the reaction shows a buffering effect due to presence of the gangue constituents.

Assuming that all the materials have relatively similar chemical behaviour, the following equations show the reactions that occurred in the leaching process:

Acid consuming reactions:

$$CuFeS_2 + 5/4\ O_2 + 5H^{+} = Cu^{+2} + Fe^{+3} + 2S^{0} + 5/2\ H_2O \tag{14}$$

$$FeS_2 + 3/4\ O_2 + 3H^{+} = Fe^{3+} + 2S^{0} + 3/2\ H_2O \tag{15}$$

$$CaCO_3 + 2H + SO_4^{-2} = CaSO_4 + CO_2 + H_2O \tag{16}$$

Acid generating reactions:

$$S^{0} + 3/2\ O2 + H_2O = 2H^{+} + SO_4^{-2} \tag{17}$$

$$Fe^{+3} + 3/2\ H_2O = 3H^{+} + 1/2\ Fe_2O_3 \tag{18}$$

$$Fe^{+3} + 7/2\ H_2O + 2/3\ SO_4^{-2} = 5/3\ H^{+} + 1/3 Fe_3(SO_4)_2(OH)_5 \cdot 2H_2O \tag{19}$$

So, the net reaction of the leaching can be written as:

$$\begin{gathered} CuFeS_2 + 2FeS_2 + CaCO_3 + 45/4\ O_2 + 37/6\ H_2O = Cu^{+2} + Fe_3(SO_4)_2(OH)_5 \cdot 2H_2O + \\ CaSO_4 + 1/3S^{0} + 8/3\ SO_4^{-2} + 10/3\ H^{+} + CO_2, \end{gathered} \tag{20}$$

Where the pyrite/chalcopyrite ratio is 2, and 11.25 moles of acid is required to oxidate 1 mol of copper. Therefore, the stochiometric factor is already defined, σ=11.25. Note that the stochiometric factor can be distinct for different mineralization and alteration. For example, bornite-biotite interactions and chalcopyrite may have different stochiometric factor. However, due to the lack of information about the components, this has not been considered in this study.

As the permeability of the mine is very low (around 2%) it is needed to apply fragmentation using blasting to improve the permeability of the rocks for leaching. Then, the stopes would be irrigated with sulfuric acid or other type of environmentally friendly lixiviant. Oxygen would also be aggregated to carry out the oxidation of the ore.

6.2 Methodology

A calculation of the rate of recovery with Excel and “R” was done using the existing block model of the mine that contains the averages values of fracture frequency, UCS, density and grade of each stope.

The fragmentation of each stope was calculated according to the Swebrec function (eq. 22), and the percentage of particles smaller than sizes 1, 10, 50, 250, 500 and 1000 mm were obtained according to the selected blasting design (CSRD). The cumulative weight percentage of each class was computed to calculate the metal upgrade to apply Grade Engineering (eq. 22). The weight of the materials in the bin size ranging x_i and x_{i+1} was considered to estimate the recovery according to equation 48 and to extract all the required components. These calculations were made using the average of the weights and grades per level.

The recovery was calculated as an average of each level, using a sensitivity analyses to understand the ranges of changes in the recovery with the performed variation of each parameter. The varied parameters were temperature, time of operation, shape factor, pressure of oxygen. In this case study, as the fragmentation and grade are dictated by geology and by blast design, the values of these factors were not changed in the modellings. Indeed, these parameters may be changed to analyse the effects of other blast designs or in areas with different grades.

The model considers the leaching operation in acid conditions, pH (1-3) and assumes that all the material, particularly, all the stope is completely irrigated and all ore can be reached by the solution, regardless of the height of the load of broken material and irrigation rate. The model works for the most abundant mineral present in the stopes. However, it must be noted that although the alteration and mineralization of rocks in the stope is obvious, the geological information, at the time of study, was not enough to understand, which was the most abundant mineral inside individual stopes of IMR. Nevertheless, the assumption that all the ore body contains chalcopyrite as the most abundant material was considered in the analyses.

The model also considers sulfuric acid as the lixiviant and oxygen as oxidant, assuming that the oxygen is injected by forced aeration with pipes, same as the leaching solution. The other assumption considered by the author was that the shape factor is static for all the particle sizes, due to the lack of information about the existing material.

7. RESULTS

In the design phase, it should always be considered that the design of high stopes means a reduction in the rate of recovery. There is no practical case to show that leaching in stopes with 240 m height is doable or not. Perhaps there might be some technical engineering challenges that must be addressed. Such technical issues are beyond the scope of this research study. Proper irrigation technologies must be implemented to avoid the decreasing of recovery that can occur due to the poor flow of the solution through the material inside a stope.

