POLITECNICO DI TORINO

Collegio di Ingegneria per l'Ambiente e il Territorio

Corso di Laurea Magistrale
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Orientamento Geo-Ingegneria

Tesi di Laurea Magistrale

IMPROVEMENT OF THE DRILL AND BLAST EXCAVATION TECHNIQUE IN A CHILEAN ARTISANAL GOLD MINE

Relatore

Prof.ssa Marilena Cardu

Candidato

Dott. Simone Bez

Anno Accademico 2018/2019

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SOMMARIO

Lo scopo dello studio svolto è quello di realizzare un progetto di miglioramento della tecnica di scavo mediante Drill and Blast in una miniera artigianale di oro situata presso il paese di Chancón, vicino alla città di Rancagua (Regione del Liberatore Generale B. O'Higgins), Cile.

Durante il tempo di permanenza in Cile, pari a circa 7 mesi, sono stati raccolti dati ed informazioni presso la miniera. Tali dati sono stati analizzati e studiati criticamente da un punto di vista tecnico-scientifico. Sono state, in particolare, osservate carenze tecnologiche che rendono il processo estrattivo improduttivo, oltre che pericoloso. I sopralluoghi effettuati hanno permesso di osservare non solo gli aspetti tecnici e tecnologici ma anche quelli socioeconomici e culturali presenti nel contesto geografico dove è ubicata la miniera. Nella redazione del progetto migliorativo sono stati presi in considerazione una serie di fattori, discussi nell'ambito del presente elaborato.

Gli interventi migliorativi proposti si articolano in tre fasi distinte e progressive, al fine di evitare cambiamenti troppo radicali e bruschi che non sarebbe possibile mettere in pratica per motivazioni di carattere logistico e culturale. La prima fase di cambiamento riguarda l'introduzione di un nuovo procedimento di lavoro e di un differente sistema di innesco della volata (sistema nonel). La seconda prevede l'introduzione di un differente tipo di esplosivo per i fori di contorno ed una nuova geometria della sezione di scavo. La terza fase comporta l'introduzione di un Jumbo per la perforazione, l'incremento dello sfondo (avanzamento/volata) e la modifica degli esplosivi impiegati. Per ogni fase sono stati presentati i calcoli realizzati ed i disegni tecnici degli schemi di perforazione e di caricamento.

L'elaborato è articolato in due capitoli: nel primo si descrivono le operazioni, le tecniche e le procedure di scavo osservate nella miniera, evidenziando le problematiche riscontrate e tenendo conto della sicurezza sul lavoro. Il secondo capitolo è dedicato alla presentazione dei calcoli e degli schemi grafici di perforazione e caricamento relativi alle soluzioni progettuali proposte. Nelle conclusioni si confrontano i risultati ottenuti con dati reperiti in letteratura tecnica. In allegato sono state inserite 55 tavole progettuali che potrebbero essere illustrate agli operatori al fine di esplicitare i problemi riscontrati nell'attuale metodologia di lavoro e proporre, attenendosi ad una sequenza relativamente semplice da attuare e soprattutto efficace, modifiche essenziali utili a migliorare la tecnica attuale, sia in termini di produttività (grazie all'introduzione di un adeguato grado di meccanizzazione) sia di incremento della scurezza in cantiere (grazie alla modifica del sistema dell'innesco, del tipo di esplosivi impiegati e della sezione di scavo).

SUMMARY

The aim of the study here presented is to carry out a project to improve the D&B technique in a artisanal gold mine located in the village of Chancón, near Rancagua city (Liberator General B. O' Higgins region), in Chile.

During the time spent in Chile, which lasted about 7 months, data and information were collected at the mining site. These data were critically analysed and studied from a scientific point of view. Many drawbacks, both from a technical and organizational point of view, have been detected. These make the extraction process inefficient and unproductive, as well as dangerous. The inspections carried out made it possible to observe not only the technical and technological aspects, but also the socio-economic and cultural traits that characterize the geographical contest where the mine is located. To realize the improvement, several factors were taken into account, very one of them discussed in the present work.

The interventions are divided into three distinct and progressive phases, in order to avoid too radical and abrupt changes that would not be easily put into practice for logistical and cultural reasons. The first step consists in the introduction of a new work process and a different blast triggering system (nonel system). The second involves the introduction of a different type of explosive for the contour blast-holes and a new geometry of the excavation cross-section. The third phase involves the introduction of a Jumbo for drilling, increasing the pull and changing the explosives used. For each phase, both calculations and technical drawings of the drilling and loading schemes are presented.

The work is developed into two chapters: the first describes the operations, techniques and excavation process observed in the mine, observing the problems encountered, also taking into account the work safety. The second chapter is devoted to the presentation of calculations and drilling and charging schemes related to the proposed solutions. In the conclusions the results obtained are compared with data found in the technical literature. Attached to the project, 55 technical drawings are presented, that could be helpful also for miners to explain the main problems encountered in the current working method and to show the improvements achievable through the suggested changes.

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INTRODUCTION

I.1 - The importance of mining in Chile

The paragraph that follows synthetizes the content of the document: "*Chilean attitudes toward mining: Citizen Survey – 2014 Results*". Moffat, K., Boughen, N., Zhang, A., Lacey, J., Fleming, D. & Uribe, K. (2014).

Chile is one of the world's leading copper producers and in 2010 produced around 34% of global copper. It is also a major producer of gold, silver, iron and lithium. The development of the mining sector has contributed, in recent years, to the rapid improvement and progress of the Chilean economy. In 2012, the mining industry contributed to the 14.2% of the Chile's Gross Domestic Product (GDP) and to 59.4% of the country's total exports, with a turnover of USD $46,537 million.

According to the National Statistic Institute (INE), in 2012, the mining industry contributed to the employment of 3.3% of the total Chilean workforce, i.e. 250,800 people. The forecasts made by the Chilean Centre for Copper and Mining Studies (CESCO) have predicted that the investments that will be disbursed to the South American mining industry during the period from 2011 to 2020 will amount to $327 billion USD, of which $75 billions USD only for the Chilean mining industry.

Mining has a long history in Chile: through this story, mining has been strongly influenced by political concerns.

As already mentioned, the mining sector has certainly been a fundamental pillar in the development and progress of the Chilean economy. This was also possible thanks to the support provided by a legal and regulatory framework that contributed to the stability and security of investments, sometimes even foreign capital. On the other hand, understanding the impact that this industry has had on Chilean society in terms of human, cultural, well-being and welfare development is less simple. For this reason, the mining industry occupies an important position in Chile's political and social discourse, reflecting a complex relationship between industry, government and population. The relationship between mining and society is obviously very complex.

I.2 - Artisanal and Small-Scale Mining in Chile

The data and information presented in the following paragraph were taken from the document: "*The Socioeconomic Impacts of Artisanal and Small-Scale Mining in Chile*". Castro S.H., (2003).

From a socio-economic point of view, artisanal mining (also called Small-Scale Mining or SSM), is a very important sector in Chile. The sector counts on a total of direct and indirect workers between 10,000 and 15,000, taking into account both the artisanal miners directly engaged in the exploitation sites and those who work at the processing plants. The main raw materials extracted are gold, copper, silver and coal.

The statistics on the production of metal (year 2001) show that Small-Scale Mining produced 40,675 m^3 of Cu content, i.e. 0.85% of the total production, 2,578 kg of Au (6.04% of the total) and 11,694.2 kg of Ag (0.87% of the total). These data are visible in the following Table I.1:

Table I.1 – Metallic production statistics (Anuario de la Minería de Chile, 2001)

Mining-scale	Cu content (t)	Mo content (t)	Au (kg)	Ag (kg)	Pb content (t)	Zn content (t)
Large-scale	4,432,463	33,492	20,040.9	1,052,404.9	-	-
Medium-scale	292,924	-	20,053.6	284,567.7	1,193	32,762
Small-scale	40,675	-	2,578.1	11,694.2	-	-
Total	4,766,062	33,492	42,672.6		1,193	32,762

The gold production reached 84.1% of the total production from artisanal mining.

Statistics (year 1994) have identified approximately 1,626 artisanal mines and 281 artisanal treatment plants across the country.

Underground and open pit mines, quarries and waste mining are actually exploited in Chilean mining. Most of the mines are underground (about 92% of the work sites) and provide approximately 95% of the overall production.

76% of the work sites exploits copper. Almost 80% of the copper coming from the Small-Scale Mining sector relates to these sites. Gold is generally exploited in small mines, supplying only the 11% of the overall ore extracted from the artisanal mining industry. There are few poly-metallic mines. The mine analysed in this study exploits mainly gold, though silver and copper being also extracted as secondary products. Silver deposits are not related to the Small-Scale mining sector, whereas Molybdenum is only produced as a by-product from large-scale copper mining and is never extracted as the main raw material.

According to statistics elaborated by SERNAGEOMIN in 2001, about 2,120 artisanal miners work in copper-related mining, while 1,485 work in gold mining. In total, therefore, 3,606 miners were directly involved in the Small-Scale metal mining sector, about 9.50% of workers involved in gold and copper mining (Anuario de la Minería de Chile, 2001). Given the noteworthy contribution to the National production, gold mining is the most important sector for artisanal mining.

I.2.1 - Organization of the Chilean mining industry

The Chilean mining sector is very differentiated and therefore it's quite complicated carrying out a detailed classification. However, by simplifying, three categories can be identified based on the size of the mining sector:

- Large-Scale Mining, characterized by a production of more than 10,000 of Cu content /year;
- Medium-Scale Mining, characterized by a production of more than 200 t of ore/day;
- Small-Scale Mining, characterized by a production less than 200 t of ore/day; according to SERNAGEOMINS's Resolution n. 0408 (January 15, 1998), the Small-Scale Mining is classified as the activity that has less than 200,000 man hours worked in one year, with an average of 80 workers/year.

In addition, the mining sector can be divided according to the property into: State-owned and Private companies.

For many years, the State has provided technical and financial support to artisanal miners. This has allowed their livelihood even in times of crisis as a result of the fluctuations in the prices of mineral products.

The Small-Scale Mining has played an important role in the history of the Country for more than 2 centuries. It should be remembered that the SSM was a precursor of mining exploration. In fact, many of the large deposits that are now being exploited on a large scale have been discovered and exploited at the beginning by artisanal miners.

Artisanal mining is a traditional activity that generates employment for the low-income population. Artisanal miners can work in mines that are owned by themselves or in someone else's mines and can produce ore for direct commercialization or process in small concentration plants. The Chilean Small-Scale Mining can be divided into two categories according to the work organization:

- Formal Small-Scale Mining, where miners or plants operate with a certain level of mechanization, usually rather low. This category includes the mine analyzed in this study;
- Artisanal Small-Scale Mining, where the ore is extracted from very small sites and sometimes processed in small plants. These miners are a poor part of the Chilean society and are generally called "*pirquineros*". There is no formal organization among them, who therefore do not represent any formally recognized society.

I.2.2 - Structure of the Chilean Small-Scale Mining sector

All Small-Scale miners sell their products to ENAMI agencies who work as promotional and regulatory bodies, processors and exporters. ENAMI has the legal obligation to process the ore extracted from small producers that comply with the promotional policies of the State, selling most of their product to ENAMI, according to a tariff system.

The organisms and institutions having the task of providing institutional support for the mining sector are shown, according to their role within the system, in Figure I.1:

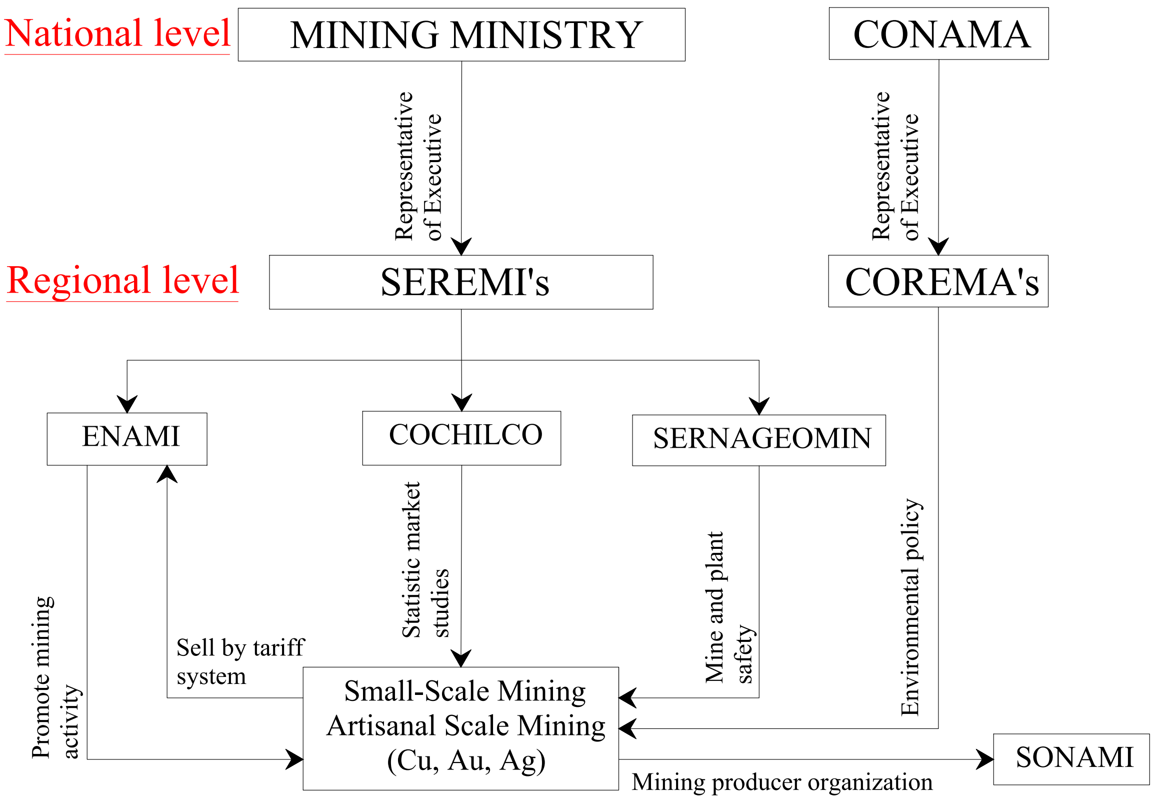

Figure I.1 - Institutional support framework for Chilean mining industry (The Socioeconomic Impacts of Artisanal and Small-Scale Mining in Chile". S.H. Castro, 2003)

The institutions shown in Figure I.1 are explained in the following paragraph.

MINING MINISTRY: As the representative of the mining executive, the Ministry has the task of planning and implementing the relevant government policies. The Ministry works practically through a Regional Ministerial Secretary, called SEREMI, which coordinates with the general main offices located in capital, Santiago. The Ministry is responsible for the National Geology and Mining Service, called SERNAGEOMIN. There is also an Environmental Unit of the Mining Ministry which has the task of promoting, both environmentally and economically, sustainable mining activities, that works in conjunction with CONAMA.

ENAMI (National Mining Company). It has the task of stimulating mining activities and contributing to the sustainability of work sites, especially for small and medium-sized mining companies. ENAMI has two main operating sectors: production and promotion. Its job is to facilitate the access of small and medium-sized companies in the global market that would otherwise be unattainable. The role played by the company in the Small-Scale Mining sector is established in accordance with the law: the obligation to purchase the entire production of the artisanal mining is imposed according to a defined tariff and the granting of subsidies in order to mitigate the effects of the fluctuations in the price of metals on the international market. ENAMI does not own mines, only buys raw materials from small and medium-sized companies to carry out processing in its own concentration plants.

SERNAGEOMIN (National Geology and Mining Service). It is a public body directly dependent by the Mining Ministry which has the task of supervising mining activities, exploration and mining research and geological services. It is also responsible for the definition of the rules on the projects for the placement of the waste dumps and disposal of industrial and leaching residues.

COCHILCO (Chilean Copper Commission), is a technical entity specialized in copper mining with functions of sectorial studies, legal and environmental consulting.

SONAMI (National Mining Society), is the company that represents the interests of its members that belong to the Large, Medium and Small-Scale Mining sectors. The Environmental Commission is focused on the topics of training, regulation and international tendencies, especially with regard to environmental aspects.

CONAMA (National Environmental Commission), is a body defining terms for studies of environmental impact and impact analyses according to the Basic Environmental Law. It coordinates the work of the Ministries and Public Services, grants permissions and supervises the compliance with the corresponding norms. It operates at regional level through the commissions called COREMA'S.

COREMA (Regional Environmental Commissions), present in all 13 regions, COREMA’s must assure compliance with the Basic Environmental Law and establishes the lines of environmental policy at national level.

I.2.3 - Characteristics of the Chilean Small-Scale Mining

The characteristics of the Chilean Small-Scale Mining are common to all situations and contexts. A low level of mechanization and consequently a wide use of burdensome manual work can be noticed. The artisanal operators have a very low technical level and the professionals are very few.

The knowledge of the deposit reserves, if present, is minimal and insufficient. The conditions of safety and hygiene are very precarious. It can be observed a deficient use of mineral resources due to a selective exploitation of high-grade ores. This is, besides being unproductive, improper from an environmental point of view. Small-Scale Mining is often characteristic of geographical areas where job alternatives are scarce. For example, the mining area of Chancón offers, as well as work in mining, agricultural work in fields and orchards but it is scarce, poorly paid and generally seasonal.

There is a direct relationship between the degree of poverty and the precariousness of work. Miners belonging to this sector are marginal and generally do not own any mining properties. Usually these workers possess little or no patrimony and use rudimentary work techniques. On average, they work in groups of up to 8 people at most, and the production per person usually does not exceed 1 t/d. This social class is generally composed of middle-aged people with low levels of education and early exclusion from the national educational system.

Almost everyone works at a site that is not owned by them and with a certain level of individualism and is usually associated with informal organizations. The working methods they use are unsafe, manual and involve low productivity. Wages are very low and unstable (in the mine that has been analysed, driller and assistant earn 380,000 Chilean Pesos per month, about $567 USD), work is often seasonal and linked to the work alternatives offered within the same mining sector.

The mining deposits are exploited selectively: the selection criterion is linked to the prices of metals on the international market. Artisanal miners in the Small-Scale Mining sector must trade their products with third parties. Since many companies do not have direct access to ENAMI, they have to sell their products to medium-scale companies or other small-scale companies larger than themselves.

Miners working in the context of Small-Scale Mining are significantly exposed to occupational accidents, occupational diseases and hygiene problems. In general, there is limited access to the health system and an extremely reluctant environment to the improvement of these conditions, even by the miners.

The main problems encountered in the Chilean Small-Scale Mining sector concern: low technological and technical level applied to the exploitation, high exploitation costs, low levels of productivity, unknown availability of the reserves, low training level and traditional and inadequate work organization.

The environmental impact of mines and artisanal processing plants afflicts water, land and the air, causing problems to the natural environment and life of the neighbouring populations. Water contamination derives from the emission of effluent liquids containing dissolved heavy metals, acids and processing reagents as well as from active and abandoned tailings deposits. This is simply due to the fact that the artisanal mining sector does not have an environmental management, and the economic resources to invest in this aspect are very limited or non-existent.

