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Master's degree in Petroleum and Mining Engineering

MSc thesis

Comparison between different excavation techniques aimed at optimizing productivity in an underground mine

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ABSTRACT

The study aims to analyze and compare the different techniques used for the exploitation of sandstone in Lochaline Quartz Sand, measuring productivity and costs. The techniques compared are drill and blast with two-stages blasting and full-face blasting, and hydraulic hammers.

For each technique the average hourly production [t/h] and the average cost/t have been calculated. The productivity has been evaluated measuring the average volume excavated in a 8 h shift, while the costs analysis involves the actual costs that occurred in the last year, from 1st October 2018 until 30th September 2019.

The full-face blasting scheme resulted as the most productive and expensive technique, while the use of the two hammers the least productive but economically advantageous. The study highlighted the inefficiency of one of the two hammers, reducing the productivity of the technique by more than 40 %. Furthermore, the analysis has justified the interruption of the two-stages blasting for the full-face blasting scheme, even though it was less expensive. This outcome is due to the fact that in the last year the unit costs of each consumable has increased. In addition, it was shown that the use of the two hammers can produce more than 70 % than the two-stages drill and blast production for 60 % of the cost.

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INTRODUCTION

This thesis aims to give an accurate comparison of two of the main conventional excavation techniques used in mining: drill and blast and hydraulic impact hammer. This study was carried out in the only operating underground silica sand mine in the United Kingdom, Lochaline Quartz Sand Ltd.

The abovementioned mine exploits one of the purest silica sands in the world, with a high percentage of quartz (99,6%) and low iron content. It provides two products: the LQS85, silica sand with average iron content of 85 ppm, and the LQS500 with average iron content of 500 ppm. The former is only produced by drill and blast, while the latter has been exploited with series of different techniques over the years, namely:

- two steps blasting, where only the lower cross-section was in use for the exploitation of LQS500
- Benching with hydraulic hammer
- and, most recently, full face by drill and blast

The study aims to analyse in detail each technique used for the exploitation of LQS500, measuring productivity and costs, and evaluating possible enhancement.

At first, a detailed description of the site and of the product is given, from the geological assessment to the description of the processing plant and the quality control performed on the product, which takes place before and during the loading of the vessel that transports the material from the site to other storage location. Then a description of the mining method used, i.e. the room and pillar, is given, focusing on ground control and ventilation issues in the mine.

Then, a characterization of drill and blast and hydraulic hammer and the description of the application of these techniques in the mine are provided. Thus, the outline of the phases of the evolution of the excavation techniques adopted is provided, from the drill and blast for the production of LQS85 to the full-face drill and blast used for the production of LQS500. All the analyses are provided with the theoretical production [t/h], and then compared.

Once the most suitable technique was identified, a cost analysis was made, taking into consideration the entire cost for each operation, by analysing all the invoices derived by each technique. For a clearer idea of the expenses, the analysis involves the actual costs that occurred in the last year, from 1st October 2018 until 30th September 2019.

For the two hammers technique, the costs considered are: a) the consumption of fuel, grease and hydraulic oil; b) the cost of the chisel; c) the operational cost considering one operator per hammer. However, in the analysis of the drill and blast, drilling and blasting were evaluated separately. The drilling expenses considered were: a) the consumption of grease, hydraulic oil, fuel, and bits; b) the chisel; c) the operator. The blasting takes into account the explosive, the detonators, the stemming, the initiating system and the operational cost, that includes the salary of three shot-fires.

At the end of the evaluation, a comparison between the techniques has been provided.

I. LOCHALINE QUARTZ SAND LTD. MINE

Lochaline Quartz Sand Ltd. produces one of the purest silica sands in the world due to its high percentage of quartz and low iron content, suitable for the production of high-quality glass.

The mine was first opened in 1940 to fulfil the need for silica sand during World War II. At that time the material was mainly used for periscope lenses and gunsights. The mine remained operative until 2008 when it was closed, and then reopened in 2012 as a joint venture between Minerali Industriali and NSG/Pilkington.

The mine is exploited by rooms and pillars, so part of the orebody is left in place to support the roof. The technique applied is a combination of drill and blast and hydraulic hammer, as will be further explained in the next chapters.

On the site are also available:

- A deposit for the explosives.
- A processing plant
- A quality control laboratory
- A workshop
- The offices

GENERAL FRAMEWORK OF LOCHALINE GEOLOGY AND DEPOSIT

Lochaline is a village in Movern, an area in the Scottish Highlands. As is possible to see in Figure I.1 it is located on the northern shore of Loch Aline, reachable using A884 road, and it is connected by ferry with Fishnish (Isle of Mull). The geology of the area includes rocks formed in three different periods. These rocks have been subjected to earth movements and fracturing until a subsequent glacial erosion has exposed the formation that is found today.

Source: Google Maps

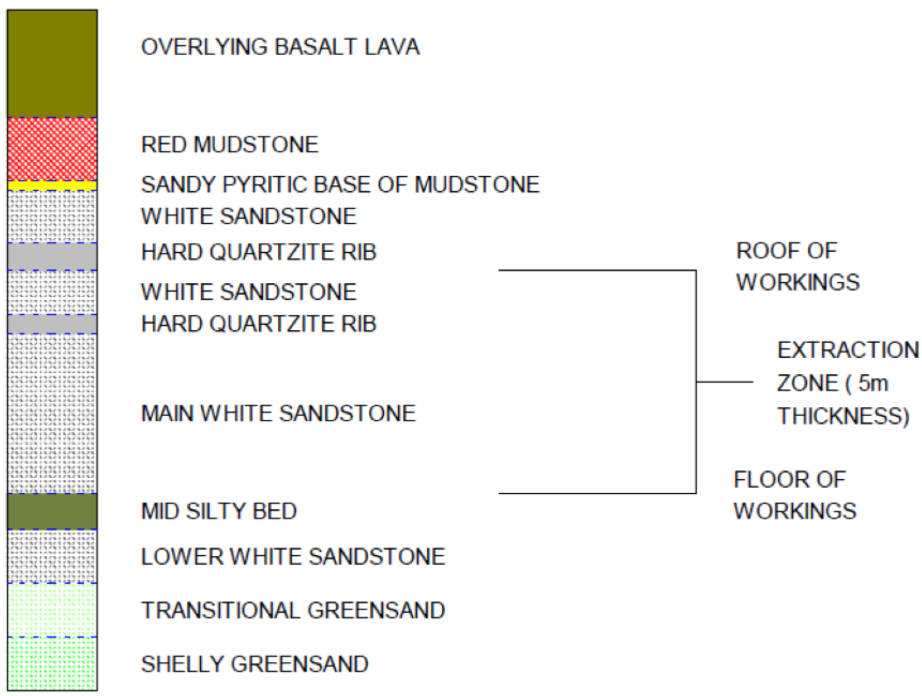
Figure I.1 Map of Movern area

The mine produces white sandstone of the Cretaceous age in a zone where most of the rock bodies are nearly horizontal. The sandstone is a very pure, well-sorted medium-grained quartz sandstone, and it is characterized by a white to pale yellow-brown colour. The total ore body consists of a seam of 5 m to 12 m of thickness. The extraction zone is a deposit of silica sand with thickness between 5 m and 7 m. The stratum has been classed by several changes of the sea level during various eras, this screening has separated the quartz from the other minerals allowing a good grade sandstone.

Cretaceous sandstone is included in the second oldest group of rocks in Lochaline, together with Jurassic sandstones and siltstones, and Cretaceous greensands. The oldest formation is a Moine schists. This latter formation has not been seen during the exploitation of the mine, even though there is some evidence that the white sandstone lies close to it.

The layers below the exploited deposit are characterized by a lower grade white sandstone and shelly greensand: these two deposits are of no interest. Immediately above the ore body, there is a thin layer (0,2÷0,4 m) of strong silicified sandstone, called "Top Hard Rib", that is where the sandstone has been hardened by silica cement. This stratum is generally characterized by semi-regular layers that, when encountered, represent an excellent roof beam for the mine. Above it, there is a layer of red mudstone that is considered as a small lode from 1 m to 3 m of thickness, even though it is not well documented. On top of the red mudstone there is a thick layer of strong basalt lava, which extends from about 5 m above the mining section up to the surface. This superficial stratum probably formed during an extended period of volcanic activity, associated with the formation of the Atlantic Ocean. This layer has preserved the quality of the ore body, by protecting it from erosion.

Figure I.2 shows the typical stratigraphy exposed in the mine.



Source: Smith T., 2019. Summary of Geological Features and Associated Mine Risk at Lochaline

Figure I.2 Lochaline stratigraphy

The Mines Regulations 2014, Regulation 32 “Duty to take ground control measures at Lochaline”, performed by Graham Daws Associates state:

"Numerous normal faults have been encountered by the mine and most of these have a NNW – SSE trend. Fault throws vary from only a few metres to over 25metres. Jointing and partial brecciation was noted in close proximity to faults, but in the mine generally joints are not well developed. Some pillars have joints but it is not clear if these are blast induced features. Occasionally major joints were seen in the roof and these were planar, rough, closed and tended to be sub parallel to the faults. Dips are generally low and towards the NW.

Where faults are encountered sub parallel joints also occur in close proximity and this breaks the integrity of the roof beam. If mudstone or basalt is introduced to the excavation then numerous bedding planes and joints are present and additional support will be required. Water can also enter the workings via a fault and this further reduces the strength of the rock mass. A systematic scheme of support is required whenever faults are encountered.

If single major joints intersect the roof or pillars and they remain closed then there is probably no hazard. However it is common for small rock fragments adjacent to become loose or detached and these need control to prevent harm to persons who work or pass. This will need to be judged on a case by case basis. Joints within pillars may need support to prevent spalling.

Where a dyke intersects the workings the basalt tends to be jointed and there can be a higher frequency of joints within the sand on the margins. Again in a similar mode to joints there is an increased risk for fragments to fall and a systematic means of control needs to be adopted.

It is thought that the basalt could be a source of water, thus any feature connecting with the basalt could introduce water to the workings. The red mudstone is probably impermeable and an effective barrier but once exposed to water en mass it could quickly deteriorated. It is not *known how the sand reacts to exposure to water but it is probably minimal.*

Generally makes of water from the basalt are not present. Water does occur where the basalt is exposed by faulting or roof falls. No assessment of water inflow rates was available. Parts of the workings are flooded but it is not clear where this water originated.

All support measures that could be exposed to water will be liable to corrosion and consequently this should be taken into account with their choice and design."

LOCHALINE QUARTZ SAND – THE PRODUCT

The principal product of the mine is a very pure silica sand, with about 99,6% of quartz. Silica Sand is used in many fields of industry mostly for the production of glass, ceramics and refractory glass.

The production of Lochaline Quartz Sand consists of two types of sand distinguished by the content of iron. The purest is called LQS85 or "white" in slang. It is characterized by a very low iron content (about 85 ppm), therefore is mainly used for the production of flint glass, but also for other uses when a uniform white sand with low iron content is needed. In Table I-1 the characteristics for LQS85 are listed, derived from a sample dried at 105°.

Table I-1 Analysis on a LQS85 sample

	Chemical Composition	**Limits**
SiO_2	99.6 %	99.5 % max
Al_2O_3	0.10 %	0.20 % min
Fe_2O_3	80 ppm	85 ppm
TiO_2	0.02 %	
CaO	0.05 %	
MgO	0.05 %	
K_2O	0.05 %	

Na_2O	0.01 %	
Loss on ignition (1100°)	0.10 %	0.30% max
Moisture	4.5 % ± 1 %	

PARTICLE SIZE (CUMULATIVE % RETAINED ON ISO TEST SIEVES)

	TYPICAL [%]	LIMITS [%]
+ 1.00	0	0.1 MAX
+ 710 μm	0	0.5 MAX
+ 500 μm	0.4	1 MAX
+ 355 μm	4	
+ 250 μm	32	
+180 μm	84	80 MIN
+125 μm	98	96 MIN
+ 90 μm	99	
- 90 μm	1	

The other product has a higher iron content (an average of 500 ppm) and it is called LQS500 or "green" in slang. The characteristics are listed in Table I-2, also obtained from a sample dried at 105°. This product, like LQS85, allows the production of transparent glass, but it is preferred for applications where the presence of iron is not relevant.

Table I-2 Analysis on a LQS500 sample

	Chemical Composition	**Limits**
SiO_2	99.6 %	99.5 % max
Al_2O_3	0.10 %	0.20 % min
Fe_2O_3	500 ppm	85 ppm
TiO_2	0.02 %	

CaO	0.05 %	
MgO	0.05 %	
K_2O	0.05 %	
Na_2O	0.01 %	
Loss on ignition (1100°)	0.20 %	0.30 % max
Moisture	4.5 % ± 1 %	

PARTICLE SIZE (CUMULATIVE % RETAINED ON ISO TEST SIEVES)		
	TYPICAL [%]	**LIMITS [%]**
+ 1.00 mm	0	0.1 MAX
+ 710 µm	0.1	0.5 MAX
+ 500 µm	0.8	1 MAX
+ 355 µm	4.2	
+ 250 µm	26.3	
+180 µm	74.7	60 MIN
+125 µm	95.4	90 MIN
+ 90 µm	99.4	
- 90 µm	1	

To obtain these products, the excavated material is treated in the processing plant, and then it undergoes quality control.

PROCESSING PLANT

The original policy of the company was to the sell the product as raw material. However, in 1974 the processing plant was opened on the mine site.

The treatment is composed of two main phases, dry and wet. The processing steps for the two sands are mostly the same; the only difference is that the LQS85 product is subjected to an additional final treatment.

The plant can process an average of 500 t/d and the final product is then stored in one of the three on- site storage facilities: Main yard, Norway yard and Fines Yard. The whole process for the white sand processing is described in Figure I.6.

Dry Treatment

The material is moved from the mine through a dumper and stocked outside. From there, using a shovel (CAT 938M), the material is transported from the temporary storage to the processing line. The first step in the plant is the *primary crusher* (Figure I.3) which has a power of 55 kW, and can reduces the dimension of the muckpile down to 200 mm. The crusher is equipped with a hydraulic hammer (Figure I.3 – in blue), that is used to reduce the size of the boulders. The material is then transported to the secondary crusher, a hammer mill that will be better explained in the next paragraphs.

The product goes then into a conveyor belt which is equipped with a cross-belt *magnetic separator* and a *metal detector*. The former is a suspended magnet fixed over the moving conveyor belt: the magnet attracts the magnetic minerals and discharges them outside the conveyor.

Figure I.3 Primary crusher

The *hammer mill* is an impact crusher, that uses a series of hammers that crush, shatter or pulverize the material by repeated impacts. The one used in the mine under study has a power of 160 kW, and the material is moved into the mill's chamber through gravity. The maximum dimension of the mill's product is about 20 mm. This material is then transported through a conveyor belt inside a silo.

The *silo* acts as a secondary deposit in order to guarantee a certain amount of material in case the first part of the plant stops. It has also the function of regulating the material's flow into the wet section of the plant.

Then, the material passes through the *vibrating-screen* that retains everything above 0,65mm. In this step the material flows or slides with water as a slurry, on an inclined frame. The groove is mounted on springs that give high frequency and low amplitude vibration thanks to an unbalanced flywheel. What passes through the A/C screen is the final product of the dry treatment and it is transported to the wet plant through the *feed pump*. The retained material is considered as reject.

WET TREATMENT

From the dry plant the slurry material is fed to the *double screw classifier* (Figure I.4) until approximately half of the tank is full. Here the largest grains sediment, whereas the fines are suspended in water. The material is then moved to the next process of the plant through a system composed of two screw conveyors. The grains suspended are considered as waste.

Figure I.4 Double Screw classifier

From the screw classifier, the material is pumped inside the *attrition cells*. The aim of this passage is the removal of the superficial layer of each grain by a rubbing action. The slurry passes through two spirals, one with a descendent flow and one with an ascendant flow. During the path, the material is cleaned from iron and impurities due to the impact between the grains inside the cells.

The product of the attrition cells is then pumped inside the *hydrosizers* that separate the fine grains from the coarser ones, by using the principle of particle settling. The separation is obtained by injecting water inside the tank, creating a rising flow which produces an overflow of the fines and an underflow of the heavy grains. In Lochaline Quartz Sand, this process is obtained with two hydrosizers: the first one separates the product from the fines ($\Phi \leq 100$ µm), while the second separates the waste from water. The water can then be reused in the plant.

The subsequent treatments depend on the type of sand produced. If the material treated is LQS500, it is pumped to a *cyclone* that separates the sand from the water by means of the centrifugal force. When the slurry is fed tangentially into the cyclone's tank, the centrifugal force tends to throw the heavier grains out through the underflow. The material of interest is in the underflow, and after the separation, it falls into the storage yard. The lighter particles and the water are pushed outside through a vortex finder and discharged as an overflow.

Instead, white sand has to pass through a further step, the *gravimetric spirals* (Figure I.5) that use the centrifugal force to separate light density materials from the heavier elements. In Lochaline Quartz Sand, they are composed of 12 spirals and are used for the separation of heavy metals (Iron, Magnesium and Titanium) from the sand. The slurry is fed from the top of the spirals and, as it travels through it, the grains start to separate according to their size and density. High-density particles travel through the inside of the spirals, while low-density grains tend to stay on the perimeter. At the end of the spiral, the two materials are separated by two splitters (Figure I.5-in red) that have to be settled by hand depending on the type of material and its reaction to the treatment. After this step, the product is pumped to the storage, where it passes through a cyclone to be separated by water (same process as LQS500).

Figure I.5 Gravimetric spirals

WHITE SAND SPIRALIZED IN MAIN YARD + GREEN SAND IN FINES YARD

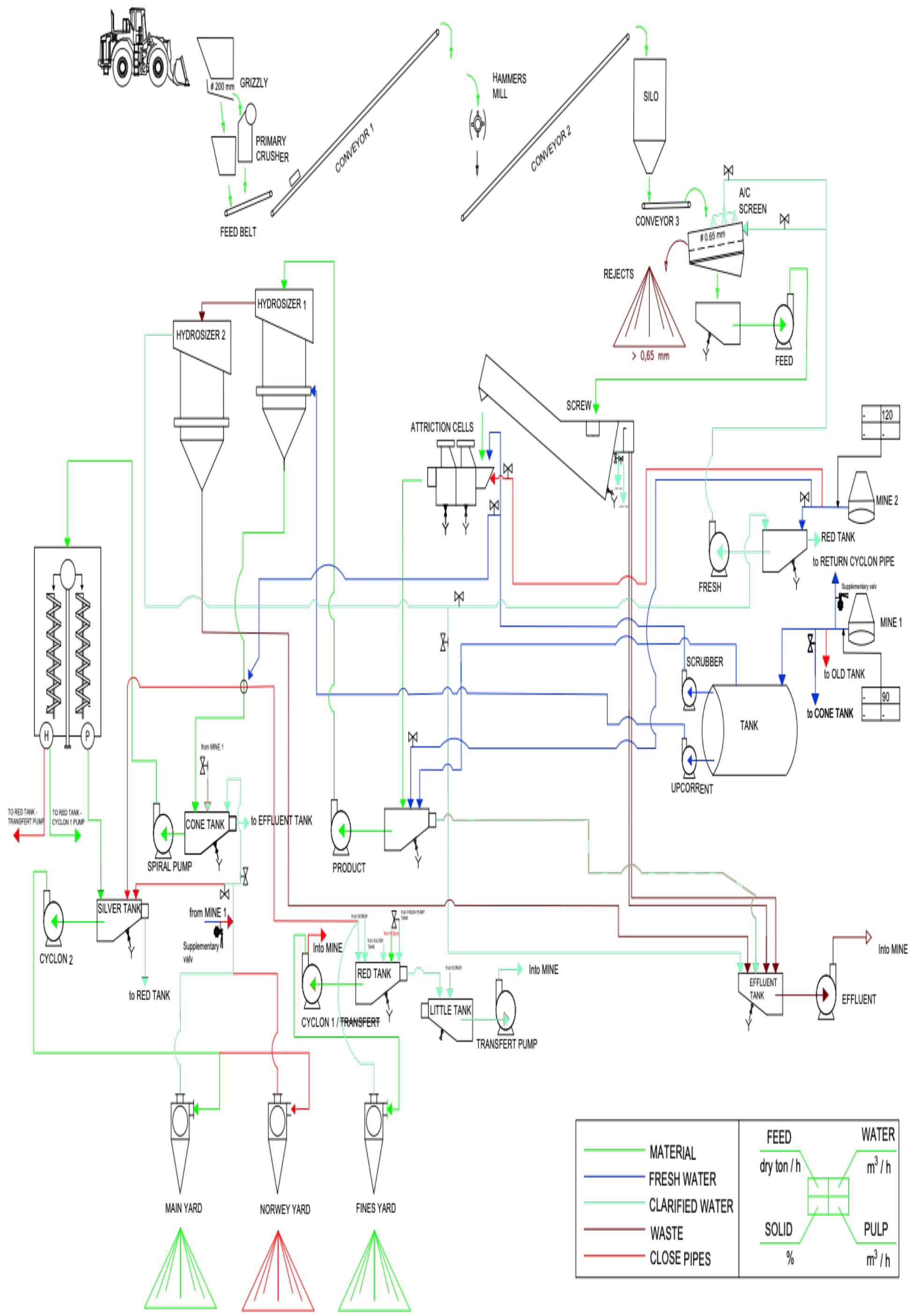


Figure I.6 Treatment Plant Flowsheet - LQS85 treatment

QUALITY CONTROL

A fully accessorized laboratory for the evaluation of the quality of the sand treated before the sale is available on site. Normally, the material is tested twice a day, but in case of high productivity, the number of controls can reach 4. Each test aims to analyse:

- Grain size distribution [mm]
- Iron Content [ppm]

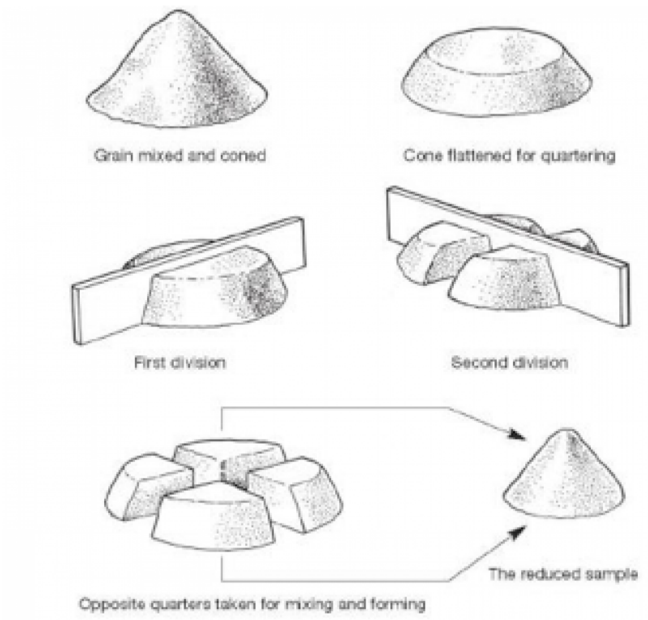
The grain size distribution is monitored to ensure that it falls in the 0,1÷0,6 mm range. This passage is also used as an indicator of the good functioning of the plant. An excess of fine grains might be due to a malfunctioning of the wet treatment, while if the grains are too big the screen might be excessively opened by wear.

The iron content defines the type of product, thus it is needed to check the quality of the product.

These tests are done on a random sample taken from 8 different zones of the stocked material at the end of the treatment. Those zones are 4 at the top of the storage and 4 at the bottom (Figure I.7). This sampling procedure ensures that the material collected is homogeneous, and representative of the whole stock. It is collected with a scoop and stored in a bucket, until the latter is full (15 kg of sand circa).

Figure I.7 Collection of the sample

Once the sample is collected, it is taken to the coning table where it is subjected to the *quartering procedure*: it consists in positioning the material in a conical shape on a flat surface and then flattening it on the table before starting the quartering. At this point, the remaining sample is divided into 4 (Figure I.9), and the 2 non-adjacent slices are removed. The 2 final slices are mixed and heaped in a cone shape so that the procedure can start again. A scheme of the sequence is given in Figure I.8. The quartering is repeated until the remaining sample weights 1 kg circa. The material is then collected in a plastic bag and taken to the lab.



