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Techno-economic analysis of scenarios to develop the extraction of limestone for the southern area of Santiago deposit located in Estepa- Spain.

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Techno-economic analysis of scheduling scenarios to develop extraction of limestone in the southern area of Santiago deposit

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Abstract

The lime is a versatile product with multiple applications in everyday life. According to the European lime association, each citizen uses around 150 grams of lime per day. It is used as a base material to produce various products in environmental, agriculture, metallurgical production, construction, and chemical industries, among others. The high range of lime applications generates many limitations based on the content criteria set by the consumers and varies according to the requirement and further use of the lime.

According to the European lime association (2017), one of the highest lime uses is on steel production; 40% of the total world production is sold to the steel industry each year. The quality requirements for the lime to be used to produce steel are strict, setting the target for the obtention of lime with a high concentration of CaO and low presence of prejudicial impurities such as phosphorus or sulfur.

Calgov lime production plant is located in the south of Spain. It supplies local and national consumers of lime and has been operating for approximately 35 years. The steel manufacturing industry is one of the major clients of the plant consuming about 50% of the total installed capacity. This industry's quality requirements are a significant concern for the plant, and the obtention of limestone with low phosphorus content one of the primary targets for the company's future.

Currently, the plant is under high pressure to find and extract limestone with low content of P2O5, below 0.039%, which is the limit set by the client. Seeking to comply with quality and quantity, the plant has performed exploratory work in the southern area of the deposit in which resources of limestone that meet this target are found.

In this work, a mine design and scheduling options for the extraction of the south area of Calgov deposit extraction is presented based on the composition requirements set for the limestone. Furthermore, a techno-economic analysis of the extraction scenarios is developed exploring different alternatives to deal with the lack of information about some areas of the deposit and alternatives to ensure a constant feed of lime to the plant, avoid over costs and increase the possible economic return.

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Abbreviations	
AL2O3	Chemical composition of aluminum III oxide in the limestone by percentage
CaCO3	Calcium Carbonate
CaO	Chemical composition of calcium oxide by percentage
CAPEX	Capital expenditures
Fe2O3	Chemical composition of iron III oxide in the limestone by percentage
LOM	Life of Mine
m	meter
m.a.s.l	Meters above sea level
mm	Millimeter
NPV	Net present value
OPEX	Operational expenditures
P2O5	Chemical composition of P2O5 in the limestone by percentage
RBF	Radial basis estimation function
SiO2	Chemical composition of silica the limestone by percentage
t	ton
UPL	Ultimate pit limit

1. Introduction

The extraction of limestone and lime production has been undertaken since ancient times; even though no one knows when humanity discovered the lime, it had been used in the construction of houses since 7500 BC. It was only until the 18th century when Black and Lavoisier carried out works to describe the lime chemical reaction, this work opened the door to understand and study its fabrication process. In this same century, Debray and Lechatelier explored its characteristics and applications; After this, in the 19th century, lime's industrial production has started. (LHOIST, 2019)

The lime is obtained by the process of calcination of the limestone. It is extracted from the mine and sized by crushing it in several stages until it reaches the required size to be fed to the kiln. It is then calcined by the heat produced in the fuel combustion process. The heat is transferred to the limestone, and it is transformed into quicklime CaO and carbon dioxide CO2. The quicklime CaO leaves the kiln and is cooled and crushed. It can be commercialized as quicklime or transformed into hydrated lime in a further process that adds water to it. The multiple uses it has in several industries make it an essential product with high demand almost everywhere. (European Lime Association, n.d.)

The use of lime is crucial for steel manufacturing to obtain homogeneous steel and remove undesirable impurities from it.

As time has advanced, the technical requirements and complex steel applications have increased, setting new requirements for better quality and different concentrations of certain elements such as sulfur and phosphorus that may affect the steel strength, hardness, ductility, and toughness. (Bell, et al., 2006) This change in the technical requirements for steel has caused an increase in the required control over the concentration of these elements in the raw materials used, setting new limitations for the lime industry. (Schifman, 2018)

To obtain the desired results in the steel, the lime used must have a low content of impurities and a high content of CaO. Below is the range of quality specifications for the lime used in the steel industry (Ponchon & Manocha, 2018)

	Specifications for the steel industry
CaO %	92% - 98%
SiO2 %	<2%
S %	<0.06%
P %	<0.06%
Al2O3 %	<0.5%
Grading	10 mm – 30 mm
Fines <5mm	10% max

To meet the steel industry's demand, the lime producers need to implement controls over the desired amount of trace elements found in the limestone; therefore, they need to identify target zones and plan the production following the strict composition needs. In this scenario, mine planning and scheduling for the extraction of limestones consist of searching for the adequate sequence to remove the material inside the pit limits that meet the processing plant capacity, the content constraints of the lime imposed by the steel industry consumers, in addition to the existent physical constraints and the mining production capacity. (Kumaral & Dowd, 2005)

Calgov is a lime producer that supplies a significant portion of its daily production to the steel industry as one of its core products. This issue directly impacts the quality of raw materials they can use as the limitations are strict. One of the main constraints in the limestone composition for this product is the phosphorus concentration. The phosphorus content in the raw limestone must satisfy the permitted limits, which for Calgov are particularly strong, requesting a limestone with phosphorus content lower than 0.039%.

1.1. Problem Statement

The lime production has been carried out in The Calgov plant for more than 30 years in which the limestone used has been a mix of own extraction performed in the north area of the Santiago deposit and limestone bought from other mines in the nearby area. The plant processing capacity has been fractioned into two lines running simultaneously. One produces lime with low phosphorus and sulfur content destined to supply the steel industry; the other is producing lime with a higher content of the mentioned trace elements for different applications.

As mentioned before, the limit imposed by the steel industry consumers regarding the phosphorus content of the limestone used for lime production is stringent, setting a maximum percentage concentration of 0.039% of P2O5. Dealing with this quality demand presents a complicated situation since the quarry, located in a sedimentary limestone deposit with several horizons in which the concentration of this element is higher than the allowed limit. The current area in which the extraction is carried out does not have reserves that meet this specification.

All the low P2O5 limestone is currently bought since the limestone extracted in the Calgov mining concession exceeds the target limit. This issue places the company in constant dependence on external suppliers of the raw material and in a risk position of not meeting the demand set by their client in case of a disruption in the other mines supply.

To reduce the risk position in which the company is now, the opening of new mining areas located in the southern portion of the same license has been considered. Exploration was undertaken to study the potential of the area for limestone extraction and to define the interest areas in which low P2O5 concentrations are found.

It is crucial to define a mine plan for the new areas that the Calgov plant wants to extract. This planning must ensure the supply of the limestone needed in compliance with the quality targets.

In addition, it needs to establish strategies for the possible blending of blocks with low phosphorus concentration with those with medium or high content to extend the life of the quarry. (Ito & Nishiyama, 2003)

The planning of the southern area can help to determine the best technical and operative sequence to extract the resources present in this area as well as provide an approach to the investments needed to carry out the extraction and the expected financial results.

1.2. Hypothesis

Based on geological mapping, exploration information, and financial information, it is possible to produce a plan and a techno-economic evaluation for the extraction of limestone with low P2O5 content required to keep the constant feed of the lime plant using the reserves found in the southern area of the Santiago deposit.

1.3. Objective

This work's main objective is to define a plan and economic evaluation of scenarios for the extraction of the southern area of the Santiago deposit currently extracted by Calgov. From this general objective, several detailed objectives have been set.

- Objective 1: Determine the pit and road design following the geotechnical parameters in the area. Based on the exploratory results and the space limitations, a pit design is made to extract the reserves found in the southern area of the Santiago deposit.

- Objective 2: Produce an extraction sequence in accordance with the current plant requirements and determine the needs of equipment for both mine and crushing plant. Furthermore, identify the space needs for the waste management

- Objective 3: Analyze the possible economic results of the extraction and define the main parameters affecting them. Based on the current cost scheme of the northern pit extraction, analyze the possible results obtained by opening the southern area, and explore different scenarios in which the extraction may be developed.

1.4. Research questions

The mine design and planning presented in the thesis work can help to have a first approach to the planning for the extraction of the new area. It will allow the visualization of the required changes in the extraction and processing of the limestone, define possible problems found for the development of the extraction in this area, and recognize the parameters affecting the financial results.

To reach the proposed objectives three research questions were defined:

1. Is it possible to ensure a constant extraction of limestone in the southern extension area that meets the steel industry requirements?
2. Which are the main bottlenecks and problems found for the extraction development?
3. What are the investments required and the possible economic results?

1.5. Methodology

The mine planning of a new extraction area requires identifying several parameters that shape the overall mining result. Collecting information about them provides the base to proceed with a further technical and economic analysis of the extraction possibilities.

The thesis development has started with collecting data regarding the geological setting, space availability, and production requirements. With this information, an analysis is made to establish the relationship between elements in the deposit and set a mine design and ultimate pit limit that allows defining the extractable portion of the overall mining license.

Further analysis is made on the equipment required and the current infrastructure to define the modifications needed and establish the base information for the Capital cost estimations.

In a second stage, the mining sequence with different extraction directions is analyzed to establish the best option considering key factors in the ore extraction, the space requirements, and the possible environmental restrictions. For both the pit design and the scheduling, GEOVIA software is used, Surpac and Minesched.

Once the extraction sequence is defined, the economic evaluation takes place by calculating possible NPV for different scenarios. Operational costs are estimated based on the current costs of extraction, and the revenues are based on the current limestone prices. Finally, an analysis of the financial results is made.

1.6. Scope

The study case for the mine planning is developed in the Calgov quarry and therefore is site-specific. The work included the next items as part of the scope:

- Data analysis
- Pit design
- Extraction preliminary sequencing
- Equipment requirement calculations
- Analysis of possibilities for crushing plant modifications

- Waste handling space requirements
- General cost calculations
- Analysis of possible financial scenarios for the long-term extraction

As mine planning is a complex process, several other items are not considered in this work. Missing base information to evaluate the influence of some parameters in the plan makes them to be out of the scope of the project.

- Environmental impacts assessment
- Legal assessment and permitting
- Detailed waste dump design
- Waste management program
- Mid and short-term planning
- Land reclamation
- Loading and truck cycle optimization
- Blasting design and optimization

1.7. General outline

The work is divided into several chapters in which all the stages in the mine plan are described. In the first chapter, an introduction to the problem is made, providing information on the work's hypothesis, objectives, methodology, and scope. In the second chapter, a literature review on mine planning and economics is presented to set the framework on which the project is based.

The third chapter describes the specific case of study and the available information. Chapter four includes exploration data analysis. It provides an overview of the exploration and geology work performed to understand the behavior of the deposit in the southern area. The block model constructed is described as well in this chapter.

Chapter five comprises the mine planning technical information, including the production requirements, the extraction scheduling, the equipment required, plant modification requirements, and alternatives, in addition to a brief approach to the waste management by establishing the amount of materials to be produced, the handling and disposing space requirements.

Chapter six includes the financial information, including the extraction costs, the economic scenarios considered, and the sensitivity analysis. Finally, chapter seven and eight include the discussion, recommendations, and conclusions.

2. Literature review

This chapter presents background information on the lime production process, lime used in steel manufacturing, mine design and planning, and economic evaluation of mining.

2.1. Lime production process

Lime is a versatile material; due to its alkalinity, it can purify and neutralize. It is used as an essential component in many industrial sectors such as agriculture, environment, steel manufacturing, paper manufacturing, among others. Lime results from calcinating limestone, which is mainly composed of calcium carbonate ($CaCO_3$). The main goal of the calcination process is to obtain quick lime (CaO). The chemical reaction that occurs inside the kiln during the calcination is stated in figure 1. (European Lime association, 2017)

$$CaCO_3 + \text{Heat} \rightarrow CaO + CO_2$$

Figure 1: Calcination reaction. (Calcinor, 2015)

The process of producing lime is shown in figure 2. It starts with the limestone extraction in the quarries either by drilling and blasting or using mechanical extraction methods. Once the mineral is mined, it is taken to a processing plant, where the sizing process occurs. The rock is broken by crushing and screening in several stages until it reaches the feed size required according to the Kiln design. The fines and undesirable particles are taken out before going into the calcination process. (National Lime Association , 2019)

The calcination process has three main stages. In the first one, the limestone is preheated using the gases that come out of the kiln. This stage takes place in the pre-heater and is where the heat transfer to the stone occurs. The second stage is calcination. It occurs inside the kiln, where the combustion of the fuel burns the limestone. The chemical reaction occurs as it moves through the kiln going from the pre-heater to towards the cooler. The applied heat turns the limestone into quick lime (CaO) and CO_2. Finally, the lime comes out of the calcination zone and reaches the cooling area where the contact with the cooling air makes it lose heat. The air used in the cooling process is warmed by heat transfer from the burned lime and used as the combustion air; this makes the process more efficient in using the heat.

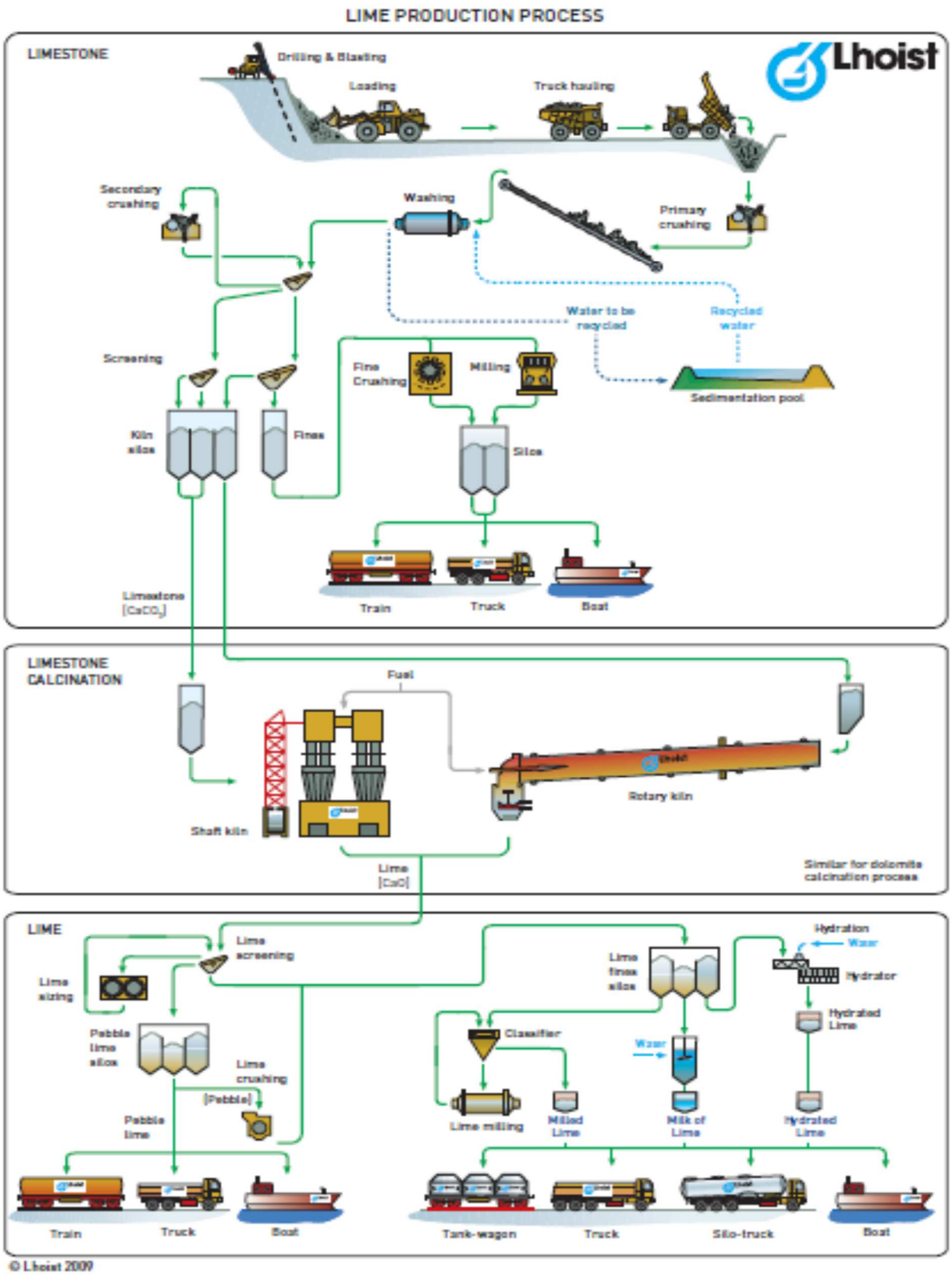


Figure 2: Lime production process. (Lhost, 2009)

The lime can be sold both as quicklime CAO or treated to produce other lime products like hydrated lime Ca(OH), milk of lime, and Precipitated calcium carbonate. All these products are obtained by adding water to the crushed lime. Figure 3 shows the production cycle of these products.

Figure 3 shows the lime production cycle, as it can be seen to produce hydrated lime, pulverized water or steam is used to produce calcium hydroxide. The milk of lime results from mixing hydrated lime with water to form a homogeneous suspension and the precipitated calcium carbonate is obtained by hydrating high calcium quick lime and adding Carbon dioxide to make the slurry (milk of lime) react to produce highly pure $CaCO_3$. (European Lime Association, n.d.)

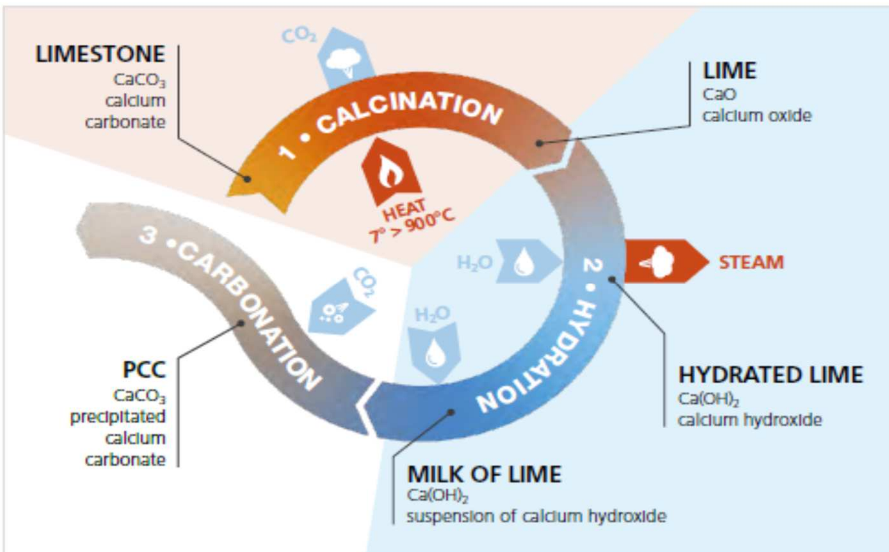


Figure 3:Lime Cycle. (European Lime association, 2017)

2.2. Lime in the steel manufacturing

Quick lime is highly important in the steel manufacturing process. The European lime association estimates that approximately 40% of the total lime production was destined to meet this industry's demand during 2017. Lime is used to remove impurities present in the iron ore and other raw materials used for steel manufacturing giving the right composition to the final steel product. According to figures of the National lime association in the US, for one ton of steel, it is needed up to 70 kg of lime. (National Lime Association , 2019)

The quality of the lime used is a determinant factor for the success of steel manufacturing. It directly relates to the slag quality affecting the production results, the cost, and revenues obtained from the steel production. Quick lime is used in many phases of the steelmaking process; it is used mainly as a flux agent but also to balance the acidity in the agglomeration process before sintering and as an essential element in the desulphurization process.

The raw materials used in steel production usually contain impurities and undesired elements such as sulfur, phosphorus, silica, among other oxides that affect the overall metallurgical result of the steel production process. Therefore it is necessary to remove them as much as possible. The lime can remove these impurities to a certain level, but the efficiency of the reactions occurring during this purification process will rely on the lime quality and reactivity. (Manocha & Ponchon, 2018)

Calcium oxide (lime) performs three primary functions during steel processing. The first one is in the formation of slag. It captures the undesired elements, protects the metal from the atmosphere components such as nitrogen and hydrogen; besides, it acts as isolation allowing to maintain the temperatures of the process and, therefore, to have more efficient use of the energy. (CALCINOR, 2015)

The second function is in phosphorus removal. Phosphorus content in the steel affects the ductility. It makes it easy to fracture during the handling and when performing cold works with it. The quick lime extracts the phosphorus by forming calcium phosphate, which is eliminated with the slag. Finally, the lime helps in the sulfur removal. Its addition lowers the percentage of sulfur in the steel composition and therefore improves the properties of it. The presence of sulfur content causes fragility in the steel, making it prone to form cracks.

Lime is used in every stage of the process, both in the primary steelmaking and secondary steelmaking. It is used in the blast furnace during the fusion process for the first casting. Also, in the electric arc furnace for the secondary fusion and the ladle furnace for the refining process. (CALCINOR, 2015)

2.3. Mine Design

The mine design process deals with the geometric and space constraints present in the deposit and the legal definition of available land. It starts by analyzing the information obtained in the exploration process to define the structures and disposition of the ore in the interest area and finish with the establishment of an ultimate pit in which the extraction is carried out based on some technical and economic parameters.

According to the Society for Mining, Metallurgy, and Exploration (SME), the mine design process is a decision-making process in which is necessary to collect information on the resource availability (Mineral Inventory), operation parameters, existent or new alternatives for the extraction and processing, information on mining and environmental laws (mining rights), and economic conditions. All this information is analyzed in the search for the best configuration of the pit. (Whittle, 2011)

2.3.1. Block modeling

A block model represents the ore body in which it is discretized in a three-dimensional arrangement of blocks, each block representing a spatial square or rectangular volume in the deposit with specific geological, physical, and geochemical characteristics. The blocks are arranged one adjacent to the next one, and the size of each block is variable for each deposit. Block modeling is one of the first steps taken for the creation of mine design and plan. It provides the base and crucial information for determining the amount of resources in the mining area and the reserves that can be extracted.

The block model construction is made using the information obtained in the exploration sampling and the geostatistical analysis made on the exploratory data. The estimation of each block composition is obtained by geostatistical simulation (interpolation) of the sampled data points. It allows the geologist and engineers to get information on the entire orebody based on the scarce data obtained in the sampling campaign. (Reyes Jara , 2017)

The main objective of the estimation process is to reproduce as accurately as possible the in-situ content of the blocks and to quantify the in-situ variability of the rock attributes following the spatial distribution of the orebody. Several estimation methods are used for the geostatistical simulation and the determination of the blocks' attributes; some of the most common ones are Kriging, Radial based function estimations, inverse distance method, gaussian simulation, among others.

One of the main assumptions made in mine planning and scheduling is the accuracy of the model and its ability to represent the real distribution of the ore and waste in the deposit. Therefore, the success of the mine plan and financial results are significantly impacted by the deviations of the model from reality. (Kumaral & Dowd, 2005)

2.3.2. Mine design process

Once a deposit block model is obtained, the mine design phase begins by delimiting the part of the deposit of especial interest and for which the extraction is designed. This delimitation obeys not only economic parameters but also geological, geomechanical, operational, and environmental variables, among other factors that affect the extraction development.

The main objective to be reached in the mine design process is to determine the geometrical design of the extraction that allows maximizing the amount of ore extracted while reducing the waste generated and keeping the safe operation of the mine. Each step taken in the design process requires an analysis of the associated risks and knowledge of the variables interacting in the result.

2.3.2.1. Slope design

The slope design process seeks to establish the pit configuration that allows carrying the extraction process in a safe way. In the design process, the pit slope angles are determined as steep as possible to maximize the extraction of resources and increase the economic results of the mine. it is essential to keep an equilibrium in which over steepness must be avoided to prevent instabilities that lead to safety risks and therefore affect the profitability of the mining activity. (Knight Piesold.Ltd, 2015)

The stability of the mining benches, ramps, and roads are highly dependent on the structures present on the rock and the mechanic interaction between them. Hence, slope design is determined mainly by the nature of the rock mass in which the mine will be developed. The geological structures as faults and joints, the rock constituents and their mechanical behavior, the weathering state, and some external factors as water and other perturbations caused by human activities. The stability of the mining benches, ramps, and roads are highly dependent on the structures present on the rock and the mechanic interaction between them. The knowledge of rock mass behavior allows us to monitor and control the deformation and failure processes in the mine. (Fleurisson, 2012)

The first step in the slope and mine angle determination is collecting geological and geomechanical information on the rock mass to identify the structures present in the area, the rock's strength, and the stress conditions present. This information is the base for the rock mass characterization. The second stage is to analyze the possible deformation and failure mechanisms acting on each zone of the interest area. Finally, with the data collected in the previous two phases, the slope design is established; the benches and berms configuration is defined, the slope angles are determined, as well as the need for monitoring and reinforcement.

The geological and geomechanical information is obtained by sampling, measurement, and field observation mixing methodologies used in geological exploration, geotechnical studies, and rock mechanics. Information on the rock structures, discontinuities, faulting, and weathering conditions is collected during the fieldwork in a systematic survey of the outcrops and provides the necessary data to follow up on the analysis. Furthermore, the geological structures are measured and carefully assessed to establish their orientation and stability behavior. (Fleurisson, 2012)

Some of the factors measured and analyzed in the slope design work are:

- Rock type and weathering conditions (weathering horizons, from very weather rock in the surface to an unweathered rock in rock mas bottom)
- Geological structures: faults, discontinuities, beddings, Shear plains, and joints.
- Rock mass quality: rock mass strength and rock quality classification based on geomechanical properties and classification, for example, the rock mas rating system, the cohesion, friction angles, deformation, among others.

- Water and climate conditions: the amount of surface and groundwater in the area, thermic stress due to season changes, runoff water, and drainage possibilities.
- Stress conditions: the rock mass has a preliminary stress regime is modified with the excavation of roads and the extraction activities. The stress state change through time can lead to a loss of rock mass quality affecting the overall slope stability and causing changes in the final pit wall.

The ultimate pit limit configuration must follow the guideline obtained in the slope design and the stability study. Also, the constant stability survey of the pit walls and roads must be made to ensure safe operation in the entire life of the mine. (Kliche, 2011)

All the analyses made and geomechanical information led to the construction of the pit design and the stability modeling process in which the possible failures are studied to establish the critical deformation events and prevent large failures that may occur during the extraction. The geomechanical model of the pit must identify the kinematical behavior of the rock by establishing the interaction between discontinuities, joint families, and mine elements as benches and openings. It must foresee the occurrence of failure mechanisms such as plane failures, wedge failures, topping failures, and circular failures; also, the scale in which they affect the extraction by determining the volumes of unstable rock and the risks associated with these failure mechanisms. (Seedsman, 2011)

Figure 4 shows graphically the type of slope failures that can be present. The planar failure (a) occurs typically in sedimentary rock mass in the bedding joints, also in the presence of foliation planes, schistosity planes, and in the contact between clay and bedrock. In the wedge failure (b), two or more structures form a detached block that slides at the intersection of one of the two planes. The topping failure (c) is caused by the loss of strength in the toe of the bench. The circular failure (d) is present in conditions of rotational shear and general surface failures in this type of failure; any geological structure controls the failure.