The main reason why a reduction of the rate of recovery may occur in high stopes is the compaction induced by the weight of the huge quantity of material supported by the lower portion of the stope. The compaction affects the permeability producing a poor flow of the leaching solutions over the surface of particles, and therefore the valuable particles of mineral will not be reached by the solution. The presence of fines is also an important factor. The movement of material after the extraction of the swelled portion of the stope can produce fines migration and fines due to the secondary impacts between particles which is like a crusher or mill in the processing plant.

The main factor and parameters considered in this study are as follow:

Fragmentation: To improve the rate of recovery inside a stope it is always essential to achieve a proper fragmentation. This means that a proper fragmentation is required to allow the circulation of the lixiviant reaching all the particles and its composition, mainly in the valuable part, where the metal is present. The fragmentation controls factors such as the size and shape of the particles. The presence of fines also affects the recovery due to the clogging inside the fractured material when the fines particles migrate. Additionally, most of the fines have an abundant presence of clays and weak minerals, which can result in higher acid consumption.

The blasting is a critical operation to achieve the fragmentation of the rock materials in the stopes. Therefore, in this study we selected a blast design for the stope which can result in a finer fragmentation. This can help to improve the rate of recovery. The selected blasting method was the Central Swedish Ring design. There are a few limitant factors that must be considered in blast design in order to reduce the risk of leakage of lixiviant and the stability of the stopes. The cost of drilling and blasting per stope was also considered in the analyses.

Mineralogy: The presence of different minerals in an environment which is aimed to be leached is indeed a governing factor in the recovery of IPM. This is generally explained by different chemical and physical compositions of minerals, which affect the duration of leaching, and the difficulty of the mineral to be leached as the ore minerals are generally more refractory and the waste rocks affect negatively the operation (acid consuming, passivation, among others).

Leaching time: The leaching time can affect the rate of recovery and generally the higher the duration of leaching the higher is the recovery. Leaching time also depends on a few other parameters such as fragmentation, mineralogy, temperature, leaching solution, pH, and the design of the leaching operation. Leaching time must be considered on the mine planning because it highly affects the economics of the mining projects.

Temperature: The required temperature for leaching highly depend on the mineralogy. Generally, when temperature is increased the recovery of leaching increases as well. There are some cases where in a range of temperatures waste particles can be dissolved and affect the oxidation of the mineral. The temperature is also controlling the range of work of the lixiviants, where, for example, in bioleaching, and chemical reactants, some microorganisms can die and lixiviants can decompose at certain temperatures. For an underground operation, the rise of the temperature as the depth increase shall be considered in the stope leach design. The surrounding environment of the operation, where crew members and machines are operating, should also be considered to design the temperature of leaching for IMR.

Leaching solution: As it was discussed before, the leaching solution plays a dominant role on the rate of leaching. The lixiviant is always considered under a best lixiviant-ore interaction selection according to the mineralogy and material characteristics. In general, as the lixiviant has higher concentration the leaching operation can be improved. It works similarly with the increment of the irrigation rate. However, an excess of these components can also negatively affect the recovery of leaching.

pH: Generally, the pH in leaching works in a lowest possible range (acidic regime) to allow the chemical reaction to occur faster. In cases where the presence of carbonate is high, ammonium leaching could be carried out. However, in the next stages of this operation a lower pH is needed to carry out the separation of the valuable mineral from the solutions.

Height of the pile of broken rock: A higher pile height means losses in recovery due to the compaction and problems in the movement of the leaching solution.

Aeration: Oxygen is always needed to oxidize the ore; therefore, aeration is the way to bring oxygen to the materials for leaching.

Hydrogeology: The hydro-geomechanical behaviour of the ground considered for the design of stopes for leaching operation is a key factor should be considered in design. This factor can affect the circulation of fluids though a stope. Therefore, permeability and spatial distribution of discontinuities and the ore material into a stope are also effective parameters affecting the distribution of the fluid flow into the broken rock materials.

For the case study of this research "Mine A", a numerical model was developed to estimate the rate of recovery within a certain range of parameters. As explained before the fragmentation of rock materials was calculated through the Swebrec function. The Grade Engineering arrangement was also considered in the analyses. The variable parameters considered in the analyses are as follow:

The time of leaching was considered based on the previous works done by Mining3. The performed pre-feasibility studies showed that 2 years (730 days) can be considered for the leaching of minerals through stopes. Therefore, in the sensitivity analyses done in this thesis the author assumed a time range of 365 (1 year) to 1095 (3 years) days.

