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COLLEGE OF GEOLOGICAL, MINING AND METALLURGICAL ENGINEERING

"Design of a Tunnel Reinforcement System Based on the Three-Dimensional Rock Mass Model of ASNV6 Vein in *Caylloma* Mining Center"

COURSE:

GEOLOGY APPLIED TO CONSTRUCTIONS

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Abstract

Safety at work is the most important factor that must be taken into account in a work environment, in the specific case of mining, tunneling works require fairly precise and concrete considerations when building a tunnel or analyzing the type of support it requires to be stable.

Within this problem, this project emerged, with the aim of demonstrating that it is possible to make a geomechanical model (lithological, statistical and structural) through different software (Leapfrog, Dips, Unwedge, etc.) starting from a previous database, since either to build a mine environment or to improve its conditions.

This objective was met through the work methodology that was used, which consisted in starting with the creation of a lithological model that allows us to identify the different types of material that are in place, with this lithological model it was possible to build a numerical model that helped to estimate the values of each geotechnical parameter of the RMR and find a value of Q, a structural analysis was also carried out to determine families of discontinuities that could cause the formation of wedges in the tunnels and a possible collapse, finally this project analyzed 3 tunnel sections (validation of the methodology) and recommend adequate support according to the in-situ conditions.

Introduction

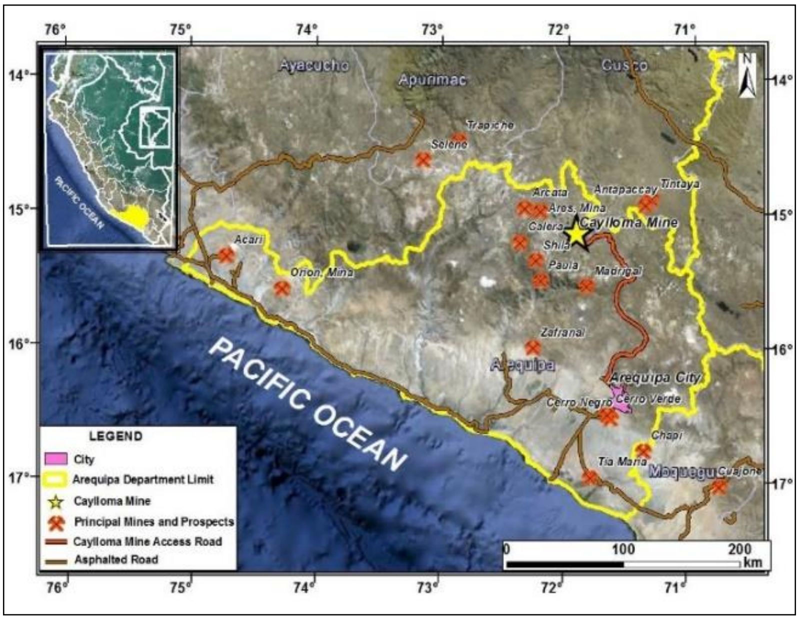
1. DESCRIPTION OF THE PROJECT

1.1 Property Description and Location

The Caylloma Mine is located in the Caylloma District, 225 road-kilometers north-northwest of Arequipa, Peru (Figure 1). The property is 14 km northwest of the town of Caylloma at the UTM grid location of 8192263E, 8321387N, (WGS84, UTM Zone 19S). The location of the mine is shown in Figure 1.

Figure 1 Map showing the location of the Caylloma Mine

Figure 1 Location of the Caylloma Mine



1.2 History

The earliest documented mining activity in the Caylloma District dates back to that of Spanish miners in 1620. English miners carried out activities in the late 1800s and early 1900s. Numerous companies have been involved in mining the district of Caylloma but limited records are available to detail these activities.

The Caylloma Mine was acquired by Compania Minera Arcata S.A. (CMA), a wholly owned subsidiary of Hochschild Mining plc in 1981. Fortuna acquired the mine from CMA in 2005.

Production prior to 2005 came primarily from the San Cristobal vein, as well as from the Bateas, Santa Catalina and the northern silver veins (including Paralela, San Pedro, and San Carlos) with production focused on silver ores and no payable credits for base metals.

1.3 Geology and mineralization

The mine is within the historical mining district of Caylloma, northwest of the Caylloma caldera complex and southwest of the Chonta caldera complex. Host rocks at the Caylloma Mine are volcanic in nature, belonging to the Tacaza Group. Mineralization is in the form of low to intermediate sulfidation epithermal vein systems.

Epithermal veins at the Caylloma Mine are characterized by minerals such as pyrite, sphalerite, galena, chalcopyrite, marcasite, native gold, stibnite, argentopyrite, and silver- bearing sulfosalts (tetrahedrite, polybasite, pyrargyrite, stephanite, stromeyerite, jalpite, miargyrite and bournonite). These are accompanied by gangue minerals, such as quartz, rhodonite, rhodochrosite, johannsenite (manganese-pyroxene) and calcite.

Underground operations are presently focused on mining the Animas ASNV6 and Animas NE veins.

1.4 Resistance parameters

The resistance of the rock massifs is related to the resistance of the matrix rocky and its discontinuities, being very variable; in addition to the conditions geoenvironmental that subject the rock massif. The presence of tectonized areas, altered or of different lithological composition, they represent areas of weakness and anisotropy with various behaviors and resistant attributes. These circumstances determine a great complexity in the evaluation of the resistance of the massifs Rocky.

The resistance can be assessed in terms of the maximum effort it can withstand for certain conditions and in terms of their resistant properties, c and ϕ, parameters that are usually needed for the project calculations of the engineering works.

According to the degree of fracturing of the rock mass, its behavior and properties resistant will be controlled by:

- The resistance of the rock matrix (isotropic or anisotropic).
- The cut resistance of a family of discontinuities.
- The cut resistance of 2 or 3 families of discontinuities (provided they are representative in the massif).
- The overall resistance of a rocky block system with behavior isotropic

In surface and underground excavations, both mass excavation works how stability and mechanical behavior problems are directly related to the strength of the material and the presence of discontinuities. The evaluation of the resistance of the rock matrix or a discontinuity can be carried out with laboratory tests or in situ. The dimensions and conditions Rock massifs cannot be reproduced in the laboratory, and their resistance must be evaluated by indirect methods.

Subsequently to establish the means that control the resistance of the rock mass (either one or more families of discontinuities, the rock matrix, an unusual sector of weakness, etc.) its evaluation can be carried out through the following procedures:

- Empirical methods based on experiences and laboratory tests.
- Indirect methods based on quality indices (classifications geomechanics).
- Mathematical modeling and subsequent analysis.
- Physical modeling.

Breaking or resistance criteria form the basis of empirical methods, and allow to evaluate them, determining the resistance of the rock massifs from of the acting forces and the properties of the rock material, providing:

- The response of the rock intact to various stress conditions.
- The prediction of the influence of discontinuities on behavior of the massif.
- The prediction of the overall behavior of a rock mass.

Quality indices defined by geomechanical classifications allow estimate resistance approximately by establishing correlations between classes of rock and resistant parameters c and ϕ of the rock mass.

Mathematical models allow estimating resistance from modeling numerical of the behavior of the massif, of its physical and mechanical properties, of the law of behavior and influencing factors (tensions, water). These models present their maximum utility in performing a posteriori or back analysis, which consist of numerically modeling the deformations and the breaking process of a real rock massif (from knowledge of the characteristics and mechanism of breakage), and thus obtain the resistant parameters corresponding to breakage or a certain level of deformations of the massif.

The physical models consist of build scale models with different natural or artificial materials (for example, with plaster paste elements, blocks of rigid material, mixtures of sand and clay and binding elements, etc.), and subject them to loads to observe their behavior. The aforementioned methods allow obtaining, more or less approximate, the resistance of the rock massifs, depending on the information and The available data. Empirical criteria and mathematical modeling based on a posteriori analysis are those that provide more values representative, the determination of the characteristic resistant parameters of the rock massifs, c and ϕ, is the most conflicting point. Of the procedures cited, only mathematical and physical models consider behavior deformation of the massifs.

1.4.1 Water conditions

The presence of infiltration water in the faults comes from wet to wet and occasionally in specific sectors it is exposed as a drip, causing instability in contact with the mineralized structure.

1.5 Exploration, drilling, and sampling

Since Fortuna took ownership of the property in 2005 the principal exploration conducted at the deposit has been surface and underground drilling, to explore the numerous vein structures identified through surface mapping or geophysical surveys conducted by Bateas, or for infill purposes to increase the confidence level of the Mineral Resource estimates.

As of August 31, 2018, Bateas had completed 1,296 drill holes on the Caylloma Mine totaling 225,361.80 m since the company took ownership in 2005.

Bateas has used a number of different drilling contractors to carry out exploration and definition drilling since it took ownership of the mine in 2005. Both HQ (63.5 mm) and NQ (47.6 mm) diameter core were obtained, depending on the depth of the hole. Ground conditions are generally good with core recovery averaging 94 %.

Proposed surface drill hole collar coordinates, azimuths and inclinations were designed based on the known orientation of the veins and the planned depth of vein intersection using geological plan maps and sections as a guide. Once the coordinates have been determined, the location of the collar is located in the field using differential global positioning system (GPS) instruments. The drill pad is then prepared at this marked location. Upon completion of the drill hole, a survey of the collar is performed using Total Station equipment, with results reported in the collar coordinates using reference Datum WGS84, UTM Zone 19S.

The geologist in charge of drilling is responsible for orienting the azimuth and inclination of the hole at the collar using a compass clinometer. Downhole surveys are completed by the drilling contractor using survey equipment such as a Flexit or Reflex tool at approximately 50 m intervals for all surface drill holes and for underground drill holes greater than 100 m in length. Bateas assesses the downhole survey measurements as a component of the data validation.

Drill holes are typically drilled on sections spaced 40 to 60 m apart along the strike of the vein with surface drilling focusing on exploring the extents of the Animas.

The relationship between the sample intercept lengths and the true width of the mineralization varies in relation to the intersect angle between the steeply-dipping zone of mineralized veins and the inclined nature of the diamond core holes.

Geotechnical logging is conducted prior to cutting of the core and involves the collection of drill core recovery, rock-quality designation (RQD) and others RMR’s parameters data. Information is recorded in the field using the Maxwell LogChief application.

Bulk density samples have been primarily sourced from drill core with a limited number being sampled from underground workings. Bulk density measurements are performed at the ALS Global Laboratory in Lima using the OA-GRA09A methodology.

1.5 Data verification

Bateas staff follow a stringent set of procedures for data storage and validation, performing verification of data on a monthly basis. The operation employs a Database Administrator who is responsible for overseeing data entry, verification and database maintenance. A separate Database Auditor is responsible for performing a detailed independent review of the database on a quarterly basis and submitting a report to Fortuna management detailing the findings. Any issues identified are immediately resolved by the administrator.

1.6 Mineral Resources and mining methods

The 2018 Mineral Resource update has relied on channel and drill hole sample information obtained by Bateas since 2005. Mineralized domains identifying potentially economically extractable material were modeled for each vein and used to code drill holes and channel samples for geostatistical analysis, block modeling and grade interpolation by ordinary kriging or inverse distance weighting.

Net smelter return (NSR) values for each mining block take into account expected commercial terms, the average metallurgical recovery, the average grade in concentrate and long term projected metal prices. Mineral Resources take into account operational costs and have been reported above a US$ 50/t NSR cut-off value for veins wider than two meters and amenable to extraction by semi-mechanized mining methods (Animas, Animas NE, Nancy, and San Cristobal veins); or above a US$ 135/t NSR cut-off value for veins narrower than two meters regarded as amenable to conventional mining methods (all other veins).

The mining method employed at the Caylloma Mine is cut-and-fill which is commonly used in the mining of steeply-dipping orebodies in stable rock masses. Cut-and-fill is a bottom up mining method that consists of removing ore in horizontal slices, starting from a bottom undercut and advancing upwards. The operation bases its mining plan on a mix of mechanized, semi-mechanized, and conventional extraction methods based on vein width and rock quality.

1.7 Sismicity

There are two major branches of seismic protection technologies. One corresponds to seismic isolation, which consists of incorporating a flexible interface between the building and the foundation floor, through the use of seismic insulators. These are elements that allow decoupling the movement of the building from the movement of the ground, considerably reducing the seismic demand on the structure. They act as a "filter" that reduces the effect that the soil transmits to the building during an earthquake.

The other branch corresponds to the dissipation of energy. In this case the energy dissipators, which are similar to the shock absorbers in a car, allow to reduce the vibrations in a building product of an earthquake, capturing part of the energy that the earthquake introduces to the structures and transforming it into other forms of energy (heat, etc.) These heatsinks are distributed in the body of the building, typically in shafts resistant to seismic forces.

A fracturing process in a rock mass within the mining activity involves a dynamic disturbance, which induces rebalancing mechanisms that give way to deformation processes after overcome a certain threshold of resistance within the rock. These rebalancing processes generate openness of pre-existing structures and / or failures, that is, breaks in the massif from which a certain amount of energy is released and transmitted in the form of elastic waves that propagate through the medium; these waves are captured by the sensors that make up a monitoring network. In mining, the study and understanding of the behavior of the disturbed massif is of vital importance, since its adequate control is transformed into a tool that allows monitoring and continuity of the mining production process, in other words, provides an operation safe for both personnel and the machinery and infrastructure involved For the present study, static conditions were considered.

1.8 Erosion control

Many of the techniques used to stabilize the slopes and prevent landslides, also related to morphological adaptation, serve to alleviate the erosive problem. In any case, some specific measures must be applied to correct the erosion of the surfaces to be revegeted.

This type of measures is aimed at the stabilization of slopes, including surface remodeling movements, drainage treatments and additional surface protection to the vegetation cover, if deemed convenient. The suitability and design of these works depends on the hardness of the substrate and the final slope of the surfaces.
Among the constructive measures against erosion to consider, are the creation of small terraces and terraces in the areas of greatest risk of erosion as a possible way to reduce the slopes and the length of decline, curbing surface runoff.

As for drainage, care should be taken that they are not an element of aggression for soil stability. The slopes of disassembly or excavation will be more susceptible to erosion in the lower area of the decline while the debris and fillings will erode more easily in the coronation. In addition to the ditches at the base of the excavation slopes, it is important that the headwaters, embankments and landfills have a guard gutter.

The drains must be channeled to the natural channels or, failing that, to the foot of the embankments, but, protecting the drain point, with a cobblestone based on gravels or gravels, in order to absorb and disperse the energy of the discharge stream.

As a measure of correction prior to revegetation, it is necessary to mention the surface scarified of the land in horizontal lines, to break the small or medium grooves already formed. Another system, when the grooves have acquired a greater dimension, consists in slightly fanning the slope, and arranging plant branches, capable or not of germinating, in small terraces; bundles of branches are attached to stakes that are stuck in the hearth.

The problems derived from the excess of water that produce waterlogging and prevent certain uses, must be corrected with the measures or works of superficial or internal drainage that in each case are convenient.

1.9 Risks

There are several aspects that are coincident in underground mining, especially those risks associated with the following variables:

- Fire
- Rock fall
- Rolling Equipment Traffic
- Explosives handling
- Pique and chimney development

- Compressed air
- Water
- Other

Fire Risks

Causes of fire inside an underground mine are welding work, in motorized vehicles, wood fortification, coal dust, electrical installations, tapes conveyors, garbage accumulation, etc.

These fires produce a lot of toxic fumes, according to the systems of ventilation of the work. Topic to be discussed in depth "Mine Rescue".

Rock fall

The fall of rocks or slabs is due to instability of the land due to the characteristics of the rock around the excavation. Although it also influences the form and excavation dimensions and operational aspects such as over-excavation due to blasting poorly designed The most dangerous operations that require specialized work in Mining and underground excavations of civil works has long been the unleashed and the support as reinforcement measures, to achieve greater security in operations Unitary units of mine exploitation.

In the mines of Europe and North America they have been continuously tested and developed different rock support systems since the early fifties.

Rock fall control has included detailed geological studies, innovations in rock mechanics and mechanics, the introduction of different types of anchor bolts, the testing and development of mechanized equipment for untied and bolted rock, as well as techniques for spraying with cast concrete. The occurrence of accidents in the mines, especially due to the fall of rocks, is not more than the consequence of the gap between the new geomechanical technologies of support and traditional empirical mining that exists in the country, where the production of raw ore about any other approach, including the lives of workers. The highlight of these new innovations is to approach the objective of accidents zero, which results in the cost of operation. As important as the above, is the cost of the quality of support which is negligible versus the opportunity cost they cause accidents, due to the bad work technique and the lack of geomechanical criteria in the exploitation of a mine.

Basically the support aims to:

- The safety of people who remain in the cavity for some reason and the protection of the equipment found there.
- Ensure that the cavity can fulfill the function for which it was excavated.
- The basic structural element is the perforated rock massif.
- This structure should be verified and reinforced eventually taking full advantage of the rock as active material.

In mining, due to size, complexity and relative position, the excavations have particular stability problems, since the miners are busy much of the time in extract minerals, the support of productive excavations must be the main task and not contrary.

To reach a rational, economical and safe solution, it is necessary to consider the global context economic aspect and technical feasibility then we must represent the reality of a technical model that reduces the problem of falling rocks in favor of essential phenomena, it is necessary to define the system, inform us about the scope of the Geomechanics on resistant materials, determine external actions and raise the sizing criteria through security concepts. The main difficulty is the heterogeneity, anisotropy and discontinuity of the mass Rocky Parameters that are not the total domain of the miner.

2. THEORETICAL FRAMEWORK

2.1 Geomechanical model

To obtain the final geomechanical model, the lithological modeling was first performed, which will be explained in this section of the report. Based on the generated lithological model, the geomechanical model was developed, which is one of the objectives of the project to finally make the analysis of the support required for each section through which the tunnels pass. Where the generated block model has dimensions of 5x5x3 m.

2.1.1 3D lithological model

From the geological logging prepared by the engineer Christian León, for the 15 drills that have been used in this project we have performed the lithological modeling in Leapfrog Geo.
Initially there were 19 lithologies of the 34 recognized by the engineer, loaded from the Excel file. Once the drills were loaded, they were selected and summarized in four main groups, which are the following: Toba (TUF), Tobaceous sandstone (TSS), Toba Lapilli (LPT) and andesite flow (ANF) **[1]**.

In the Table 1, taken from the presentation of the engineer León **[1]**, you can see the 34 lithologies described by which for this project they were summarized in 4 main lithologies plus one that is the Vein.

Table 1 Table of the 34 lithologies

Litología			
COL	Coluvial	CG	Conglomerado
TUF	Tufo o toba	SS	Areniscas
LPT	Toba Lapilli	SH	Lutitas
LPS	Roca Lapilli	LST	Limolitas
TBX	Brecha de toba	QTZ	Cuarcita
PBX	Brecha piroclástica	HBX	Brecha hidrotermal
TCG	Conglomerado tobáceo	QV	Venillas de cuarzo
TFB	Brecha tobácea	CV	Venillas de calcita
TSS	Areniscas tobácea	RcV	Venillas de rodocrosita
TST	Limolitas tobácea	RnV	Venillas de rodonita
TMS	Lodolitas tobácea	QRnV	Venillas de cuarzo-rodonita
ANF	Andesita de flujo	QCRnV	Venillas de cuarzo-calcita-rodonita
ANI	Andesita intrusiva	RnQCV	Venillas de rodonita-cuarzo-calcita
RHY	Riolita	QCRcV	Venillas de cuarzo-calcita-rodocrosita
DAC	Dacita	GSV	Venillas de sílice gris
RHD	Riodacita	ZM	Zona mineralizada
VCL	Volcanoclástico	Vt/EST	Veta/Estructura

In the figure 2 the drills entered with the initial 19 lithologies are observed, then in figure 3 We see the drills summarized in 4 lithologies.

Finally, from the drills with the summarized lithologies we use the Geological Models option to create the three-dimensional bodies shown in figures 4, and 5.

Figure 2. Drills in which the 19 lithologies initially charged are observed. ***[1]***

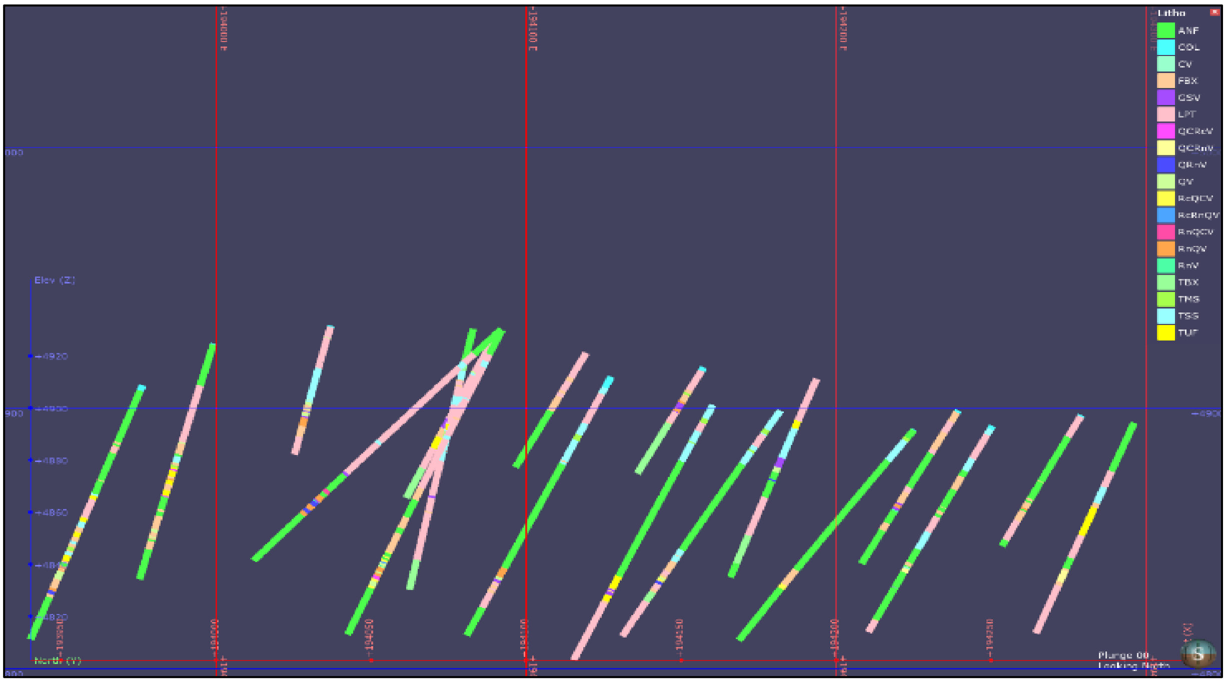


Figure 3. Drills with the lithology summarized at 4 (ANF, TUF, TSS and LPT). ***[1]***

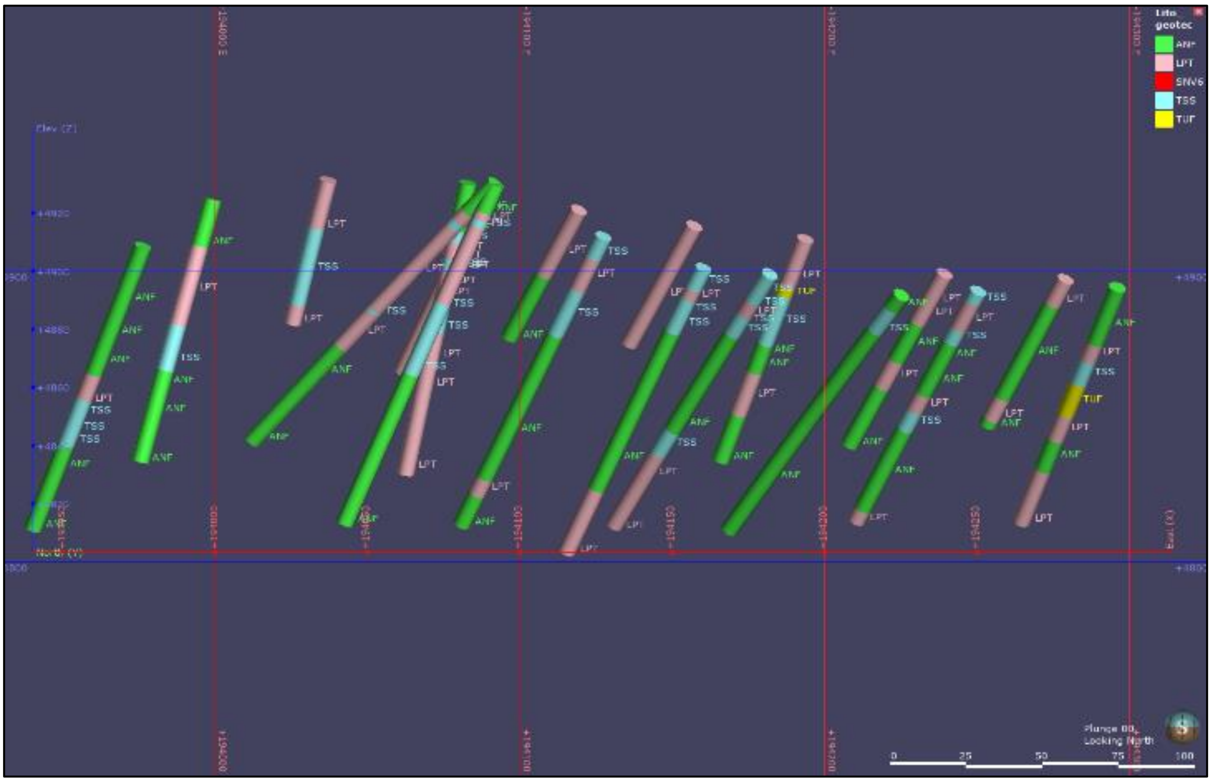


Figure 4. Plan view of the bodies modeled from the drills summarized.