Source: www.911metallurgist.com

Figure I.8 Quartering scheme

Figure I.9 Division of the sand sample

In the lab, the material is further divided into 4 samples taken randomly, one of 60 g used to verify the distribution of the grains, and the other three of 25 g are used to determine the iron content. All the samples are then dried in oven at 110°.

The three 25 g samples take about 30 minutes to be completely dry: at this point, the first sample is weighed up to 20 g, placed in the milling bowl and milled for 10 minutes. When removed, 8 g of the powder are mixed with 2 g of Hoechst wax and milled again for 1 minute to be pulverized. Then, the sample is pressed into a pellet and analysed with the *X-Ray Fluorescence machine* (XRF procedure) to evaluate the iron content.

The XRF is a technique that is used to determine the chemistry of the material by measuring the fluorescent X-ray emitted by the sample when invested by a primary X-ray source. This analysis takes about 10 minutes. The second sample is analysed using the same procedure as the former.

The third sample is analysed to save time in case the two results of the XRF show a difference of more than 10 ppm, and the sand must be reanalysed.

The 60 g sample takes about 1 hour to completely dry. After the sample has cooled down, it is weighed, and 50 g are analysed with the *agitator*. The equipment used at Lochaline Quartz Sand is composed of 7 sieves (710 µm; 500 µm; 355 µm; 250 µm; 180 µm; 125 µm; 90 µm). The material is agitated for 10 minutes, each sieve is individually weighed and each weight recorded. An example of the resulting grain size distribution curve is given in Figure I.10, where it can be noticed that the abovementioned range is respected.

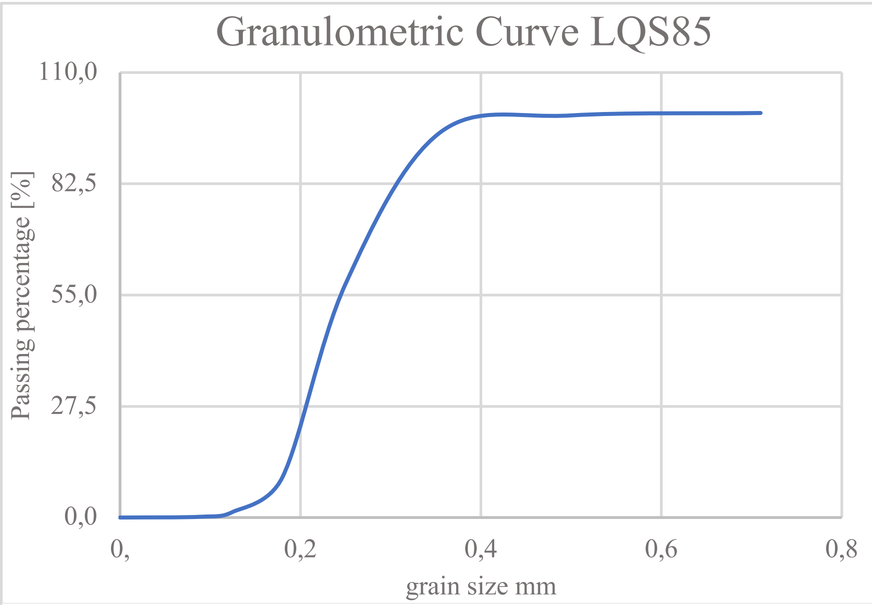


Figure I.10 granulometric curve LQS500

At this point, the tests are over, and the results are registered. Some parts of the non-used samples are kept in plastic bags and stored for six months in case of complaints about the quality of the sand, to reproduce the analysis. The whole tests take about 2 hours.

Loading of a Vessel

Since the available space for storage is limited, every two-weeks the product is transported by ship from the yards to other locations in the UK. The material is loaded on shovels and dumped inside a hopper to load the conveyor belt. The carousel is equipped with a fixed magnet to capture any present metallic object and a scale. From the main conveyor belt, the sand passes to an extender conveyor belt. The material then falls from the edge of the carousel inside the boat's storage (Figure I.11).

Figure I.11 Loading of the sand inside the vessel

During the loading of the vessel, the extender conveyor can move back and forward to load the material in a homogeneous way, to guarantee the loading is performed in an efficient way. The complete loading of the ship is obtained with the coordination of the vessel's manoeuvres, the conveyor belt movements and the sea tides.

During the loading the material is tested for detecting its moisture content. To be saleable, the water content should be lower than 5%, and then the material is analysed before and during loading to ensure that it is adequately drained. This is done thanks to two procedures: a first test before loading the vessel and a second during the load by using a "Speedy" protimeter moisture meter.

Before the loading, a sample of about 2 kg of material is taken, of which 400 g , are , dried in the oven at 110° for about an 1 hour. Then, the sample is weighed again. The difference between this second measure and the former one shows the water content in the sand. This measure can also be used to calibrate the speedy protimeter test.

During the loading of the vessel, the test is repeated approximately every 100 t of material. The sample is taken with a plastic scoop from the conveyor belt until reaching 6 g of material. The sand collected is transferred to the sample flask (Figure I.12). Then 2 scoops of calcium carbide are put in the flask's cap. At this point, the flask is carefully closed and shaken energetically for 5 seconds. Since the reaction generates acetylene, this process takes place in a ventilated area. Following this operation, the moisture is measured and the result can be read at the bottom of the flask. Then the flask and the flask's top are thoroughly cleaned with brushes. Since the calcium carbide reacts vigorously with water, it is important to follow this process carefully and in a dry environment. The results of the tests are recorded on the bill of loading which is given to the vessel's crew, and a copy is retained by the company. The spent product is deposited in a bucket.

Figure I.12 Speedy Protimeter test flask

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II. Exploitation Method

The correct choice of the mining method aims to maximize the recovery of the mineral, thus the return on investment and the the company profits, without compromising the safety of workers. It is defined considering the deposit characteristics and the required productivity, and it influences the whole organization of the exploitation area. For every mining deposit there is generally more than one method available: the best choice is the most economically viable and suitable to the exploitation conditions.

The first important distinction is between open pit and underground exploitation. Surface mining methods have lower development costs, quicker start-up time, and lower accident rates, but they are also less selective and have a larger footprint on the environment than underground methods. Thus, some deposits are mined only by surface methods while others only by underground methods, and there are also some deposits for which it is economic to start as an open pit and then change to underground exploitation. The choice of the best method depends on many parameters, such as dimensions and depth of the deposit, geomechanical characteristics, but also on environmental, social, and political conditions.

Hartman's classification (1987) of mining methods states that the most suitable method should be based on the geometry and depth of the deposit and on the ground conditions of the orebody. According to this selection, there are three main types of underground mining methods:

- *Caved*, that consist in exploiting the desired material triggering a caving effect on the surrounding zone. This method involves a high magnitude of displacement but low costs. It is generally used in moderate to weak rocks. Longwall mining, sublevel caving and block caving belong to this class
- *Supported*, involves all those methods that imply support after the exploitation, thus filling methods. Cut and fill stoping, stull stoping and square set stoping belong to this class.
- *Unsupported,* or naturally supported, are those methods that allow a safe environment during the exploitation where generally no support is needed where

only in some cases local support is necessary. These methods are suitable for strong and competent deposits. Room-and-pillar mining, stope-and-pillar mining, shrinkage stoping, and sublevel stoping belong to this class.

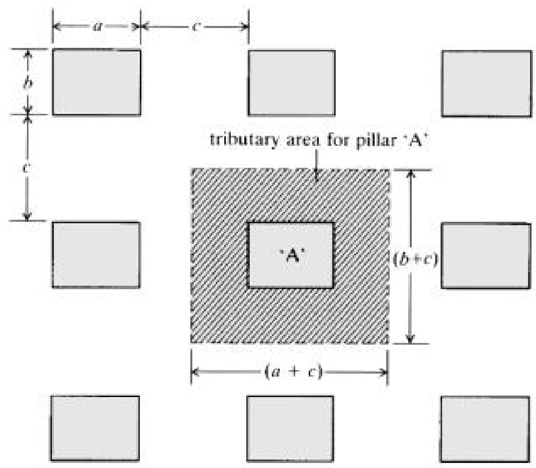
When choosing an underground method, three main issues must be taken into account:

- Ore grade control, because the economic value of the product depends on this parameter, thus the profitability of the mine.
- Ground Control: the stability of the mine is a priority to guarantee a safe working area for workers and to assure the continuity of the exploitation.
- Ventilation, to guarantee a safe working environment for workers, especially when excavating in assessments containing noxious substances, such as asbestos or quartz, or gases such as methane.

The method adopted by Lochaline Quartz and Sand is Room and Pillar. A description of the method and its application is given below.

ROOM AND PILLAR

Room and Pillar is an unsupported mining method whereby the ore body is exploited by developing a set of horizontal openings called *rooms*, and leaving in place part of the deposit. The left-behind material is called *pillar*, and it acts as a support for the roof (Figure II.1). It is generally realized in flat homogeneous ore bodies but can also be realized in deposits with dip angle up to 45°.



Source: Brady and Brown, 1992

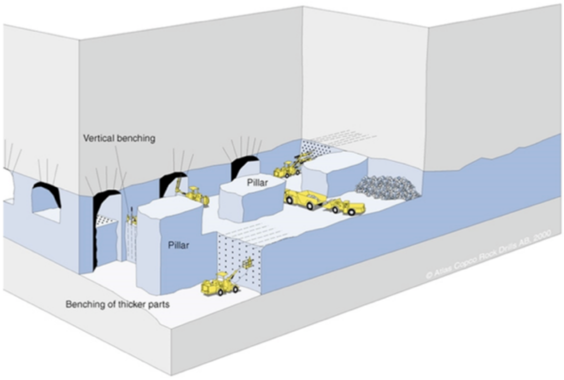
Figure II.1 plan view of a room and pillar mine

The general aim of room and pillar is to recover the maximum amount of ore without compromising the stability of the whole mine. This means that the pillar left behind is as small as possible. The dimensions of the rooms and the pillars depend on the strength of the ore body, the roof and the floor.

This method has been widely used for the last 150 years, both in soft materials and hard rock-masses, because of its versatility, low costs and safety. The versatility is due to the possibility of applying some variations to the general pattern in the case of bad quality material and possible instabilities.

There are three typical varieties of the room and pillar method, depending mostly on the geological asset of the ore body.

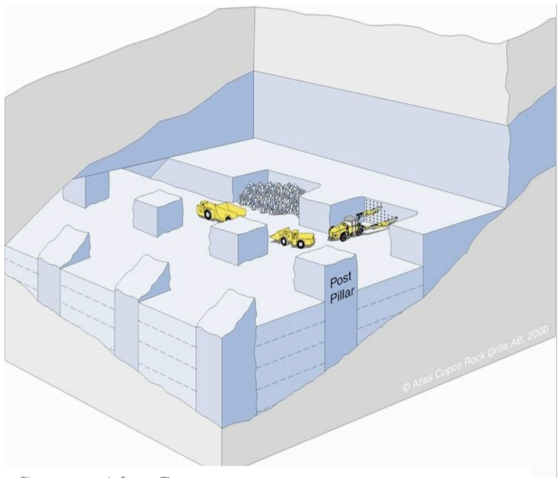
C*lassical room-and-pillar mining* is generally applied to flat bedded deposits (Figure II.2). It requires minimum development work; this is because the excavation of the roadways can be generally combined with the excavation of the ore body, and the roadways for transportation and communication are established inside production stopes.



Source: Atlas Copco

Figure II.2 Classic room-and-pillar mining scheme

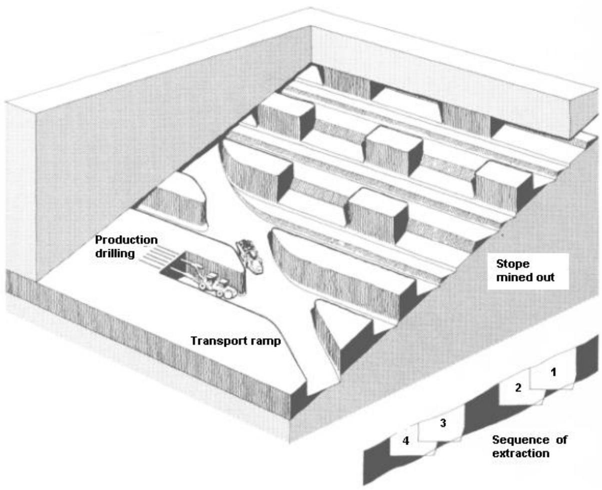
P*ost-room-and-pillar mining* is generally applied to ore bodies with a dip angle of 20° ÷ 55° and large vertical extension. This variation allows the excavation of the deposit in horizontal slices from bottom to top, where each excavated stope is then backfilled (Figure II.3). In this way, the pillar has a better supporting ability and allows a higher recovery than the classic room-and-pillar method. Generally, production is obtained with mechanized mining methods. Post-pillar mining is a good combination of the room and pillar method and the cut and fill method.



Source: Atlas Copco

Figure II.3 Post room-and-pillar mining scheme

The third variety is *step room-and-pillar mining* that is applied when the ore body has a very steep dip and rubber-tired vehicles cannot travel. It is generally used in tabular deposits with 2 m ÷ 5 m of thickness and dips of 15° ÷ 30°. The stope is oriented along the strike and is crossed by a series of parallel transport drifts. This configuration allows the excavation of the ore body and to collect it to the transport drifts with trackless vehicles (Figure II.4).



Source: Atlas Copco

Figure II.4 Step room-and-pillar mining of inclined orebody

One of the key parameters for the success of the room and pillar method is the design of the pillar. The pillars should be big enough to guarantee the stability of the roof, but not too big so as to avoid reducing the profitability of the mine by leaving excessive valuable material in place. The cross-section depends on the characteristics of the material. Pillars can be cylindrical, rectangular or squared, depending on the mine's needs and the excavation technique. Generally they follow a regular pattern, but in some cases their geometry can be conditioned by the geology of the ore body, with the objective of obtaining the best recovery.

The dimensions and shape of the pillar depend on many factors, such as:

- The type of material: in hard rock-masses, the pillars are generally smaller than in softer materials.
- The geology, as mentioned above.
- The stresses in the pillar due to the weight of the overburden ($\sigma_z = \gamma \times z$).
- The stresses in the pillar due to the excavation of the rooms which are affected by the excavation technique.

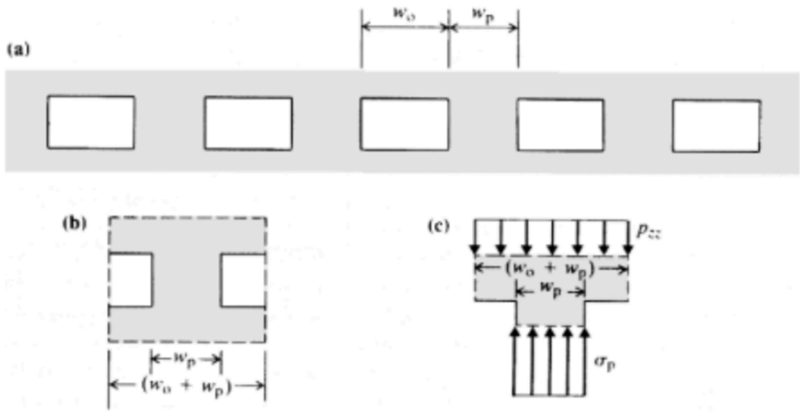
In order to obtain the optimum pillar dimension, the design is generally strength-based. One of the most used design methods is the *Tributary Area Method*. This method provides a first-order estimation of the average axial pillar stress, which is measured as the stress of the rock prism weighing on the pillar. The pillar stress depends on the pillar width (w_p), the opening width (w_o) and the vertical stress of the pre-mining stress field acting on the pillar (σ_z):

$$\sigma_p = \sigma_z \cdot \left(\frac{w_p + w_o}{w_p}\right) \tag{II-1}$$

In case of squared room and pillar system (Brady and Brown, 1985), the formula (II-1) becomes:

$$\sigma_p = \sigma_z \cdot \left(\frac{w_p + w_o}{w_p}\right)^2 \tag{II-2}$$

A scheme of all the parameters entering the tributary area method is shown in Figure II.5.



Source: Brady and Brown, 1992

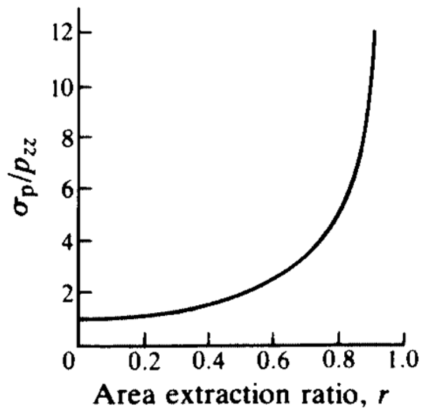
Figure II.5 Tributary Area Method Scheme

A parameter of practical interest for the determination of the average pillar stress, especially in uniform thickness pillars, is the *area extraction rate* (r). This coefficient is the ratio between the area extracted and the total area of the ore body. By taking Figure II.1 as a reference, the area extraction ratio is defined as:

$$r = \frac{[(a+c)(b+c) - ab]}{(a+c)(b+c)} \qquad \text{II-3}$$

By this way the (II-1) can be written as:

$$\sigma_p = \sigma_z \cdot \left(\frac{1}{1-r}\right) \qquad \text{II-4}$$



Source: Brady and Brown, 1985

Figure II.6 Variation of the extraction ratio with the normalized pillar strength

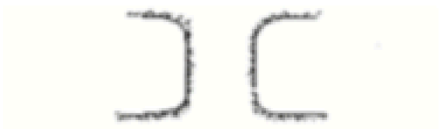
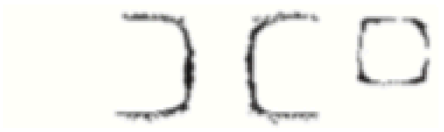
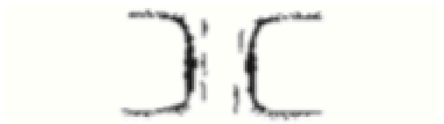
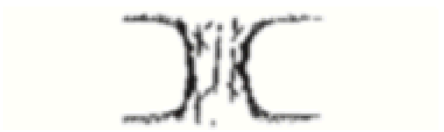
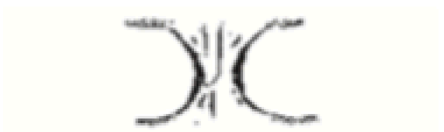
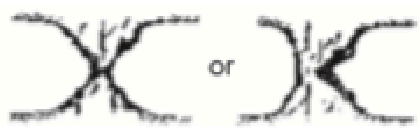
This coefficient can also be used to have an indication on the dimensions of the pillars. In fact, by increasing the extraction ratio, therefore reducing the pillar dimensions, the stresses acting on it will increase (Figure II.6). When the recovery factor is equal or greater than 75%, the slope of the curve increases steeply. Therefore, it is recommended not to reduce the dimensions of the pillars or increase the width of the span, in order to avoid damage to the pillars. An extraction ratio greater than 75% is rare and used only in the supported methods. Characteristics values for the recovery factor for three different deposits are given in Table II-1.

Table II-1 Characteristics recovery factor

Stone and aggregates	75 %
Coal	60 %
Potash	50 %

It is clear that the tributary area method is a simple estimation that gives an average value of the state of axial stress in the pillar. The actual in-situ stress should be defined with a method that considers the strength or peak resistance of the pillar to the axial compressive strength.

The strength of the pillar is related to its volume and geometry. It is difficult to estimate how the pillar stress can damage the pillar, especially in the long term, but the conditions of the pillars can be monitored in-situ with monitor instruments and visual checks. Carmark (2001) defined a rating system (Figure II.7) from 1 (no visual stress) to 6 (pillar failure). When spalling on the corners and fractures on the wall (rating 3) are noticed, the pillar must be reinforced. The suggested reinforcement is a grouted rebar that will avoid further deterioration, allowing the pillar integrity for a long period.

Pillar Rating	Pillar Condition	Appearance
1	No indication of stress-induced fracturing. Intact pillars.	
2	Spalling on pillar corners; minor spalling of pillar walls. Fractures oriented subparallel in walls and are short relative to pillar height.	
3	Increased corner spalling. Fractures on pillar walls more numerous and continuous. Fractures oriented subparallel to pillar walls and lengths are less than pillar height.	
4	Continuous, subparallel, open fractures along pillar walls. Early development of diagonal fractures (start of hourglassing). Fracture lengths are greater than half of pillar height.	
5	Continuous, subparallel, open fractures along pillar walls. Well-developed diagonal fractures (classic hourglassing). Fracture lengths are greater than half the pillar height.	
6	Failed pillar, may have minimal residual load-carrying capacity and be providing local support to the stope back. Extreme hourglassed shape or major blocks have fallen out.	

Source: Carmack et al. 2001.

Figure II.7 System for rating pillar strength

ROOM AND PILLAR IN LOCHALINE QUARTZ AND SAND

Lochaline Quartz and Sand deposit is a shallow nearly horizontal sandstone deposit of 5 m ÷ 8 m thick, as reported in chapter 1. The material has relatively homogeneous geomechanical properties. This geological configuration allows an easy exploitation with continuous room and pillar method. In fact, the mine is developed by leaving squared pillars extended on one level only. The mine has a wide extension divided into eight areas that are inter-connected through drifts. When the advancement intersects a fault or there is an excessive inflow of water or the distance from the muckpile to the plant becomes uneconomical, the mine generally decides to stop the exploitation in that direction and to start a new area.

As much as possible, the pillars follow a regular pattern (Figure II.12), but sometimes, due to the geology and/or the stability of the roof, the cross-sections may change in shape and dimensions.

GROUND CONTROL

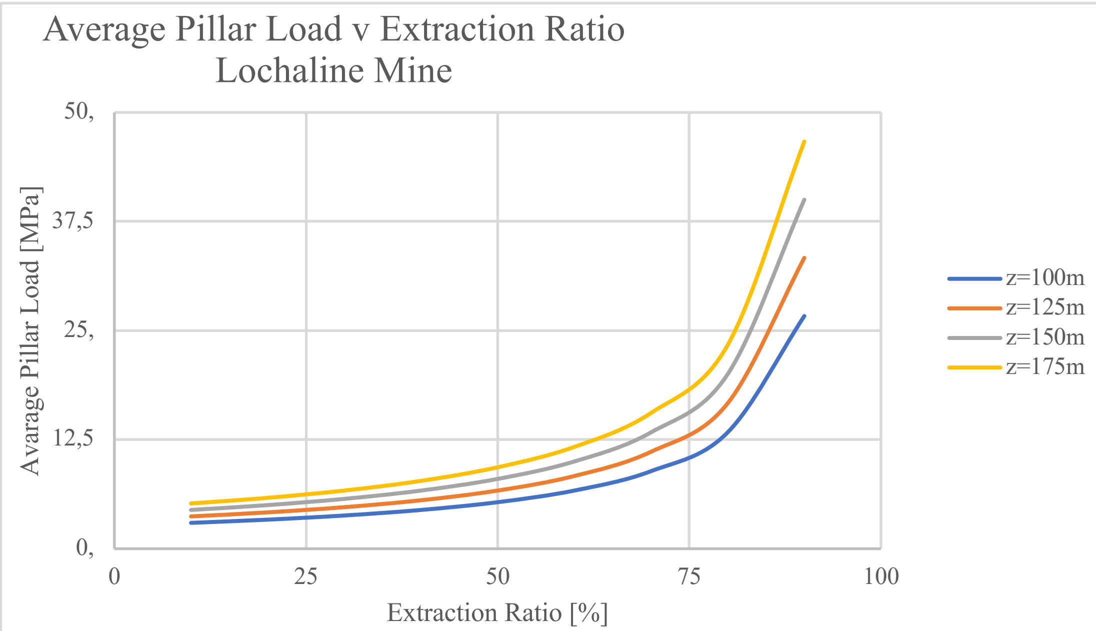
The mine layout entails the exploitation in intact rock strength at shallow depths (up to 165 m), thus vertical loads are low, expected to be in the range 2,75 MPa ÷ 3,75 MPa. The stress regime is considered isotropic. Thanks to this favourable configuration and to no significant interaction with adjacent works, the room can be safely excavated in any direction without major effects on the horizontal stress.