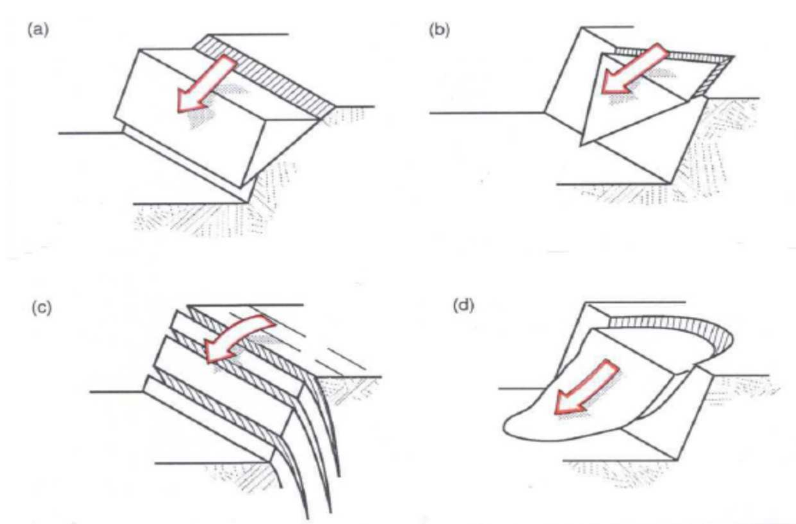


Figure 4: Types of slope failure. (Fleurisson, 2012)

The final pit design must be made based on the geomechanical model results considering the predefined safety factors for the bench design and the slope angles. The safety factor ensures the stability and equilibrium of the forces acting in the mine slopes. The pit design must consider the inaccuracy in the data obtained in the sampling and measuring process and therefore use preferably the lower values found for the mechanical properties of the rock mass. In practice, the safety factors used vary in function of the exposure time to the failure risk, working with factors of about 1.3 for short term stability analysis and 1.5 or even higher values to guarantee long term stability. The selection of the safety factor is defined by the design engineer based on the experience, mechanical properties results, and type of work to be developed in the area. In all the cases, the safety factor must be superior to 1. (Fleurisson, 2012)

In addition to the slope design, several actions can be useful to improve pit stability and increase the slope angles to maximize the extraction. Examples of this are the evaluation of different extraction directions and methods to minimize the stress concentration in the operation areas. Also, the proposal and estimation of activities as reinforcement in areas with high risk, drainage systems to control the water effect on the stability and monitoring systems may increase overall stability. All these elements and systems designed to increase the slope stability carry on a cost for the production, making it necessary to find an equilibrium between the cost and the benefits obtained.

The final geometry and parameters to establish are shown in figure 5. the overall slope angle, the inter ramp angle, and the bench angle are set as a result of the slope stability analysis and calculations. The bench height, berm width, and the size of the ramps are determined by joining the stability criteria with the production criteria related to the equipment size and the operations need.

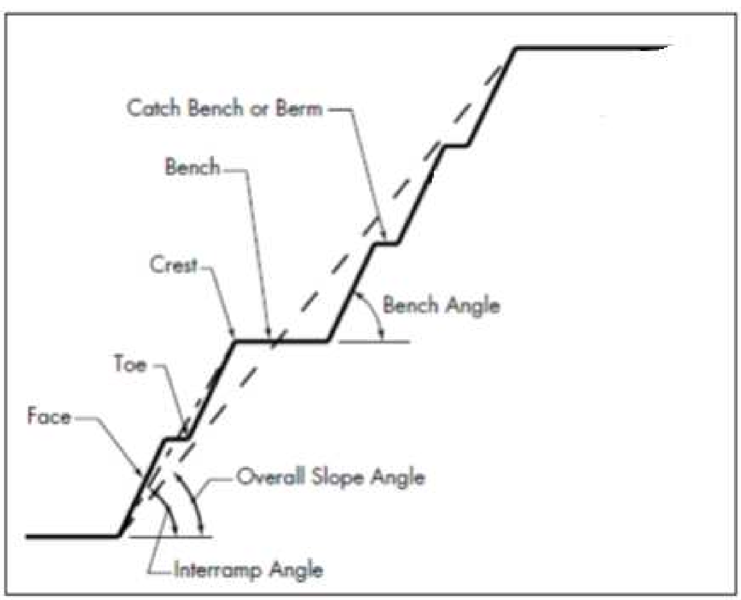


Figure 5: Pit design variables. Modified from Slope stability, Mining engineering handbook, Kilche 2011, p 497.

2.3.2.2. Road design

The haul roads are one of the most important components of an open pit mine. They provide access to the material of interest, waste and serve as the route to take it to the processing plant or, in some cases, even to the waste dumps. The adequate planning and construction of the mining roads can represent a crucial point in cost reduction, transport efficiency, and mine safety. (Kliche, 2011)

In the road design, several parameters must be considered to get a functional and efficient road system. The hauling roads of a mine built a system or network to provide flexibility and efficiency to the overall operation. The road's geometric design needs to be done according to the road function, traffic type, service level (main road or secondary road), traffic volume, and general land conditions (weather, soil resistance, inclination, among others). (Kaufman & Ault, 2001)

The haul roads need to have a proper width and gradient following the local legislation and the operation particularities. In Spain, the mining code establishes the maximum mining road grade to be 20 %. Any mining service road can overpass this limit, but the grade can be lower depending on every specific mine site conditions. In addition to this disclosure, the haul road needs to be wide enough to ensure the safe operation and transit of the mining trucks.

The width of the road is determined by the wider truck used in the mine. Figure 6 shows the factor to determine the width of the road. It changes as a function of the traffic volume (1 or several lanes). In the road design, it is also needed to consider the runoff water drainage to keep the road in the right operational conditions. Furthermore, it is necessary to consider the construction of safety berms on the sides. These berms should have a high of approximately 66% of the wheel diameter. The space requirement for these berms depends on the type of material used for its construction and the natural angle of repose of it (Thompson, 2011).

Number of Lanes	Factor x Width of Largest Truck on Road
1	2
2	3.5
3	5
4	6

Notes

For switchbacks and other sharp curves and/or roads with high traffic volumes or limited visibility, a safe road width should be designed with an additional 0.5 x vehicle width.

A four-lane road is recommended where trolley-assist systems are in use.

Figure 6: Road Width factor. (Thompson, 2011)

The mining roads usually must cross through areas with severe topographic conditions, and the construction of curves, switchbacks, and complicated turns cannot be avoided. Therefore, it is also essential to consider the inclusion of the superelevation to compensate for the centrifugal forces' effect allowing the trucks to have a fluent drive in the curves. The superelevation should not overpass 5 to 7%, and it needs to be established based on the curve radius and the speed of transit. (Thompson, 2011)

2.4. Mine Planning and scheduling

The mine planning and scheduling objective is to find the optimal mining conditions and sequence based on different limitations set by the physical distribution of the ore, the geological conditions of the area, the environmental and policy restrictions, and the operational and production constraints. The mine plan must seek to meet several economic targets, including maximizing the value and the income obtained by the mining activity while meeting the customer expectations in quality and quantity, ensuring the safety of the operation, and avoiding or mitigating the environmental impacts.

The long term mine planning approach is to design and plan the extraction of the available reserves to maximize the life of mine, meet the processing plant requirements, and ensure an optimal mining process development. It is a complex process due to the high number of variables involved in the process. Furthermore, it must deal with uncertainties in the spatial distribution of the ore and waste due to the little information obtained in the geological sampling process and the variability of the elements in the deposit. (Benndorf & Dimitrakopoulos, 2013)

The final layout of the mine, called ultimate pit limit (UPL), is generated based on the ore grade (quality targets), the available mining area established in the mining license provided by the government, the environmental and social restrictions, and the spatial location of ore and waste in the deposit. All these issues, in conjunction, determine the maximum valuable pit that can be extracted. (Sattarvand & Niemann Delius, 2013)

As mentioned before, the open-pit mine planning involves the interaction of multiple variables related to each other in a synergy in which a change in one of them can change the behavior of all the others, causing a failure in the entire plan. At the beginning of the project, it is necessary to establish the quality target ranges to distinguish the ore and waste material and define the reserves.

The definition of the targets is based on economic variables as the price of mineral, the mining and processing cost, and the market conditions. With the targets, an ultimate pit limit is defined, a production schedule is created, and an estimation of the economic results, costs, and revenues is calculated for the sequence. The financial results obtained are a new set of variables used for the target calculations getting back to the beginning of the process. (Sattarvand & Niemann Delius, 2008)

It is an iterative process in which multiple conditions can change the result, making it difficult for engineers and qualified professionals to evaluate the interaction of all the variables and get an optimal result. Due to the complexity of the process, some computational tools have been created and continue evolving, looking forward to help in the evaluation of scenarios and variables in the mine planning process. (Benndorf & Dimitrakopoulos, 2013)

Several software packages and optimization tools help the engineers in the design and evaluation of the mining production schedules to determine the best possible sequence counting the processing plant and mining equipment capacities.

The mining schedules are generated by periods in which the extraction of specific zones of the mine is done. For every period, a polygon is generated, all the blocks contained in it are extracted, obtaining a sequence of small pits that build the overall schedule of the mine. (Benndorf & Dimitrakopoulos, 2013)

For the extraction of each polygon in the sequence, roads and ramps, and other auxiliary designs are made to allow access to the mineral. Thus, the long term mine planning establishes the minable reserves per period in a production schedule. The mining activity is then developed as close as possible to the mining plan until reaching the ultimate pit limit. Changes in the fixed sequence of extraction affect the financial results of the mining activity. (Sattarvand & Niemann Delius, 2013)

2.4.1. Scheduling of the extraction

The open pit scheduling deals with when a block must be extracted (time for the extraction) to achieve the best financial results from the mining activity. It means the sequence of extraction that maximizes the net present value of the entire operation. The problem solution is ruled by a set of different constraints that need to be satisfied to meet the objective, such as shape constraints related to the area required for the operation and the geomechanics conditions (stability issues of the deposit). Precedence constraints reference the block's position inside the deposit whit respect to all the blocks surrounding it, which blocks must be extracted previously to release the desired block. Operational constraints, capacities for both mining equipment and processing plant. Grade constraints, minimum, and maximum desirable elements content. Besides, in some cases stockpiling constraints based on the storage capacity are added to the problem.

The problem is highly complex not only because of the natural complexity in the deposit but also due to the high number of variables to consider. Some variables interacting in the problem are the number of blocks in which the deposit is discretized, time expressed in the number of periods for which the extraction is planned, physical and operational constraints. All this led to having up to millions of variables to evaluate when seeking for a solution.   (Espinoza, et al., 2013)

As mentioned before, the scheduling problem deals with the extraction sequence of the blocks estimated in the block model. Even though numerous extraction sequences may be possible, only one is optimal in getting the lowest cost possible while meeting the quantity, quality, and physical constraints.

Currently, any specific method is commonly accepted or marked as the best practice for doing the extraction sequencing of a mine. There is a wide variety of methods, software proposed, and multiple approaches are taken when looking at the different models to solve the mine scheduling. Some of the methods studied for the problem-solution are heuristic methods, linear programming, dynamic programming, mixed-integer programming, among others. All the previously mentioned methods rely on several assumptions made to simplify the problem, and they are only valid under the assumed conditions. (Kumaral & Dowd, 2005)

The assumptions for which the models that are stated as valid are set in the initial discretization of the deposit in blocks, each with specific characteristics. All the scheduling formulations and constraints applied are expressed as dependent on this block definition, and the characteristics of each block are assumed as constant over time.

The characteristics set in the block model construction and estimation are considered an accurate representation of the reality even though this may not be true due to the uncertainty in the estimation and the geological conditions. The above means that the models are important tools in understanding and solving this complex problem to a certain extent, but they fail to be 100% accurate and represent the reality of mine operation. (Espinoza, et al., 2013)

Due to the high complexity of the task and regardless, the method used, only sub-optimal (near-optimal solution) can be found. The main idea is to get the best near-optimal schedule among all the possible schedules. The estimation uncertainty, the representation, and modeling of the geological process and the computing capacities are limitations and still represent a challenge for the problem solution. (Kumaral & Dowd, 2005)

Currently, some software helps engineers in the evaluation of the scheduling possibilities, allowing them to have a previous simulation on the mining activity and its results. This computational tool, combined with the criteria, knowledge, and expertise of the geologist and mine engineers, can lead to schedules that meet the extraction objectives and give the best possible use of the available reserves.

2.5. Mining economic evaluation

The economic evaluation of a mine extraction based on the reserves obtained in the exploration results is a complex process since it involves many assumptions on how the production and the market will be in the long term. Additionally, it is a crucial step for the development of the project since it helps in the determination if it is possible to perform the extraction under the current established technical conditions.

The economic evaluation of mine extraction involves the work with many variables with high uncertainty. The information on which it is based is often limited or based on engineering assumptions on the contained materials (ore and waste) in the methods and results expected from the extraction activity. It is necessary to be careful when doing estimations to avoid underestimating costs or overestimating the revenues.

The financial analysis of the project is made based on the annual cash flow estimations of the entire life of the project. It is built with the possible costs and revenues for each year. The economic analysis is then made based on the net present value economic indicator (NPV) at selected interest rates. (Orae, et al., 2011)

The annual cash flow is calculated based on equation 1, in which the revenues and costs are included to obtain the financial result of the company. Using the previous result, the NPV is calculated based on equation 2

$$Annual\ cash\ flow = Revenue - Costs \quad (1)$$

$$NPV = \frac{Annual\ Cash\ flow}{(1+i)^n} \quad (2)$$

$$n = time\ (number\ of\ years)$$

$$i = discount\ rate$$

In equation 1, the costs include both capital expenditures or capital costs (CAPEX) and operational expenditures or operational costs (OPEX).

The NPV calculation needs to be evaluated in different scenarios to establish the economic results in the best and worst conditions the project may face throughout its life. Furthermore, a sensitivity analysis must be made to reduce the uncertainty and lead to the right decision regarding the future of the extraction. This sensitivity analysis is a valuable tool to see the effects of the different project variables on financial results. It helps in the identification of the most crucial factors that must be controlled to avoid affections in the overall project. It also helps to clarify the impact of specific changes on the variables and lead to fast decision making in the future. (Allen, 1986)

2.5.1. Cost estimations

A mining extraction project entails different expenditures that need to be included in the cash flow forecast. These costs can be classified into two categories, capital costs and operational costs. The capital costs (CAPEX) include the investments made both in the beginning and during the project life for the equipment and infrastructure. These costs are usually based on detailed engineering studies and need to be generally approved by the company’s financial directors or board managers. On the other hand, the operational costs (OPEX) are expenditures that include the resources to keep the production going each year. The OPEX is also called working costs and is controlled by the operation managers. (Allen, 1986)

For the cost estimation, several tools and methods have been used. Even though the methods are different, they all seek an accurate estimation of the mining operation's real costs. In many cases, the cost estimation includes an additional factor or a contingency established to cover the uncertainty and reduce the risk in the evaluation.

Some of the strategies used in the cost estimation are:

- Use of historical information or actual costs of a mine operation. The cost estimation can be made based on data obtained from the operation of other mine areas in the same deposit or with information obtained about mines of the same commodity. Special care needs to be taken to adapt the cost to a particular project since the conditions may vary significatively. (Stebbins & Leinart, 2011)

- Cost estimation guides and cots references. Estimated operational and capital costs are provided in many guides, reference books, and quotations of the equipment providers. These tables are obtained by an analysis of the industry based on information obtained from current operations. Some examples of this are the 'Mine costs guides' published by InfoMine and the 'Reference manuals and buyers guide' published by the Canadian Mining Journal.

- Cost estimation is based on a scaling rule depending on the production rate and the influence of the type of facilities and equipment used. O'Hara and Suboleski proposed this method; they had expressed the cost as a constant multiplied to factors powered to values between zero and one; they represent the most important variables affecting the costs and the rate at which they change. (O'Hara & Subiloeski, 1992).

- Cost estimation based on detailed engineering in which the equipment's size, the rate of use of consumables and wear parts, the labor, and other associated costs are known in advance. In this case, unitary costs can be estimated in detail. This type of estimation is usually one of the most accurate methods, but it requires a high amount of information and many individual studies to reduce uncertainty. It can only be made after all the decisions regarding the project development are made. Even with all the information gathered, some deviations can be expected, and contingencies must be considered. (Allen, 1986).

An accurate estimation of costs is highly important in the choice of continuing or not mine development. It provides the basis for the financial expenditure, which is usually high at the beginning of a mining project. It is determinant in evaluating the profitability of the project and the decision of acceptance or rejection.

2.5.2. Estimation of revenues

The revenue estimation is made based on the expected production and mineral selling prices. It needs to be high enough to cover the estimated costs and provide profits to the investors under the current and future market conditions. It is calculated, including all the possible earnings obtained by the sales of both main mineral products and by-products obtained in the processing stage.

The estimation of the possible revenues obtained by a mining project is a process that involves a study of the market conditions for the commodity not only in the present but also through the history to establish tendencies and make a possible forecast for the upcoming years. It is a risky process since the price can be volatile and may change several times through the project's life. Furthermore, the commodity price is determined by the supply and demand variation, variables that are not within the control of the company. (Allen, 1986)

Part of the risk may be mitigated by the signature of initial selling agreements in which a price is fixed for a certain amount of time. It may be a good move if the selling price drops, avoiding unexpected losses; on the other hand, it may also be prejudicial in the cases where the price increase causes the opportunity of getting higher revenues may be gone. In any case, the mining project needs to deal eventually with the commodity market variables to succeed.

The decisions regarding the evolution of the project must be based on an analysis of the price fluctuation, and this means the evaluation of different revenue scenarios to establish the breakeven point and see if the project can deal with the lowest prices or if changes need to be done to it to make it profitable. The revenues estimation needs to be continuously updated and reviewed following the change in the conditions; adjustments made at the right time to the process may help the project to succeed when the prices drop. (Runge, 2011)

3. Case of Study: Southern area Santiago deposit. Long term planning

3.1. Regional geology

The mine is in the town of Estepa in the southern part of Spain. The reserves occur in a part of the "Sierra de Estepa" (Estepa Mountains). it forms part of the external sub-betic system that corresponds to the southern area of the Betic mountain range. (Ortiz, et al., 1995)

The external sub betic system comprises a curved fold and thrust mountain range formed by Jurassic marine carbonates and cretaceous marls and marly limestones. The deformation of the external betic zone is described as a sequence of sedimentary beds and faults that strike NE-SW. (Pedrera, et al., 2012)

The main geological units found in the Estepa formation are part of a large antiform body trending N-S. An arrangement of 400 m to 700m of micritic, oncolytic, and oolite limestone forms its center. It was formed during the early and mid-Jurassic. On top of it, a thin layer of about 25 m of marly limestone was deposited during the late Jurassic. In addition to a significant layer of 650 m of marl, marly limestone, and limestone with high silex content from the upper Cretaceous to Paleocene. (Ortiz, et al., 1995)

In figure 7, the stratigraphic column of the external sub etic zone shows the Jurassic-Cenozoic sequence of limestone found and characteristic of the Estepa Region. It shows the stratigraphic sequence of the calcareous materials in the "sierra de Estepa" (Estepa mountain range). (Pedrera, et al., 2012)

The thickness of the beds can vary depending on the location. The lower part is composed of 20 to 20 m thick dolomitic breccias. An oolitic platform of limestone is defined as Camarena formation with a thickness that varies between 200 – 700 m formed in the Jurassic. Above this platform, a sequence of marl and marly limestones were deposited between the Cretaceous and Paleocene in the deep part of it; it has a low content of silex and is found with a thickness of about 0 to 400 m (Ammonitico Rosso and Caretero formations) (Castro, et al., 1990). The deposited material transitions towards an increase of the silex content in the upper part of the unit; even some silica nodules can be found in the rock. The transition has an approximate thickness of 200- 300 m. Finally, a unit composed of calcareous sands, limestone, marls, and calcarenites is placed above the previous units. (Pedrera, et al., 2012)

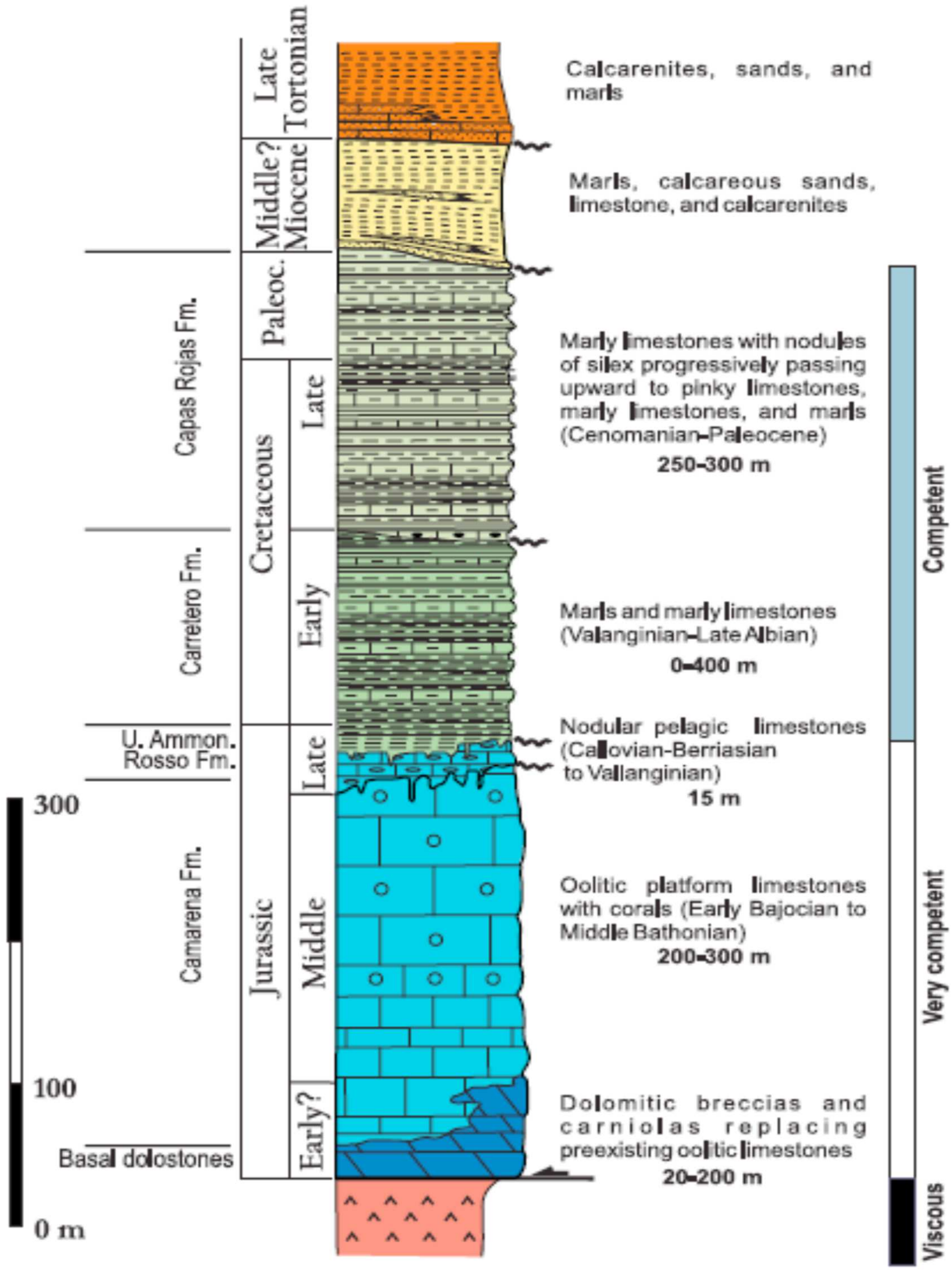


Figure 7:stratigraphic units from the external sub betic in the "Sierra de Estepa." (Pedrera, et al., 2012)

Geochemically, the limestone found in the Estepa region is very homogeneous with a low content of silica below 1% and with the presence in different low proportions of trace elements such as Magnesium, potassium, phosphorus, Aluminum, iron, and Sulphur. Some trace elements are attributed to the marine origin of the formation and the presence of bioclasts in it. Despite the low geochemical variance, textural variation allows the definition of different petrographic groups in the area. Micritic limestone, fine-grained oncolytic limestone, coarse-grained oncolytic limestone, oolitic limestone, and detritic limestone are found in the area.

The physical properties of the limestone beds in the “Sierra de Estepa” are in general favorable for its use in the construction industry. The stone extracted has low porosity, a density that ranges from 2.5 to 2.7 $t/m^{3,}$ and a medium to high mechanical resistance. (Ortiz, et al., 1995)

Despite the presence of trace elements that can be detrimental for the industrial use of the stone for some applications, in the area surrounding the town of Estepa, several quarries carry on the extraction of the limestone for lime, cement, and ornamental stone production.

3.2. Geology of the current quarry

The reserves under current extraction are part of the mining concession number 7146 called “Santiago mining concession.” It is located about 1.5 km from the town of Estepa in the Becerro mountain. According to studies made by geologist Fernando Menendez, the sedimentary sequence in the mine is folded and faulted, indicating the presence of a thrust system in the area. This system is also described in the work performed by Pedrera et al. (2012), in which a description of the main structures in the area is presented.

Currently, the deposit is divided into two main parts with different characteristics and settings. The north part of the deposit, which is currently mined, has been in operation for more than 30 years; the south part of it is intact, and its reserves are the objective of the mine planning for the future years. The 2 zones are shown in figure 8, including the current pit, the south extension area, and the main fault.

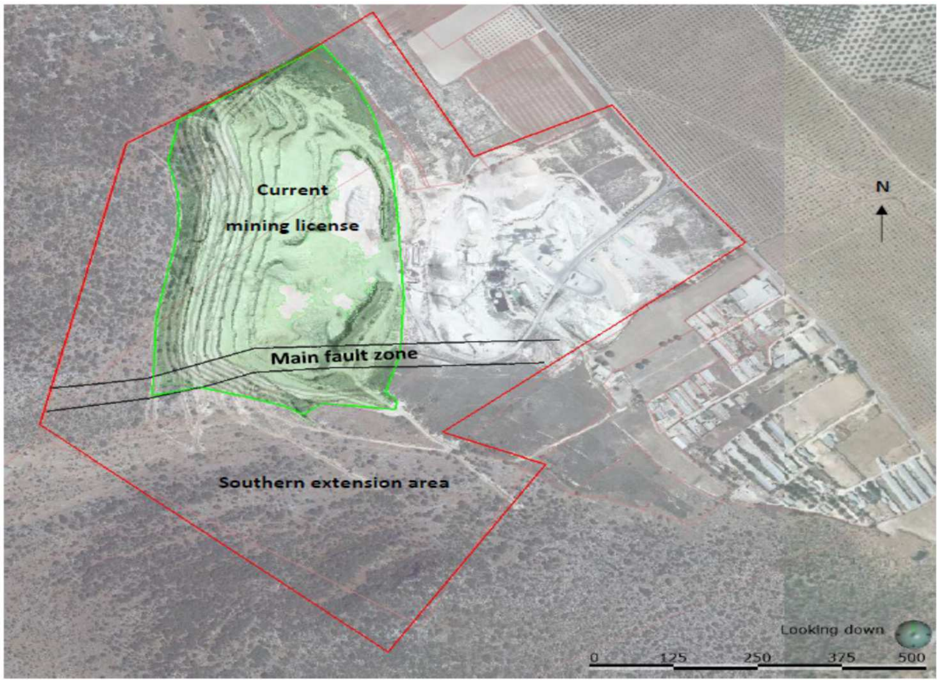


Figure 8: Aerial view of the mining concession. The current pit showed in green. The extension area and fault are also delimited. (Pirotte, 2019)

The northern area of the deposit, currently being mined, is described as an open asymmetric syncline with the southern area overturned. Although any deep exploration has been previously done, a surface sampling in quarry faces, as well as structural information collection, was made to complement the information on the chemical composition of the rock collected as part of the extraction and production controls in past years. The information shows the limestone in this area following the synclinal structural sequence. In addition, the presence of trace elements such as P2O5 is assumed as stratigraphically controlled.

In the quarry area, several samples were taken from the extraction faces. Some destructive drillings were done, and dust samples collected, allowing to have superficial knowledge regarding the rock's chemical composition. The layers in this area are massive and homogeneous. Furthermore, the structural measurements indicate that the limestone layers are nearly sub-vertical with a general orientation of dip of 80 to N80E (Menendez , 2007)

The quality of limestone extracted in the current open faces is suitable for the production of lime as it has a high calcium carbonate ($CaCO_3$) content of over 55% and low content of most of the undesirable trace elements. The rock composition in this area is very constant with a low silica content (below 0.5%), low Aluminum and Iron content (both below 0.1%). The only trace element that places a problem to produce lime and only for specific applications is the P2O5 content that is very variable in the area and can be higher than 0.2 %. The P2O5 content limits the possibility of using the limestone from the actual active quarry to produce lime to be used in the steel industry.

The southern part of the deposit consists of a sedimentary bedding sequence in which at least five geological units (5 different beds) outcrop and can be visually identified on the surface. The disposition of the sequence in this area is congruent with the open synclinal structure described previously. The units found in this area differ by their chemical content of trace elements and type of rock. Besides, in some beddings, the presence of clay and other contaminants is visible. (Menendez , 2007)

The area has been explored both on the surface and with the use of core drilling to obtain detailed information about bedding's distribution and geochemical content of the different units. The interpretation of the information obtained in the field observation and the exploration campaigns led to the development of a geological model of the southern area. In addition, a study of the distribution of the conflict element P2O5 has been made to evaluate the possibility of the extraction of Low P2O5 limestone in this area.