Despite the aforementioned problems caused by Small-Scale Mining, it plays a very important role in the Chilean society. In fact, it offers an opportunity for work and income in remote rural areas where there is often little productive activity, making it the only alternative for working in sectors suffering from poverty and poor job options. It therefore has a strong impact on local communities, defining a dynamic contribution to their economy. However, it should be noted that this activity offers low wage levels, is unstable as it depends on climatic factors or fluctuations in the price of metal, lacks of capital and is socially and legally conflictive (compliance with wage standards, benefits, work sites installation permissions, mining concessions, etc.).

From a socio-economic point of view, the artisanal mining is characterized by high levels of poverty, scarce education, health and lack of work alternatives. It is strongly characterized by a deep-seated cultural mining tradition in which a movement to change is not desirable in the short term.

Artisanal miners can be divided into two categories according to their characteristics:

1) Artisanal miners whose presence in the sector is linked to fluctuations in the price of metal. Their work is itinerant and sporadic. Generally, this category is made of a group of young miners working on other activities or of unemployed miners living in mining areas or migrating to them, taking advantage of periods when the price is high by linking to the mining sector through very short mining rent contracts;

2) Miners whose main activity is mining, and who remain in the mining sector to which they belong, despite the difficult conditions linked to the price of the mineral. This group is generally made up of old miners, whose continuous presence is explained by cultural reasons that constitute their tradition, as well as a low aversion to the risky nature of this type of work and the lack of working alternatives for subsistence.

In general, the level of education of artisanal miners is low and depends on their age. The intermediate education held by artisanal miners in the age ranging between 14 and 29 years is 8 years, for those ranging between 30 and 45 years is 6 years, and for those in the 45- 65 range, is only 5 years. A total lack of education can be found in 3.9% of artisanal miners (Muñoz, 1999). Most of the Small-Scale Miners are aged between 26 and 40 and have a family of 4 to 7 members.

Generally speaking, artisanal mining is characterized by a wide range of limitations, including: a high percentage of renters with short-term rent contracts, poor or no knowledge of potentially exploitable reserves, a mining property situation that does not permit development and improvement; very limited or no access to funding.

The artisanal mining works with high grades ores and the high cost of production is the result of several factors: use of artisanal and rudimentary technologies, royalty payments to property owners, large distances travelled to the concentrations plants, etc. In Chile there are no indigenous groups associated with mining activities, a situation that is present in other Latin American countries.

I.3 - The artisanal mine analysed

The mine analysed in this study, called "*La Palmera*", is located in the Chancón mining sector, about 25 km away from the city of Rancagua, in the province of Cachapoal, in the 6th region, named the Gen. Liberator General Bernardo O'Higgins, in Chile. The UTM coordinates of the mine are: North 6.229.783 and East 329.424. The mine is located at an altitude of 553 m.a.s.l., at the entrance to the access level 0. The location of the mine is shown in Figure I.2.

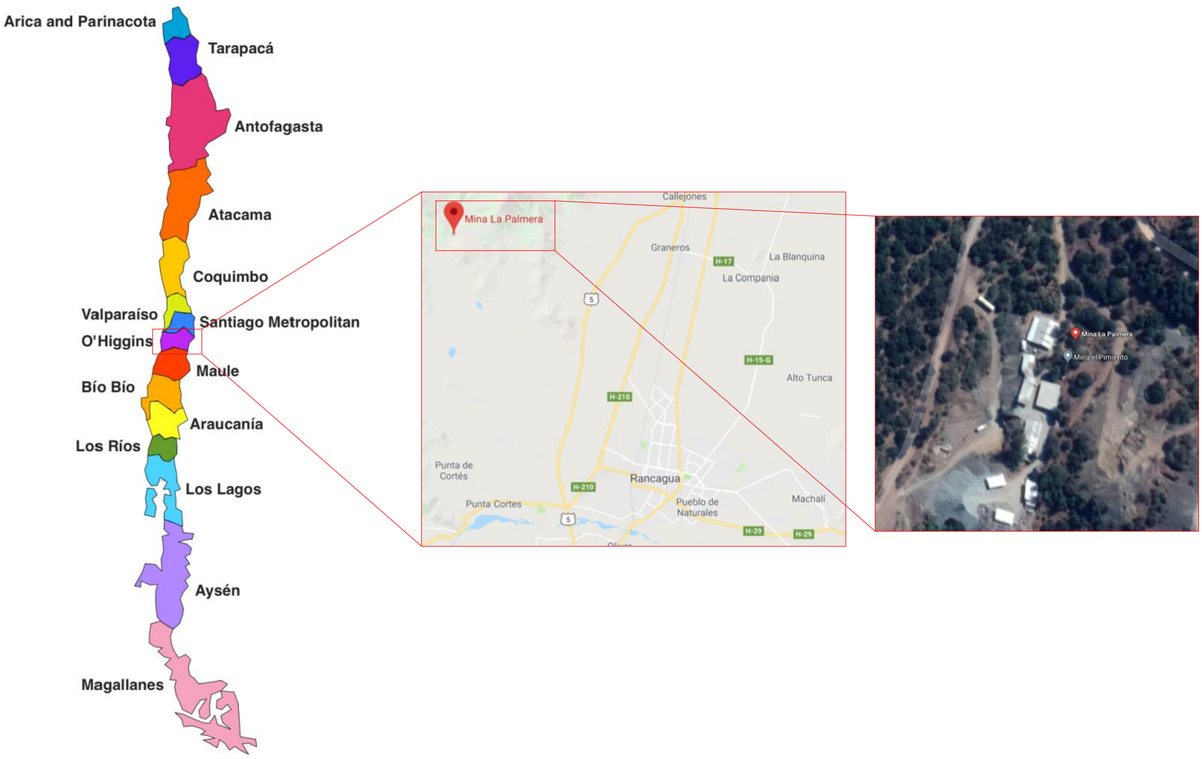

Figure I.2 - Geographical location of the "La Palmera" mine. The image of the regions of Chile is taken from [1], the satellite image on the right is taken from Google Maps

This mine, formerly called "*El Pimiento*", is a small gold mine, within the other many small mines in the sector. The mine changed its name from *Mina* "*El Pimiento*" to *Mina* "*La Palmera*" due to the change of ownership. A photograph of the access road to the Chancón mining sector is shown in Figure I.3.

The mining sector of Chancón contains a total of about 40 mines, all belonging to the Small-Scale Mining sector. The mine analysed is the largest mine in the sector even if all the companies operate, regardless of their size, according to the same method of work and using the same excavation techniques.

Figure I.3 - Access road to the Chancón mining sector. The image is taken from [2]

The exploitation of the "*La Palmera*" mine began 22 years ago, when José Jiménez Navea, mining engineer graduated at the University of Santiago, became the owner of the plot of land. In 1997 Manuel Olivares Duque was hired as a mining manager due to his 26-year mining experience. His main function was to oversee the production of gold and copper, cooperation in the practical training of new generations of miners, also giving the opportunity to universities, institutes and schools, to visit the mine and observe and operations of the excavation on site. In 2016, José Jiménez Navea died, leaving his sons José Matías and Joaquín as owners. In 2017 Manuel Olivares left his post as administrator of the mine, due to the aggravation of the symptoms of silicosis, contracted during the years of work in the mines. Currently, Joaquín Jiménez is responsible for the new one level under construction, called "Coni-Coni".

The geology of this district is composed of stratified volcanoclastic and volcanic rocks of andesitic composition and is part of Ovalle's Formation. There are numerous dykes with a diacitic and ritolytic composition. In this area, andesite or porphyric andesite predominates, which could be affected by hydrothermal alterations. In the district of Chancón there are mainly geological structures with NS, NE-SW and NW-SE prevalent direction that correspond to the veins containing economically viable mineralization. The mineralization in the veins is characterized by gold, silver, copper, lead and zinc sulfides and also includes a mixed zone of oxidized copper and sulfides (chrysocolla, atacamite, galena, sphalerite) and occasionally hematite, with gangue composed mainly of silica and clays. The vein exploited in this mine is called "Anita".

A photograph of the blasted rock is shown in Figure I.4. As it can be seen, the rock is highly mineralized with quartz, pyrite and chalcopyrite.

Figure I.4 - Mineralized blasted rock

There are numerous tributaries in the area that converge forming two streams: the “Anita” stream and “La Mina” stream, which in turn converge into a stream that feeds the “La Cadena” stream, natural limit between the municipality of Rancagua and that of Graneros. This water resource is extracted from natural wells, and its drainage is done by means of open channels and by pipes. The mine has numerous water infiltrations in all sectors, one of them being totally submerged for a portion.

The mine is developed in three main sectors, and each sector has been exploited because of the high-grade ore.

- Sector E-W: Dip-direction: E-W and Dip 70 N;
- Central Sector: First order structure E-W to N-W, presence of pyrite, chalcopyrite, gold, hematite and calcite. The ore grade exploited in the area ranges from 4 to 6 g/t;
- Sector N50-70E: Has gold grades that fluctuate between 2 to 14 g/t.

In this mine, as typical of the Small-Scale Mining sector, the methods of exploitation are various since the knowledge of the disposition of the veins and the technology for their location are reduced. The level that is currently under exploitation produces monthly about 500 t of blasted rock with a gold content ranging from 2 to 7 g/t and copper sulphate as a by-product with a copper concentration of about 0.7%. In some sectors of the mine, the vein is thin and takes almost vertical orientation. The method employed involves the construction of ramps (called "*clavos*", i.e. nails) rather narrow and sloping. A photograph taken inside one of them, is shown in Figure I.5, and allows to observe how they are narrow and sloping.

Figure I.5 – Photograph taken inside one "clavo"

The ore extracted is sold to various plants depending on the concentration of gold extracted during the month. It is sold directly, thus limiting the purchase by the companies that carry out the treatment at a price fixed according to the concentrations reached. This does not allow to derive the economic profits that the mine could have due to the presence of a gold, silver and copper and zinc treatment plant. A photo of a heap of blasted rock extracted from the mine is shown in Figure I.6. As it can be observed, there are many large blocks.

It should be noted that the payment for the ore takes into account only the gold, copper or silver grade and does not take into account the size of the blasted rock. Being the blast performed in an artisanal manner, without a drilling scheme, a blasting pattern and without a detonation sequence, the grain size distribution of blasted rock is extremely wide, ranging from blocks to pulverized rock. The payment of the ore extracted also on the basis of the regularity and the correct size of the blasted rock, which would allow a significant monetary saving during crushing and processing phases, would encourage the exchange towards the use of improved techniques.

Figure I.6 - Heap of blasted rock extracted from the mine

The mine is equipped from a logistic point of view with precast structures placed outside the entrance of the mine. Rooms that are used as kitchen and dining room, bathrooms, bedrooms and office are in the main container. A smaller container is used as a locker room. Many sheds are used for parking vehicles and cars and for maintenance. These spaces are shown in the following pair of figures: an aerial photo and a map, shown in Figure I.7. The elements according to the following description are numbered in the same figure:

1. Dirt road for internal transit to the mining sector of Chancón;
2. Truck loading area;
3. Access road to the mine;
4. Road inside the property;
5. Shed for parking and repair of vehicles and machineries;
6. Shed for parking vehicles and machineries;
7. Container for services (kitchen and dining room, pantry, bathrooms, bedrooms and an office);
8. Locker room container;
9. Access to the mine;
10. Hill;
11. Fuel tank;
12. Slope made of compacted barren rock;
13. Heaps of barren rock;
14. Metal ramp for the storage of the mineralized rock.

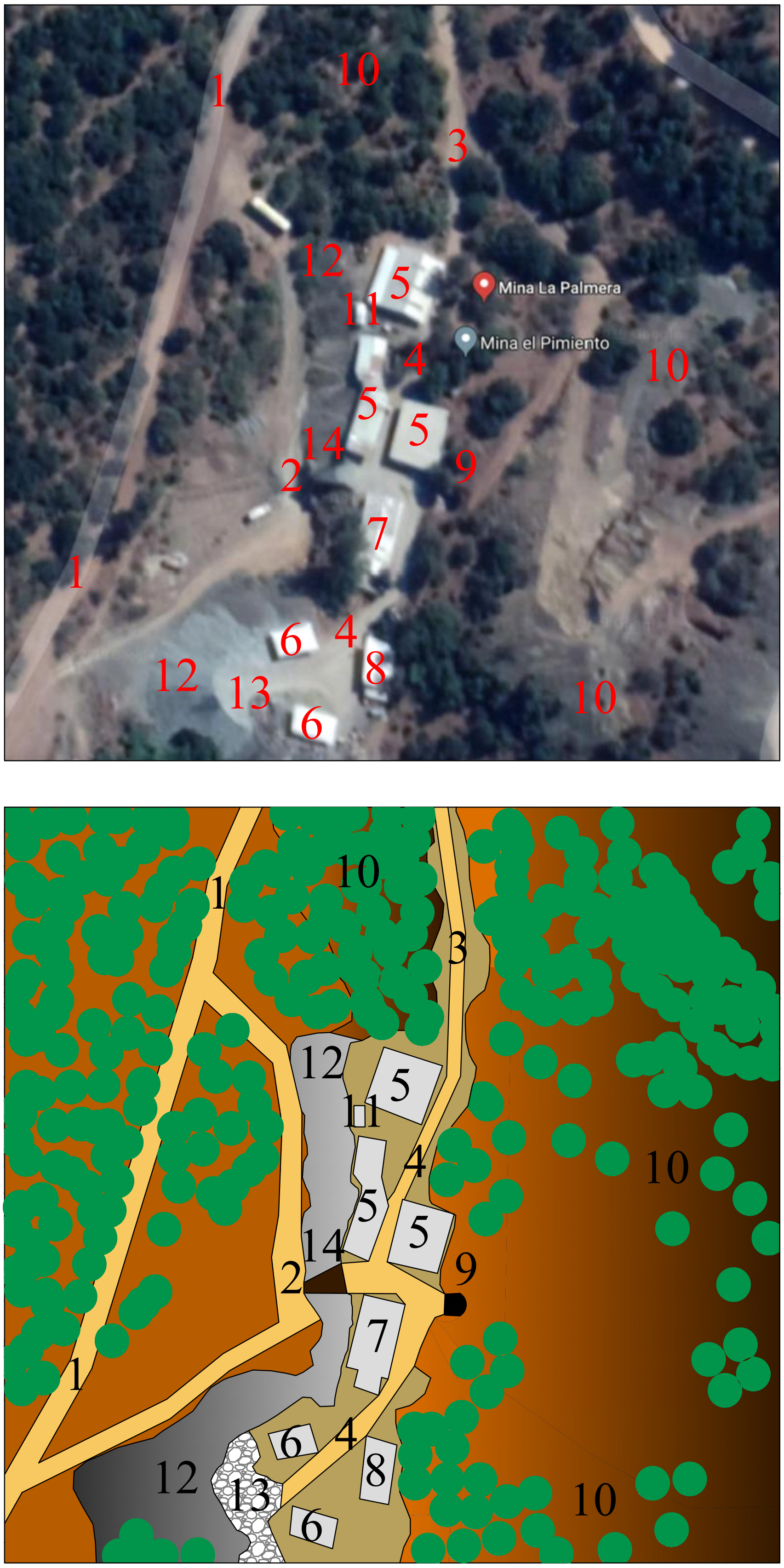

Figure I.7 - Aerial photo and map of the space outside the mine. The satellite image above is taken from Google Maps

I.4 - Structure of the improvement project

The first phase (Phase A) of the project has been performed directly on the work site. During the nearly 7 months spent in Chile, surveys at the artisanal mine were carried out.

Numerous factors were observed during the inspections:

- Observation of the general context:
    - Cultural aspects:
        - Educational level;
        - Local ideological and cultural aspects;
        - Chilean rural society;
        - Local traditions and beliefs;
        - History;
    - Social aspects:
        - Relation between the miners;
        - Relation between the miners and the managers;
        - Relation with parties outside the mining company (others mining companies, providers, clients, neighbouring populations);
    - Economic aspects:
        - General economic context of the country, region, village, mining sector;
        - Economic relationships between the subjects involved in the sector;
        - Cash flow within the company;
        - Invested capital and capital available for investments;
        - Style and quality of life;
        - Worker's salary;
- Observation of the logistic equipment:
    - Structures and installations:
        - Inside the mine;
        - Outside the mine;
- Observation of the technologies used:
    - Machinery:
        - Drills;
        - Compressors;
        - Pumps;
        - LHD shovels;
        - Pipes;
    - Explosives and Triggering systems:
        - Booster explosive;
        - Column charge explosive;
        - Detonators and fuses;

- Observation of the techniques, methodologies and work safety aspects:

    - Auxiliary operations:
        - Ore prospecting;
        - Bureaucratic management;
        - Maintenance;
        - General management of the mine;
        - Mining ventilation and dewatering;
        - Services lines;
        - Scaling;
        - Management of the explosives;

    - Drill and Blast:
        - Drilling schemes;
        - Charging of explosives;
        - Transport of blasted rock.

This last observation phase involved the measurement of relevant quantities. The expected results from this phase were:

1. Observations and quantification of the problems;
2. Definition of a hierarchy of problems;
3. Evaluation of possible solutions and practical application methods.

The most important aspects that must absolutely be improved urgently concern work safety, especially with regard to the prevention of rock collapses from the roof and walls of the tunnels and the improvement of safety in the use and handling of explosives. Photos that show a collapse occurred in the tunnel that caused various damages are shown in Figure I.8 and Figure I.9. Photos showing the absolute lack of safety in the use of explosives are shown in Figure I.10 and Figure I.11.