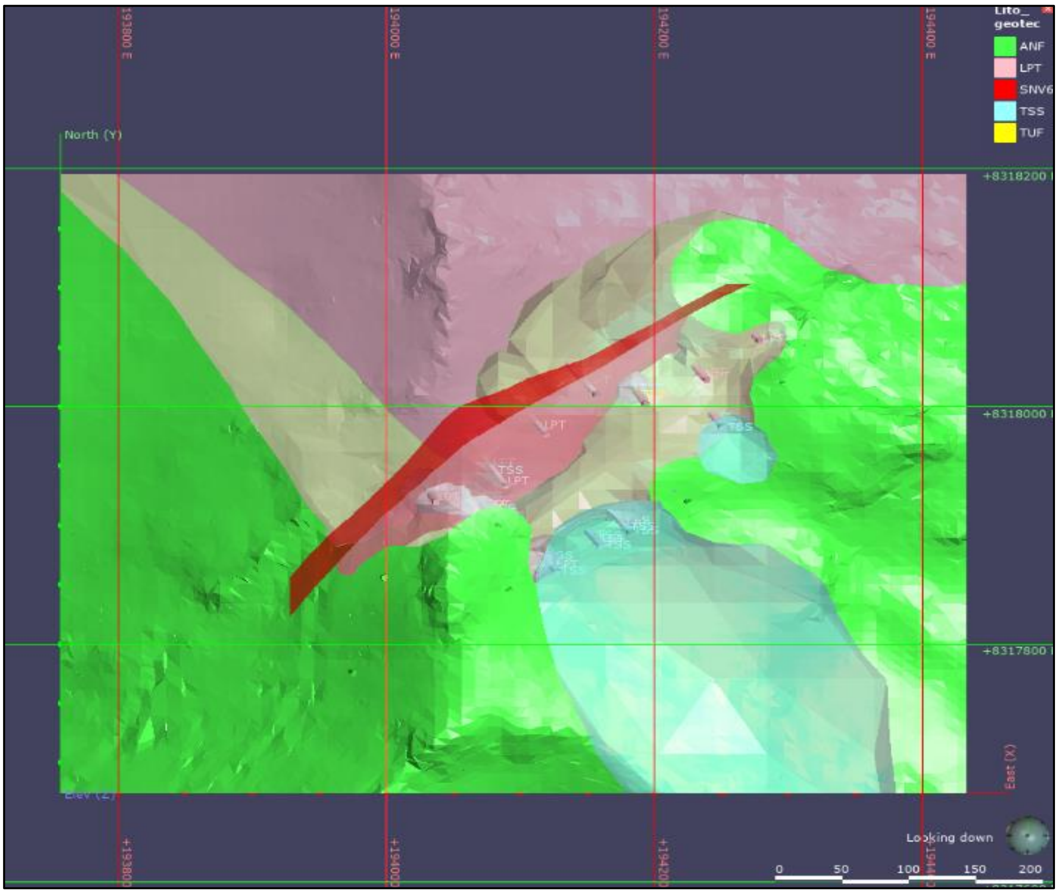
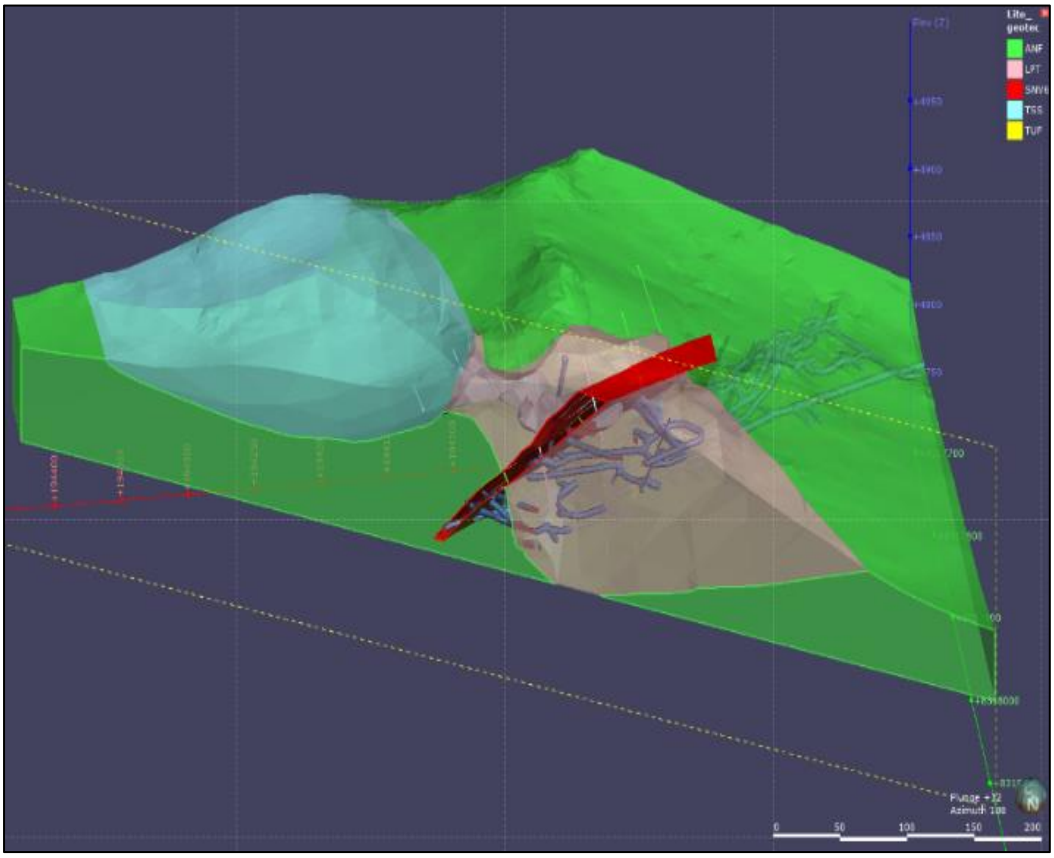


Figure 5. View facing the SE of a NO-SE cut of the modeled bodies.



2.1.2 Geomechanical model

For the elaboration of the geomechanical model the classification of the rock massifs has been carried out, this through mainly two classifications: Bieniawsky RMR and the Barton Q index.

These classifications are in turn supported by the collection of information on the 15 holes used in this Project, which was in charge of the engineer Christian León Ruiz. **[1]**

2.1.2.1 Collection of information

This section will describe the different parameters that were taken into account for obtaining data and their subsequent use in the classification of the massif by the systems already mentioned.

2.1.2.1.1 Determination of R.Q.D.

The quality of rock R.Q.D. It can be determined from pieces of control rocks larger than 10 cm recovered in boreholes or from Jv joints that indicate the number of joints per m3 observed in an outcrop.**[3]** For the first case the first formula is used:

Equation 1 RQD formula

$$\boxed{R.Q.D. = [\sum (\text{Rubble} > 10cm) / (\text{drilling total})]}$$

x100

For the second case, the following formula is used:

Equation 2 Second case RQD formula

$$R.Q.D. = 115 - 3.3xJv$$

Table 2. Mass clasification according to RQD.

2.1.2.1.2 Parameters for RMR Classification

The parameter that defines the classification is the so-called RMR index (ROCK MASS RATING), which indicates the quality of the rock mass in each structural domain from the following parameters:

1. Simple compressive strength of the intact rock, that is, the part of the rock that does not have structural discontinuities.

2. R.Q.D. This parameter is considered of great interest, to select the lining of the tunnels.

3. Spacing of the diaclases or discontinuities, which is the distance measured between the discontinuity planes of each family.

4. Nature of the Diaclases which consists in considering the following parameters:

-Opening the faces of Discontinuity.Your heading and diving.

Quality Index R.Q.D. (%)	***Quality***
0 -25	Muy mala
25 – 50	Mala
50 – 75	Regular
75 – 90	Buena
90 - 100	Excelente.

-Roughness.

-Hardness of the faces of Discontinuity.

-Fill of the Boards.

5. Presence of Water, in a rocky massif diaclased, water has a great influence on its behavior, the description used for this criterion are: completely dry, humid, water at moderate pressure and water at strong pressure.

6. Orientation of discontinuities.

To obtain the RMR Index of Bieniawski the following is done:

Step1: The 5 calculated variables or parameters are added, that results in an index value.

Step 2: Parameter 6 that refers to the orientation of the discontinuities, this classification considers that this parameter is unfavorable, therefore, when this index value of the orientation of the discontinuities is obtained, this is subtracted from the index value obtained when the first 5 parameters are added, when performing this operation the RMR INDEX is obtained and that value is searched in the table described later in the guide.

1st PARAMETER CLASSIFICATION FOR HEALTHY ROCKS RESISTORS

Figure 6. Graph to calculate the RMR index for Simple Compression Resistance.

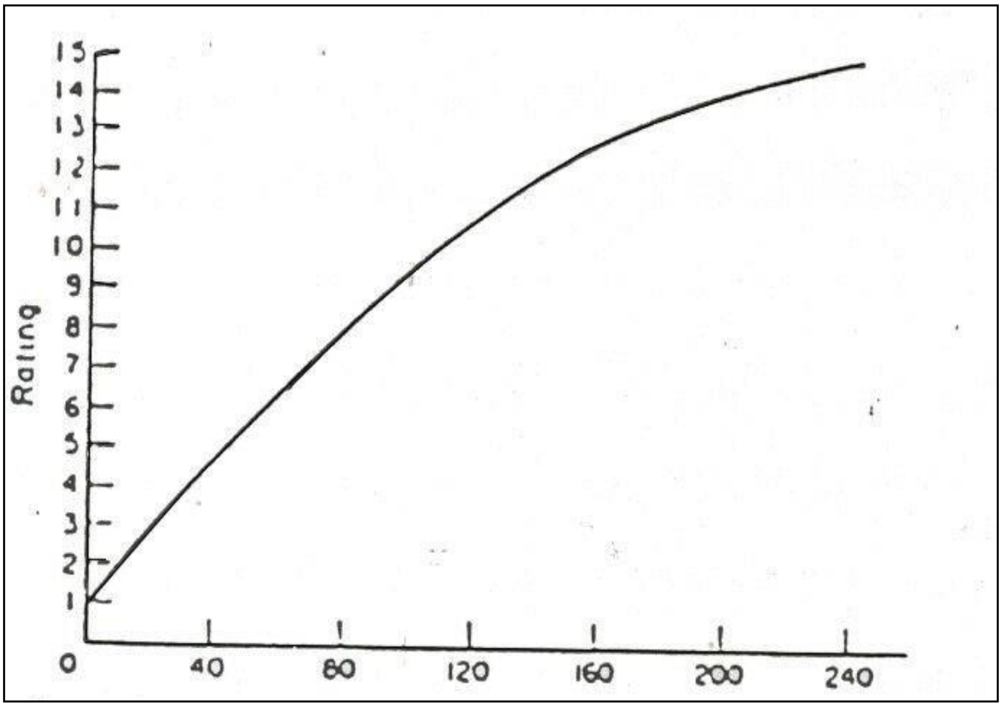


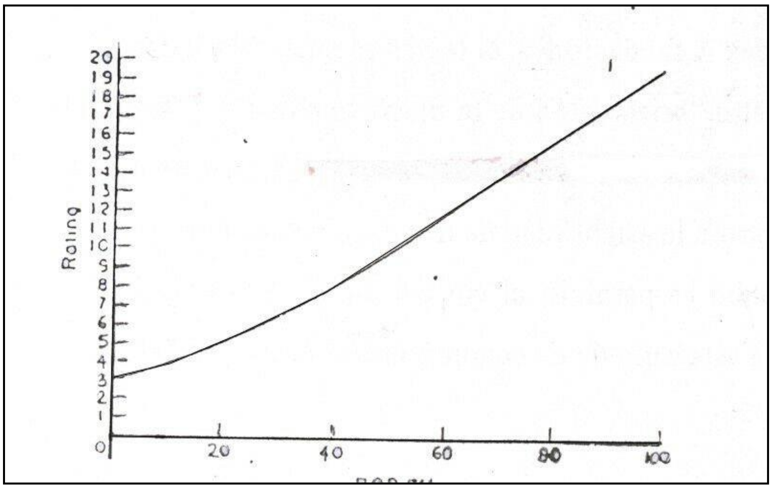
Table 3. Score for uniaxial resistance values.

DESCRIPTION	SIMPLE COMPRESSION RESISTANCE (Mpa)	Score for the RMR System
VERY HIGH	>200	15
HIGH	100 – 200	12
REGULAR	50 – 100	7
LOW	25 – 50	4
VERY LOW	10 -25 3 – 10 1 - 3	**2** 1 0

2nd PARAMETER TO CALCULATE THE RMR. R.Q.D. CALCULATION

The R.Q.D. It is calculated as indicated at the beginning of this guide, when you have the value, you should look for the index for the calculation of the RMR, and for this the following graph is used:

Table 4. Score for the RQD parameter in the RMR system.



Quality Index R.Q.D. (%)	**Score for the RMR System**
0 -25	3
25 – 50	8
50 – 75	13
75 – 90	17
90 - 100	20

Figure 7. Graph to calculate the RMR index, for the R.Q.D parameter.

3rd PARAMETER TO CALCULATE THE RMR. SPACING THE DISCONTINUITIES.

The spacing of discontinuities is classified according to the table below:

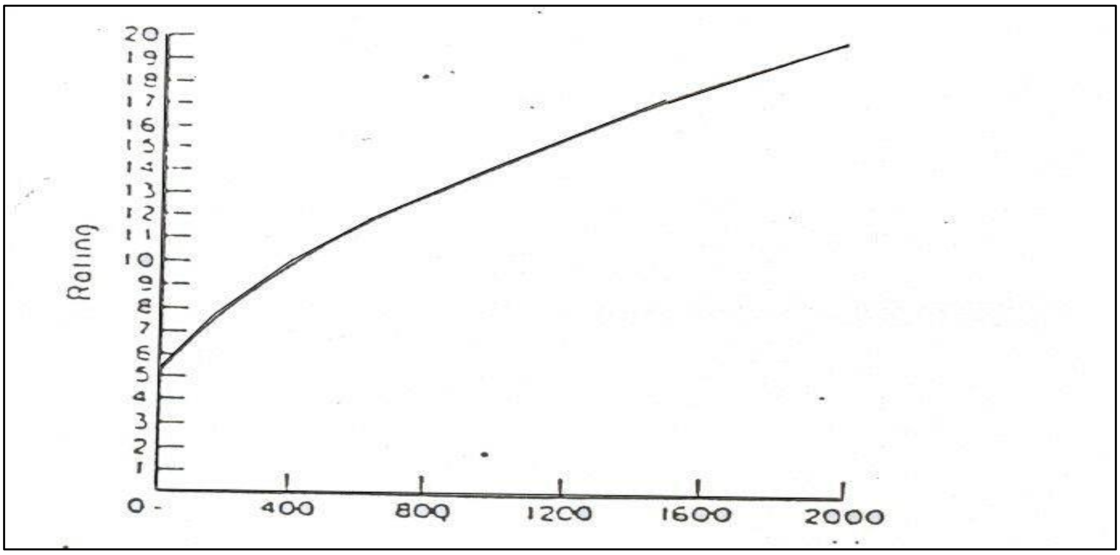
Table 5. Score of the openings of the disagreements, for the RMR system.

Description	**Board spacing**	**Type of rock mass.**	**Score for the RMR System**
Very wide	> 3 m	Solid	20
Width	1 – 3 m	Massive	15

Moderate Closed Mind	0.3 – 1 m	In blocks	10
Closed	50 – 300 mm	Fractured	8
Very closed	< 50 mm	Crushed	5

The following graph is used to calculate the range:

Figure 8. Graph to calculate the RMR index, based on the discontinuity spacing parameter.



4th PARAMETER TO CALCULATE THE RMR. NATURE OF THE BOARDS.

Table No. 6 shows the classification according to the openings of the discontinuities.

Table 6. Score of the openings of the disagreements, for the RMR system.

Grade	**Description**	**Face separation**	**Score for the RMR System**
1	Open	> 5mm	0
2	Moderately open	1 – 5 mm	1
3	Closed	0.1 – 1 mm	4

4	Very closed	< 0.1 mm	5
5	Does not have	0	6

Table N °7 shows the classification according to the continuity of the discontinuities.

Table 7. Score for continuity of the family of discontinuities.

Grade	**Description**	**Continuity (m)**	**RMR Range**
1	Very small	< 1	6
2	Little	1 – 3	4
3	Half	3 – 10	2
4	high	10 – 20	1
5	Very high	> 20	0

Table N ° 8 shows the classification according to the roughness of the discontinuities

Table 8. Punctuation texture for discontinuities.

Description	RMR Range
Very rough	6
Rough	5
Slightly rough	2
Soft	1
Fault mirror	0

Table N ° 9 shows the classification according to the filling of the discontinuities.

Table 9. Scoring for the type of filler and its thickness.

Grade	**Description**	**Score for the RMR System**
1	Soft fill> 5 mm	0
2	Soft fill <5mm	2
3	Hard filling> 5mm.	2
4	Hard filling <5 mm	4
5	none	6

Table N ° 10 shows the classification according to the weathering of the discontinuities

Table 10. Score for the degree of meteorization of discontinuities.

Grade	Description	**Score for the RMR System**
1	Decomposed	0
2	Very weathered	1
3	Moderately weathered	3
4	Slightly weathered	5
5	Not weathered	6

To calculate the RMR according to the nature of the discontinuities, the average of the sum of the RMR obtained in the 5 tables described above is taken, in addition to adding the value of 15 points for the assessment of the dry condition of the area.

Rock mass quality in relation to the RMR Index

Table 11. Score for the RMR system, which has 5 scales

CLASS	QUALITY	RMR RANGE
I	Very good	100-81
II	Good	80-61
III	Fair	60-41
IV	Poor	40-21
V	Very Poor	< 20

2.1.2.1.3 Barton Q classification

The Tunnelling Quality Index, or Q system, was developed by Barton and his collaborators in 1974 (12) and in later years (45) (46) (47), and also uses 6 parameters to estimate the behavior of the rock mass:

• Rock Quality Design (RQD).

• Family number of meetings or discontinuities (Jn).

• Roughness of the joints (Jr).

• Degree of alteration of the joints (Ja).

• Presence of water (Jw)

• Tension state of the rock, Stress Reduction Factor (SRF).

In this system there is an assessment for each of the 6 parameters which are replaced in the following formula:

However, for the end of the project we have seen the need to use one of the

Equation 3 Q index formula

$$\boxed{Q = \left(RQD / Jn\right) \cdot \left(Jr / Ja\right) \cdot \left(Jw / SRF\right)}$$

proposed Correlations for the RMR and Q of Barton.

The correlation between the two classification systems was proposed by Bieniawski in 1976 through a linear regression of 111 sets of RMR and Q data from 62 cases from Scandinavia, 28 from South Africa, and 21 from North America, Europe and Australia, with a coefficient of correlation of R2 = 0.59 (R = 0.77). (Gutierrez F., 2017)

Having the following formula:

Equation 4 Correlation between RMR and Q index

$$\boxed{RMR=9.LnQ+44}$$

From where clearing we have the formula to find the Barton Q, which was used for calculations of the block model estimates in the Leapfrog Geo software.

Equation 5 Correlation between Q index and RMR

$$\boxed{Q=e^{(RMR-44)/9}}$$

3. PROCESS METHODOLOGY

3.1 Geomechanical classification

In the mineralized structure, between the 4800 and 4820 levels the quality of the rock is mainly average, between the 4820 and 4840 level the quality of the rock ranges from medium to good and between the 4840 and 4860 levels the rock quality decreases on average to bad, mainly because of the onset of the presence of oxides. In addition, this model considers the presence of the faults to the floor and ceiling, in that case the presence of these produces laxations.

In the boxing rock, between the 4800 and 4820 levels the rock quality ranges from medium to good, this information is linked to the interior mine detail lines; between the 4820 and 4840 levels the rock has a poor quality to regulate with good quality sectors. Between the 4840 and 4860 levels the rock quality goes from good to regular mainly, with bad quality sectors.

3.1.1 Lithological aspects.

The lithostructural domains show locally the presence of a type of predominant lithology constituted by volcanic rocks of the flow type, Andesitas, in the which is located entirely the Ánimas vein.

Mineralization in the Ánimas vein shows a well-marked structural control defined by failures to the floor and ceiling of the mineralized structure. Another feature important mineralized structure refers to the sinuous nature of the grain with different ranges in horizontal and vertical projection, from intermediate to low dive (42-50 °), it can also be seen that the dip decreases towards the surface. Mineralization at level 6 of the Ánimas vein consists of an assembly mineralogical consisting of rhodonite, quartz, pyrite, sulphides and tetrahedrite; a particular feature are the bands with the highest concentration of silver and are located towards the roof of the grain; the floor and ceiling boxes of the mineralized structure Lithologically they are composed of andesites and andesitic tuffs.

3.1.2 Structural aspects.

At the local level the most relevant characteristics of the average structural aspects of failures (major structures) and discontinuities (minor structures) Present in the rock mass that involves the areas evaluated are as follows:

3.1.2.1 Failure systems.

The geomechanical characteristics that the faults present in the Ánimas vein are:

variable spacing greater than 1 - 2 m, with a persistence ranging from about meters to tens of meters, as for the opening they are characterized by being open (> 5 mm), the surfaces of the faces are smooth or flat and occasionally have fault mirrors and certain undulations, they also have brechoid fillings, oxides and clays, superficially are very altered and the presence of water from infiltration in these faults comes from wet to wet and occasionally in sectors point is exposed as a drip, causing instability in contact

With the mineralized structure. It is necessary to specify that in the floor and ceiling boxes of the mineralized structure the failures are very sporadic (isolated); however in the structure mineralized have a continuous exposure in the course of the pit constituting a destabilizing agent of rock mass excavations (pit).

3.1.2.2 Fracturing systems.

The geomechanical characteristics of the discontinuities in the floor and roof box they have spacings in the box between 20 - 60 cm, while in the structure Mineralized average 6-20 cm. In the boxes the persistence varies between 3 to> 20 m, the faces appear slightly to moderately altered, the opening is varied from narrow to extremely narrow, you can see stuffed with various types highlighting oxides with a smooth consistency (occasionally they occur filled with clays, which are the product of the alteration of the same rock towards the wall of the same discontinuity generated by hydrothermal processes and influence later of the infiltration of meteoric waters) and also they appear without filling, in As for the shape and roughness of the walls it varies between flat to wavy and rough to slightly rough.

While in the mineralized structure persistence varies between 1 and 10 m, the faces are moderately altered (there is a greater intensity towards contact with the mineralized structure, this being likely be associated with the influence of hydrothermal processes), the opening is varied from narrow to very narrow, as regards the filling, different types of oxides are observed, Calcite of soft consistency and hard consistency quartz, in terms of shape they present from wavy to flat and the roughness of the walls varies from rough to slightly rough.

3.1.2.3 Degree of fracture

In the logging of drill witnesses it was determined that the average spacing between invoices for the encasing rock is 0.178 m considering it as very rock fractured. While the grain has an average spacing between fractures of 0.122 m which represents a very fractured rock. Of the values obtained are considers that the average spacing between fractures of the rock mass is 0.172 m establishing itself as a very fractured rock.

In the mapping in the interior mine through a detail line it was determined that the

Average spacing between fractures for the encasing rock is 0.263 m considering it as fractured rock. While the grain has a spacing average between fractures of 0.173 m which represents as very fractured rock. From the values obtained are considered to be the average spacing between fractures of the Rocky massif is 0.238 m, establishing itself as a very fractured rock. In the mapping of outcrops by detail line it was determined that the

Average spacing between fractures average for the encasing rock is 0.173 m considering it as a very fractured rock. While the grain has a average spacing between fractures average of 0.407 m which represents as fractured rock Of the values obtained, the average spacing is considered between fractures of the rock massif is 0,180 m establishing itself as a very rock fractured.

3.2 3D rock mass model

For 3D modeling, the Leapfrog Geo software was used, where all the information obtained from the analysis of the RMR parameters and the calculated Q index was inserted.

Subsequently, to generate the Estimate, a “declustering” was performed for each RMR parameter, then the numerical model was elaborated, where it was estimated inversely of the distance to the cube, to finally obtain the block model for the 4 main bodies: the Vein (ASNV6), andesite flow (ANF), Tobáceas Sandstone (TSS) and lapilli tuffs (LPT), which will be shown below.

We can see in the figures 9, 10 and 11 that most of the blocks have an RMR between 40 and 60, which is of medium quality of the massif.

Figure 9. Plan view of the block model for the RMR of the three main bodies (ANV6, LPT and ANF). The legend can be seen in the upper right.

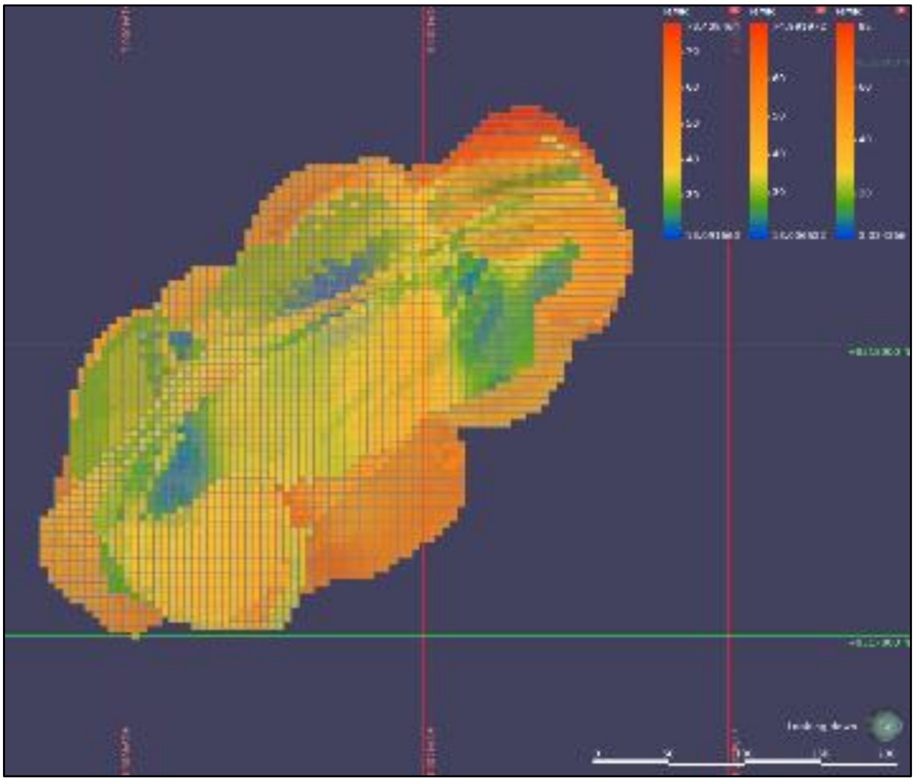
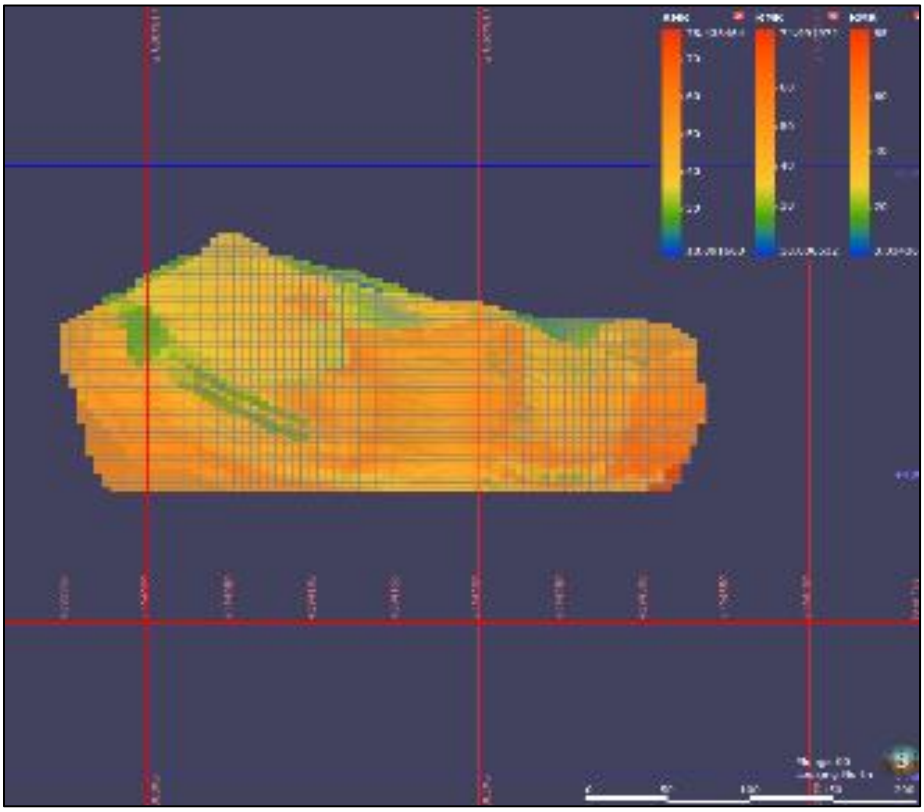


Figure 10. Northward view of the block model for the RMR of the three main bodies (ANV6, LPT and ANF). The legend can be seen in the upper right.

Figure 11. Northward view of the block model for the RMR of the three main bodies



(ANV6, LPT and ANF). The legend can be seen in the upper right.