Between the northern zone and the southern extension zone, a fault is found. It comprehends a wide area of about 20 m with sharp edges and a mixture of clays and limestones with different textures. The fault is visible from the open quarry area. It is oriented dip 55º to N20E. A more detailed study of the fault is required to know its displacement and the influence over the stratigraphic unit and disposition of the limestone layers.

3.3. Mining method

Calgov carries out the extraction of limestone from the locally denominated Santiago deposit. The plant has been operating for 35 years, using material extracted from the limestone in the northern area of the deposit. The northern pit has been extracted based on geological information of the deposit obtained from the multiple outcrops in the area and the results obtained in the previous extraction. There is not a detailed geologic model or block model of this area.

The extraction is carried out using the quarrying method in open cut mine. The mine has a bench configuration of 20 to 30 m high and 5 m berms. The roads have a maximum slope of 15º; they are built in the pit's perimeter and made with conditions only to transit track chained equipment. The bench angle is approximately 75º, and the pit has nine benches, having the bottom pit at level 590 and the last bench at level 730.

The mining is undertaken using drilling and blasting for rock fragmentation. The blasted material has intermediate handling in which it is thrown from the upper benches to the bottom pit using a backhoe for this process. The mineral is loaded and transported from the bottom pit to the crushing facilities. The operation is planned based on the material available in the bottom pit and the throwing activity. In every moment of the mining activity, the backhoe is situated at the opposite end of the area where the loading is taking place to avoid possible accidents.

The mine operates only during the day in one shift of 8 hours, 5 days of the week. During the weekend, preventive maintenance is carried out in the equipment and the crushing facilities. Depending on the plant's lime production, an extra shift is set on the weekend for the backhoe operation to ensure the constant availability of limestone at the bottom of the pit. Figure 9 shows a view of the current pit under extraction.

Figure 9: Calgov S.A. Northern Pit. Taken from Google satellite images.

3.4. Processing

The limestone extracted in the mine is processed to have the desired size by several crushing and screening stages. All the fine-grained material, between 0 to 35 mm, is separated and sent to the fines waste disposal in this process. Only material with granulometry between 35 to 120 mm is used to feed the lime kilns.

The limestone is fed into the crusher with an approximate size of 900 mm or smaller. Once in the plant, a primary screening stage takes place; the material is classified before entering the crusher, taking the fines out of the process. The material is fed into the primary crushing stage; after this first size reduction stage, it passes through the second screening stage and is classified into 3 flows; the coarse-grained material (bigger than 120 mm) goes to a secondary crushing stage, the material between 0 – 35 mm leaves the process, and the other continues to the kiln feed stockpile. The secondary crushing is in a closed circuit with the second screening.

A more detailed description of the processing is given in chapter 5.8, in which the current system conditions are explained.

3.5. Southern area geology

The geologic model for the south extension area is based on field observations, structural measurement of the limestone layers, and the exploration results from two campaigns carried in 2010 and 2016. A total of 830 meters were drilled, and the cores were analyzed with samples collected every 1 to 2 m. Several field measurements of structures found in the area were made, also a description of the type of rock found in each area.

In the southern area, the geological model comprises a total of 8 geological units or domains identified in the exploration campaign. Each unit has homogeneous Chemical and physical properties. The units are named Q1, Q2, Q3, Q4, Q5, Q6, Q7, and Q8; Q1 was deposited at the top of the sequence, and Q8 is the deepest layer. The geologic unit differentiation was made based on the exploration results, the rock types found, and the deposit's structural setting. (Menendez, 2016)

Figure 10 shows the location of the exploratory drilling in the area. In total, 10 core drillings were made. The geologic units are also shown. Some units as Q1 are intercepted only by one exploration drilling.

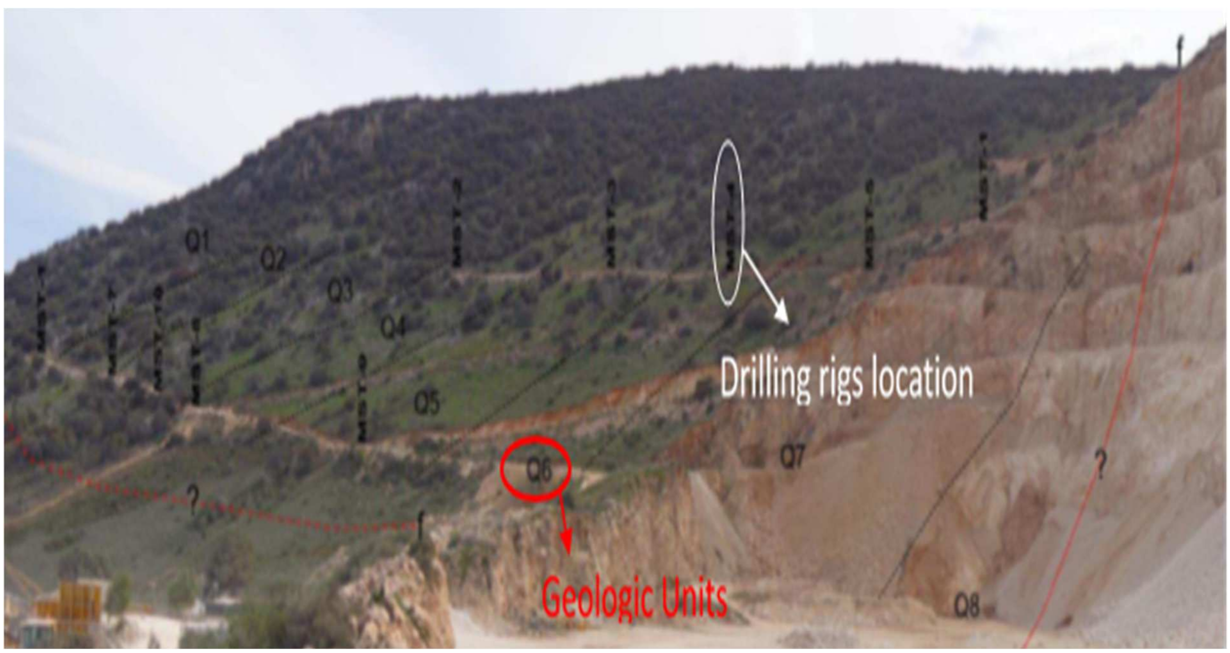


Figure 10: Location of the exploratory drilling.

Figure 11 shows how the geological units described in the geological model are distributed in the surface area. Unit Q1 is located at the top of the sequence and Q8 at the bottom of it. The domains follow a constant monoclinal structure dipping 30º to 40º to N100E. Almost all the layers outcrops in the area except for units Q7 and Q8 that outcrops in the bottom of the current open pit near the main fault. The geologic domain subdivision is only valid for the southern area of the deposit. Further studies need to be done in the fault zone and northern zone to identify how the disposition of the limestone beds change. (Pirotte, 2019)

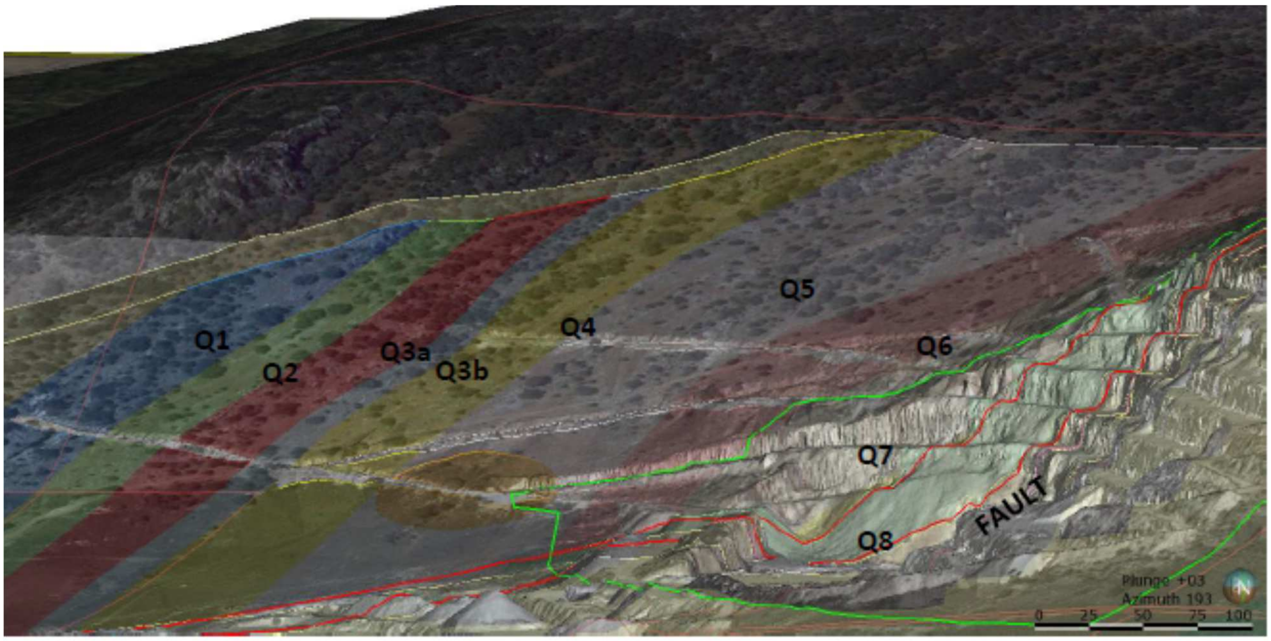


Figure 11: Geological units in the southern area

Table 1 shows the mean composition of each one of the geologic units. It also specifies the lithology, type of rock and texture found, and the possible products under the plant processing options.

Table 2 shows the chemical composition of the limestone for each product processed in the plant. It is essential to have reserves of limestone to meet the quality required by the current clients.

Table 1: Average composition of the geological units. modified from Menendez,2016

Unit	Average content								Description	Posible products
	CAO	MgO	Fe2O3	S	SiO2	Al2O3	MnO	P2O5		
Q1	55.43	0.24	0.03	0.01	0.10	0.04	0.00	0.07	High: P2O5. Low: Al2O3,Sio2,Fe2O3, Mn,Mg	Precipitated calcium carbonate
Q2	55.35	0.27	0.05	0.01	0.21	0.10	0.00	0.02	High spots of SiO2,Al2O3,Fe2O3. Low P2O5.	Precipitated calcium carbonate. Lime for steel industry.
Q3	55.34	0.23	0.04	0.01	0.18	0.09	0.00	0.07	Variable contetnt of MgO and Fe2O3. Medium to high P2O5.	Precipitated calcium carbonate
Q4	55.11	0.33	0.08	0.01	0.39	0.18	0.00	0.02	Medium content of MgO. High spots of SiO2,Al2O3,Fe2O3. Low P2O5.	Lime for steel industry
Q5	55.44	0.21	0.04	0.01	0.13	0.07	0.00	0.05	Variable contetnt of MgO and Fe2O3. Medium to high P2O5.	Precipitated calcium carbonate
Q6	53.64	0.53	0.32	0.03	1.87	0.67	0.01	0.01	High content of SiO2,Al2O3,Fe2O3, P2O5.	Waste
Q7	55.03	0.39	0.09	0.02	0.43	0.18	0.00	0.01	Midium contetn of Fe2O3 and MgO. Low Mn and P2O5	Lime for steel industry
Q8	55.61	0.18	0.02	0.01	0.03	0.02	0.00	0.01	Low conternt of SiO2,Al2O3,Fe2O3, P2O5.	Precipitated calcium carbonate. Lime for steel industry.

Table 2: Quality specifications for different products. Modified from Menendez, 2016

Limestone quality to produce		
Component	**PCC**	**Lime for the steel industry**
CaCO3	>50	> 55
Al2O3	< 0.5%	< 0.5%
Fe2O3	< 0.5%	< 0.5%
MgO	< 0.3%	< 1%
P2O5	< 0.3%	< 0.039%
SiO2	< 1%	< 0.5%
Mn	< 0.02%	

The geological model is modeled in 3D using Leapfrog Geo implicit modeling software. It was made using the drill hole database, the stratigraphic unit intervals already described, and the structural points measurements. In the 3D model, a division is made in unit Q3 displayed in two subunits Q3A and Q3B; this distinction is made due to the difference in rock type found in the top and bottom of this unit. Even though the chemical composition is homogeneous in the entire unit 3, a change in the rock texture is found. In the bottom sub-unit 3A, the rock texture is oolitic fine-grained; on the other hand, in the top, Sub-unit 3B, the rock texture change to be micritic and marly.

An area with a soft clay filling was identified in the exploration results in the Q4, this area is a karstic cavity, and it is modeled as a separate circular unit. Figure 12 shows the 3D geological model for the south extension area. it shows first the overall model in an orthographic view and below a down view of it, including the exploration holes location. (Pirotte, 2019)

Figure 12 shows the geological model of the southern area of the deposit, the picture in the top of the interest units are displayed showing how they are distributed in the zone and above the elevation of the bottom level of the pit. The bottom pictures show a top view of the area in which the current mine limit and the new area to plan are shown. Besides, the drilling location is included to clarify the extension of the exploration work made.

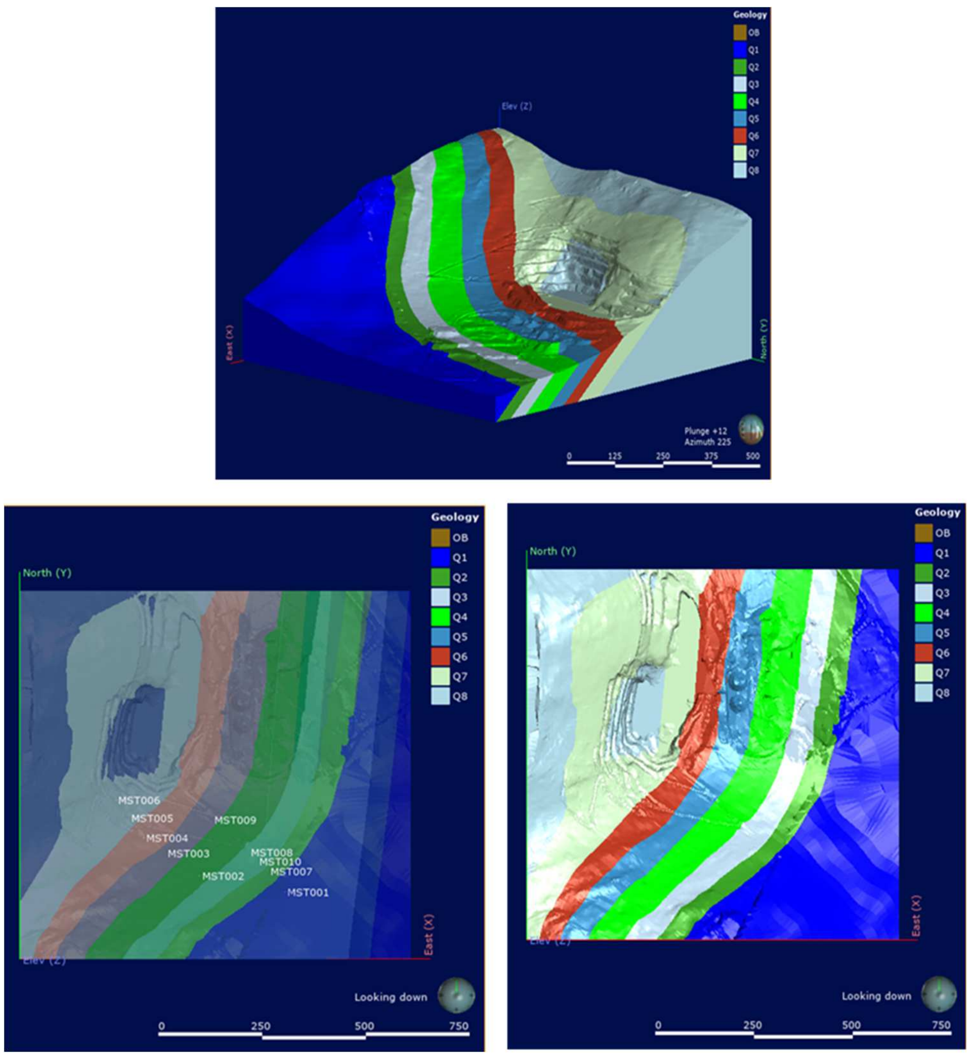


Figure 12: Geologic model of Calgov southern area. The model was developed by Nicolas Pirotte in 2019. Leapfrog modeling software

3.6. Geomechanics

A geomechanical study of Calgov southern area was made in 2010. It focused on knowing the rock mass geomechanics and the main discontinuities in the area. The rock mass was classified using the rock mass rating system (RMR) proposed by Bieniawski in 1974 and the modification of it published in 1989.

The classification is made by dividing the total area into small regions defended by a main structural feature as a fault, discontinuity family changes, or rock type changes. The RMR system uses six main parameters for the rock mass classification (Hoek, 2006)

- Uniaxial compressive strength
- Rock quality designation RQD
- Spacing of discontinuities
- Condition of the discontinuities
- Groundwater conditions
- Orientation of discontinuities

The RMR is calculated by the addition of the values obtained for each one of the parameters mentioned. It rates the rock mass quality assigning a value between 0 and 100. If the obtained value is below 20, it indicates very poor rock with a need for constant support. On the other hand, values above 81 indicate good quality rock with any or very few support required. The tables used as a guide for the RMR calculations is showed in Annex 1. (Hoek, 2006)

In the fieldwork, a total of 3 areas were studied, 5 structures were identified, one stratification plane named S0 (the stratification of the layers is parallel in the area), and 4 joints families named J1, J2, J3, and J4. The RMR is calculated based on the observations made; the results are shown in Table 3. The lower values of cohesion and friction angle for the rock type ranges were taken from the Bieniawski classification guide to keep a conservative approach.

Table 3: Rock mass rating Calgov south area. Bieniawki system

Structure	Dip	Dip direction	RMR	Cohesion (MPa)	Angle of friction	Rock classification
S0 (Bedding plane)	40	N100	74	0.3	35º	Good rock
J1	70	N255	71	0.3	35º	Good rock
J2	72	N51	69	0.2	25º	Good rock
J3	43	N77	59	0.3	35º	Fair rock
J4	80	N160	74	0.3	35º	Good rock

Some failure mechanisms were observed during the fieldwork in the outcrops and the north pit area. The main ones are small wedge formation between the joints J2, J3, and J4. The wedges observed can be controlled by slope control and monitoring. An analysis of stability is needed when the final pit configuration and extraction directions are determined to prevent the possible development of other failure mechanisms with the extraction, in addition, to determine the safety factor for the design of the slopes and the general expected stability.

A more detailed stability analysis is also needed to determine the interaction between the different joint families and the bedding plains. The main objective of it must be to prevent the possible occurrence of significant failures and take stabilization actions where needed.

4. Data analysis and block model

4.1. Exploratory Data Analysis

The geochemical information from the sampling of the two exploration campaigns is joined in a drill hole database consisting of 530 samples assayed for the chemical composition analysis of elements as $CaCO_3$, AL_2O_3, Fe_2O_3 SiO_2, MgO, P2O5. The samples were regularized to obtain a homogeneous sample distribution with information of composition data every 2 m obtaining 411 composites. In table 4, the summary of the composites per domain is showed, including the total length sampled and the mean composition per domain. The domains are kept as the same geologic units defined in the geological modeling (Q1 to Q8). Leapfrog Geo software was used for the statistical analysis, the videography, and the other estimation process.

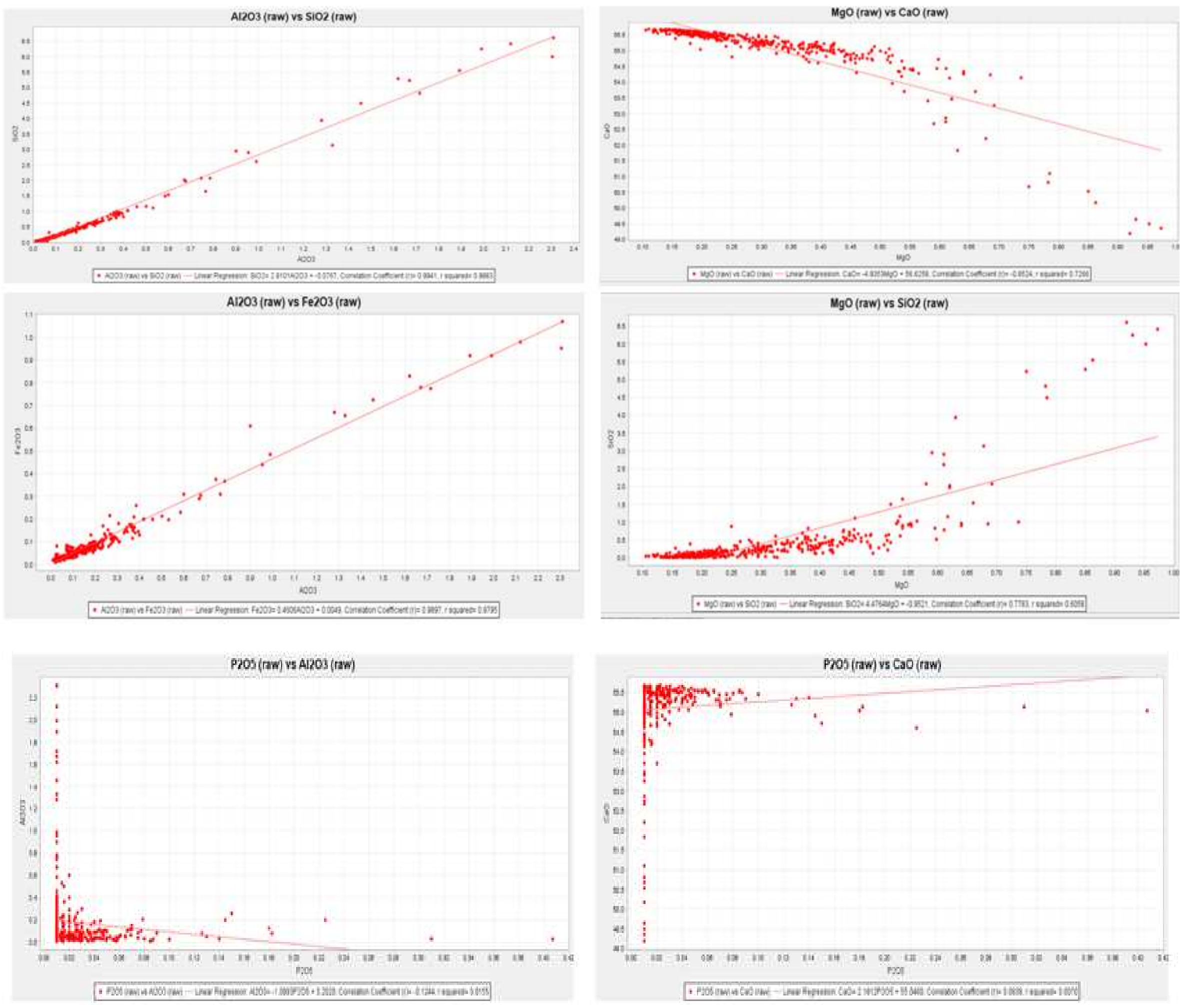
Table 4: Composites per geological unit

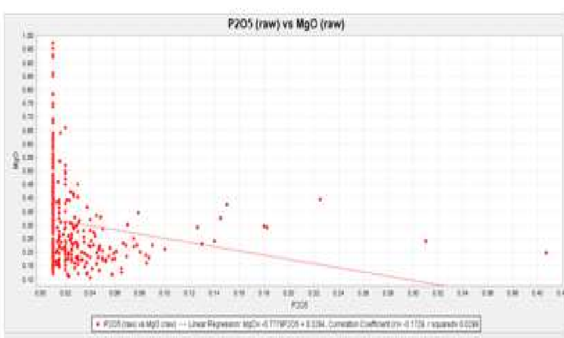
Name	Count	Length	AL2O3	CAO	FE2O3	MGO	P2O5	SIO2
Q1	14	28	0.026	55.481	0.023	0.211	0.074	0.071
Q2	16	29.6	0.137	55.213	0.068	0.306	0.024	0.295
Q3a	35	62.2	0.103	55.332	0.043	0.250	0.043	0.214
Q3b	29	49.69	0.047	55.480	0.032	0.207	0.056	0.079
Q4	62	116.91	0.193	55.065	0.080	0.349	0.015	0.418
Q5	44	80.9	0.066	55.436	0.038	0.218	0.045	0.132
Q6	47	91	0.672	53.624	0.315	0.535	0.012	1.886
Q7	90	175.6	0.186	55.002	0.096	0.385	0.011	0.461
Q8	74	145.5	0.025	55.601	0.021	0.188	0.013	0.032

Statistical analysis of the composites is made to understand the distribution and possible correlation between the deposit elements. Each element was studied separately; histograms, box, and whisker graphics, correlograms, and main statistics calculations were made and are shown to illustrate the distribution of the components in the deposit.

Figure 13 shows the correlograms and the correlation coefficient between the main elements in the deposit. It shows a high direct correlation between Al2O3, FE2O3, and SiO2 with a correlation coefficient of 1. A good correlation of these elements with the MgO content is also present in the distribution. This correlation behavior between the previously mentioned 4 components can be interpreted as the presence of clay mineral particles intimately blended in the carbonate matrix of the rock; according to the company geologist, the clay particles were originated in the early sedimentation process of the rock. (Pirotte, 2019)

On the other hand, an inverse correlation between the previous components and the CAO is present with a correlation coefficient near to -1, showing an increase of the carbonate content as the others decrease. The P2O5 content does not show a correlation with any of the elements in the deposit. The distribution of the elements in the deposit is highly controlled by the bedding of the deposit.





Correlation Coefficient Table						
	Al2o3	Cao	Fe2o3	Mgo	P2o5	Sio2
Al2O3	1	-0.9912	0.9897	0.7979	-0.1244	0.9941
CaO	-0.9912	1	-0.9861	-0.8524	0.0838	-0.9898
Fe2O3	0.9897	-0.9861	1	0.7834	-0.1158	0.9919
MgO	0.7979	-0.8524	0.7834	1	-0.1729	0.7783
P2O5	-0.1244	0.0838	-0.1158	-0.1729	1	-0.1175
SiO2	0.9941	-0.9898	0.9919	0.7783	-0.1175	1

Figure 13: Correlation between the deposit components.

Figure 14a shows the histogram and the fundamental statistical indicators for the aluminum distribution in the entire deposit. Also, in the box plot in figure 14b, the distribution is showed per domain. As can be seen, the histogram is right skewed; the variance is low in the deposit and in most of the units. High values are only present in unit Q6, which is classified as waste due to its composition.

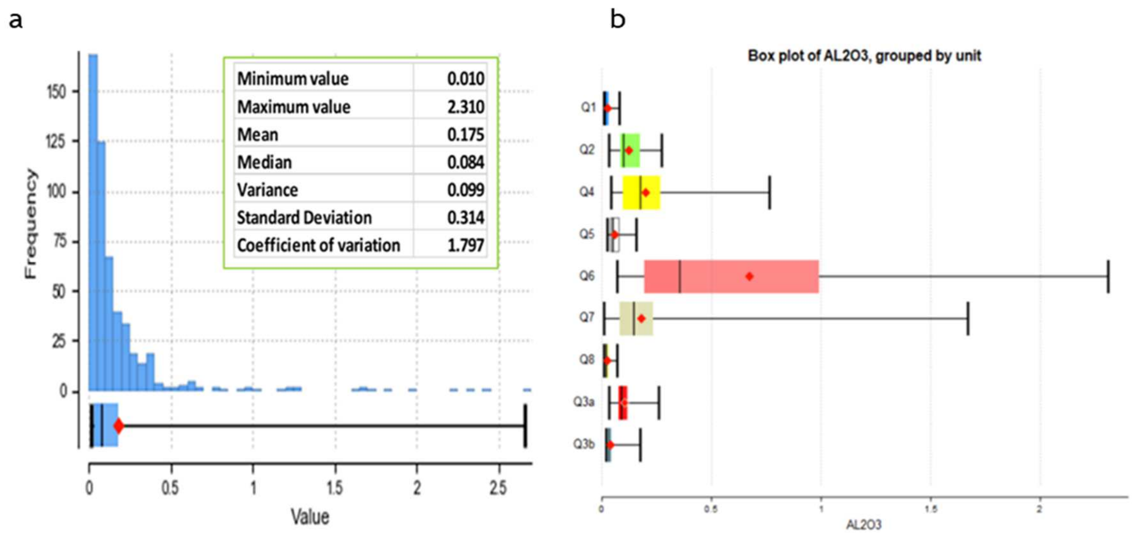


Figure 14: (a) Al2O3 histogram, (b) Al2O3 distribution per domain.