The pressure considered in the analyses was extracted from the experiment reported in Braun et al. [52]. This study simulates the conditions of in place mining method. The pressure values varied between 200 and 600 psi with an average around 400 psi. The pressure of oxygen is related with the aeration that will be injected into a stope.

The average temperature was set to 80 °C to be able to takes advantage of the favourable chemical reaction showed in previous chapters. For the sensitivity analysis, the temperature was varied from 40 to 120 °C. It should be noted that higher temperatures can increase the rate of recovery, but the associated risk of the operation increases as temperature increases and for temperatures over 120 °C it can result in the dissolution of unwanted rock components in leaching.

Shape factor: A minimum value showed in the experiments of Braun et al. [52], was 0.17. The maximum value is 1, which considers that the particle is perfectly spherical. For simplicity it is assumed that the value of this factor for blasted rock particle is around 0.6 which is the rounded number of 0.585 (the average between both maximum and minimum values). This range of shape factor was considered in the analyses due to the lack of information related to the shape of particles generated by blasting of rocks in the stopes.

Shape factor can highly affect the recovery. However, this parameter is strictly dependent on the material itself and cannot be changed to increase the recovery. Then, the sensitivity analysis for shape factor was completed to understand its influence over the rate of recovery due to the lack of information presented about the material characteristics and its shape factor values.

In summary, the fixed parameters used for the numerical calculation of the rate of recovery are presented in Table 21.

Table 21 Fixed parameters used to calculate recovery

Parameter	Unit	Symbol	Value
Time	days	t	730
Time	s	t	63,072,000
Rock density	gr/cm^3	ρ_r	2.60
Mineral density	gr/cm^3	ρ_p	4.20
Tortuosity	-	ı	2.00
Diffusivity of oxygen in water	cm^2/s	D	2.5E-05
Molecular weight	g/mol	M	183.53
Boltzmann's Constant	m^{2}kg/s^{2}k	k_b	1.38E-23
Planck’s constant	m^{2}kg/s^2	h	6.63E-34
Henry’s constant	moles/cm^{3}psi	k_h	5.87E-08
Pressure	psi	pO_2	400
Temperature	°C	T(C)	80
Temperature	K	T(K)	333.15
Fraction of cathodic surface of $CuFeS_2$ chemically active	-	S	0.5
Total number of moles of surface sites per cm^2 on the surface of $CuFeS_2$	moles/cm^2	k_o	9.55E-10
Energy of activation	kcal/mol	ΔE	18
Constant of gases	kcal/mol K	R	1.98E-03
Stochiometric coefficient	-	σ	11.25
Shape factor	-	ø	0.6
Surface area of sulfide mineralization per unit volume of waste	cm^{-1}	a_{sulf}^{R}	80

Note from Table 22 that the PSD of the muckpile was used to calculate the weight of each range of particle size analysed in this study. The mean of the bin sizes was used for defining the parameters for each class range. The grade was adjusted (upgraded) according to the Grade Engineering theory, showing higher grades in the finer portion of the muckpile of broken materials.

It should also be mentioned that particles with an average radius of 0.025 cm were completely leached at all the tested times of operation. On the other hand, the coarser particles show a very low rate of leaching at the tested conditions. This shows that it is needed to extend the time of leaching and increase the temperature to rise the rate of recovery.

The biggest sizes, which are defined as the range of particles with a radius over 37.5 cm were not considered for the estimation of recovery, because the weight of this portion was estimated as less than 1% and the recovery that such large particles showed is negligible.

According to the results, the PSD of the particles should the most homogenous possible, in order to aim for a good recovery. It means that the particles shall carefully be blasted or fragmented to maximise a specific bin size to avoid the issues caused by fines, such as ponding. The low rate of recovery observed in coarser fragments is due to the impossibility of the solution to attack the minerals in the core of these particles.