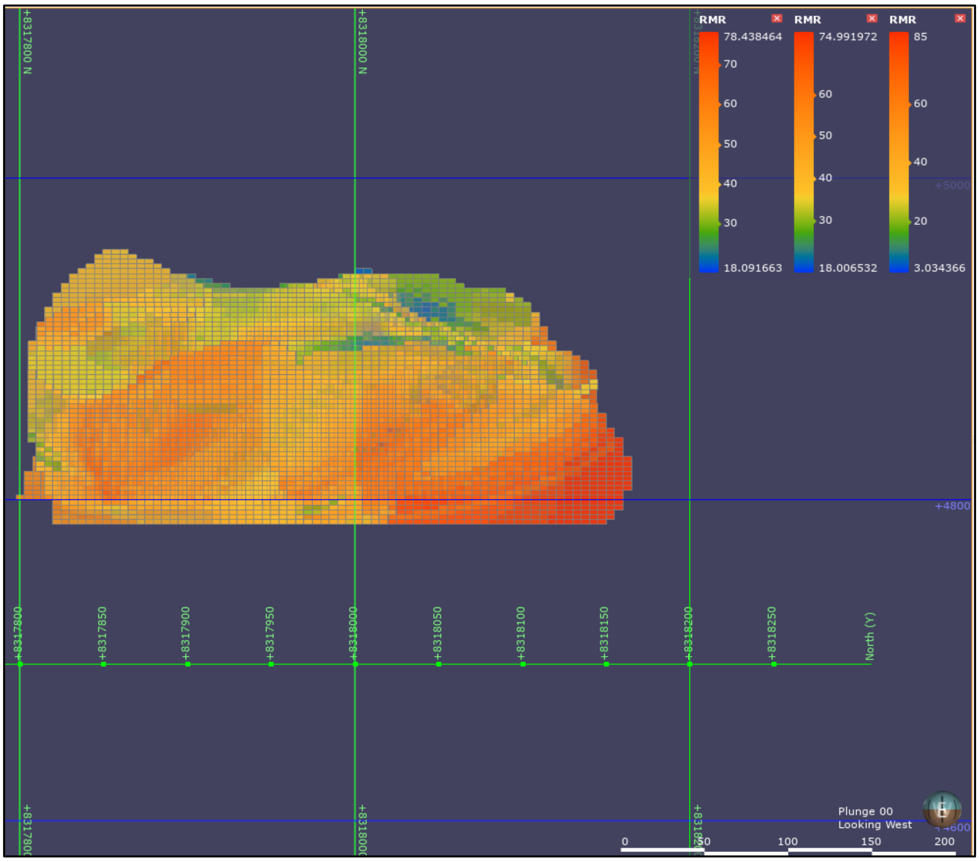
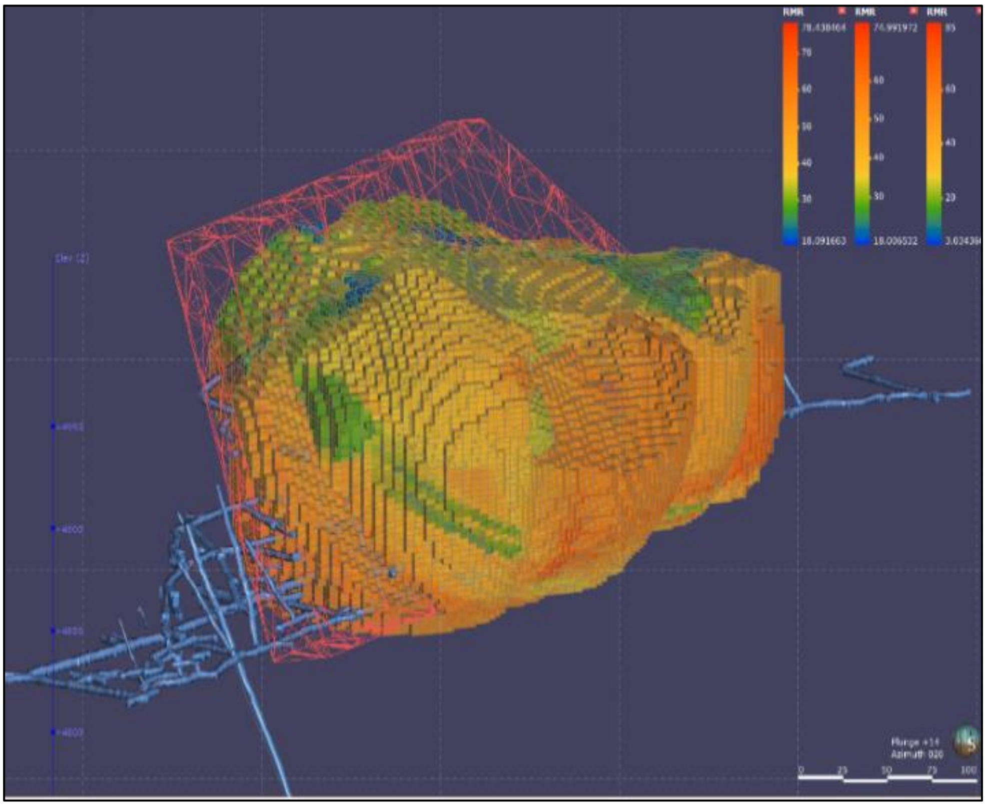


Figure 12. View towards the NE of the vein (red color), with the tunnels (Mesh provided by the resources area of the Bateas company).



3.2.1 **Exploratory analysis of the RMR data.**

- Once the lithologies were determined, the data corresponding to the parameters that make up the RMR (OPEN, RIM, RQD, RELL, RUGOS, SD, ALT, LD) were analyzed.
- The analysis included determining the number of samples, as well as finding the minimum and maximum value, in such a way that it allows us to identify some wrong value, since each parameter is within a certain range, for example the RQD has a score of 0 to 20, and if we find a value greater than 20 in that parameter, it means that the data was taken badly and must be corrected.
- The analysis also included the determination of the mean, standard deviation and the coefficient of variation.

In the table 12 it’s noted all the parameters taken from the Leapfrog software in the Statistics tool

Table 12 Statistics Compilation

LITOLOGIA	PARAMETRO	N DE MUESTRAS	MIN	MAX	MEDIA	D.E	CV
ANF	Open	198	0	6	1,255	1,129	0.899
	RQD	200	0	20	9,068	5,911	0.652
	LD	198	0	15	2.95	1,230	0.417
	Relleno	198	0	6	2,538	1,705	0.672
	RIM	199	0	12	4,408	3,013	0.681
	Rugosidad	192	0	6	1,041	1,215	1,167
	SD	197	0	20	11,221	5,473	0.488
	Alteracion	197	0	6	3,188	2,052	0.644
ASNV6	Open	85	0	6	1.364	1.246	0.913
	RQD	87	3	17	6.254	4.988	0.797
	LD	86	0	20	2.807	1.748	0.623
	Relleno	85	0	6	2.29	1.723	0.752
	RIM	88	0	12	5.005	4.473	0.894
	Rugosidad	85	0	6	1.242	1.417	1.14
	SD	87	0	20	9.027	5.059	0.56
	Alteracion	84	0	6	3.054	2.241	0.734
LPT	Open	117	0	6	1,589	1,339	0.842
	RQD	121	0	20	10,327	5,788	0.560
	LD	123	0	20	4,156	4,242	1,021
	Relleno	116	0	6	2,622	1,692	0.645
	RIM	123	0	12	4,281	3,403	0.795
	Rugosidad	117	0	6	1,124	1,218	1,084
	SD	123	0	20	11,737	5,379	0.458
	Alteracion	117	0	6	3,148	2,061	0.654

- Once the previous analysis was completed, each of the parameters was composed to interpret its behavior, in most cases it was composed at a distance of 10.

Table 13 shows the composites that were used and the statistical data that were obtained from it.

Table 13 Composited Statistics

LITOLOGIA	PARAMETRO	N DE COMPOSITOS	MIN	MAX	MEDIA	D.E	CV
ANF	Open	465	0	6	1,278	1,079	0.844
	RQD	468	0	20	9,195	5,739	0.624
	LD	465	0	15	2,956	1,134	0.384
	Relleno	465	0	6	2,579	1,594	0.618
	RIM	467	0	12	4,471	2,929	0.655
	Rugosidad	441	0	6	1,041	1,215	1,167
	SD	451	0	20	11,218	5,206	0.464
	Alteracion	463	0	6	3,227	1,939	0.611
ASNV6	Open	140	0	4	1.333	1.171	0.879
	RQD	141	3	17	5.587	4.514	0.771
	LD	140	0	9	2.718	1.235	0.455
	Relleno	140	0	6	2.156	1.531	0.71
	RIM	140	0	12	4.729	4.187	0.885
	Rugosidad	140	0	5.65	1.26	1.275	1.013
	SD	139	0	20	8.627	4.654	0.539
	Alteracion	140	0	6	2.949	2.128	0.721
LPT	Open	372	0	6	1,574	1,295	0.823
	RQD	381	0	20	10,286	5,659	0.550
	LD	385	0	20	385	4,172	0.996
	Relleno	371	0	6	2,599	1,640	0.631
	RIM	385	0	6	4,261	3,334	0.783
	Rugosidad	372	0	6	1,115	1,141	1,023
	SD	380	0	20	11,777	5,133	0.436
	Alteracion	372	0	6	3,138	2,020	0.644

- The graphics of this chapter are in the appendix

3.3 Estimation of geotechnical parameters

- With the finished lithological model, it was necessary to create a numerical model that allows us to make a block model that is not only based on isovalues, but results that are estimated from nearby data, it is for this reason that the numerical model It was believed that 2 estimation methods were used, the first one was optional for each of the cases and it was declustering, while the second method was the Inverse of the distance to the cube, for this last one a search ellipsoid is needed, the which have distances and preferential directions, these distances and directions were taken from planes studied on the lithological model in which we find certain trends that could indicate the behavior and give us an idea of the direction of the search ellipsoid.

Equation 6 Inverse distance formula

$$z_j = \frac{\sum z_i / d_{ij}^{\beta}}{\sum 1 / d_{ij}^{\beta}}$$

- In this way, the following table was prepared indicating the directions, distances, minimum and maximum # of samples that were taken for the search ellipsoid of each parameter.

  Table 14 shows the preferred distances and directions of the search ellipsoid used to estimate each parameter, as well as the minimum and maximum number of samples.

Table 14 Distances and directions for the search ellpisoid 1/3

Lito	Var	Ejes	Dist	Dir	# Min. Comp.	# Max. Comp.	# Max. x sondaje
LPT	ALT	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	LD	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	OPEN	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	RELL	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	RIM	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	RQD	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	RUGOS	1	90	0	2	7	2
		2	60	0			
		3	90	90			
	SD	1	90	0	2	7	2
		2	60	0			
		3	90	90			

Table 15 Distances and directions for the search ellpisoid 2/3

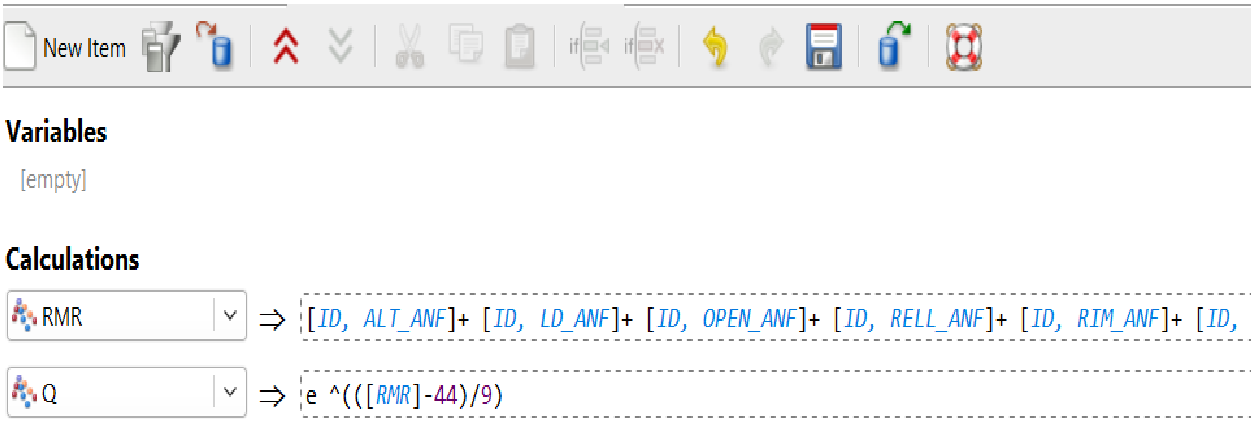
ANF	ALT	1	90	90	2	16	2
		2	90	46.44			
		3	60	90			
	LD	1	90	90	2	16	2
		2	60	61.36			
		3	60	90			
	OPEN	1	90	90	2	10	2
		2	60	63.13			
		3	60	90			
	RELL	1	90	89.9	2	10	2
		2	90	48.73			
		3	90	89.9			
	RIM	1	120	90	2	14	2
		2	90	46.36			
		3	60	90			
	RQD	1	90	82.56	2	20	2
		2	90	226.5			
		3	60	82.23			
	RUGOS	1	90	90	2	16	2
		2	90	54.42			
		3	60	90			
	SD	1	90	90	2	16	2
		2	90	46.54			
		3	60	90			

Table 16 Distances and directions for the search ellpisoid 3/3

ASNV6	ALT	1	90	44.14	2	7	2
		2	90	144.7			
		3	60	136.5			
	LD	1	90	49.8	2	7	2
		2	90	143.6			
		3	60	146.5			
	OPEN	1	90	50.3	2	7	2
		2	90	142.1			
		3	60	36.68			
	RELL	1	90	51.34	2	7	2
		2	90	142.1			
		3	60	152			
	RIM	1	90	52.85	2	7	2
		2	90	139.6			
		3	60	150.9			
	RQD	1	90	51.1	2	7	2
		2	90	141.7			
		3	60	157.6			
	RUGOS	1	90	48.82	2	7	2
		2	90	140.5			
		3	60	139.5			
	SD	1	90	52.13	2	7	2
		2	90	143.4			
		3	60	152.2			

- Once the numerical model of each parameter was determined, the block model was created, said block model as a whole (all the parameters included) allowed us to calculate the sum of them and thus obtain an RMR for each block determined, there is also a correlation between RMR and Q, this calculation was also made in the model and finally each block had a value of RMR and Q, which later would serve to analyze the type of support that can be used.

Equation 7 RMR formula and Q index formula



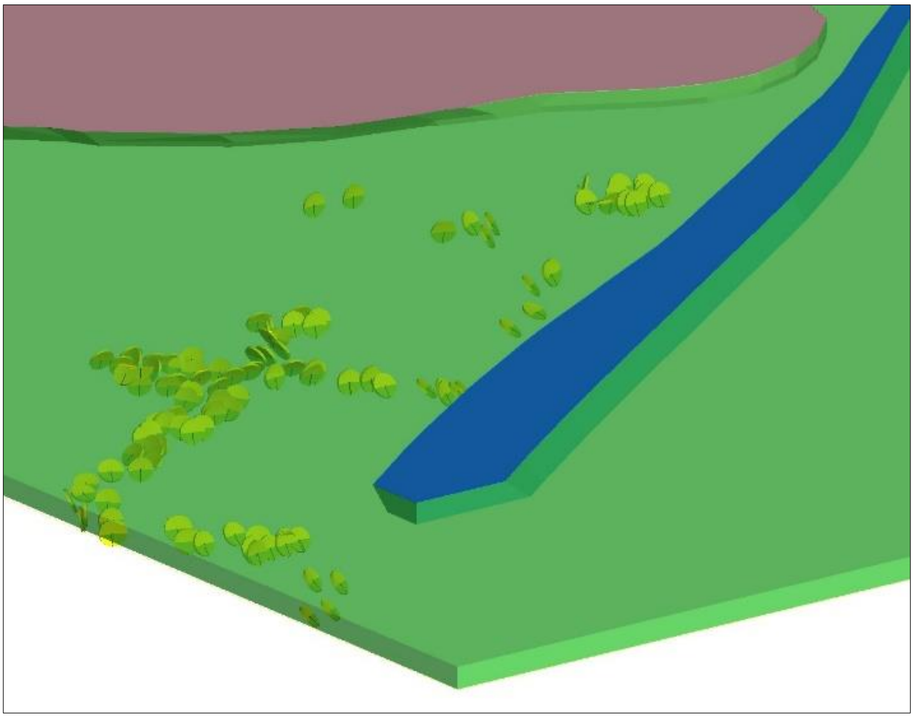
3.4 Structural Analysis

We have information of a structural mapping carried out at the 4800m level indicating the faults and joints. This structural information was uploaded to the Leapfrog software to perform an analysis by lithology. At this level the structural information cuts mainly 2 lithologies: the porphyritic andesite (ANF) and the vein (ASNV6), so we have separated into 4 groups to make the analysis of structural systems separately. The groups are: Joints in the porphyritic andesitic, joints in the vein, faults in the porphyritic andesite and faults in the vein. To determine the main families, we have used the Dips software.

3.4.1 Joints in porphyritic andesitic (ANF)

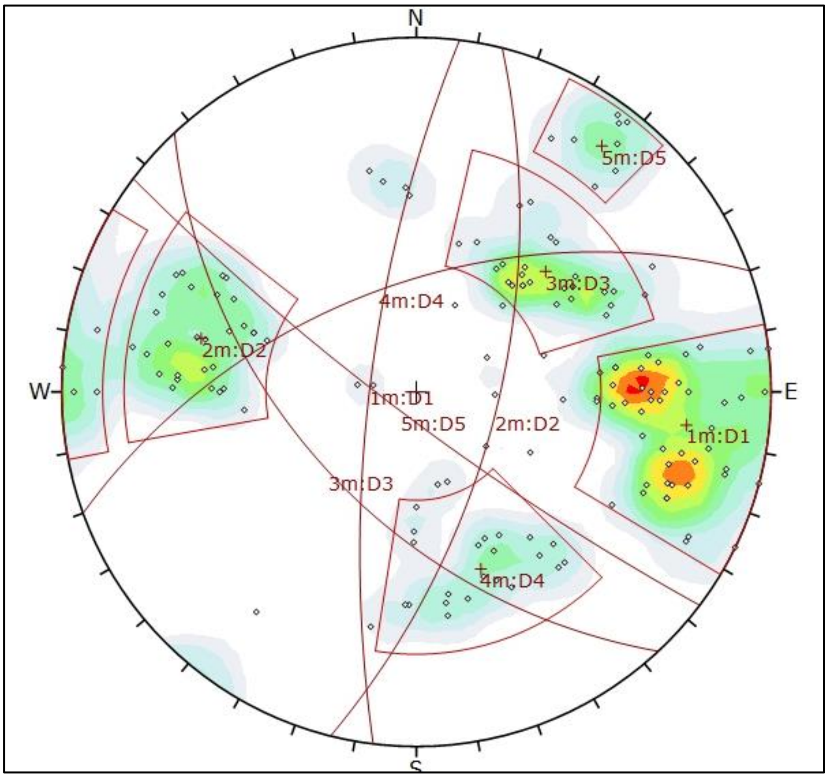
Of the total joints that were imported from the structural mapping at a height of 4800m, those that cut the porphyritic andesite were separated using the Leapfrog software.

Figure 13. Joints (in yellow) inside the porphyritic andesite (ANF).



The structural data of the joints that cut the porphyritic andesitic was uploaded to the Dips software to determine the main families.

Figure 14. Stereographic projection of the joints' poles, they were grouped in 5 families.



Five main families were determined, which are presented below indicating the number of poles per family.

Table 17. Results of the Structural system for the joints in porphyritic andesite.

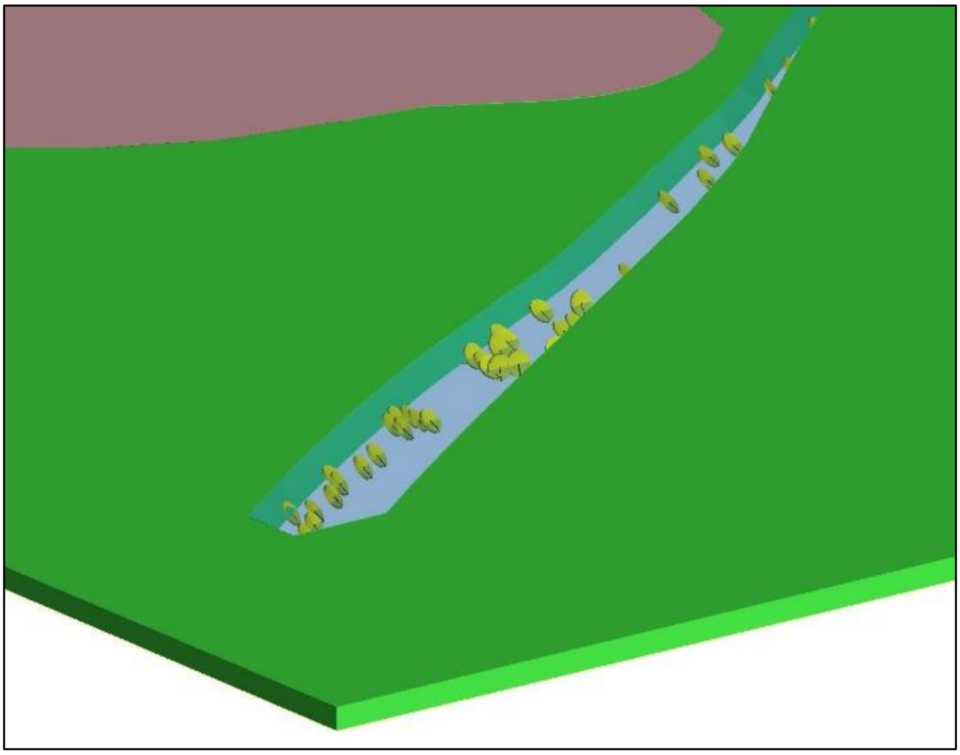
ID	***Set***	***Azi.***	***Dip***	***Poles$_{set}$***	***Poles$_{total}$***	***%***	***K***
1	J1	187	75	49		32%	28
2	J2	14	64	29		19%	42
3	J3	137	53	26		17%	36
4	J4	250	56	21	154	14%	29
5	J5	127	82	8		5%	143
Other				21		**14%**	
Total				**154**		**100%**	

The percentage of poles that do not belong to any family must be less than 30%, to have a good distribution of families, for this case it is 14%. The value of the K Fisher will be smaller as more data are grouped in a family.

3.4.2 **Joints in the vein (ASNV6)**

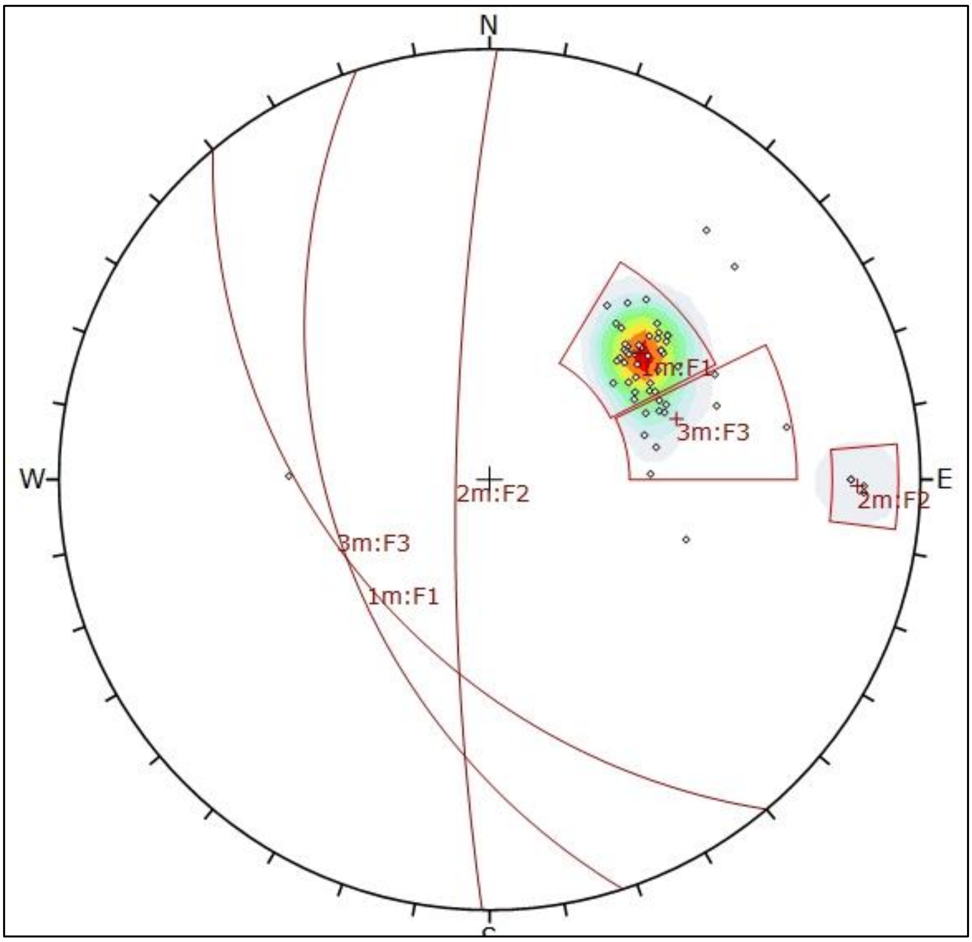
Of the total joints that were imported from the structural mapping at a height of 4800m, those that cut the vein were separated using the Leapfrog software.

Figure 15. Joints (in yellow) inside the vein (ASNV6).



The structural data of the joints that cut the vein was uploaded to the Dips software to determine the main families.

Figure 16 Stereographic projection of the joints' poles, they were grouped in 3 families.



Three main families were determined, which are presented below indicating the number of poles per family.

Table 18. Results of the Structural system for the joints in the vein.

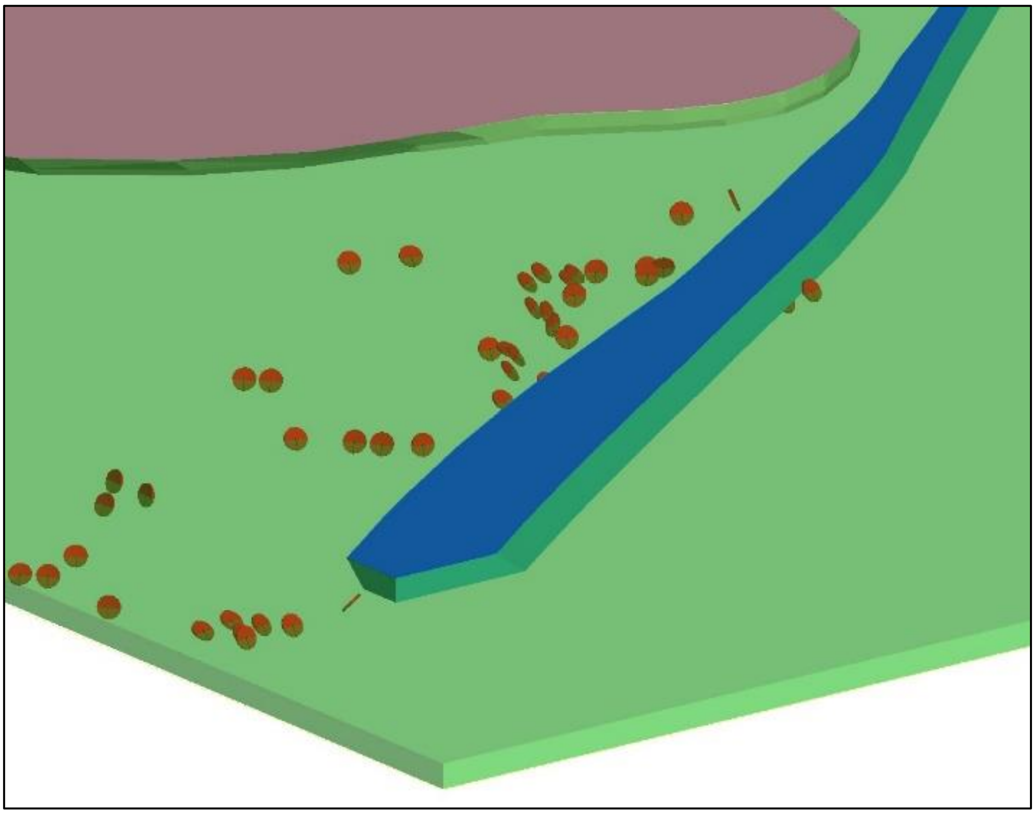
ID	***Set***	***Dip***	***Dip Dir***	***Poles***$_{set}$	***Poles***$_{total}$	***%***	***K***
1	F1	49	230	39	58	67%	149.64
2	F2	81	271	4		7%	2947.1
3	F3	49	252	11		19%	56.95
Other				4		7%	
Total				**58**		**100%**	

The percentage of poles that do not belong to any family must be less than 30%, to have a good distribution of families, for this case it is 7%. The value of the K Fisher will be smaller as more data are grouped in a family.