The distribution of FE2O3 is presented in figure 15 a and b. The distribution of this element is very similar to the distribution of AL2O3. The histogram has the same right skewness, and the variance is even lower than the one of the AL2O3. The content of Fe2O3 in the deposit is low, with high values only in unit Q6.

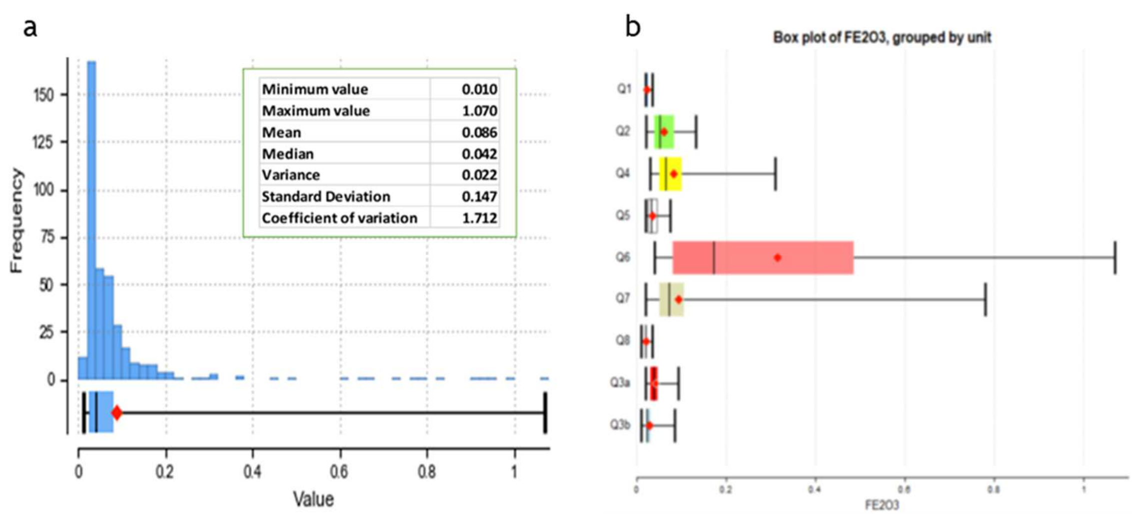


Figure 15: (a) Fe2O3 histogram. (b) Fe2O3 distribution per domain

Figure 16 a and b have the distribution of the SIO2 in the deposit and units. This element shows a higher variance than the AL2O3 and FE2O3, even though it has the same origin and is part of the clay founded in the rock matrix. Like the previous ones, unit 6 shows higher content of it. As can be seen, unit Q3b has a lower content of SiO2 than unit Q3a due to the presence of more clay in it.

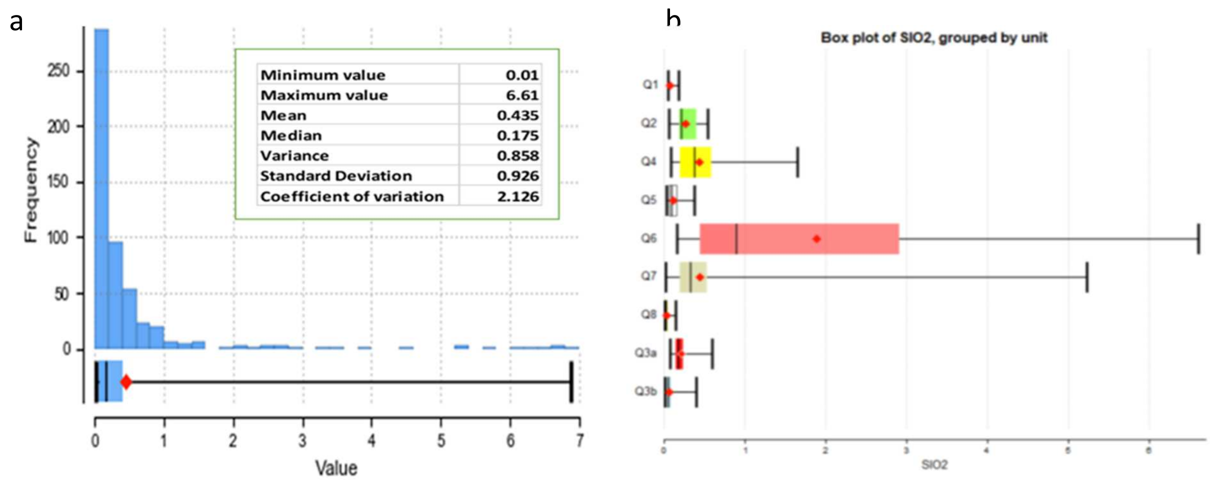


Figure 16:(a) SiO2 histogram. (b) SiO2 Distribution per domain.

CaO distribution is shown in figure 17; opposite to the other elements, the histogram is left-skewed, indicating a predominance of a high content of CaO, the core element for the lime production. The overall variance is higher than the other elements, but it is still low when analyzed per geological unit. The deposit is homogeneous in its composition of CAO, and the geochemical changes follow the beddings.

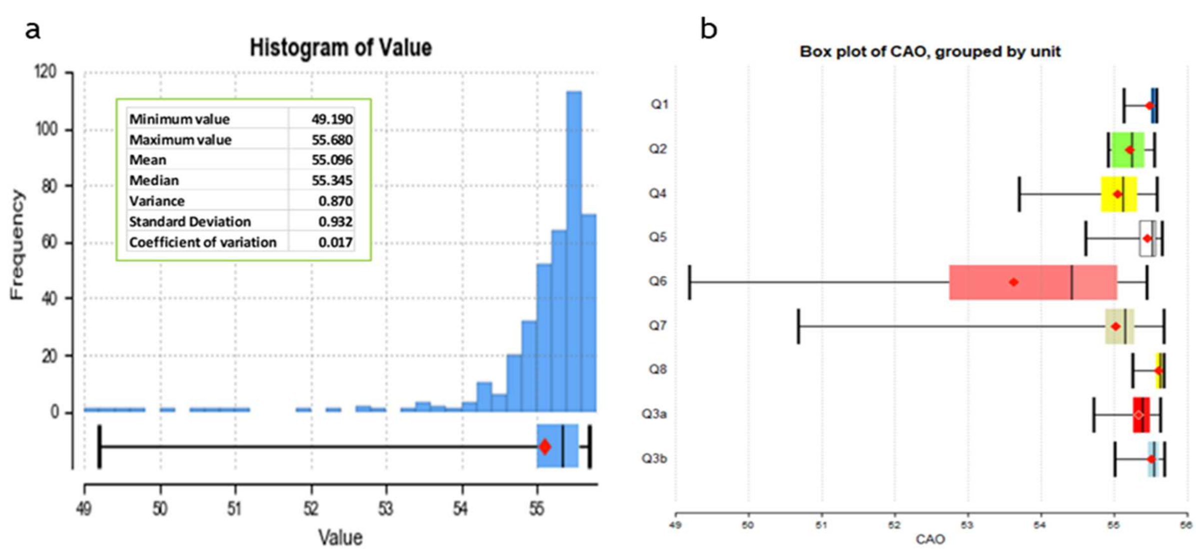


Figure 17: (a) CaO histogram. (b) CaO Distribution per domain.

In figure 18 a and b, the MgO distribution is presented; it has a different behavior than the previous elements showing a more spread distribution in the deposit. The overall variance is low; on the other hand, when looking at the box plots, a good contrast of the content of this element is found, implicating a stratigraphic controlled distribution of this element. The origin of MgO in the deposit is not clear; it is attributed to two origins. The first one is that the clay minerals contain some Mg; the second one may be from a low dolomitization level. (Menendez , 2007)

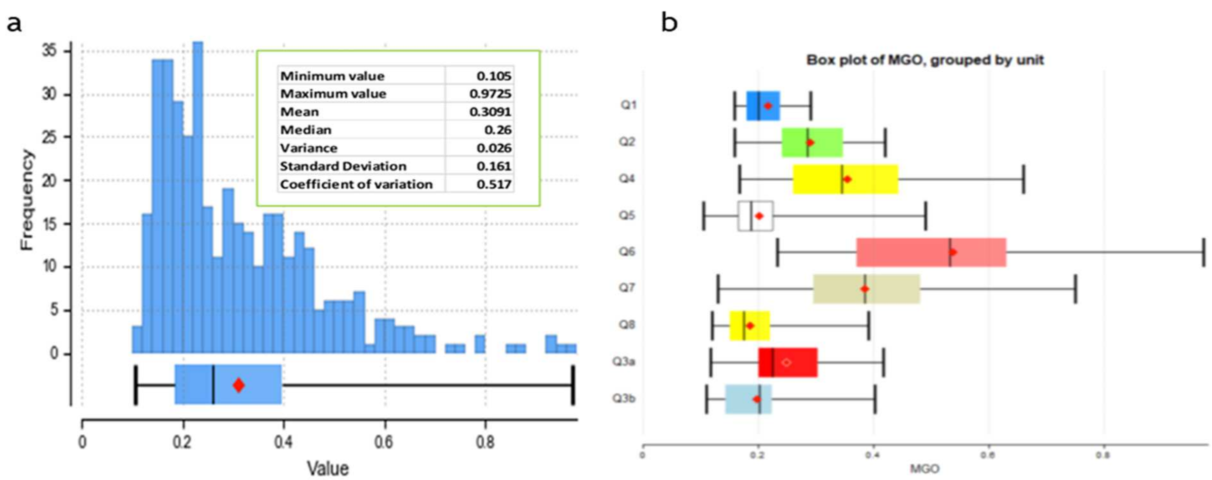


Figure 18: (a) MgO histogram. (b) MgO Distribution per domain

The P2O5 distribution is more complex to be interpreted due to the lack of correlation between it and the other components. Figure 19a shows the histogram of it. Even though it is right-skewed, it has no relation of been originated with the clay elements. It can be designated to have originated from sea level drops and aerial exposure of the sediments that lead to the creation of har grounds parallel to the bedding. It can also be formed in a strata bound deposition of volcanic ashes that bring contents of P to the deposit. (Menendez , 2007)

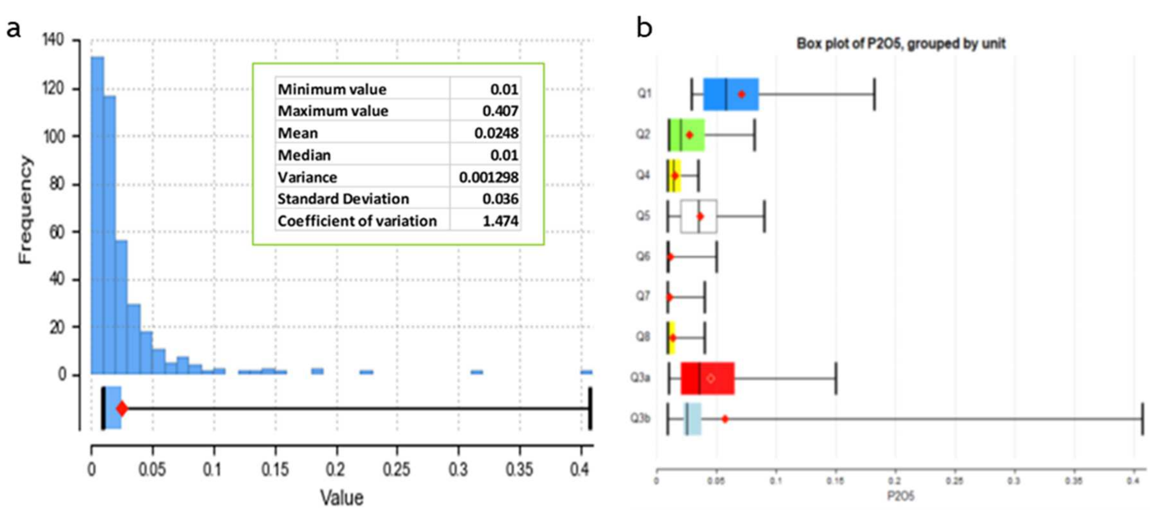


Figure 19:(a) P2O5 histogram. (b) P2O5 Distribution per domain

4.1.1. Variography

Getting knowledge of the deposit variability and possible anisotropy is the key to get an accurate estimation of the block's ore content and, therefore, for the construction of a successful mining plan. Getting variograms of the deposit components allows knowing the area in which the samples are related and the deposit spatial continuity. (Akeju & Afeni, 2015)

A variogram analysis is made to know the spatial variability of the limestone deposit in the southern extension area. Omnidirectional variogram, as well as variogram maps changing the direction of variography calculations, were constructed, and models were fitted for each one of the variables of the deposit. The Al2O3, Fe2O3, SiO2, and CaO have been fitted with a spherical model while the MgO and P2O5 with an exponential model.

Figure 20 shows the omnidirectional variograms for the limestone components. The data was normalized for the variogram construction. For most of them, the sill is reached near 1; it means that the sill is reached close to each element variance. The ranges are between 14 m and 40 m, and the only variable that has a nugget present is the P2O5 with a value of 0.2.

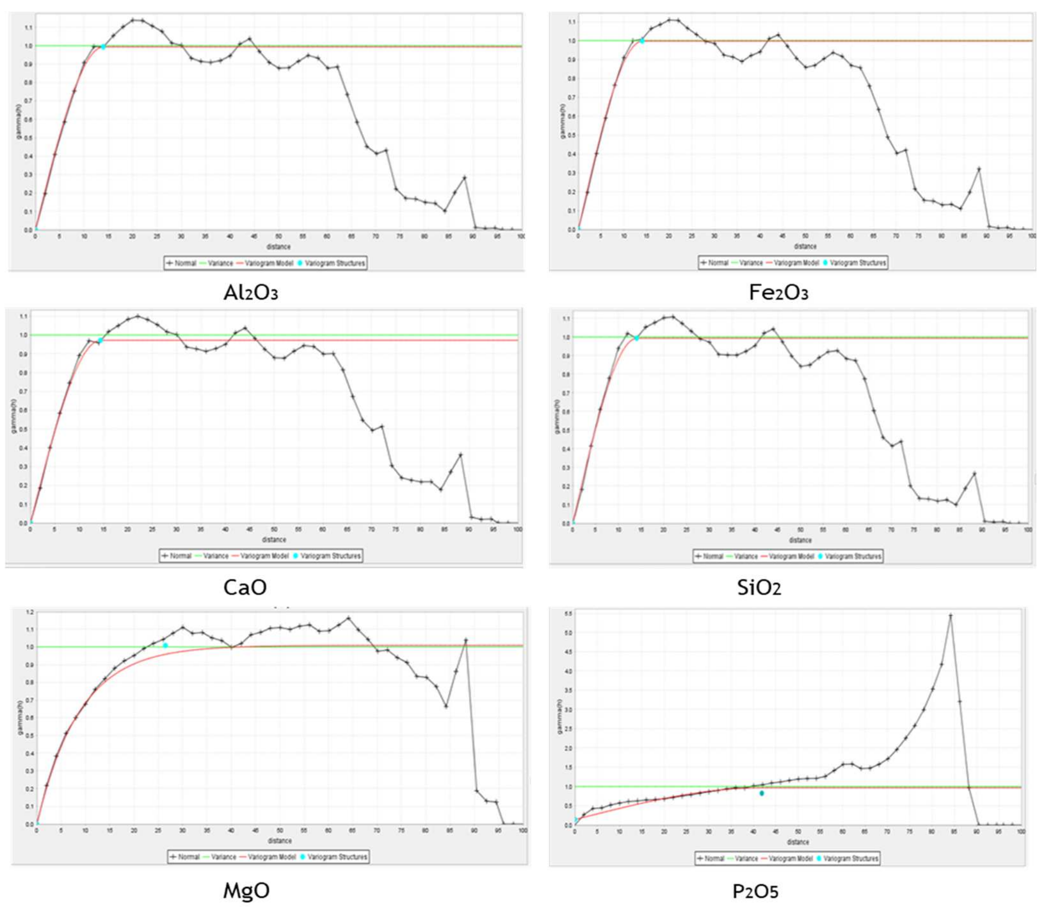


Figure 20: Omnidirectional variograms of the deposit main components

Table 5 shows the structures found in the theoretical model fitting process. In addition to the omnidirectional variograms, variogram maps with changes in the direction were studied to determine possible anisotropy in the deposit. The obtained maps were almost equal for all the different directions, and therefore it is concluded that the deposit does not show spatial or zonal anisotropy in the components for the southern area.

Table 5: Structures models found in the variogram fitting.

Element	Nugget	Sill	Range	Fitted Model
CaO	0	0.96	14.5	Spherical
Al_2O_3	0	1	14	Spherical
Fe_2O_3	0	0.9	14.3	Spherical
SiO_2	0	0.98	14	Spherical
MgO	0	1	26.4	Exponential
P_2O_5	0.2	0.8	41	Exponential

The deposit composition is highly homogeneous with low variance in the geochemistry. This variance is, in most cases, lower or near to one, especially when analyzing the composition per domain. Variography has not been made per domain due to the lack of data. The count of composites in some of the predefined geological and structural domains is too low. Besides, some of the domains are intersected only by 2 or 3 cores, and therefore doing an accurate geostatistical domain is not possible. An increase in the sampling is required to validate with higher certainty the spatial configuration on the beddings.

Despite the little information obtained for some of the domains, the low variances in the composition per domain show that the pre-established stratigraphic units efficiently separates the total samples into populations with specific chemical characteristics.

4.2. Block model

The block model is based on the geological model and the subdivision of domains established before in the geological modeling process. The deposit has been divided into blocks with a volume of 50 m^3 and measures of 5m x 5m x 2m in X, Y, Z axis, respectively. The density is set to 2.5 t/m^3 for all the blocks and lithology types.

The homogeneity of the deposit layers, the results of the variograms maps where no anisotropy is found, and the assumption that all the chemical components of the deposit were formed in the sedimentation process following a stratigraphic control without secondary contamination processes affecting the deposit makes it possible to do the estimation of the content of each block using the parameters found in the omnidirectional variograms. Furthermore, the estimation process was done following the beddings and giving high importance to the structural trend.

The estimation was made using Leapfrog Geo multi-domain interpolant following the structural trend input into the process. The units (Q1, Q2, Q3a, Q3b, Q4, Q5, Q6, Q7, Q8) do not dip homogeneously in the full extension, which means small variations in the dip are present; Consequently, slight changes and adjustments need to be made in the process. The Leapfrog Geo tool allows realizing estimations with variable orientations, allowing to consider the strike and dip changes in the estimation process for each domain. The interpolation process follows a series of sub interpolations for each domain in which the parameters can be changed. (Seequent limited, 2018).

In the estimation process, a strong influence is given to the structural trend; the strength assigned to it was 4. It means that the search ellipsoid used in the estimation process has 4 times more weight in the directions parallel to the beddings than in those perpendicular to them.

The estimation for all the elements is made using a sill of 1 and no nugget effect. The P_2O_5 nugget effect is set to 0 even though in the variograms it shows the presence of this structure. This decision is made to generate more sharp stratigraphic contrast in the estimation. The drift is set to be constant, forcing the estimation to remain close to the mean at significant distances from the composites and avoiding it to turn into 0, causing an overestimation of the quality.

Whit the estimation parameters set as previously described, the blocks near the composites receive a value very close to the nearest composite data and blocks far from composites receive a value close to the mean of the composite population. No estimation is made for the CaO component; it was calculated based on the results for the other elements.

The final block model of the southern extension area has a total of 126184 blocks. It includes all the material contained in the area owned by Calgovsa. This model is used as a base for the further calculation of resources and reserves for the mine planning. Figure 21 shows the block model colored by domains, and figure 22 shows the block model colored by the possible destination of the block. Both figures present a zoom of the blocks in the interest area on the right.

Figure 21 shows a top and isometric view of the block model built for the south area of the Santiago deposit. It is colored by the domain (geological unit) to which each block belongs. The block colored in white are of unknown quality and mainly located in the north area (current pit).

In figure 22, the top and isometric view of the block model is shown. It is colored by the P_2O_5 content indicating the possible destination of the block. The blocks colored in green have low P_2O_5 (below 0.039%) and can produce lime for the steel industry. The blocks in yellow have a medium content of P_2O_5 (between 0.04% and 0.05%), and the blocks in red are waste and have a P_2O_5 content higher than 0.05%. Finally, the blue blocks have unknown composition; they are located either in the north area or are part of the fault zone.

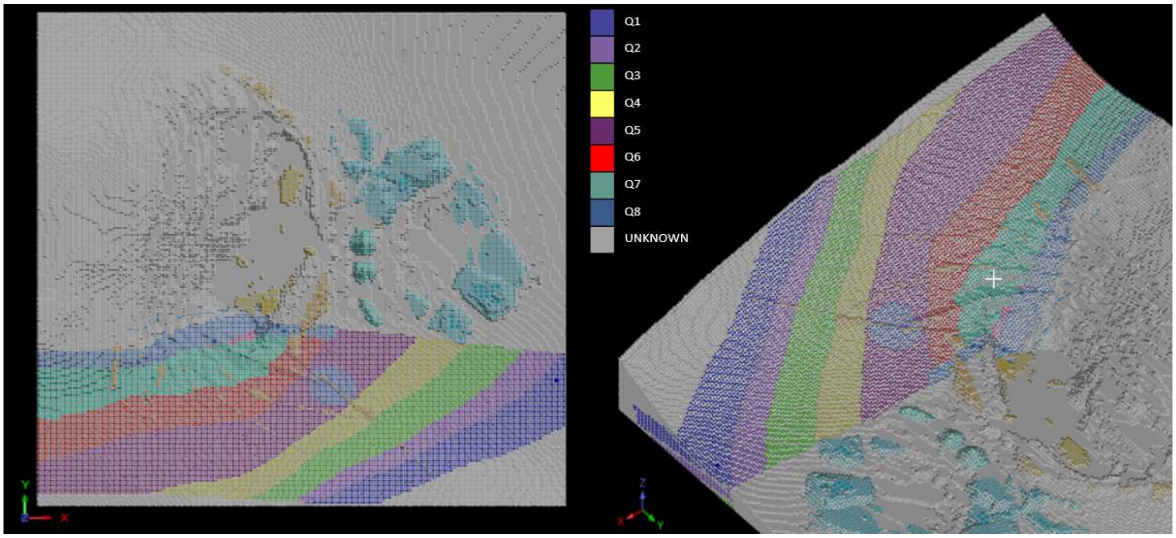


Figure 21: Top and isometric view of the block model colored by domain.

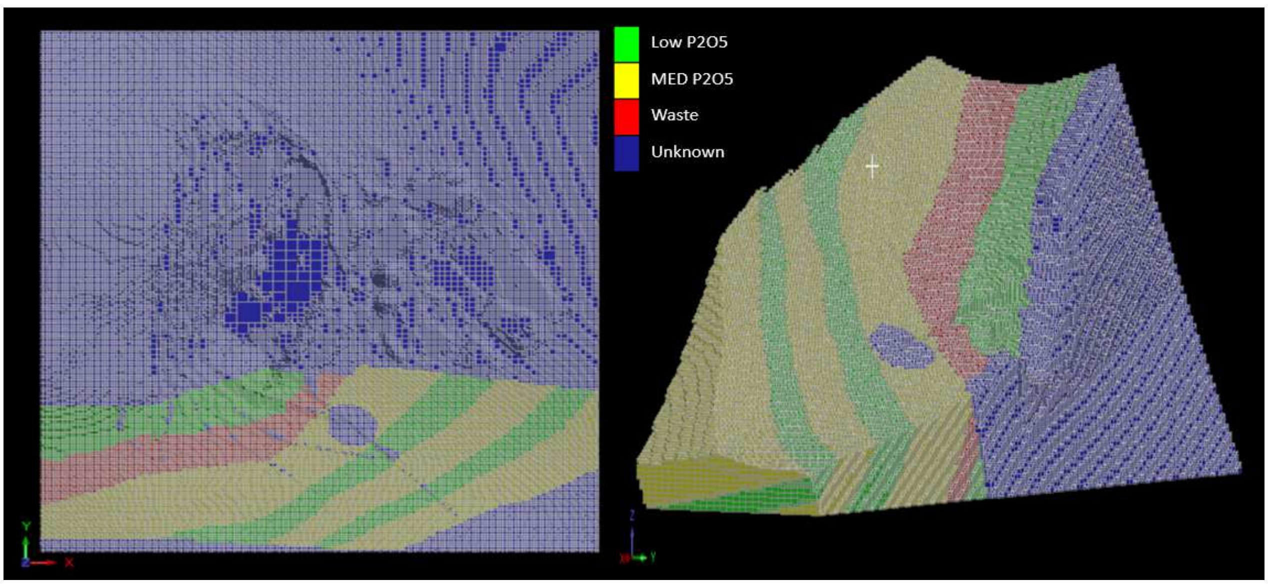


Figure 22: Top and isometric view of the block model colored by the destination of the material. Zoom of the southern area on the right.

5. Mine design and planning

The mine design process starts by studying the physical constraints present in the area that can affect the extraction, the geological structures present to determine their orientation, and possible response to the perturbations generated by the extraction and the collection of geotechnical data in the area.

Even though the deposit is extensive and the mining license in the south area covers 212 Ha, the mine design has been limited to cover only the area of the deposit that lies inside the current land owned by the company, an area of 66.11 Ha. Both the pit and the access roads are designed to lie inside this area. This decision has been taken because the negotiation of new land may take several years, and about 80% of the current measured resources are inside the owned land.

5.1. Resources and reserves

The most valuable asset for a company that uses mineral resources for its transformation is the reserves found in the deposit and their availability to be used for production. It is, therefore, of high importance to know its extent. Having an overestimation of the quantity of reserves can lead to the total failure and bankruptcy of the mining company. Contrary, underestimating them can lead to wrong plant and production sizing and the wrong decision about the investments and the expected results on the extraction. (Arteaga, 2014)

As mentioned before, the mining concession has a bigger extent, and therefore there may be more resources and reserves than the ones present in this report. To get a clear idea of the total amount of reserves, exploration needs to be done in all areas; a particular focus must be placed on areas outside the owned land and in the area under current extraction (North pit) in which any previous exploratory work has been done until now.

5.1.1. Resources

The resources classification at Santiago deposit is made based on the location and the existing geological studies. Only the southern area of the deposit is used for the resources estimation.

The material found in the south area of the deposit has been classified based on the P2O5 content, dividing the total resources into Low P2O5 limestone with P2O5 content below 0.039%, Medium P2O5 resources with content between 0.04 to 0.05, and high P2O5 for the resources with content higher than 0.05%.

In figure 23, the estimated resources for Calgov southern area are presented. They are sorted by the P2O5 content. There are 23 Mt of measured and indicated resources of low P2O5 content limestone and 8 Mt of Medium P2O5 Content in the southern extension area. Moreover, there are 3 Mt of limestone with unknown content of P2O5 in the area. This material requires further geochemical study to determine its possible destination.

The inferred resources are considered those resources inside the limits of the owned land but outside of the area considered for the exploration and extraction planning. It is assumed that the domains keep the same density and structural characteristics if the deepness is increased to overpass the level 590 m.a.s.l. The total inferred resources following this consideration are 40 Mt of Low P2O5 limestone and 19 Mt of mid P2O5 limestone.