Figure I.8 – Photo of a collapse occurred in a tunnel (1)

Figure I.9 - Photo of a collapse occurred in a tunnel (2)

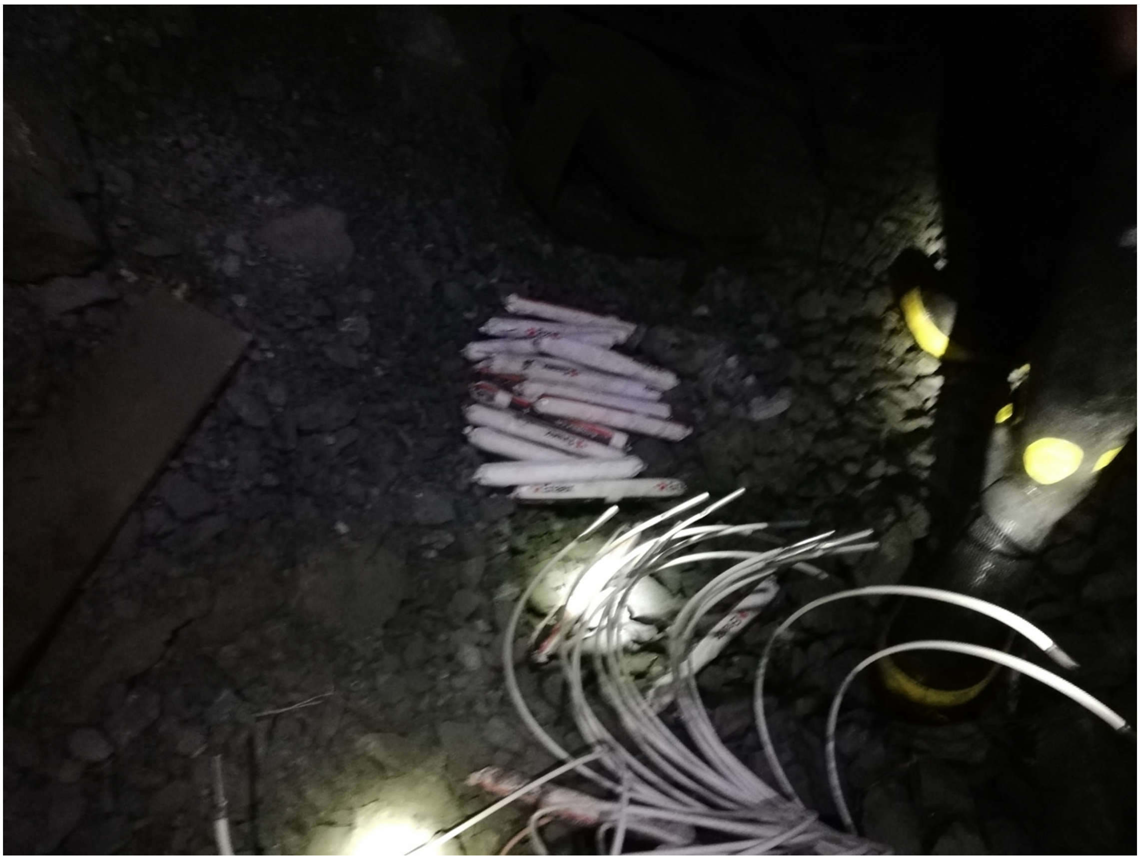

Figure I.10 – Fuses, detonators and booster cartridges before loading into the hole

Figure I.11 - Loading of primers

The second phase of implementation of the project (phase B) concerned the design and planning of the improvement intervention. This phase has been subdivided into 3 distinct sub-phases, so as not to include changes that are excessively abrupt and therefore impossible to put into practice practically:

- Phase 1:

    - Excavated sections with defined shape and dimensions;
    - Project of the drilling scheme according to a scientifically proven methodology;
    - Implementation of the non-electric ignition system;

- Phase 2:

    - Improvement of the shape of the excavation section;
    - Decoupled explosive for the contour holes (walls, roof) using a different type of explosive;

- Phase 3:

    - Doubling of the design pull;
    - Drilling performed using a Jumbo;
    - Use of more efficient explosives;
    - Increased production.

During the implementation of these phases, checks must be carried out at all stages. The project can be modified and improved during the implementation, generating a dynamic improvement process.

- Phase 4:

    - Qualitative and quantitative evaluation of the results obtained:
        - From a technical point of view;
        - From an economic point of view;

- Phase 5:

    - Making miners capable of managing the new working methodology and technology independently.

The phase B, due to temporal and logistical reasons, was not completed entirely. The work presented in the thesis refers only to the proposition of improvement solutions for the artisanal mine (the first 3 phases).

The expected results from this phase are:

1. Application of more reliable work system;
2. Improvement of working and life condition of workers;
3. Improved productivity;
4. Increase of the mining revenues and wages for workers;
5. Increased work safety.

The third phase of the project (phase C) should be implemented when the second phase is already completed, and the miners are able to teach and disseminate information to other mining companies in the mining sector.

This phase consists of the application of the improvement project to all the mines of the mining sector. This phase could also be divided into three sub-phases:

- Phase 1: Demonstration of the results obtained and dissemination of the information;
- Phase 2: Application of the improvement project to all the mines in the sector;
- Phase 3: Evaluation of the results obtained in the overall mining sector.

Each improvement project must be modified and adapted to the conditions of each specific mine. Checks must be carried out constantly.

The results that are expected from the implementation of this phase are the following:

1. Establishment of a collaborative relationship between the various mining companies of the sector, with the suppliers and with the state company that buys the ore;
2. Improvement of the economic conditions and quality of life of the whole sector;
3. Technological and socio-cultural development;
4. Creation of new infrastructures.

The conceptual map illustrating the various phases, the description of the phases and how they are put into practice and the expected results is shown in Figure I.12.

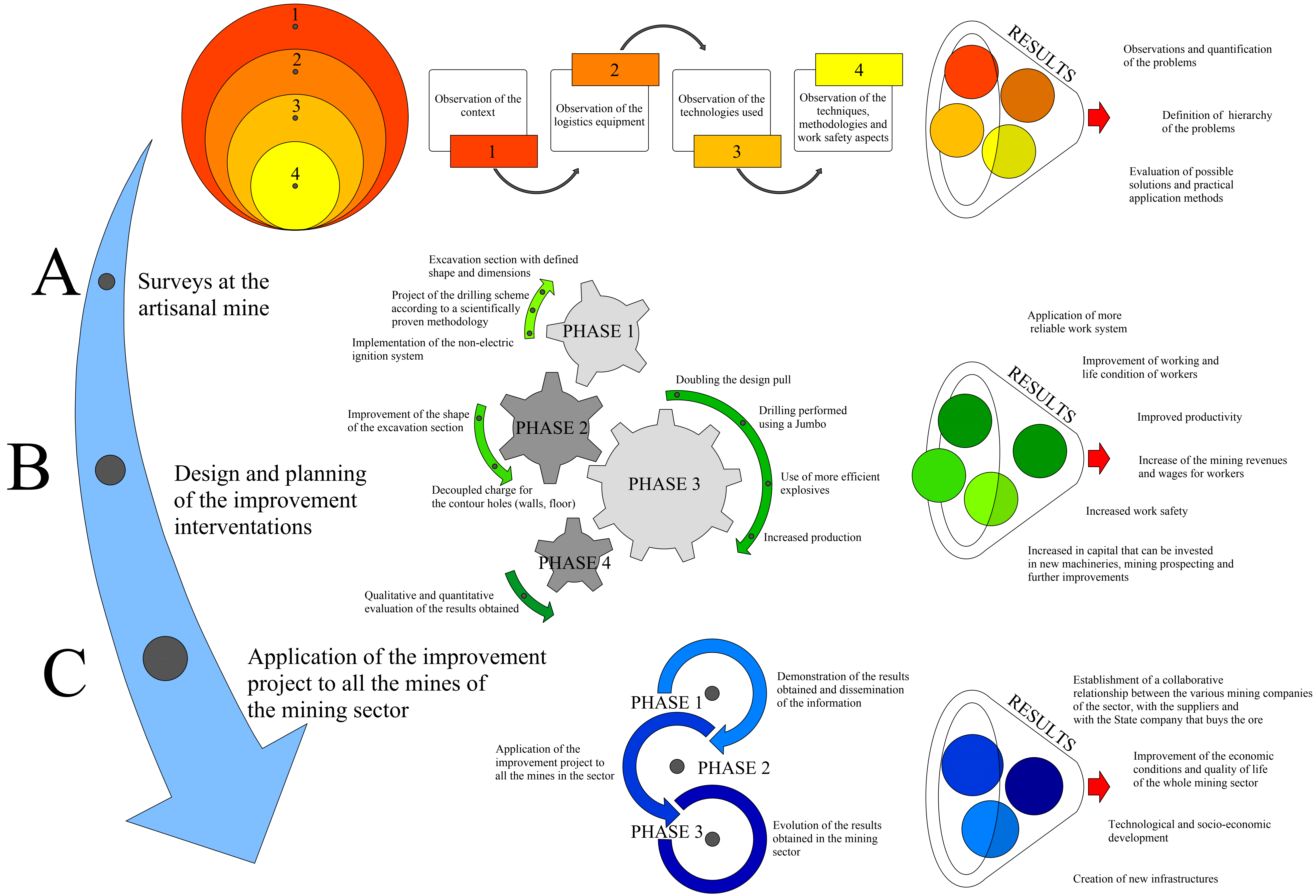

Figure I.12 -Concept map on the structure, the arguments and the results expected from the thesis

1 - DESCRIPTION OF THE CURRENT SITUATION IN THE ARTISANAL MINE

1.1 - Procedure used to collect information and data

This chapter is devoted to the explanation of the working methods adopted in the artisanal underground mine in Chile.

During the period spent over there, geometric surveys were carried out using simple instruments. In Figure 1.13, the operation of measuring the length of a blast-hole by means of a measuring tape is shown. In addition, technical information was obtained regarding the equipment and materials used, as well as measuring the times needed for the various work phases.

Figure 1.13 - Measuring the length of a blast-hole with a measuring tape

Interviews were also conducted with the personnel involved in the work operations at the various hierarchical levels within the mining company. These interviews were carried out with the aim of investigating and obtaining both technical and socio-cultural information, in such a way as to understand a context of work so different from that of the developed countries. Particular attention was also paid to work safety: the procedures adopted in the mine were observed, sometimes highlighting many shortcomings.

It should however be specified that in the artisanal mine there is no type of programming. The work is conducted according to the rules of experience and without following any type of project, especially as regards the excavation phase with explosives. There is no global planning of work, nor at the level of the single work phase. This has made data collection work difficult.

1.2 - Description of the working methodology in the mine

The search for the ore to be extracted is carried out not by surveys but by making production blast with prospecting purposes. This aspect is undoubtedly inefficient and wasteful. The research focuses on points of the mine where the presence of the mineral is highlighted by signals that the miners have been able to learn and interpret thanks to their experience (for example by observing the colour of the water coming out from the holes during drilling, or by the colour of the rock or the oxidation layer covering it). It should be noted that, however, nothing is supported by scientific basis or standardization, so this process is absolutely inefficient and unreliable.

Moreover, it should also be noticed that all artisanal mining companies work in the same way, according to this ancient and inefficient practice. Information and experience are handed down from generation to generation with pride and dedication. This helps to create a reluctant environment for change, information and progress. Very often the search for ore veins is concentrated in areas of the mine that have been already exploited. This produces tunnels having a variable and irregular geometry (even meters) with obvious problems related to ventilation and constant dangers of collapse, sometimes of considerable size. Figure 1.14 shows the miners inside an inclined shaft excavated for prospecting purposes.

Figure 1.14 - Drilling phase inside an inclined shaft to perform production blast with prospecting purposes

The working day starts at about 8 am. Some workers live in the lodgings at the base camp of the mine (equipped containers), others reach the workplace every day from their home located in the city of Rancagua or in the town of Chancòn by car or motorbike.

Arrived at the workplace, the workers go to a container housed in the outer square of the mine where they wear personal protective equipment consisting of overalls, gloves, boots, dust mask with filters, helmet equipped with earmuffs and lantern.

The first phase of the working day consists in the control and maintenance of the equipment that will be used during the shift. The drills, the compressors, the pumps, the LHD shovels are checked, and the drilling bits are sharpened when necessary (Figure 1.15). The equipment used for maintenance is housed in a shed near the entrance of the mine. However, in the mine there are no qualified technicians involved in repairs and maintenance. For this reason, readjustments are usually performed by the same miners when necessary: repairs are hand-crafted and of poor quality. Only in case of serious problems that compromise the progress of work, the company resorts to professional technicians for targeted repairs. This situation has repercussions not only on the condition of the machines, but also on the time taken by the miners for long and ineffective repairs. Often the machines are in a very poor state of preservation, with strong limitations on productivity and high frequency of breakages (for example all LHD shovels are affected by serious losses of lubricating oil). The materials used in the repairs are often inadequate and of poor quality, partially limiting the damage for a short time (for example, the water pipes are repaired with rubber strips made with air chambers of exhausted tires).

Figure 1.15 - Miner during the sharpening phase of a tool

In the mine, the progress of the work takes place in two different excavation faces. The miners work in pairs: a miner handles the servo-sustained pneumatic drills while the other helps him in the various auxiliary operations. In total, therefore, 4 miners work inside the mine.

The first operation carried out by the miners consists in an inspection of the tunnels and excavation faces inside the mine. It is quite frequent that, during the blast of the previous shift or during the absence of workers, pieces of rock detach from the roof of tunnels already excavated. Often such collapses cause truncations to the water and compressed air supply pipes, which must be repaired. The inspection is carried out while the other workers carry out the maintenance and control work on the machines and equipment. In Figure 1.16, the effects of a collapse are visible: fragments of rock detached from the roof of the tunnel have been found in the middle of the passage during the morning inspection.

Figure 1.16 - Photograph taken during an inspection. Fragments of rock fallen from the roof of the tunnel are visible

Those collapses are sometimes of considerable size. The fact that miners conduct daily inspections shows that they are anything but infrequent. The reason why they occur is due to the incorrect execution of the blasts (lack of timing and contour blast-holes), to the absence of supports (shotcrete, electrically welded mesh, rock bolts, etc.), to the lack of geomechanical surveys and monitoring. The high risk of collapse is unacceptable, as it puts the lives of miners every day at risk. This is the aspect where it’s necessary to intervene with urgency, in order to ensure sufficient conditions of safety.

Once the inspection has took place, the following phase consists in removing the blasted rock obtained from the blast carried out during the last shift of the previous day (mucking). The operation is performed by means of LHD shovels. This phase is quite long, lasting about 2 hours. Overall, it takes about 30 minutes for each of the 4 loading and unloading cycles necessary to take the blasted rock out of the mine. Attention must be paid during this operation: it can often happen that, due to the poor execution of the blast, unexploded primers (cartridge + detonator) are found. Great caution has to be applied not only during mucking, but also during the scaling phases. Miners should be advised not to touch these cartridges and not to remove the detonator with the hands, being not allowed to do that, even though it occurs regularly. The ore is stored in a metal ramp (Figure 1.17) until it is full and then is loaded into trucks which haul it to the processing plant. The barren rock is stored in a dedicated area (external square of the mine) in heaps.

Figure 1.17 - Photo of the access to the metal ramp where the ore is stored

Once the blasted rock has been removed, the next phase consists in water pumping (it is often gathered at the excavation face). The presence of water is almost exclusively due to infiltrations through permeable layers or to an inflow of water along fractures, dislocations, cavities, having communication with the outside. The water inflows in the mine are directly related to the progress of underground works. The removal of water takes place thanks to immersion pumps connected to the pipes which are fixed to the tunnel walls.

The pump used for the drainage of the excavation face is shown in the Figure 1.18.

Figure 1.18 - Pump used for the drainage

Once this operation has been done, the miners carry out the scaling.

The scaling, as shown in Figure 1.19, is done by hand, using a steel bar to remove the blocks of rock that are visibly unstable. This operation lasts approximately 30 minutes. However, it should be noted that the result of scaling is not good at all. Since the rock around the tunnel is very damaged due to the poor quality of the blast, not all blocks are removed and especially the biggest are not removed, remaining dangerously unstable on the roof of the tunnel.

Figure 1.19 – Scaling phase

The following operation is the preparation of the excavation face: it consists in removing the fragments of rock at the base of the face that have not been removed by the LHD shovel and in the regularization of the work surface in order to prop the servo-support of the perforator. Often some fragments of rock (especially in the lower part of the section, where the explosive works in the worst conditions due to the presence of water and for geometric reasons, related to the presence of many solid corners) have only been fractured and must be removed manually. This operation is carried out, as visible in Figure 1.20, thanks to a manual shovel and pickaxe for about an 1.5 hours (sometimes more). It is very heavy and tiring operation, that could be cut out by correctly using the timing and by removing the blasted rock with greater caution by the LHD shovel.

Figure 1.20 - Removal of rock fragments with shovel and pickaxe near the excavation face

Drilling is performed with a pneumatic servo-supported rotary percussion drill. The drilling diameter is 1.5” (38.1 mm) and is executed with linear or crosswise cutting bits. The drill bars are steel-made, have a length of 1.80 m and are used to drill holes up to a length of 1.50 m. The drilling phase is shown in Figure 1.21 and Figure 1.22. The photograph of a hole just drilled is shown in Figure 1.23.

Figure 1.21 – Perforation with pneumatic servo-supported drill

Figure 1.22 - Drilling phase in an inclined shaft

There is no predetermined drilling scheme, as there is not a blast design. The operator performs drilling according to his own experience, according to a rough and irregular pattern with a spacing of about 1 m. Blast-holes are not classified in functional groups, and the concept of cut is almost unknown. The holes are not drilled perpendicular to the face, but with a certain inclination, to make easier the practical execution of the hole. This creates a random and often inefficient drilling pattern. Although the drilling operation is wet, the working environment, due to the absence of a forced ventilation system, is very dusty, as visible in Figure 1.24.

Figure 1.23 - Hole just drilled (diameter 1" 1/2)

Figure 1.24 - Photo taken during the drilling phase. As visible, the air is very dusty

The drilling phase is performed by two miners: a qualified (experienced) driller and an assistant.

The operation begins with the connection of the drilling machine to the compressed air and water pipes. The point where the hole is to be made is chosen, and the bit is positioned on that point. The assistant holds the bar in place by holding it tightly between the hands until a section of hole is drilled with a depth such that the bit can no longer slide on the rock surface. This operation is rather dangerous and should be avoided. Once the bit has penetrated the rock to a depth of about 5 cm, the assistant leaves the grip of the bar and the driller continues the execution of the hole along its whole length. The weight of the drill is supported by the servo-support, so that the drill must only drive the machine by holding the hands on the support. The hole is drilled with the maximum pull possible in those conditions.

Often, during the execution of the hole, it may happen that the base of the servo-support slides due to the slippery ground (humid because of the drilling water, and irregular). The assistant has the task of blocking the servo support with the foot so that it does not move, as shown in Figure 1.25.

Figure 1.25 - Perforation assistant blocks the base of the servo-support with one foot

Some operators do not wear personal respiratory protective devices despite they have been supplied to them. This is because they have not been duly informed about the risk of contracting silicosis and the consequences of this behaviour, as well as a lack of supervision by the staff.

The drilling bits are re-sharpened only when they are so worn out that they cannot penetrate the rock. Greater attention to bits maintenance could help reducing dust and increasing productivity.

Problems often occur during the drilling phase, due to pipe, bar and bits breakage. These drawbacks are mainly due to lack of organization, maintenance and order on the excavation site. These events, as well as slowing production are often dangerous for operators. Once the drilling phase has been completed, the perforator is detached from the water and compressed air pipes and is brought to a location distant from the excavation face. The excavation face after the drilling phase is shown in Figure 1.26.