3.4.3 Faults in porphyritic andesitic (ANF)

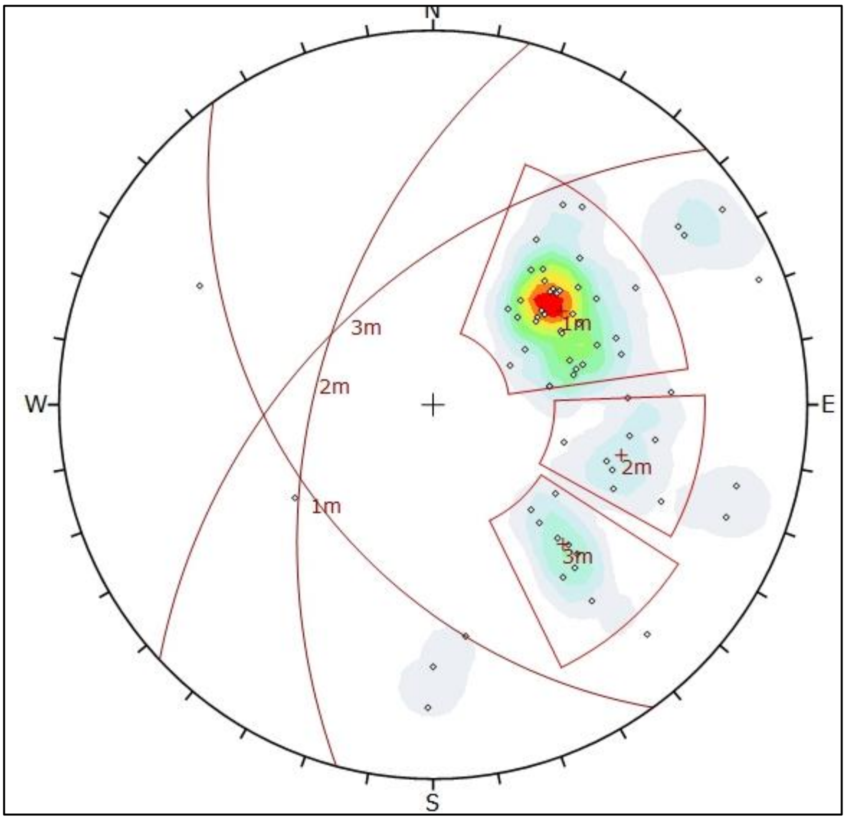
Of the total faults that were imported from the structural mapping at a height of 4800m, those that cut the vein were separated using the Leapfrog software.

Figure 17. Faults (in red) inside the porphyritic andesite (ANF).



The structural data of the faults that cut the porphyritic andesitic was uploaded to the Dips software to determine the main families.

Figure 18. Stereographic projection of the faults' poles, they were grouped in 3 families.



Three main families were determined, which are presented below indicating the number of poles per family.

Table 19. Results of the Structural system for the faults in the porphyritic andesite.

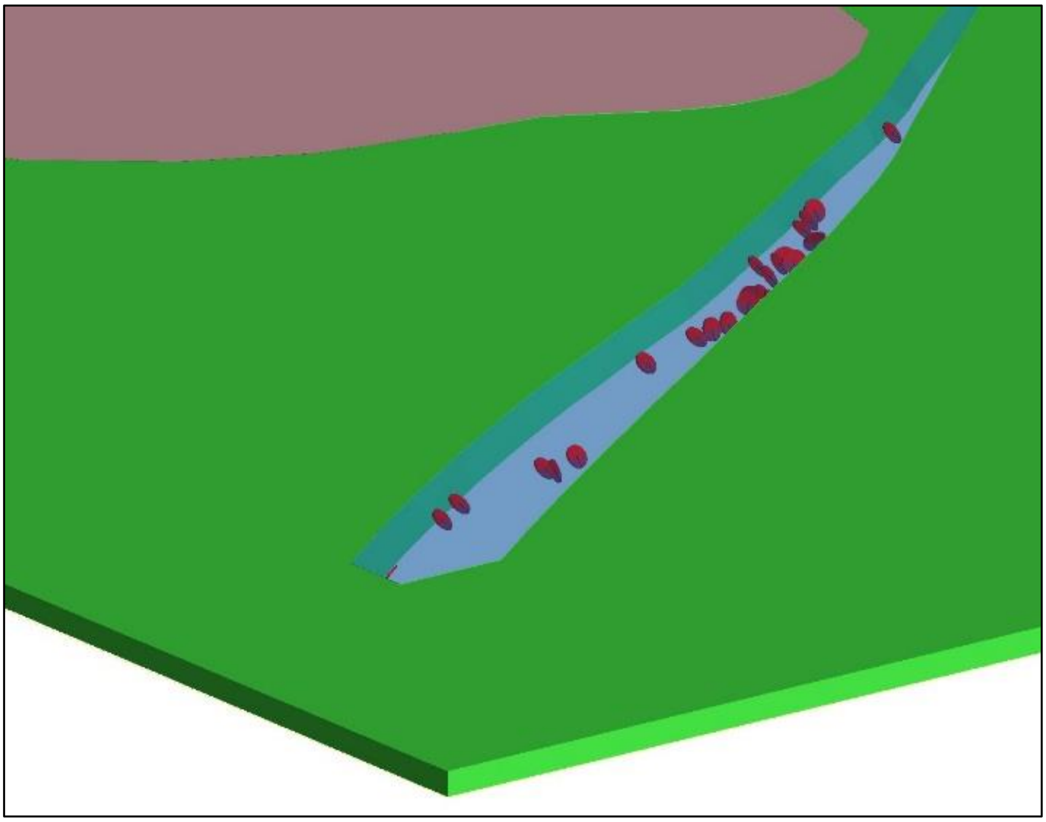
ID	Set	Azi.	Dip	$Poles_{set}$	$Poles_{total}$	%	K
1	F1	144	46	36	66	55%	34.58
2	F2	195	55	8		12%	56.76
3	F3	227	54	9		14%	74.61
Other				13		20%	
Total				**66**		**100%**	

The percentage of poles that do not belong to any family must be less than 30%, to have a good distribution of families, for this case it is 20%. The value of the K Fisher will be smaller as more data are grouped in a family.

3.4.4 Faults in the vein (ASNV6)

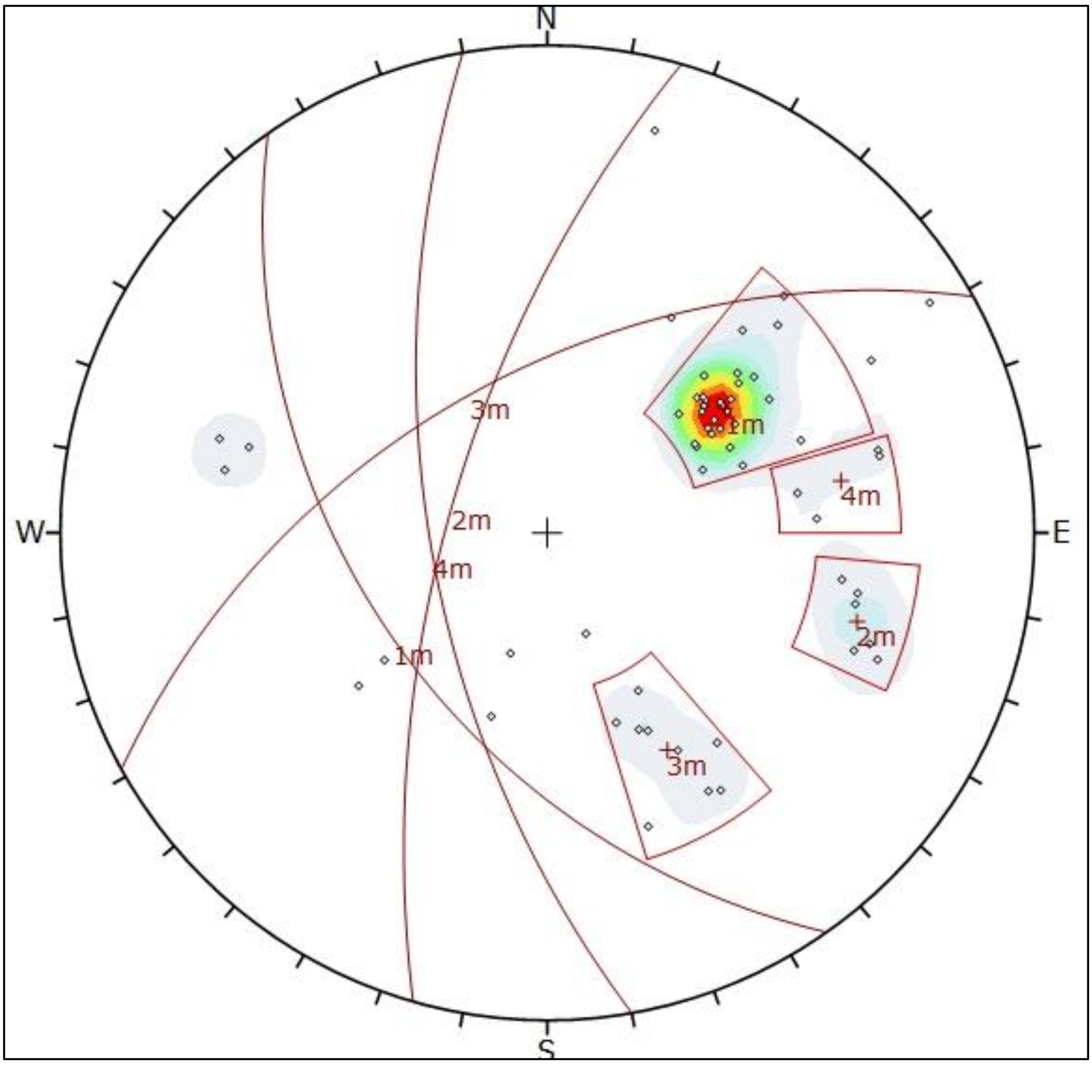
Of the total faults that were imported from the structural mapping at a height of 4800m, those that cut the vein were separated using the Leapfrog software.

Figure 19. Faults (in red) inside the vein (ASNV6).



The structural data of the faluts that cut the vein was uploaded to the Dips software to determine the main families.

Figure 20. Stereographic projection of the faults' poles, they were grouped in 4 families.



Four main families were determined, which are presented below indicating the number of poles per family.

Table 20. Results of the Structural system for the faults in the vein.

ID	***Set***	***Azi.***	***Dip***	***Polos$_{set}$***	***Polos$_{total}$***	***%***	***K***
1	F1	145	48	31	62	50%	80.5
2	F2	196	67	6		10%	191.7
3	F3	241	54	9		15%	58.7
4	F4	170	63	4		6%	80.3
Other				12		19%	
Total				**62**		**100%**	

The percentage of poles that do not belong to any family must be less than 30%, to have a good distribution of families, for this case it is 19%. The value of the K Fisher will be smaller as more data are grouped in a family.

3.5 Support analysis for 3 tunnel section

- To perform the sustainability assessment, the sections of 3 tunnels were taken in the block model generated in Leapfrog, after analyzing these sections the following data were obtained:

Table 21. Results of the structural data and other parameters for the 3 tunnels.

	Azimuth	High	Width	Q	ESR	High/ESR
Vein Tunnel	53	3	3	7.5	1.6	1.875
ANF1 Tunnel	86	3.5	3	1.5	1.6	2.1875
ANF2 Tunnel	337	3	3	28	1.6	1.875

- The azimuth was obtained with the help of the Structural Modeling tool <New planar structural data, with this tool an address was drawn to each of the tunnels and the azimuth was obtained.
- The height and width of the tunnels were obtained from measurements with the Leapfrog ruler tool
- The value of Q was obtained from the previous block model, that block that intersected the tunnel section provided the value of Q for the space it occupies, it is worth mentioning that for the tunnel in the vein it intersects with the model of ASNV6 blocks and for the ANF tunnels intersected with the block model with the same name.
- The ESR factor was obtained from the following table:

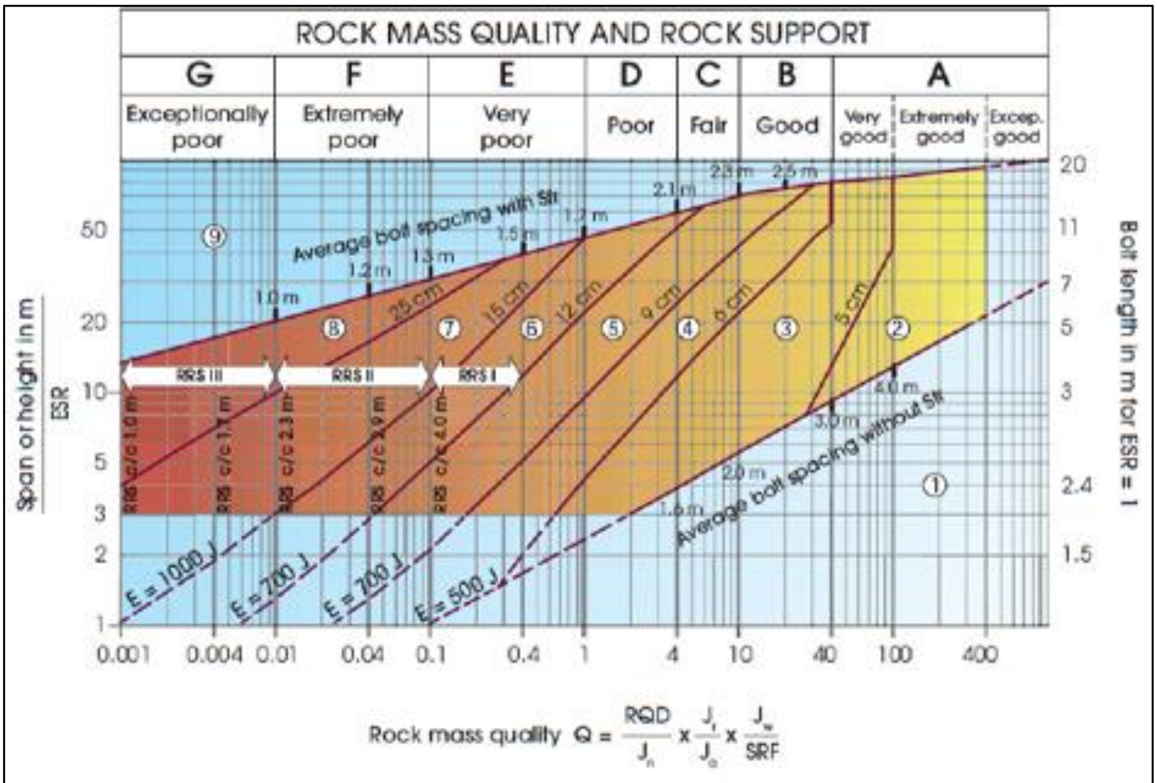
Table 22. Values for the ESR factor. ***[4]***

Excavation category	Excavation type	ESR
A	Temporary mine openings	3–5
B	Circular shafts	2.5
	Rectangular shafts	2.0
C	Permanent mine openings, water tunnels for hydropower, pilot tunnels drifts and headings for large openings	1.6
D	Storage rooms, water treatment plants, minor road and railway tunnels, surge chambers, access tunnels, etc.	1.3
E	Power stations, major railway tunnels, civil defense chambers, portals, intersections, etc.	1.0
F	Underground nuclear power stations, railway stations, sports and public facilities, factories, etc.	0.8

It is observed that the chosen factor is 1.6

- The following table was used to determine the support in each tunnel:

Figure 21. Figure to determine the support for each tunnel. ***[4]***



Support Categories

1. Unsupported or spot bolting
2. Spot bolting, SB
3. Systematic bolting, fibre reinforced sprayed concrete, 5-6 cm, B + Sfr
4. Fibre reinforced sprayed concrete and bolting, 6-9 cm, Sfr (E500) + B
5. Fibre reinforced sprayed concrete and bolting, 9-12 cm, Sfr (E700) + B
6. Fibre reinforced sprayed concrete and bolting, 12-15 cm + reinforced ribs of sprayed concrete and bolting, Sfr (E700) + RRS I + B
7. Fibre reinforced sprayed concrete > 15 cm + reinforced ribs of sprayed concrete and bolting, Sfr (E1000) + RRS II + B
8. Cast concrete lining, CCA or Sfr (E1000) + RRS III + B
9. Special evaluation

For this reason, the Height / ESR value was obtained

- Once the parameters were obtained, we will place them in the table that indicates the support according to the Barton Q index, the following results were obtained:

  Vein Tunnel

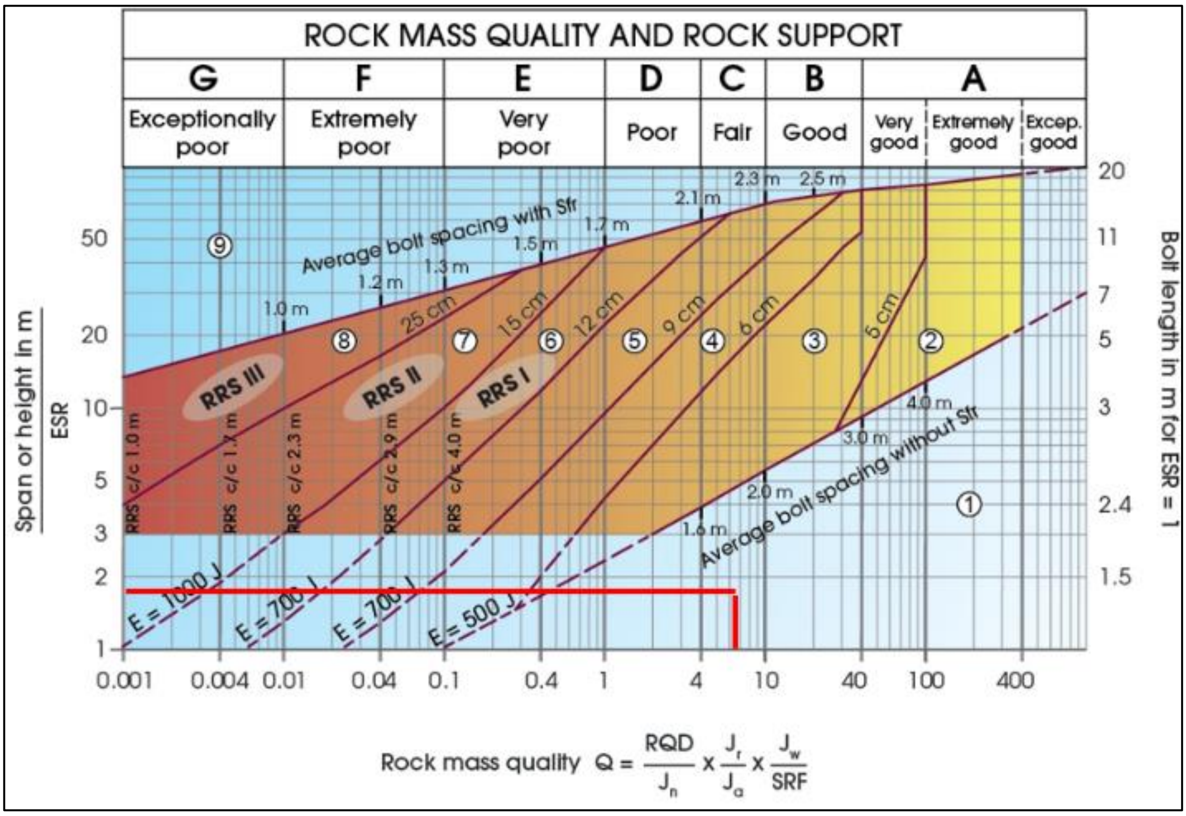


Figure 22. Support for the Vein Tunnel is category 1.

ANF1 Tunnel

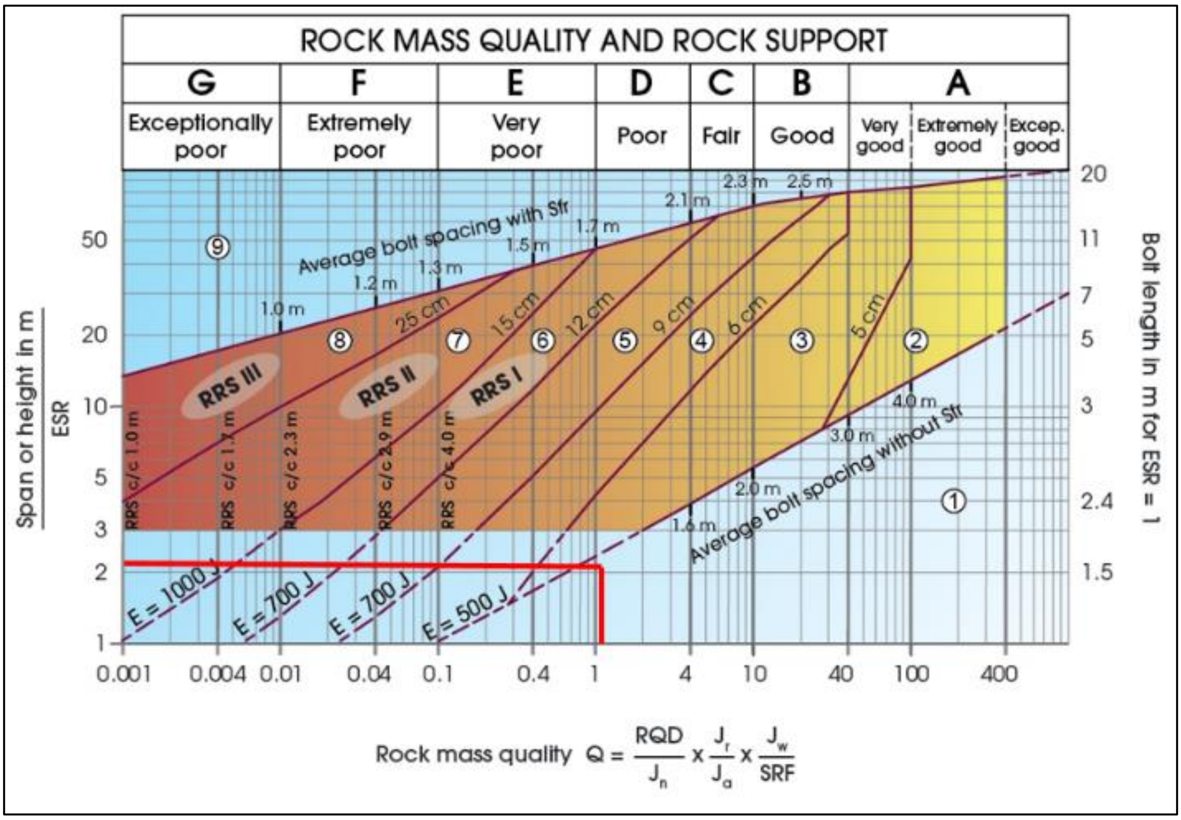


Figure 23. Support for the ANF1 Tunnel is category 1.

ANF2 Tunnel

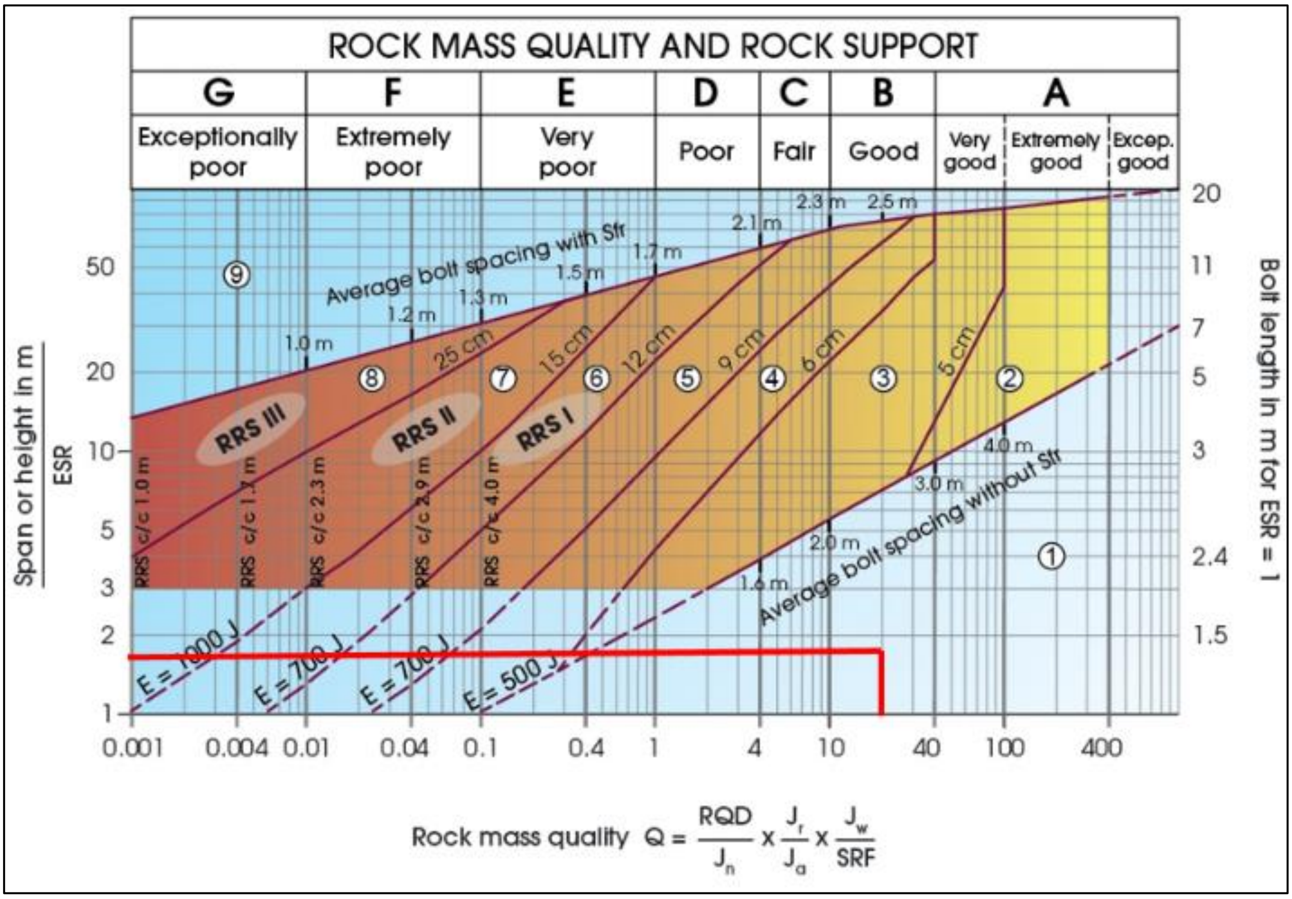


Figure 24. Support for the ANF2 Tunnel is category 1.

- It can be seen that for the 3 tunnels it is necessary to apply support, but now we will take into account the main families of diaclases previously analyzed.
- This analysis will allow us to define the wedges that are formed in each tunnel and be able to assign a type of support for each one, this whole process will be carried out with the support of the Unwedge software.

3.6 Support analysis in Unwedge

- To start the analysis, the software is opened and the project is configured, it is assigned a name and the units with which it will work, in our case metric units will be used and the tension in tons/m².