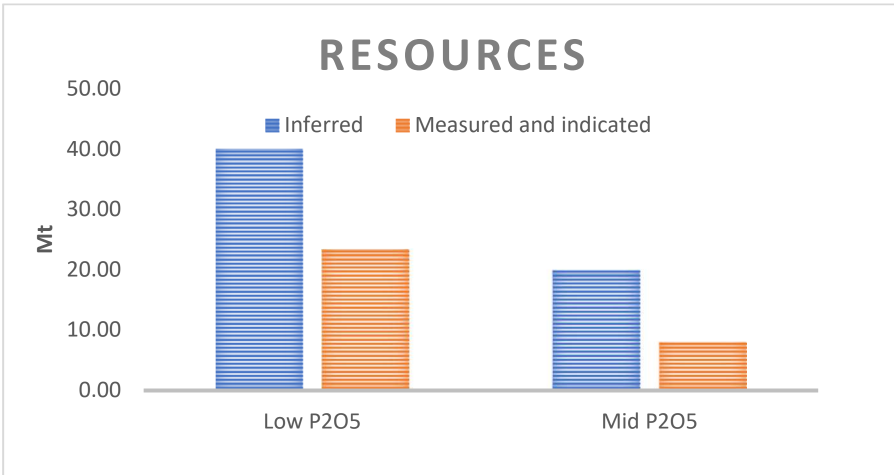


Figure 23: Resources for Calgov southern area

Further exploration is required to classify the resources found in the Calgov deposit with higher accuracy. More knowledge is required, especially in the areas that lie outside of the owned land and in the north area. A study of the influence of the fault located in the intersection of the north and south pit can determine the destination of the unknown material mentioned before. It is necessary to perform exploration works to know the behavior of the deposit below 590 m.a.s.l.

5.1.2. Reserves

The reserves considered for the extraction of Calgov southern area are calculated based only on space constraints established by the mining and environmental authorities, the production space restrictions for hauling roads, and the constraints regarding the extension of the owned land and the limits approved for the extraction in the environmental impact assessment.

The Local authorities had set a maximum extraction level for the mining activities in the “Sierra de Estepa” area of 745 m.a.s.l. This restriction forces every extraction to operate only below this level. Currently, the environmental permit allows Calgov to extract only up to the current level of the north bottom pit, so up to level 590 m.a.s.l. These two restrictions provide a vertical margin for the extraction of 155 m. Besides these restrictions, the development of the mining activity can be done only inside the owned land; this means that every activity, including the setting for auxiliary roads and hauling roads, must lie inside the owned area.

In figure 24, the current reserves considered for the southern extension area are showed. A total of 9.5 Mt of limestone with low P2O5 content and 3.5 Mt of limestone with mid P2O5 content are considered for the mine planning and extraction.

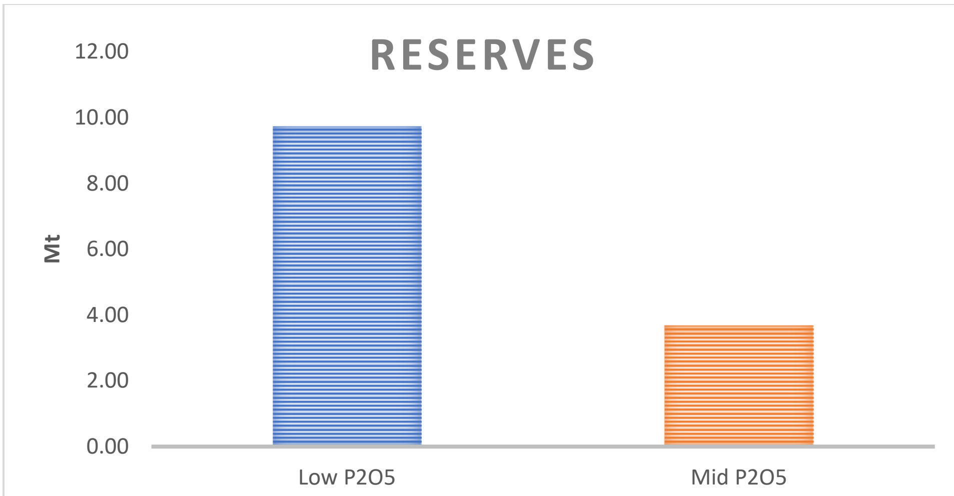


Figure 24: Reserves in the south area of Santiago deposit.

The requirement of using the area inside owned land for the construction of permanent roads causes a depletion of reserves since part of the valuable resources must be left in place for future transit and reclamation activities. It has been decided to design and plan the mine so that any land outside of the current owned area is used.

Even though in Spain the mining activity is of public interest and it is possible to use lands outside the owned area to build the mining roads, it is necessary to perform a new environmental impact assessment, in addition, to undergo a negotiation process with the neighbors of the mining activity to rent or buy the land and with the local authorities to change the current use of the land. This process of getting new land or to proceed with an adjacent land occupation may take several years and involve many formalities that may delay the overall project.

5.2. Pit Limit

An initial pit limit has been set to cover the reserves inside the owned land and using as limit the elevation and the area covered in the environmental permit. The maximum elevation the mining in the area can reach is up to 745 m.a.s.l. The pit is designed to reach its deepest point at an elevation at which the current extraction has its bottom pit; this means up to level 590 m.a.s.l.

The pit design has been made as an extension of the current north extraction and trying as much as possible to match the pit benches configuration and use common roads and services areas looking towards optimizing the space and the operation.

The field observations and the rock mass classification for the southern area indicate good rock with small wedge creation between the joint families (chapter 3.6). It has been decided to use benches with a maximum high of 20 m; when considering the areas in which higher instability is present due to the crossing of the joints families and the bedding structures, it has been decided to lower benches 2 and 6 to 10 m to increase the stability and to ensure a conservative design. However, it may be necessary to place special care in unstable areas to ensure the safety of the quarry.

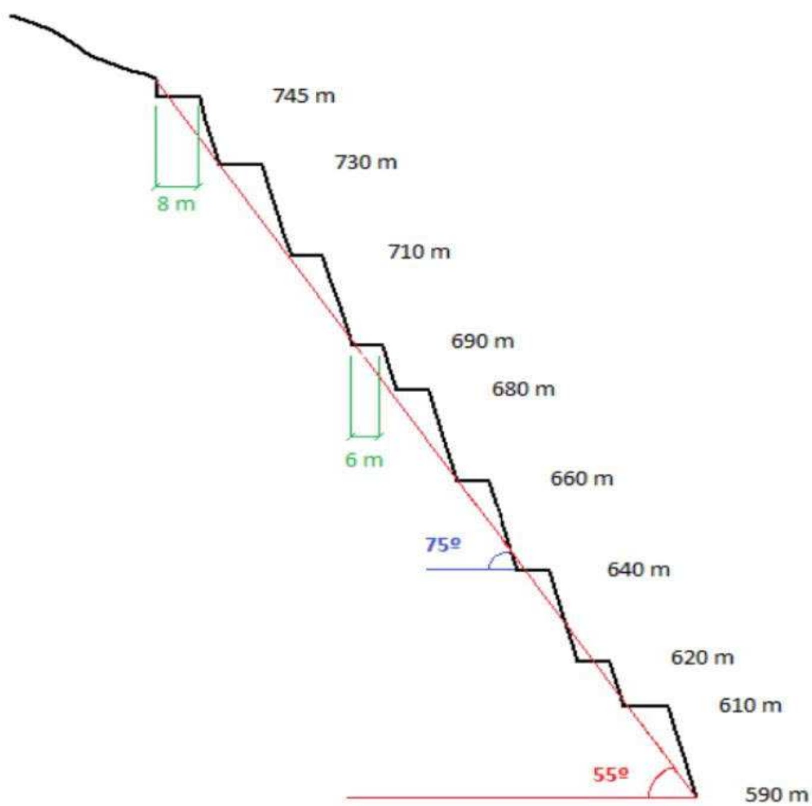


Figure 25: Final pit wall for the south extension.

Figure 25 shows the configuration of the final pit wall design. The maximum overall slope angle is 55° degrees, and the bench angle is 75. The bench height is 20 m for benches 1,3,4,5,7,8, 10 m for benches 2 and 6, and 15m for the last bench. The berms will be approximately 6m; this berm width is only for the configuration of the final wall. During operation, it is highly recommended to keep berms of 10 m or more to ensure safe operation.

In the future, it is possible to increase the deepness of the pit by extracting the stone located below the bottom pit since there is any environmental or operational restriction over the deepness the pit can reach. To do it is necessary to perform hydrogeology studies on the water level and geological exploration.

Furthermore, a new study on stability needs to be performed to establish the possible areas in which reinforcement is needed and define the slope monitoring and control strategies.

5.3. Mining method

The mining method for the south area extraction is kept the same as the one developed in the north pit (descendent benching) with drilling and blasting for the extraction and the combination of loader and trucks for loading and transport. The operation of throwing the material to several levels down is planned to have a modification; the material throwing operation is going to change, looking towards a reduction in the height and the number of times the material is handled, making the operation more efficient.

It is proposed for the southern extraction to have loading stations in bench 3 (640 m.a.s.l.) and the bottom pit. With this small change, it is expected to reduce the time it takes to have available material for the loading operation, reduce the fragmentation, and, therefore, the fines generation previous to the crushing, optimize the equipment use, and have a more flexible operation. It also can allow creating temporary space that can be used in a further operation. It is also proposed to keep a berm width during the operation of a minimum of 10 m to ensure the safe operation of the equipment.

Even though the mining method used is not optimal, it causes an increase of fine particle production in the process of throwing the material several levels down. It has been decided to be kept as the same currently used for the north area because the south area is considered to be an extension of the north pit. The current mining method used is already approved by the Spanish mining authorities and the environmental authorities. Any change of the current methods requires a change in the mining permits and falls into delays due to required documentation, studies, and approval times.

The fines generation can be partially compensated by developing a blasting design that minimizes the fines generation. Part of the fine material is produced during the blasting operation; the design variables such as burden, spacing, explosive change, and timing of the blasting can be changed to produce more efficient use of the blasting energy and reduce the amount of fines generated in the blasting.

5.4. Road Design

The road design has been made, considering the maximum allowed slope by the Spanish mining authority, which is 15% for any mining road. The design has been made for truck use up to the third bench and track chained equipment up from this level.

The road design is located in the border area of the pit but inside the owned land. Since the space restriction is involved and the land is steep, it has been necessary to design a road with multiple curves making it more complex and less efficient for circulation. The road has a width of 13 m approximately in the first part, from level 590 m.a.s.l. to level 640 m.a.s.l and 8 m from this level upwards. In table 6, the main characteristics of the road design are presented.

Table 6: Road characteristics

Bench	Level	road length (m)	slope (%)
1	590-610	213	8.4
2	610-620	80	12.7
3	620-640	139	14.5
4	640-660	139	14.3
5	660-680	240	10
6	680-690	100	11.1
7	690-710	160	13.6
8	710-730	200	14.4
9	730-745	112	14.2

In the future, if the possibility of performing load and transport of limestone from the upper benches is taken as an option, it is necessary to perform a redesign of the road to allow the circulation of trucks and loaders from level 3 upwards.

5.5. Production requirements

The information about the mine extraction and production is based on the historical record of the past three years of the crushed material, kiln feed material, fine-grained waste produced, and transformation ratio. All this information is currently followed and kept as part of the control strategy and KPI of the plant work.

The information about mine equipment work and efficiency is calculated based on the material passing through the crusher and on the record of loads registered by the operators. Information on the cycle times was taken for the current extraction conditions and is used as a basis for the future calculations and schedule time forecast.

The density of the rock is taken as 2.5 for the blasted rocks in all the domains. This value is taken from the geology information on the area (Pirotte, 2019). A detailed density testing must be performed to establish the rock's real density and have a more accurate calculation.

There is no study of blasting results regarding the fragmentation and swell factor; hence this factor has been taken from values reported in the literature. According to Roseke, 2013, the swell factor generated in the blasting for the limestone is 65%.

5.5.1. Extraction requirements

The plant is currently producing two lime products differentiated in the amount of allowed S an P in the final product. Therefore, a differentiation in the raw materials fed needs to be done since the beginning of the process. The first product is lime with content of P ranging between 0.03% and 0.08% and S content of 0.05% to 0.3%; the second product is a lime of high quality with the content of P below 0.01% and S below 0.01%.

The limestone with low content of P_2O_5 is currently bought from external mines and transported to the plant. It arrives already crushed and meets the requirements for the plant feed. It is used in Kiln 04 (H4) in which natural gas is used for the combustion to ensure the low concentration of P and S; on the contrary, the limestone with a higher content of P_2O_5 is extracted from the north pit, crushed, and used to feed kiln 03 (H3).

The amount of material to be extracted is based on the production capacity of the kilns, and therefore on the stone feed necessity. Figure 26 shows the amount of crushed stone is shown in blue, and the kiln feed in orange. It considers the process done to feed the H03 since only the feed of this kiln is extracted from the mine.

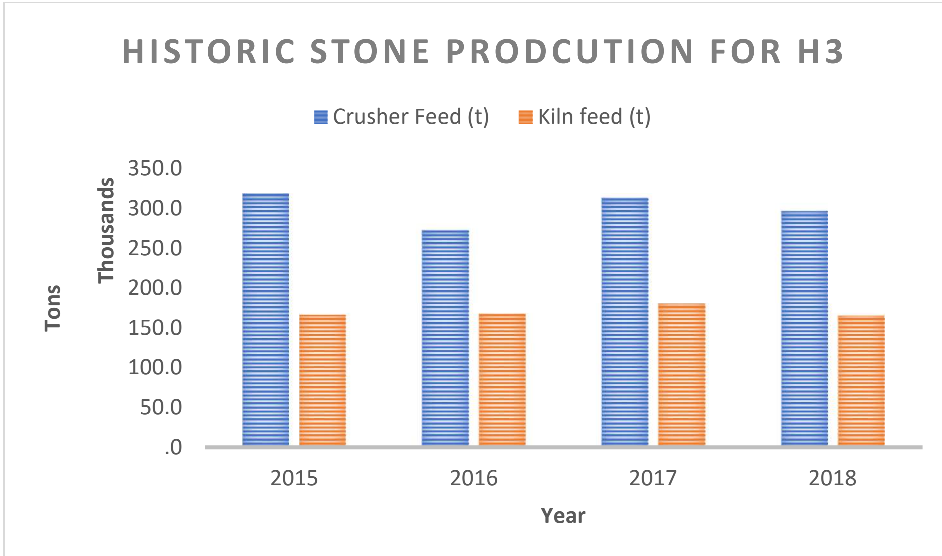


Figure 26:Historic crusher production and Kiln feed

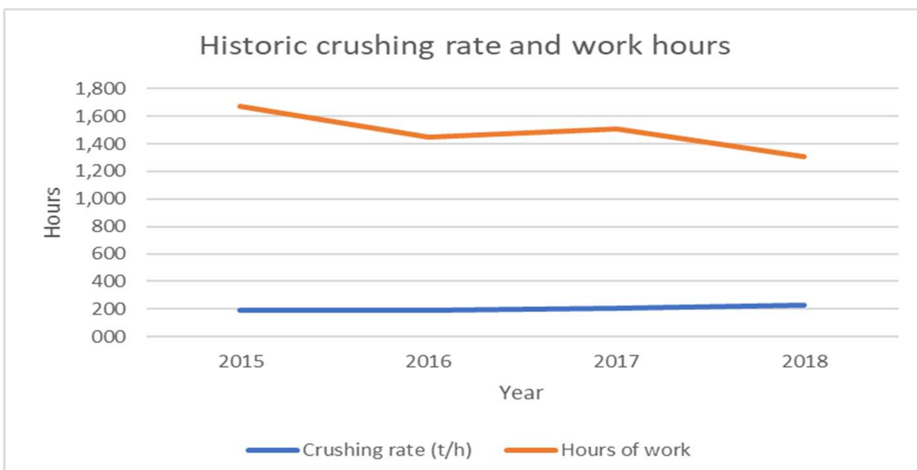


Figure 27: Last 4 years crusher rate and hours worked.

The information in figures 26 and 27 only shows the fraction of feed that is processed on-site. However, the material requirements calculation shown in table 6 is made assuming the feed both kilns since the final objective is to establish planning that includes the extraction of limestone with Low P contents from the southern area of the current mining license.

Based on the information obtained, it is possible to see that out of the mined and crushed reserves, only approximately 64% meet the size requirements to be fed in the kiln. The remaining 36% is fine-grained material not used in the process and considered aggregates and "fine waste" (material that meets the quality requirements for production but is below the size to be fed to the kiln).

Finally, the feed to product relation in the kiln is estimated to be 57%; this means that to produce 1 ton of lime is necessary to feed 1.754 tons of limestone. With these process indicators, it is possible to estimate the required extraction per year to ensure the constant feed of the process.

Table 7 present the calculation of stone requirements per year for each Kiln. Also, it establishes the amount of material to be extracted to meet the feed requirements. For the calculations, it is assumed operation of the kiln of 365 days (7 days per week) and the operation of the mine and crushing system to be 250 days per year (5 days per week).

Table 7: Stone requirements per kiln.

	H3	H4	Total	Comments
Kiln production (t/day)	350	270	620	
Kiln feed (t/day)	614	474	1,088	Product/feed rate 0.57
kiln feed (t/year)	224.123	172.895	397.018	365 days of kiln work
Stone feed to crusher requirement (t/year)	350.192	270.148	620.340	Fines fraction 36%
Stone feed to crusher requirement (t/day)	1.401	1.081	2.481	250 days crusher work

Table 8 shows the required hours of crusher work for the estimated production for both kilns. Additionally, it shows the available hours assuming shifts of 8h and the number of shifts requirement for both the current crushing rate and the nominal crushing rate that the crusher can reach (420 t/h). An efficiency factor of approximately 90% is applied to the nominal rate obtaining 380 t/h. It is crucial to highlight modifying the conveying system after the crusher is needed to reach the nominal rate.

Table 8: Required crushing hours and shifts

	Crushing rate	Required hours (h/y)	Available hours in a year (8 hours/shift)	# of shifts required
Actual	203,33	3055,25	2000	2
Nominal	380,00	1632,47	2000	1

The mine planning and scheduling are made, having a fixed target the feed for the Kiln 4 (H4). This means that the further scheduling scenarios simulation target is to ensure a minimum extraction of 270.148 of limestone with P2O5 content lower than 0.039%. The limestone extraction with content higher than this is planned to be sent to the Kiln 3 (H3) to complement the material extracted from the current pit under extraction (north pit). Only the stone classified as Q6 in the block model is entirely waste.

5.6. Life of mine calculations

The life of mine estimation (LOM) is based on the required production to feed the current plant necessities and the available reserves. It can change along the time when adding newly found reserves due to exploration studies or when the plant needs change both due to an increase of production capacity or to a decrease in the production rate in a certain period. It is essential to understand that the capital cost involved in the enlargement of the plant is usually higher than the additional operational costs of building a bit oversized plant.

The Life of mine calculation defines the time frame in which the extraction is going to be developed, and it provides a valuable indicator for the investors since the mining activity requires significant investments in equipment and therefore, the return over the investment and the profits made are a key decision parameter (Lhoist Mining Team, 2019).

For the southern area of Calgov, the LOM is calculated for reserves of limestone with low P2O5 and Mid P2O5 content. Even though the life of mine for Mid P2O5 material is presented, its extraction will advance at a variable rate since it is planned to be added to the north pit extraction for feeding the H04 kiln and mined mainly to provide access to the low P2O5 material.

The life of mine calculation is shown below. They are simple calculations based on the total amount of reserves found and the production rate already existing in the plant. It indicates the time frame in which mining activity can be carried on; It sets a maximum time when the current reserves will be depleted if the production is carried on under the current or designed production conditions. Nevertheless, it does not consider the economic variables such as cost or revenues in the calculation. The final decision on how many years the mining activity will continue has to include the economic evaluation of the extraction.

- Life of mine low P2O5 reserves

$$LOM = \frac{9.5\ Mt}{270.150\ {}^{t}/_{y}} = 35\ y$$

- Life of mine mid P2O5 reserves

$$LOM = \frac{3.5\ Mt}{350192\ {}^{t}/_{y}} = 10\ y$$

When economic variables are evaluated, a decision must be made on the optimal time to keep the mining activity going on. The economic results may lead to the consideration of extracting only part of the total reserves, the decision to increase the plant capacity to get a higher profit and reduce the life of the mine, or the decision on not developing the mining activity at all.

Since Calgov already has an installed plant with a defined capacity, commercial agreements for its products, and favorable market conditions, it has been decided to perform the initial mine planning based on this result. Since the life of mine obtained is 35 years, and there may be additional reserves in other areas of the mining concession, it is recommended to evaluate the possibility of increasing the plant capacity to take the maximum advantage of the favorable market situation. This option should be considered for its study in the future.

5.7. Scheduling

The mine scheduling process is made using the mine planning software from Geovia, MineSched. The block model is constrained to include only the blocks inside the ultimate pit established for the southern area. A new attribute has been assigned to the blocks to group the previous classification by geological unit into the 3 possible materials to be extracted:

1. Low P limestone: includes material from units Q2, Q4, Q7, Q8. It means the material with P2O5 Content lower than 0.039%, and content of all the other elements is lower than 0.7%
2. Mid P limestone: includes material from units Q1, Q3, and Q5. In these units, the content of P2O5 is higher than 0.039%, and the content of all the other elements is lower than 0.7%
3. Waste: this includes the geological unit Q6 in which the material is mostly clay type, and even though the content of P2O5 is low, the content of SiO2, Al2O3, MgO is high (higher than 1%)

The mean composition of the units is shown in table 3, chapter 4. With this classification of the block model, the mining area is divided into small, intercalated areas representing the geochemical distribution of phosphorus in the mine. The scheduling process is going to consider the removal of sequence lots of blocks. Each lot consists of the required material to keep the phosphorus concentration been feed into the crusher between the required limits. (Ito & Nishiyama, 2003)

Different extraction directions have been considered and simulated to evaluate the best operational option to develop the extraction of the southern area. Two scenarios consider the extraction of the blocks with the advance been in opposite directions; for both scenarios, it has been considered to start the extraction in bench 3 level 660 m.a.s.l going upwards first and leaving the start of extraction of the levels below to an advanced stage of the mine life.

5.7.1. Description of scheduling options considered

<u>Advance to the north</u>

It considers the start of the extraction in the southeast end of the owned land. It means opening a new satellite pit with an advance towards the north; it will be connected with the northern pit towards the end stage of the mine extraction.

This extraction sequence requires a high preparation cost at the beginning involved in the road and access construction for the new mining area; besides, it requires starting from the beginning with the extraction of the vegetable cover and the overburden material.

The extraction scheme mentioned above can be done by temporarily extending the south road of the northern pit to create the first entry and, further, when the extraction advances, create the final road for the satellite pit. It has as advantages the start of the extraction of reserves from Q1 and Q2, which are suitable for lime production (med P2O5 content and low P2O5 content, respectively).

In addition, the unknown material present in the fault area (limit between north and south area) is going to be extracted at the end of the mine life, and therefore, it gives more time for a detailed study on the quality of the limestone of this zone and for the decision on how to deal with it.

On the other hand, the disadvantages are the requirement of a high amount of development before the start of the extraction. Second a higher cost at the beginning of the project due to the longer transport distances to reach the crusher, finally the possibility of having longer times and more complex processes in the permit obtention since it is an opening of a separated pit from the one existing, therefore requires a new complete environmental impact assessment (EIA) and labor planning approval from the government.

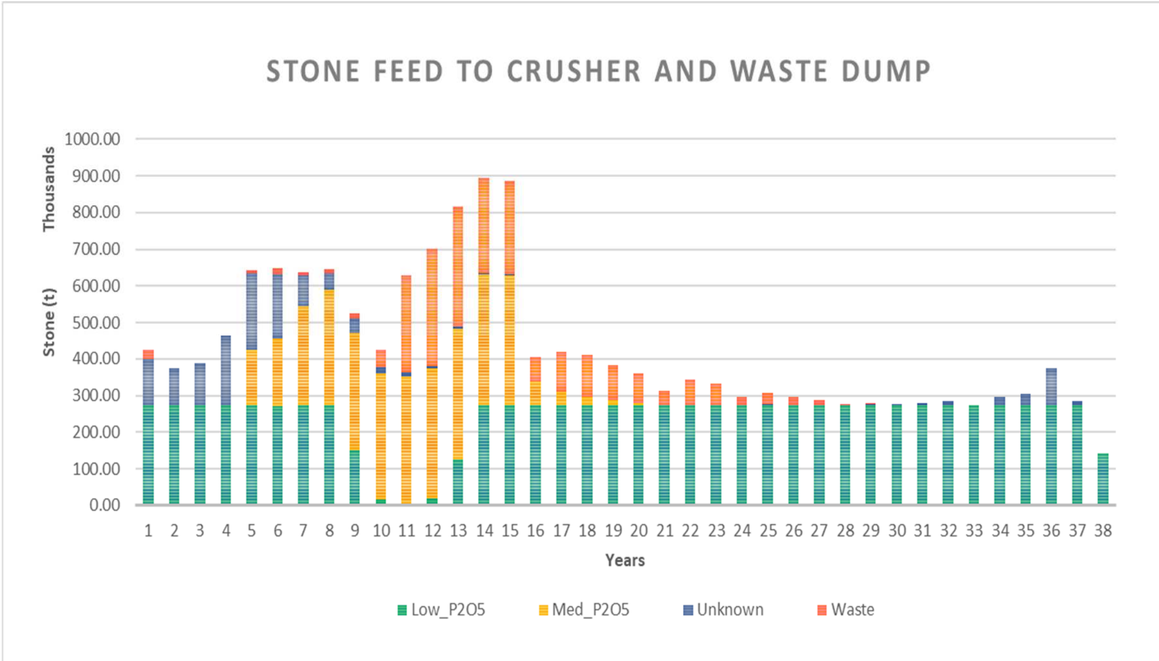


Figure 28: Amount of stone coming from quarry to crusher or waste dump when advancing towards the north.

In figure 28, the extraction for this option is shown. As can be seen, this schedule option shows a problem between years 9 to 13 in which the extraction of med P2O5 material and waste is high, and therefore the target extraction of low P2O5 limestone is not reached. This issue changes in the further years, and the extraction continues reaching the target production for all the other years

Advance to the south

The second scheduling option considered is the extraction of the southern reserves starting in the limit of the south area with the north pit. It means to perform the extraction of the south area as an extension of the current extraction developed in the north. The advance for this extraction sequence is towards the south, expanding the current pit until it reaches the maximum size allowed limited by land ownership.

This option has low development costs since the current road may be used to start with the extraction, and it can be expanded with the advance of the quarry; therefore, the transport cost will start being low and increase as the road gets longer with the mine advance. Furthermore, the lowP2O5 limestone extraction can start from the beginning with the extraction of parts of units Q7 and Q8 that already outcrops in levels 2 and 3 of the north pit. The permit obtention for it is less complicated since the extraction can begin with the presentation of a modification on the labor plan and area affected by the extraction; no specific environmental impact assessment is needed.

The main disadvantage of this sequence is the necessity to deal in the early stages of the project with the unknown material found near the fault. It is necessary to define the possible use of this limestone and decide whether it will be taken to waste dump, to the production of lime with mid P2O5 content or if it has the quality to be destinated to the Low P2O5 lime production. The study on the chemical characteristics may make a difference in the costs and investment incurred at the beginning of the extraction.

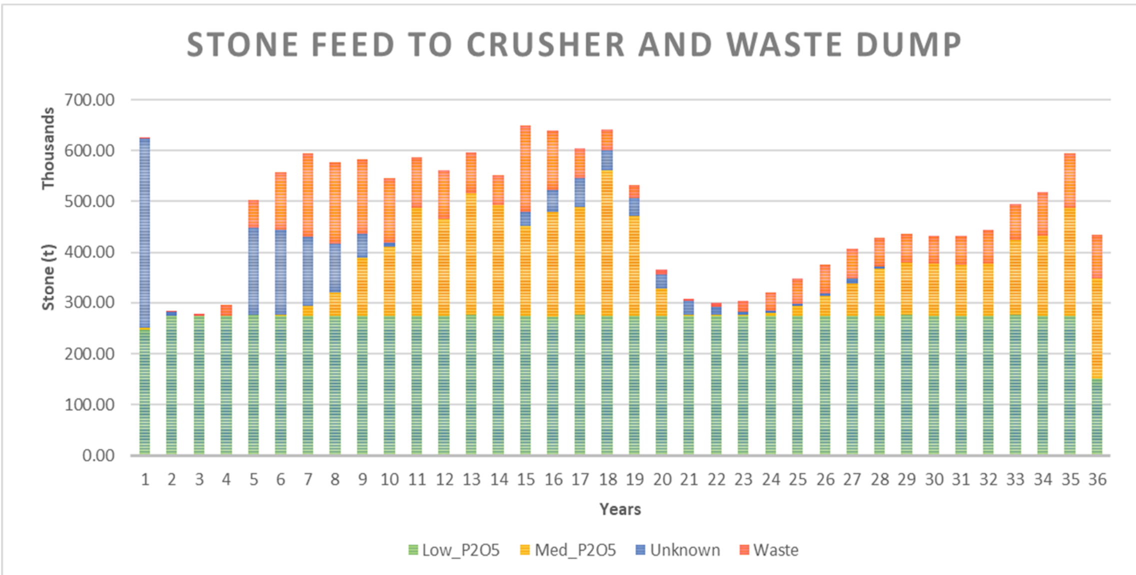


Figure 29: Stone feed per year to crusher and waste dump when advancing to the south.