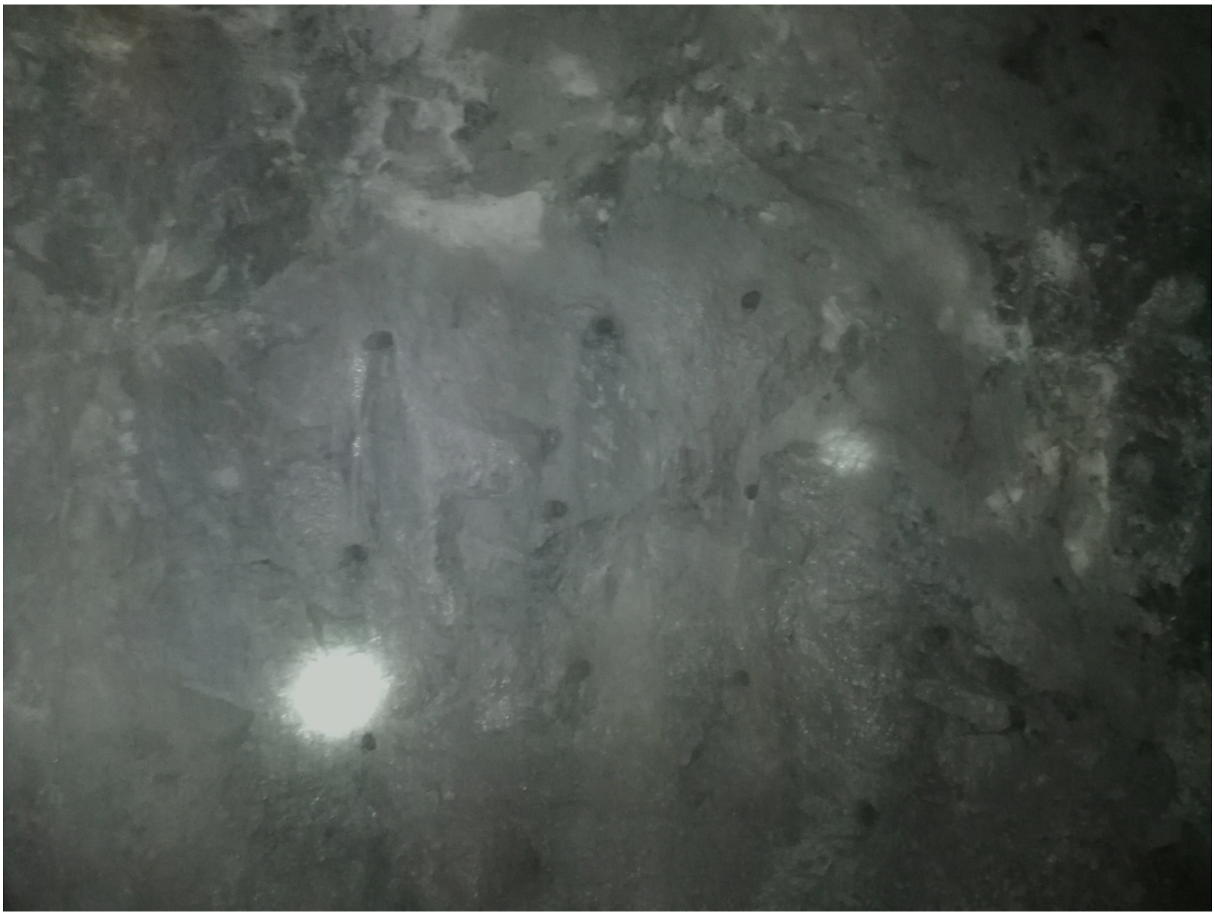

Figure 1.26 - Excavation face at the end of the drilling phase

The following phase consists in charging the explosive into the holes, using a cartridge of Emultex CN® as bottom charge and bulk ANFO premium® (pneumatically loaded) as column charge. Ordinary detonators (Riocap®) are used, triggered by safety fuse (Riofuse®). The fuse is previously cut into pieces with length 2.50 m, greater than the minimum limit imposed by Chilean regulations and enough to allow miners to safe escape (sometimes the miners have to go down ladders or walk climbing traits with ropes along routes that are difficult to access, especially for the older workers). The photo of a charged hole is shown in Figure 1.27.

Figure 1.27 - Photo of a charged hole: the pink powder is bulk ANFO, it's visible also the slow burning fuse coming out of the hole

The explosives are taken from the storage room (an underground room placed inside the mine and closed by a door) and brought to the excavation face by means of a backpack. The transport of detonators and booster cartridges takes place in the same backpack: this is very dangerous and should be absolutely avoided. Detonators and explosive cartridges must be transported and stored separately. ANFO is stored and transported in 25 kg bags. In the meantime, the holes are cleaned by compressed air. At this point the detonators (already connected to the fuse) are inserted inside the boosters. The picture showing the insertion of an ordinary detonator into a booster is shown in Figure 1.28.

Figure 1.28 - photo of a miner inserting a detonator inside a booster cartridge

The primers are inserted into the holes by means of a plastic tube, letting the fuse come out. While charging the primer, the miners wrap around their neck the cartridges with their detonators and fuses, as shown in Figure 1.29: this operation is very dangerous and must be avoided. A meeting with the workers has to be urgently planned in order to explain and implement the safety rules for handling explosives. ANFO is pneumatically loaded, as visible in Figure 1.30. The operator charges the hole with the ANFO for a length of about 1 m and stops when the noise produced by the grains of explosives settling on the bottom becomes more low-pitched A portion of hole of about 0.5 m is left empty. The fuse extending from the hole is knotted to facilitate the ignition. In the lower part of the excavation section, in order to prevent contact between ANFO and water (since this explosive has a poor resistance to water) plastic tubes are placed, that are loaded pneumatically. In these holes, as visible in Figure 1.31, two detonators (and relative fuses) are inserted to guarantee the initiation. The excavation face at the end of the charging phase can be seen in Figure 1.32.

Figure 1.29 – The miner, with the fuses wrapped around the neck, during the insertion of the primer in the holes

Figure 1.30 - Pneumatic charging of the holes with bulk ANFO

Figure 1.31 - In the holes at the base, plastic pipes protect the ANFO from contact with water

Figure 1.32 - Photo of the excavation face before ignition of the fuses

Once the charging is complete, the materials and the equipment are removed from the face. The operators inside the mine are warned of the imminent initiation of the blast. The fuses are ignited by a gas torch or a lighter, as shown in Figure 1.33. The ignition is done according to a random order decided by the operator, to make the operation as easy as possible. In low light conditions and due to the presence of dust in the air, it is very easy to forget to ignite some fuses.

Figure 1.33 - Miner during the phase of ignition of the fuses

All operators have to go outside the mine. Since the blast does not take into account the timing, the fuses trigger the detonators without a pre-established sequence. This can be clearly understood by listening to the sounds produced by the detonation of the individual blast-holes. The bangs can be heard in a random sequence: sometimes interspersed with tens of seconds and sometimes in rapid succession.

The ventilation, necessary for the removal of gaseous products and dust produced by the blast, occurs during the lunch time and is natural. The duration of this phase is approximately 1 hour. The natural ventilation circuit is quite efficient, made by means of vertical shafts placed with an interspace of 90 m.

After the ventilation phase, the second shift of the day can begin, and then the steps of mucking, scaling, preparation of the excavation face, drilling, charging and blasting are repeated. The working day ends with the triggering of the second blast.

The following Table 1.2 shows a summary of the working day, describing the duration of each phase. The data were used to plot the chronogram shown in Figure 1.34.

Table 1.2 - Definition of the times of each work phase

Working phase:	Duration of the phase (h)	Start time	End time
Maintenance and control of equipment and machines	1.0	08:00	09:00
Mucking of the blasted rock	1.5	08:00	09:30
Pumping of the water	0.5	09:30	10:00
Scaling	0.5	10:00	10:30
Preparation of the excavation face	0.5	10:30	11:00
Drilling	1.5	11:00	12:30
Charging	0.5	12:30	13:00
Ventilation	1.0	13:00	14:00
Mucking of the blasted rock	1.5	14:00	15:30
Pumping of the water	0.5	15:30	16:00
Scaling	0.5	16:00	16:30
Preparation of the excavation face	0.5	16:30	17:00
Drilling	1.5	17:00	18:30
Charging	0.5	18:30	19:00

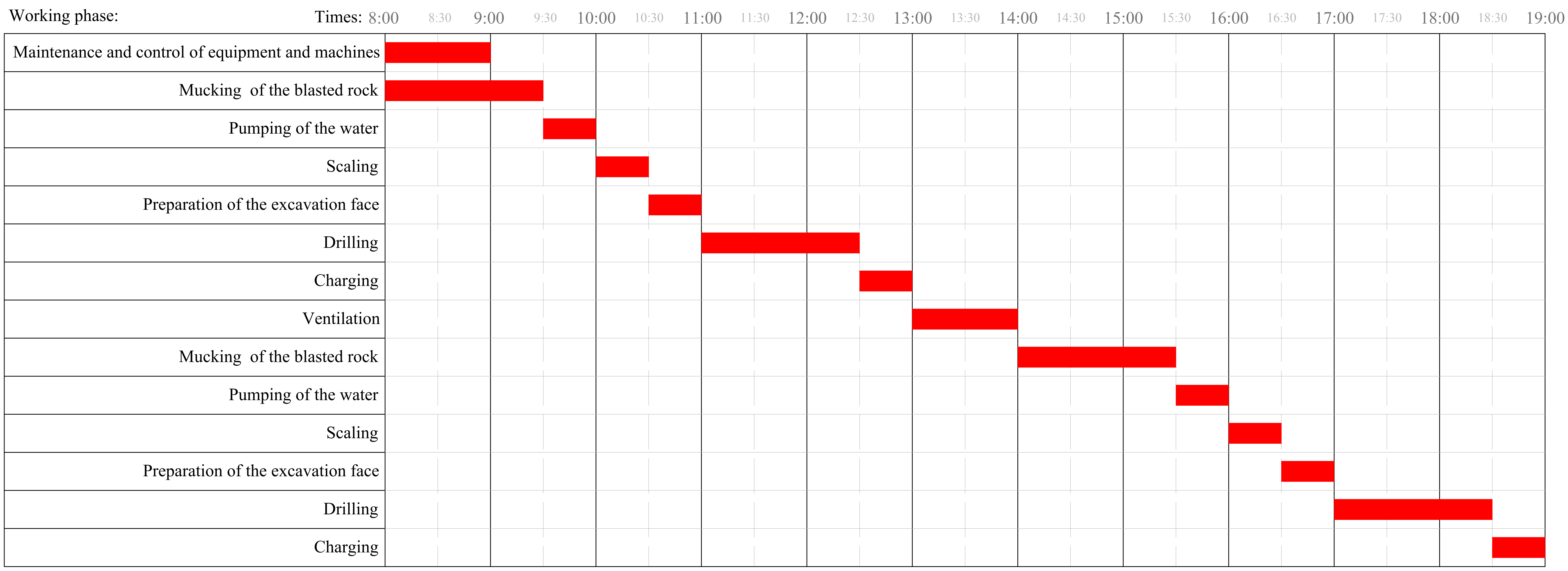

Figure 1.34 - Chronogram representing the temporal organization of a working day in the artisanal mine

In the summary described, 4 blasts are daily produced. However, this is just an example. The estimate of the amount of rock extracted, daily or weekly (working days are from Monday to Friday) is impossible, as there is no rigid schedule. Sometimes only two blasts are performed, sometimes none. The excavation section is absolutely variable and there is no default drilling scheme. For this reason, calculating the volumes of blasted rock is impossible.

Inside the mine there is a great confusion, due to the total lack of a schedule of work shifts and activities to be carried out; also, the control by the supervisor (mining engineer) is a drawback, as he essentially leaves total freedom and self-management to the miners, without placing objectives of production and safety. For example, only at the beginning of the work shift the miners decide what to do and where to run the blast, almost randomly. Furthermore, each blast is different compared to the others. The whole mine is very poorly managed.

The result of such behaviour is a disordered and unproductive work environment. Being the results of the blast very bad, each subsequent blast is an attempt to remedy the previously errors: regularizing or expanding the walls of the tunnels, removing pieces of rock that are unsafe or trying to more exploit the vein. The need of implementing radical changes is therefore very evident in order to increase both productivity and work safety.

1.3 - Analysis of the data collected in the surveys

This paragraph will show the data collected mainly in the course of the surveys carried out during the time spent in Chile. A production blast was observed in a recently opened tunnel in the mine. The data collected are exposed and critically commented in order to understand the most serious technical deficiencies and to evaluate the possible solutions that can be adopted in order to improve the technique of excavation.

The survey was conducted during the work phases, without interrupting, slowing down or influencing the miners. During the inspection, the significant lengths related to the drilling scheme and to the charging were measured using a tape measure (length of the hole, charged length, length of the fuses, etc.). The orientation of holes has been measured through the use of a compass. The times used for the execution of each operation have been measured with a stopwatch. During the various phases of the work, photographs were taken, and the most relevant were included in the report as documentation.

The excavation face was observed and detected before starting drilling. The excavation section has an extremely irregular contour, observable in Figure 1.35. The section measures approximately 235 cm in height and 230 cm in width.

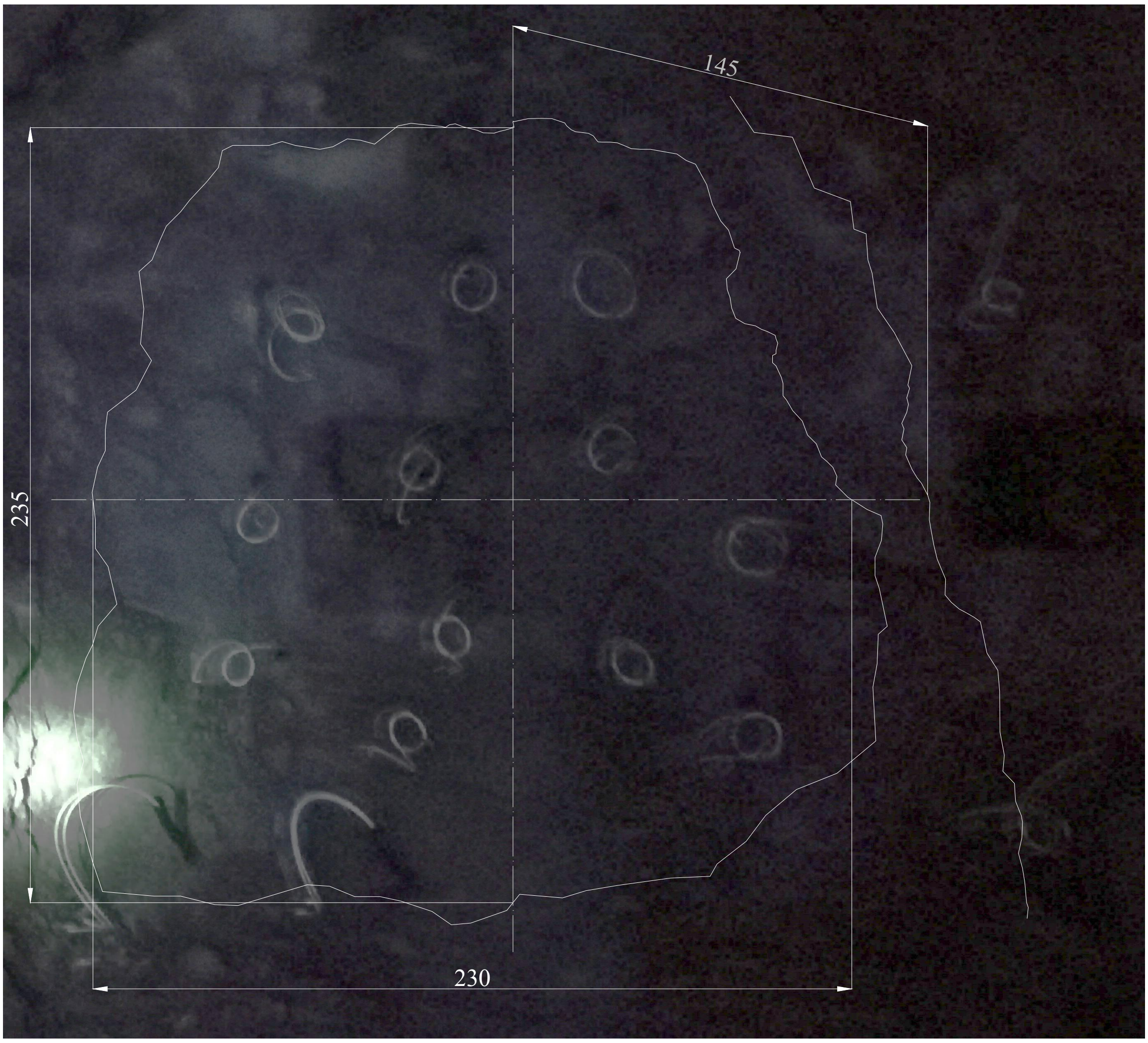

Figure 1.35 - Dimensional survey of the excavation face

Along the 4 sections (two verticals and two horizontal) the heights of the irregularities and the depths of the cavities that were found on the rock cavity were measured. The position of the relief sections on the excavation face is shown in Figure 1.36.

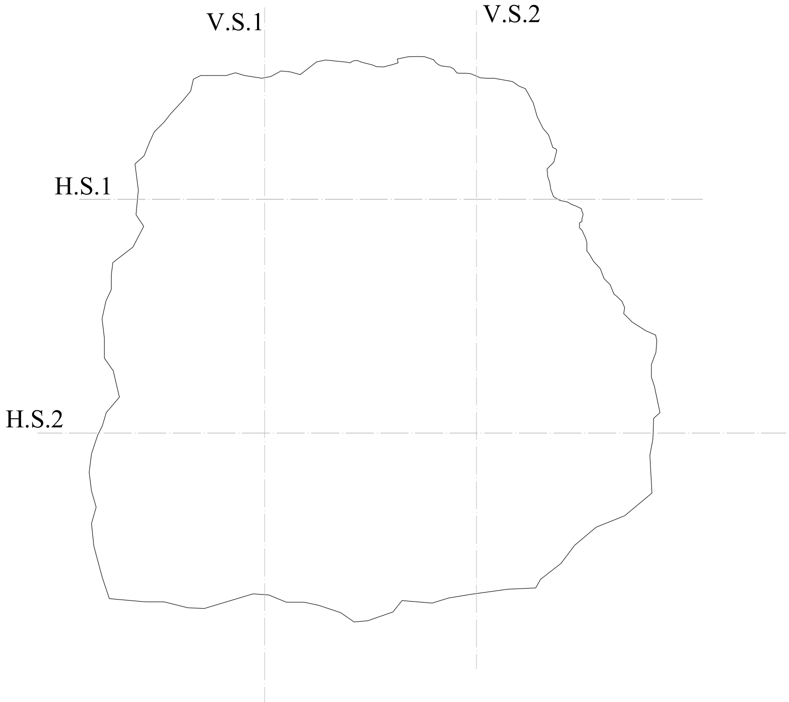

Figure 1.36 - Position of the relief sections on the excavation face

The data collected during the survey are shown in Table 1.3 (horizontal sections), and in Table 1.4 (vertical sections).

Table 1.3 - Data of the geometric surveys carried out on the horizontal sections

Horizontal Section 1		Horizontal Section 2	
Distance between reliefs (cm)	Height of the profile (cm)	Distance between reliefs (cm)	Height of the profile (cm)
18	-4	14	-2
12	0	45	0
15	1	13	1
10	-5	44	20
25	-2	10	10
19	-4	24	5
10	0	14	-14
11	10	20	-10
10	0	20	-4
17	-9	9	-6
13	-4	17	-1
22	-2	10	-3

Table 1.4 - Data of the geometric surveys carried out on the vertical sections

Vertical Section 1		Vertical Section 2	
Distance between reliefs (cm)	Height of the profile (cm)	Distance between reliefs (cm)	Height of the profile (cm)
8	6	7	1
5	0	24	0
10	-14	25	-10
17	-6	12	0
17	-5	14	12
30	-6	23	0
15	-8	20	-25
20	-2	17	-10
13	-4	32	-20
17	0	25	0
9	2	25	20
14	3		
49	20		

The graphic representation of the sections measured is shown in Figure 1.37 for the horizontal sections and Figure 1.38 for the vertical sections.