Table 22 PSD per Level and its corresponding Rate of recovery

LEVEL	**FRAGMENTATION (PSD DATA)**			
	Particle radius (cm)	Grade (%)	Weight (%)	Recovery (%)
3800	0,025	2,34	1,12	100,00
	0,275	2,23	5,19	81,23
	1,5	1,86	21,79	4,09
	3,75	1,57	23,20	0,63
	8,75	1,28	33,91	0,11
	18,75	1,21	10,02	0,02
	37,5	1,19	4,60	0,01
	Total (%)	**1,19**	**99,83**	**6,41**
3560	0,025	2,43	1,16	100,00
	0,275	2,31	5,38	81,69
	1,5	1,91	22,31	4,13
	3,75	1,62	23,31	0,63
	8,75	1,32	33,45	0,11
	18,75	1,26	9,40	0,02
	37,5	1,23	4,84	0,01
	Total (%)	**1,23**	**99,84**	**6,66**
3320	0,025	2,12	1,11	100,00
	0,275	2,02	5,15	79,91
	1,5	1,68	21,60	3,96
	3,75	1,42	23,07	0,61
	8,75	1,16	34,04	0,10
	18,75	1,10	10,31	0,02
	37,5	1,07	4,56	0,01
	Total (%)	**1,07**	**99,83**	**6,25**
3080	0,025	2,03	1,10	100,00
	0,275	1,93	5,12	79,30
	1,5	1,60	21,51	3,89
	3,75	1,36	23,04	0,60
	8,75	1,11	34,10	0,10
	18,75	1,05	10,43	0,02
	37,5	1,03	4,52	0,01
	Total (%)	**1,03**	**99,83**	**6,17**

The results of the analyses are summarized in Table 23:

Table 23 Summary of the recovery per level and Mine total recovery

	STOPES	AVERAGE
LEVEL 3800	724	6.41
LEVEL 3560	1063	6.66
LEVEL 3320	1238	6.25
LEVEL 3080	725	6.17
Mine Total Average Recovery (%)		**6.38**

The average recovery at 730 days, 80 °C, with a fixed shape factor of 0.6 and a pressure of 400 psi for the entire mine was 6.38 %.

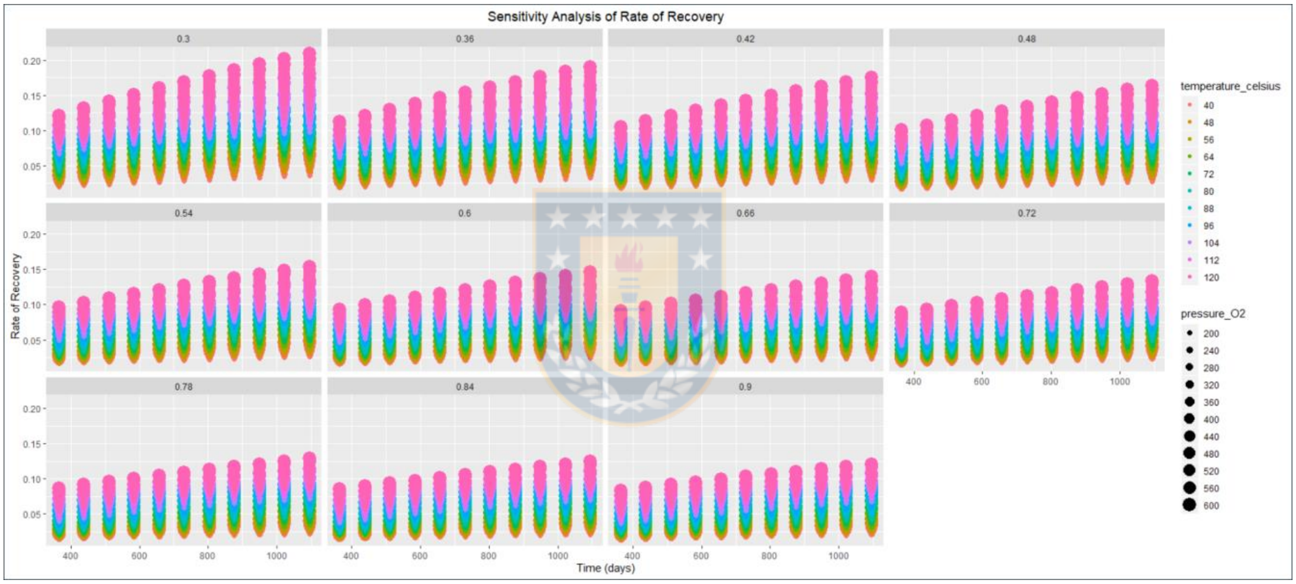


Figure 27 Plot of the Sensitivity Analysis Results

Figure *27* shows the results of the sensitivity analysis for the rate of recovery of stope leaching. The plots are separated for different shape factors to have a better visualization of the changes in the recovery.

As can be noted from the plots, the temperature is the most influential parameter affecting the rate of recovery. These plots show a difference in the recovery of almost 15% when the temperature increases from 40 to 120 As was mentioned before, the analysis shows that as the temperature increase the rate of recovery increases. However, a maximum temperature should be considered to make the operation safe and to avoid the passivation of the ore or activation of unwanted materials.

The analyses show that the time of leaching can also affect the recover, especially where the shape factor is low, showing a variation of almost 10 % when the time is increased 2 years. The operation time can be longer, but it should be considered based on the economics of the project.