The mine adopts squared pillars, whose dimensions, as well as their span, can vary depending on the depth of excavation and on the opening height: 5 m for the production of LQS85 and 7 m for the production of LQS500. The suggested dimensions, depending on the depth of excavation, are listed in Table II-2. Those results were obtained considering a height of 5 m, normally adopted for the exploitation of LQS85, and 7 m, used for the exploitation of LQS500, and a maximum room width of 7,5 m, in order to guarantee the stability and sufficient space for the manoeuvres of the machines.

Table II-2 Suggested pillar sizes depending on the depth of exploitation

Depth [m]	Pillars Size [m x m] 5 m height	Pillars Size [m x m] 7 m height
100	7 x7	8 x 8
110	7 x 8	8 x 9
120	8 x 8	9 x 9
130	8 x 9	9 x 10
140	9 x 9	10 x 10
150	9 x 10	11 x 10
160	10 x 10	11 x 11

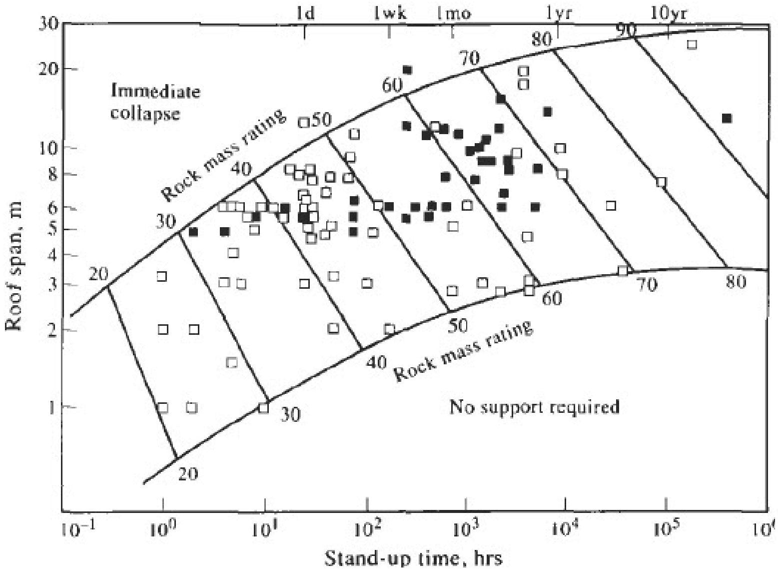
The mine usually adopts pillars with cross-section of 10 m x 10 m. With this figure, the recovery reaches 67,3% and the pillar stress ranges between 8,15 MPa and 14,27 MPa, depending on the depth, as it can be seen in Figure II.8. These dimensions assure an



Source: Graham Daws Associates - Notes on a visit to Lochaline Mine 22nd February 2017

Figure II.8 Variation of the pillar stress with the recovery factor acceptable safety factor (≥ 1,6 as requested by the international standards) up to depths of 150 m. For higher depths, pillars of 11 m x 11 m are suggested, but they may cause operational problems. A proposed alternative is to use pillars of 10 m x 12 m, with a reduction of the extraction ratio to 64,8%. At the moment, the working area has a depth between 105 m and 125 m.

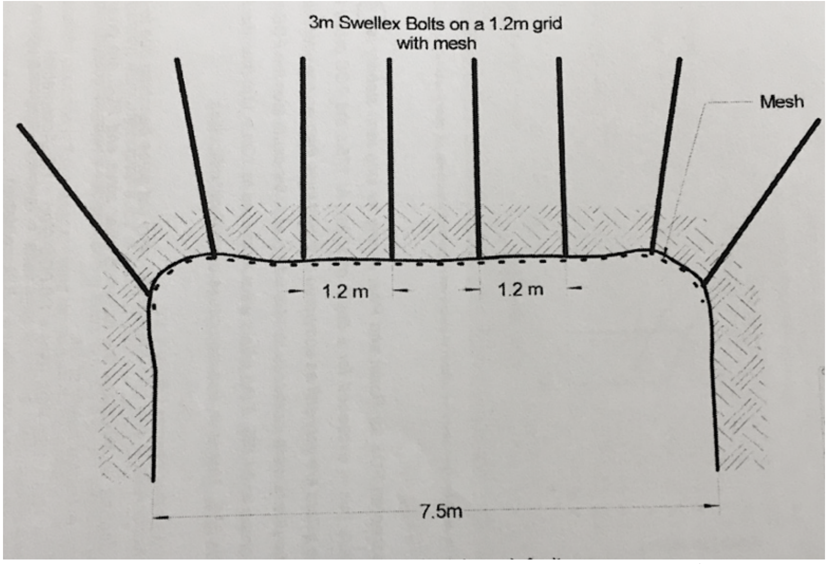
The shallow depth and the intact rock strength make it very unlikely that floor movements or pillar failure will occur. As for the roof, generally, no support is needed. This was confirmed by the evaluation of the rock mass rating (RMR) with the GRMCS empirical method (Geomechanics Rock Mass Classification System) proposed by Bieniawski (Figure II.9 – 1989). This method correlates the stand-up time with the roof span and the RMR. These data have been taken from evaluations of the roof of the old parts of the mine first opened in 1940. From this historical evaluation, an opening of 20 m has resulted standing and stable for at least 10 years. The RMR is 90, which, according to Bieniawski's classification (1989), is characteristic of very good rock that can be excavated at full face with a 3 m pull and without systematic support.



Source: Bieniawski, 1989

Figure II.9 Stand Up Time from GRMCS

In the mine under study, local supports are applied when there is an intersection with a fault or dykes, when scaling is not sufficient to guarantee the safety of the area, or when there are problems of small instabilities such as debris fall from the roof. Graham Daws Associates Ltd. (2016) after a geotechnical study, proposed the supporting scheme suggested in Figure II.10, composed of swellex bolts and wire mesh.



Source: Graham Daws Associates - Notes on a visit to Lochaline Mine 3rd August 2016

Figure II.10 Wire mesh + bolting support model scheme

Swellex dowels are fully connected bolts, that do not need any grouting insertion. This type of element is composed of a steel tube folded into a C Shape. No pushing force is required during the installation and it is activated by injecting high-pressure water (≈ 30 MPa) into the pipe: the water expands the element until it is in contact with the borehole's walls, creating a strong connection with it.

An important aspect of ground control is the monitoring of possible instabilities. In the mine, this is generally accomplished through a visual check by means of a daily and a weekly inspection route. The inspection path changes according to the evolution of the mine. If any spalling of pillar side (Figure II.7 – rating 2) are experienced, the area is immediately isolated and access prevented until remedial work has been done. If specific features need monitoring, the following measures can be applied according to the suggestions of Graham Daws Associates Ltd.:

- Roof to floor convergence measurements from convergence pins
- Wooden wedges lightly tapped into open joints across the roof, followed by visual observation then used to check if wedge falls

- Steel or brass pins set on either side of a joint line in the roof or rib. The measurements are obtained by means of digital callipers.

If any movement is monitored, or wooden wedges fall, a support system should be installed after consulting an experienced geotechnical engineer.

A further monitoring technique adopted by Lochaline Quartz Sand is the single point rotatory Tell Tale (Figure II.11). This device is installed every time the span between the pillars is greater than 12 m. If the measurement on the tale is higher than a certain figure, called action level, mining activity should be interrupted and the area examined. If no immediate cause is detected, some action must be taken. If the area is not operating, it should be fenced off and declared an “exclusion zone”. If the hazardous area is operative, a system of rock bolts should be installed.

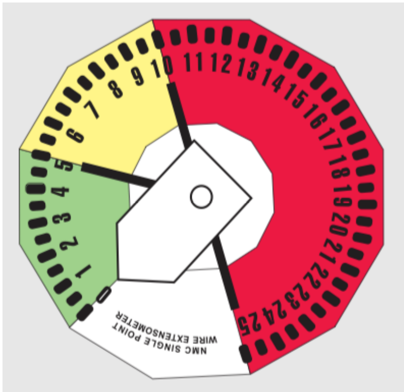


Figure II.11 Single Point Rotatory Tell Tale - Scheme

The action level depends on the thickness of the roof beam and on the strain shaft limit of the sandstone roof, that is the point at which failure of the rock starts. In fact, the action level is the product between the strain shaft limit and the anchoring length of the single point Tell Tale, the suggested values are listed in Table II-3.

Table II-3 Suggested Action Levels

ROOF BEAM THICKNESS [m]	ACTION LEVEL [mm]
0,9	2,5
1	3,25
1 ÷ 1,5	4,75
1,5 ÷ 2	6,5
2 ÷ 2,5	8

In case rock bolts have to be installed, it is suggested that a pattern of at least 1 bolt per 1,5 m^2 is chosen. The length of the bolts should be at least twice the thickness of the roof beam, and in any case longer than 2,4 m.

VENTILATION

Ventilation is an important issue in underground mines because it ensures a safe environment for workers in order to ensure good working conditions with reasonable comfort (temperature and moisture). The goal is to guarantee a sufficient airflow in the working area to provide oxygen and dilute the eventual contaminants to safe concentrations and then remove them.

Historically, ventilation was required only to provide a sufficient airflow to replace the O_2 consumed by workers (≈35 m^3/h/man). Nowadays it is also used to deal with dust, temperature and gases. It has to guarantee that the concentration of noxious substances is below the safety limits, and that the concentration of oxygen is higher than 19%. Generally, the amount of air needed for the dilution is higher than that needed to replace the oxygen consumed by both workers and machines.

The most common types of pollutants in underground mines are listed below; they can arise from natural conditions (depth, geology, gases contained in the rock, etc.) or be due to engineering decisions (mining method, grain size of the muckpile, types of power used, number of vehicles in the mine, etc.) :

- Gases
- Product of combustion
- Dust
- Heat and humidity
- Radioactive solid and by-products
- Diesel particulate

In order to guarantee a successful ventilation both in quality and quantity of the air, it is important to know the possible causes of pollutants and to manage them. The general rule is to control emission and dispersion of contaminants, dilute the substance and protect the workers with a suitable equipment.

Mine ventilation is designed by applying the principles of fluid mechanics and thermodynamics to the flow of air. The air should enter the mine from the atmosphere through openings (i.e. adits, shafts, ecc.), it flows around the mine and it exits by creating a differential pressure between the intake and return openings of the mine. To guarantee this process, the mine should have a ventilation system composed of interconnected airways, fans and control devices. Fans induce the airflow in mines. There are two main types: main fans and booster fans. Main fans handle the whole mine airflow and are commonly installed on the surface, but it is possible to install them underground. Booster fans boost airflow energy to allow the circulation of air over greater distances. Generally, they are installed in specific areas of the mine.

The mine under study uses as the main fan, an axial fan that provides an airflow of about 39 m^3/s. It is positioned in adit 9, area C (Figure I.12 – orange circle). The airflow follows a nearly horseshoe path, passing through the open working areas, exiting through the main entrance of the mine, adit 7 in area B (Figure I.12 – blue circle). This path is guaranteed using brattices to close openings and direct the flow to the desired areas. Inside the current working area, area 3, the ventilation is assured by using a booster axial fan, positioned in the Giorgio Drift, approximately in the middle of the area.

The ventilation aims to assure a presence of O_2 equal or greater than 19%, a temperature between 10° and 15°, and the suggested limits of the air velocity (Table II-4), considering that in working areas the lower threshold limit is 0,3 m/s :

Table II-4 Recommended maximum air velocity

AREA	**VELOCITY [m/s]**
Working Faces	4

Conveyor Drifts	5
Main Haulage Routes	6
Smooth Lined Main Airways	8
Hoisting Shafts	10
Ventilation Shafts	20

Source: McPherson's ventilation planning

As the mine advances, the airflow changes so as to assure a safe environment for the workers and also to have a quicker fume extinction after a blast. It is checked monthly with a hand-held instrument. The check consists of measuring the airflow in determined points whose cross-section is known. Each measurement provides the temperature and the average airspeed. Based on the results obtained, general improvement plans are considered and the closure or opening of some rooms is planned.

In the mine under study, there is no hazard due to dangerous gases, as methane is absent. Due to the room and pillar geometry, there is a constant airflow with no build up and dangerous accumulation of CO_2. This is also guaranteed by the fact that, according to UK's regulation, the ventilation must be guaranteed by fans when using diesel vehicles, such as the dumpers and land rovers, in the mine.

Since the material exploited is a sandstone for the production of silica sand with very high concentration of quartz, the inhalation of silica dust represents a hazard for Lochaline Quartz Sand mine. To reduce the risk, the workers in presence of dusty environments have to use sealed vehicle systems with air conditioning and filtration. If an operator is not working in a sealed vehicle system, they must wear the appropriate PPE. The use of water is the preferred method for a better control of dust, such as during drilling.

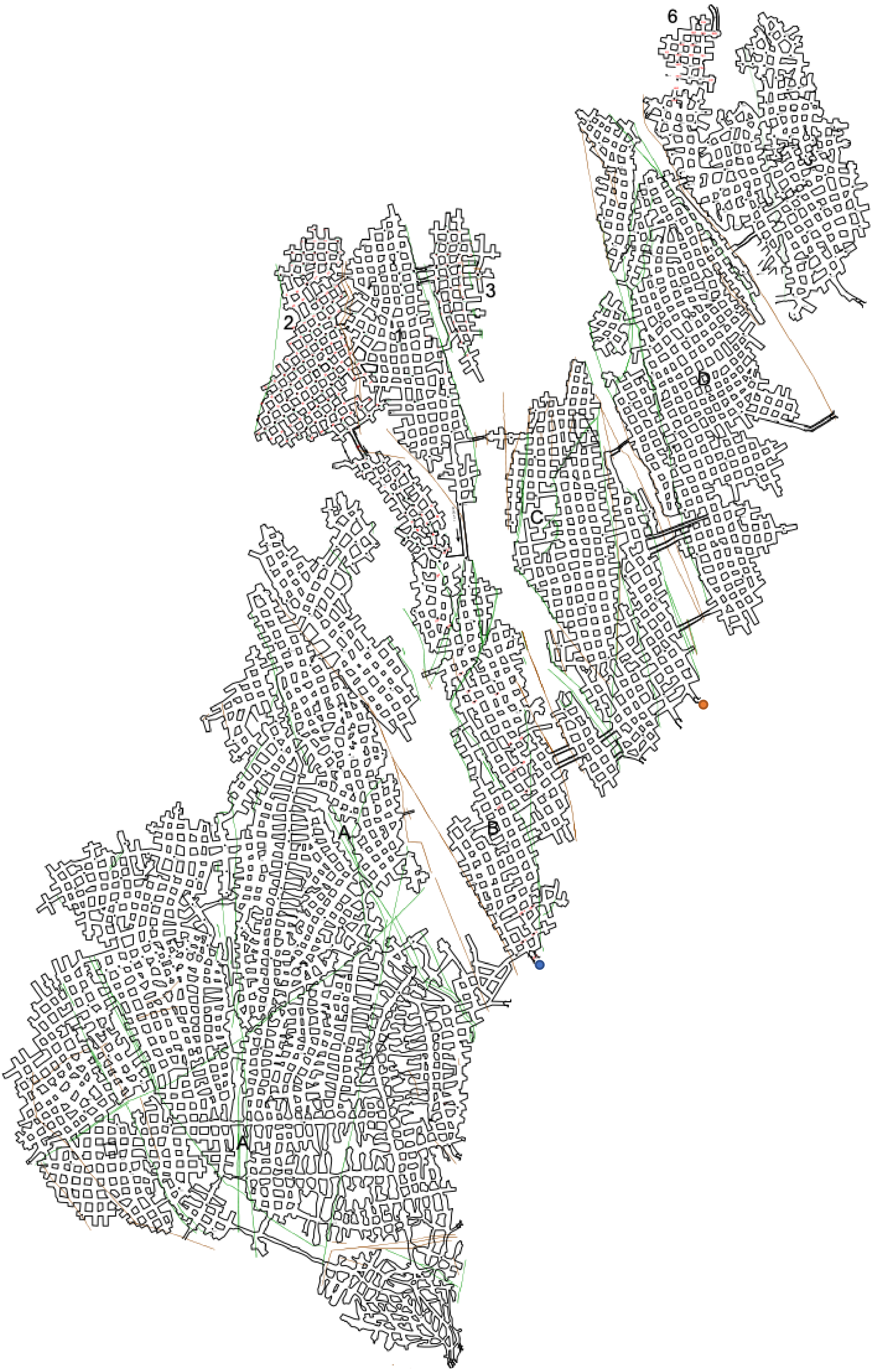


Figure II.12 Lochaline Quartz Sand Mine Map

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III. EXCAVATION TECHNIQUE

The excavation technique consists of the type of equipment chosen to carry out the exploitation. The choice of technique must be the most suitable for the site, both on a technical and economic level. For the exploitation of hard rock, conventional excavation techniques are generally used, and two main families can be distinguished:

- Drill and blast
- Mechanical excavation, that includes the machineries that are able to break the rock using mechanical cutting tools, i.e. the road header and the hydraulic hammer.

The techniques used in Lochaline quartz Sand are drill and blast and hydraulic hammer. In this chapter they will be explained in detail.

DRILL AND BLAST

Drill and blast is an excavation technique widely used both in mining and civil industries because it allows the excavation of high volumes of rock in a short amount of time. It is very versatile, allowing the easy excavation according to different geometries.

The explosives allow a very rapid chemical reaction that produces a high pressure shock wave, heat and gaseous products. To be productive, the explosive must be controlled, therefore the reaction needs to happen in a designated time. The reaction should start only after the initiation which can be either thermal, mechanical or electrical.

Wyllie and Mah (2004) describe the action of the explosive on the rock as following:

> *When an explosive is detonated, it is converted within a few thousandths of a second into a high temperature and high pressure gas. When confined in a blast hole, this very rapid reaction produces pressures, that can reach 18 000 atm, to be exerted against the blast hole wall. This energy is transmitted into the surrounding rock mass in the form of a compressive strain wave that travels at a velocity of 2000–6000 m/s.*

As the strain wave enters the rock surrounding the blast hole, the material for a distance of one to two charge radii (in hard rock, more in soft rock) is crushed by compression (Figure III.1, (a)). As the compressive wave front expands, the stress level quickly decays below the dynamic compressive strength of the rock, and beyond this pulverized zone the rock is subjected to intense radial com- pression that causes tangential tensile stresses to develop. Where these stresses exceed the dynamic tensile breaking strength of the rock, radial fractures form. The extent of these fractures depends on the energy available in the explosive and the strength properties of the rock, and can equal 40–50 times the blast hole diameter. As the com- pressive wave passes through the rock, concentric shells of rock undergo radial expansion resulting in tangential relief-of-load fractures in the immediate vicinity of the blast hole.

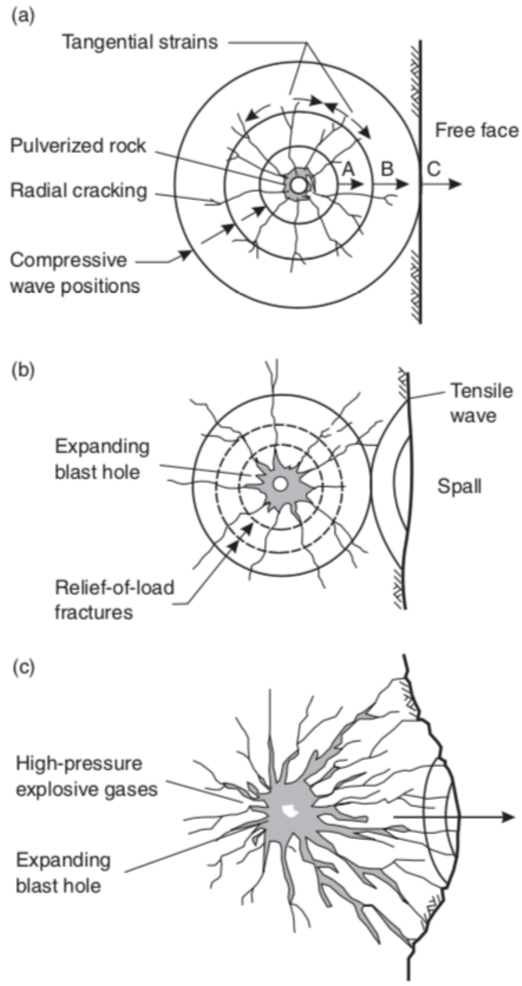


Figure III.1 Mechanism of rock breakage by explosives

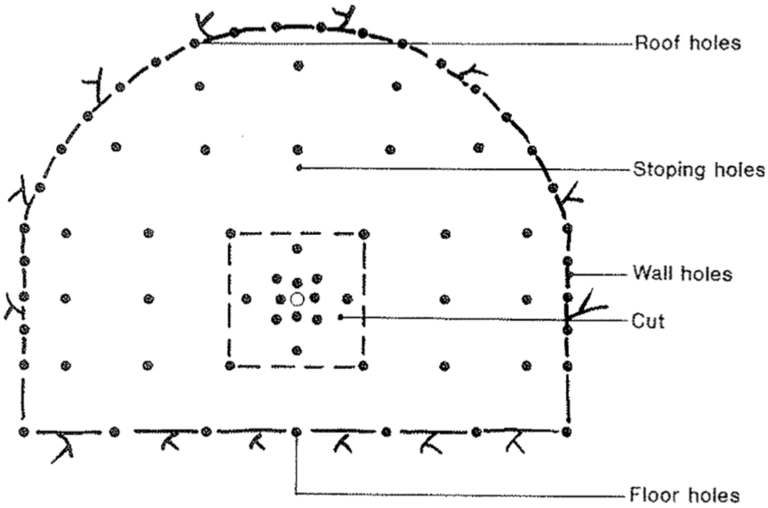
These concentric fractures follow cylindrical surfaces, and are subsequently created nearer and nearer to the free face. When the compressive wave reaches a free face, it is reflected as a tensile strain wave. If the reflected tensile wave is sufficiently strong, "spalling" occurs progressively from any effective free face back towards the blast hole. This causes unloading of the rock mass, producing an extension of previously formed radial cracks (Figure III.1, (b)). Rock is much weaker in tension than compression, so the reflected strain wave is particularly effective in fracturing rock.

The process of fracture formation due to strain wave energy typically occurs throughout the bur- den within 1 or 2 ms after detonation, whereas the build-up of explosive gases takes in the order of 10 ms. As *the rock is unloaded due to radial expansion and reflection of the compressive wave, it is now possible for the expanding gases to wedge open the strain wave-generated cracks and begin to expel the rock mass (Figure III.1, (c)).. This stage is characterized by the formation of a dome around the blast hole. As wedging action takes place due to the heaving and pushing effect of the expanding gases, considerably more fracturing occurs due to shear failure as the rock mass is expelled in the direction of the free face. In highly fissured rocks, fragmentation and muck pile looseness are caused mostly by expanding gases.*

In the quarry under study the technique used is tunnelling blasting. Tunnelling consists of the excavation of volumes of rock characterized by the cross section and the pull, that is the length of the advancement. The excavation is guaranteed by the round, a set of charged holes that work together to blast the said volume of rock.

The tunnel blasting is characterized by only one available free surface; therefore, a second free surface has to be created for an efficient blasting. This operation is called attack stage and consists of creating a cavity, generally in the central part of the cross section, called the cut. The most common cuts are parallel hole cut, V-cut and fan-cut. Once the new opening is formed, the stoping begins. This phase consists of the blasting of a determined number of charges properly located with respect to the cut, until the desired contour is reached. The final stage is obtained with the stoping or production holes and the contour holes (roof, floor and wall holes).

Figure III.2 represents a general layout of the blasting scheme, with the nomenclature of each functional group of holes.



Source: Olofsson, 1991, Applied explosives technology for Construction and mining

Figure III.2 Blasting scheme and nomenclature for tunelling

The success of drill and blast lies in the executive blasting design project of the round, or blasting scheme, that sets:

- Hole diameter
- Hole length, that influence the pull of the blast.
- Hole direction (inclination)
- Hole position
- Initiation system, which defines the type of detonator
- Charging consists of a defining the type and quantity of explosive. The amount of explosive used changes for each functional group of holes.
- Timing, that is the delay sequence chosen.