Figure 29 shows the stone extraction per year with this advance scheme. The target production of low P2O5 is reached almost every year, and the extraction of mid-P2O5 Stone and waste is distributed through the mine life. As it can be seeing a significant portion of the unknown material has to be extracted in year 1 to provide access to the low P2O5 stone. Also, an increase in the extraction from year 5 to 19 is present, with approximately half of the total extraction being mid P2O5 stone and waste; during these years, both kilns will be fed with stone extracted mainly from the southern area.

Even though the possibility of producing a blend of materials has been considered looking to increase the possible earnings and give the best use of the reserves, it has been disregarded and was not included as a schedule to evaluate. This decision was made due to space restriction and the requirements of this option to produce stockpiles with different qualities. As mentioned before, the available space for the development of the process is minimal. Furthermore, this option requires a change in the mining method used, and more administrative work is needed for the approval, and therefore the start of the extraction may be delayed.

The blending of materials requires high care with the mix and may require performing changes also in the quality controls made to the kiln feed to ensure that the adequate blend is used. This change affects the sample taking process and its frequency, but it may also be required to increase the investment in new quality control equipment and significant modifications in the further process.

Out of the two advance options, the development of the extraction starting in the north pit limit and advancing towards the south is the best operational option since it represents the fastest path to get access to the highly valuable material with a low P2O5 content of units Q7and Q8. Also, the preparation, development, and timing to obtain the necessary approvals needed to start the production are lower than the opposite option.

When comparing the extraction of both schemes, it can be seen from figures 28 and 29 that the south advance has a more uniform distribution of types of materials extracted, allowing to have continued access to low P2O5 stone and to extract the mid P2O5 and waste across the mine life. Contrary the north advance scheme shows a lack of the core material (low P2O5 reserves) in years 9 to 12 and high extraction of mid P2O5 and waste in these years.

To completely define which mining scheme is the best option to perform the extraction, an economic evaluation must be done to establish which one provides the best NPV (Net present value). The economic evaluation of the proposed extraction schedules is presented in chapter 6.

5.8. Current processing conditions

The crushing system has been studied to define the adaptations required to reach the target production of both limestones with low P2O5 content and mid P2O5 content. Information was gathered regarding the current state of the crushing and conveying system, and some options for changes are disclosed. The presented options are only conceptual of the available possibilities. The final decision on which option should be chosen had to be based on a more detailed study on the available space, the investment required, and the economic return.

5.8.1. System description

The trucks feed the limestone to a hopper that discharges in a metallic apron feeder with a size between 0 to 900 mm. It enters into the first screening stage, in which it is classified into 3 particle sizes. The material between 0 and 30 mm is taken out to the fines material stockpile and finally to the fines dump. The limestone with granulometry between 30 to 100 mm is taken to the second screening stage, and the sizes bigger than 100 mm are fed into a hammer crusher.

The output of the hammer crusher joins the conveyor belt that carries the 30mm – 100 mm stone and is taken to the secondary screening process, where it is classified again in 3 particle sizes. The lower one 0- 35 mm is taken to the fines dump; the fraction with size between 35mm to 120mm goes to the Kiln fed stockpile, and the stone bigger than 120 mm is fed into a secondary crushing stage in which a roller crusher in a closed circuit with the second screening stage reduces the particles size again.

In figure 30, the diagram of the process is shown. In it, the size distribution of fed and output is included as well as general current mass flow. In red, the waste 0-35mm material is showed and in blue, the circuit up to the kiln feed stockpile. In green, the auxiliary conveyor belt transport for the material coming from other mines (material bought to feed the kiln at a higher cost)

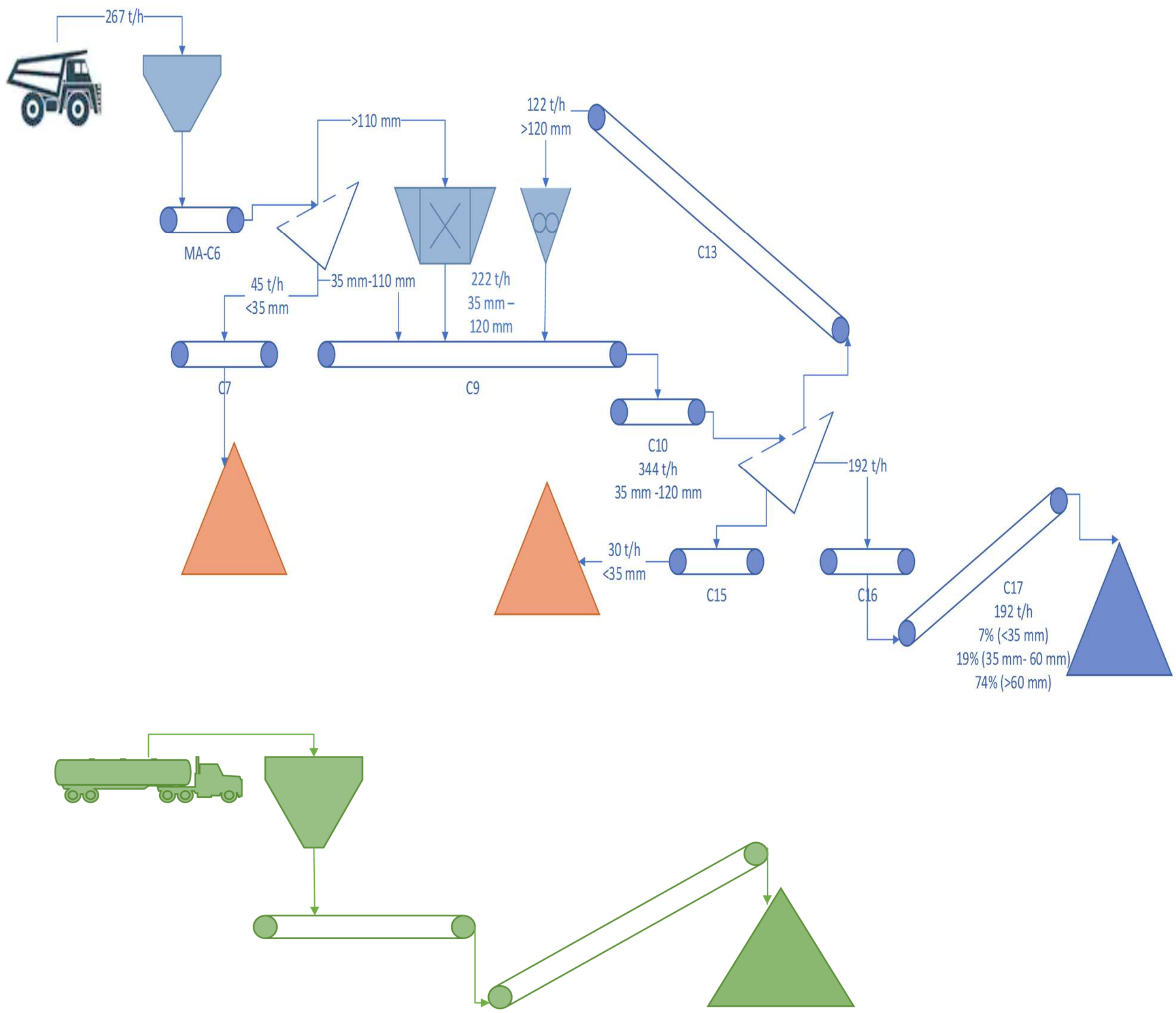


Figure 30: Mass flow of the crushing and screening

The low P2O5 limestone arrives from the external mines in road trucks and is discharged in a hopper connected to a conveyor belt system that takes it into the low P2O5 stockpile that feeds the H4. It arrives at the site already crushed and with the required size.

The system is designed to handle a nominal processing rate of 430 t/h, but it is currently used at a maximum of 260 t/h due to some problems in the conveyors and transport facilities. The entire system is stopped for programmed maintenance once a week. The crusher is in good mechanical conditions, and the motors are big enough to produce at its nominal capacity. The roller crusher is also in good condition, and it only requires the replacement of some of the wear parts. The screens are also in the right conditions, with the meshes and motors well maintained. (Madejon, 2018)

Only the limestone used to feed H3 is crushed on site. There is no possibility of crushing the Low P2O5 limestone since any infrastructure connects the hammer crusher and the screening sequences to the stockpile that feeds the H4.

5.8.2. System bottlenecks

In total, 2481 t/day are required to feed both kilns when adding both types of materials. To get the required amount of crushed stone, it is necessary to operate the crusher at a higher rate of a least 360 t/h. To do this, it is necessary to find the points in the system that are limiting the capacity of it.

In a first analysis made in 2018 of the conveying systems, two conveyor belts were identified to be the ones causing the overall system to be producing less than the possible capacity of the crusher. (Madejon, 2018)

The conveyors are undersized and were not designed to operate under the maximum capacity of the system. These conveyors are the C16 and C17 in the previous diagram of the process (figure 30). Both conveyors are at the end of the process and correspond to the ones that take the material from the second screening stage to the kiln feed stockpile.

All the other components of the crushing system are oversized for the actual processing rate, and they can operate at a higher rate. In table 9, the capacities of the system and some other specifications are showed.

Table 9: processing equipment capacity

Equipment	Capacity
Primary Crusher	430 t/h
Metallic apron	900 t/h
Secondary crusher	175 t/h
C7	100 t/h
C9	700 t/h
C10	700t/h
C13	300 t/h
C15	200 t/h
C16	251 t/h
C17	205 t/h

In addition to the previously mentioned bottlenecks, it is necessary to find a way to connect the current crushing and screening process with the stockpile where the feed for the H4 is stored. It is necessary to invest in an upgrade to the current system and add some new transport infrastructure in the crushing plant.

5.8.3. Proposals for modification

Four options are explored to reach the target amount of crushed stone. The first two consider the modification of the current crushing installation and some extra conveyors to the system. The third and fourth one considering leaving the current crushing and transport infrastructure as it is now and using additional resources to crush the low P2O5 limestone. The options are briefly explained. A more detailed engineering study and design are required to know the total amount of investment required for each case.

5.8.3.1. Options with modification of the current infrastructure

According to the calculations made on the conveyor system capacities and based on table 8 in the previous chapter, it is necessary to increase the capacity of conveyors C16 and C17 to handle approximately 305 t/h to take the crusher to its nominal production. It means it is needed to increase the capacity of the C16 by 21 % and the capacity of the C17 by 49%. To do this, mainly the entire conveyor infrastructure (idlers, pulleys, housings, and engine) needs to be changed to increase the size of the conveyor and therefore increase the capacity. (Metso, 2018)

In addition to the increase in capacity, it is necessary to connect the Low P2O5 Stockpile with the crushing system. It can be done using different mechanisms that allow changing the discharge direction of the flow when moving through the system. These mechanisms include bypasses and conveyor belt diverters. This option allows discharging in a new conveyor going from C16 or C17 to the stockpile. It requires the redesign of conveyors C16 and C17 to place the bypass and requires installing a new band in the area between the two stockpiles. Currently, this area is used for the transport of spare parts and maintenance of the conveyors.

Another option is to use reversible discharge band systems at the end of the C17 to allow the discharge in both stockpiles. This option must include the change of conveyor belts C16 and C17 and the installation of the belt at the discharge point of C17. The installation of additional infrastructure for the discharge band is required.

These options need to be carefully designed since they need to be located after the control size stages. It means at the end of the process where the C16 and C17 are located. It must be studied in detail to choose the best configuration since the current inclination of the conveyor belts is high, approximately 18º, and the space availability in the area is not enough to reduce the inclination in the new conveyors installed.

5.8.3.2. Options for keeping the current infrastructure.

A completely different approach to the production of crushed low P2O5 limestone from the southern extension area is to keep the current infrastructure, including the conveyors C16 and C17, with their current capacity. Two options are possible to achieve the additional crushing needed once the southern extension area starts its mining.

The first option is to invest in a mobile crushing system; it can crush the material directly in the loading benches. The crushed stone can be taken either by conveyor or trucks to the current hopper that feeds the low P2O5 limestone stockpile. The loading benches in which the mobile crusher can be functioning are the bottom pit and bench 3 at level 640.

This option is convenient if the selected scenario consists of performing the extraction towards the south and advancing in the benches above bench 3 before beginning the lower benches extraction. This sequence can allow having enough space for the installation of this type of equipment that requires about 90 m2. furthermore, the movements of the crushing and conveying system can be reduced if the extraction is focused on one area for several years. (Metso, 2017)

This method has several advantages like improving the transport system, increasing the crushing capacity of the plant, and provide flexibility to the operation, and having a backup system to continue the production if the current installation suffers a big failure. Among the disadvantages are the investment required that can be high depending on the type of crusher and the configuration chosen for the auxiliary operations. Besides, it requires space in the benches to operate and to store the crushed material in temporary storage.

The second option consists of modifying the current shifts of the mine to duplicate the crushing time. This option can allow crushing one material type in the first shift and the other in the second shift. It does not require the investment in the additional mining equipment that the other options require, but it needs to hire a complete shift of operators, in total 4 more people. It is a critical point to consider since the increase in operation time implies more exposure to safety risks and higher operation costs. Furthermore, this option still entails the addition of a bypass to the conveyors C16 or C17 to connect the low P_2O_5 stockpile.

5.9. Waste generation

The presence of waste material presents a challenge for the plant since it must take some actions that represent a cost to add to the process. Some of the items to count for in waste management are:

- Find an extraction scheme to avoid dilution and keep as much as possible a constant relation of ore-waste extraction.
- Waste storage requires using some areas to handle the material, temporary facilities, and final disposition spaces. It is essential to optimize the space need while considering the chemical and physical properties to avoid significant environmental impacts.
- If possible, create a strategy for the use and final disposition given to the waste.

Calgov, in its regular operation, generates two types of waste. The first type includes all the material that does not meet the desired chemical composition for the production. This material corresponds to limestone with an abundance of clay, high content of SIO_2, AL_2O_3, FE_2O_3, and limestone with elevated concentration of trace prejudicial elements as P_2O_5 or S. In the southern extension area, this type of waste is found only in the Q6 geological unit.

The second type of waste corresponds to limestone particles that meet the chemical composition targets, but due to its fine-grained size, it cannot be used in the process under the current conditions. It is generated as a result of mining activity and processing of the stone through the grinding and size classification stages. This material causes significant losses to the process, not only in the form of valuable reserves sent to the waste but also an increase in the waste management cost and the amount of space needed. Currently, this portion of the material corresponds to 36% of the total extracted reserves. It comprises material sizes between 0- 35 mm and is stored and, when possible, sold as a whole mix.

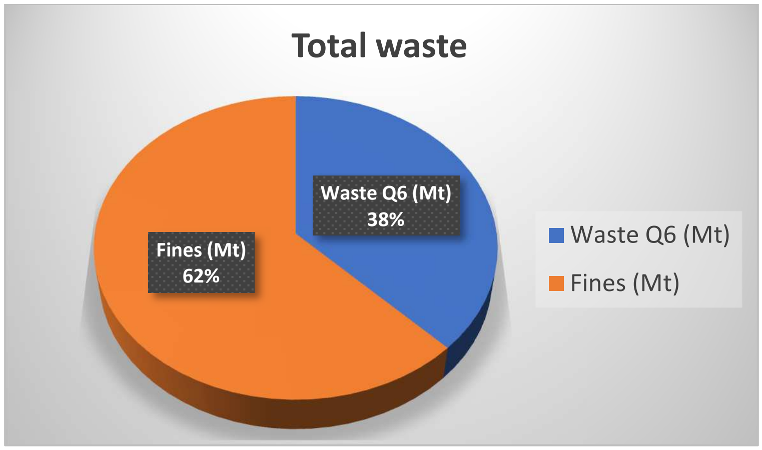


Figure 30:Total waste generated if the entire reserves are extracted and processed under the current plan conditions (assuming the same overall recovery)

Currently, waste production represents a bottleneck to Calgov since the space availability inside the land owned by the company is very limited. The rate of fines production is higher than the rate at which it is consumed by the companies that buy it. Due to this, temporary storage on one side of the plant was created. It has been growing over time, and it can reach the maximum storage capacity if nothing is done. Furthermore, the waste production of both types of materials is going to increase by the opening of the south area and the increase in stone production.

From figure 31, it is possible to see that the highest proportion of the total waste generation, 62%, corresponds to the fines generated during the extraction and processing of the limestone. Therefore, any effort to increase the fraction of reserves recovered or valorized must be made. The fine-grained material can have a high impact on the waste problem.

Figures 32 and 33 show the waste generated per year and cumulative in the project life if all the current reserves are extracted. It is planned to generate approximately 7.3 Mt of waste, 2.5 Mt of waste coming from the extraction of Q6, and 4.8 Mt from fines. The rate at which the fines waste is generated may vary through the project life since it is directly proportional to the processing operation and the plant feed.

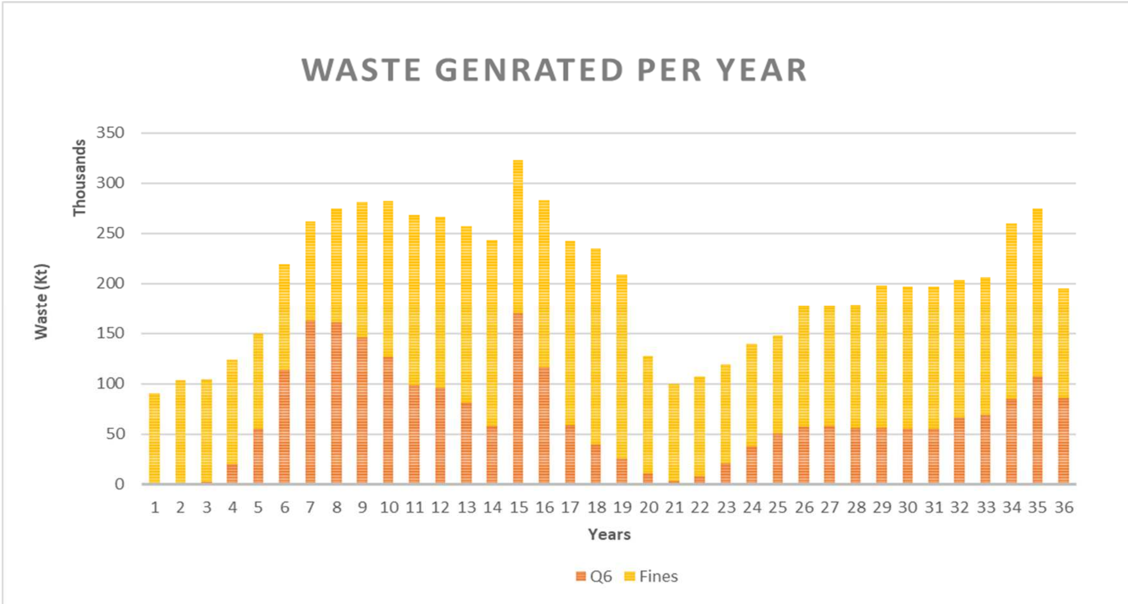


Figure 31: Amount of waste generated per year classified by the fine portion generated during extraction and processing operations and the waste extracted as part of unit Q6

Figure 32: Total waste generated across the project if all the current reserves are extracted.

It is important to remark that it is impossible to fill the area open for the current mining in the north pit since it still contains valuable reserves and will remain under production for several years. In addition, there is any current existent space suitable for temporary waste storage.

To size the approximate space needs for the waste disposal, a waste dump place is simulated, assuming an angle of repose of 36° and a bench height of 10m. A conservative approach must be taken regarding the repose angle to ensure the stability of the fill location. The possible waste dump location is unknown, and therefore, only an estimation of approximately space needs is possible. The waste dump design needs to be adjusted following the topography, the stone characteristics, and conditions of the area designated for this use. (International institute for environment and development , 2002)

The waste fill simulation showed the necessity for approximately 11 hectares for the storage of material originated from Q6 extraction, assuming a density of 2 t/m^3 and 35 hectares for the fine waste storage with a density of 1.8 t/m^3. Space estimations are made separately for both materials since there may be possible in the future for the commercialization of a fraction of the fine waste.

Further studies need to be performed to establish the real repose angles and the swell factor for both materials to have a more accurate estimation of the spatial needs for waste dumps. It is important to know that geotechnical and geomechanical studies to establish the rock parameters can lead to the optimization of the waste dump configuration and its safety. (Castro, et al., 2006)

Some actions can be taken to reduce the amount of waste generation, such as:

- Establishment of a secondary process to perform classification by size of the fines portion to valorize the different sands and gravels obtained for the construction market.
- Improve the blast design and performance of the explosives to reduce the fines obtained.
- Reduce the fragmentation of the stone in the handling before the crusher by reducing or eliminating the throwing operation. It is necessary to study the possibility of transporting the stone directly from the blasting to the crusher.

All these actions need to be carefully considered, and a cost-benefit study must be made before deciding to implement any of them

5.10. Equipment requirements

The extraction of the southern area is planned as an expansion of the mine production to avoid purchasing raw material from other mines. This expansion makes it necessary to invest in new equipment to handle the new material production and complement the equipment already used in the northern area extraction.

The equipment must scale based on the production requirements, the amount of waste to be handled, and the geometrical configuration of the mine roads and benches. Since it can be used in both the south extraction area and north current opened pit, it needs to be fitted to the existing mine.

The loading equipment needs to be useful in both mining areas. In addition, there is a space limitation when considering the throwing operation and the road width and accessibility from level 3 upwards. The equipment considered for truck loading is a wheel loader able to provide flexibility to the operation and be moved easily to different locations. The equipment used for material handling is an excavator.

The company already owns 2 backhoe excavators Cat 336, for the material handling process; therefore, this equipment can be used for the south extraction. The mine currently owns a Cat 972H with an approximate capacity of 10 tons for the trucks loading. It is not enough capacity to support both extractions; furthermore, it has been overhauled once and is at the end of its productive life.

A calculation was made to acquire a new loader. The option to upgrade to a loader with a large bucket capacity is a good option since it is planed only to reach up to level 3. Also, it can serve as auxiliary equipment for the throwing activity in the lower levels. For this reason, loaders with a bucket capacity of 15 tons or more are considered to be more convenient.

Even though the south road design can be done to support bigger trucks, the north pit is already built, and there is not project to modify the existing roads. Hence the equipment necessity has been calculated for both areas using trucks with a nominal capacity of approximately 80 tons.

In table 10, the results of the calculation for the equipment required are shown. The extraction of unknown material found at the very beginning of the mine life is planned to be performed entirely by outsourced equipment

Table 10: Equipment needs

Loading	
Estimated bucket capacity 18 t	
Amount of extraction per year	1115000
Number passes required per truck	4
Cycle time (time to load one truck)	5
Number of loaded trucks per year	21000
Number of loaders needed	1

Transport	
Estimated truck nominal capacity 80 t	
Amount of extraction per year	1115000
Estimated cycle time	13
Required cycles per year	16397
Possible cycles per year per truck	8308
Number of trucks needed	2

6. Economic Calculations

The financial evaluation of the different extraction options can allow the right decisions regarding the execution of the project and have an initial look into the possible results of the entire operation. It is necessary to estimate the costs and revenues for the project and perform a sensitivity analysis in which the possible outcomes of it are evidenced if something changes in the future.

6.1. Costs

The extraction activity cost includes all the expenditures to carry out the mining activity and the further size reduction and clarification of the stone. They are classified into two main types, Capital Costs (investments required) and operating costs. A further division of the total cost is made to know which items have a more substantial influence on the extraction cost per ton.

6.1.1. Capital costs

The capital expenditures comprehend the cost of the extra equipment necessary for developing the southern extraction area. As mentioned in the equipment requirement chapter 5.10, it is required to increase the fleet to ensure the total production; moreover, it is necessary to modify the conveyor belt system after the crusher to reach the maximum production and eliminate the bottleneck represented by the very limited transport system that has reached its limit of production.

In table 11, an approximate of the possible capital costs are detailed. Since no decision has yet been taken regarding the crushing area upgrade, this cost is shown as a wide range. More details on the possible options were detailed previously in chapter 5.8. (Infomine USA, Cost Mine Division, 2018).

The permit cost is an estimation based on the administrative fees that have to be paid to the Spanish authorities and the possible expenditures due to the development of technical and environmental reports on the exploitation results. This cost is given as a range since it varies based on the permit type and the requirements made by the authorities over the reports presented. (Gobierno de España, 2020)

Table 11: Investment required for southern extraction project

Equipment	Estimated investment ($)
Truck	700,000
Loader	900,000
Crushing upgrade	900,000 - 1,5000,000
Waste dump, land acquisition	20.456 Eur/ha (Ministerio de Agricultura, 2020)
Preparation	80000
Permits	5000 -10.000000

6.1.2. Operating costs

The operation cost is estimated with the current conditions of extraction. It is assumed that the same mining method is going to be used in the southern extension area. According to this, it is possible to use the current cost information to make an approximate forecast of the cost for the extraction in the new area.

The cost calculations are assuming the same shift will be kept so that the labor costs will remain the same as they are now. They will increase only with the regular salary increments per year but not because of new hiring.

Table 12 shows the cost of the operation. It considers the fragmentation by drilling and blasting on each bench. An initial ore and waste handling by throwing it into the loading places located in the bottom pit (590 m.a.s.l) and the bench 3 (640 m.a.s.l) using a backhoe. A loading and transport cycle using a loader and a truck. The total extraction cost of limestone is 4.67 $/t. The cost estimation has been made using the information on the current extraction costs and the Cost Mine, mining cost estimators guide. (Infomine USA, Cost Mine Division, 2018).

Since the cost information is taken with the extraction results for 2019, an estimation is made to get the possible actual cost. In the past 3 years, the increase in mining and processing cost has been 11%, 9%, and 9.6% for 2017 to 2019, respectively. Therefore, an increase factor of 10% is applied to the actual cost to get a cost forecast for 2020, obtaining a total operation cost of 5.14 $/t

Table 12: Cost per unitary process

Process	Cost 2019 ($/t)	Cost 2020 ($/ton)
Drilling and blasting	0.86	0.95
Backhoe (material preparation)	0.60	0.66
Loading	1.06	1.17
Transport	0.84	0.92
Crushing	0.91	1.0
Preparation (Before feeding the kiln)	0.40	0.44
Total	4.67	5.14

The only waste produced consists of the fine material obtained due to the rock fragmentation, crushing, and screening process. This material corresponds, as mentioned before, to approximately 36% of the total rock been extracted. It is temporarily handled in a stockpile near the plant and sold as an aggregate for the construction industry. The cost of extraction of it is already included in the previous cost calculation. The cost of storage and handling (transport by belt conveyor and disposal) is 0.4 $/t.

It is sold at a price of approximately 3 $/t. (Junta de Andalucia, 2019). It is sold as a whole mix; any classification by size is done. The consideration of a further process of classification of these aggregates can allow having a variety in the product range and possibly increase the sales and a rise in the revenue obtained for this material.

The southern area has the additional extraction of waste from the unit Q6. This material is going to be blasted and transported to a waste dump. It means that the crushing and transformation cost is not involved. On the other hand, it requires temporary storage in the mine, and final disposition costs, increasing the overall cost of extraction for the southern area.

Table 13 shows the cost estimation for waste extraction and handling. It is only an estimation based on the possible operational costs of the waste facilities. The costs do not include land adequation and the permits required for the waste disposal; these costs are included in the initial investment needed. Specific studies need to be performed to establish the best location and the adaptation needs of the waste dump place. (Infomine USA, Cost Mine Division, 2018).