In case that the shape factor is close to 1, the particles get hard to be leached because of the circular shape of the fragments. In other words, such fragments do not have favourable boundary conditions to allow the lixiviant to attack the core of the particles.

Pressure has a behaviour over the recovery relatively similar to time, showing a difference of 10% in the recovery between the lowest and highest value considered in the analyses. The change in the pressure is a constrain and makes it difficult to reach high values, because of the operational or design problems that a high pressure inside an underground stope might produce.

The low rate of recovery is explained by the low kinetics of the chalcopyrite plus the coarse p_{80} obtained from blasting. It should be noted that only chalcopyrite was considered for the calculations of the rate of recovery in the above-mentioned analyses. However, the presence of bornite could rise the rate of recovery due to its enhanced leaching kinetics.

8. CONCLUSION

The parameters analysed in this thesis are those that affect the most the rate of recovery. These parameters include but are not limited to fragmentation (Particle Size Distribution), mineralogy, leaching time, temperature, leaching solution and its components, pH, height of the pile of broken material, irrigation rate, aeration and hydrogeology.

The fragmentation has a large effect on the recovery. This is because both the Particle Size Distribution, PSD, and the shape of particles are crucial parameters governing the rate of chemical leaching processes. The fragmentation achieved from blasting of the stopes in underground mining generally has a characteristic of $P_{80} \geq 200$ mm. This should be considered in the selection of a kinetic model.

Of the four blast designs tested to identify the ranges of the particle sizes (the particle size distribution) resulting from each blast design, the Central Swedish Drift Design was the best option to achieve a finer fragmentation.

A systematic study is needed to characterise the ore sample before starting any treatment over broken ore for leaching. Large scale tests are also essential to understand the behaviour of the material during the leaching process. However, the high cost associated with such tests may considerably limit their applications.

Mineralogy is a fundamental parameter in analysing the leaching, for example, primary sulphides such as chalcopyrite and bornite reduce the rate of recovery because they are too hard to be leached. This can significantly increase the required time to obtain a good recovery.

A longer leaching time generally increases the rate of recovery. The reason is that time allows the chemical leaching solution to penetrate the core of rock fragment and chemical reactions can then be triggered in areas with high concentration of ore, inside the particles. The time of leaching should be considered in the planning of the IPM operations and on its economy.

Temperature is the other governing factor in the rate of recovery. Generally, increases of temperature improve the leaching rate. However, it is noted that in some cases the changes of the temperature can activate the reaction of unwanted materials, consequently, may adversely affect the reaction of bioleaching components.

The selection of the lixiviant and oxidant is generally performed according to the environment and the geological and mineralogical conditions of the ore. The concentration and irrigation rate of the solution are also important factors affecting the kinetic of leaching.

This comprehensive literature study and analyses also showed that aeration is a governing factor in leaching as it provides enough oxygen to carry out the oxidation of minerals. In general, the increase in the concentration of lixiviants produces a reduction in the pH. The pH value controls a good leaching yield. In some cases, when exists abundant carbonates surrounding the valuable metal the ammonia leaching is convenient, working in a basic range of pH. The height of broken material affects the recovery due to the possibility of channelling, ponding, increasing of the pressure over the fragments and consequently the bulk density of the material, affecting negatively the fluid flow through the mass of fragments.

A mixed kinetic model, whose fitting was tested previously in similar material conditions was used to estimate the recovery of leaching in the “Mine A” stopes. The grade was adjusted using the grade engineering arrangement. The design for blasting previously selected was also used. The author does not recommend using a recovery model without a previous implementation of an analysis of material and leaching tests. This is, because it can provide a good understanding and knowledge of the material aimed to be leached, to put in work the model with the correct data.

Due to low rate of recovery of copper obtained, the operation could not fulfil the economics requirements. On the other hand, in mine recover appears as a very low emission and low-contaminant method, therefore, if an operation can produce a green copper product, the price of this commodity might rise due to the rising commitment with caring for the environment from the industries in the world.

9. RECOMMENDATIONS FOR IN-PLACE MINING LEACHING PROCESS

Taking into account some problems and constrains that influence the rate of recovery for leaching in IPM, the following considerations are recommended from the author to improve the yield of the process.

The presence of fines in the ore, especially swelling type clays, can cause unacceptably low percolation and flow rates. Countermeasures are:

- Pre-wetting the ore with water or barren leach solution
- Agglomeration with strong lixiviant or polymer
- Minimizing the vertical free fall ore during heap/stope building

The optimized time for an operation must be selected according not only to improve the recovery, also costs must be analysed.