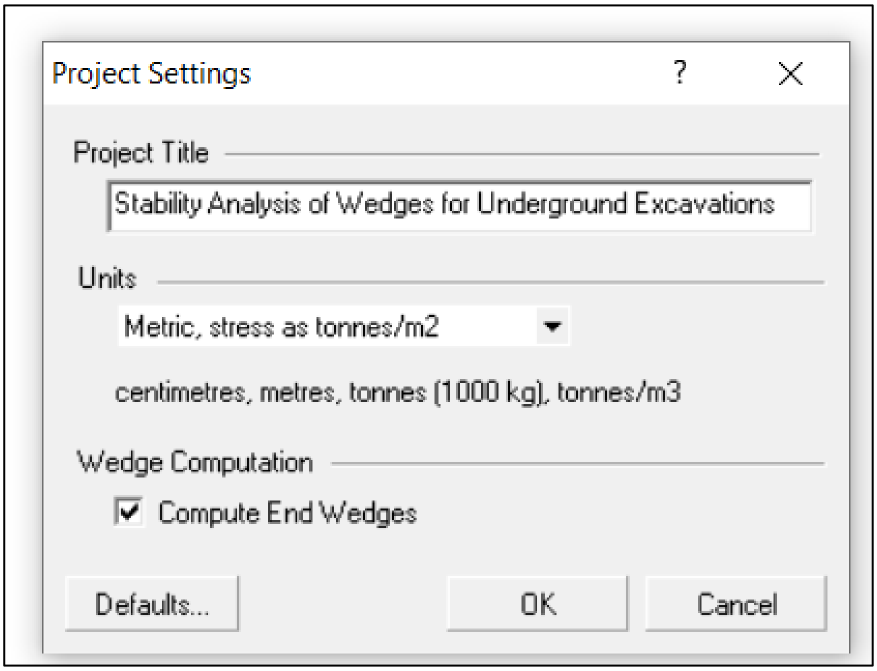
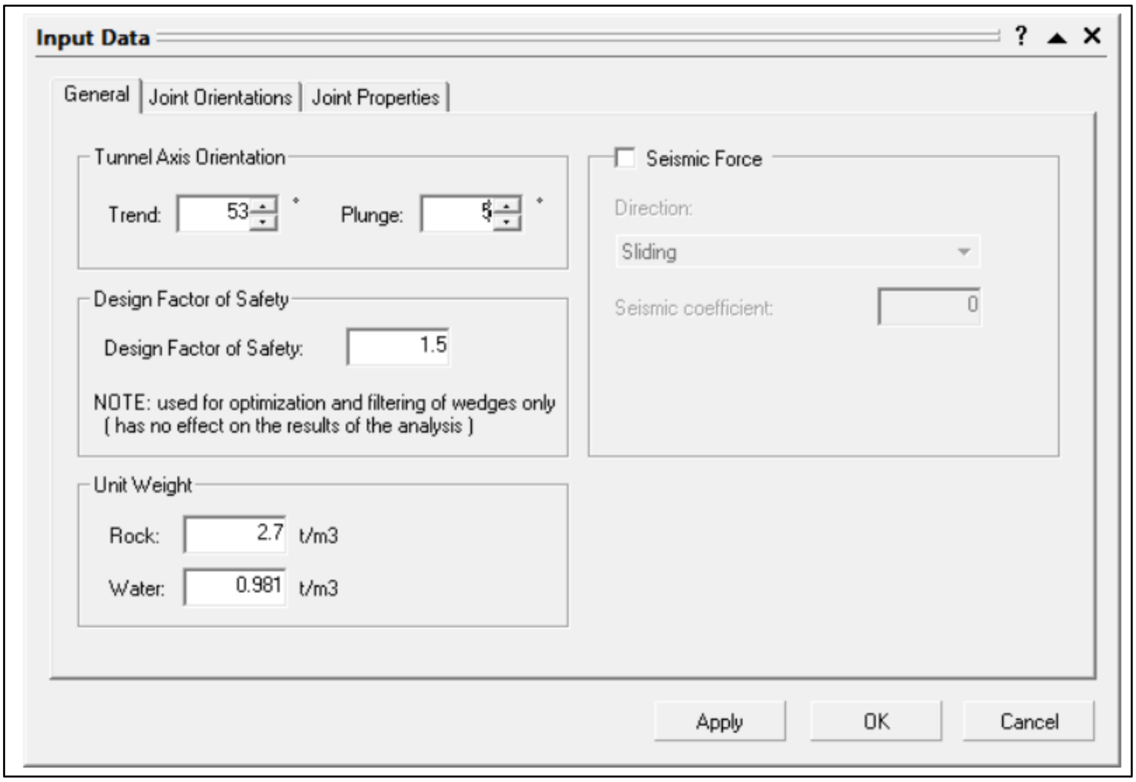


Figure 25. Support for the ANF2 Tunnel is category 1.

- Then we go to the Imput data window, in the general tab the tunnel address (53°) is completed and its plunge (5°) in FS desgin is assigned 1.3, which means that it will be taken as a minimum value for the value of FS, the units of weight for rock and water are not changed.

Figure 26. Support for the ANF2 Tunnel is category 1.

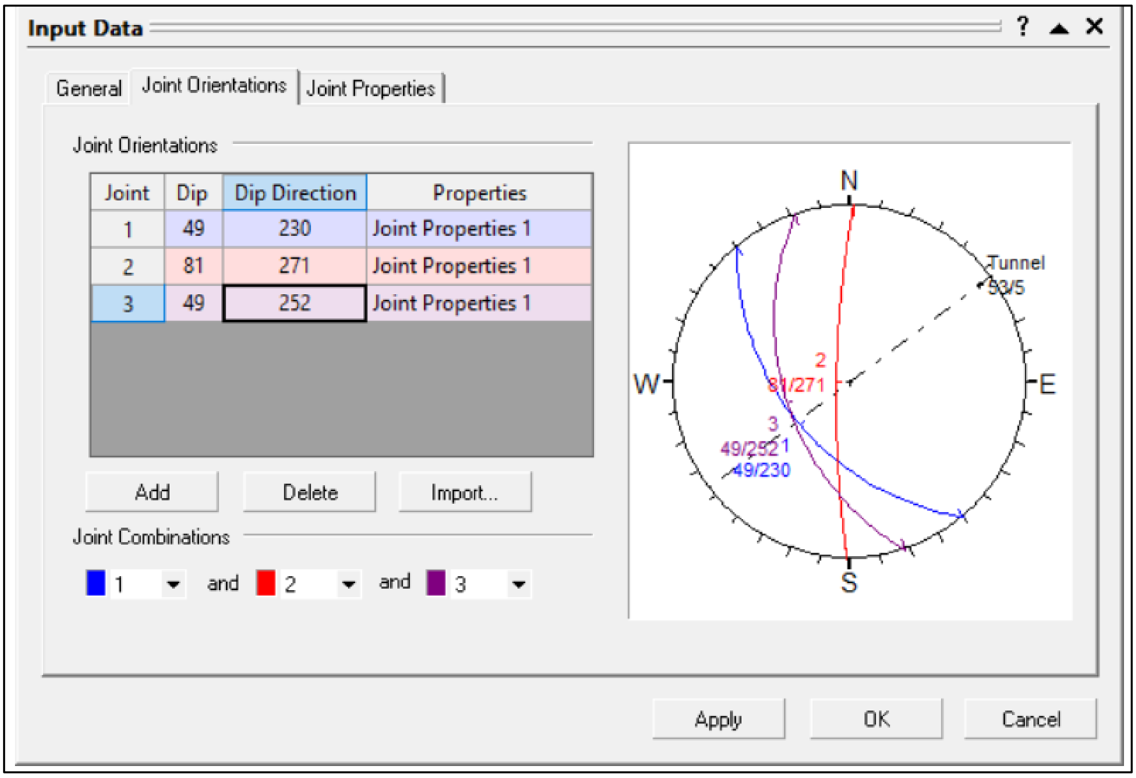


- In the tab joint orientations, we will complete with the data of the families of diaclasas previously identified, in the case of the Veta tunnel, there are 3 families:

Table 23 Dip and Dip direction for the joints in Veta

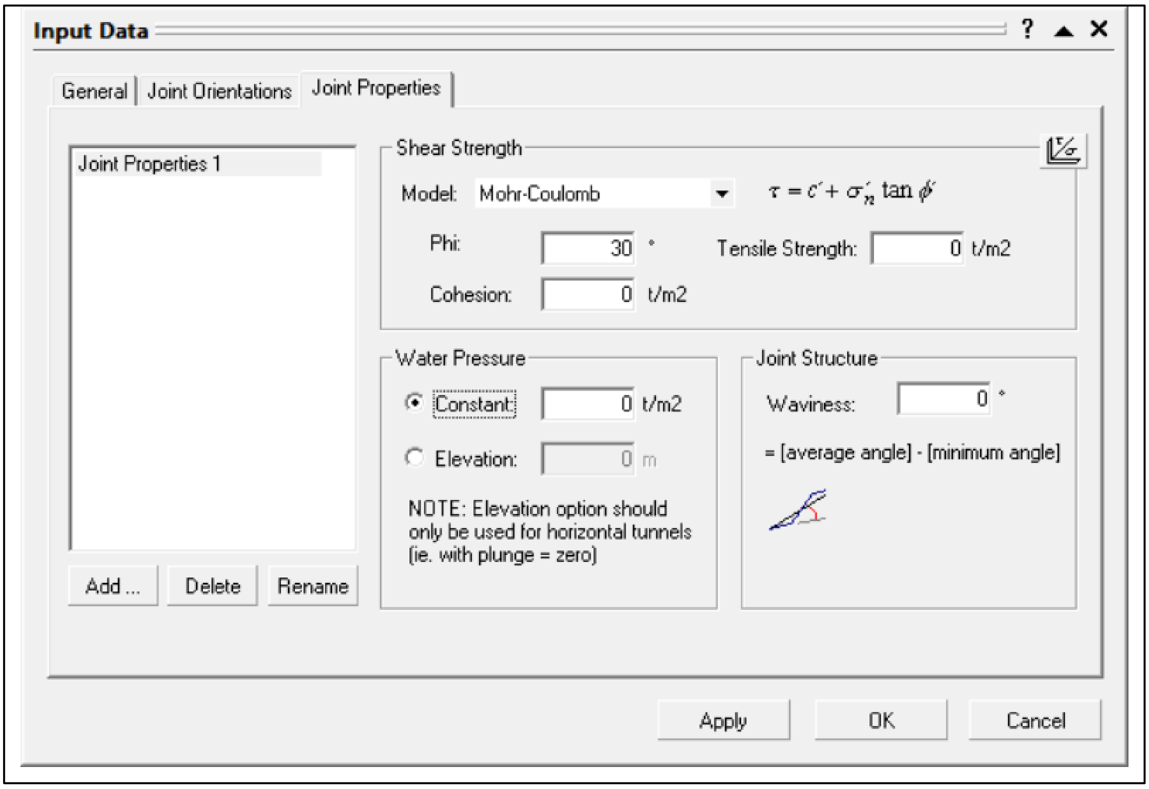
Joint	Dip	Dip Dir
F1	49	230
F2	81	271
F3	49	252

Figure 27. The structural data of the joints in the vein were uploaded.



- In the Joint properties tab, the model to be used will be that of Mohr-Coulomb and the parameters to be used will be the values shown in the image.

Figure 28. For shear strength, phi is 30º and cohesion is 0 t/m².

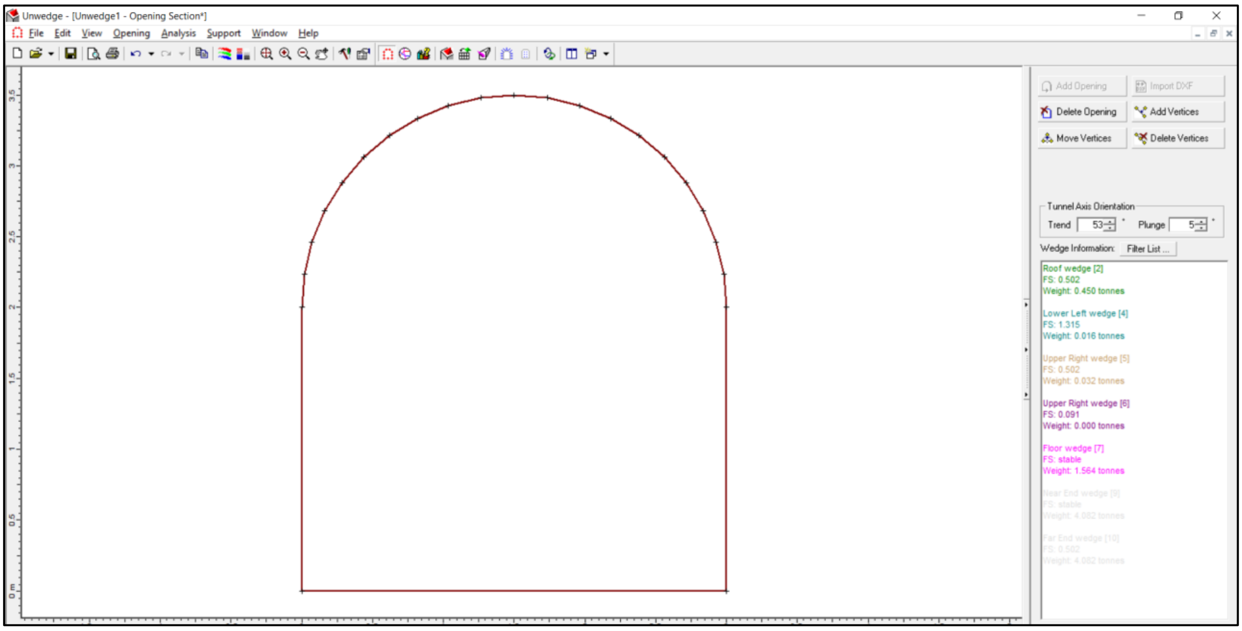


- Once the imput data window is completed, the tunnel profile where the joints will be applied will be drawn, again we will use the following table for the tunnel height and width data.

Table 24. Geometric values for the tunnel's construction.

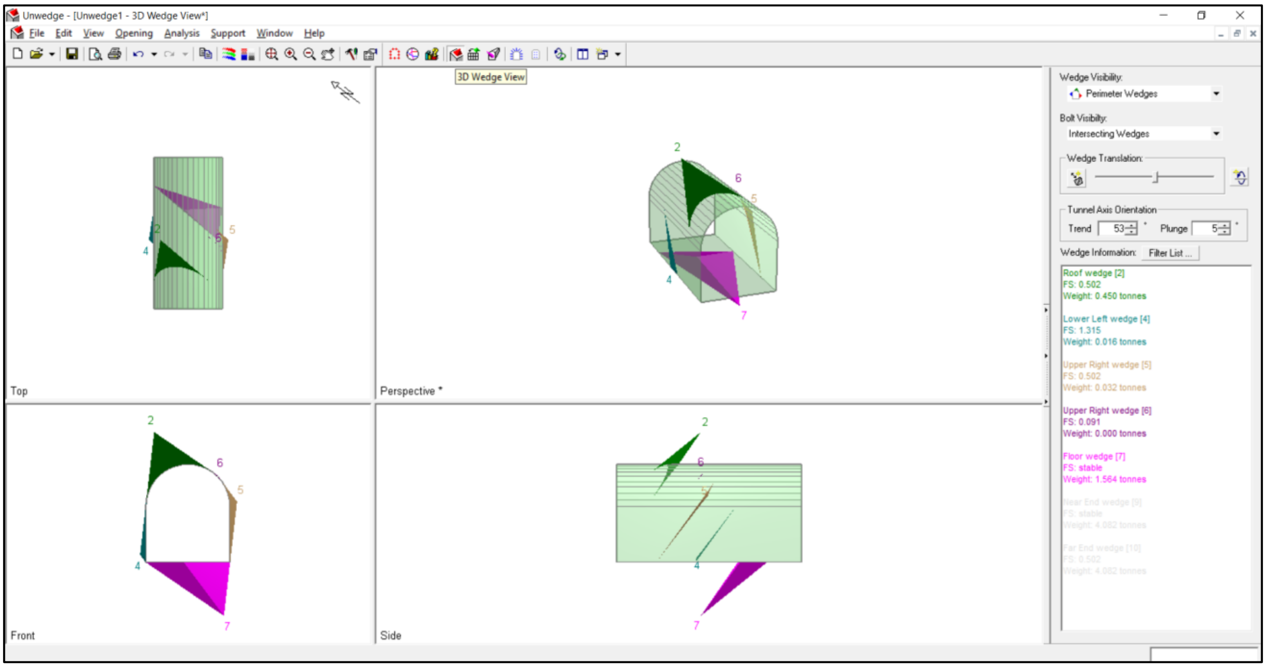
	Azimuth	High	Width	Q	ESR	Alt/ESR
Vein Tunnel	53	3	3	7.5	1.6	1.875
ANF1 Tunnel	86	3.5	3	1.5	1.6	2.1875
ANF2 Tunnel	337	3	3	28	1.6	1.875

Figure 29. Draw of the tunnel.



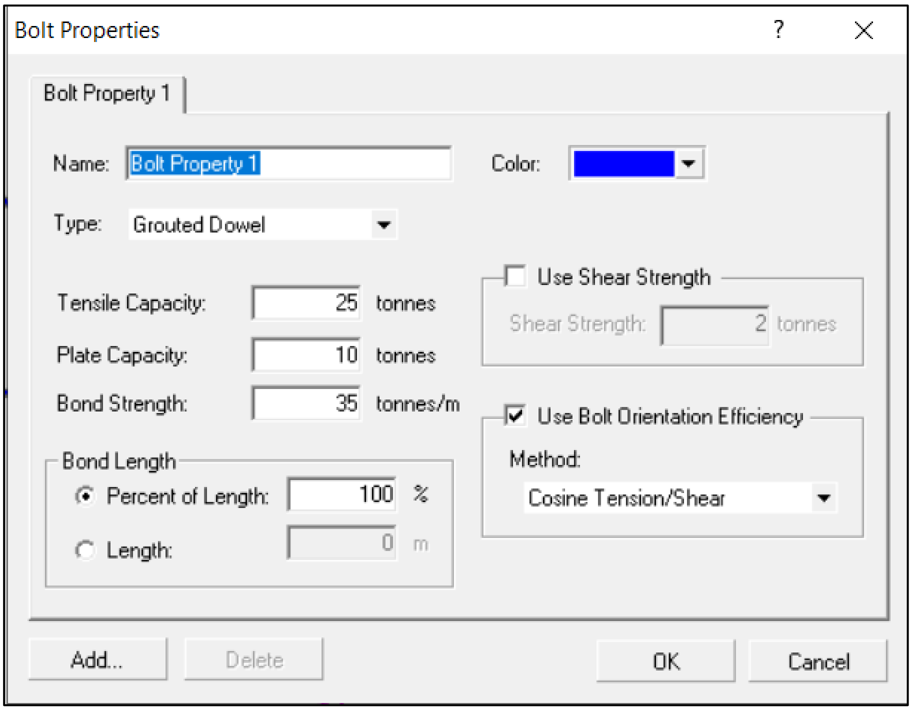
- Once the tunnel is plotted, we go to the 3D wedge view option, which will allow us to observe the wedges that are formed from the families of discontinuities.

Figure 30. Wedges formed by the intersection of the main joints' families.



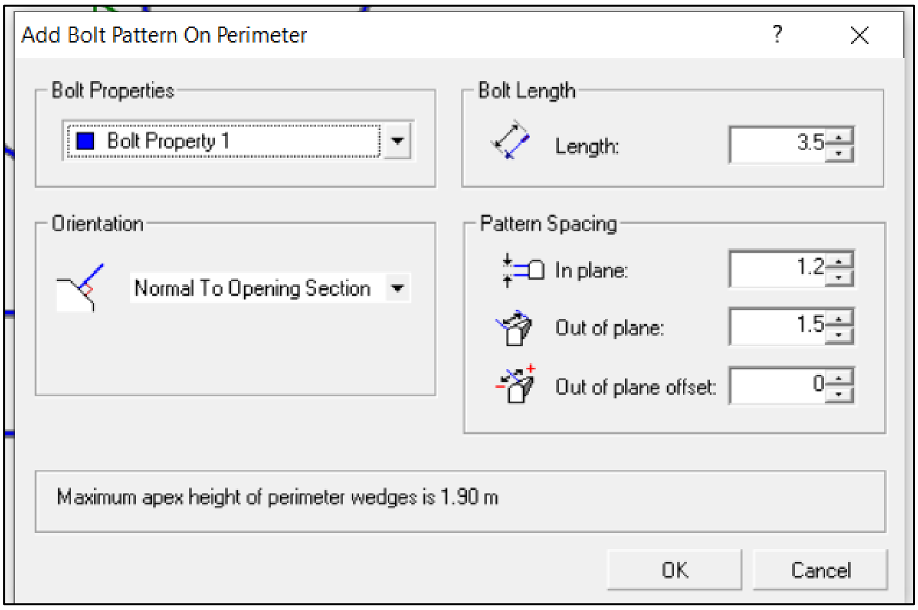
- It is observed that wedges are formed, therefore, we will need to apply a type of support that will depend on the dimensions and position of the wedge formed.
- We will start applying bolts in a systematic way, for this we go to switch to perimeter support designer <bolt properties, in this window we choose the type of bolt (Split set) and other parameters that are observed in the image:

Figure 31. Applying bolts for the sustenance of the tunnel.



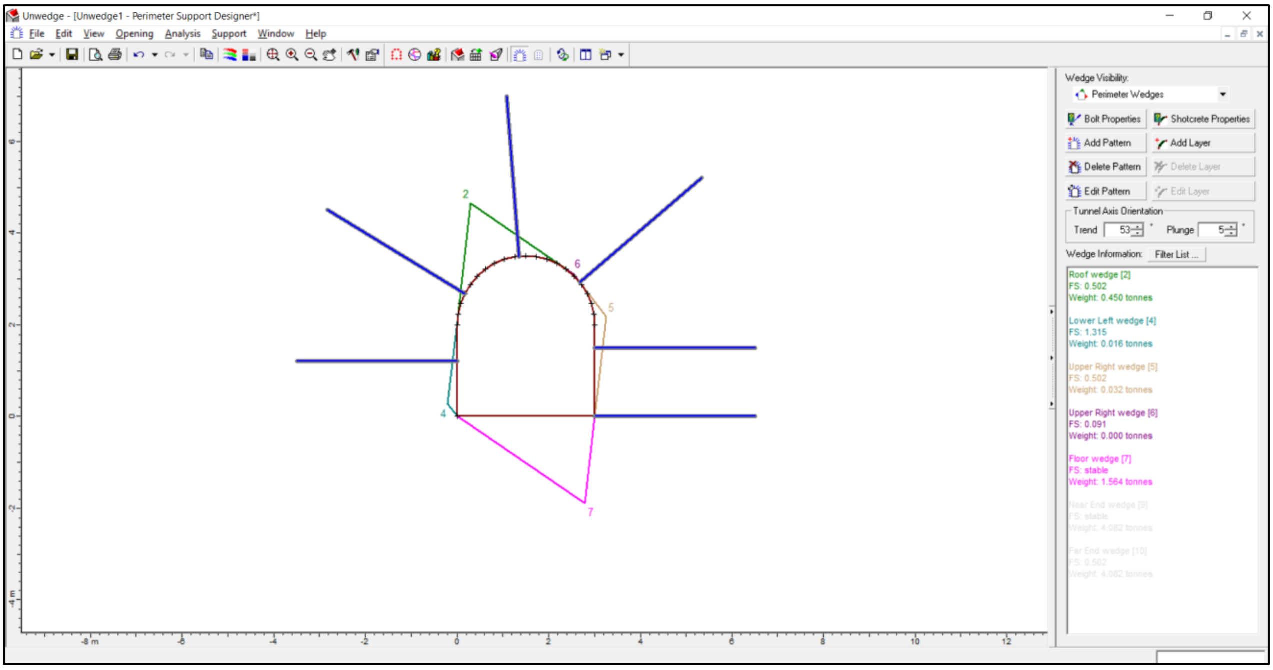
- To add a bolt pattern to the tunnel, we go to Add bolt pattern, there we indicate the bolt length and the bolt spacing.

Figure 32. Values for bolt length and the pattern spacing.



- Finally, we draw the outline of the tunnel that will be occupied by the bolts and a result like this is obtained:

Figure 33. Location of the bolts around the tunnel.



- Once this process is finished, it is observed that the FS of several wedges is still below 0, so a coating with a shotcrete layer will be needed, for this we are going to shotcrete properties, in this window we configure the thickness values and shear strength, for these values it is preferable to take the lowest possible but that at the same time provide us with an FS greater than 1.3 approximately.

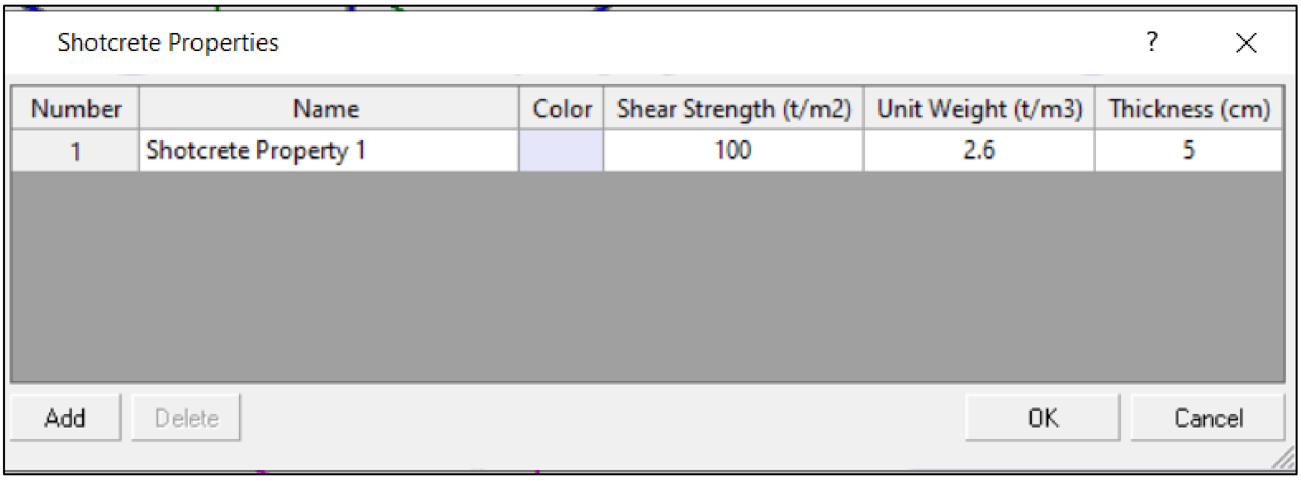
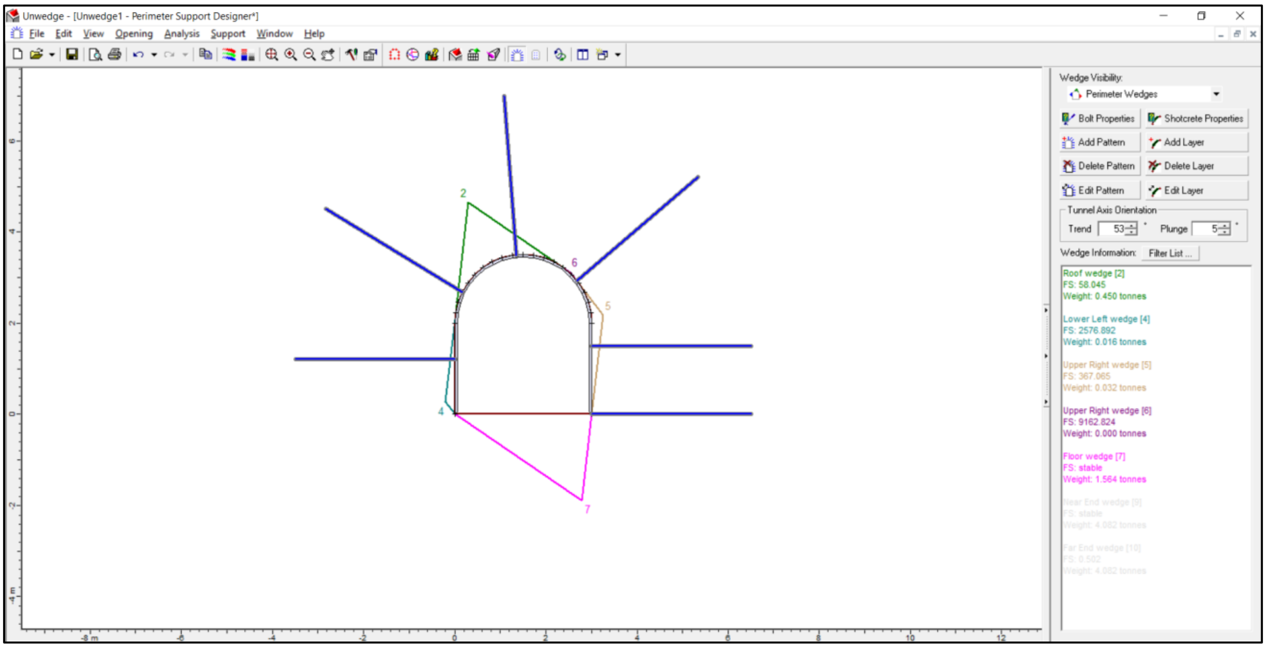


Figure 34. Values for applying shotcrete around the tunnel.

- Once the shotcrete is configured, we will add a layer with the option Add shotcrete layer, we mark the surface that we want to cover and accept.