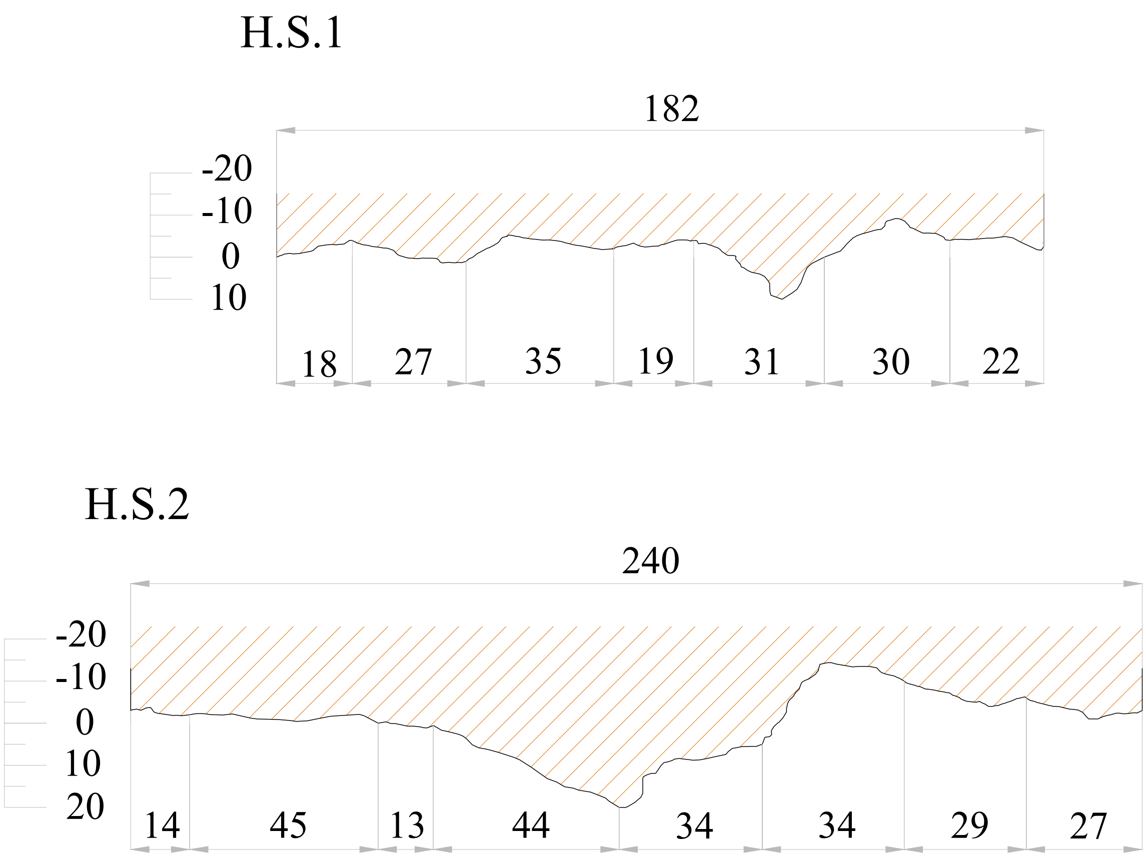

Figure 1.37 – Geometrical relief of the horizontal sections of the excavation face

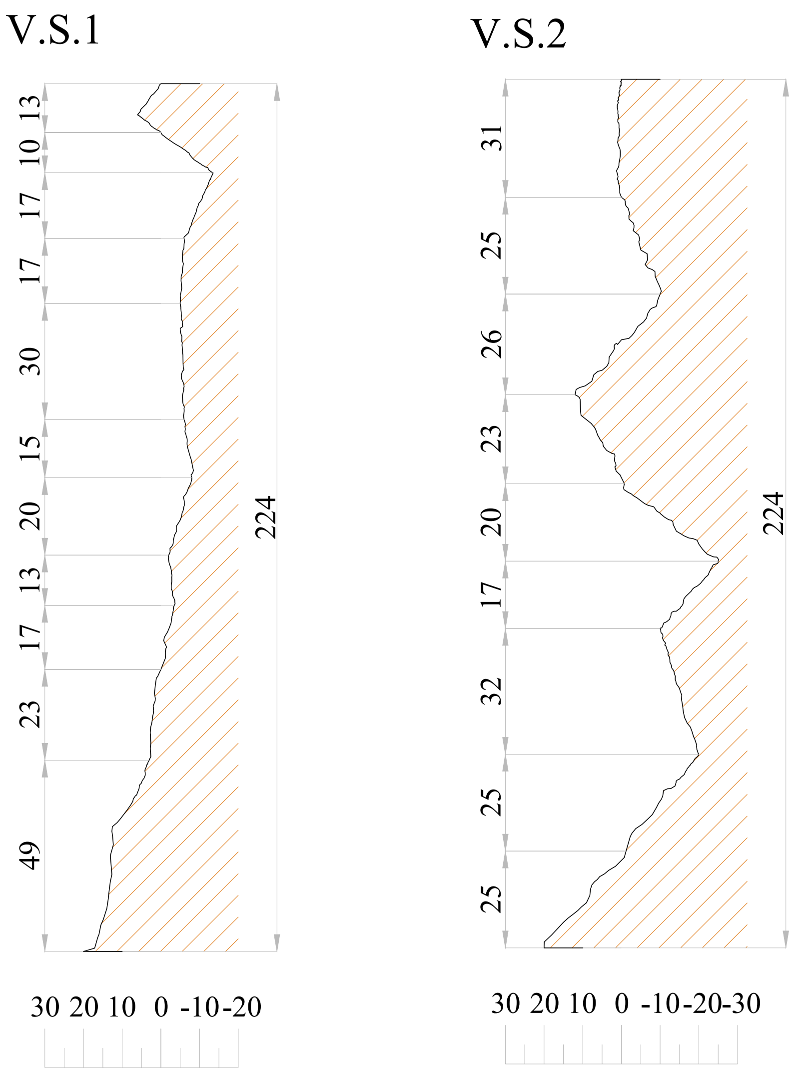

Figure 1.38 - Geometrical relief of the vertical sections of the excavation face

The relief showed that the excavation face was very irregular, with roughness of even 30 cm. This is certainly due to the ineffective and incorrect use of explosives: lack of a drilling scheme, lack of functional blast-hole groups (especially for the contour), and no timing, due to the inefficient initiation system made by safety fuse.

This is a problem not only because it makes drilling more difficult, but also the use of explosives, since the value of spacing and burden is difficult to quantify, and the amount of charge/unit volume cannot be evaluated correctly. Without a proper geometry of the excavation face, the irregularities will propagate, becoming ever larger and requiring corrective adjustments.

Looking at the relief of the vertical sections (Figure 1.38), it can be noticed that, generally, in the central part of the section the cavities are quite deep, while in the lower contour there are very evident protrusions. This is due, in addition to the incorrect drilling scheme (where the holes are more concentrated in the centre of the section), also to the conditions in which the explosive detonate: in the lower part, the ANFO is loaded into plastic tubes with a diameter of 1", a value much lower than the suggested critical diameter and less than the drilling diameter; in addition, in the lower part of the section, there is always the presence of water that, in any case, goes in contact with the bulked ANFO. The length of the holes in the lower part of the excavation section is smaller, due to the greater difficulty in drilling with the servo-support drilling machine in that position.

As for drilling, a total of 17 holes were made. 3 holes were made in the rocky wall placed at a distance of 145 cm with respect to the excavation face. These holes were made in order to enlarge the excavation section, to correct the previous blast. The representation of the position of the holes made on the excavation section is shown in Figure 1.39.

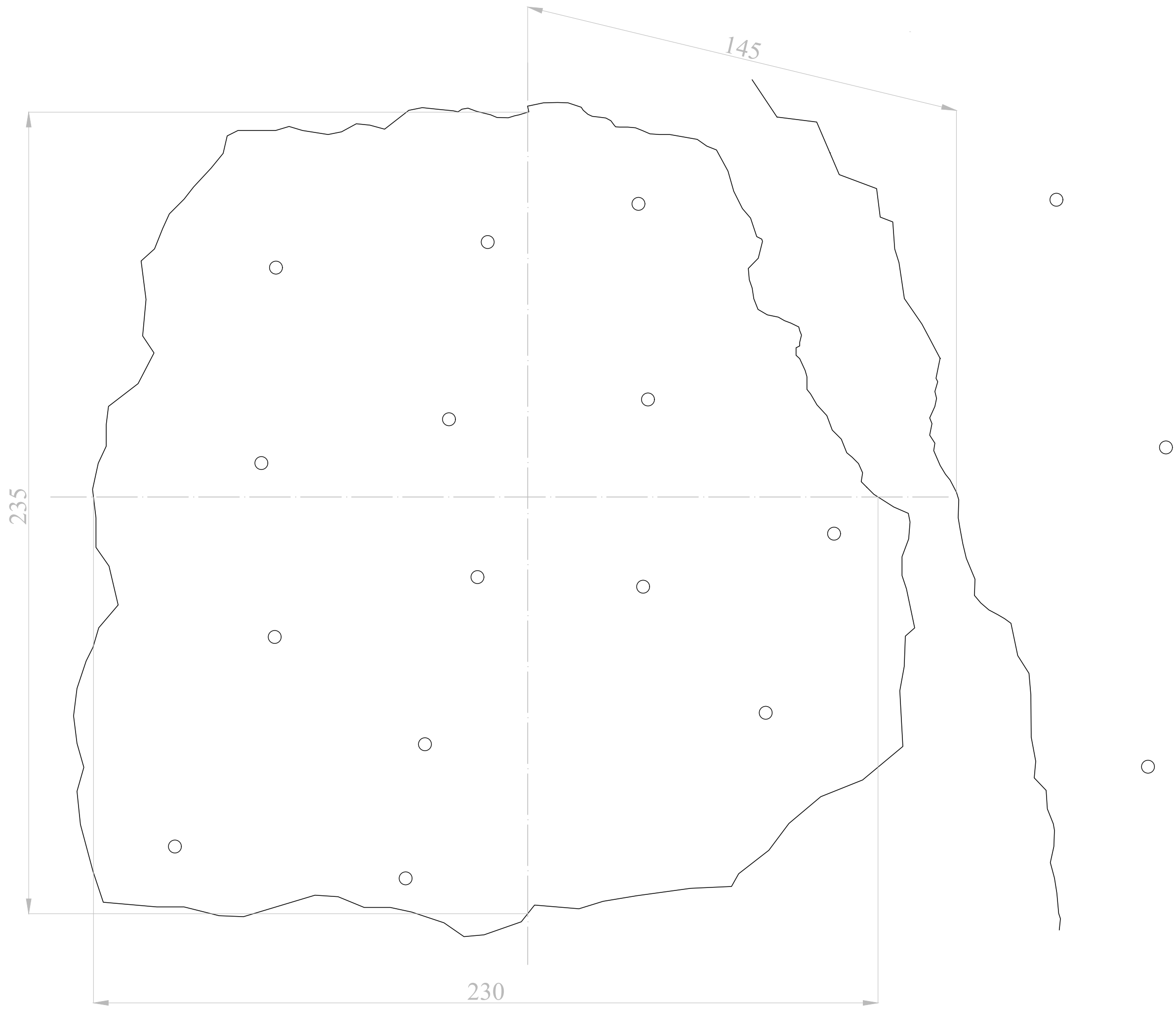

Figure 1.39 - Representation of the excavation section and blast-holes

The drilling length is approximately 130 cm. The operator never measures the length of the holes, he only looks at the penetration depth of the bar within the rock. As a result, each hole has a different length. The holes located at the bottom of the section have a shorter length, <90 cm. This is due to the greater difficulty in drilling those holes.

The drilling order is random and chosen directly by the operator. The number, position, inclination and length of the holes are established during the drilling phase by the miner. The image showing the drilling order is shown in Figure 1.40.

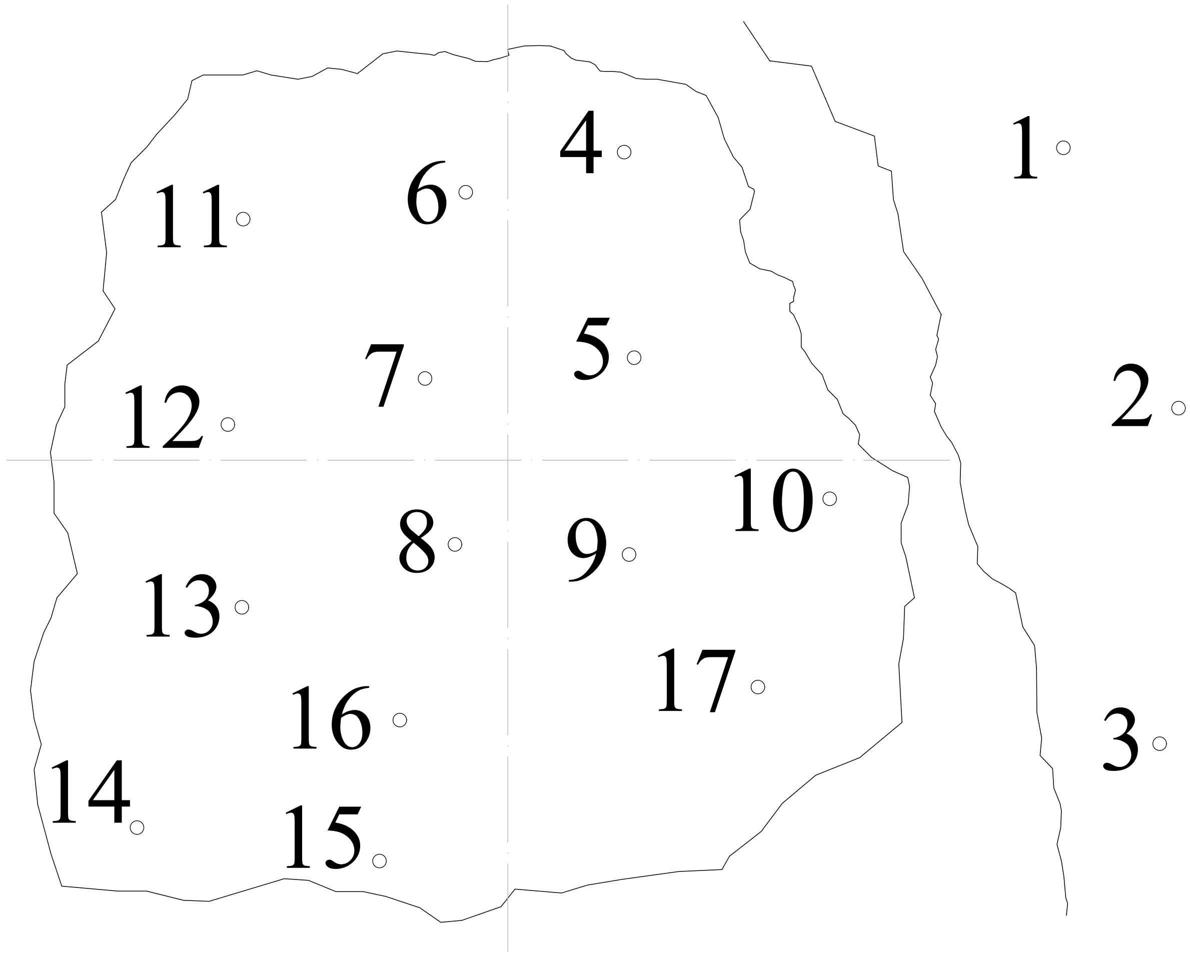

Figure 1.40 - Drilling order

There isn't a project of the drilling scheme to be followed during the work. Since there is no possibility of adopting a detonation timing, the cut is not noticeable. The holes are drilled according to the miner's experience. The distance between the holes is about 60 cm. The exact position of the hole is also chosen taking into consideration the greater ease in positioning the drilling machine and the bit on the rock surface so that it does not slip. The result is an irregular pattern, where some holes are closer and others too distant (Figure 1.39).

The orientation of the holes has been measured by means of a compass, measuring the angle with respect to the horizontal plane and the angle with respect to the vertical plane. The holes are not drilled all orthogonally to the excavation face, but with an inclination defined by the ease of drilling.

Observing the horizontal sections, shown in Figure 1.41, it can be noticed that all the holes have an inclination towards the right. This makes it clear how the resulting section will be slightly moved to the right. Observing the vertical sections, it can be noticed that the holes in the upper part of the section are inclined upwards, those in the centre are almost horizontal and those positioned in the lower part of the section are inclined towards the bottom. These inclinations are due to greater drilling convenience.

The absence of precision in drilling, in establishing the drilled length and the orientation of the holes, provides as a result of the blast a very irregular excavation face. Since there is no blast timing, it is impossible to establish a detonation order and therefore to predict the progression of the blast.

Table 1.5 shows the data measured during the drilling phase.

Table 1.5 -Data concerning drilling

Hole number by drilling order (-)	Length (cm)	Angle on the horizontal plane (°)	Angle on the vertical plane (°)
1	135	15	30
2	132	10	0
3	121	20	-10
4	124	30	5
5	120	25	10
6	118	20	30
7	119	30	20
8	117	20	0
9	112	10	0
10	127	35	-5
11	123	0	10
12	126	20	10
13	123	10	-5
14	82	10	-35
15	88	20	-40
16	108	-5	-10
17	114	15	-20

Figure 1.41 shows the projections of the blast-hole sections on the horizontal plane. This image allows to observe the inclination of the holes measured on the horizontal plane. Figure 1.42 shows the projections of the holes in the vertical plane.

In both figures, the cross-section shown at the top allows to locate the position of the sections and the drilling order.