A further arrangement for IPM was discovered by the Author, this arrangement is highly dependent on economics of mine, fulfilling characteristics of IPM as environmentally friendly, in situ operations and grade engineering. This proposal is based on the experimental results presented by Madsen [10], where a higher recovery of minerals was achieved by removing the finest and coarsest part of the ROM fragmentation. This is explained because the remotion of fines allows to improve the permeability into the heap, dump, stope, avoiding channelling, ponding and acid consumption from clays. On the other hand, the remotion of coarse rocks that does not have a good recovery since they have a low degree of freedom of the ore. It also allows the air to circulate through the material providing the needed oxygen for oxidation of the mineral. Therefore, the proposal of the Author to make a higher rate of recovery is the following:

The proposal is called Rock Filling, which has a similar arrangement to stope leaching. The difference is the extraction of the entire stope instead of 20-30 % of the bottom of the stope in “normal” Stope Leaching, separating the 20-30% finest and coarsest part of material and sending the remaining part to fill with rock an already extracted stope. (see Figure 28)

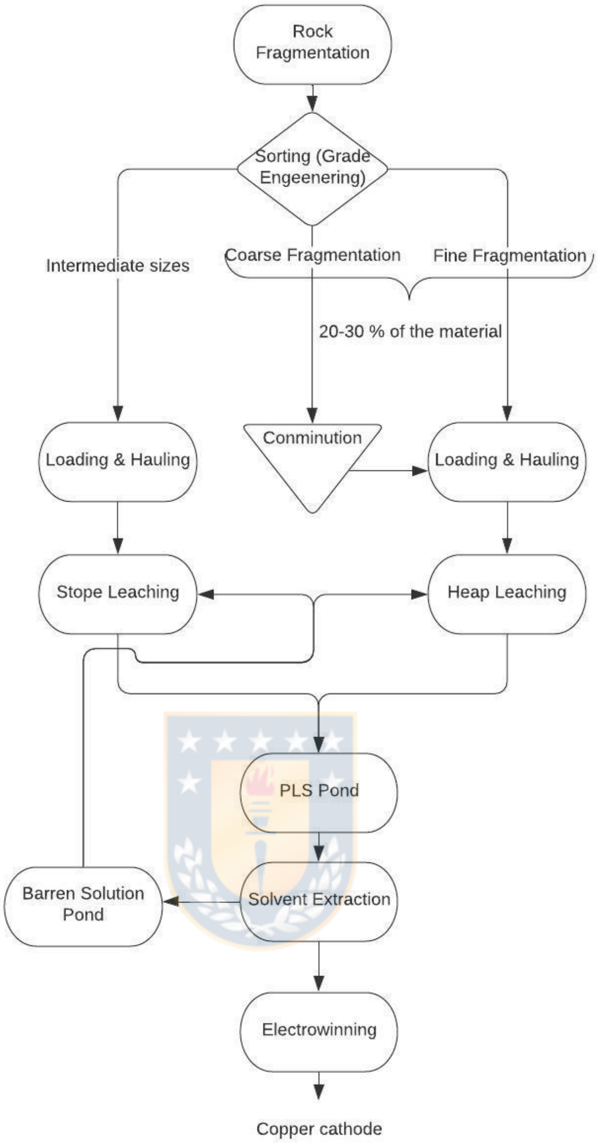


Figure 28 In mine recovery diagram for rock filling, case to improve the recovery in sulfides.

Table 24 Rock Filling and Stope leaching comparison

Characteristic	Stope Leaching	Rock Filling
Recovery	• ~2-30%	• At least 5% more than stope leaching, up to 40 % more.
Costs	• Economically cheaper due to mining costs decrease by developments are reduced and material movement is less, consequently less infrastructure and equipment are required. • A higher operational cost and Leaching infrastructure cost are required due to the necessity of technology solutions to improve recovery, as pipes for oxygen and solution injection and extraction. Lixiviant and water consumption increase due to presence of fines and the oxygen requirements.	• Could result in expenses for infrastructure due to more developments are required for rock filling. • More hauling of material is required. • Less leaching solution and ventilation is required because the recovery is highly effective since properties of material into stopes such as higher permeability.
Acknowledgment of Stope and Material	• There is not a good understanding of the stope in the operation and can be produced stability problems or leaks due to possible fractures in the stope walls.	• The stopes can be explored by drones, checking the walls, looking for failures or possible leaking potential. It can improve mine recovery because in the case of instability presence, the stope is only backfilled with cement and may consider the option of conventional mining, sending all the extracted material to existing processing plants.
Comminution and sorting	• Only the 20-30% of the bottom of the stope is extracted. Hence extracted ROM material must be sent to crushing and sizing stages to achieve a proper fragmentation for posterior processes such as Heap Leaching.	• The entire stope is extracted, only the 20-30% fines material is separated and sent to next processes. It must be noted that the finest part of the stope may contain the high-grade mineral as the mineral is weaker than rock, therefore, is liberated at blasting. • A huge comminution is not needed for the coarse part except for over size material which is difficult to handle. • Less crushing and milling stages are needed for fines due to most of the valuable mineral is already in a small particle size.
Time	• A shorter time of the cycle for material extraction per stope is needed. This is because of less material movement. However, a longer time to achieve an aimed recovery is needed due to the slower rate of Leaching. The time must be considered for the final NPV	• A shorter Time for Leaching is required due to better Leaching rate and yield. But, a longer time of the cycle of material movement due to the extraction of the entire stope is needed.