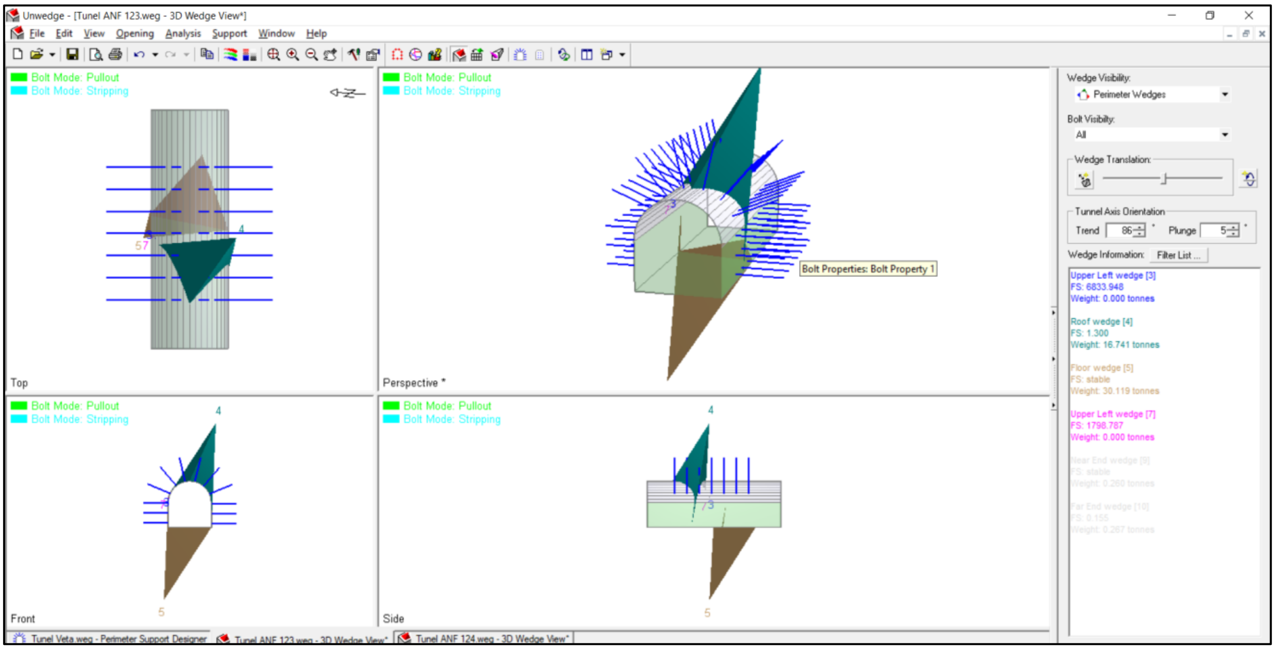
Figure 35. Location of the shotcrete around the tunnel.



- It can be seen that the FS of all wedges increased considerably, which indicates that the tunnel is stable.
- The previous case included 3 families of discontinuities, but the next 2 cases (Tunnels ANF1 and ANF2) include 5 families of discontinuities, which 3 in 3 will form up to 10 different types of wedges; The support analysis for these cases is similar, the difference is that the support applied to each tunnel must stabilize the 10 cases that are formed.
- The results obtained for the last 2 cases were:

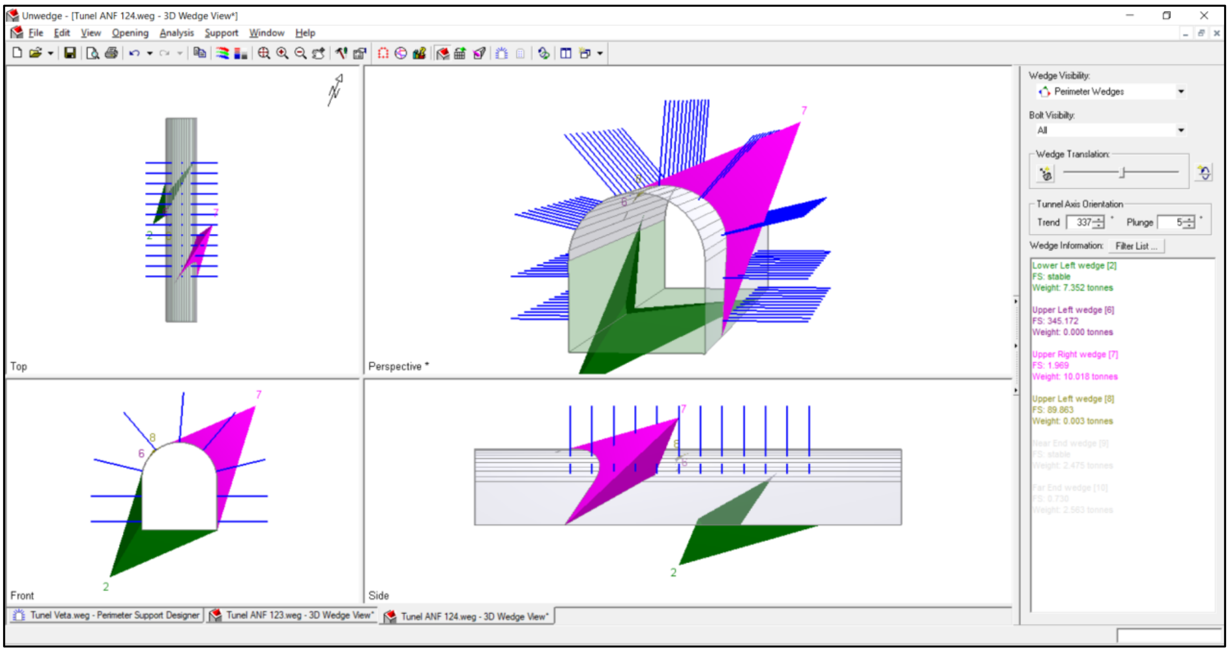
ANF1 Tunnel

Figure 36. Sustenance for the ANF1 tunnel.



ANF2 Tunnel

Figure 37. Sustenance for the ANF2 tunnel.



4. DISCUSSION OF RESULTS

4.1 Conclusions

- The use of the software (Leapfrog, Dips and Unwedge) facilitated the realization of the project, due to the speed with which the data is processed and the different views that it provides us with the designed models, these views allowed us to perform a more analysis exhaustive in all directions and to be able to relate lithologically, strategically and geometrically the data that were provided to us.

- Barton's Q classification is generally used to get an idea of the support to be used in a tunnel, but this concept must be accompanied by the additional support that should be added when families of discontinuities that directly affect the tunnel are present, within this tunnel. context, the use of Leapfrog software allowed us to assign values of Q to different blocks through which tunnels passed, from this idea the use of Unwedge software was very useful, because it allowed us to define the support to be used to stabilize the wedges that were formed due to the discontinuities analyzed in the structural model.

- The methodology used is quite useful, because it combines different factors that affect the stability of a tunnel, for example factors of the lithological, structural and geomechanical type, for each factor a different model was made and in the end all were unified providing us A clear idea of the area studied.

4.2 Recommendations

- To carry out a more complete study, it would be very helpful to corroborate the research with a field work that allows us to physically identify the characteristics of the area to be studied, for practical reasons the information used in this research.
- The Mohr-Coulomb model was used to calculate the shear strength, within this model the parameters of cohesion and friction angle will be used, these parameters were assumed as 0 and 30 respectively, but for a more exhaustive study, they can use correlations between the RMR with cohesion and friction that would allow us a more thorough analysis.

References

[1] Leon, C (2017) “Evaluación Geomecánica del Macizo Rocoso en la Unidad Minera Bateas - Nivel 6 Veta Ánimas”, Lima.

[2] L. González de Vallejo, M. Ferrer and L. Ortuño, Carlos Oteo (2002) INGENIERÍA GEOLÓGICA. PEARSON EDUCACIÓN, Madrid.

[3] Hatzor, Y. and Goodman, R.E. 1992. Application of block theory and the critical key block concept in tunneling; two case histories. In Proc. Int. Soc. Rock Mech. conf. on fractured and jointed rock masses, Lake Tahoe, California, 632-639.

[4] Quevedo, D (2018) “MODELAMIENTO GEOMECÁNICO DE LA VETA ÁNIMAS EN MINA CAYLLOMA – AREQUIPA”, Lima

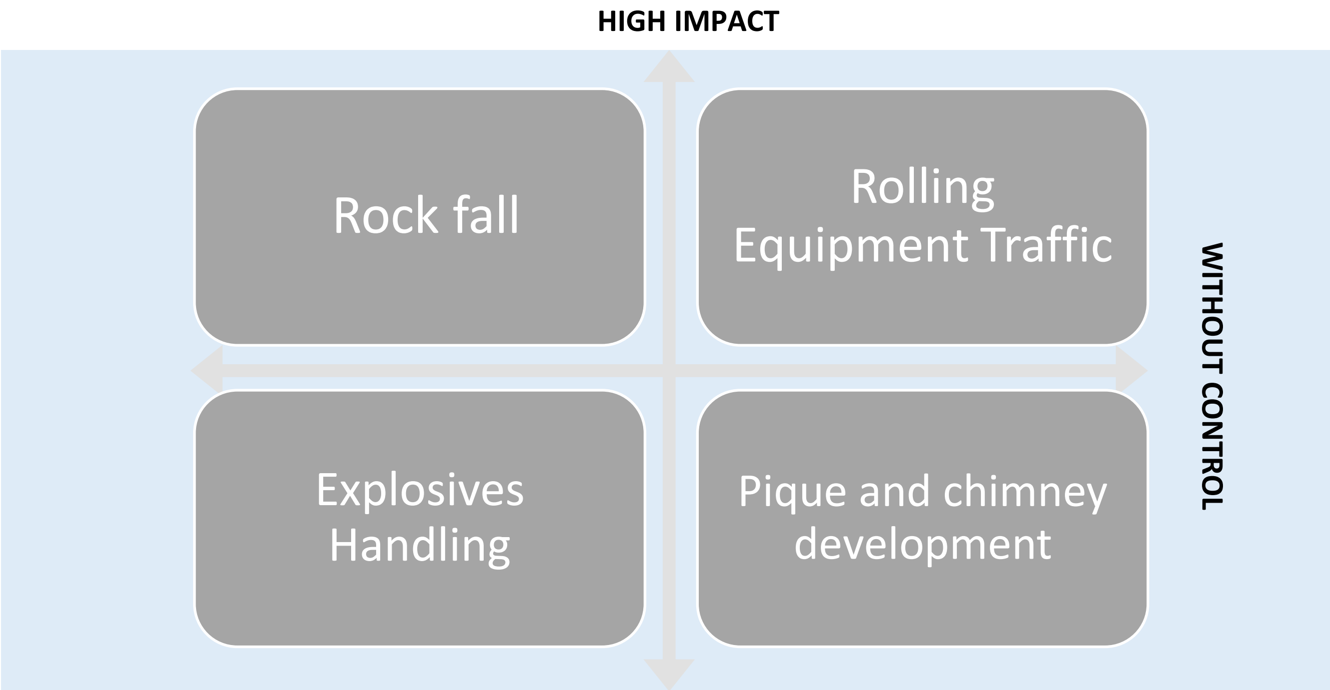
[5] Tyler, D.B., Trueman, R. and Pine, R.J. 1991. Rockbolt support design using a probabilistic method of key block analysis. Proc. 32nd U.S. Symp. Rock Mechanics, Norman, Oklahoma, 1037-47.

[6] Decreto Supremo N° 023-2017-EM, Art. 213-214

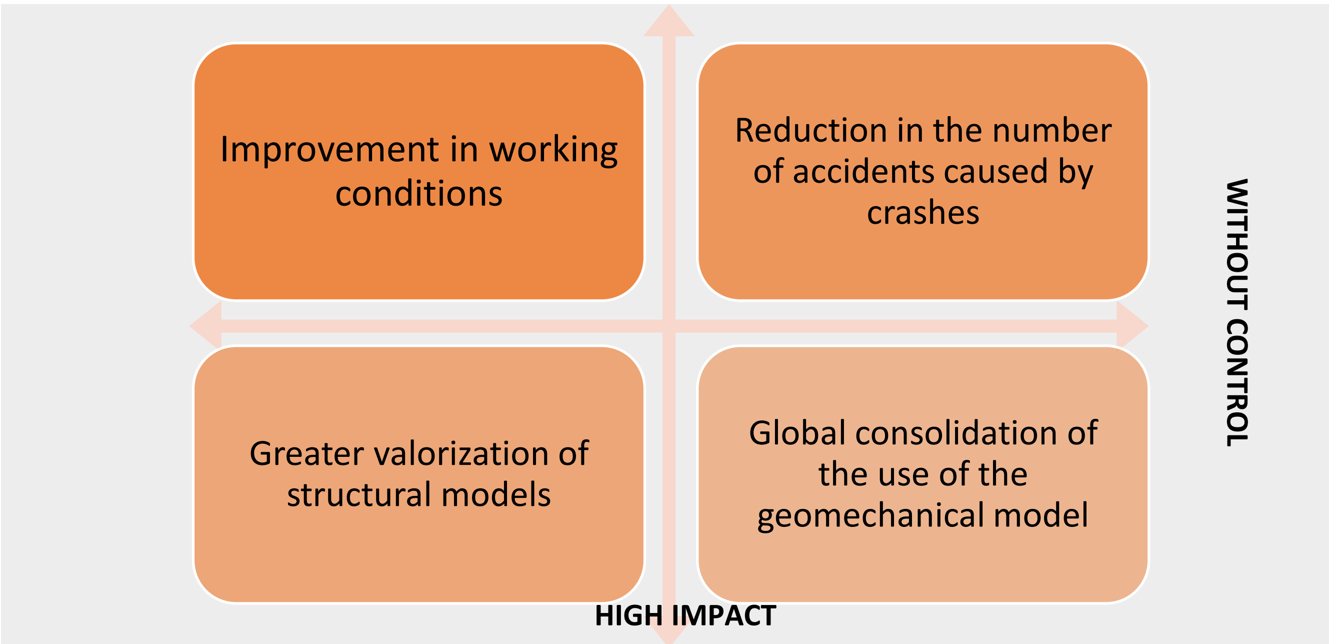
[7] Decreto Supremo N° 024-2016-EM

4.4 Appendix
4.4.1 Appendix 1: Prioritization of Risks and Opportunities

Risk Prioritization



Prioritization of opportunities



4.4.2 Appendix 2: Permission Management

For this management the following regulations are shown

a) Pre-operation permits
   - Decreto Supremo N° 024-2016 – Aprueban Reglamento de Seguridad y Salud Ocupacional en Minería. [7]
   - Decreto Legislativo N° 662
   - Decreto Legislativo N° 757
   - Decreto Supremo N° 024-93-EM – Aprueban Reglamento del Título Noveno de la Ley General de Minería, referido a las Garantías y Medidas de Promoción a la Inversión en la actividad minera.

b) Operation permits
   - Ley N° 30299 – SUCAMEC
   - Decreto Supremo N° 023-2017 – MINAM
     - Titulo cuarto/ Gestion de las operaciones mineras:

       ART. 213.- En la ejecución de las labores mineras horizontales, inclinadas o verticales y otras, se procederá a su sostenimiento sistemático inmediato, sobre la base de los estudios geomecánicos, antes de continuar las perforaciones en el frente de avance, aplicando el principio de “labor avanzada, labor sostenida”, en lo que sea aplicable.

       ART. 214.- En las etapas de exploración y explotación, - incluida la preparación y desarrollo de la mina-, el titular de actividad minera debe tener en cuenta:

       a) Que, de acuerdo al estudio geomecánico efectuado, el plan de minado debe considerar las condiciones más desfavorables de la masa rocosa del depósito mineralizado, para elegir el método de explotación de menor riesgo que permita la seguridad de los trabajadores y maquinarias, así como: una alta recuperación del mineral, la estabilidad de las excavaciones y la buena productividad.

       b) Registrar mensualmente los ensayos y pruebas de control de calidad, respecto de no menos del uno (1 %) del sostenimiento aplicado en dicho periodo.

       c) Registrar el monitoreo por estallido de rocas en base a la frecuencia de reportes de incidentes de este tipo, y en base a las labores sometidas a altas presiones por carga litostática.

       d) Los PETS relativos a temas geomecánicos deben incluir los materiales y estándares de acuerdo al trabajo realizado y deben ser actualizados por el área de Geomecánica de acuerdo al cambio de las condiciones geomecánicas de las labores.

e) Que, durante la ejecución del plan de minado debe establecerse una relación de comunicación técnica y profesional entre las áreas de geología, geomecánica, mina y el Gerente de Seguridad y Salud Ocupacional. Dicha comunicación debe permanecer durante todo el proceso de explotación, a efectos de prevenir el desprendimiento de rocas, especialmente cuando se atraviesa zonas de gran perturbación estructural.

f) Que los avances de las labores mineras no deben exceder lo establecido en el plan mensual de minado, salvo modificación previa del mismo.

g) Que se mantenga el ancho y la altura de los tajeos dentro de los parámetros establecidos en los cálculos de la geomecánica desarrollados para cada unidad de operación.

h) Que el diseño de la sección y gradiente de las galerías y otras labores tengan en cuenta las características estructurales del macizo rocoso, sus propiedades geomecánicas, la utilización que tiene, y los elementos de servicio (agua, aire comprimido, cables eléctricos, ductos de ventilación) requeridos.

i) Que todas las galerías y otras labores cuenten con refugios peatonales cada cincuenta metros (50 m) y las galerías principales de transporte cuenten, además, con áreas de cruce de los equipos motorizados con sus respectivas señalizaciones y/o semáforo.

j) Que, en tramos de ciento cincuenta (150) a doscientos (200) metros, se construya accesos laterales adicionales o cruces para los vehículos considerando el vehículo más grande de la mina para facilitar el pase de los vehículos de ida y vuelta, considerando además un área necesaria para la construcción de cunetas para casos de drenaje o deshielo.

k) Para la ejecución de las operaciones mineras subterráneas y superficiales, el titular de la actividad minera debe acreditar que cuenta con la asesoría de un profesional ingeniero, especializado y con experiencia en geomecánica, para cada Unidad Minera o Unidad de Producción. [6]

- Decreto Supremo N° 014-92-EM - Aprueban el Texto Único Ordenado de la Ley General de Minería

c) Mining inspection

- Ley N° 27474 – Ley de Fiscalización de las Actividades Mineras
- Decreto Supremo N.º 082-2002-EF

4.4.3 Appendix 3: Mining Industry Standards and Guides

Standards

- Barton Q index geomechanical classification
  The Barton Q classification is one of the most commonly used geomechanical classifications in rock massifs along with the RMR classification of Bieniawski. **[2]**
  Both are widely used, however, the RMR is normally used more as a geomechanical index for the evaluation of rocky massif properties while the Q index is mostly used in the evaluation of tunnel support by bolts, trusses, shotcrete, etc.
  The Q classification was developed in 1974 by Barton, Lunde and Lien based on information from numerous tunnels. Subsequently it has been reviewed several times. The one contained in this post is the Q Barton Classification, 2000.
  This geomechanical classification allows estimating geotechnical parameters of the rock mass and, more importantly, design supports for tunnels and other underground excavations.
  The Q index varies between 0.001 and 1000, classifying the rock mass as:
    - 0.00 and 0.01: exceptionally bad rock
    - 0.01 and 0.1: Extremely bad rock
    - 0.1 and 1: Very bad rock
    - 1 and 4: Bad Rock
    - 4 and 10: Middle Rock
    - 10 and 40: Good rock
    - 40 and 100: Very good rock
    - 100 and 400: Extremely good rock
    - 400 and 1,000: exceptionally good rock

  It is calculated using 6 geotechnical parameters according to the following expression:

  $$Q = (RQD / Jn) \cdot (Jr / Ja) \cdot (Jw / SRF)$$

  The three terms of Barton's Q expression represent the following:

  (RQD / Jn): Block size
  Jr / Ja): the cut resistance between the blocks
  (Jw / SRF): influence of the tension state

  From the score obtained in each block we can know which term has more or less weight in the evaluation of the Q index and therefore its influence on the quality of the rock mass.
- RMR geomechanical classification
  The RMR geomechanical classification allows obtaining a quality index of the rock mass from resistance of the intact rock, degree of fracturing and diaclassing of the discontinuities of the massif, presence of water and the orientation of the discontinuities with respect to the study element: tunnel, slope or foundation. **[2]**

The RMR index ranges from 15 to 100 points from which the rock mass can be classified into 5 categories.

The geomechanical parameters that influence the RMR index are:

Matrix Rock Resistance
It is measured from the simple compression break test of rock witnesses or from the point loading test. The score ranges from 0 to 15 points depending on the resistance of the rock.

RQD
It assesses the degree of fracturing of the massif according to the universally known RQD from 3 points for a RQD value of less than 25% to 20 points for a RQD value of more than 90%.

Separation between diaclases
As the statement says, punctuate the spacing between discontinuities. The score reaches values of 20 points for diaclases separated more than 2 m and a minimum value of 5 for diaclases spaced less than 6 cm.

Diaclases state
It allows to punctuate the state of the diaclases through persistence or length of the discontinuities, opening, roughness, presence of filling and alteration of the joints. The maximum value is 20 points while the minimum is 0 points.

Presence of groundwater
It measures the water leaks in the massif, the water flow and the humidity present in the discontinuities. The score reaches a value of 15 for a dry rocky massif and a value of 0 for when water is flowing between the joints with a flow greater than 125 l / min or the water pressure / main tension ratio is greater than 0, 5.

- International Organization for Standardization
  ISO 14001- "Environmental Management System"
  It is an internationally accepted standard that indicates how to put an effective environmental management system in place. It is designed to help organizations stay commercially successful without ignoring their environmental responsibilities. The ISO 14001 standard is applied in the environmental risk assessment nuance.
- Global Mining Guidelines Group
  Global Mining Guidelines Group (GMG) is a network of representatives from mining companies, OEMs, OTMs, research organizations, consultants and regulators from around the world who collaborate to face the challenges facing the mining industry. The guide for the Implementation of Autonomous Systems in Mining was used
- Friction Angle
  It is a property of granular materials which has a simple physical interpretation, as it is related to the angle of repose or maximum angle possible for the slope of a set of said granular material. In any granular material, the angle of repose is determined by the friction, cohesion and shape of the particles but in a material without cohesion and where the particles are very small in relation to the size of the set, the angle of repose coincides with the internal friction angle.

It is important to determine the stability of slopes, the strength of a foundation or for the calculation of the earth thrust.

- Cohesion
  It is the attraction between particles, caused by molecular forces and water films. Therefore, the cohesion of a soil will vary if its moisture content changes. Cohesion is measured kg / cm2. Clay soils have high cohesion of 0.25 kg / cm2 to 1.5 kg / cm2, or more. The silty soils have very little, and in the sands the cohesion is practically nil.
  When it is added to the earth, even the one with a high sand content, water plays a fundamental role in soil cohesion, due to surface tension. That property provides a weak bond between the grains of land to cause cohesion. Minerals such as salt and caliche favor the cohesive properties of the soil. Long-term pressure can also increase cohesion. Although water plays a fundamental role in the cohesion process, if the amount is excessive, it can cause the land to lose this property. The soil loses its plasticity and becomes almost liquid.

Guides

- The Mining Association of Canada
  Since 1935, the Canadian Mining Association (MAC) has been the national voice of the Canadian mining industry. MAC promotes the industry nationally and internationally, works with governments on policies that affect the sector and educates the public about the value that mining brings to the economy and daily life of Canadians. The members of this association represent the majority of Canadian production of basic and precious metals, uranium, diamonds, metallurgical coal, extracted oil sands and industrial minerals, and are actively involved in mineral exploration, mining, smelting, refining and semi-manufacturing.
- JORC 2012: Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves
  The JORC Code is an Australian code for Reporting on Mineral Resources and Reserves and Reserves, it establishes minimum standards, recommendations and standards for Public Information on exploration results, Mineral Resources and Mineral Reserves in Australia. This has been written by the Joint Reserve Committee of Mena of "The Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia". The Joint Mineral Reserve Committee was established in 1971 and published several reports making recommendations on the classification and Public Information of Mineral Reserves before the first disclosure of the JORC Code in 1989.
- National Instrument 43-101
  NI 43-101, is the national instrument for the Mining Project Disclosure Standards, which are owned or explored by companies that report these results in Canadian stock markets. This includes foreign-owned mining entities that are listed on stock exchanges supervised by the Canadian Securities Administrators, even if they only operate in over-the-counter derivatives or other instrumented securities.

4.4.4 Appendix 4: Support Application

4.4.4.1 Shotcrete

The shotcrete is reinforced with steel fibers or electro welded mesh; The foregoing is determined by the type of excavation and the construction process adopted; the two reinforcements with steel fiber due to the need for a quick application and that this be of a single launch; if the availability of time permits the site conditions are appropriate, it could be reinforced with electro-welded mesh.

-The application process of the concrete launched begins with the profiling of the surface to be protected; Depending on the type of rock and the desired finish, this profiling is executed at the time of excavation or after it in processes such as manual or mechanical combs with back-hammer.

-The surfaces either of bare rock or previously coated with shotcrete, which are to receive shotcrete, are cleaned of loose or loose material, dust, mud or any other material that contaminates or decreases the adhesion between the concrete and the surface.

-The cleaning is done with high pressure air and water jets or by any other method that shows similar results.

-The method and cleaning operations chosen are carried out in such a way as to avoid loosening, cracking or fragmenting the surface that will receive shotcrete.

-The surfaces are kept moistened from the moment the cleaning is finished until when the concrete thrown is applied.

4.4.4.2 Bolts

Bolt placement: in the case of bolts that are used in places where the use of reinforcement mesh is required, it is attached to the bolts by means of the nuts and plate that are placed with each bolt.

-The bolt nuts are located in such a way that, at least the first 4 cm of the thread are free after the installation of the bolt and any steel mesh supported by it.

The bolts are tensioned in a short time after installation; said tensioning is carried out with a controlled torque impact wrench, so that the bolt takes a tension of not less than 50% and not more than 90% of its creep limit.

-Where the rock or concrete surface thrown is not perpendicular to the direction of the hole using beveled or semi-spherical washers on the support plate and a flat washer between these washers and the hexagonal nut.

-The anchoring of type C bolts is obtained by grouting or cement mortar injected into the hole prior to the installation of the metal tube in such a way as to guarantee the filling with grout or mortar of the internal space of the tube and between it and the walls of the perforation.

4.4.5 Appendix 5: Support maintenance

4.4.5.1 When the concrete plant is on the surface

Shotcrete transport with mixer trucks

The determining factor is the distance to the work front. To travel long distances, it is necessary to use additives to stabilize the shotcrete mixture. In addition, more mixers are needed at a greater distance to maintain projection performance.

Therefore, the transport distance has a great impact on the costs of mixing, transport and equipment maintenance. In addition, due to the increase in the mixers fleet, the accesses of the mine are congested, which may have an impact on the normal flow of operations.

Shotcrete transport through a pipe

It requires a higher initial cost and planning in its installation. It is necessary a person who is dedicated exclusively to coordinate the reception and dispatch of the shotcrete inside mine and a perfect communication between outside and inside.

The advantage of this method is the reduction of the fleet of transport equipment and the consequent decongestion of the accesses. When traveling shorter distances, the operating costs of the equipment are lower.

When preparing the shotcrete mixture, additives should be used so that it does not disintegrate or homogenize it again with a mixer truck or using a remixer.

4.4.5.2 When the concrete plant is underground

In this case the materials are lowered dry and mixed in the underground concrete plant.

Truck transport

As the volume of dry material is larger, more trucks are needed to transport them, which increases access traffic.

Transportation through a pipe

It has the same advantages and disadvantages as in the case of the concrete plant on the surface.