The specific consumption of explosive or powder factor (P.F.) can be used to characterize the round. This parameter is equal to the ratio between the amount of explosive (Q) and the volume of rock to be blasted (V) ((III-1).

$$P.F. = \frac{Q}{V}\left[{}^{kg}/_{m^3}\right] \qquad \text{(III-1)}$$

The specific consumption of explosive is mainly influenced by the cross section, more than the properties of the rock. A smaller cross section will be blasted with a higher P.F. Generally speaking, bench blasting will have a lower powder factor than tunnelling excavation: this is because in a tunnel blasting the cut works in non-optimal conditions, therefore a higher charge density is needed to be successful.

Mancini and Pelizza (1969), based on statistical analysis of a wide collection of data on civil and mining tunnels driving, proposed the correlation formula (III-2) to predict the P.F. when the rock type, the explosive type and the round type are known.

$$PF \cong \left(\frac{10}{S} + 0{,}6\right) \cdot A \cdot B \cdot C \tag{III-2}$$

where

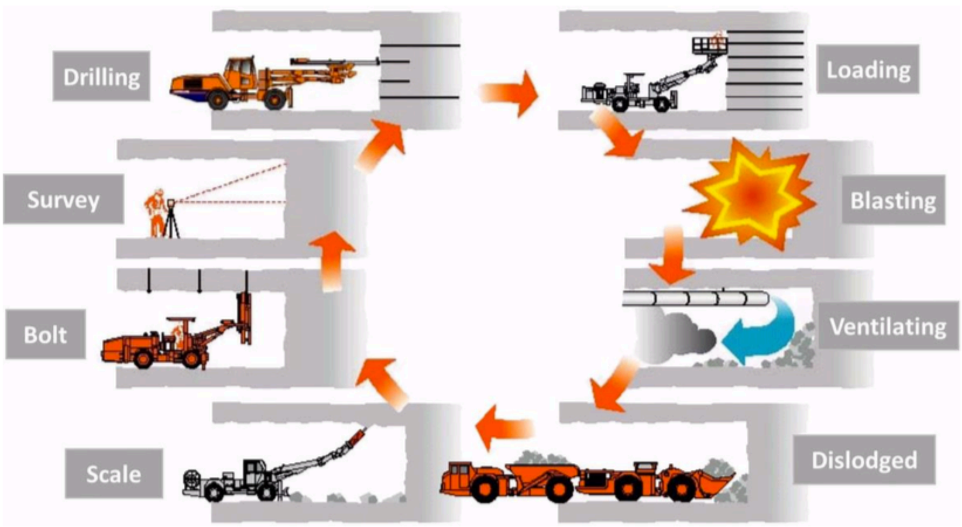
S= blasting surface [m^2]

A = empirical coefficient depending on the type of rock, based on the Protodyakonov class (Boky, 1967);

B = empirical coefficient depending on the type of explosive;

C = empirical coefficient depending on the type of cut.

Drill and blast in underground consists of a series of cyclic operations. The steps are the following (Figure III.3):



Source: http://www.railsystem.net/drill-and-blast-method/

Figure III.3 Drill and Blast scheme

- *Drilling* consists in performing the drilling pattern on the face. The drilling pattern is defined before the operation, by using the parameters mentioned before. The design depends on many factors, such as the type of explosive, the type of rock and the maximum size of the muck.
- *Charging.*
- *Blasting*.
- *Fumes Clearing* is the amount of time needed for the fumes to escape from the face. The time needed for this operation depends on many factors such as the type of explosive used and the ventilation inside the mine. Generally, to avoid stopping the production for too long, the blasting is done at the end of the shift or before a scheduled break.
- *Loading and transport* of the mucking from the face.
- *Scaling* consists the identification and removal of unstable blocks.
- *Stabilization and monitoring* consist of the stabilization of the roof and walls with supports and checking for movements and displacement of the rock.

DRILL AND BLAST IN LOCHALINE QUARTZ SAND

The blasting scheme of the mine under study depends on the type of sand produced, as will be shown in the following chapter. When drilling a full face, the general number of blast holes varies from 44 to 55 to excavate a cross-section of maximum 7,5 m width and a variable height. The theoretical pull is 3 m, respecting the limit of 3,5 m given by the Mining Regulation 2014 (Regulation 32).

The drilling is performed with a two booms jumbo, the Atlas Copco 282s (characteristics in Appendix A). The two booms allow a reduction in the overall drilling time (Figure III.4)

Figure III.4 The two booms drilling the cut

The tool used for the drilling is a button tool type (Figure III.5), with all-round drill bit, ideal for medium-hard/hard rock formations. It can create holes of 38 mm and variable lengths, up to 3,4 m. The bit has front and side channels for the inflow of high-pressure water, used for the discharge of debris inside the hole.

On average, drilling a hole takes about 40s. Since the number of holes in the drilling pattern varies depending on the exploitation of LQS85 or LQS500, as will be further explained in the next chapter, drilling the whole round requires between 40 minutes to 60 minutes. This time average takes into consideration all the manoeuvres of the boom and the possibility of problems arising during the operation, i.e. the presence of hard rib or faults that require a slower excavation to avoid wear on the bit and damage to the rod.

Figure III.5 Button bit

The drilling process is composed of:

- check of the roof conditions by drilling a hole in it. This will give a clearer idea of the distance between the sandstone and the hard rip and mudstone.
- Scaling of the face, using high-pressured water and the rods, to remove small unstable blocks, allowing a more regular blasting face and a safer working environment for the shot fires.
- Drilling of the cut.
- Drilling of the roof and the production holes.
- Drilling of the floor.

After the drilling, each hole is then charged. The explosive used for the blast is the Senatle™ Powerfrag™, that is an emulsion packaged in 32 mm cartridges of 400 g. The technical properties are listed in Table III-1. Emulsions are composed of 90% of a concentrated solution of nitrates in water and 10% of oils, waxes and paraffins. They are characterized by a high velocity of detonation, good fume characteristics and allow a good efficiency in wet and dry environments. The cartridges are bought in boxes, each box contains 60 sticks. The boxes are stored in the mine deposit.

Table III-1 Explosive Technical Properties

	SENATEL™ POWERFRAG™

Density [g/cm^3]	1,15÷1,23
Diameter [mm]	32
Hole type	Wet and dry
Velocity of detonation [m/s]	3500÷5300
Explosive heat [kJ/kg]	3191
CO_2 Output [kg/t]	184
Gas Volume [l/kg]	929

The initiation system used consists in electrical detonators Dynadet™ – C1- 25 ms by Orica. The characteristics are listed in Table III-2. Electric detonators are a versatile type of initiator. In cases where there is the presence of water, some attention must be paid to the end part of the leading wires making sure they do not come into contact with water because it may cause a dispersion of electricity, thus the exploder would not be able to set off the blast. To avoid this problem the producer provides the leading wires with a plastic waterproof cap. Lochaline Quartz and Sand mine generally uses 11 delays with a delay sequence of 25 ms.

Table III-2 Detonator Technical Properties

	DYNADET™ - C1- 25ms
Shell	Aluminium
Base Charge	RDX/PETN
Delay Interval [ms]	25
Delay numbers	1 - 20, 24, 32, 40, 48, 56, 64, 72, 80
Wire Diameter [mm]	0,6
Wire Length [m]	6
Total Resistance [Ω]	1,9 ± 0,3

Once the initiation circuit is completed, before blasting it is necessary to check with an Ohmmeter whether the resistance is correct. The resistance for each component is listed in Table III-3. The exploder used is the Beethoven Mk 7, which ensures the correct initiation of a maximum of 5 blasts at the same time.

Table III-3 Resistance values

	RESISTANCE
Detonator	1,6 Ω
Shot firing cable	0,06 Ω/m
Round	70÷90 Ω

For the stemming, clay dummies are used.

The mine generally exploits an area with no nearby target buildings: this allows no limit to the charge per delay. Since the exploder does not allow more than 5 blasts per time, and generally the maximum number of holes that blast with the same delay per round is 7, the general CPD is taken as 70 kg/delay.

The charging operation is carried out by three shot-firers and a forklift with a platform (Figure III.6), for the charging of the higher holes. The entire operation lasts an average of 45 minutes.

Knowing the type of rock, the explosive used, the blasting scheme and using the Mancini and Pelizza formula ((III-2), a reference value of the P.F. can be obtained, for the production of the two sands. The blasting scheme for LQS85 should be P.F. = 0,91 kg/m^3, while for the production of LQS500 the reference value should be P.F. = 0,81 $kg/m^{3.}$

Figure III.6 Charging of the roof holes

To guarantee the required fumes clearing time, the blasting generally happens at the end of a shift or mid-morning, depending on if the shot-fires work on the night or day shift. There is no rule on the time needed for this operation and it depends on the various factors already mentioned. The Mine Regulation 2014 suggests waiting at least 1h before entering the blasting area.

To ensure safety, after the fumes clearing, an analysis of the air is performed by checking the concentration of CO, CO_2, NO_2 and O_2 in the air. The measurements are then compared with the E40/2005 Workplace exposure limits (2018). If the concentration levels are below the limits, the scaling operation begins. In the mine, after a visual check, the scaling can be carried out manually or mechanically, depending on the roof's conditions. The mechanical scaling is performed with a hammer mounted on a CAT 308.

The hauling and loading of the muckpile is carried out after the scaling using one or two shovels and four dumpers. The shovels load the muck on the dumpers that transport the material to the production plant. This operation is the same both for drill and blast and hammer exploitation and is not considered in the study. The machines used are:

- CAT 938K shovel
- Volvo L90G shovel
- Two CAT 730 dumpers
- CAT 725 dumper
- Bell 25E dumper

HYDRAULIC IMPACT HAMMER

The hydraulic hammer is a very versatile machine that can be used for many purposes such as local demolitions, breaking oversize boulders and excavation of tunnels. The principle behind this machine is simple. A piston is moved against an element that acts as a pick on the rock. To guarantee the necessary energy to break the material, the hammer is equipped with an oil accumulator able to supply the needed oil volume in a short time.

According to Gertsch (2000), the rock cuttability can be expressed with two parameters: specific penetration (SP) that is the amount of indentation obtained for a given applied force F, and specific energy (SE) defined as the energy required to excavate a unit volume of rock. Some studies have shown that specific energy is inversely proportional to impact energy (Wayment and GrantMyre, 1976). The equations that show these relationships are given by Hughes (1974).

$$E_i = \frac{MV^2}{2} \quad \text{(III-3)}$$

$$SE = k/\sqrt{E_i} \quad \text{(III-4)}$$

where

SE=specific energy

E_i=impact energy

M=piston's weight

V=piston's velocity

K= constant

From equations ((III-3) and ((III-4), a direct link between the impact energy and the weight of the piston can be noticed. If the breakage of the rock requires great energy, then high frequency is needed. Therefore, a heavier piston and carrier should be used. A general indication of the hammer's class weight depending on the UCS of the rock is given in Table III-4.

Table III-4 Indication of hammer's weight based on the type of rock

Rock Classification	**Hammer Weight**
Soft rocks/ Deeply stratified rock masses	200 ÷ 500 kg
Compact Rock masses with detachment surfaces	500 ÷ 1000 kg
Hard Rock	500 ÷ 2000 kg

The hammer can be installed on any type of excavator, but it is important to match the size of the carrier with the weight and the power of the hammer, to guarantee the safety and the efficiency of mining operations. Generally, the carrier should be a least 10 ÷ 20 times heavier than the hammer, and it is better to use a crawler machine to guarantee major support. The suggested carrier weight class is normally given by the hammer producers.

IMPACT HAMMERS IN LOCHALINE QUARTZ SAND

The principal use of high impact energy hammers in Lochaline Quartz and Sand is the excavation of the lower cross-section of the two-stages exploitation for the production of LQS500. The production is carried out by the use of two hammers, the Atlas Copco HB2500 (Table III-5- Figure III.7) and the Atlas Copco MB700 (Table III-6 - Figure III.8) Both hammers have a chisel tool installed.

Table III-5 Technical specification hydraulic hammer HB2500

HB 2500	
Service weight	2500 kg
Oil flow	170 l/min ÷ 220 l/min
Operating pressure	160 bar ÷180 bar
Impact rate	280 blows/min ÷ 580 blows/min
Working tool diameter	155 mm
Working length of tool	640 mm
Carrier weight class	27 t ÷ 46 t
Hydraulic input power	66 kW
Sound power level, guaranteed	121 dB(A)

Figure III.7 HB2500

Table III-6 Technical specification hydraulic hammer MB700

MB 700	
Service weight	750 kg
Oil flow	80 l/min ÷ 120 l/min
Operating pressure	140 bar ÷170 bar
Impact rate	370 blows/min ÷ 840 blows/min
Working tool diameter	100 mm
Working length of tool	510 mm
Carrier weight class	10 t ÷ 17 t
Hydraulic input power	34 kW
Sound power level, guaranteed	117 dB(A)

Figure III.8 MB700

The two hammers are mounted on the Hyundai R320 excavator and the CAT 312 excavator. The specifications of the excavators are given in Appendix A.

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IV. EVOLUTION OF GREEN SAND EXPLOITATION

At the beginning, Lochaline Quartz Sand ltd. produced only LQS85, but recently it has also extended the production to green sand. In this chapter the evolution of the techniques used for the exploitation will be described, focusing on the production of LQS500. At the end, a comparison will be done between the productivity of each technique.

The analysis was carried out considering the productivity in one-year time, from the 1st October 2018 to the 30th September 2019.

For drill and blast the average number of round/day has been calculated. The value was obtained by dividing the total number of rounds blasted during the study for the actual days of blast.

1. FULL FACE ADVANCEMENT – LQS85 PRODUCTION

The exploitation of LQS85 consisted in excavating a 5 m height section in the white sandstone deposit, as shown in Chapter I, with a maximum width of 7,5 m. The technique adopted was drill and blast with a room and pillar scheme, as explained in Chapter II. This technique is still in use for the production of the "white" sand.

The cut chosen is the V-cut. This cut consists in the making of 2 to 4 holes forming a V with an opening of at least 60°, a more acute angle requires a higher charge concentration. The number of Vs depends on the width of the face and the depth of the pull. Next to the V holes there are a series of inclined holes called helpers or auxiliary holes. In this configuration the cut holes occupy most of the width of the cross section. A general scheme of the V-cut is given in Figure IV.1.

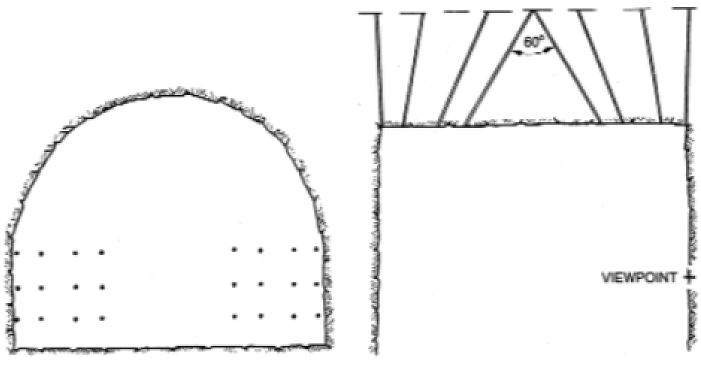

Figure IV.1 V-cut theoretical scheme

The cross-section exploited in the mine is 7,2 m x 4,5 m x 3 m with the blasting scheme shown in Figure IV.2. The cut consists in 3 rows of cut with 2V (Figure IV.2 - blue holes). The angle of the cut is between 35÷39°, while the burden between the two blastholes is of 1,1 m. The actual pull is of 2,85 m. This pull respects the limit given by Mine Regulation 2014 (Regulation 32), that states that the advancement should be lower than 3,5 m.

The middle row does not have any auxiliary blasthole, while for the other two rows there are 14 helpers each (Figure IV.2, azure holes). The contour holes consist of 6 holes in the roof and 7 holes in the floor (Figure IV.2, green holes). With a totality of 49 holes, and a Drilling Density (D_1) of 1,56 holes/m^2.

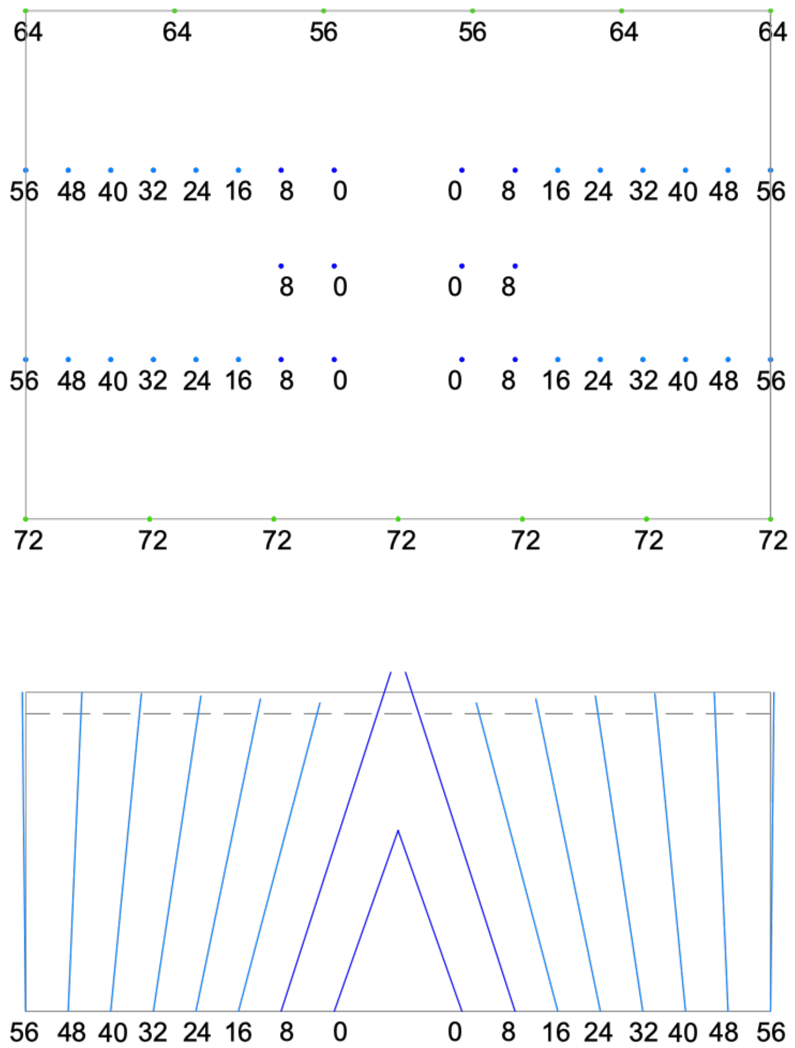

Figure IV.2 Blasting scheme for the production of LQS85

The charging scheme changes based on the functional groups of the hole. The First V, which has a length of 2 m, is charged with three cartridges (Figure I.3 – right). The floor holes are charged with six cartridges (Figure IV.3 – left), while all the other blastholes

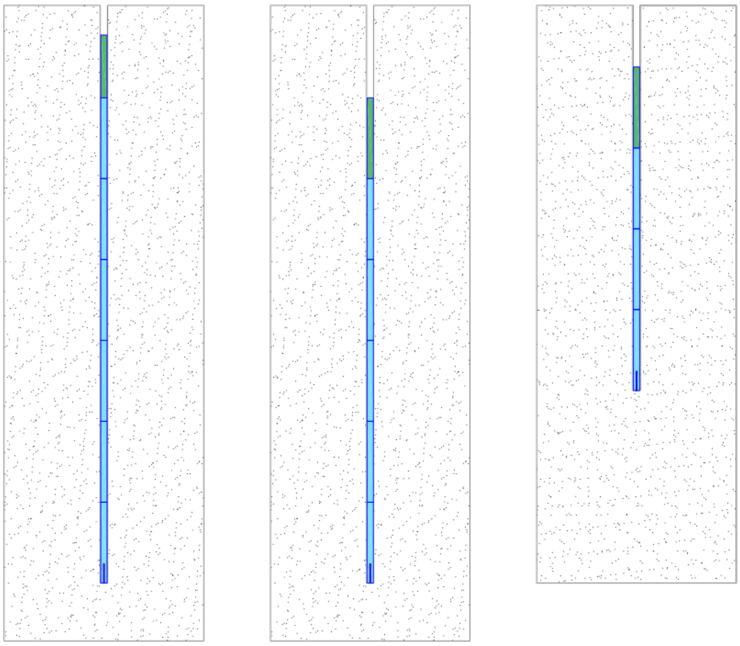

Figure IV.3 Charging scheme

are charged with five cartridges (Figure IV.3 – centre). In all the blastholes the detonator is positioned at the bottom of the hole (Figure IV.3 - in blue), while the stemming at the top of the hole (Figure IV.3- in green).

The blasting scheme used by Lochaline Quartz Sand is consistent with Olofsson's method (1991). This method gives an indication on the position of the blastholes and the amount of explosive suggested for each hole for a V-cut blasting scheme. It is based on the charge concentration [kg/m], and on the type of charge used.

PRODUCTION

The production for the drill and blast is calculated as the volume of rock blasted, taking into account the cross-section and the pull of the blast. For the calculation, 90 % of the theoretical pull has been considered, obtaining a real pull of 2,85 m.

As already mentioned, the cross-section is 7,2 m x 4,5 m, allowing the exploitation of 92,34 m^3 of white sandstone. Knowing that the density of white sandstone is 1,8 t/m^3 the production is 166,21 t/round. Considering the charging scheme shown in Figure IV.3, the total amount of explosive used is 108 kg/round.

With this scheme, the P.F. should be 1,17 kg/m^3, which is higher than the empirical predicted value obtained with the Mancini and Pelizza formula in Chapter III which is 0,91 kg/m^3, but not unreasonable. By analysing the actual production of LQS85 throughout the considered year (Figure IV.4), the average calculated P.F. is equal to 1,21 kg/m^3 (Figure IV.4 – orange line).

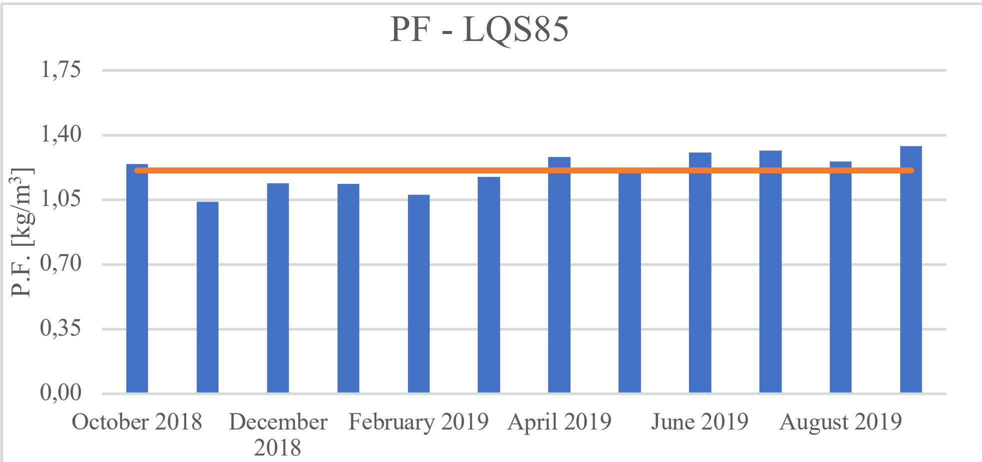

Figure IV.4 Average P.F. distribution - LQS85 production

From the analysis of the production of LQS85 during twelve months, 1/10/2018 ÷ 30/9/2019, the recorded rounds per day is, in average, 3,5. Thus, knowing that each shift lasts 8 h, the average production results as listed in Table IV-1:

Table IV-1 LQS85 hourly and daily production

Production [t/round]	166,21
Rounds x day	3,5
Average Daily Production [t/day]	581,74
Daily shift [h]	8

Hourly Production [t/h]	**72,72**

By multiplying the number of faces blasted in the studied period by the ideal volume excavated, the real production of LQS85 can be easily found. From this study, an average of 68,3 t/h are obtained, that is 94 % of the predicted value.

2. Lower Cross-Section of a Two Stages Blasting Scheme

Two years ago, the demand for sand with higher iron content initiated the production of LQS500, and thus lead to the exploitation of the sandstone located below the already exploited white sandstone. Consequently, a new exploitation technique was required. The first proposal was a two-stages advancement with drill and blast.