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11. APPENDIX

11.1 Appendix A

Interviewed people

- Ebrahim Fathi (CSIRO)
- Isaac Dzakpata (Mining3)
- Ewan Sellers (Mining3)
- Paul Lever (Mining3)
- Laura Kuhar (CSIRO Perth)
- Tania Hidalgo (CSIRO Perth)
- Gay Walton (CSIRO Perth)
- Rebecca Meakin (CSIRO Perth)
- Robbie McDonald (CSIRO Perth)
- Mario Rossien (CSIRO)

11.2 Appendix B

Table 25 Cost Unit CSRD

Cost Unit			
Parameter	**Symbol**	**Unit**	**Value**
Drifting + Support	UCe+s	\$/m	1200.0
Charge	UC_c	\$/kg	1.0
Drilling	UC_d	\$/m	21.5
Storing and Transporting Explosive Charge	$UC_{s\&tc}$	\$/kg	0.6
Loading Charge	UC_{lc}	\$/hole	10.0
Initiator	UC_i	\$/hole	35.0
Booster	UC_b	\$/hole	6.5

Table 26 Cost Unit CS

Cost Unit			
Parameter	**Symbol**	**Unit**	**Value**
Shaft Sinking/Raise Boring	UC_{rb}	\$/m	5000.0
Charge	UC_{c}	\$/kg	1.0
Drilling	UC_{d}	\$/m	21.5
Storing and Transporting Explosive Charge	$UC_{s\&tc}$	\$/kg	0.6
Loading Charge	UC_{lc}	\$/hole	10.0
Initiator	UC_{i}	\$/hole	35.0
Booster	UC_{b}	\$/hole	6.5

Table 27 Cost Unit DRD

Cost Unit			
Parameter	**Symbol**	**Unit**	**Value**
Drifting + Support	UCe+s	\$/m	1200.0
Charge	UC_{c}	\$/kg	1.0
Drilling	UC_{d}	\$/m	21.5
Storing and Transporting Explosive Charge	$UC_{s\&tc}$	\$/kg	0.6
Loading Charge	UC_{lc}	\$/hole	10.0
Initiator	UC_{i}	\$/hole	35.0
Booster	UC_{b}	\$/hole	6.5

Table 28 Cost Unit VCR

Cost Unit			
Parameter	**Symbol**	**Unit**	**Value**
Drifting + Support	UCe+s	\$/m	1200.0
Charge	UC_{c}	\$/kg	1.0
Drilling	UC_{d}	\$/m	21.5
Storing and Transporting Explosive Charge	$UC_{s\&tc}$	\$/kg	0.6
Loading Charge	UC_{lc}	\$/hole	10.0
Initiator	UC_{i}	\$/hole	35.0
Booster	UC_{b}	\$/hole	6.5

Table 29 Total Costs per Stope CSRD

Parameter	Symbol	Unit	Value
Preparation of Drifts	C_{dp}	$/Stope	192000
Charge	C_{c}	$/Stope	608861
Drilling	C_{d}	$/Stope	1955334
Storing and Transporting Explosive Charge	$C_{s\&tc}$	$/stope	365,316
Loading Charge	C_{lc}	$/Stope	35,280
Initiating Devices and Boosters	$C_{i\&b}$	$/Stope	369,516
Total Blasting Cost	TC_{b}	$/Stope	3,334,307
Total Fragmentation Cost (Blast + Drifts)	TC_{f}	**$/Stope**	**3,526,307**
Percentage of Drifting Cost per Stope	PC_{rb}	(%)	5.4
Percentage of Blasting Cost per Stope	PC_{b}	(%)	94.6