4.4.6. Appendix 4: Study Limitations

Assumptions

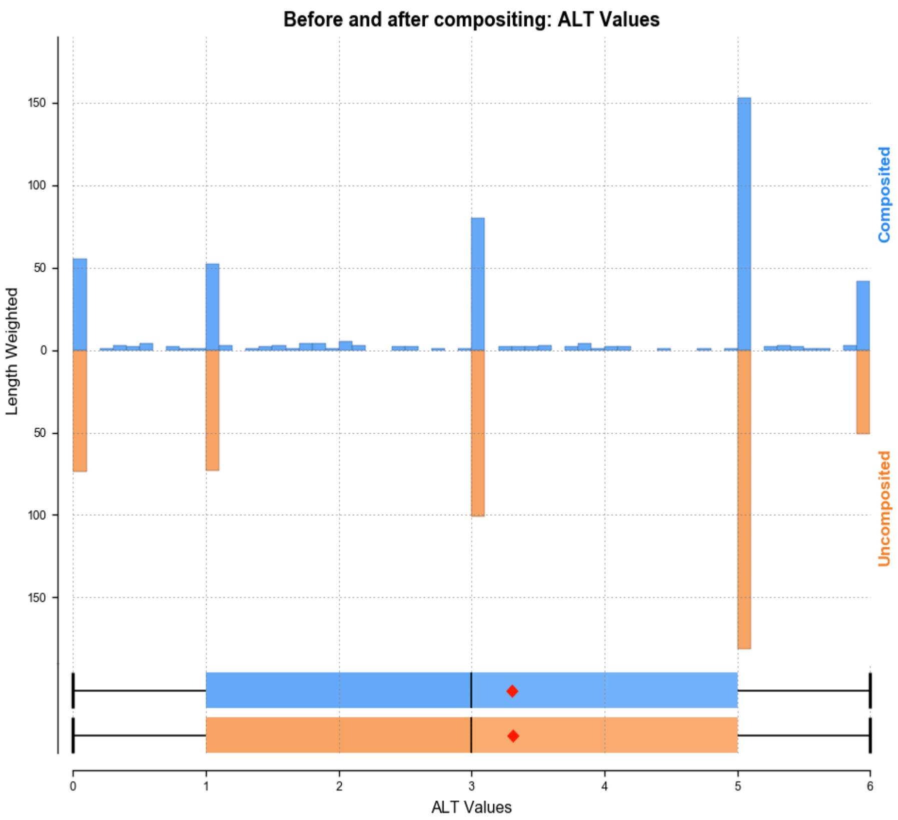
The present work presented some limitations due to the little information available about the project. Not having enough information, some information was assumed in order to develop the conceptual study of the Caylloma project. Among this supposed information are:

- Unique values of cohesion and friction angle were taken at values 0 and 30 ° respectively, because there was not enough information to corroborate values for each lithology.
- Dry conditions were assumed to determine the RMR value, which meant adding a rating of 15, this assumption was considered because there was no hydrogeological information that allowed us to ensure water conditions.
- Initially, the lithological data provided us with information on 19 different lithologies, but because most of them were practically negligible, they were not taken into account, and this amount of lithology was reduced to 4 main ones.

Participation of guide teachers

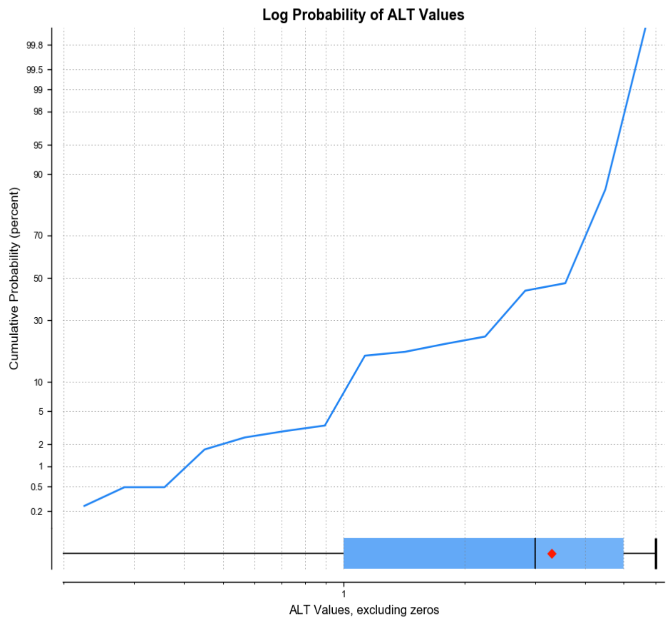
This project was carried out with the help of the teacher Jose Gutierrez Ramirez, responsible for the course, as well as the theoretical part and the practical part. It is a great help in carrying out the conceptual study of the Caylloma project.

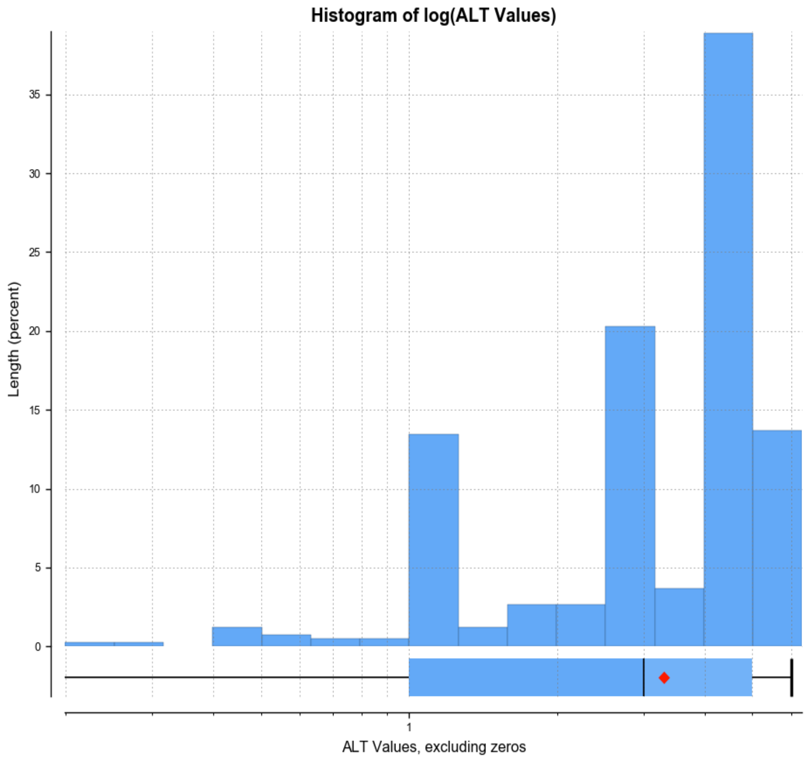
This led to an educational and professional development of the team responsible for preparing the conceptual study of the Caylloma project.

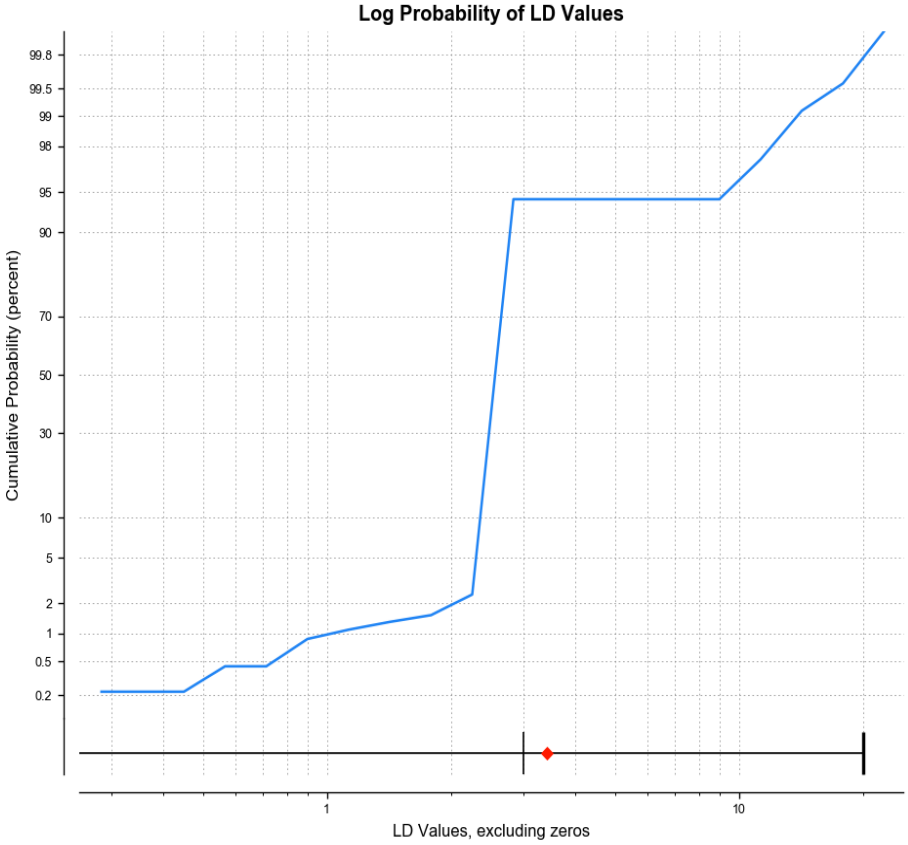


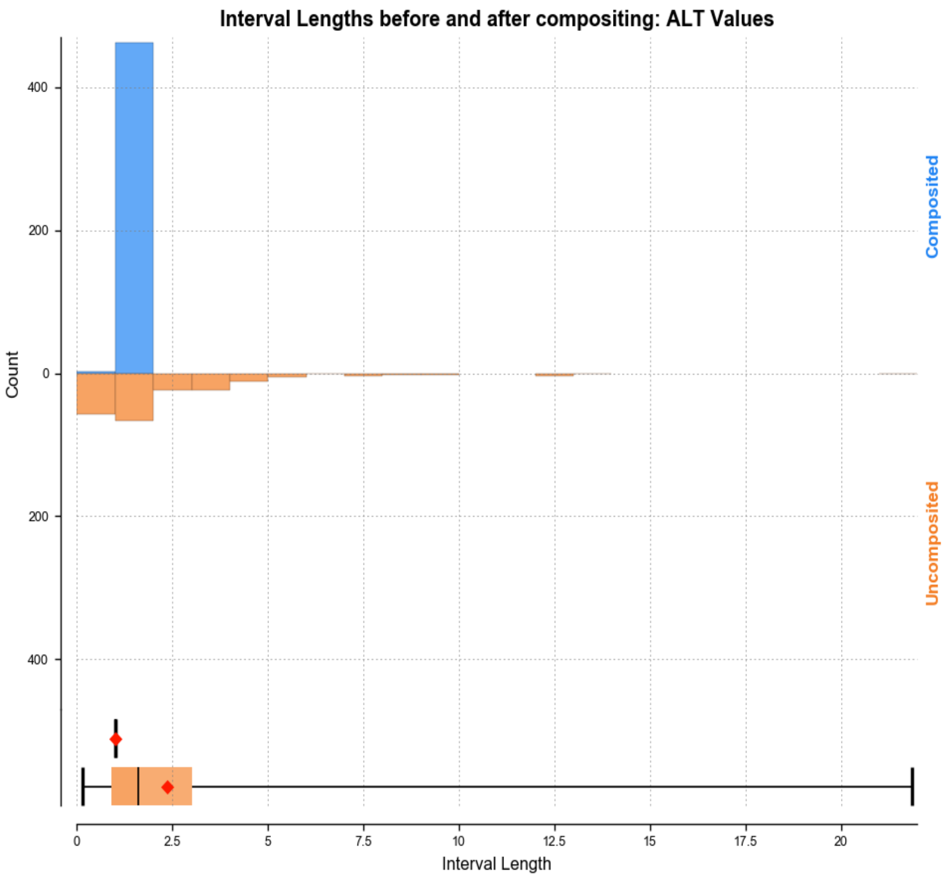
Graphics

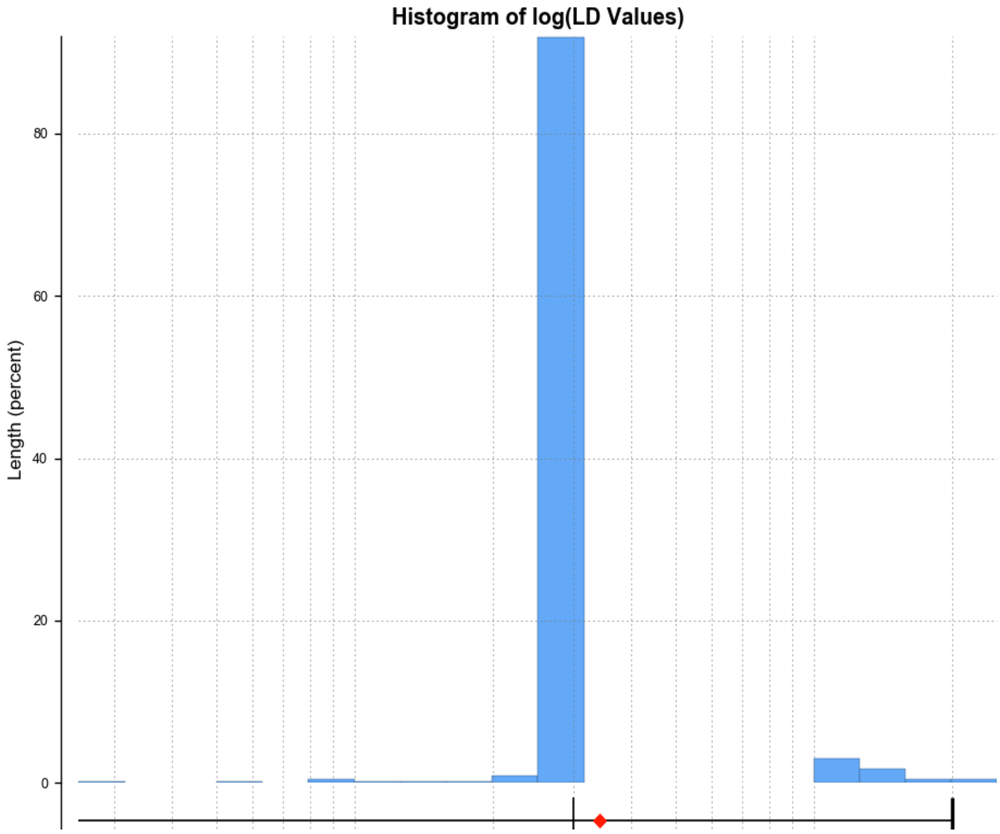
Below are the graphs Log probability, Histogram and comparative of composited and non-composited values for quantities and spacing; for each of the factors that make up the RMR.

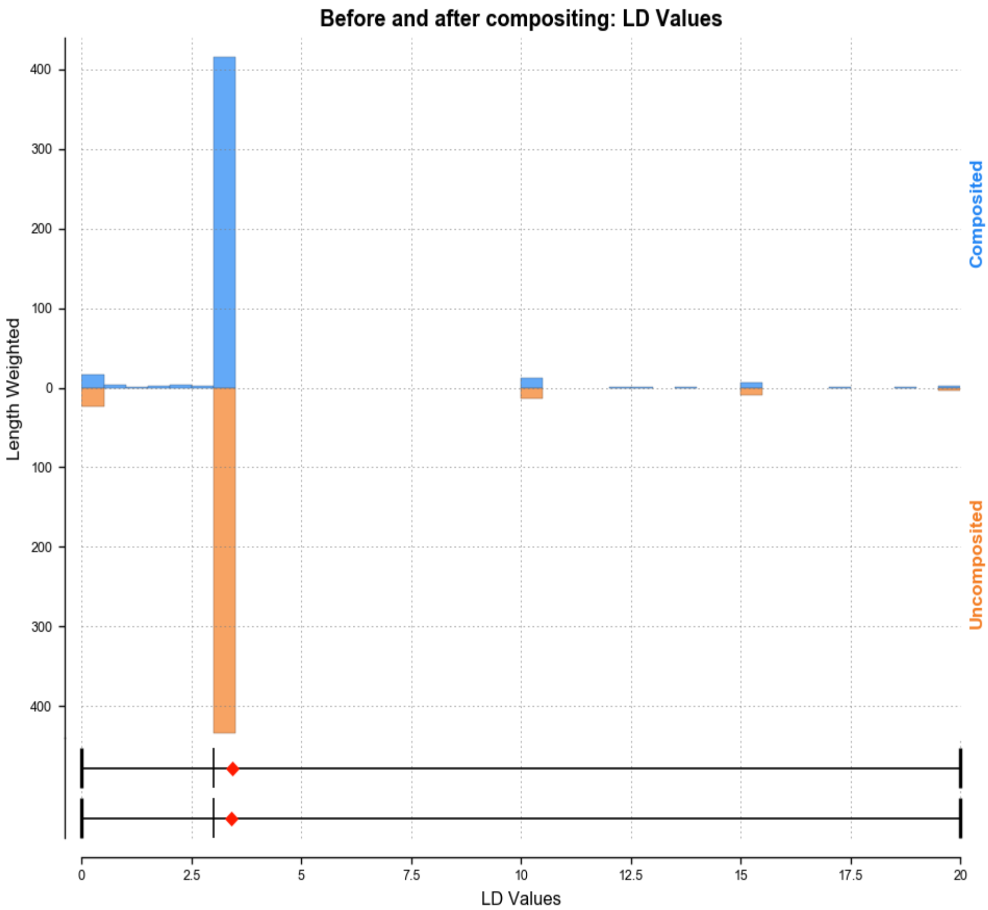


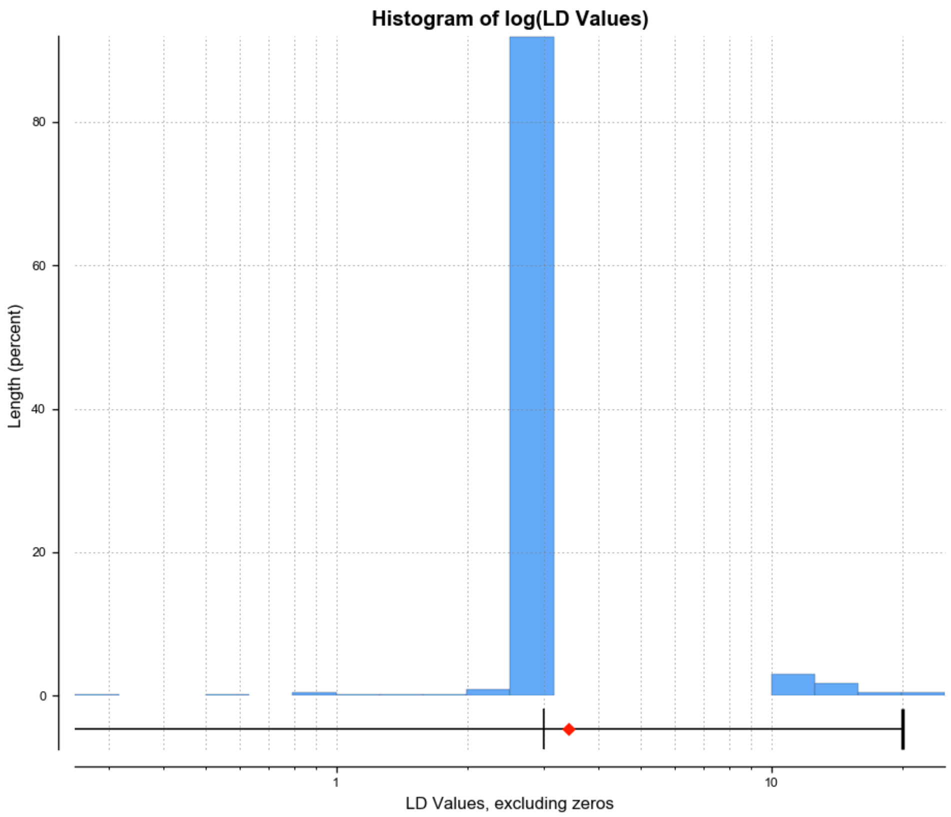



Log Probability of LD Values
Cumulative Probability (percent)
99.8
99.5
99
98
95
90
70
50
30
10
5
2
1
0.5
0.2
1
10
LD Values, excluding zeros

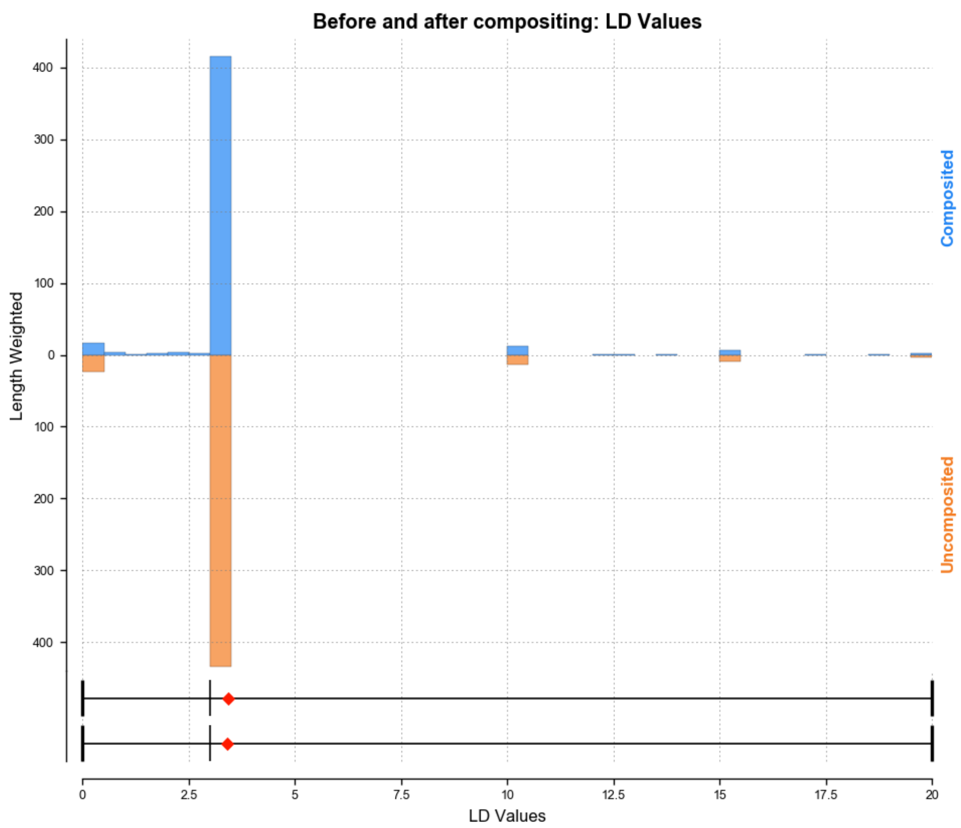

Interval Lengths before and after compositing: ALT Values
Count
400
200
0
200
400
Composited
Uncomposited
0
2.5
5
7.5
10
12.5
15
17.5
20
Interval Length


Histogram of log(LD Values)
Length (percent)
80
60
40
20
0

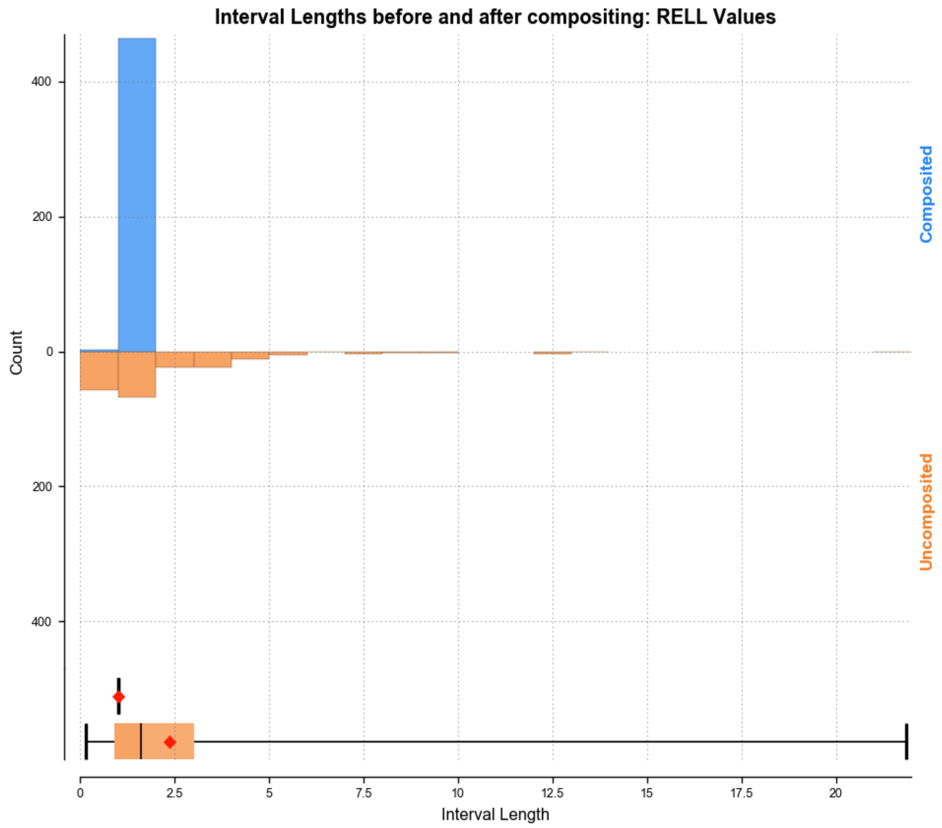

Before and after compositing: LD Values
Length Weighted
400
300
200
100
0
100
200
300
400
Composited
Uncomposited
0
2.5
5
7.5
10
12.5
15
17.5
20
LD Values

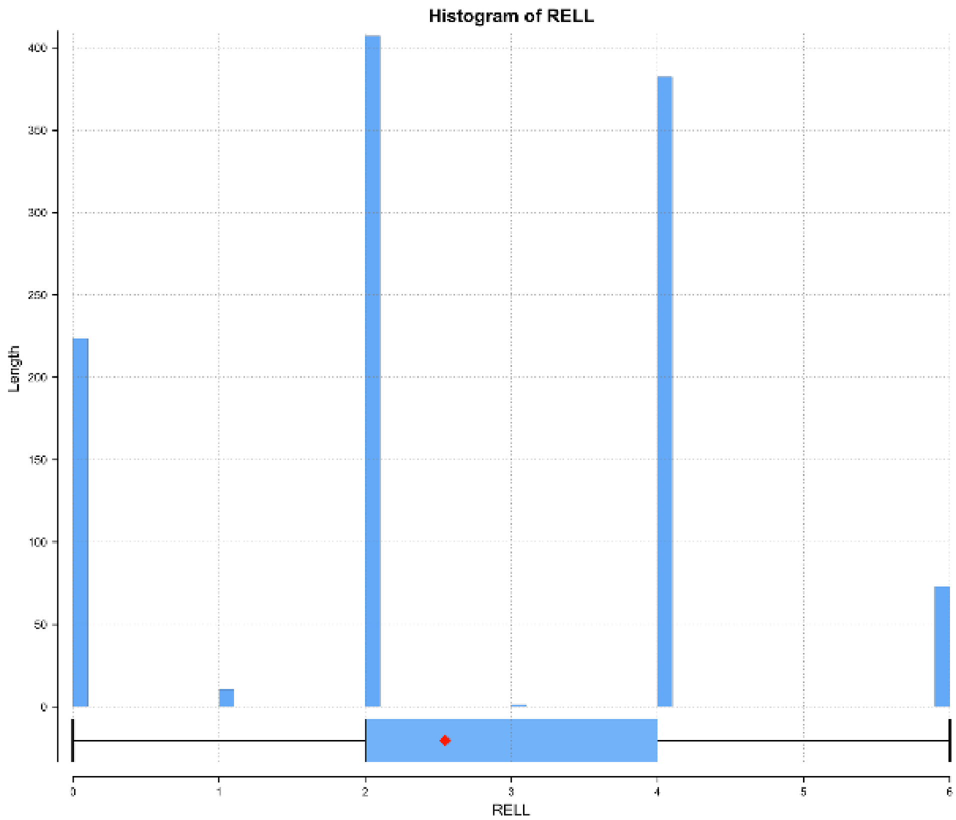


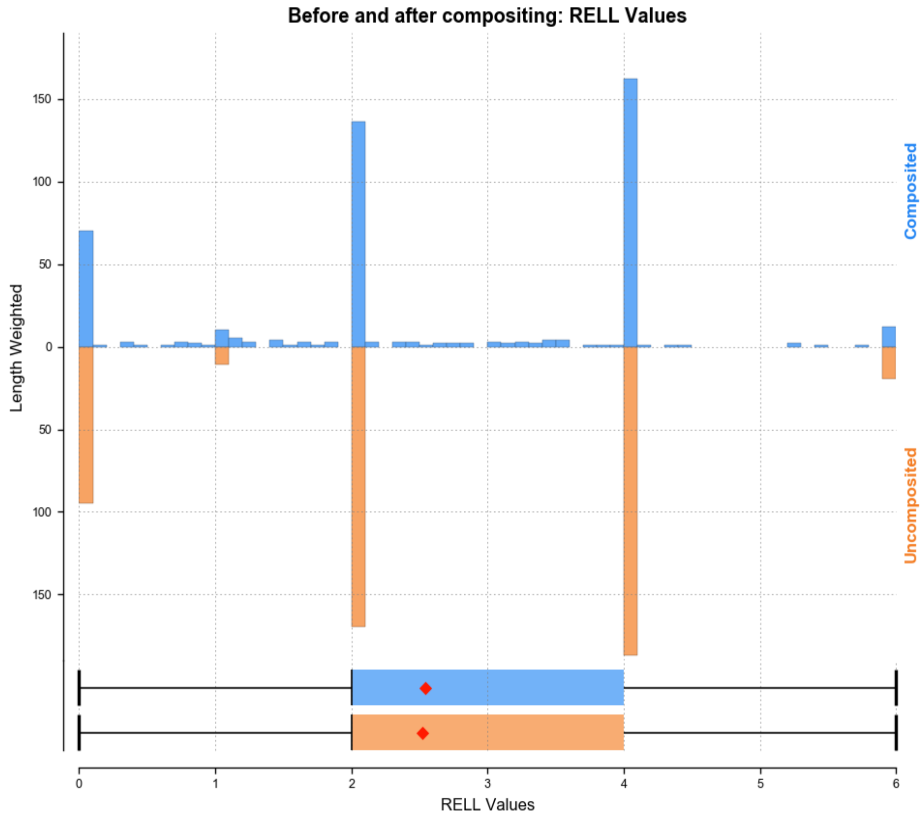
Histogram of log(LD Values)
Length (percent)
80
60
40
20
0
1
10
LD Values, excluding zeros