The decision taken was to deepen the tunnel up to 2 m by exploiting the deposit in two times. First, white sandstone was to be exploited with the same scheme described above (Full Face Advancement – LQS85 production), followed by the excavation of the lower grade sandstone. The lower cross section consisted on a volume of about 7,2 m x 1,7 m x 3 m. The round was composed of 3 rows with 6 holes/row, with 18 charged holes totally, as shown in Figure IV.6. Each hole is charged as for the floor holes in the aforementioned blasting scheme (Figure IV.3 - left). Thus, the blast consists in 43,2 kg of explosives with 3 delays.

The powder factor (P.F.) is 1,17 kg/m^3. By analysing the actual production between the 4th of September 2017 and the 24th of May 2018 (Figure IV.5), the average P.F. is 1,18 kg/m^3 (Figure IV.5 - orange line). Thus, the actual consumption of explosive is reasonable for the volume of rock that has to be exploited.

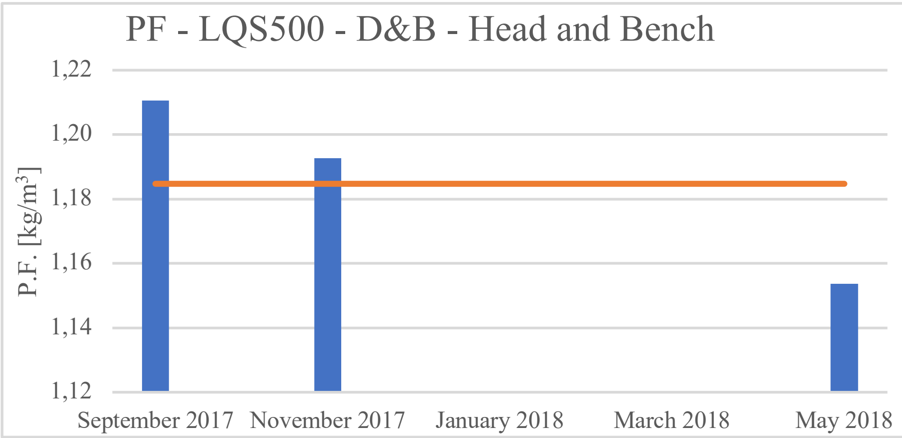

Figure IV.5 Distribution P.F. - LQS 500 - D&B - Head and Bench

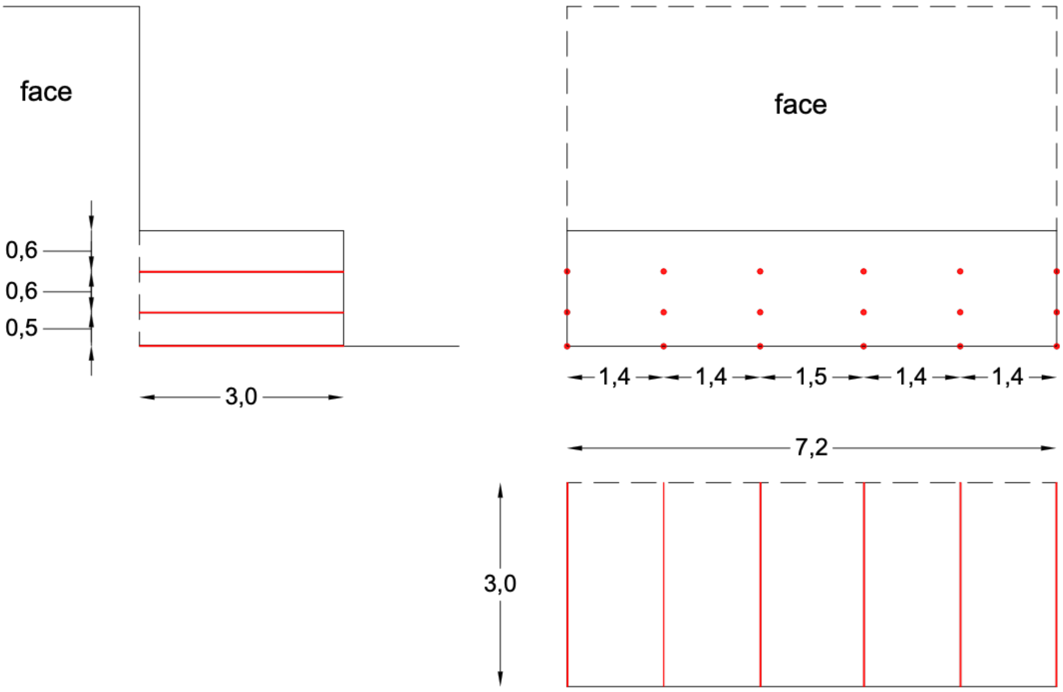

Figure IV.6 Scheme of the bench exploitation with drill and blast

PRODUCTION

With this scheme each round consisted in 36,72 m^3 of rock to be excavated. Considering that the density of "green" sandstone is 2,2 t/m^3, the relative production was of 82 t/round. Calculations were made on the basis of an 8h shift and an average of 3,5 round per day. The estimated daily and hourly production are reported in Table IV-2.

Table IV-2 Production analysis for LQS500 drill and blast benching

Production [t]	82
Rounds x day	3,5
Average Daily Production [t/day]	287
Daily shift [h]	8
Hourly Production [t/h]	**35,88**

In the period under study, by knowing the amount produced and the actual time worked, the production is 23,01 t/h. Therefore, with this technique the productivity reaches the 64,15% of that which had been predicted.

All the data related to the technique described have been taken from a previous study made in 2018 that compared the benching exploitation by drill and blast and hydraulic hammer.

3. Two-stages Advancement with Drill and Blast and Impact Hammer

The company invested in a hammer to exploit the sandstone used for the production of LQS500 in the working area and also to excavate the lower cross-section in older rooms where there is no white sandstone left. The hammer exploits section of 7,2 m x 2 m, deepening the already excavated tunnels.

The study made in 2018 showed that the exploitation with the hammer was economically advantageous in respect to the drill and blast technique. Therefore, a new hydraulic hammer was bought. The two types of hammers used in Lochaline Quartz Sand are described in Chapter 2.

Theoretical Production

There are no standards for the evaluation and prediction of the productivity of hydraulic hammers, but many studies attempt to analyse the performance of the machine. In this study, a graphical method (Indeco) as well as an empirical method (Tuncdemir, 2008 and Bilgin et all, 1996, 1997, 2002) were performed.

The graphical method described by Indeco correlates the class of the hammer with the uniaxial compressive strength of the rock. Depending on the final use of the machine (demolition, trench, ecc.) an indication of the hourly production is given.

For the evaluation of the uniaxial compressive strength, an average value from geotechnical studies achieved from 2002 up to February 2019 have been considered, obtaining $\sigma_c = 100$ MPa (Figure IV.7). Since these graphs are based on Indeco's hammers, the most similar to those in use in Lochaline are the HP4000 and the HP1200.

From this analysis, the HM2500 should be able to provide a production of 190 m^3/shift, while the MB700 of 15 m^3/shift. These values have to be taken only as indicative.

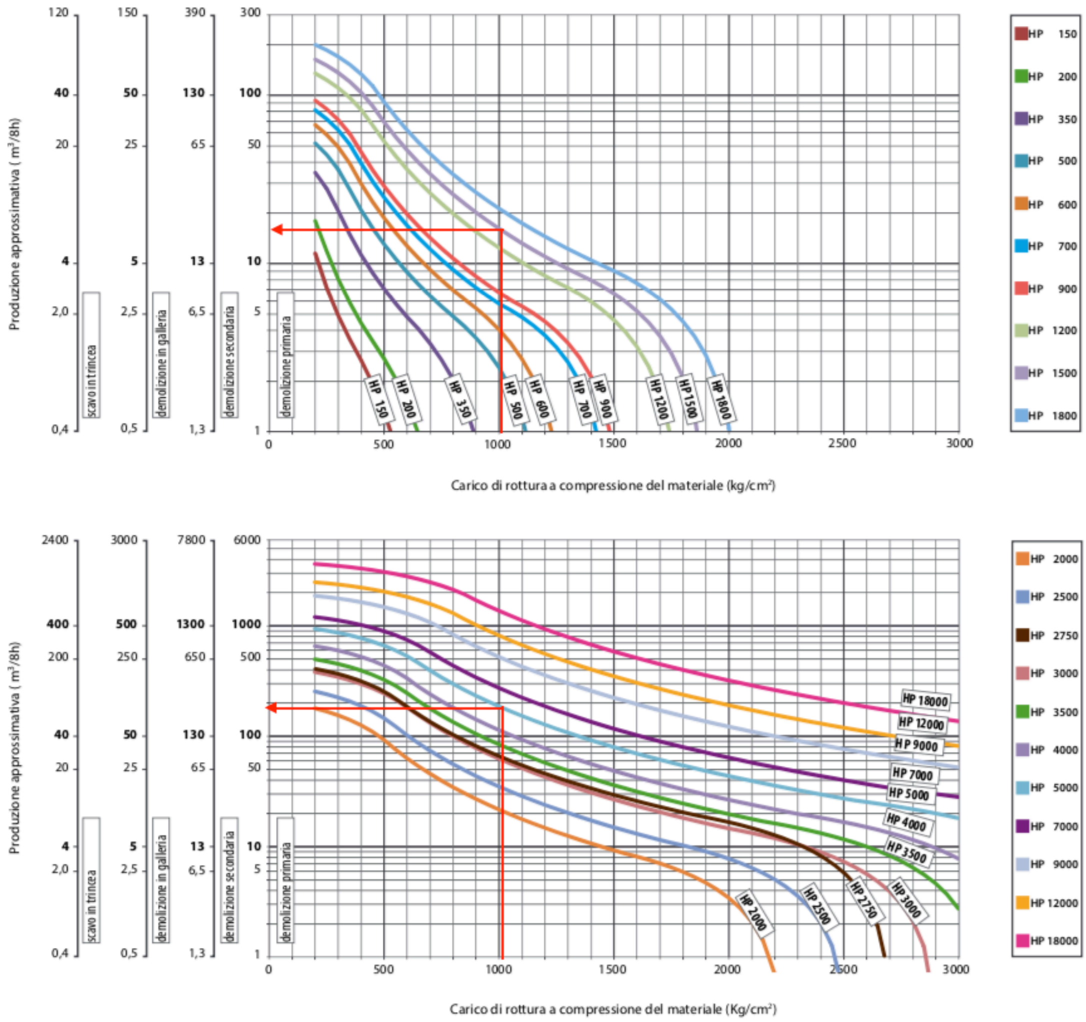

Figure IV.7 Indeco's indicative values for the production with hydraulic hammer

The Tuncdemir method (2008) is related to the output and input power, obtained with the following equations:

$$P_{input} = Q \cdot p \tag{IV-1}$$

$$P_{output} = n \cdot E_i \tag{IV-2}$$

$$\eta = P_{output}/P_{input} \tag{IV-3}$$

where:

P_{input} = input power [kW]

Q = oil flow [m^3/s]

p = oil pressure [kN/m^2]

P_{output} = output power [kW]

E_i = single blow energy [kN·m]

n = blow frequency [blow/s]

η = hammer efficiency

To evaluate the parameters above, the following empirical equations (Tuncdemir, 2007), obtained from the analysis of 600 impact hammers available in the industry, are used:

$$E_i = 2{,}4718 \cdot W_{hammer} - 27{,}774 \quad \text{(IV-4)}$$

$$EW_{max} = 0{,}015 \cdot W_{hammer} + 3{,}2343 \quad \text{(IV-5)}$$

$$EW_{min} = 0{,}0094 \cdot W_{hammer} + 1{,}3485 \quad \text{(IV-6)}$$

$$P_{output} = 0{,}0187 \cdot W_{hammer} + 7{,}1016 \quad \text{(IV-7)}$$

$$P_{input} = 0{,}0187 \cdot W_{hammer} + 11{,}837 \quad \text{(IV-8)}$$

where

E_i =single blow energy [J]

W_{hammer} = operational weight of impact hammer [kg]

EW_{max} = maximum recommended weight of excavator [t]

EW_{min} = minimum recommended weight of excavator [t]

The maximum reachable hourly production [m^3/h] is evaluated by Bilgin et al. (1996, 1997, 2002) as the Instantaneous Breaking Rate (IBR). The IBR is given by the empirical formula:

$$IBR = 4{,}24 \cdot P_{output}(RMCI)^{-0{,}567} \quad \text{(IV-9)}$$

$$RMCI = \sigma_c \cdot \left(\frac{RQD}{100}\right)^{\frac{2}{3}} \quad \text{(IV-10)}$$

where

RMCI = Rock Mass Cuttability Index

σ_c = Uniaxial Compressive Strength [MPa]

RQD = Rock quality designation [%]

The prediction of the performance of the two hammers has been evaluated with this latter method. All the parameters regarding the hammer characteristics (oil flow, operating pressure, impact rate) are average values taken from the brochures listed in Table III-5 and Table III-6. The single blow energy has been calculated from the empirical law (I4). The uniaxial compressive strength is taken, as per Indeco's graphical analysis, of 100 MPa. The RQD has been evaluated as the average value of logs analysed in 2002. Thus, using the (IV-1), (IV-2), (IV-9) and (IV-10)the following results listed in Table IV-3 have been obtained.

Table IV-3 HB2500 Theoretical Production on the left; MB700 Theoretical Production on the right

HB2500		**MB700**	
Oil flow [l/min]	195,00	**Oil flow [l/min]**	100,00
Operating Pressure [bar]	170,00	**Operating Pressure [bar]**	155,00
Impact Rate [blows/min]	430,00	**Impact Rate [blows/min]**	605,00
Hammer weight [kg]	2500,00	**Hammer weight [kg]**	750,00
E [kN*m]	6,18	**E [kN*m]**	1,85
P_{input} [kW]	55,25	**P_{input} [kW]**	25,83
P_{output} [kW]	44,29	**P_{output} [kW]**	18,69
Efficiency [%]	80	**Efficiency [%]**	72
UCS [MPa]	100,00	**UCS [MPa]**	100,00
RQD [%]	91,93	**RQD [%]**	91,93
RMCI [MPa]	94,54	**RMCI [MPa]**	94,54

IBR [m^3/h]	18,79	**IBR [m^3/h]**	7,93
Theoretical production [t/h]	41,30	**Theoretical Production [t/h]**	17,45

Theoretically, the HB2500 and the MB700 hammers are able to provide respectively an efficiency of 80% and 72%, and an hourly production of 41,30 t/h and 17,45 t/h. Using the model of a 8h shift, the daily production of the two hammers, taken separately, and together, will be (Table IV-4):

Table IV-4 Theoretical Daily Production

	HB2500	**MB700**	**BOTH**
Theoretical production [t/h]	41,30	17,45	58,80
Shift [h]		8	
Theoretical production [t/d]	330,78	139,62	470,41

PRODUCTION

In the mine, the two hammers work at the same time. Since the actual tons produced were not registered because most of the muckpile was left in place after the excavation and not weighed by the shovels, an average hourly production has been estimated. The evaluation has been done for both hammers, by measuring the excavated volume and the time needed to do it. For the assessment of the volume, at the beginning of the shift the initial position of the hammer (Figure IV.8) was marked on the wall, and then at the end of the working hours the geometrical excavated volume was measured.

Figure IV.8 Marks on the pillar for the measurement of the hammer's excavated volume

By multiplying the values obtained by the specific weight of the green sand, 2,2 t/m^3, and dividing the result for the hours worked, the average production in t/h is obtained. The final results, listed in Table IV-5, are an average of the estimated measurements.

Table IV-5 I5 Hammers Hourly Production

	HB2500	**MB700**	**BOTH**

Volume Excavated [m³]	76	46,25	122,25
Hour worked [h]	16,33	8,75	25,08
Hourly production [t/h]	12,48	13,19	25,67

By comparing the theoretical and the calculated production (Figure IV.9), it is noticed that the HB2500 has an efficiency of less than 30 %, while the MB700 of more than 70 %. The difference between the productivities may be caused by the fact that during the study an unfavourable geology was found, forcing the hammers to reduce the production to avoid damage to the tool.

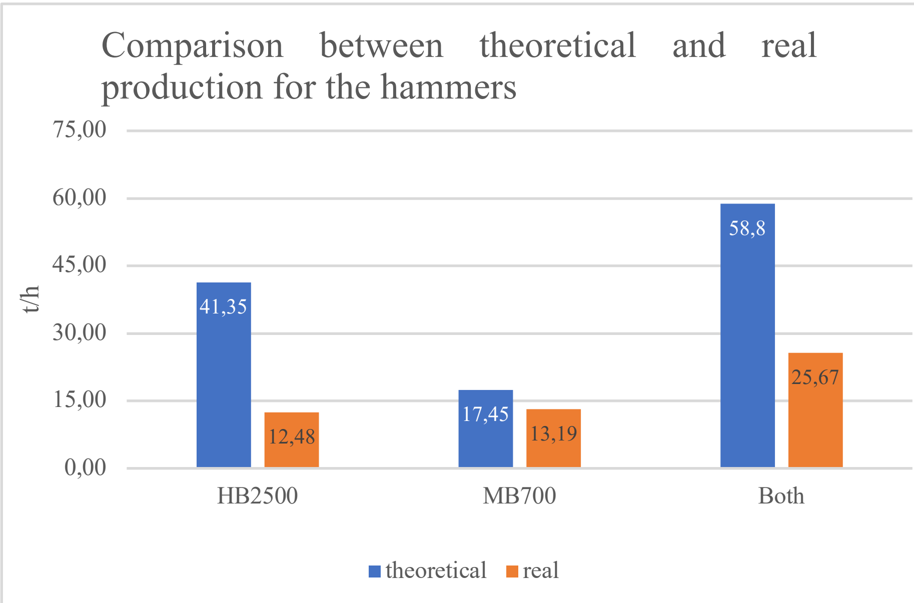

Figure IV.9 Comparison between theoretical and real production for the hammers

Therefore, from the analysis resulted that the two hammers produced less than 50 % of the expected production rate.

4. FULL FACE ADVANCEMENT – LQS500 PRODUCTION

In the last few months, it was decided to avoid using the lower cross-section of the two-stages blasting for the production of LQS500 and move to a full-face advancement with drill and blast by enlarging the round used for the exploitation of white sandstone. In this way, the excavated material is a mix of pure white sandstone (iron content ≤ 85 ppm) and lower graded sandstone (iron content up to 800 ppm), with an average content of iron ≤ 500 ppm, ideal for the production of the desired sand.

The new round results in higher production than the previous one, with a cross section of 7,2 m x 6 m. The blasting scheme differs from that used for the production of LQS85 with the addition of one row of stoping holes, as shown in Figure IV.10. The holes are charged in the same way as for the full-face round for the LQS85 (Figure IV.3).

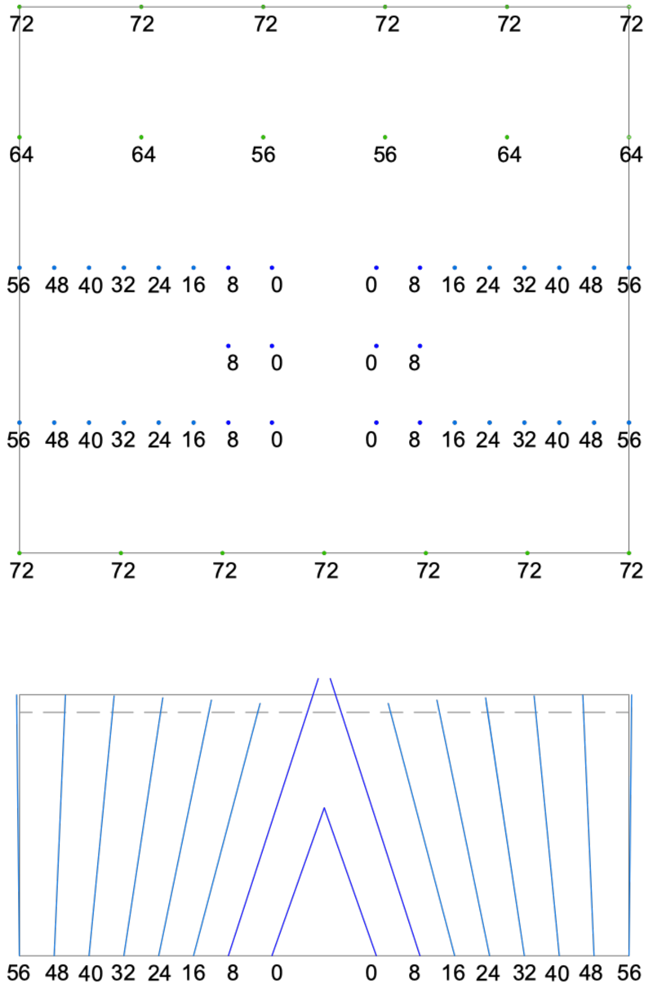

Figure IV.10 Blasting scheme for the production of LQS500

The blasting scheme consists of 55 holes charged with 298 cartridges, thus the total amount of explosive used is 122,4 kg/round.

PRODUCTION

In the studied year the mine was excavated in two different zones for the production of LQS500. For the first six months, a reduced cross-section was used. The blasting scheme was the same as that for the production of LQS85. Thus, in that period each round consisted of 92,34 m^3 of material. Considering that the material exploited is composed of approximately 70% white sandstone (γ= 1,8 kg/m^3), and for the remaining 30% of low grade sandstone (γ = 2,23 t/m^3) the production of each round resulted in 178,12 t/round.

The P.F. for this blasting scheme is 1,17 kg/m^3, the same as LQS85. By analysing the consumption of explosive and the theoretical production per each day in the period under study (Figure IV.11), the resulting P.F. is on average 1,09 kg/m^3 (Figure IV.11 – Orange line).

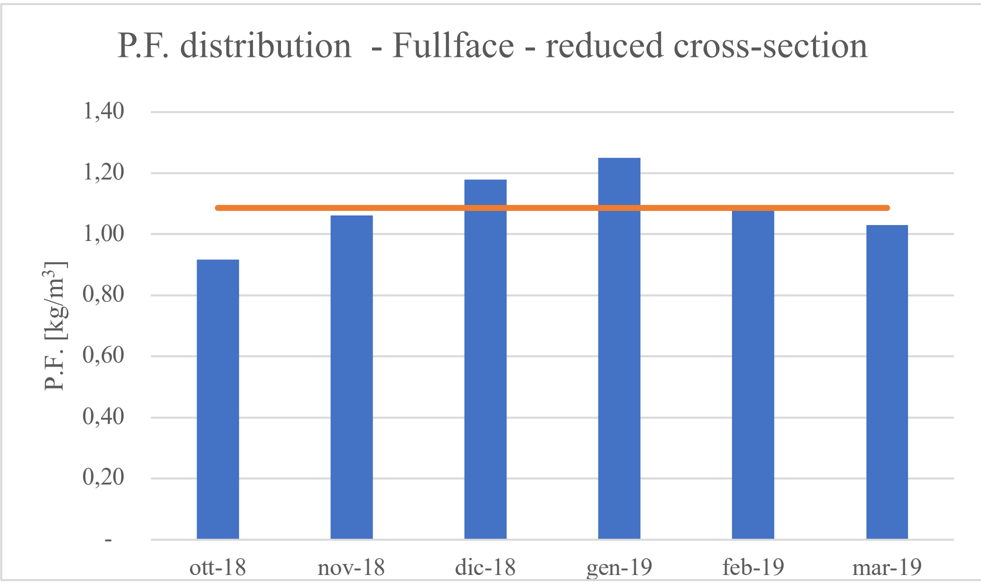

Figure IV.11 Distribution of the P.F.- D&B - full face - reduced cross-section

According to the blasting diary, the average number of rounds/d in the given period was 1,5. Thus, knowing that each shift lasts 8 h, the estimated hourly and daily production is shown in Table IV-6.

Table IV-6 Production analysis for LQS500 production - full face advancement - reduced cross section

Production [t/round]	178,12

Rounds X Day	1,5
Av. Daily Production [t/day]	267,19
Daily Shift [h]	8
Hourly Production [t/h]	**33,40**

The recorded production for the period between the 1st October 2018 and the 21st March 2019, was analysed considering the tons exploited based on the volume blasted and the actual days worked. The result was 38,27 t/h, 15 % more than the expected value.