The same cost increment factor of 10% is applied to the waste operation cost to get the forecast of cost for 2020, obtaining a total waste handling cost of 4.4 $/t

Table 13: Cost of waste handling process

Waste handling cost	Cost 2019 ($/t)	Cost 2020 ($/t)
Waste extraction cost	3.3	3.63
Waste transport cost (transport from the temporary stockpile to the final waste dump)	0.3	0.33
Waste storage and handling	0.4	0.44
Total waste handling cost	4	4.4

6.1.3. Overall cost division

The total cost of the extraction includes direct and indirect costs; as mentioned before, the information is based on the current extraction costs. Figure 34 shows a general distribution of the costs, as it can be seen the costs with higher weight in the distribution are the labor, supplies, energy, and the fuel and lubricants.

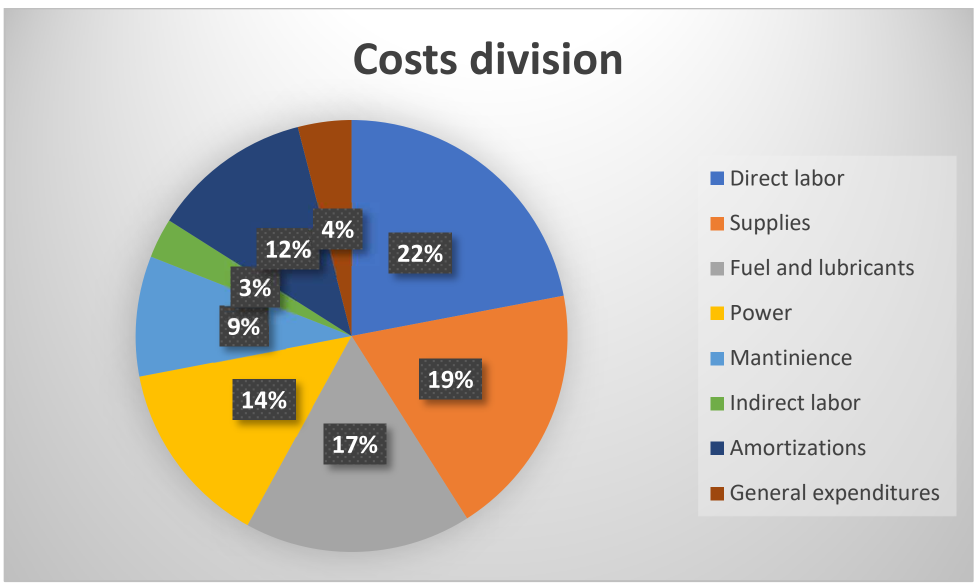


Figure 33: Mining cots division at Calgov plant

The highest cost portion is labor; direct labor corresponds to the equipment operators and field personnel in the crushing area. The indirect labor corresponds to the administrative and laboratory staff, and they are included as a percentage of the total indirect labor costs incurred in the lime production.

The other cost with a strong influence on the overall cost per ton is the supplies. It is related to all the wear parts of drilling and crushing as well as the blasting supplies. It can be reduced by implementing continuous control over the operation and use of consumables and having constant supervision over the results of the blasting to improve the parameters and avoid the overproduction of fines or the waste of energy from the explosives. It is important to note that the blasting should be designed following the target objective. In the case of lime producers as Calgov, to avoid the over fragmentation and obtain a higher portion of sizes over 90 cm.

The fuel and lubricants can be optimized by improving the cycle times, performing road maintenance, mining equipment preventive maintenance, and avoiding the waiting times of the loading equipment. It is essential to mention that due to the low production carried on in the crusher currently the loader waiting times are high; moreover, the current mining method implies higher costs in this item due to the re-handling of material done by the backhoe that has to throw the material several times to reach the bottom pit.

Energy cost can also be reduced in the crusher if it is taken to its nominal capacity increasing the efficiency of the overall system. The amortization is based on the assets current value, and the general expenditures comprehend lab testing, technical and administrative support.

The maintenance cost is derived from the preventive and corrective actions performed in the mine and crushing area. It can be improved by analyzing the failures and, based on them, develop actions to avoid costs and undesired stops. Preventive maintenance is a key step towards having efficient production.

6.2. Revenues

Calgov plant receives profits on the sale of lime as a final product. Also, it sells fines as aggregates, all mixed 0 - 35 m. These two products need to be included when calculating the total result of productive activity. As the revenues are obtained by the sales of lime, and there is no record of the selling price of the crushed limestone in the plant, the net present value (NPV) calculation is going to be made based on the price at which the plant would have to buy the limestone in case it could not be extracted in the own quarry. This price is 8.2 \$/ton for the low P2O5 limestone and 7.6 \$/ton for the limestone with med P2O5 content. The aggregates sales prices taken from the local market and based on the data provided in "*Base de costs para la construcccion en Andalucia*" (Junta de Andalucia, 2019). The price for this hole in one mix is 6.88\$/m^3.

Using the cost information provided in the previous chapter, and the mentioned prices for the materials, the NPV of different situations and outcomes are estimated.

6.3. Economic evaluation of mining schemes

The evaluation of the possible economic results of the south reserves extraction has started with the cash flow and NPV calculation for the two mining schedules presented in chapter 5,7. These calculations allow having a clearer view of which is the scheme that should be followed to get better financial results.

Table 14 shows the parameters used to calculate the NPV. Equations 1 and 2, described in chapter 2.5, were used. The initial investments vary according to the development needed for the extraction. The overall NPV of the project is obtained to compare both options.

Since the initial investment may vary significatively by the time the project begins and depends on the plant modification decisions, the NPV calculation to evaluate the best mining scheme is done, setting an investment cost based on the actual prices for the equipment. In chapter 6.4, an analysis of how the fluctuation of the initial investment cost affects the overall result is made.

The cost had been assigned per process; the waste material is extracted but not processed; the fines produced during the fragmentation and screening had an extra cost of transportation, storage, and handling. When advancing towards the south, the initial investment has an extra expenditure due to the necessity of dealing with the material of unknown quality first.

According to the report "Limestone Market – Growth, Trends and Forecast 2020-2025" released by Mordor Intelligence and Market watch advisory firms, the limestone industry is expected to have a constant annual growth rate of 3.6 % in the upcoming years. This rate indicates the average growth rate for long term investment. Consequently, this rate of 3.6% is set as the interest rate (discount factor) for the NPV calculation and project evaluation. (Mordor Intelligence, 2020)

The fines production rate and the sales rate of this whole mix material had been set under the current situation of the plant. They may vary over time, depending on the market of aggregates in the plant area.

Table 14: Parameters used for the cash flow and NPV calculation

NPV parameters						
Stone class	Price	Cost		Fine material production rate	Initial investment Advance towards the north	Discount factor
Low P2O5	8.2	Extraction and processing	-5.14	36%	3550000	3.6% (Mordor Intelligence, 2020)
Med P2O5	7.6			Fine material sales portion	Initial investment Advance towards the south	
Unknown	7.6	Waste handling	-4.4	10%	3500000	
Whole Mix	6.88	Fine processing	-0.77			

Figure 34 shows the NPV result obtained for the extraction when advancing towards the south; in the first 4 years, the result is negative due to the high initial investment required. It includes a high portion of the unknown material that needs to be removed at the beginning of the project. It starts providing revenues higher than the considered discount rate at year 5 and continually increases over the years until year 25, when a second capital injection is needed to renew the equipment.

Once the second capital investment is made, the NPV falls and starts growing again slower than the previous years. The total NPV reached is $11,67 Million for the entire life of the mine 36 years of extraction, but until year 25, 87% of the total possible NPV is already reached.

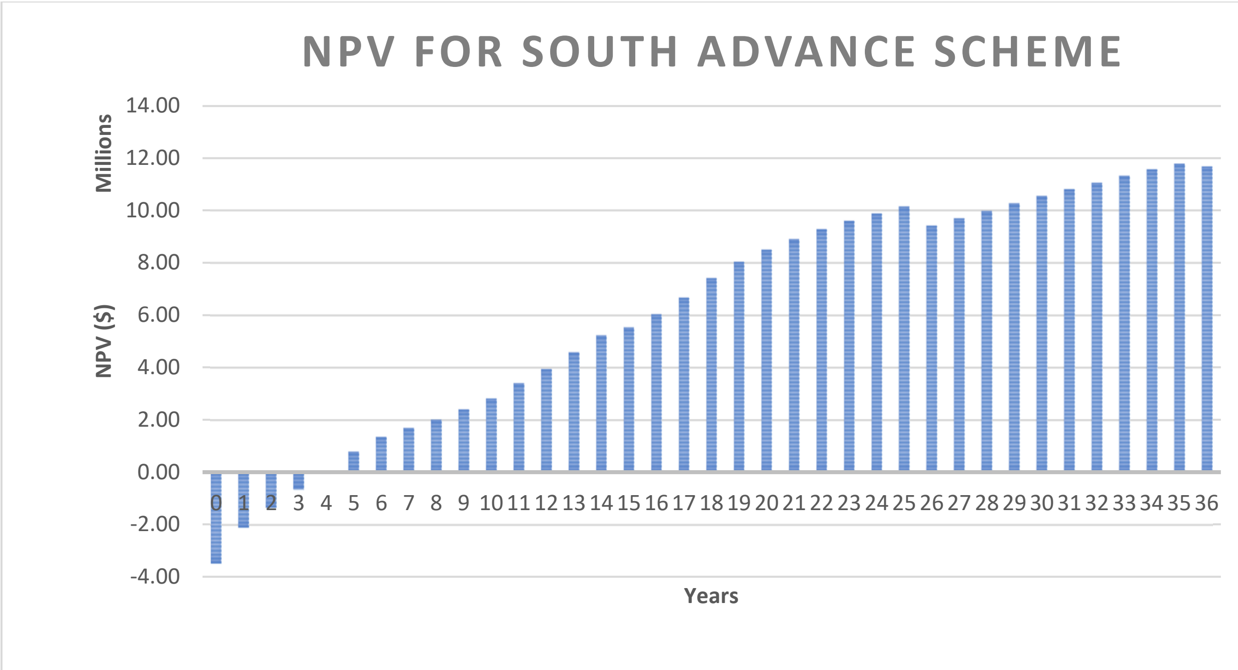


Figure 34: NPV for scheme advancing towards the south

Figure 35 shows the NPV results of the project if it advances towards the north. In the beginning, the investment is high and includes all the equipment needed and the road opening from the existing plant in the north pit to the far south end of the extraction. The project starts providing revenues higher than the discount rate at year 4, and it keeps an ascending tendency until year 9. From years 10 to 12, NPV decreases due to the lack of low P2O5 reserves open for the extraction; therefore, an increase in the mid P2O5 stone and waste extraction is present. A high cost is incurred during this period because the stone needs to be obtained from adjacent extractions, and the lime production is put at the risk of relying on the possible supply of materials from other market competitors. After this period, the NPV starts growing again but at a slower rate, and it never reaches the peak of $5,71 Million obtained in year 9.

For the north advance scheme evaluation, it is estimated that the second investment takes place also in year 25. And the total possible NPV under these conditions is $5,65 Million.

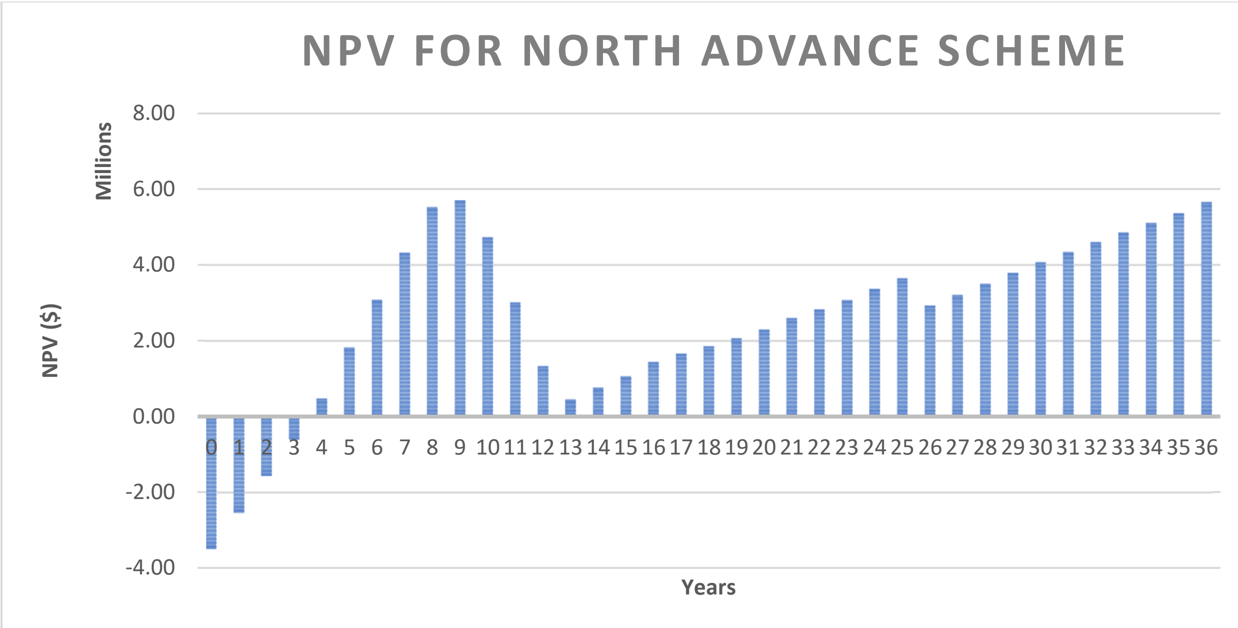


Figure 35: NPV for scheme advancing towards the north

6.3.1. Selection of advance scheme.

According to the information presented in the previous section, the best economic option based on NPV is the mine extraction starting in the north and advancing towards the south (Figure 34). This matches with the extraction scheme that is considered as the best one for the operation in chapter 5,7.

Focusing on this selected option, the rate at which NPV grows is reasonable until year 25; therefore, it has been decided to take this time frame as the target LOM. After year 25 and under the current market conditions, the NPV growing rate is not as attractive as before, and the investment necessary to replace the equipment is not considered an option. Figure 36 shows the NPV generated until this year, in total $10.14 Million. A further economic evaluation must be done if the market conditions change or if changes in the company make it necessary to evaluate if this decision is kept or if it is reasonable to carry on the extraction until year 36.

The NPV calculated and the extraction sequence may vary highly if the unknown material is defined to be either ore or waste; hence, it is essential to perform studies on its characteristics to apply the right classification and assign the cost of its extraction, storage, and transport. Also, to evaluate the profit that may be obtained from it.

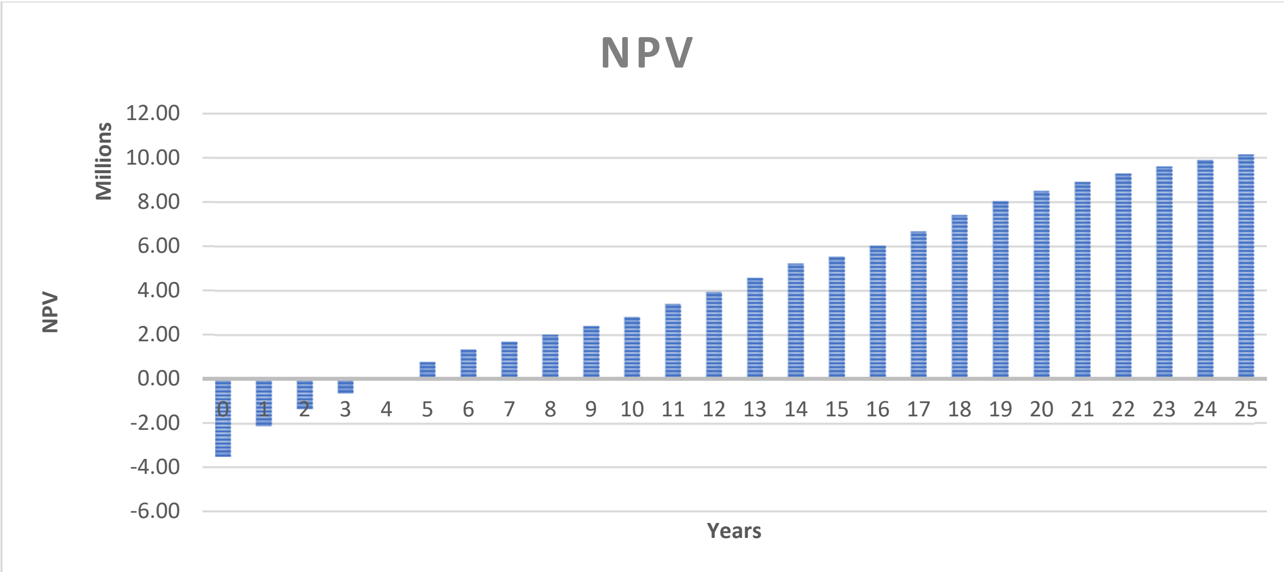


Figure 36: NPV for south advance until year 25

6.4. Evaluation of possible scenarios in the selected Scheme (NPV based)

As discussed in sections 5.7 and 6.3, the scheme is considered the best option both operationally and economically is advancing towards the south and the extraction of reserves until year 25.

Nevertheless, there is high uncertainty associated with the extraction of the limestone with unknown quality located near the fault and with the possible initial investment for the project. It is necessary to analyze different scenarios to avoid overestimating the project results and to understand the possible situations the project may face in its start.

6.4.1. Possible scenarios for the initial investment

The initial investment needed (CAPEX) is not entirely defined because the crushing and screening plant modification has not yet been designed, and decisions concerning it need to be carefully analyzed. In addition to it, the location and final design for the waste facilities are unknown; therefore, the investment cost on land acquisition and adequation for it is only an estimation based on the amount of waste generated and the space needs.

To avoid been too optimistic regarding the required investment, two scenarios were simulated. The first one is made considering only the modification of the current crushing and screening plant using, to a certain extent, the current infrastructure and the installed crusher. The second one includes the assumption of buying a new crusher and the change of the screening and conveyor system. In addition, to evaluate this second scenario, a rise of 20% was added to the land cost.

Figure 37 shows the results of both scenarios as the initial investment raises the time for getting earnings higher than the expected rate is longer. And therefore, the NPV value is lower.

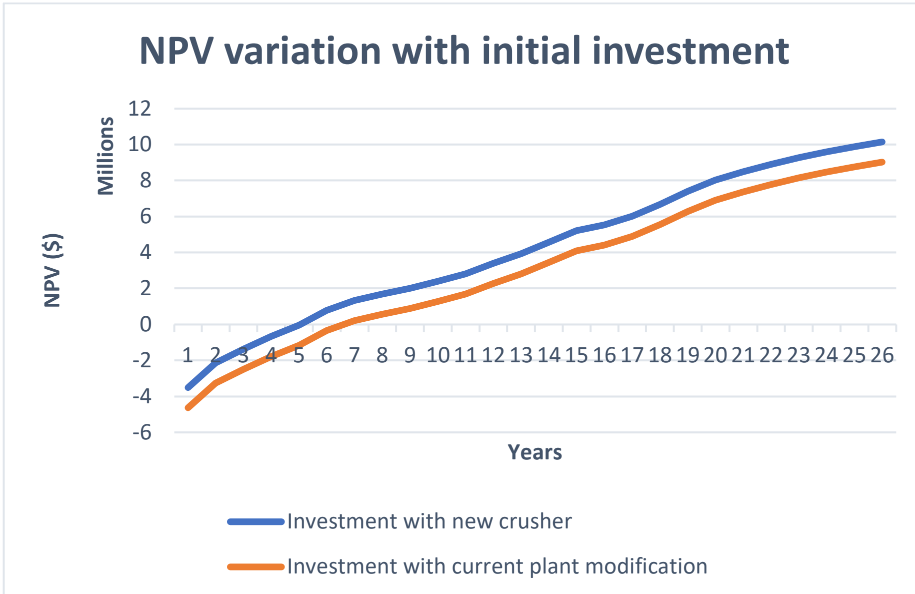


Figure 37 Effect of the initial investment in the project results

6.4.2. Possible scenarios for the unknown material extraction

The unknown material is located in the intersection between the north and south areas of the deposit. It partially covers the reserves of units 7 and 8 (low P2O5 content); hence, it is needed to extract this material to access them. The removal of it must be done at the very beginning of the mine life. Then it is necessary to know how this issue may change the financial results of the project.

Several scenarios had been evaluated in which different portions of this material are considered either ore or waste. Figure 38 shows how the classification of the unknown material affects the NPV generation. The classification of ore and waste portions of the unknown material are varied as well as the ore classification as Low or Mid P2O5. The continuous line represents the evaluation for ore being Low P2O5, and the dashed lines represent ore being Mid P2O5.

As can be seen, if all the unknown material located in the north area of the deposit is waste, the cost of removing, handling, and storing it is very high, and the project may not generate the expected income and may not be attractive to invest in it. The same happens when considering 70% and 50% of the material to be waste. The project only starts to provide revenues higher than the expected minimum rate after 8 years or even more, and therefore it may be a better option to invest in other projects.

When looking at the other two situations, 70 % and 100% of the unknown is ore, the project becomes very attractive, and investing in the opening of the new mining area becomes an excellent option not only to generate good income but also to avoid the risk or been highly reliant on other stone suppliers.

Further studies on the fault zone and the possible mixture of materials of this area need to be made to define its final destination; it is assumed that since it is in contact with both north reserves and south low P2O5 reserves, it has enough quality to be mixed and fed with the mid P2O5 stone and therefore can produce lime with standard quality specifications. Better results may be obtained if a significant portion of it has low P2O5 content.

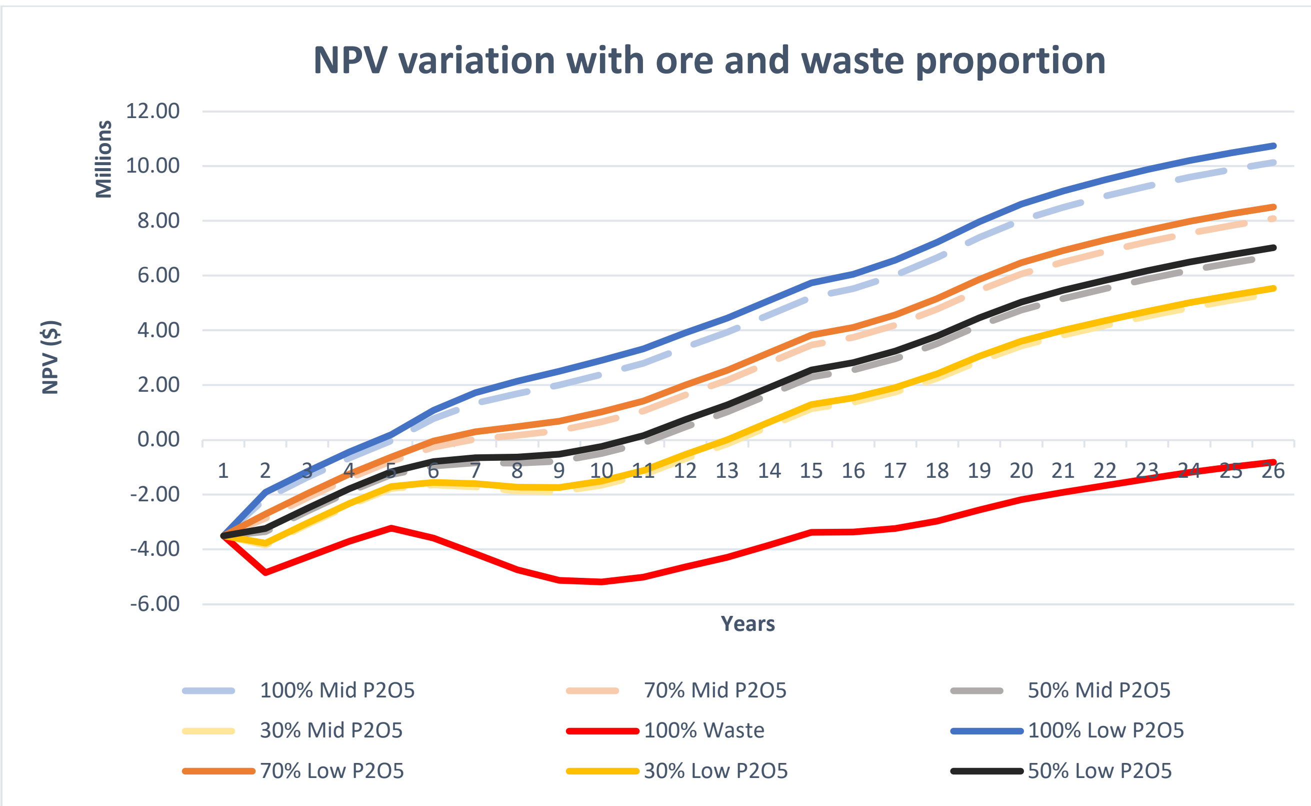


Figure 38: Comparison between the NPV variation when considering the classification of unknown material as ore or waste.

6.5. Sensitivity analysis

Several variables may affect the outcome of the project, and their behavior cannot be predicted with certainty for the entire mine life. Consequently, it is necessary to evaluate the uncertainty in the input variables of the project by performing a sensitivity analysis to determine what would happen with the NPV if any of them change. This analysis allows the company to be aware of how possible changes in the project can benefit or affect the overall result; besides, it allows to know under which conditions the project may be at risk.

The sensitivity analysis for the project evaluation is made for price, cost and extraction rate variations, fine material production, and aggregate mix sales (byproduct). Figure 39 shows the results obtained. As can be seen, the economic results of the extraction project are highly sensitive to variation in the cost and price set for the stone. On the contrary, the changes in production or sale of byproducts seem to affect a lower proportion of the project outcome.

It is essential to understand that as the stone is used for self-consumption and further production of lime, therefore, the limit price at which the plant considers the project as a good option to invest is the cost they are paying to other suppliers to obtain the raw stone. The leading company objective is to maximize and ensure the availability of stone for the plant operation; hence selling the extracted stone to other interested industries is not considered.

In figure 39, a decrease in the extraction rate generates a significant impact on the project since it can cause a breach of contracts and loss of clients. For this reason, if an issue occurs in the extraction, the missing material needs to be covered by getting it from the local market at a very high price. Contrary, if the extraction rate is increased, the impact is low because the lime plant already has an installed capacity, and the exceeding material must be stored or left for the future.

Even though the positive impact of a possible market price raise is showed, it would only represent a favorable situation for the plant if it is decided to sell the surplus product generated in the mine. Meanwhile, the variable with the highest impact over the project results is the extraction cost. A raise in the mining and crushing cost can make buying the stone to other mines a better option since the own reserves can be kept for more favorable conditions.

An increase in fines sales can be very beneficial to the project since it can provide an extra outcome from byproducts of the process and diminish the waste sent to waste dump. A similar effect has fine-grained material production; even though the economic effect is not as significant compared with other variables, it represents a significant loss for the company since part of the reserves can not be used for lime production. In addition, it increases the space need for both temporary storage and waste dump. Any effort looking towards reducing fine material produced and valorizing it after it leaves the process is important to obtain better results.

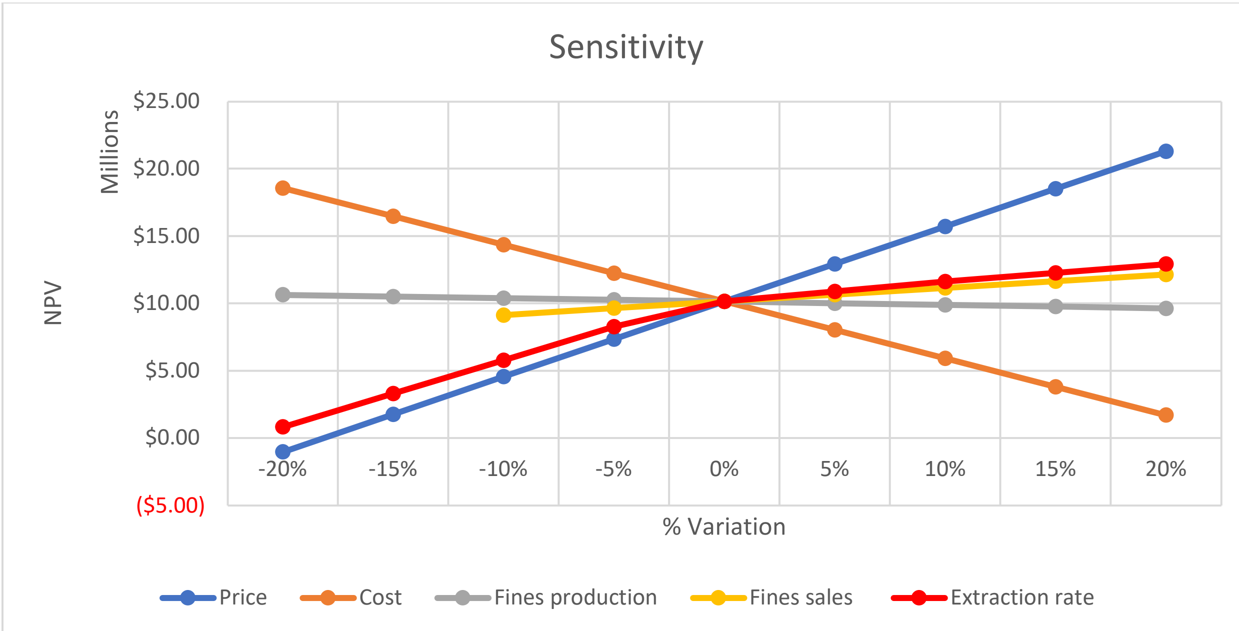


Figure 39:Sensitivity analysis

7. Discussion and Summary

7.1. Exploration and geological mapping

The exploratory work made in the Santiago deposit currently under extraction by Calgov has shown potential for the extraction of limestone with low P2O5 content, which is the raw material to produce lime used for the steel manufacturing industry and which is currently one of the core products for Calgov plant.