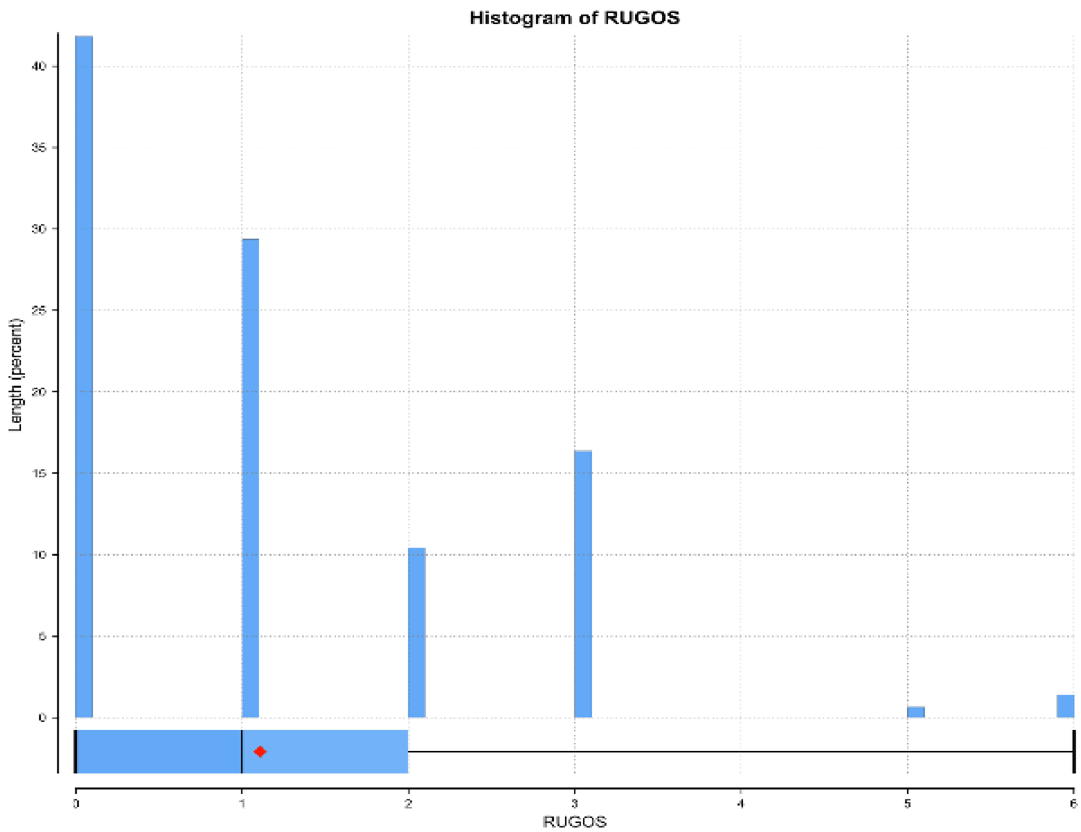
Before and after compositing: LD Values
Length Weighted
400
300
200
100
0
100
200
300
400
Composited
Uncomposited
0
2.5
5
7.5
10
12.5
15
17.5
20
LD Values


Interval Lengths before and after compositing: RELL Values
Count
Composited
Uncomposited
Interval Length
0
2.5
5
7.5
10
12.5
15
17.5
20
400
200
0
200
400

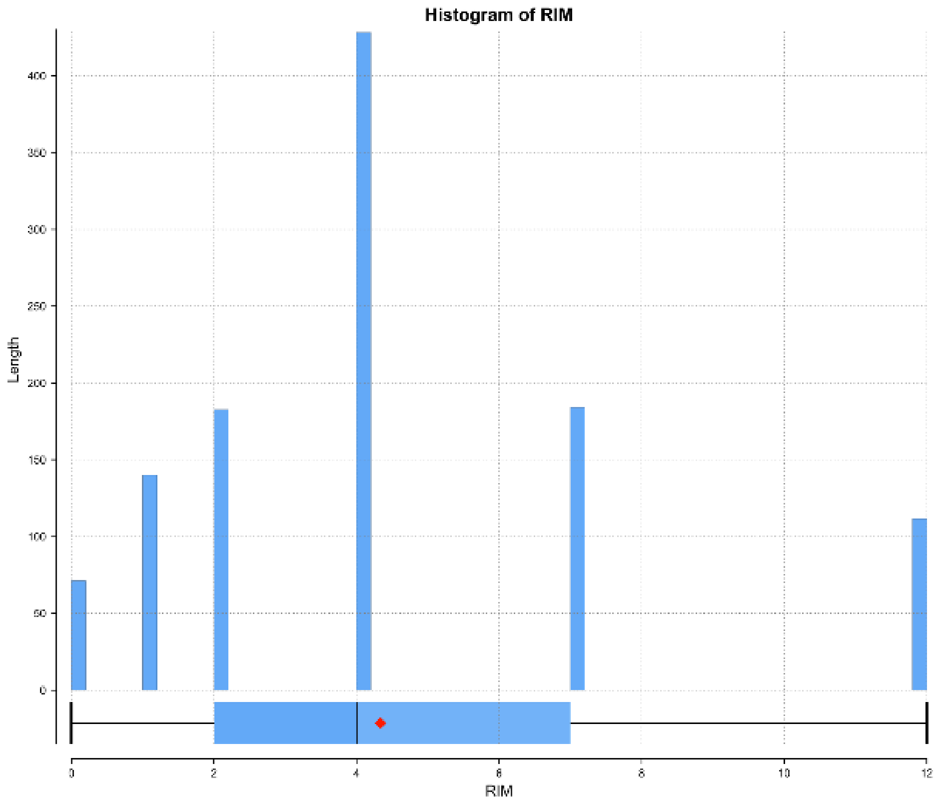

Histogram of RELL
Length
RELL
0
50
100
150
200
250
300
350
400
0
1
2
3
4
5
6



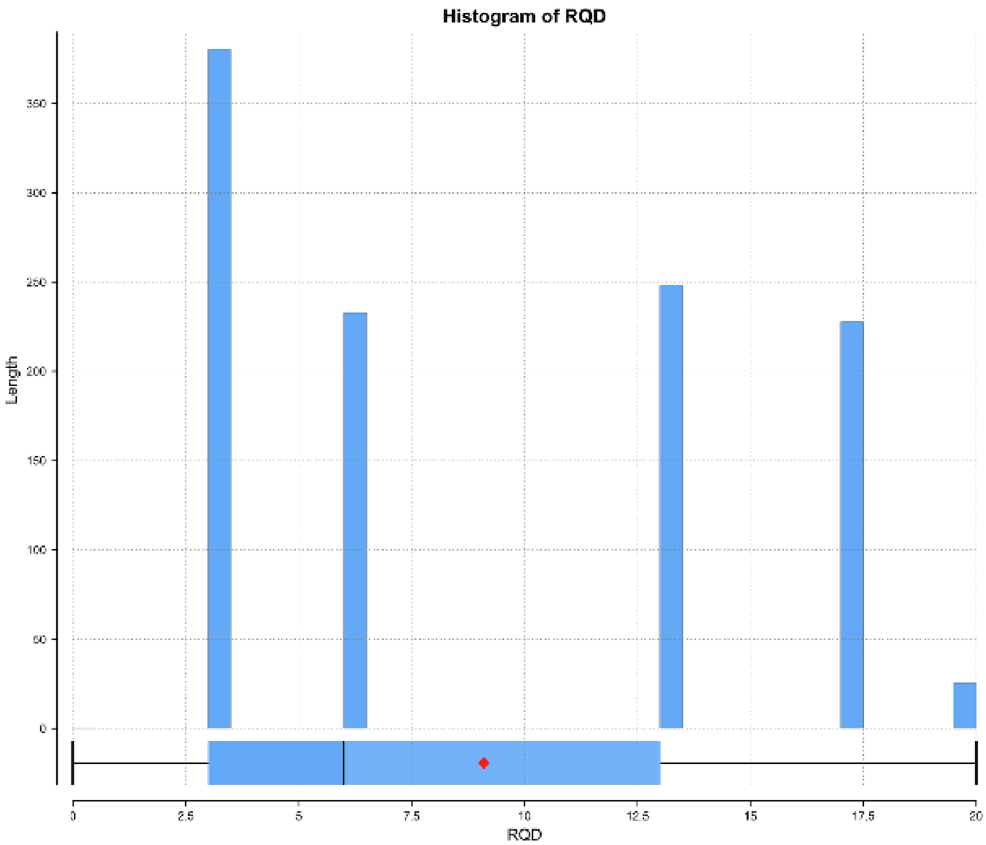
Before and after compositing: RELL Values
Length Weighted
150
100
50
0
50
100
150
Composited
Uncomposited
0
1
2
3
4
5
6
RELL Values



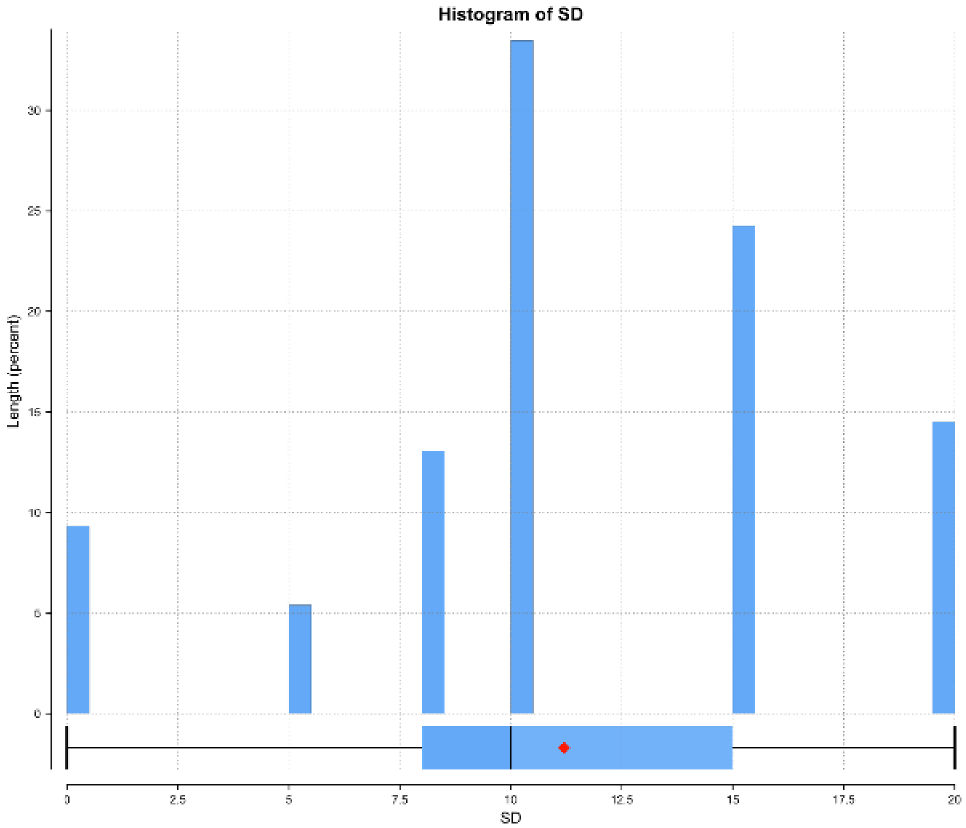
Histogram of RUGOS
Length (percent)
40
35
30
25
20
15
10
5
0
0
1
2
3
4
5
6
RUGOS

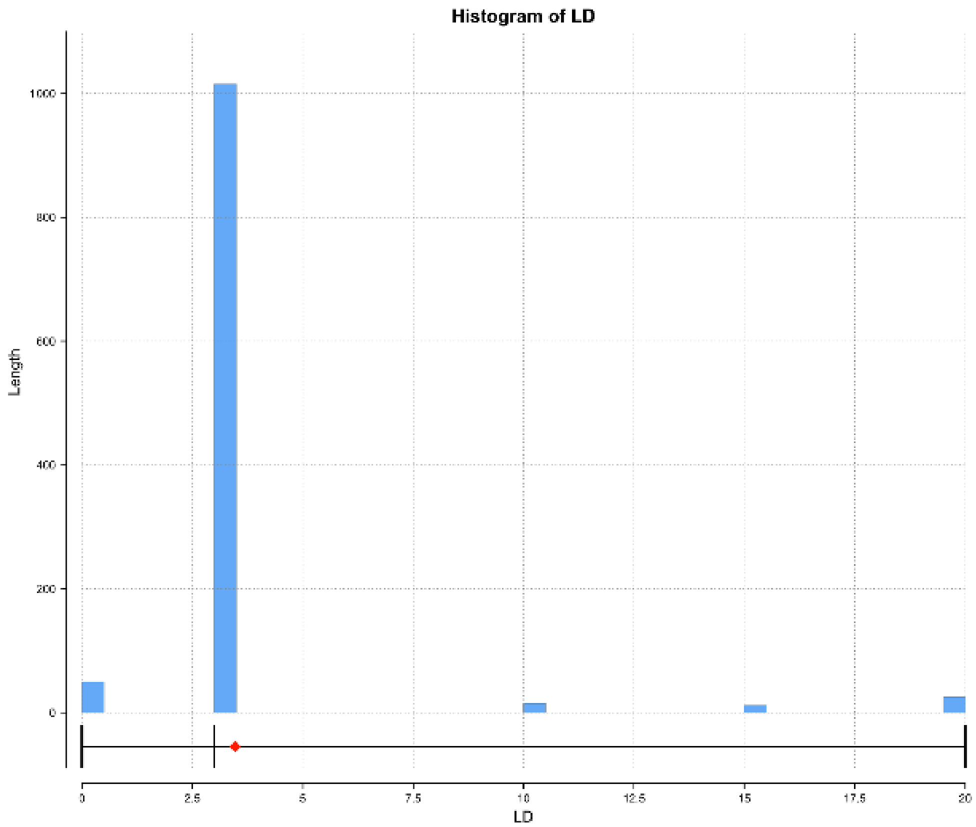


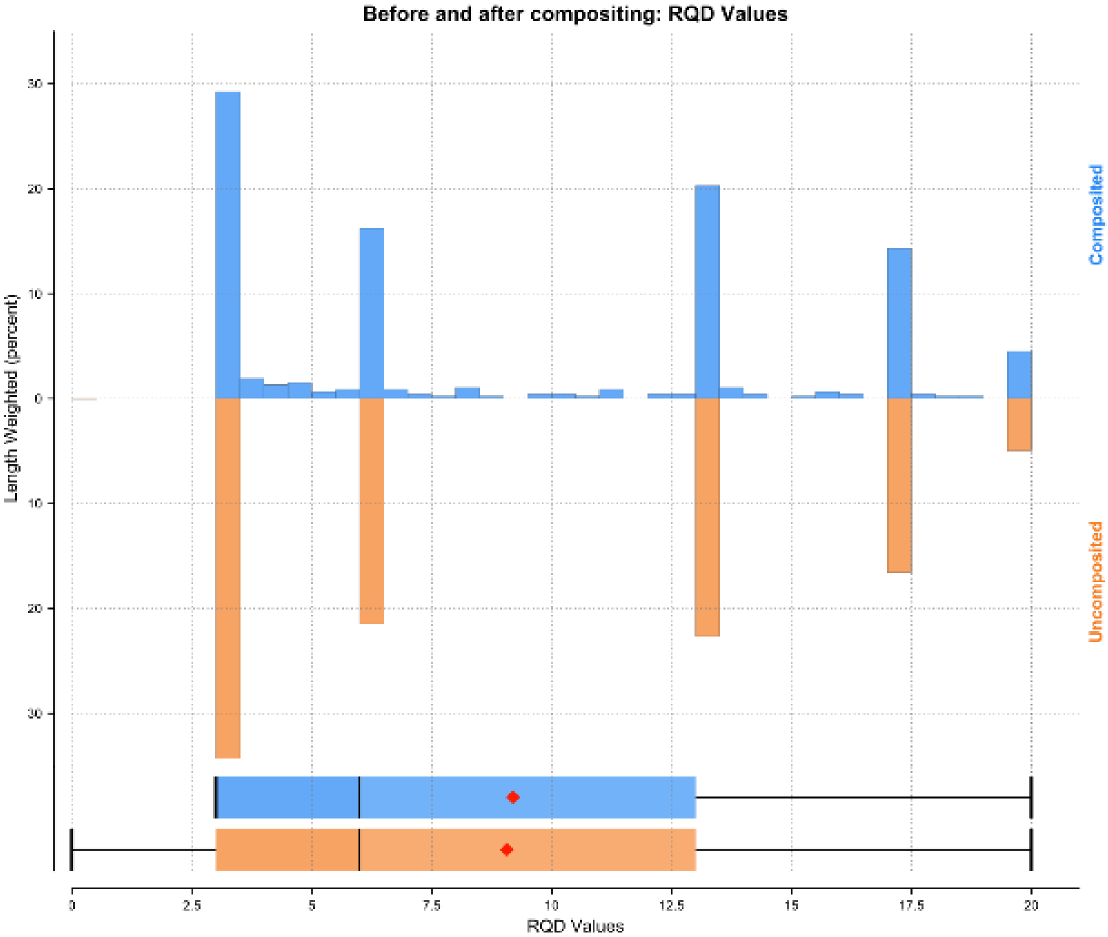
Histogram of RIM
Length
0
50
100
150
200
250
300
350
400
0
2
4
6
8
10
12
RIM



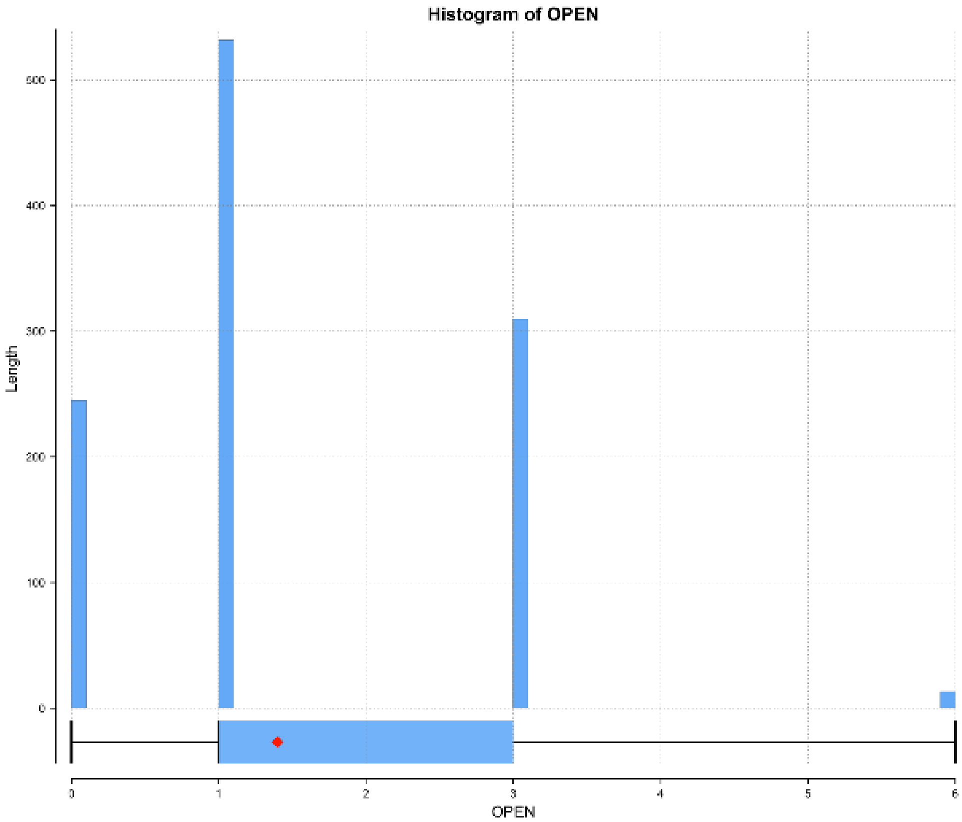
Histogram of RQD
Length
0
50
100
150
200
250
300
350
0
2.5
5
7.5
10
12.5
15
17.5
20
RQD


Histogram of SD
Length (percent)
0
5
10
15
20
25
30
0
2.5
5
7.5
10
12.5
15
17.5
20
SD


Histogram of LD
Length
0
200
400
600
800
1000
0
2.5
5
7.5
10
12.5
15
17.5
20
LD



Before and after compositing: RQD Values
Length Weighted (percent)
Composited
Uncomposited
RQD Values



Histogram of OPEN
Length
OPEN

APPENDIX A

Engineering Standards Applied in the Project

ISO 14689:2017
Identification, description and classification of rock
This standard specifies the rules for the identification and description of rock material and mass on the basis of mineralogical composition, genetic aspects, structure, grain size, discontinuities and other parameters. It also provides rules for the description of other characteristics as well as for their designation. The standard applies to the description of rock for geotechnics and engineering geology in civil engineering. The description is carried out on cores and other samples of rock and on exposures of rock masses.

ACI 506R-16
American Concrete Institute
Guide to Shotcrete
This guide provides information on materials and properties of both dry-mix and wet-mix shotcrete. Most facets of the shotcrete process are covered, including application procedures, equipment requirements, and responsibilities of the shotcrete crew. Other aspects, such as preconstruction trials, craftsman qualification tests, materials tests, finished shotcrete acceptance tests, and equipment, are also discussed.

ASTM STP984
Rock Mass Classification and Tunnel Reinforcement Selection Using the *Q*-System
This standard provides an overview of the *Q*-system and documents the scope of case records used in its development. A description of the rock mass classification method is given using the following six parameters: core recovery (RQD), number of joint sets, roughness and alteration of the least favorable discontinuities, water inflow, and stress-strength relationships. Examples of field mapping are given as an illustration of the practical application of the method in the tunneling environment, where the rock may already be partly covered by a temporary layer of shotcrete. The method is briefly compared with other classification methods, and the advantages of the method are emphasized.

ASTM D2487 - 17e1
Standard Practice for Classification of Soils for Engineering Purposes (Unified Soil Classification System)

This standard classifies soils from any geographic location into categories representing the results of prescribed laboratory tests to determine the particle-size characteristics, the liquid limit, and the plasticity index.
The various groupings of this classification system have been devised to correlate in a general way with the engineering behavior of soils. This standard provides a useful first step in any field or laboratory investigation for geotechnical engineering purposes.

Geotechnical Design Standard – Minimum Requirements
Manual
Department of Transport and Main Roads
General requirements. Performance requirements. Unreinforced embankments. Reinforced embankments. Cut slopes. Unreinforced cuts. Reinforced cut slopes. Deep foundations. Retaining structures. Embedded retaining walls. Reinforced concrete cantilever retaining walls. Soil nailed walls. Reinforced soil structure RSS walls. Gabion retaining walls. Boulder retaining walls.

ASTM STP984
Rock Mass Classification and Tunnel Reinforcement Selection Using the *Q*-System
This standard provides an overview of the *Q*-system and documents the scope of case records used in its development. A description of the rock mass classification method is given using the following six parameters: core recovery (RQD), number of joint sets, roughness and alteration of the least favorable discontinuities, water inflow, and stress-strength relationships. Examples of field mapping are given as an illustration of the practical application of the method in the tunneling environment, where the rock may already be partly covered by a temporary layer of shotcrete. The method is briefly compared with other classification methods, and the advantages of the method are emphasized.

ISO 19434:2017
Mining — Classification of mine accidents
This standard establishes a classification of mine accidents by their origin or causes, by the type of accident, and by their results or consequences. The latter includes only the accidents resulting into consequences on people, not equipment or machinery.
Different categories of causes, types and consequences of mine accidents are briefly defined, and a 3-digit code is assigned to each category. These can be combined to ultimately allocate a unique 15-digit code to each type of mine accident. This code can then be used in statistical analysis. Similarly, an allocated code clearly shows to which categories of causes, type of accident and resulting consequences the mine accident belongs to. The standard is applicable to all surface and underground mines.

APPENDIX B

Multiple Constraints, Restrictions and Limitations Considered in the Project

Geological and Geotechnical Constraints

Geological and geotechnical problems face several technical constraints that must be carefully considered for a successful implementation of the project: stability of the opening during excavation, tunnel-induced displacement field, constraints for alignment, thrust zone, shear zone, fault zone, shallow overburden, complex geometry, recent global formations, fills, weathering, groundwater.

Availability of Geological and Geotechnical Data

Part of the required soil and rock information was obtained by laboratory testing of soil and rock samples, as well as from in-situ analysis. Remaining information was obtained from other sources such as reports from technical reports Peruvian Geological Institute, Geological Engineering Chapter of Peruvian Engineers Association, Geological, Mining and Metallurgical Peruvian Institute IGMMP. All required information and data was finally found and made available, and applied to the project.

Uncertainties and Risks

No geotechnical project is risk free. Risk are managed, minimized, shared, transferred or accepted, but they cannot be ignored. Geological and geotechnical concepts with respect to structure could be uncertain. On the other hand, economic evaluations have uncertainties related to cost estimation, changing conditions in economically viable sites, changes in geotechnical technology, fluctuations in costs and market conditions, political situation, community relations, etc. All these issues must be carefully analyzed in order to ensure the profitability of the project for the most conservative economic conditions and diversity of scenarios. In this project, all these issues have been considered from a conservative scenario and criteria.

Safety Considerations

Geological and geotechnical activities present diverse safety issues that must be taken into account in the development of the project. It is important to comply with safety standards pointing to satisfy proper safety levels considering their impact in the project budget. Care of human life, well-being and safety is an important issue to take into account throughout the different stages of the project and its life-cycle.

Environment and Sustainability

Geological and geothecnical activities face diverse and broad environmental issues at both local and global levels which could affect the project sustainability. The project considers environmental issues such as potential effluents spills,

soil, air and water pollution, habitat protection and biodiversity. The project also considers community relations with local people as an important stakeholder of the project.

Schedule

The project must be completed in one academic semester. It is estimated the project requires an average of 150 hours of teamwork with 4-5 students per team. Considering that, besides the senior design project course, students are enrolled in 3-4 additional courses in the academic semester, students have to plan ahead in order for identify all required activities, distribute the tasks among all team members and, finally, integrate all partial tasks to configure the final project.