From the 21st March 2019, the exploitation moved to another area that allowed the excavation with the cross section previously explained. For this figure, each blast produces 123,12 m^3, and therefore 234,85 t/round.

This blasting scheme should be characterized by a P.F. of 0,99 kg/m^3, that is comparable with the predicted value (0,91 kg/m^3) obtained with the Mancini and Pelizza formula (Chapter III). The actual specific consumption of explosive from the year analysed is shown in Figure IV.12, with an average P.F. of 0,8 kg/m^3 (Figure IV.12 – orange line).

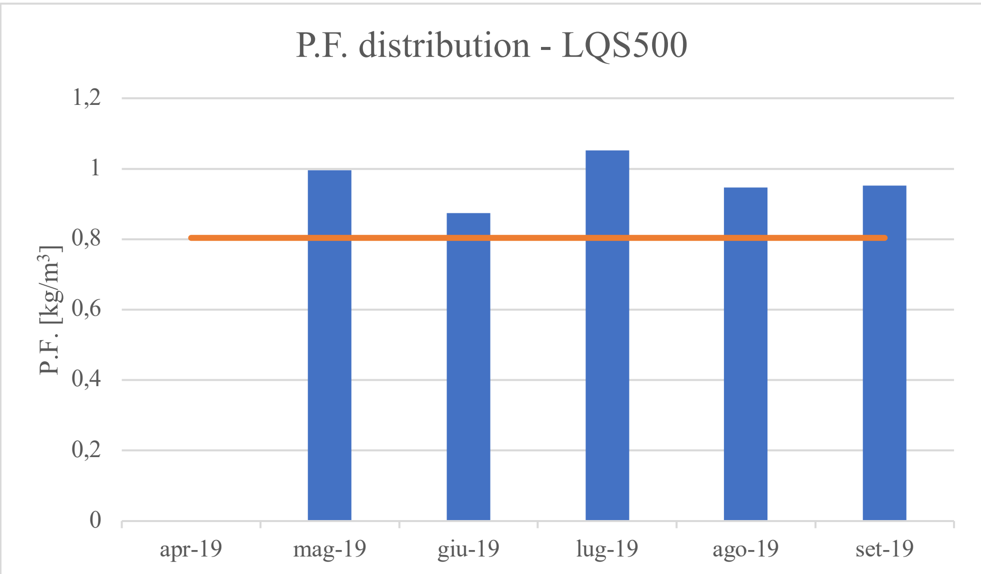

Figure IV.12 Distribution of the P.F.- D&B - full face - Usual cross-section

The study showed an average of 2,5 rounds/day. The expected daily and hourly production are listed in Table IV-7.

Table IV-7 Production analysis for LQS500 full face advancement

Production [t/round]	234,85
Rounds X Day	2,5
Av. Daily Production [t/day]	587,13
Daily Shift [h]	8
Hourly Production [t/h]	73,39

According to the analysed blasting diary, the actual production was of 70,79 t/h, equal to 96 % of that expected.

Benching Cross-Section During The Studied Period

From the blasting diaries during the studied year it resulted that some benches were excavated. These rounds had no production value but were excavated to enlarge the cross-section in order to move from the production of LQS85 to LQS500.

The blasting scheme used changes from the one described in Lower Cross-Section of a Two Stages Blasting Scheme. The cross section has a variable height depending on the needs and a width of 7,2 m. The general scheme is shown in Figure IV.13.

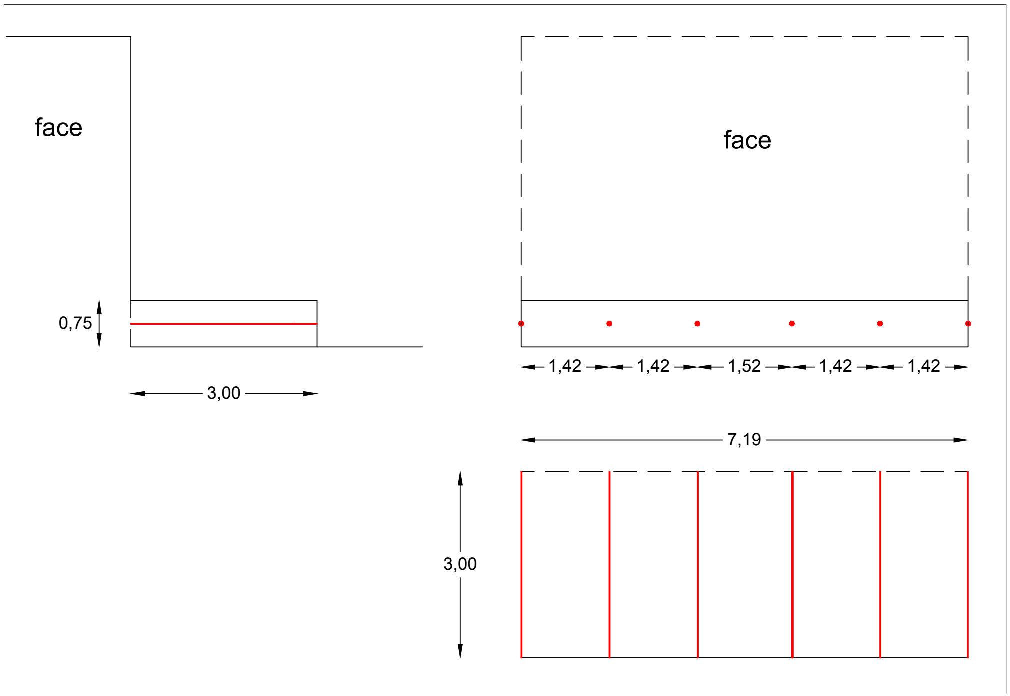

Figure IV.13 New blasting scheme for bench blasting

COMPARISON BETWEEN THE THEORETICAL PRODUCTIONS FOR LQS500

The study gives an indication of the expected production for each technique used in the mine for the exploitation of LQS500. The results are given in Figure IV.14.

The most productive technique is drill and blast, with an average production of 71,3 t/h when using a full-face blasting scheme, followed by the two-steps cross-section blasting

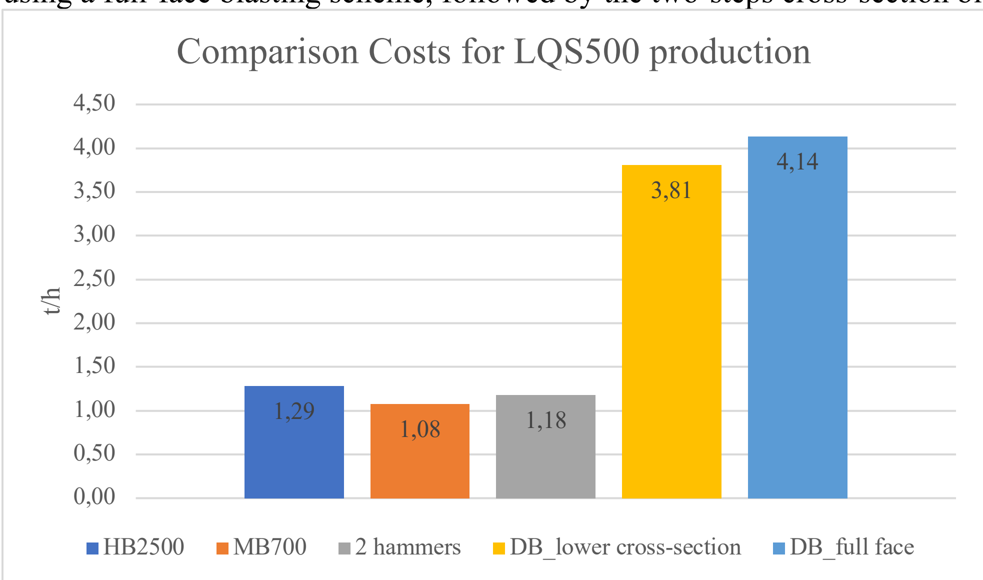

Figure IV.14 Comparison between the average production of the different techniques scheme (35,88 t/h). This result was predictable as drill and blast is known to be a very productive mining technique.

The hammers, instead, are less productive with 25,67 t/h, obtained by summing up the single productions, 13,19 t/h for the MB700 and 12,48 t/h for HB2500. The HB2500 has a production of less than 30% of that predicted by Tuncdemir and Bilgin et al. Therefore, the inefficiency of the hammer reduces the productivity of this technique by 43,56 %. If the HB2500 worked with a 70 % efficiency as does the MB700, the use of two hammers together would produce more than the two-stages blasting.

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5. COST ANALYSIS FOR LQS500 PRODUCTION

In the present chapter, a general overview of the costs is presented, in order to evaluate the best solution on an economic point of view. For a good comparison, each technology has been analysed obtaining the average cost/t. The costs taken into consideration are those that have occurred in the course of the studied year, between the 1st October 2018 and the 30th September 2019.

The costs for loading and transport of the muckpile outside the mine have not been considered, since they are not dependent on the type of sand produced or on the technique used for the exploitation.

DRILL AND BLAST COST ANALYSIS

For a general overview of the costs, deriving from drill and blast operation, the costs that have been taken into account are those suggested by Lusk and Worsey (2011):

Drilling Costs	**Explosive Costs**
Fuel	Detonators
Operator	Explosive
Lubrification	Stemming
Maintenance	Initiation system
Drill tool	Operator

Since the study for the lower cross-section of the two-stages blasting is a year old, and many incomes have changed, the descriptions of the costs have been analysed separately.

COSTS FOR LOWER CROSS-SECTION OF THE TWO-STAGES BLASTING

The drilling cost, the fuel, lubrication and maintenance have been considered as a single cost of £ 3 per round. The drilling tool expense is considered taking into account the changing of the tools every 8 rounds (consumption = 25 %). The operational costs are the hourly salary of the driller and two shot firers.

For ease of reading, the costs have been divided into blast (Table 5-1), drilling (Table 5-2) and operational costs (Table 5-3).

Table 5-1 Cost Analysis for the Blast – D&B – Lower cross-section

EXPLOSIVE	
Consumption [kg]	43
Cost [£/kg]	3,6
Total Cost [£/round]	154,8

DETONATORS	
Number	18
Unit Cost [£]	1,45
Total Cost [£/round]	26,1

STEMMING	
Number	14
Unit Cost [£]	0,35
Total Cost [£/round]	4,9

ELECTRIC CABLE	
Used meters [m]	100
Cost [£/m]	0,1077
Total Cost [£/round]	10,77

Total Cost [£/round]	196,57

Table 5-2 Cost Analysis for the Drill – D&B – Lower cross-section

CONSUMPTIONS	
Total Cost for Electricity, Gas and Oil [£]	3

TOOLS	
Consumptions [%]	25
Unit Cost [£]	50
Total Cost [£/round]	12,5
Total Cost [£/round]	15,5

Table 5-3 Cost Analysis for the Operators – D&B – Lower cross-section

OPERATIONAL COST

Shift [h]	8
Hourly Cost [£/h]	14,6
Daily Cost [£]	116,8
Operator Cost Per Round [£/round]	33,37
Operators	3
Total Cost [£/round]	**100,11**

To evaluate the average cost/t, each item has been multiplied by the number of rounds/day, 3,5. The results and their distribution are listed in Table 5-4 and represented in Figure 5.1. The final result is an average of 3,81 £/t.

Table 5-4 Percentage distribution of the costs – D&B – Lower cross-section

	COSTS X ROUND	PERCENTAGE
EXPLOSIVE	154,8	49,59 %
DETONATORS	26,1	8,36 %
STEMMING	4,9	1,57 %
CABLE	196,57	3,45 %
DRILL	15,5	4,97 %
OPERATARS	100,11	32,07 %
TOT	312,18	100

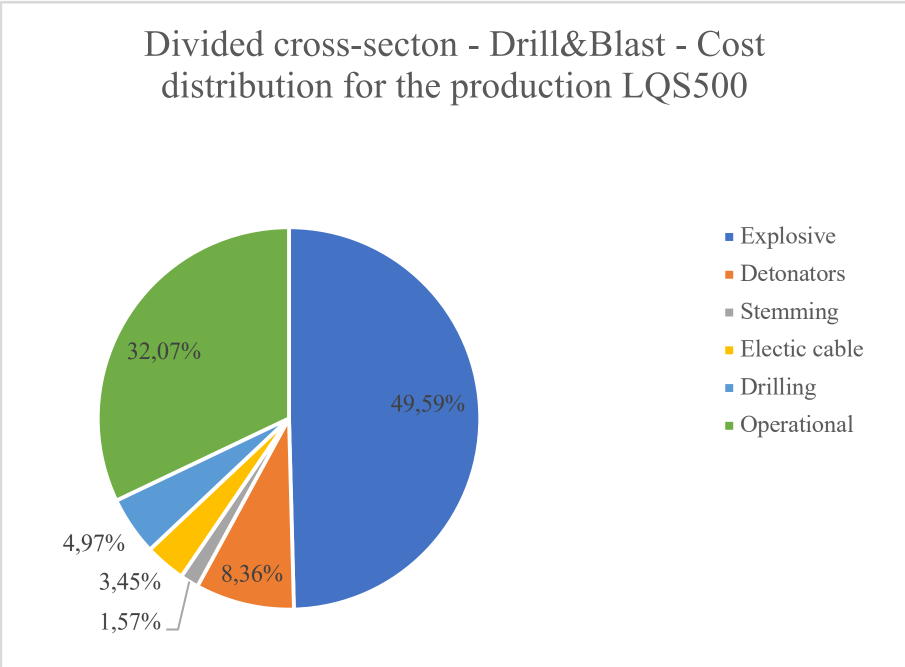

Figure 5.1 Percentage distribution of the costs – D&B – Lower cross-section

COSTS FOR FULL FACE

The analysis for the production costs of LQS500 by drill and blast with a full-face blasting scheme uses data related to the period 1/10/2018 ÷ 30/9/2019. It is worth to highlight that, compared to the two-steps blasting scheme, many costs have changed and most of them have increased.

The drilling costs were analysed including all the actual consumables and expenses faced throughout the studied year. Thus, the following considered are:

- Consumable parts, such as rods, tools, and so on.
- Grease
- Fuel, nearly negligible cost because the machine does not move from underground and stays in the proximity of the working area
- Hydraulic Oil
- Operator

In this study, the maintenance of the jumbo has not been considered because the machine is covered by an insurance supplied by Epiroc, formerly Atlas Copco. This insurance is a monthly cost plus a possible surplus depending on the hours worked by each boom of the jumbo. Since the mine has only one jumbo that is used for both the LQS85 and LQS500 exploitations, the maintenance cost is independent of the technique used.

The costs have been divided into fuel, grease and hydraulic oil consumption (Table 5-5), service costs (Table 5-6) and operational costs (Table 5-7).

Table 5-5 Cost Analysis for the fluids – Drilling - Full face

	FUEL	GREASE	HYDRAULIC OIL
Consumptions [l]	224	24	1.516,9
Unit cost [£/l]	0,58	2,364	1,02
Total Cost [£]	129,92	56,74	1.547,23

Table 5-6 Costs for the services Analysis – Drilling - Full face

	CONSUMABLE
Total cost [£]	70.462,29

Table 5-7 Cost Analysis for the Operator – Drilling - Full face

OPERATOR	
Hourly cost [£/h]	22,5
Days worked	234
Shift [h]	8
Daily Cost [£/day]	180
Total Cost [£]	42.120,00

To include this study in the cost analysis of the full-face blasting scheme, the total cost has been divided by the worked days. The number has been found considering 5 days/week for a year, and 10% of stops due to holidays or breakage of the machine. The percentage has been evaluated on the basis of the actual days worked during the analysed year.

According to this study the daily cost for the drilling was 506,68 £/day. The percentage distribution is shown in Figure I.2 - Table I8.

Table 5-8 Percentage distribution - drilling cost

Fuel	0,11%
Grease	0,05%
Oil	1,35%
Consumables	61,64%
Operational	36,85%

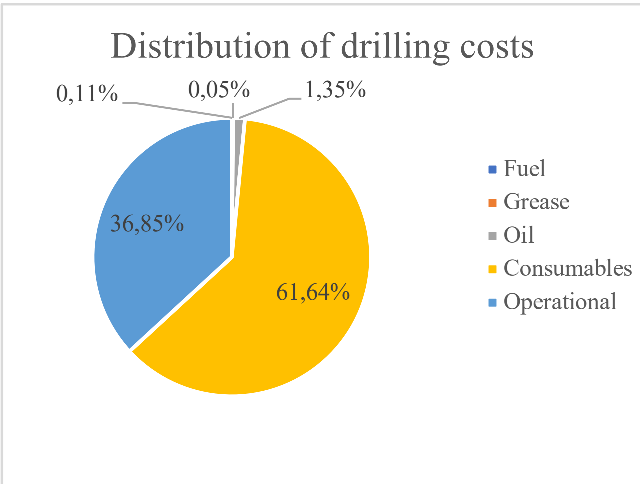

Figure 5.2 Cost Distribution - Drill and Blast - Drilling costs

For the remaining part of the analysis, a distinction has to be made between two periods, due to the change of the cross-section moving from one area to another.

Ideal production – reduced section

As already described in Chapter IV, during the first six months (1st October 2018 - 21st March 2019), the exploitation of LQS500 was carried out in an area that allowed the blasting of a reduced cross-section, similar to that used for the production of LQS85. The resulting production was of 43,55 t/h, with an average of 1,5 rounds/day.

The blasting scheme was made by 49 holes, charged with 108 kg of explosive. For the stemming, generally 7 clay plysters per round were used. To close the blasting circuit an electric cable was used. In average 300 m per round are used, therefore 3 reels of 100 m each per round have been considered.

The cost analysis based on the expected production is listed in Table 5-9, Table 5-10 and Table 5-11.

Table 5-9 Cost analysis for the blast - Full Face - reduced cross section

EXPLOSIVE	
Consumption [kg]	108
Cost [£/kg]	3,7
Total Cost [£]	399,6

DETONATORS	
Number	49
Unit cost [£]	2,51
Total Cost [£]	122,99

STEMMING	
Number	7
Unit cost [£/dummy]	0,44
Total cost [£]	3,08

ELECTRIC CABLE	
Packages x day	3
Unit cost [£]	5,8
Rounds x day	1,5
Cost [£/Round]	11,60

The drilling costs include the driller's salary, increased by 50% because he works during the night shift. The operational cost only refers to the cost of the shot-fires. Normally, three workers are responsible for charging, but since the introduction of the double shift, the hidden times have been reduced. Thus, after the blast, they are employed on other tasks. For this reason, the numbers of the workers taken into account is 2,5 instead of 3. The hourly cost is equal to the average of the three salaries, all increased by 25 %, to cover the cost of the night shift (two weeks per month).

Table 5-10 Cost analysis for the drilling - Full Face - reduced cross section

DRILL	
Daily cost [£/day]	488,53
Round x day	2
Cost X Round [£/round]	325,69

Table 5-11 Cost analysis for the Operators - Full Face - reduced cross section

OPERATIONAL COST	

Shift [h]	8
Hourly cost [£/h]	21,91
Rounds x day [rounds]	2
Operator Cost Per Round [£/round]	175,25
Operators [-]	2,5
Total Cost [£]	292,08

Therefore, the ideal cost per tons results in 6,48 £/t, with the percentage distribution described in Figure I.3 and Table I12.

Table 5-12 Expected percentage distribution of the costs – D&B - Full face - reduced cross-section

	COSTS X ROUND	**PERCENTAGE**
EXPLOSIVE	399,6	34,6%
DETONATORS	122,99	10,65 %
STEMMING	3,08	0,27 %
CABLE	11,06	1 %
DRILL	325,69	28,20 %
OPERATARS	292,08	25,29 %
TOT	1155,04	100

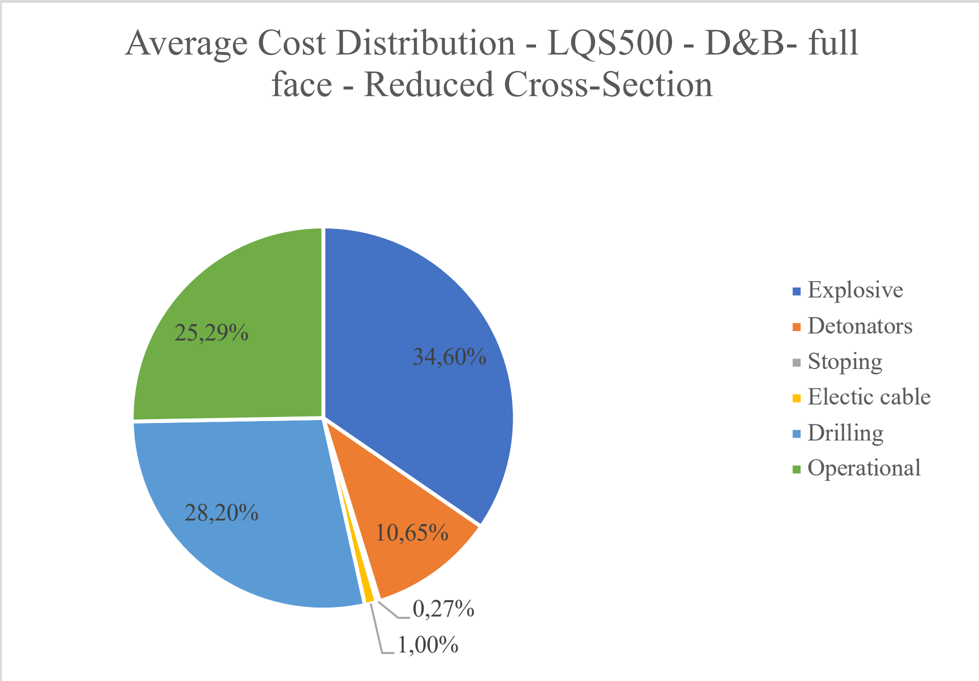

Figure 5.3 Percentage Distribution of the Costs - D&B - full face analysis, reduced cross-section

Real production – reduced section

The real cost of the reduced blasting scheme was made by analysing the actual consumption of explosives and detonators and multiplying the number of faces blasted by the theoretical production (161,5 t/round). The cost amounts to 5,83 £/t.

The percentage distribution of each item is given in Table 5-13 and Figure 5.4.

Table 5-13 Real percentage distribution of the costs – D&B - Full face - reduced cross-section

	COSTS X ROUND	PERCENTAGE
EXPLOSIVE	421,79	35,47 %
DETONATORS	134,99	11,35 %
STEMMING	3,08	0,26 %
CABLE	11,60	0,98 %
DRILL	325,69	27,39 %

OPERATARS	292,08	24,56 %
TOT	1189,23	100

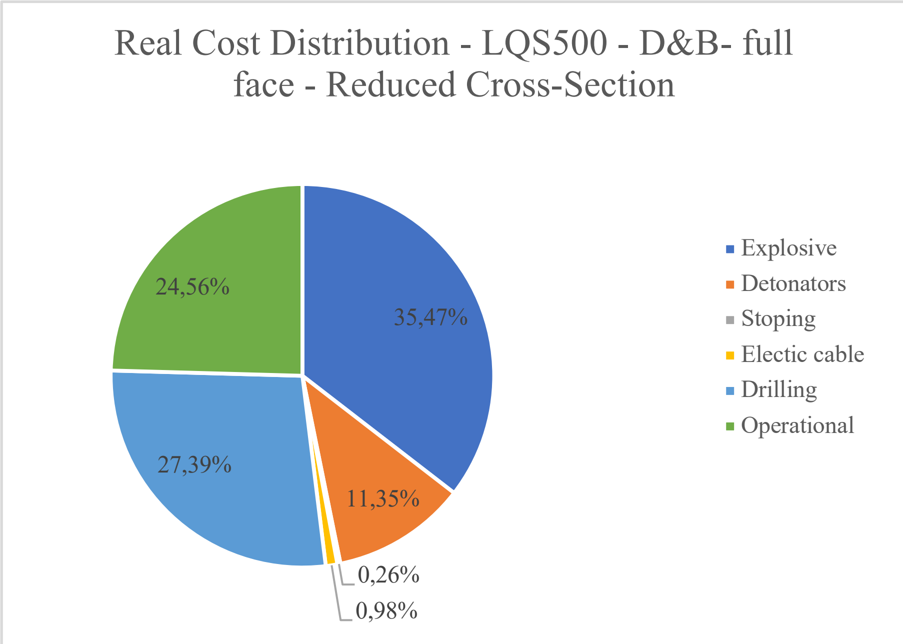

Figure 5.4 Percentage Distribution of the Costs - D&B - full face analysis, reduced cross-section

Ideal production – Actual section

For the last six months of the analysis, the exploitation moved to an area where it was possible to use the round explained in Chapter IV. This blasting scheme produces 285,12 t, using 122,4 kg of explosive in 55 holes.