Table 30 Total Costs per Stope CS

Parameter	Symbol	Unit	Value
Shaft Sinking/Raise Boring	C_{rb}	$/Stope	1,200,000
Charge	C_{c}	$/Stope	157,680
Drilling	C_{d}	$/Stope	402,233
Storing and Transporting Explosive Charge	$C_{s\&tc}$	$/stope	94,608
Loading Charge	C_{lc}	$/Stope	12,070
Initiating Devices and Boosters	$C_{i\&b}$	$/Stope	100,181
Total Blasting Cost	TC_{b}	$/Stope	76,772
Total Fragmentation Cost	TC_{f}	**$/Stope**	**1,966,772**
Percentage of Raise Boring Cost for Each Stope	PC_{rb}	(%)	61.0
Percentage of Blasting Cost for Each Stope	PC_{b}	(%)	39.0

Table 31 Total Costs per Stope DRD

Parameter	Symbol	Unit	Value
Preparation of Drifts	C_{dp}	$/Stope	576,000
Charge	C_{c}	$/Stope	608,861
Drilling	C_{d}	$/Stope	1,955,334
Storing and Transporting Explosive Charge	$C_{s\&tc}$	$/stope	365,316
Loading Charge	C_{lc}	$/Stope	35,280
Initiating Devices and Boosters	$C_{i\&b}$	$/Stope	369,516
Total Blasting Cost	TC_{b}	$/Stope	3,334,307
Total Fragmentation Cost (Blast + Drifts)	TC_{f}	**$/Stope**	**3,910,307**
Percentage of Drifting Cost per Stope	PC_{rb}	(%)	14.7
Percentage of Blasting Cost per Stope	PC_{b}	(%)	85.3

Table 32 Total Costs per Stope VCR

Parameter	Symbol	Unit	Value
Preparation of Top Sill	C_{dp}	$/Stope	1,920,000
Charge	C_{c}	$/Stope	660,113
Drilling	C_{d}	$/Stope	681,120
Storing and Transporting Explosive Charge	$C_{s\&tc}$	$/stope	396,068
Loading Charge	C_{lc}	$/Stope	316,800
Initiating Devices and Boosters	$C_{i\&b}$	$/Stope	131,472
Total Blasting Cost	TC_{b}	$/Stope	2,185,573
Total Fragmentation Cost (Blast + Drifts)	$\mathbf{TC_{f}}$	**$/Stope**	**4,105,573**
Percentage of Raise Boring Cost for Each Stope	PC_{rb}	(%)	46.8
Percentage of Blasting Cost for Each Stope	PC_{b}	(%)	53.2

UNIVERSIDAD DE CONCEPCIÓN – FACULTAD DE INGENIERÍA

Departamento de Ingeniería Metalúrgica

Hoja Resumen Memoria de Título

Título:	INVESTIGATING THE PARAMETERS GOVERNING THE RATE OF LEACHING OF PRIMARY SULFIDES COPPER DEPOSITS IN IN-MINE RECOVERY
Nombre Memorista:	BRAYHAN ANTONIO SALAZAR SANHUEZA

Modalidad	
Concepto	
Calificación	
Fecha	
Prof. Fernando Parada L.	

Profesor(es) Patrocinante(s)	
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Comisión (Nombre y Firma)

Prof. Asieh Hekmat	Prof. Maria C. Ruiz	Dr, Ebrahim Fathi

Resumen

Debido a que el beneficio económico se reduce con el aumento de costos necesarios para la extracción del mineral en depósitos de baja ley con condiciones geomecánicas y geológicamente complicadas, además de la presencia de mayores restricciones medioambientales, se espera que existan pocos métodos de explotación viables en el futuro.

In-Mine Recovery, IMR aparece como una técnica amigable con el medio ambiente y que puede cumplir con los estándares necesarios para desarrollar minería en ambientes subterráneos con características complejas

El objetivo de este estudio es investigar el efecto de los parámetros que controlan la recuperación del metal en lixiviación. Parámetros tales como fragmentación, mineralización, temperatura pH, aeración, solución lixiviante, tiempo de lixiviación y altura de pila de material.

El caso de estudio considerado en esta investigación corresponde a la operación de una mina subterránea de cobre ''Mine A'' con sulfuros primarios. Los resultados mostrados para el caso de estudio son solo útiles para la estimación aproximada de la recuperación de cobre. Según el análisis de sensibilidad desarrollado, la temperatura es el factor con mayor efecto en la recuperación. Presión y tiempo son también factores altamente influyentes que deben ser analizados para un plan de implementación de IMR en sulfuros primarios.