The results obtained are based on the information from 10 core drillings in which 8 geological units were identified. These units have different geochemical compositions and were classified following the use they can have in lime production. Out of the 7 units have the required composition for lime production, 4 suitable to produce high-quality lime with lowP2O5 content, and the other 3 useful to produce lime with quality to be used in the production of lime with different applications. Only one unit was classified as waste due to its composition.

A fault zone was identified in the limit of the north area, located at the intersection where the area of interest for this work begins. It has not been studied, and the material in this zone is classified as unknown until further studies help to determine the composition and the effect of it on the project.

7.2. Data analysis, block modeling, and resources estimation

In the statistical analysis of the deposit, it was found that it has a low variability inside the domains. This geostatistical study was made using the drilling information. Some of the domains were intersected only by one or two of the drillings, and therefore, the information is not enough to have an adequate level of certainty of their spatial extension and chemical behavior.

To establish the possible extraction of the limestone, an initial reserve estimation needs to be made. Using the information collected in the exploration and data analysis, a block model is built. It is the first version of the southern area block model and needs to be updated once further information is gathered. It is built following the same units defined in the geological map of the area.

The resources and reserves estimation also provide an initial approach to the possible extraction. The resources are taken as all the ore (both with low P2O5 content and mid P2O5 content) that lie inside the owed area and between 590 m.a.s.l. and 745 m.a.s.l. Reserves have been defined as all the useful material inside the proposed pit. These resources and reserves estimations are initial estimations made following space restrictions. They are not based on the classification criteria set by an expert or defined under the market and economic evaluation of the project and therefore are only indications of the possible amount found and extractable. Further definitions and classification criteria must be set to define the total reserves accurately.

7.3. Pit Design and sequencing

The pit design has been made based on a first geomechanical estimation of the rock mass quality on the southern area stability and in observations of structures in the natural outcrops in this area. A final pit wall has been defined to keep general slope stability, any particular or zonal instabilities had been studied, and further geomechanics work needs to be done to have more details when considering the operation stability.

The berms are designed with a maximum angle of 75 and an overall slope angle of 55. The bench height is 20m, and Two intermediate levels have been reduced in height to 10m with an increase in the final berm width to reach the desired angle and maintain stability. The south area is considered an extension of the current extraction, and therefore it is required to perform studies on the stability of the fault zone at the intersection of both areas.

The road design is made for dumper transit up to bench 3 and only for backhoe excavator transit to the upper levels. The space constraints for the pit and road design are strict, and the use of the land owned by the company must be optimized. Both pit and road are inside the mentioned perimeter. The roads must have a maximum slope of 15% following the legislation set by the Spanish mining authorities.

Different extraction directions have been considered in the sequencing process, but only one is chosen for being the one that allows reaching the annual desired kiln feed. The area is fractioned into two elevation zones in which direct loading will be performed, bench 3 and the bottom pit. The extraction begins with the extraction in level 3 to produce a platform big enough to ensure safe operation, and only after it is reached the extractions of material from levels below this bench 3 starts.

The extraction sequence will advance to the south, and the extraction of low P_2O_5 as target ore is approximately constant over the LOM. the extraction of limestone with mid P_2O_5 content and waste varies over time.

The LOM to extract all the reserves is 36 years; however, a decision was made to perform the extraction only for 25 years due to economic reasons.

7.4. Equipment and plant requirements

The southern area extraction requires the modification of the crushing and processing plant to reach the target production to feed both Kilns; the current crushing rate of 200 t/h must be duplicated, and the current installation is not suitable for this purpose. Even though the crusher installed has enough capacity to carry out the work, the conveyor and screening system afterward is undersized and poorly distributed, setting a bottleneck to reach the goals.

Two options for the plant modification are possible, the first one is redesigning and installing new conveyors and screens with bigger capacities, and the second one is considering the investment in a mobile crusher. Both options require careful technical and economic evaluation before decision making.

New mine equipment is required to perform the southern extraction with the investment needed for 2 trucks and one wheel loader.

7.5. Waste management

Two types of waste will be produced during the south area extraction; the first one is stone that does not meet the chemical composition required and need to be taken from the mine directly to a waste dump. It is coming from geological unit 6, which has high P_2O_5, Al_2O_3, and FE_2O_3. It counts for 2.5Mt.

The second type is material that, after passing through the extraction, crushing, and screening process, is too fine-grained to be used in the Kiln. It gets out of the process after the screening phase and corresponds to 36% of the total stone feed to the crusher. It is estimated to be in total 4.8Mt in the entire LOM.

There is no existing waste dump, and there is no space inside the owned land for waste storage. The fine-grained material encloses sizes between 0 to 35 mm approximately, and further classification processes can be made to get commercial aggregate and valorize this portion of the produced stone and reduce the required space for waste storage.

7.6. Economic evaluation

The economic evaluation of the project provides a first estimation of the expected results. It explores different scenarios the extraction may face. Since the project still has many uncertainties and technical and logistic decisions need to be made, it can change, and the results may differ in reality.

The evaluation of expected financial results and scenarios started with the possible cost study; the CAPEX is established, estimating the equipment acquisition cost, plant modification cost, permits required, land preparation, and land acquisition cost for the waste dump facilities. An initial investment enclosing these items is estimated to be approximately $ 3.5 million.

The operational cost, OPEX, is a projection of the possible costs based on the study of the past 3-year operational cost of the mine currently under extraction with a cost per ton processed of $5.14. The waste handling cost is estimated based on the approximate cost of transport for the waste based on the fees currently charged for transporting from plant to temporary waste dump places. The Cost of waste handling is 4.4 $/t.

The revenues are calculated based on the market prices for limestone in Andalucia province and the aggregates price set by the construction industry in Spain. They are 8.2$/t of high-quality stone, 7.6 $/t of mid-quality stone, and 6.8$/t for the aggregates mix.

Using the cost and price information, the Project NPV is estimated to be approximately 10 Million in the 25 years of operation. The discount rate to perform the NPV estimation is 3.6%. The NPV is highly dependent on the initial investment decision, the destination of the unknown stone, and the extraction cost variation.

An evaluation of financial scenarios is made for the initial investment and the unknown material classification to avoid overestimation. For the initial investment, a worst-case in which a rise of 20% over the estimated investment is evaluated a result of an NPV of $ 9 million is reached, and the project starts providing earnings higher than the discount rate only after 7.5 years; it means 1 Million less when compared with the best case evaluated and 1.5 years more to start getting a positive NPV.

The scenario evaluation for the unknown material destination and its impact over the project was made, assigning it with fractions of waste and ore and evaluating both results for Low P2O5 been the ore and Mid P2O5 been the ore. The obtained result shows that the project is only attractive for investment if 70% of the unknown material is ore (either Low P2O5 or Mid P2O5).

To finish the economic evaluation of the project, a sensitivity analysis is made to see which conditions have a higher effect on the expected results. Since 100% of the extraction is for own consumption and stone is not considered to be sold to other companies, the main variables affecting the project results are the rise in operation cost and the decrease of the extraction rate. A raise in the kiln feed cost can put the project in the position of nonbeing economically viable, and a decrease in the extraction rate can place the plant is a risk position of not complying with the customers requirements or under the risk of placing the production in the dependence of outside stone providers.

8. Recommendations and Conclusions

8.1. Conclusions

This work aimed to define a plan for the extraction of the limestone reserves of the southern area of the Santiago Deposit. It was made focusing mainly on keeping the extraction of limestone with Low P2O5 constant to ensure the continuity in the lime plant operation.

Based on the geologic map and the information obtained in the drilling campaigns, the southern area of the Santiago deposit is suitable for extracting the required amount of low P2O5 limestone to feed the kiln H04 to ensure compliance with the current plant customers. It has, in addition, some reserves of Mid P2O5 that can be used to feed the kiln H03.

A mine design was made, considering the available space, pit, and road lie inside the company's land. A total of 9 benches will be extracted, leaving berms of 10 m during operation and 5 to 9 m at the final pit wall with an overall slope angle of 75%. In total, 900 mt of the road must be constructed to perform the extraction.

The extraction plan considers the extraction of 6.9 Million tons of LOW P2O5 limestone and 2.4 Million tons of Mid P2O5 limestone and is planned to be carried on for 25 years. Even though the available resources allow continuing the extraction of 11 years more, so for 36 years in total, the investment needed to replace the equipment is high, and 87% of the total possible NPV is reached until year 25.

The mine scheduling process has shown a constant feed of low P2O5 limestone throughout the life of mine, and a variable extraction of Mid P2O5 stone, waste, and unknown material; The advance of the mine is going to be towards the south and is planned to start at the limit with the current north pit under extraction.

A total of 5.6 million tons of waste will be generated during the mine life. Out of them, 3.8 million tons are fine aggregates, and 1.8 million tons are waste (stone with a composition not suitable for lime production). Under the current conditions, 36% of the material is lost as fine aggregates and sales of only 10% of it; the space need for the waste dump is 35 hectares.

The economic evaluation of the extraction shows a good result for the project with a total NPV of $10 million under the current market conditions for the lime industry. The initial investment for the project is high, with approximately $4 million needed for the equipment and the modification of the crusher and screening plant.

The project is highly sensitive to the unknown material classification. If it has a portion higher than 30% waste, its extraction and storage costs as waste are high, and the project will not be attractive. It is also highly sensitive to the cost increase and the extraction rate decrease. Changes in these two variables can significantly vary the outcome and turn over the project to be economically unfeasible.

The main problems the project can have are related to the uncertainty of the reserves and block model; for some of the units, the quantities of exploration data is low; in addition to it, the unknown material may represent a challenge, and no information has been collected from it yet. Operationally the space limitations are high, and the uncertainty over the waste dump location and design may change significatively the cost and the financial results.

8.2. Recommendations

In the development of this work, several gaps have been found regarding the amount of information available and the possible outcomes of mine extraction in the south area. It is important to work further in the obtention of more information for technical and economic evaluation. Some of the key areas in which further research is needed previously of a final decision are mentioned below.

- It is necessary to complement the exploration information and perform further studies on the composition of the deposit. Currently, the extraction performed in the north area is supported only by the historical data of the material extracted. There is no exploration drilling in this area, having a complete uncertainty on the deposit below the current benches.

  Furthermore, it is necessary to increase the data for the southern extension area. Even though some information is based on the 10 exploratory drillings made in the past years, it is not sufficient since some units like Q1 and Q2 are intersected only once and have a high uncertainty over their geochemical composition.

  Until now, the information gathered allows to develop an initial geologic map and block model to start evaluating a future extraction, but more information is needed to reduce the uncertainty of the current models. The fault zone has not been studied at all; deeper drillings are needed to define the behavior of the units near the base pit.

- Even though the rock mass has a generally good quality and the need for reinforcement is low, a more detailed geomechanical study needs to be done for both the north pit and south extension project to establish the possible zones in which wedges or failures can be developed in the daily operation. It is also necessary to define a slope monitoring and control system focusing on the fault area.

- Hydrogeology studies need to be performed to determine the water base level and define the possibilities of making a deeper pit and recover reserves below the current pit bottom. There is any legal or environmental issue that defines the base level for the pit so that it will be only dependent on the water base level.

- It is essential to complement the existent block model with the historical information of the production drilling to contrast and complement the result obtained in the exploration. Currently, the drilling dust samples are collected and analyzed, but the information is not linked with the extraction r the mining plan.

- It is necessary to evaluate the cost of building the roads outside the owned land by agreeing with the neighbors. The space availability is low, and the roads are built on top of the material of possible interest causing the reserves to be reduced. The possibility of doing the load and transport from the benches may be a good option to reduce the fines fraction and increase the flexibility in the mining operations.

- Currently, 36% of the total extracted reserves are lost in fine-grained materials and set to temporary waste facilities. Finding alternatives for the use of this portion of the mine production, either by using it in the process or selling it for the aggregates/construction industry, can help to reduce environmental and space problems as well as obtain better financial results.

- It is necessary to analyze the mining method used and selected for the south extension area. Eliminating the throwing activity can help to reduce the fines material and have a more efficient extraction. Also, the loading and transport operations can have more flexibility for the production if they go directly into the blasted materials.

- Further studies need to be done to define the right design of the waste dumps according to their location and the stone handling process.

- Further studies are needed to establish the right drilling and blasting patterns to reduce the fine material production. These studies should be done separately to north and south zones since the bedding orientation is different and the extraction advance sequence is also different

- Blending materials may increase the resources and get the maximum advantage of the low P2O5 limestone. Due to space limitations and the existence of a kiln that can use the Mid P2O5 stone, this scenario was disregarded. Despite this, it may be useful to evaluate this possibility since it can provide more value to the reserves.

The project still has many uncertainties in the block model quality environmental impact assessment, waste dumps location and design, land reclamation, and process optimization that were out of the scope but need to be considered and carefully studied. A new economic evaluation should be made once the technical studies are complete to get a more accurate result over the expected NPV and project development.

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Annex 1 RMR Table classification

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence) Rating	< 1 m 6	1 - 3 m 4	3 - 10 m 2	10 - 20 m 1	> 20 m 0
Separation (aperture) Rating	None 6	< 0.1 mm 5	0.1 - 1.0 mm 4	1 - 5 mm 1	> 5 mm 0
Roughness Rating	Very rough 6	Rough 5	Slightly rough 3	Smooth 1	Slickensided 0
Infilling (gouge) Rating	None 6	Hard filling < 5 mm 4	Hard filling > 5 mm 2	Soft filling < 5 mm 2	Soft filling > 5 mm 0
Weathering Ratings	Unweathered 6	Slightly weathered 5	Moderately weathered 3	Highly weathered 1	Decomposed 0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

Strike perpendicular to tunnel axis		Strike parallel to tunnel axis	
Drive with dip - Dip 45 - 90°	Drive with dip - Dip 20 - 45°	Dip 45 - 90°	Dip 20 - 45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip - Dip 45-90°	Drive against dip - Dip 20-45°	Dip 0-20 - Irrespective of strike°	
Fair	Unfavourable	Fair	

* Some conditions are mutually exclusive . For example, if infilling is present, the roughness of the surface will be overshadowed by the influence of the gouge. In such cases use A.4 directly.
** Modified after Wickham et al (1972).

Annex 2 Quinquennial extraction of stone

Reserves extracted inside extension pit				
Years	**Low_P2O5 (Mt)**	**Med_P2O5 (Mt)**	**Unknown (Mt)**	**Waste (Mt)**
1-5	1.347	0.002	0.554	0.080
5-10	1.374	0.317	0.453	0.714
10-15	1.374	1.039	0.027	0.505
15-20	1.373	0.955	0.203	0.252
20-25	1.375	0.028	0.057	0.123
25-30	1.375	0.402	0.021	0.284
30-36	1.526	0.923	0.000	0.469

Annex 3 Roads design images

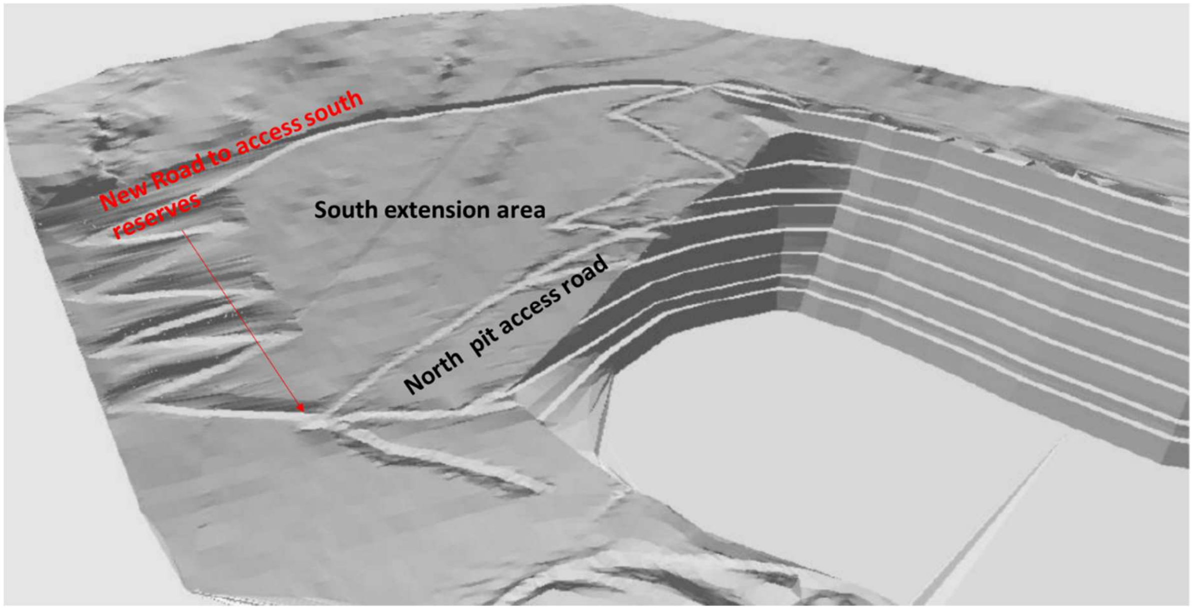

Figure 40: Current and the new road used for the mine extraction

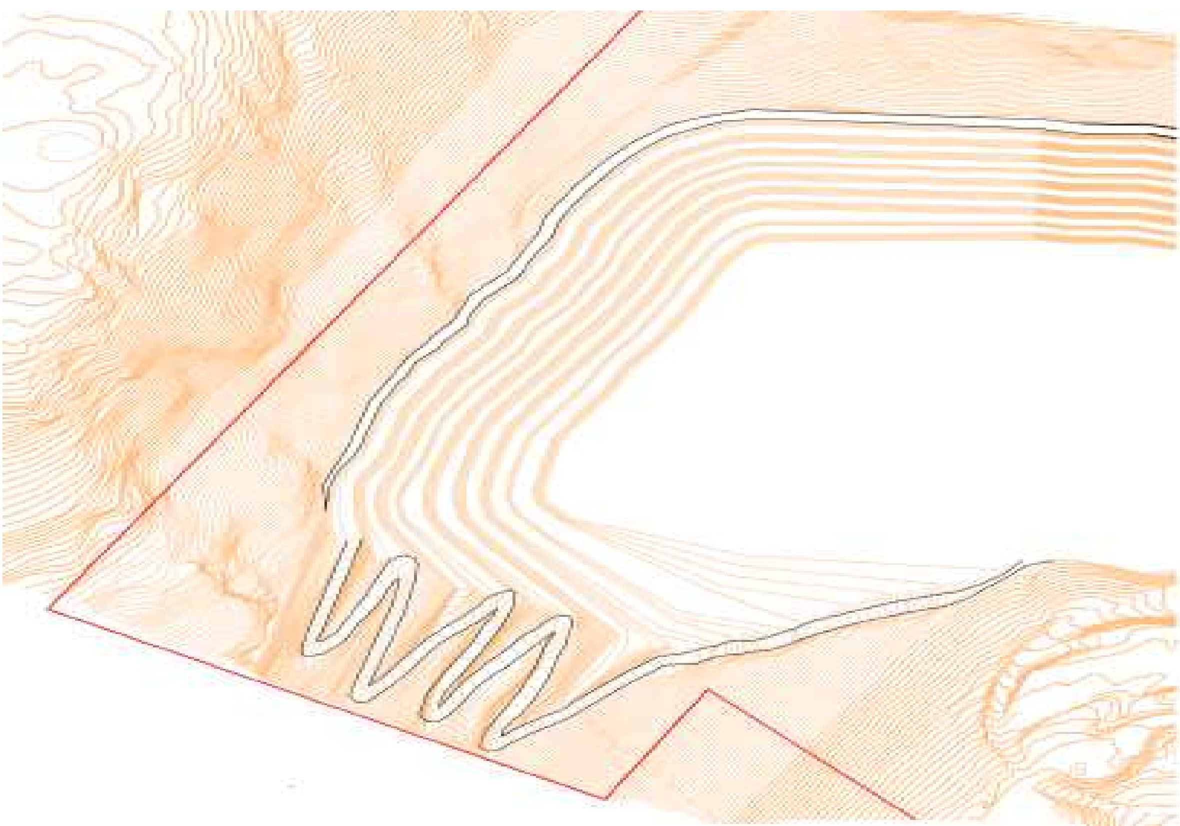

Figure 41: road for the south extraction.

Annex 4 Pit layout

Current pit under extraction (north area)

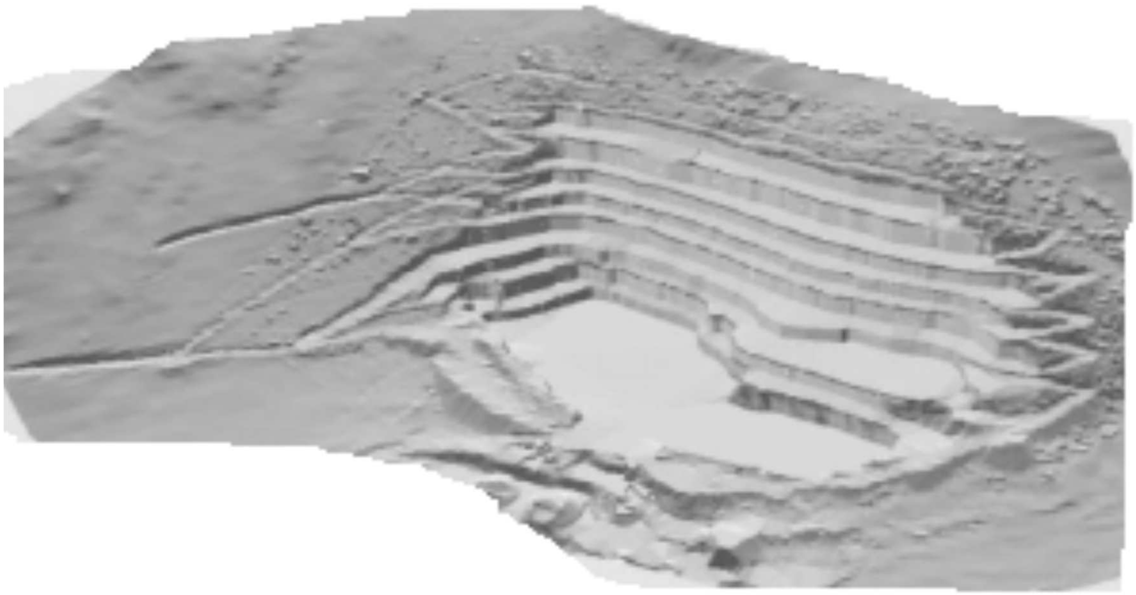

Figure 42: Actual pit

Final approved pit for the north area

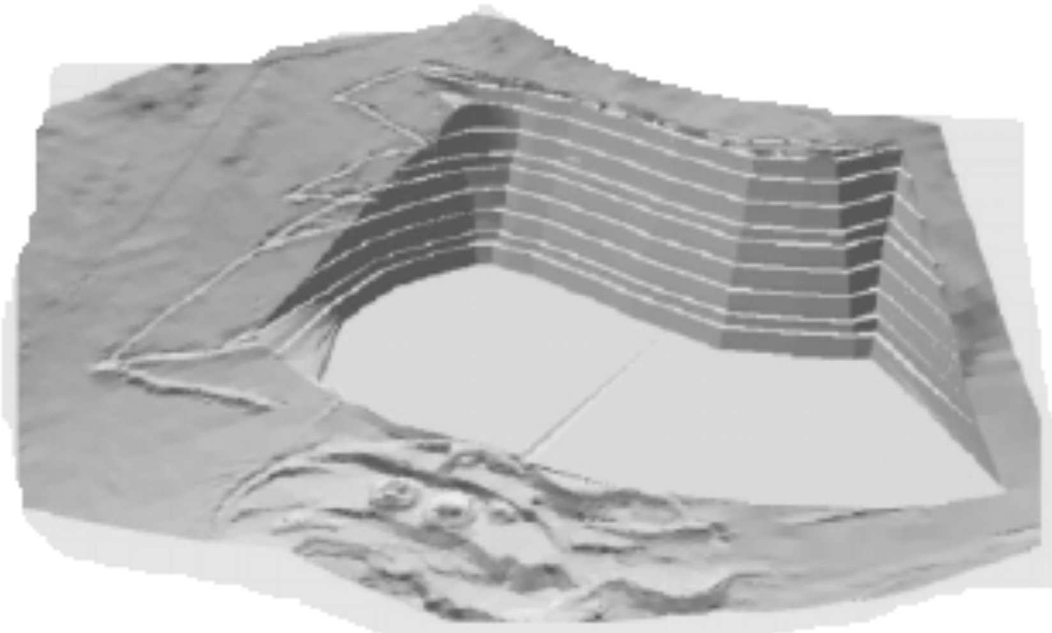

Figure 43: Final pit for the north area. Already approved by the mining authorities

Final project view: north pit + south extension area

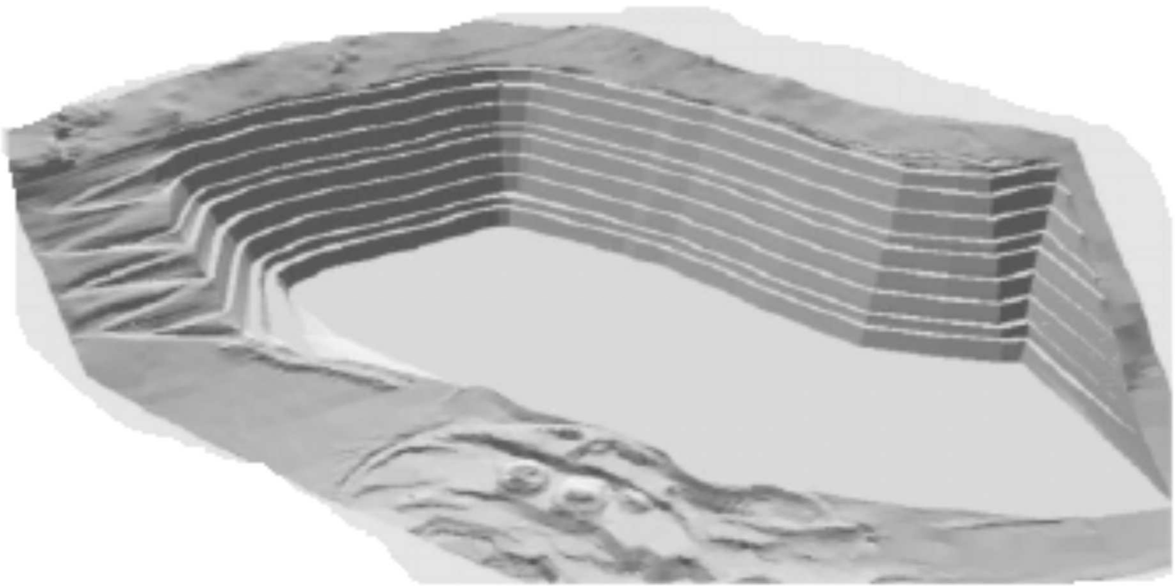

Figure 44: Ultimate pit including both north area and south area