Knowing the unit cost per each category, the ideal cost/t can be obtained. The results of the analysis are listed in Table 5-14, Table 5-15 and Table 5-16.

Table 5-14 Cost Analysis for the Blast -D&B - Full face - ideal production

EXPLOSIVE	
Consumption [kg]	122,4
Cost [£/kg]	3,7

Total Cost [£/round]	452,88

DETONATORS	
Number	55
Unit Cost [£]	2,51
Total Cost [£/round]	138,05

STEMMING	
Number	7
Unit Cost [£/Dummy]	0,44
Total Cost [£/round]	3,08

ELECTRIC CABLE	
Packages X Day	3
Unit Cost [£]	5,8
Round x day	2,5
Total Cost [£/round]	6,96

Table 5-15 Cost Analysis for the Drilling -D&B - Full face - ideal production

DRILL	
Daily Cost [£/day]	488,53
Round x day	2,5
Total Cost [£/round]	195,41

Table 5-16 Cost Analysis for the Operational cost – D&B - Full face - ideal production

OPERATIONAL COST	
Shift [h]	8
Hourly cost [£/h]	21,91
Rounds x day [d]	2,5
Total cost x operator [£/operator]	175,25
Operators	2,5
Total Cost [£/round]	175,25

The average costs/t obtained as final result is 4,14 £/t, with the distribution given in Table 5-18 and Figure 5.5.

Table 5-17 Percentage distribution of the costs – D&B – Full face

	COSTS X ROUND	**PERCENTAGE**
EXPLOSIVE	452,88	46,61 %
DETONATORS	138, 05	14,21 %
STEMMING	3,08	0,32 %
CABLE	6,96	0,72 %
DRILL	195,41	20,11 %
OPERATARS	175,25	18,04 %
TOT	971,63	100 %

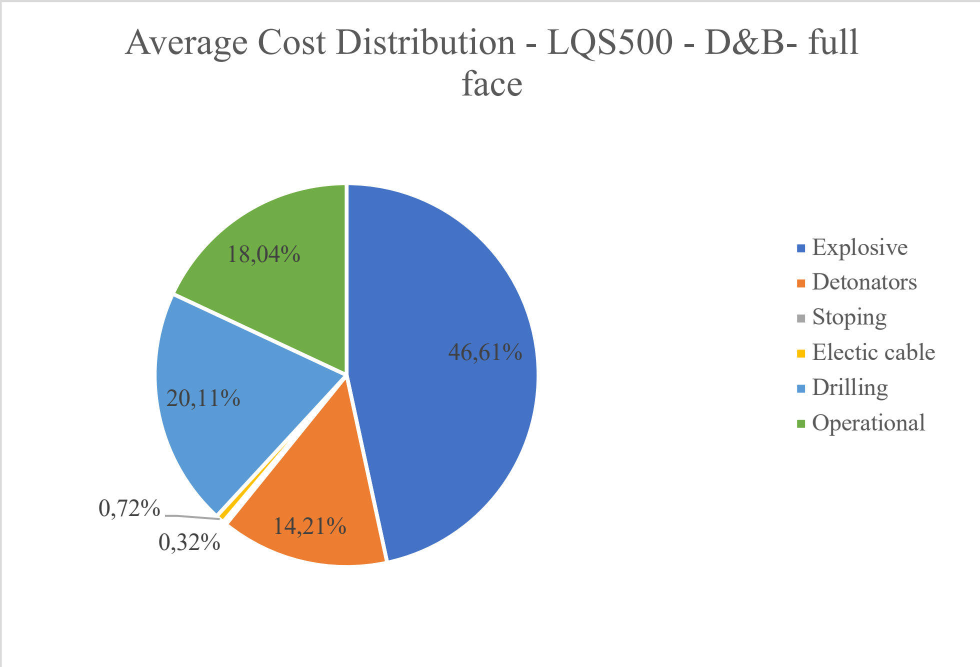

Figure 5.5 Ideal Percentage Distribution of the cost - D&B - full face

Real production – actual section

As for the Real production – reduced section the explosive costs have been analysed by considering the total consumption of explosive and of detonators. The real cost per tons estimated is 4,13 £/t, with the distribution given in Table 5-18 and Figure 5.6.

Table 5-18 Percentage distribution of the blast - Full face – real production

	COSTS X ROUND	PERCENTAGE
EXPLOSIVE	438,69	45,84 %
DETONATORS	128,64	13,44 %
STEMMING	3,08	0,32 %
CABLE	7,12	0,74 %
DRILL	200,03	20,90 %
OPERATARS	179,39	18,75 %
TOT	956,95	100 %

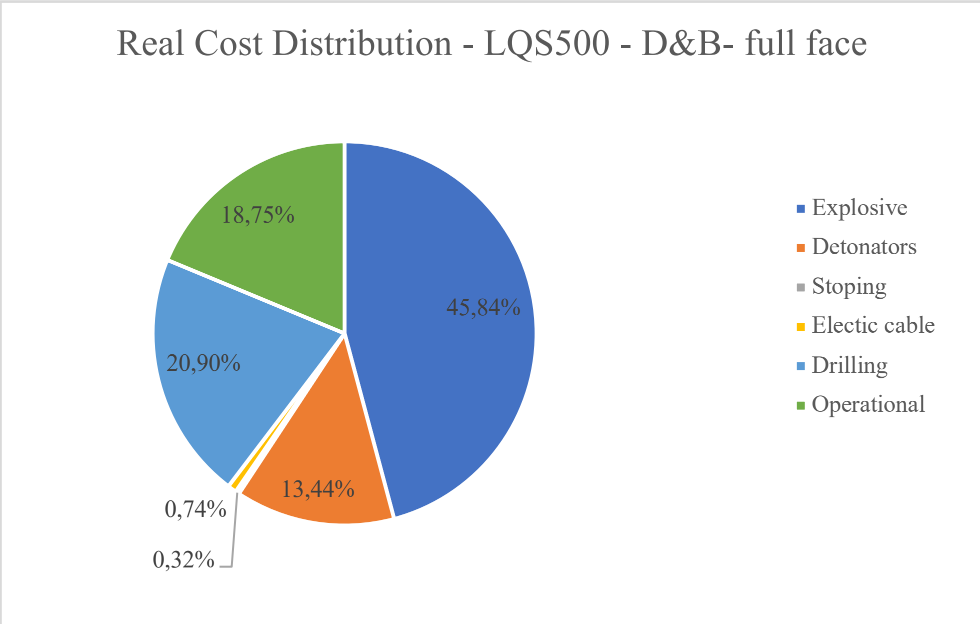

Figure 5.6 Percentage distribution of the blast - Full face – real production

For a comparison of the technique's cost, only this latter cost has been considered, since the exploitation of LQS500 with the reduced cross-section is considered as an irregularity in the production.

COMPARISON BETWEEN THE BLASTING SCHEMES

As already said, the P.F. is a good parameter for the evaluation of the unit cost of the production. By comparing (Figure 5.7) the average values obtained by the studies, it can be stated, as expected, that the full-face blasting scheme requires a lower consumption of explosive than the lower cross-section of the two-steps blasting.

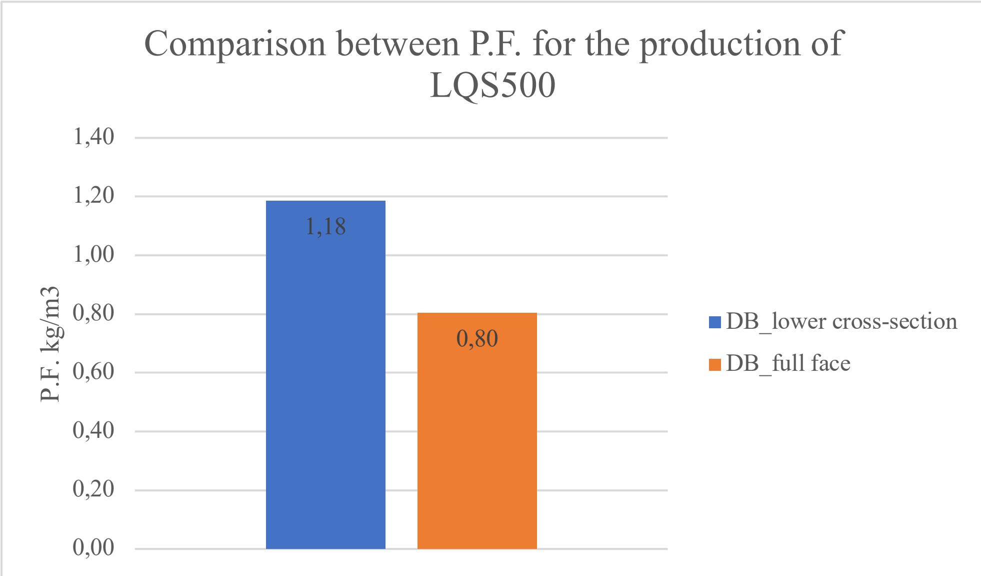

Figure 5.7 Comparison between the P.F. of the two blasting schemes

Therefore, the new configuration is expected to be cheaper than the bench scheme, even though the study does not reflect this result (Figure 5.8).

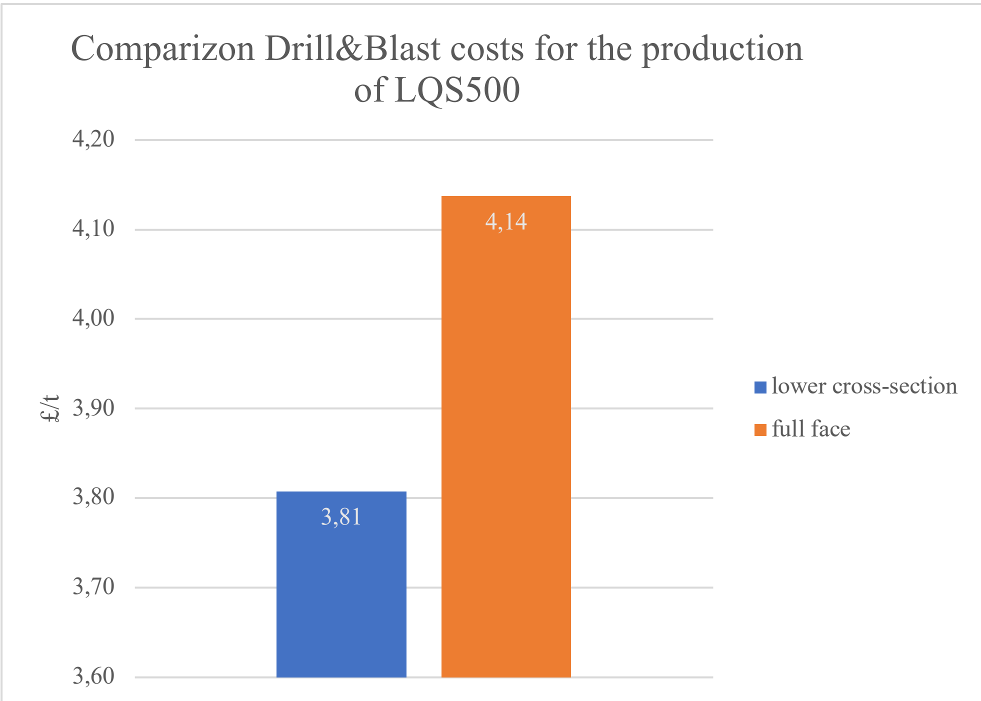

Figure 5.8 Comparison between the cost/t of the two blasting schemes

This incongruence is mainly due to an increase of the price of the following items:

- Explosive, with an increase of 2,7 %.
- Detonators, with an increase of 73 %.
- Operational costs, due to the introduction of the night shift and the consequent increase of 50 % over the hourly salary. The driller always works at night. The shot fires work during the night shift two weeks per month.

Furthermore, the benching was analysed for an average of 3,5 rounds per day, whereas the full face for 2,5. By increasing the number of rounds per day to 3,5, the average cost per tons produced will decrease to 3,68 £/t.

HIGH ENERGY IMPACT HAMMER COST ANALYSIS

For this analysis the following costs have been considered:

- Fuel
- Hydraulic oil
- Grease
- Tool
- Operator

The average cost of fuel was evaluated at 0,55 £/l. Since the consumption of fuel is monitored for all the machinery in the site, the actual consumption was evaluated. As for the consumption of oil and grease, an average value based on the total demand of the mine's equipment was considered. The costs were: 1,02 £/l for the hydraulic oil and 13,40 £/l for the grease. This final cost is considerably higher if compared with the grease used for the driller (2,36 £/l). This occurs because the grease used for the hammers is the Atlas Copco chisel pastel, that costs nearly 3 times more than the regular grease used for the other machines on the site. The two hammers consume on average half a cartridge per day (250 g) each.

The two hammers work 5 days/week for 8 h shifts. In the analysis, 85 % of this time was calculated to account for holidays, machine breakage and the lack of availability of the operators.

The cost of the hammer tools has evaluated based on the orders, considering that the tool for the HB2500 costs £ 364, while the one for the MB700 costs £ 285. For each order, the transport cost of £ 30 was added.

COSTS FOR HAMMER HM2500

The costs derived through the excavation with this machine were analysed in the mentioned study made in 2018, obtaining 1,16 £/t. The invoices have been analysed again in the period between 1/10/2018 and 30/9/2019, as shown in Table 5-19, Table 5-20 and Table 5-21.

Table 5-19 Cost analysis for the machine consumptions - HB2500

	FUEL	**GREASE**	**HYDRAULIC OIL**
Consumptions [l]	7.560,00	55,25	2985,79
Unit Cost [£/l]	0,58	13,4	1,02
TOTAL COST [£]	4.384,80	740,26	3.045,51

Table 5-20 Cost Analysis for the tool - HB2500

TOOL	
Number	2
Unit Cost [£]	365
Transport Cost [£]	30
TOTAL COST [£]	760

Table 5-21 Operational Costs Analysis - HB2500

OPERATIONAL COST	
Total worked hours [h]	1.768,00
Unit Cost [£/h]	11

TOTAL COST [£]	19.448,0

The distribution of the costs is illustrated in Figure I.9 and Table I22.

Table 5-22 Percentage Distribution of the Costs - HB2500

Fuel	15,45 %
Grease	2,61 %
Oil	10,73 %
Consumables	2,68 %
Operational	68,53 %

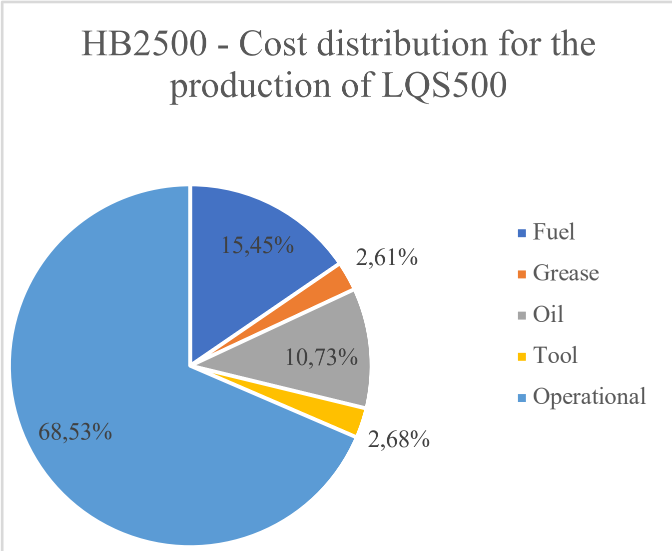

Figure 5.9 Percentage Distribution of the costs - HB2500

By considering the average production obtained with the HB2500 of 12,48 t/h, and an average of 221 working days per year, the average cost/t comes to 1,29 £/t. Therefore, in the last year the costs for the exploitation with the HB2500 have increased by 10 %.

COSTS FOR HAMMERS MB700

The MB700 was bought as a result of the 2018 analysis, when the advantages of bench excavation with hydraulic hammer became evident. The study aims to estimate the total cost of this second hammer and the cost per tons produced with this machine. As in the analysis previously mentioned, the period taken into consideration is from 1/10/2018 to 30/9/2019.

The results are listed in Table 5-23, Table 5-24 and Table 5-25.

Table 5-23 Cost analysis for the machine consumptions - MB700

	FUEL	GREASE	HYDRAULIC OIL

Consumptions [l]	5.818,00	55,25	30,33
Unit Cost [£/l]	0,58	13,40	1,02
TOTAL COST [£]	3.374,44	740,26	30,94

Table 5-24 Cost Analysis for the tool – MB700

TOOL	
Number	5
Unit Cost [£]	285,00
Transport Cost [£]	30,00
TOTAL COST [£]	1.515,00

Table 5-25 Operational Costs Analysis – MB700

OPERATIONAL COST	
Total worked hours [h]	1.768,00
Unit Cost [£/h]	11,00
TOTAL COST [£]	19.448,00

Obtaining the percentage distribution in Table 5-26 and Figure 5.10.

Table 5-26 Percentage Distribution of the Costs – MB700

Fuel	13,44 %
Grease	2,95 %
Oil	0,12 %
Consumables	6,03 %
Operational	77,46 %

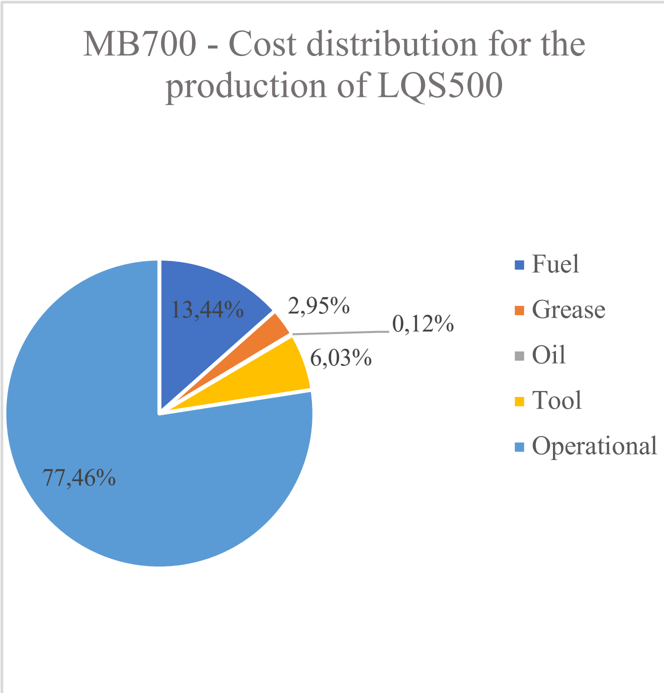

Figure 5.10 Percentage Distribution of the Costs - MB700

The average cost amounts to 1,08 £/t, considering 221 working days with a production of 13,19 t/h.

COMPARISON BETWEEN THE HAMMERS

The study shows that the MB700 is cheaper and more productive than the HB2500, even though the latter is heavier and should guarantee a greater production. The HB2500 allows a production of 12,48 t/h with an average cost of 1,29 £/t, against the 13,19 t/h, with an average cost of 1,08 £/t, obtained with the MB700.

By comparing the percentage distribution of the costs for the two hammers (Figure 5.11), it is evident that the main difference between the expenses lays in the hydraulic oil consumptions. In fact, the HB2500 uses a significant amount of hydraulic oil that increases the total costs by 10,73%. This consumption is due to the fact that the hammer is mounted on a Hyundai 30T digger, which is a large machine (the boom is more than 3 m height) and thus it can easily impact against the roof in underground, and consequently damage the hydraulic oil pumps, causing leakages.

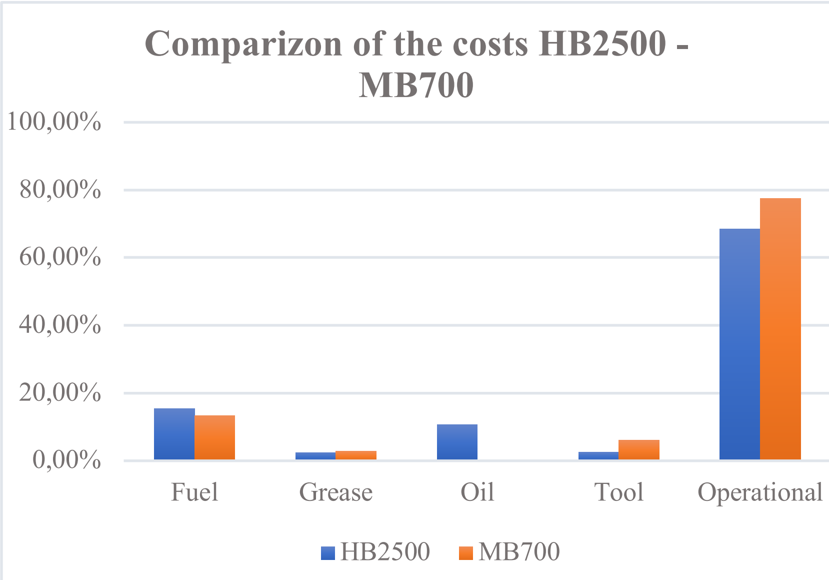

Figure 5.11 Comparison of the cost's distribution between HB2500 and MB700

COMPARISON OF THE COSTS

The costs analysis provides an evaluation of all the costs derived from each of the techniques used at Lochaline Quartz Sand mine. Each technique was evaluated based on an average hourly production and on the analysis of the actual consumption and expenses linked to each technique, thus obtaining the average cost £/t.

By comparing the results (Figure 5.12), it is evident that drill and blast is more expensive than exploitation with hammers. In fact, the full-face blasting scheme costs 4,14 £/t against the 1,18 £/t costs derived by using the two hammers working at the same time. This latter cost was obtained by dividing the total cost derived by both hammers by the total tons produced by MB700 and HB2500. Instead, the lower cross-section of the two-steps blasting has a cost of 3,81 £/t, even though this parameter is not fully comparable with the other because of the large increase of costs from 2018 to 2019.

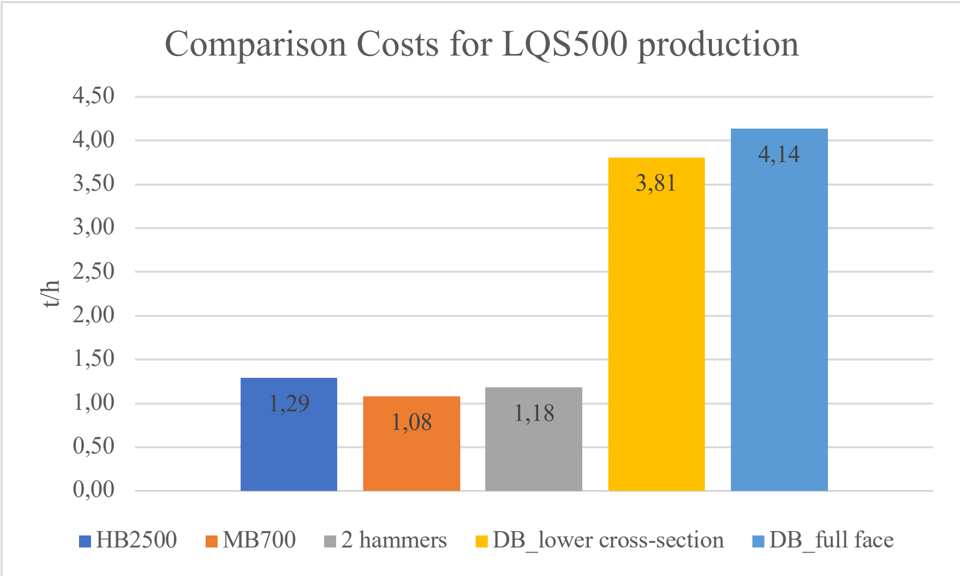

Figure 5.12 Comparison of the costs [£/t] for each technique

The huge difference between the two techniques is mainly due to the fact that the exploitation with drill and blast requires daily costs derived from the use of explosives and detonators. Another important aspect is that blasting requires a higher number of operators, one driller and three shot fires against one operator per hammer. Furthermore, the operational cost of drill and blast is increased by the work carried out during the night shift.