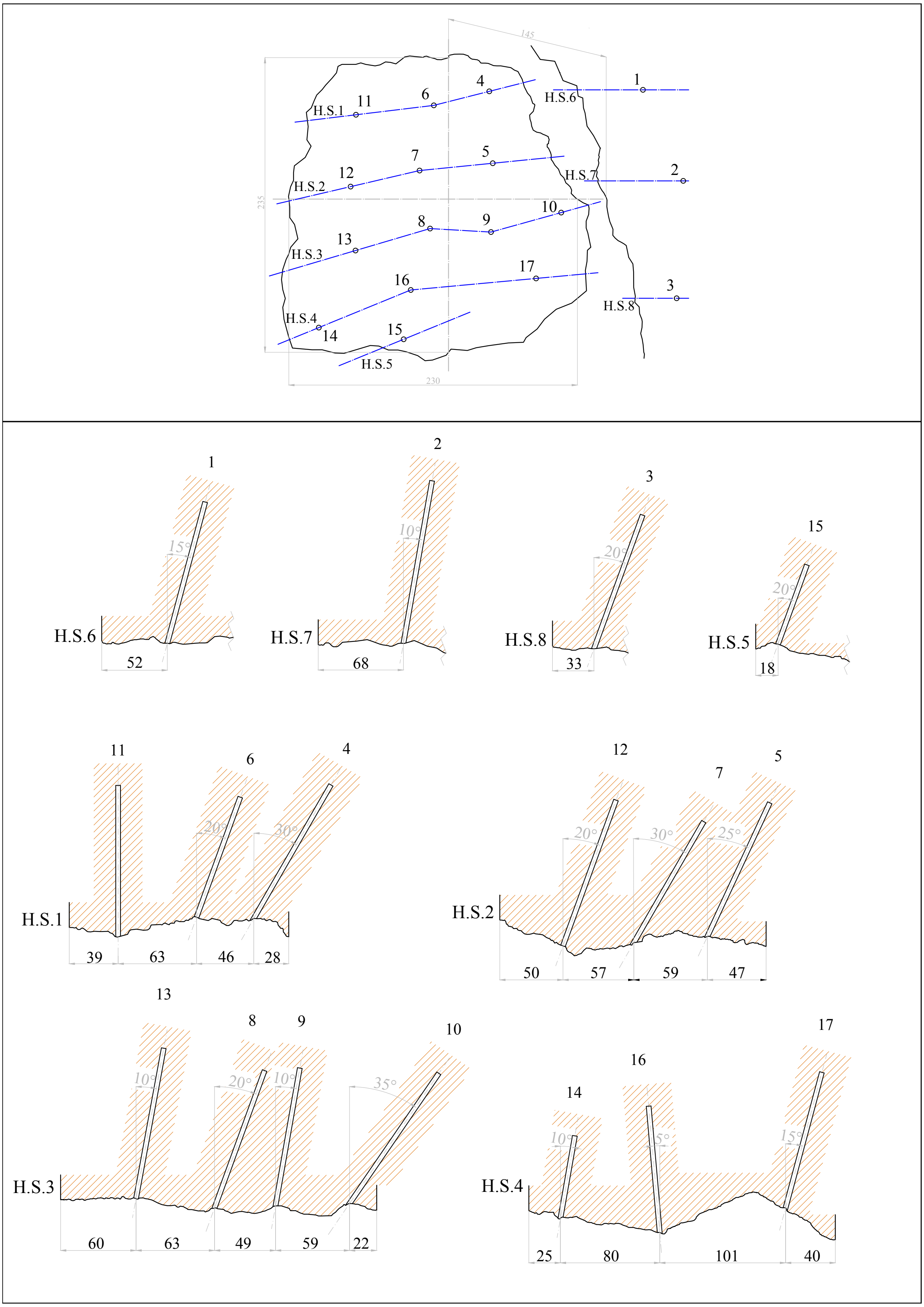

Figure 1.41 - Projection of the section of the holes on the horizontal plane

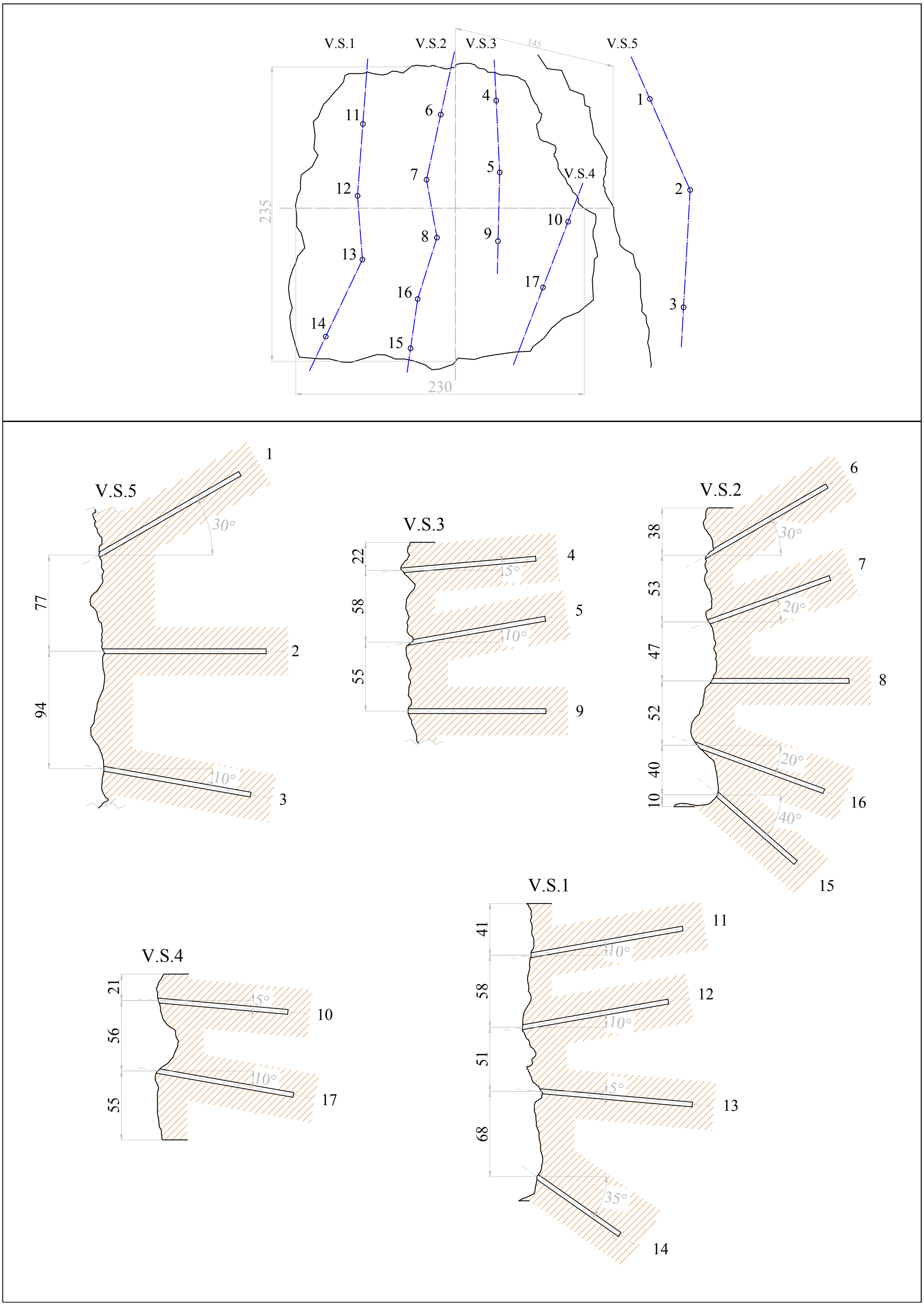

Figure 1.42 - Projection of the section of the holes on the vertical plane

The measurement of the drilling time allowed to observe that, on average, a single hole requires about 4.5 minutes. However, inconveniences (breakage of bars and tools, of the drilling machine or of the water or compressed air supply pipes) can very often cause delays and lengthening of the drilling times. A pause of about 1.5 minutes was recorded between the drilling of one hole and the next, due to positioning the bar to drill the next hole. The drilling speed is about 23 cm/minute, that is rather low, making it clear how the drilling phase limits the production and therefore needs to be fastened.

Drilling times data are shown in the following Table 1.6.

Table 1.6 - Data related to drilling times

Hole number by drilling order	Length (cm)	Drilling start time	Drilling end time	Drilling time
1	135	0' 0"	04' 27"	4' 27"
2	132	5' 07"	09' 44"	4' 37"
3	121	10' 14"	15' 02"	4' 48"
4	124	16' 15"	20' 47"	4' 32"
5	120	21' 08"	25' 43"	4' 35"
6	118	26' 29"	31' 12"	4' 43"
7	119	31' 44"	36' 10"	4' 26"
8	117	36' 36"	41' 23"	4' 47"
9	112	41' 48"	46' 23"	4' 35"
10	127	47' 01"	52' 03"	5' 02"
11	123	53' 15"	57' 11"	3' 56"
12	126	57' 39"	61' 56"	4' 17"
13	123	62' 24"	65' 52"	3' 28"
14	82	67' 05"	70' 29"	3' 24"
15	88	71' 32"	75' 36"	4' 04"
16	108	76' 04"	80' 51"	4' 47"
17	114	81' 37"	86' 39"	5' 02"

After the drilling phase, the preparatory operations for charging take place. The holes are cleaned with a compressed air jet; the detonators, the booster cartridges and the bag of ANFO are taken from the storage area. Subsequently, primers are prepared by inserting the detonator into the booster. This operation takes about 15 minutes. The insertion of the primer and the ANFO in the hole takes about 12 minutes.

Figure 1.43 shows the drilling times for each hole and the time necessary for completing the loading by means of red bars. As it can be seen, the drilling speed is almost constant during the whole phase.

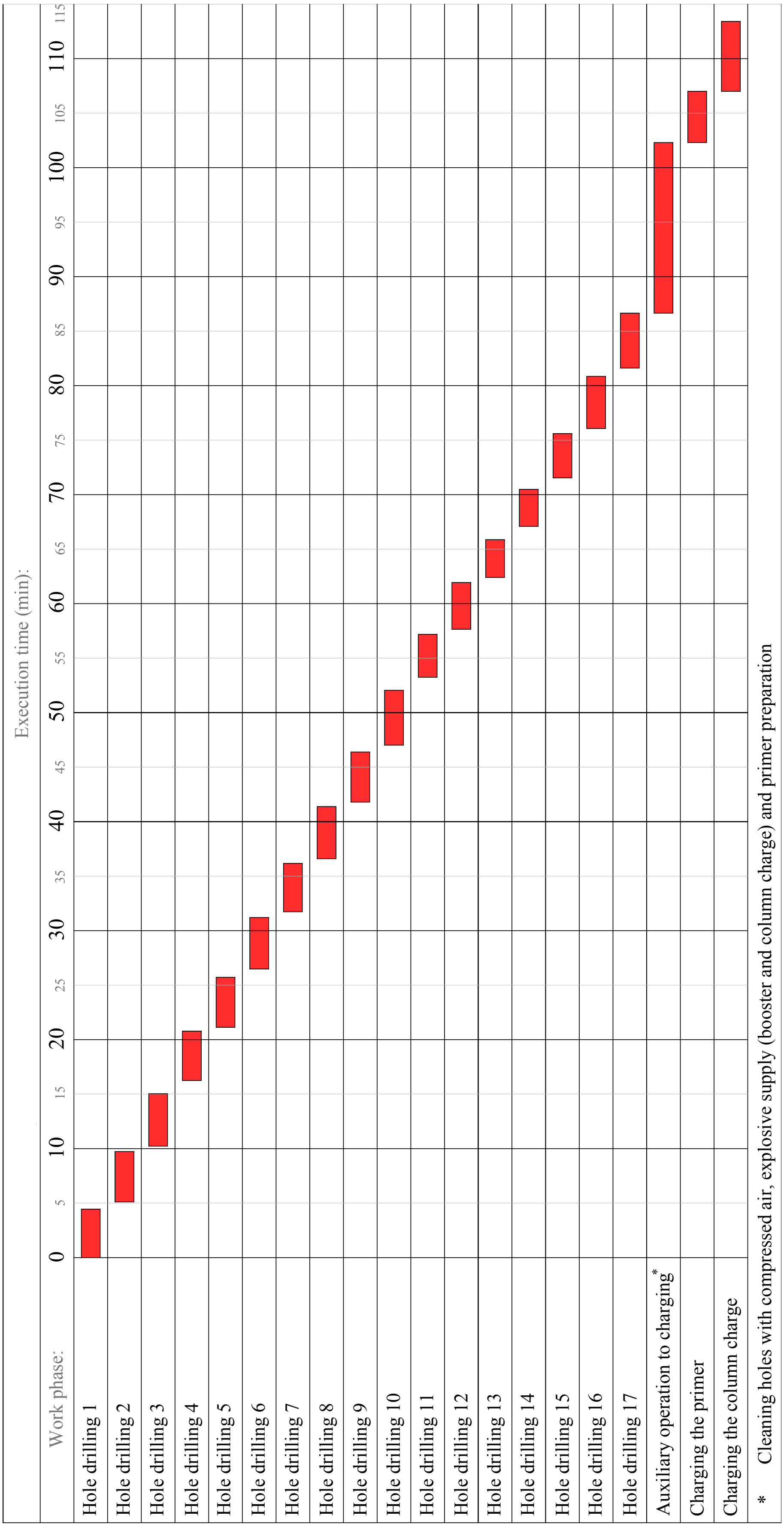

Figure 1.43 - Diagram of drilling and charging times

The holes are charged according to the methods already described. Table 1.7 shows the data measured during the survey and related to charging.

Table 1.7 – Data related to the charging

Hole number by drilling order	Total length (cm)	Charged length (cm)	Non charged length (cm)	Length of fuse out of the hole (cm)	Amount of ANFO (kg)
1	135	80	55	115	0.623
2	132	77	55	118	0.596
3	121	78	43	129	0.605
4	124	81	43	126	0.631
5	120	76	44	130	0.588
6	118	77	41	132	0.596
7	119	75	44	131	0.579
8	117	76	41	133	0.588
9	112	74	38	138	0.570
10	127	75	52	123	0.579
11	123	79	44	127	0.614
12	126	75	51	124	0.579
13	123	77	46	127	0.596
14	82	85	35	168	0.667
15	88	83	37	162	0.649
16	108	73	35	142	0.561
17	114	78	36	136	0.605

The following figures (Figure 1.44, Figure 1.45, Figure 1.46) show the charge of each hole. The booster is represented with a light blue colour, while the ANFO with a pink colour. The safety fuse is drawn with a green line connected to the detonator located at the bottom of the hole. It is possible to note how the length of the unloaded part is quite large (about 1/3 of the length of the hole): this length is left empty. This could cause the ANFO to be ejected during detonation.

In the upper box of the figure, the cross section is shown with the pieces of fuse coming out from the holes. The numbers shown in red represent the order of initiation of the fuse. It should be remembered that this order does not influence the propagation of the detonation, since the use of safety fuse, with the relative uncertainty on the combustion speed, does not allow the definition of a detonation order. In any case, the fuses are triggered according to the order decided by the miner, trying to complete the operation in the shortest time, without forgetting any strand of fuse.

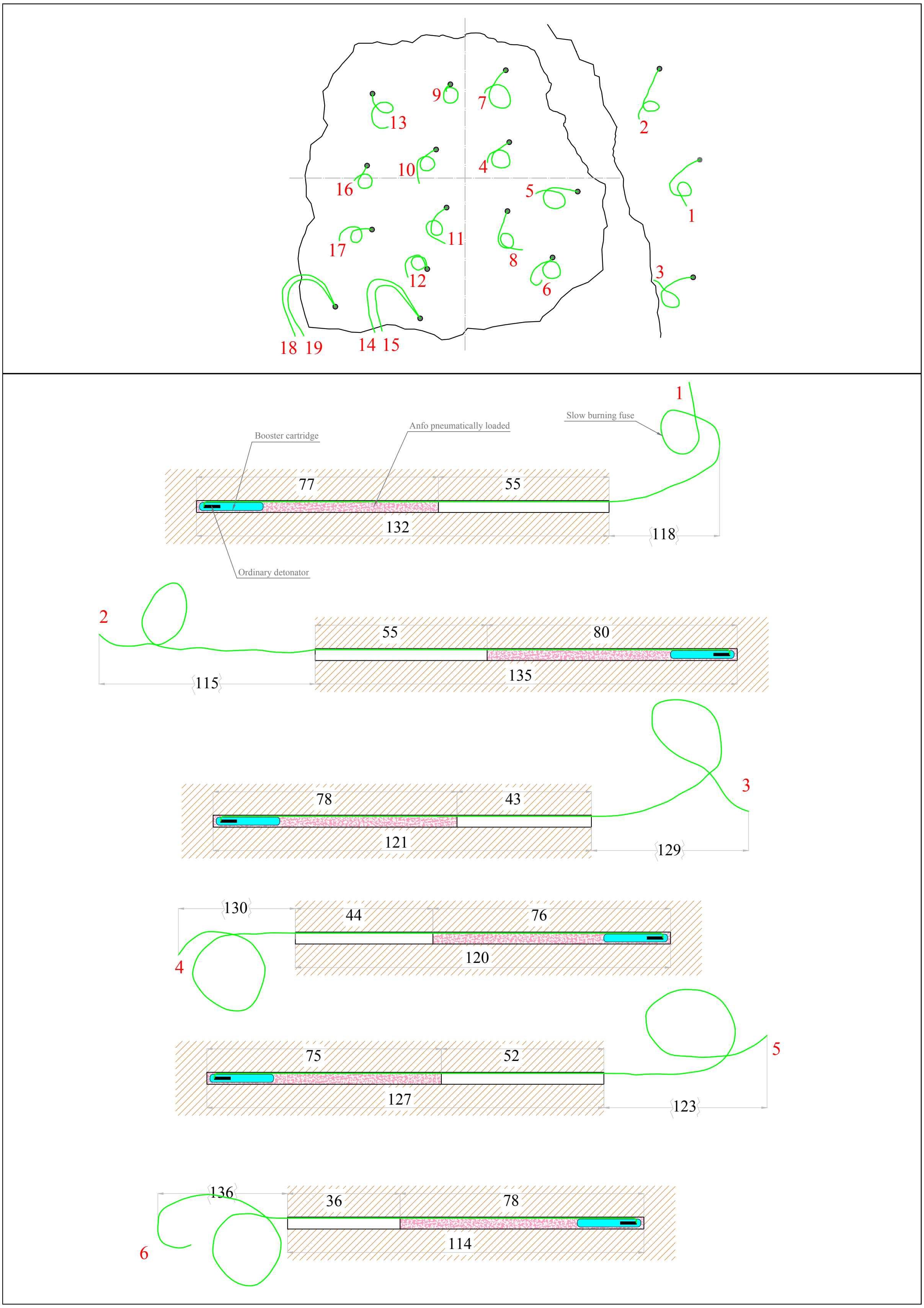

Figure 1.44 – Scheme of the charged holes (1)