Therefore, considering that the full-face blasting scheme produces 2,5 rounds per day with an average of 234,85 t/round, in a day (8 h of work) the two hammers together would provide 35% of the tons produced by the drill and blast, but at 10 % of the cost.

To improve this situation two main suggestions can be provided:

- improve the drill and blast production by increasing the average round/day from 2,5 to 3,5 (as for the two-stages blasting). The average production would increase by 40 %, while the cost per tons would be reduced by 11 % circa.
- Replace the HB2500 with a lighter hammer that can be mounted on a smaller carrier which could provide a higher production. In fact, the smaller machine would ensure a massive reduction in the hydraulic oil consumption, thus the costs.

By substituting it with a second MB700 mounted on the same carrier, and hypothesizing the same hourly production obtained from this study, production would increase by 3 %, while the costs would be reduced by 8 %.

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CONCLUSIONS

The work here presented aimed to compare the underground exploitation of sandstone both on the production and economic points of view. The techniques analysed were the hydraulic hammer and the drill and blast, using a full-face and a two-stages blasting scheme. The techniques mentioned are those in use at Lochaline Quartz Sand.

Firstly, a brief description of the site was made, mainly focusing on the geology of the area, on the final product and the way the muck is processed for sale purposes. Then, a more detailed explanation of the mining method and the techniques used to exploit the sandstone deposit were described, both on the theoretical and practical point of view.

The company produces two types of silica sand, the LQS85 with a lower iron content and the LQS500, lower graded sand with a higher iron content. Since the former is only exploited by drill and blast with a full-face blasting scheme, the study involved only the production of the latter. The analysis of the productivity was carried out together with the description of the changing techniques employed over time for the exploitation, namely:

1. Two-stages blasting scheme with drill and blast
2. Benching with hydraulic hammers
3. Full-face with drill and blast

Each analysis was carried out with the goal of obtaining figures for the average hourly production [t/h]. As for drill and blast, the hourly production figure was obtained through an evaluation of the volume exploited with the corresponding blasting scheme, and the number of rounds per day (8 h shift). The theoretical and actual average powder factor were also evaluated. The lower cross-section requires 43,2 kg of explosive to blast a volume of 36,72 m^3 (P.F. = 1,17 kg/m^3). Considering an average of 3,5 rounds per day, the resulting hourly production is 35,88 t/h. The exploitation with the full-face blasting scheme involves white sand (70%) and low graded sand (30 %), obtaining a product with an average iron content suitable for the production of LQS500. Each round uses in average 122,4 kg of explosive for the production of 92,34 m^3 (P.F.= 0,99 kg/m^3). Considering an average of 2,5 rounds per day, the hourly productivity reaches 73,39 t/h.

With regard to the hammers, the average production was evaluated by observing the volume excavated in a shift. The company uses two hammers, the Atlas Copco HB2500 and the Atlas Copco MB700. The former provides an hourly productivity of 12,48 t/h, while the latter of 13,19 t/h, therefore, when working together, the productivity reaches 25,67 t/h.

Based on these results, the study highlighted that the most productive technique is drill and blast, using a full-face blasting scheme, followed by the drill and blast with a lower cross-section scheme, and the least productive technique is the mechanical excavation performed by two hammers, where the inefficiency of the Atlas Copco HB2500 is highlighted.

The cost analysis was carried out by considering the actual consumption of resources for both techniques, obtained by analysing all the invoices and the registers available for a period of one year (1st October 2018 - 30th September 2019), both for the full face blasting scheme and the two-steps scheme (lowering by hammers). For the two-steps cross-section with drill and blast, the values considered were obtained from a previous study performed in 2018. For the hydraulic hammers, the costs taken into account were hydraulic oil, fuel, grease, tools and operator costs; by knowing the hourly productivity and the amount of hours per year worked by the machines, the average cost/t produced was evaluated.

For both the drill and blast schemes, the parameters considered were the costs of detonators, explosives, stemming, initiation system, fuel, grease, hydraulic oil, drilling tools and the operational costs. To identify possible waste and potential inefficiencies of the techniques applied, the expected and the actual cost/t were evaluated by checking the blasting calendar. To have a global comparison only the average cost/t were considered.

The analysis of the costs revealed that drill and blast is the most expensive technique (4,14 £/t for the full face blasting and 3,81 £/t for the lower cross section blasting), while the two hammers resulted in being the most economic technique (1,18 £/t when the two hammers are used at the same time) with a saving of 70 %.

From these results, an anomaly was identified. According to the powder factor, the lower cross-section should result in a greater expense than the full-face blasting. However, this is not evident in the cost analysis of the invoices studied in the two periods. The anomaly is due to an increase of costs that has occurred during the two periods under investigation.

A further anomaly lies in the mechanical techniques production. The HB2500 should result in being more productive than the MB700, but the analysis showed the contrary. The inefficiency in using HB2500 hammer is due to the fact that it is mounted on a Hyundai 320, a big carrier that does not allow the efficient handling of the machine inside this mine characterized by small dimensions of tunnels. Damage to the hydraulic oil pipes is frequent and this leads to an increase in hidden times.

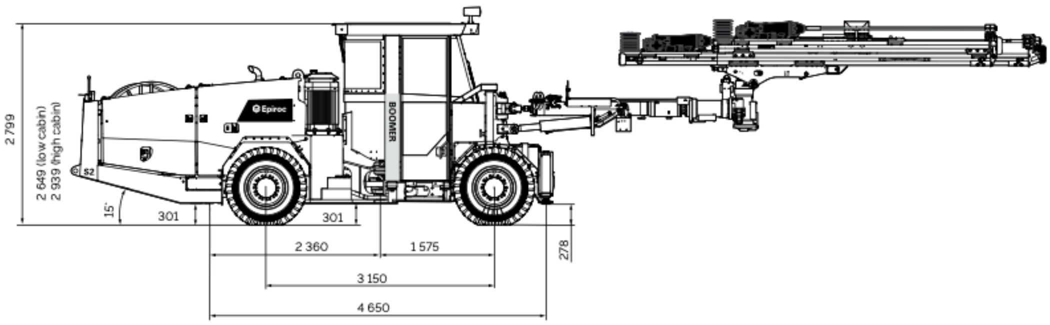
In conclusion, the study gave the predictable outcome that the drill and blast technique is more productive but more expensive than the exploitation with hammers. In fact, considering the average values obtained from the analysis, in one day of work the hammers were able to obtain less than half of the production achieved with drill and blast, but at 15 % of the cost. In addition, the study highlighted how interrupting the two stages blasting for the full-face blasting scheme resulted in an improvement both in terms of productivity and costs. This improvement could be further enhanced by increasing the number of rounds from 2,5 rounds/d to 3,5, as it’s commonly done for the production of LQS85.

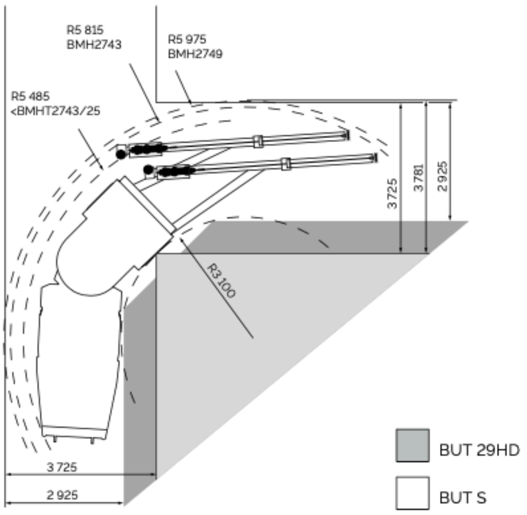
Furthermore, the study suggested investing in a smaller hammer, and thus a carrier smaller than Hyundai 320. This solution would mean to purchase a less powerful machine, but easier to manoeuvre, reducing the risk of breakages and, thus, decreasing the costs and the downtime.

APPENDIX A - MACHINE'S TECHNICAL SPECIFICATIONS

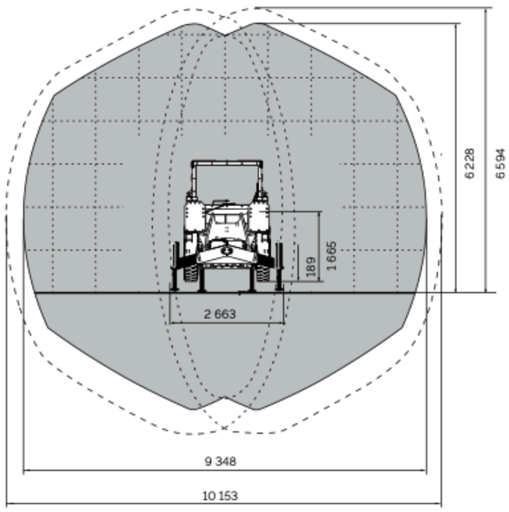
DRILLER – ATLAS COPCO 282 S

Dimensions in millimeters





Boomer S2 turning radius.



Boomer S2 coverage area.

Dimensions

Width	2 000 mm
Height with cabin	2 799 mm
Height roof up/down	2 850/2 155 mm
Length with BMH 2814 feed(s)	12 216 mm
Ground clearence	278 mm

Tramming speed

On flat ground (Rolling resistance 0.05)	>15 km/h
On incline 1:8	>5 km/h

Gross weight (depending on configuration)

Rig type	Total	Boom side	Engine side
Two boom rig	18 000-21 000 kg	11 000-12 500 kg	7 000-8 500 kg

Source: Epiroc – Boomer 282 technical specification- https://www.epiroc.com/en-uk/products/drill-rigs/face-drill-rigs/boomer-282

HYUNDAI R320

Operating weight (approximate)

Operating weight, including 6450mm (21′ 2″) boom, 3200m (10′ 6″) arm, SAE heaped 1.44m³ (1.88 yd³) backhoe bucket, lubricant, coolant, full fuel tank, hydraulic tank and the standard equipment.

Major component weight

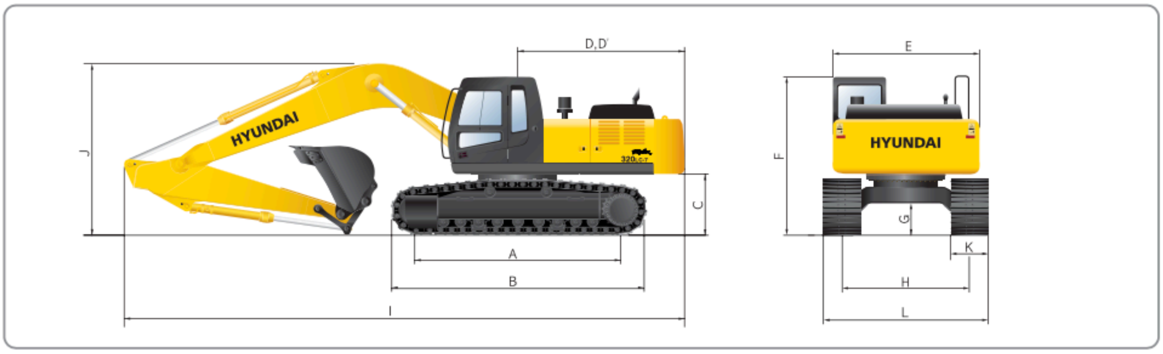
Upperstructure	8320kg (18340lb)
Counterweight	6200kg (13670lb)
Boom (with Arm cylinder)	3030kg (6680lb)

Operating weight

Shoes		Operating weight		Ground pressure
Type	Width mm(in)	kg(lb)		kgf/cm²(psi)
Triple grouser	※**600 (24)**	**R320LC-7**	**32200 (71000)**	**0.62 (8.82)**
		R320NLC-7	32000 (70500)	0.61 (8.67)
		R320LC-7 H/C	34700 (76500)	0.67 (9.53)
	700 (28)	R320LC-7	32800 (72300)	0.54 (7.68)
		R320LC-7 H/C	35300 (77800)	0.58 (8.25)
	800 (32)	R320LC-7	33200 (73200)	0.48 (6.83)
		R320LC-7 H/C	35700 (78700)	0.51 (7.25)
	900 (36)	R320LC-7	33600 (74100)	0.43 (6.11)
Double grouser	700 (28)	R320LC-7 H/C	35900 (79100)	0.58 (8.25)

※Standard equipment

Dimensions R320LC-7 / R320NLC-7



mm (ft · in)

A	Tumbler distance	R320LC-7	4030	(13′ 3″)
		R320NLC-7	4030	(13′ 3″)
B	Overall length of crawler		4940	(16′ 2″)
C	Ground clearance of counterweight		1200	(3′ 11″)
D	Tail swing radius		3330	(10′ 11″)
D′	Rear-end length		3265	(10′ 9″)
E	Overall width of upperstructure		2980	(9′ 9″)
F	Overall height of cab		3090	(10′ 2″)
G	Min. ground clearance		500	(1′ 8″)
H	Track gauge	R320LC-7	2680	(8′ 10″)
		R320NLC-7	2390	(7′ 10″)

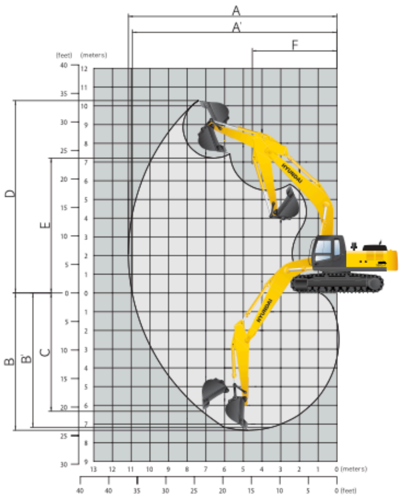
mm (ft · in)

	Boom length	※**6450 (21′ 2″)**				6150 (20′ 2″)
	Arm length	2200 (7′ 3″)	2500 (8′ 2″)	※**3200 (10′ 6″)**	4050 (13′ 3″)	2200 (7′ 3″)
I	Overall length	11230 (36′ 10″)	11100 (36′ 5″)	**10980 (36′ 0″)**	10980 (36′ 0″)	10930 (35′ 10″)
J	Overall height of boom	3640 (11′ 11″)	3670 (12′ 0″)	**3380 (11′ 1″)**	3860 (12′ 8″)	3680 (12′ 1″)

K	Track shoe width		※ **600 (24″)**	700 (30″)	800 (32″)	900 (36″)
L	Overall width	R320LC-7	**3280 (10′ 9″)**	3380 (11′ 1″)	3480 (11′ 5″)	3580 (11′ 9″)
		R320NLC-7	**2990 (9′ 10″)**	-	-	-

※Standard Equipment

Working ranges R320LC-7 / R320NLC-7



mm (ft · in)

	Boom length	※6450(21' 2")				6150(20' 2")
	Arm length	2200 (7' 3")	2500 (8' 2")	**※3200 (10' 6")**	4050 (13' 3")	2200 (7' 3")
A	Max. digging reach	10330 (33' 11")	10550 (34' 7")	**11140 (36' 7")**	11950 (39' 2")	10020 (32' 10")
A'	Max. digging reach on ground	10110 (33' 2")	10330 (33' 11")	**10940 (35' 11")**	11760 (38' 7")	9800 (32' 2")
B	Max. digging depth	6370 (20' 11")	6670 (21' 11")	**7370 (24' 2")**	8220 (26' 12")	6160 (20' 3")
B'	Max. digging depth (8' level)	6160 (20' 3")	6470 (21' 3")	**7210 (23' 8")**	8080 (26' 6")	5950 (19' 6")
C	Max. vertical wall digging depth	5980 (19' 7")	5920 (19' 5")	**6360 (20' 10")**	7260 (23' 10")	5710 (18' 9")
D	Max. digging height	10220 (33' 6")	10170 (33' 4")	**10310 (33' 10")**	10710 (35' 2")	9940 (32' 7")
E	Max. dumping height	7050 (23' 2")	7050 (23' 2")	**7240 (23' 9")**	7630 (25' 0")	6780 (22' 3")
F	Min. swing radius	4700 (15' 5")	4500 (14' 9")	**4470 (14' 8")**	4470 (14' 8")	4520 (14' 10")

※Standard Equipment

Source: R320LC-7 – Hyundai Heavy Industries – Catalogs https://pdf.directindustry.com/pdf/hyundai-heavy-industries/r320lc-7/17582-308881.html

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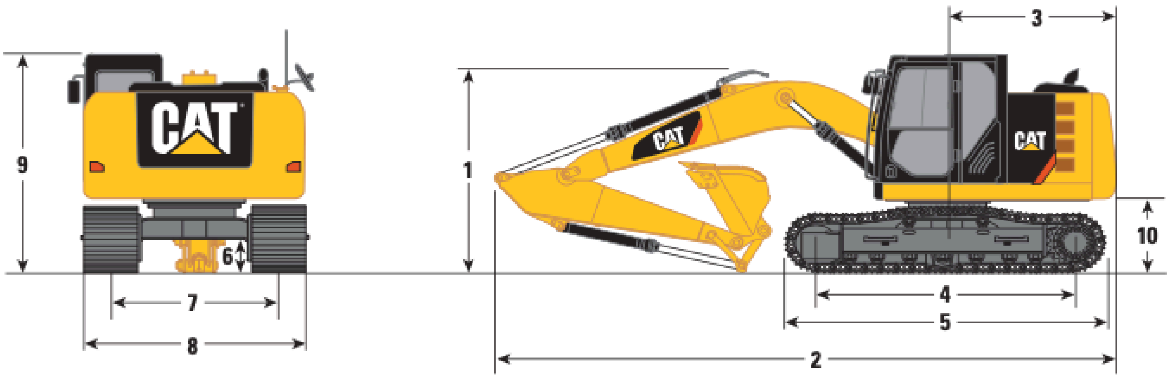
Dimensions

All dimensions are approximate.



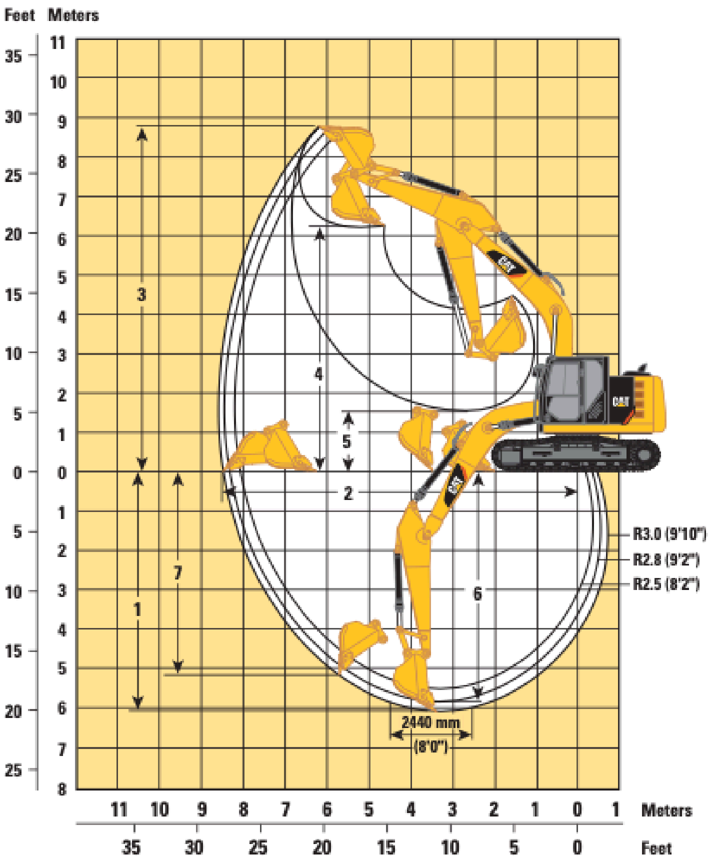
	Reach Boom 4.65 m (15'3")		
Stick	**R3.0 (9'10")**	**R2.8 (9'2")**	**R2.5 (8'2")**
	mm (ft)	**mm (ft)**	**mm (ft)**
1 Shipping Height*	2980 (9'9")	2980 (9'9")	2980 (9'9")
Shipping Height at Boom Top	2830 (9'3")	2970 (9'9")	2830 (9'3")
Shipping Height with Guard Rail	2980 (9'9")	2980 (9'9")	2980 (9'9")
Shipping Height with Top Guard	2970 (9'9")	2970 (9'9")	2970 (9'9")
2 Shipping Length			
Long Undercarriage	7670 (25'2")	7650 (25'1")	7670 (25'2")
Long Undercarriage with Blade	7960 (26'1")	7920 (26'0")	7950 (26'1")
3 Tail Swing Radius	2160 (7'1")	2160 (7'1")	2160 (7'1")
4 Length to Center of Rollers			
Long Undercarriage	3040 (10'0")	3040 (10'0")	3040 (10'0")
5 Track Length			
Long Undercarriage	3750 (12'4")	3750 (12'4")	3750 (12'4")
6 Ground Clearance	440 (1'5")	440 (1'5")	440 (1'5")
7 Track Gauge	1990 (6'6")	1990 (6'6")	1990 (6'6")
8 Transport Width			
500 mm (20") Shoes	2490 (8'2")	2490 (8'2")	2490 (8'2")
600 mm (24") Shoes	2590 (8'6")	2590 (8'6")	2590 (8'6")
700 mm (28") Shoes	2690 (8'10")	2690 (8'10")	2690 (8'10")
770 mm (30") Shoes	2760 (9'1")	2760 (9'1")	2760 (9'1")
9 Cab Height	2770 (9'1")	2770 (9'1")	2770 (9'1")
Cab Height with Top Guard	2970 (9'9")	2970 (9'9")	2970 (9'9")
10 Counterweight Clearance**	890 (2'11")	890 (2'11")	890 (2'11")

*Including shoe lug height.

**Without shoe lug height.

Working Ranges

All dimensions are approximate.



	Reach Boom 4.65 m (15'3")		
Stick	**R3.0 (9'10")**	**R2.8 (9'2")**	**R2.5 (8'2")**
	mm (ft)	**mm (ft)**	**mm (ft)**
1 Maximum Digging Depth	6040 (19'10")	5840 (19'2")	5540 (18'2")
2 Maximum Reach at Ground Level	8620 (28'3")	8430 (27'8")	8170 (26'10")
3 Maximum Cutting Height	8710 (28'7")	8590 (28'2")	8490 (27'10")
4 Maximum Loading Height	6330 (20'9")	6210 (20'4")	6100 (20'0")
5 Minimum Loading Height	1530 (5'0")	1730 (5'8")	2020 (6'8")
6 Maximum Depth Cut for 2440 mm (8'0") Level Bottom	5860 (19'3")	5650 (18'6")	5330 (17'6")
7 Maximum Vertical Wall Digging Depth	5200 (17'1")	5070 (16'8")	4840 (15'11")

Major Component Weights

	kg	lb
Base Machine (with boom cylinder, without counterweight, front linkage and track)	5120	11,290
Long Undercarriage	2600	5,730
Counterweight 2.2 mt (2.4 t)	2200	4,850
Boom (includes lines, pins and stick cylinder)		
Reach Boom – 4.65 m (15'3")	1010	2,230
Stick (includes lines, pins, bucket cylinder, and bucket linkage)		
R3.0 (9'10")	560	1,230
R2.8 (9'2")	530	1,170
R2.5 (8'2")	480	1,060
R3.0 (9'10") for Thumb	610	1,350
Track Shoe (Long/per two tracks)		
500 mm (20") Triple Grouser	1560	3,440
600 mm (24") Triple Grouser	1820	4,010
700 mm (28") Triple Grouser	2100	4,630
770 mm (30") Triple Grouser	2240	4,940
Blade		
2500 mm (8'2")	810	1,790
2600 mm (8'6")	810	1,790
2700 mm (8'10")	820	1,810

All weights are rounded up to nearest 10 kg and lb except for buckets. Kg and lb were rounded up separately so some of the kg and lb do not match.
Base machine includes 75 kg (165 lb) operator weight, 90% fuel weight, and undercarriage with center guard.

Source: Catterpillar – 312E Hydraulic excavator- http://s7d2.scene7.com/is/content/Caterpillar/C726422