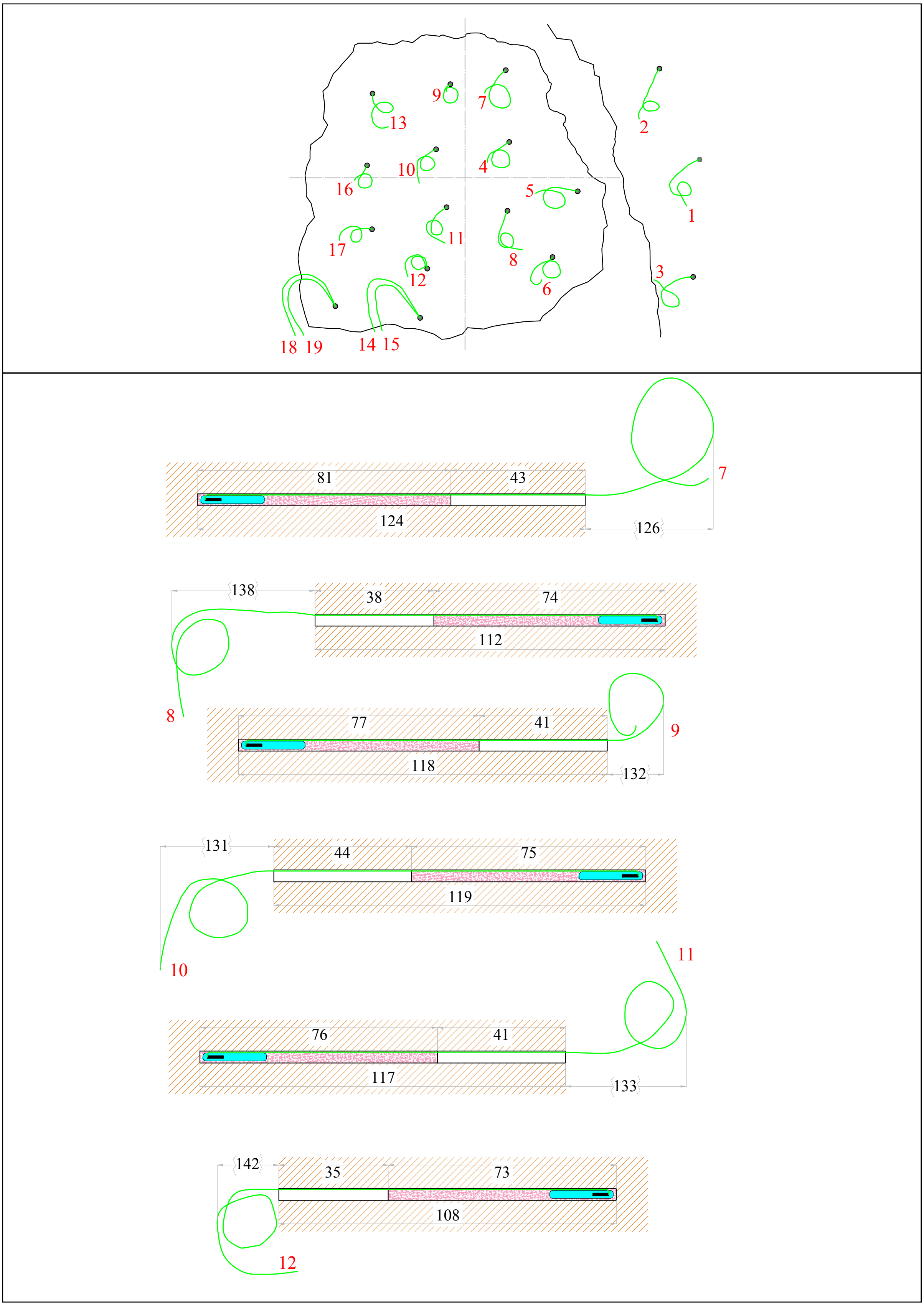

Figure 1.45 - Scheme of the charged holes (2)

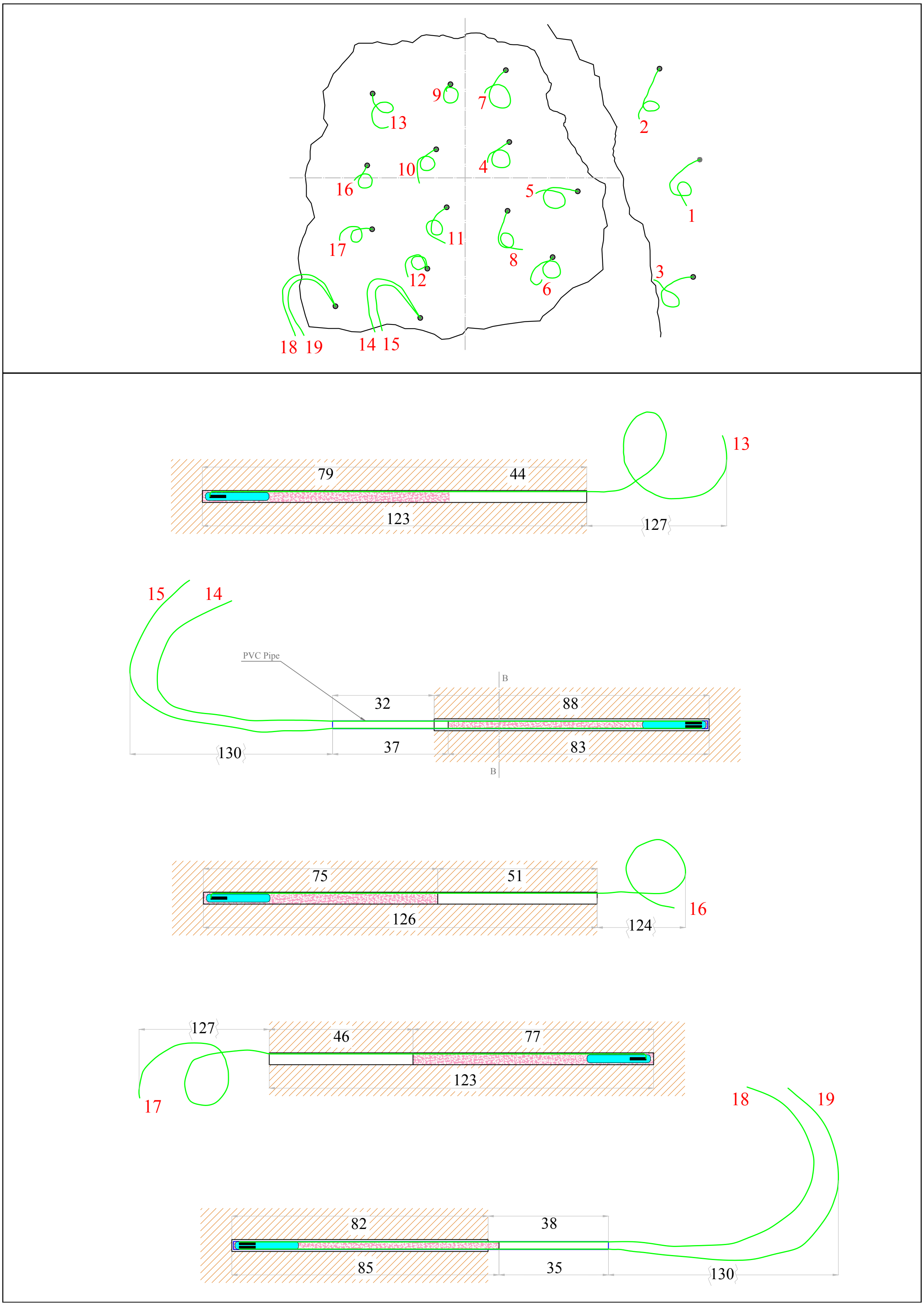

Figure 1.46 - Scheme of the charged holes (3)

After the ignition of blast and waiting for the ventilation time, the analysis of the excavation face obtained after the blast was carried out. It has shown how the profile of the tunnel obtained is very jagged and not traceable to any geometric shape. The excavation face has an extremely irregular surface. Blast-hole sections which have not been completely detonated during the blast have been detected.

In Figure 1.47, one photograph of the face after the blast is shown. The outline of the tunnel and the position where the remaining sections of holes were identified have been traced. As it can be seen, all of them are placed near the contour.

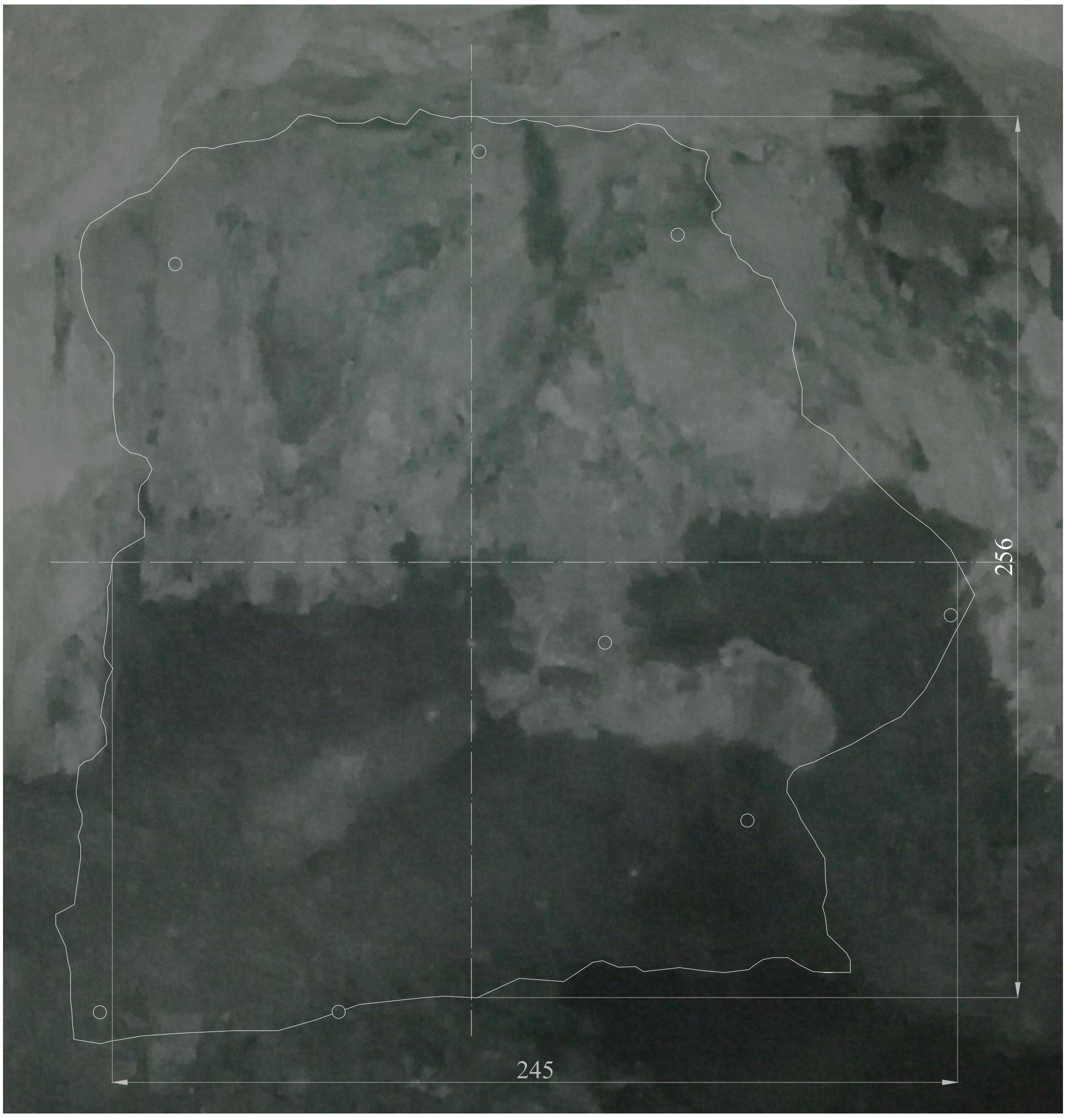

Figure 1.47 – Excavation face after the blast

The length of the remaining holes section after the blast has been measured and this allowed to calculate the efficiency (E) of the blast (1.1), making the ratio between the length of the remaining hole ($L_{remaining}$) and the drilled length ($L_{drilled}$):

$$E = \frac{L_{remaining}}{L_{drilled}} \tag{1.1}$$

In the Figure 1.48, the projections of the sections of holes remaining after the blast are shown, together with the lengths used to calculate the efficiency of the blast.

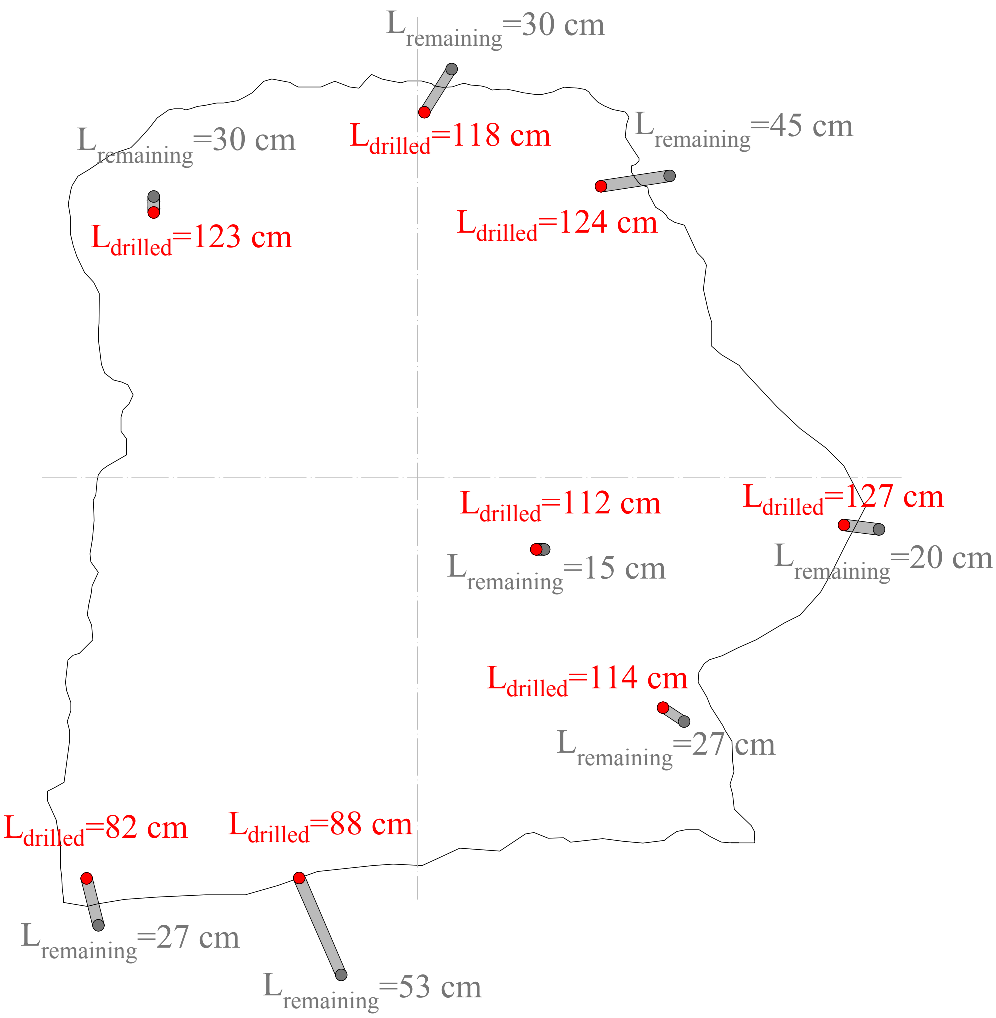

Figure 1.48 - Projection of the remaining sections of the hole after the blast

The following Table 1.8 shows the data (measured) used to calculate the efficiency of the blast.

Table 1.8 - Calculation of the efficiency of the blast

Length of the drilled hole (cm)	Length of the remaining hole (cm)	Efficiency (%)
123	30	76
118	30	75
124	45	64
127	20	84
112	15	87
114	27	76
82	27	67
88	53	40
	Average Efficiency (%)	71

The average efficiency of the blast is very low, equal to 71%. This allows to understand how the excavation technique must necessarily be improved in order to achieve a better performance. The reason why the efficiency is so low is detectable in the absence of a drilling scheme, in the absence of the timing of the detonation and in the use of explosives that are not perfectly suited to the excavation conditions present in the mine. It should be emphasized that such a low efficiency leads to a huge consumption of economic resources: drilling, explosive, work and time costs.

The same four sections (two horizontal and two vertical) that were measured before the blast were analysed, so that a comparison was made between the excavation faces before and after the blast. In the following Figure 1.49, the superimposed outlines of the sections (before and after the blast) and the position where the section profiles were measured can be seen. The projections on the vertical plane of the blast-holes are also represented.

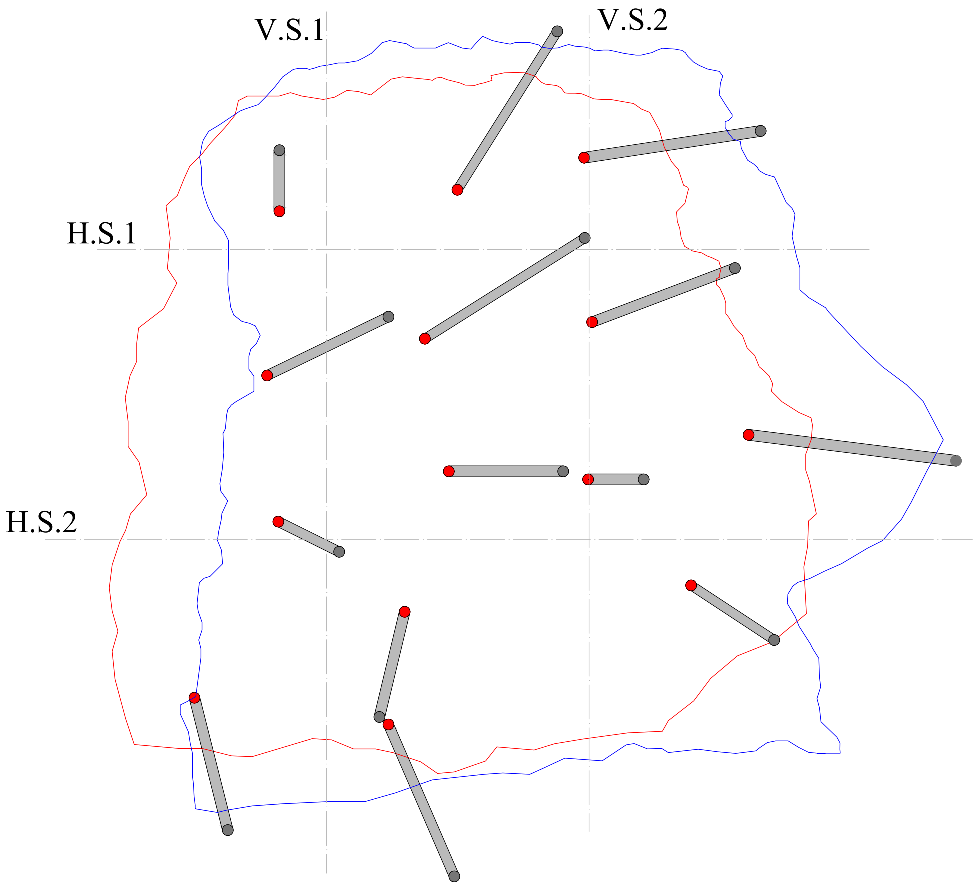

Figure 1.49 - Contour of the sections of the gallery before and after the blast and position of the sections where the profile was measured

Figure 1.50 shows the comparison of the horizontal sections of the excavation face before and after the blast. As it can be seen, it remains very irregular. The excavation section obtained after the blast is slightly displaced to the right due to the inclination of the holes. The next blast will have to regularize the section, enlarging the tunnel on the left side. As previously noticed, the work in the mine is a succession of errors and corrections that continues over the time. Due to the absence of a drilling pattern, the excavation face has a protruding and recessed profile, as the blast-holes do not cooperate each other, being sometimes too close and sometimes too distant. It is also possible to note that the contour is very jagged due to the absence of contour holes. The average progress of the excavation is about 1 m.

In the following tables (Table 1.9 related to horizontal section 1, Table 1.10 related to horizontal section 2) the data collected during the inspection and used to compare the horizontal sections of the faces before and after the blast can be observed.

Table 1.9 - Data on the relief of the excavation faces before and after the blast related to horizontal section 1

Horizontal Section 1			
Face before blasting		Face after blasting	
Distance between reliefs (cm)	Height of the profile (cm)	Distance between reliefs (cm)	Height of the profile (cm)
18	-4	15	-10
12	0	19	-2
15	1	9	0
10	-5	12	13
25	-2	14	2
19	-4	50	0
10	0	7	-8
11	10	16	0
10	0	25	10
17	-9	31	0
13	-4		
22	-2		

Table 1.10 - Data on the relief of the excavation faces before and after the blast related to horizontal section 2

Horizontal Section 2			
Front before blasting		Front after blasting	
Distance between reliefs (cm)	Height of the profile (cm)	Distance between reliefs (cm)	Height of the profile (cm)
14	-2	3	5
45	0	26	0
13	1	26	-5
44	20	35	0
10	10	9	5
24	5	9	0
14	-14	9	-5
20	-10	13	-2
20	-4	32	-15
9	-6	71	-8
17	-1		
10	-3		

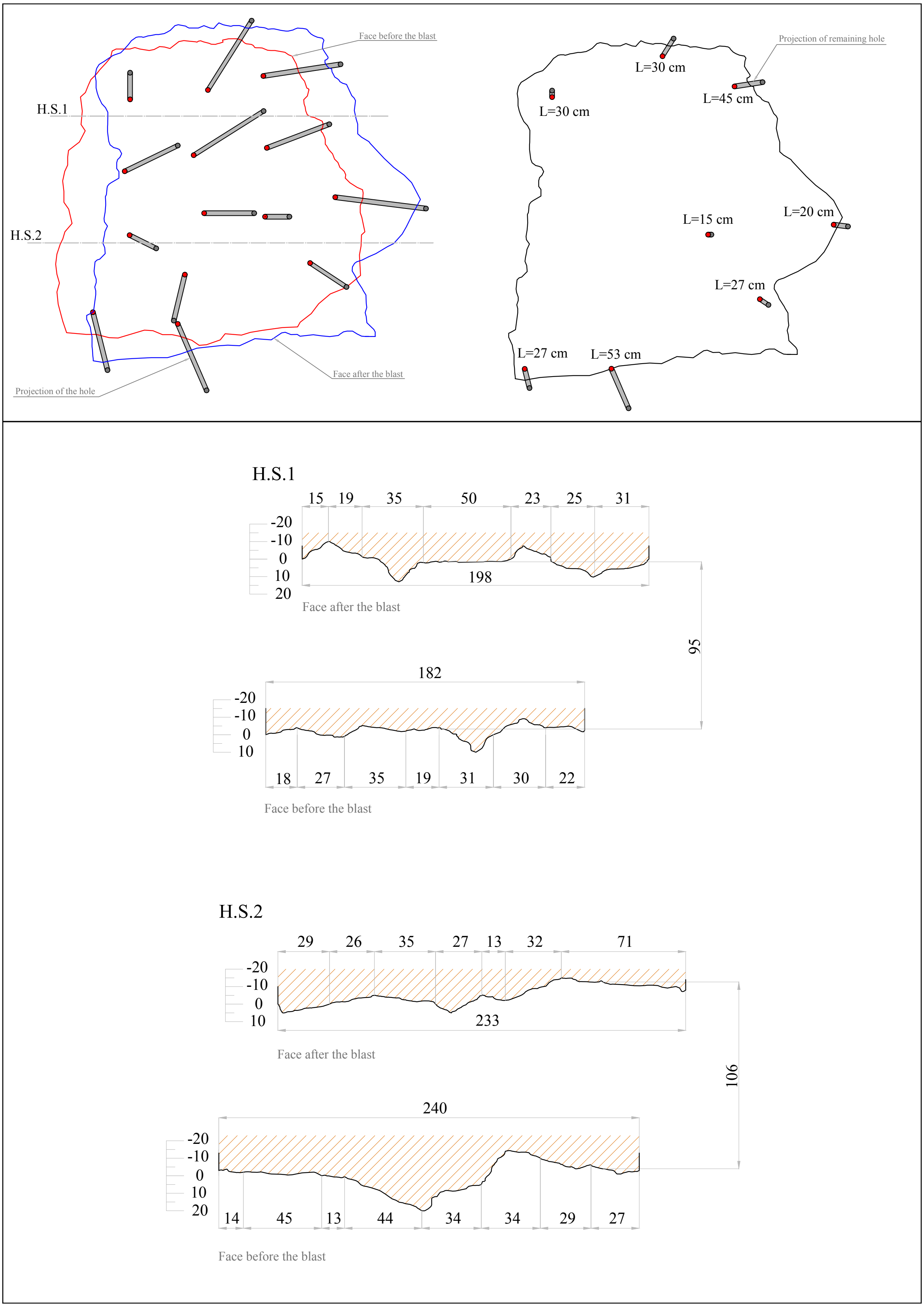

Figure 1.50 - Comparison of the horizontal sections of the excavation face before and after the blast

Figure 1.51 shows the comparison between the vertical sections. As it can be seen, the section after the blast turns out to be slightly higher. Furthermore, the profile of the sections is very irregular: where the holes detonated, there are craters, with a depth up to 30 cm, while some asperities are observed between them. This is due to the lack of cooperation between the blast-holes that were positioned too far apart. This causes problems in the drilling stage as well as making dangerous the area close to the face, since collapses of considerable size can occur. In the following tables (Table 1.11 related to vertical section 1, Table 1.12 related to vertical section 2) the data collected during the inspection and used to compare the vertical sections of the excavation faces before and after the blast can be seen.

Table 1.11 - Data on the relief of the excavation faces before and after the blast related to vertical section 1

Vertical Section 1			
Front before blasting		Front after blasting	
Distance between reliefs (cm)	Height of the profile (cm)	Distance between reliefs (cm)	Height of the profile (cm)
8	6	7	-2
5	0	14	-5
10	-14	13	0
17	-6	18	-2
17	-5	8	-3
30	-6	9	0
15	-8	9	3
20	-2	9	0
13	-4	6	-2
17	0	11	-3
9	2	24	-10
14	3	45	0
49	20	84	25

Table 1.12 - Data on the relief of the excavation faces before and after the blast related to vertical section 2

Vertical Section 2			
Front before blasting		Front after blasting	
Distance between reliefs (cm)	Height of the profile (cm)	Distance between reliefs (cm)	Height of the profile (cm)
7	1	10	0
24	0	9	-13
25	-10	14	-10
12	0	25	-8
14	12	25	-7
23	0	25	-25
20	-25	70	0
17	-10	71	20
32	-20		
25	0		
25	20		

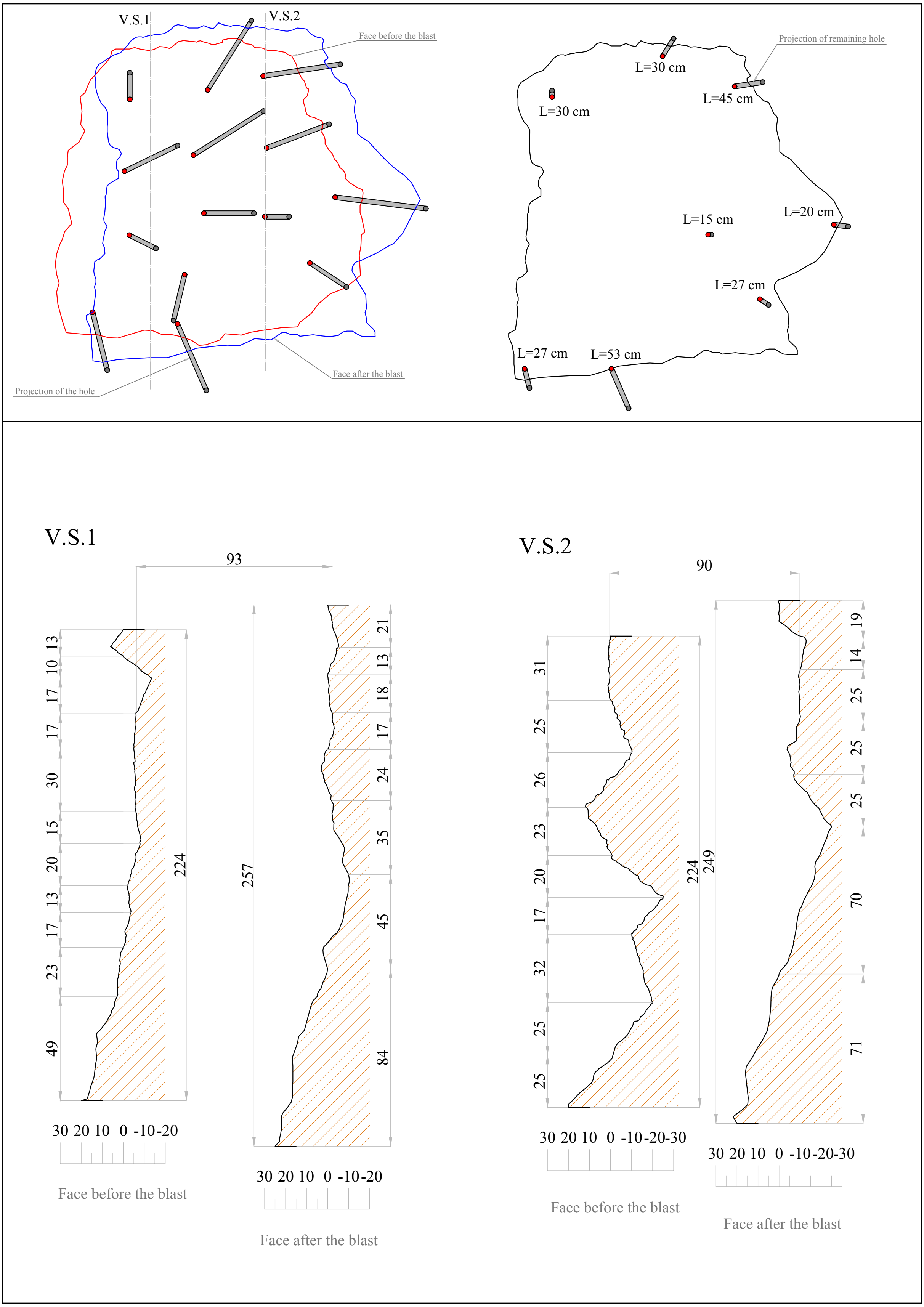

Figure 1.51 - Comparison of the vertical sections of the excavation face before and after the blast

Finally, Table 1.13, summarize the main data concerning the blast. The calculations are related to the whole examined blast. Specifically, the most important data are: volume blasted, amount of explosive, powder factor, volume to remove (using a 1.4 bulking factor), holes density, total drilled length, specific drilling, specific consumption of detonators.

Table 1.13 - Summary of the main data relating to the examined blast

Cross section (m^2)	6.851
Average measured pull (m)	1.17
Average actual pull (m)	0.83
Estimated efficiency (%)	71
Blasted volume (m^3)	8.016
Volume to be removed (m3) (B.F.=1.4)	11.222
Total amount of booster explosive (kg)	2.023
Total amount of ANFO (kg)	10.225
Total amount of explosive (kg)	12.248
Overall powder factor (kg/m^3)	1.528
Holes diameter (mm)	38.1
Holes number (-)	17
Holes density ($hole/m^2$)	2.481
Total drilled length (m)	19.89
Specific drilling (m/m^3)	2.481
Total number of detonators (-)	19
Detonator specific consumption ($detonators/m^3$)	2.370
Total slow burning fuse length (m)	47.5
Slow burning fuse specific consumption (m/m^3)	5.930

After having examined the procedure and the technique of excavation used in the artisanal mine, it is clear that the improvement changes to be introduced are many and that they must be applied both at an organizational level and at a practical executive level. The main changes that must be made are:

- Making work safer;
- Organizing and making the work environment more efficient;
- Creating greater collaboration between miners and managers;
- Regularizing the contour of the tunnels, making it safer and avoiding collapses;
- Introducing an excavation section with a defined shape and size;
- Designing a drilling scheme according to a scientific procedure;
- Using explosives suitable to working conditions;
- Introducing an initiation system that allows timing during detonation;
- Increasing the production and lowering the costs.

Having set these goals, the next chapter will be devoted to the explanation on how to reach them. In fact, the projects and calculations that lead to the definition of the improvement methodology that has been planned and which could be really adopted in the artisanal mine will be shown.

2 - DESCRIPTION OF THE IMPROVEMENTS

2.1 - Analysis of the general context

The present chapter is devoted to the description on how to improve the drill and blast excavation technique in the artisanal mine. The passage from the technique and technology currently used (consisting, as explained in the previous chapter 1, of safety fuse for the initiation of an ordinary detonator, bulk ANFO pneumatically loaded in the hole and emulsion as a booster) to the proposed solution for its improvement cannot take place in a single phase. There are indeed practical problems that don't allow it.

The first problem is related to the fact that the superintendent and the miners themselves are not convinced of the need of modifying anything, because they are totally devoid of a scientific culture and work in the same way as always, believing that the aforementioned technique is the best and the only one that can be used in the mine development. Therefore, improvements should be made little by little, trying to gain the trust and respect on behalf of the workers and managers. On the other hand, the change that for a western engineer would be obvious, for them is a huge turning point, a radical change also linked to the fact that in their opinion the technique currently used is an ethical and cultural factor deeply inculcated by experience and working tradition, often passed down from generation to generation.

Other problems are related to the supply of materials necessary for the employment of a new technique. In fact, the company that provides explosives, the only one for all companies in the district, owns a very limited range of products and supplies all its customers with them. Changing the supplying company would be complicated, being the mine difficult to reach and not very attractive economically. In fact, purchases would be too limited, and would not justify the transport costs. It should also be considered that between the supplier company and the miners there are close relationships of trust, friendship and sometimes kinship that would prevent the change towards other competitor companies. Therefore, specific requests must be made to the company, leaving the necessary time to research, purchase and supply of new products, also guaranteeing new opportunities for market expansion and business development.

Another problem is related to the education and training of miners versus new technologies. It should be kept in mind that many miners have a rather low level of education and culture and therefore the procedure should be applied slowly. It will be necessary to organize theoretical classes, also by means of Power Point presentations, demonstrations on the use of new products and finally practical training in the mine. Furthermore, the explanation of safety rules and procedures should not be forgotten.

2.2 - Definitions of the improvement phases

As a result, the change in the work system has been divided into three distinct phases, which will be applied in succession to the achievement of the pre-established results. It should be emphasized that each phase is followed by the other one in a progressive and fluid manner, without abrupt and sudden changes. In fact, when a phase is operatively applied, every measure for putting into practice the next one must already be implemented: making specific requests to the ownership of the mine, suppliers, training and instruction procedures of the workers, provision of other means and materials and explanation of the results obtained. In this way a productive and collaborative working environment is created, where everyone is an integral part of the system. It is necessary to obtain a relationship of collaboration and interdependence between the external consultant who proposes the improvement measures and who puts them into practice, without the latter becoming merely a passive subject.

- **Phase 1:** the first phase of improvement is the most delicate, as it brings about radical changes regarding the working procedure. The excavation section is square, as explicit request of the client. It should however be noted that this shape does not guarantee an optimal result in terms of regularity of the profile, especially at the corners, where it is not possible to guarantee the respect of the prefixed geometry and where there is an unfavourable concentration of lithostatic stresses. This can lead to collapses, although modest in size, which could be avoided by adopting an arched geometry for the tunnel roof. However, in order to keep a relationship of collaboration with the commissioner of the works in the mine, the improvement proceeds first by following the instructions and their requests. In the subsequent phases, having ascertained the best adaptation of the arched profile, the works will be followed in this way.

  The holes have a length of 1.5 m, performed with the servo-sustained pneumatic drills that are already in use in the mine. The drilling diameter is 1" 1/2 (38.1 mm), made using the same bits that are commonly used. The explosive is the same already in use. However, varying the drilling pattern, the amounts of charge vary. For this reason, prior arrangements must be made with the supplier. As for the bottom charge, the Emultex CN® (emulsion explosive) in 1" x 8" cartridges is used. The column charge consists of pneumatically charged ANFO Premium® (bulk ANFO). These explosives are used to charge all the holes, including the contour holes. However, the use of bulk ANFO presents problems related to the presence of water, especially in the lower holes of the section. This explosive has a poor water resistance and therefore must be protected with plastic pipes, as usually done the in previous working practice. The fact that the ANFO doesn't work in the best conditions in such small diameters is also considered, as ANFO has a critical diameter higher than 38.1 mm. However, it is preferable to continue using this explosive without introducing too many changes in the early stages of improvement. As for the contour holes, no decoupling is employed, and this can lead to an imperfect profile.

  The greatest innovation that has been introduced in this phase is linked to the non-electric (nonel) initiation system. It is very important to explain the importance of timing to hierarchically higher occupational levels, highlighting the advantages of this solution. As for the operators, it is essential to make them able to manage the nonel system, how to make the connections, the reading of the detonator labels, the sequence of installation and all the operational and safety procedures. Therefore, courses will have to be implemented to teach how to use the new technology through theoretical but also practical lessons. The benefits related to the introduction of timing are many: preventive definition of the geometry resulting from the blast, improvement of the contour of the tunnel with the consequent reduction of risk of detachment of rock fragments, increased operational safety, greater control of the grain size distribution of the blasted rock. Then, the drilling scheme has been modified by introducing functional groups of blast-holes, with different charges and a parallel hole cut, with empty holes placed at the centre of the round. The design procedure was performed using the model proposed by Olofsson S.O. (1991).

- **Phase 2:** the second phase of improvement presents a different cross section shape, that consists in arched roof and vertical walls. This geometry ensures a better regularity of the contour, preventing possible localized detachments.

The drilling length is 1.5 m with a diameter of 1” 1/2 made with a servo-sustained pneumatic drill. The explosive is the same as in the previous phase regarding cut and stoping (production) holes (for the bottom hole charge, Emultex CN® emulsion explosive is used in 1" x 8" cartridges. For the column charge, pneumatically charged ANFO Premium® is used). The same explosive is used for the floor holes.

The most important evolution in this phase is linked to the explosive used for the contour holes (walls and roof), where a detonating cord with a charge concentration of 80 g/m is proposed (Pentacord® Special 80P). As in the previous phase, the ignition system is non-electric. The drilling scheme for cut, production and floor holes is performed with the Olofsson (1991) procedure. The design of the drilling scheme of the contour holes (walls and roof) is carried out according to the empirical rules of the smooth blasting.

- **Phase 3:** this is the last phase of improvement that will be developed, once the first two will have been used and assimilated. This third phase involves the most important changes on the point of view of the management and execution of the work, as well as the means and materials used. The excavation cross section has the same shape as the one introduced in step 2.

The drilling length is 3 m and is performed using a Jumbo (model Boomer 104®) that is already owned by the mining company and located in another yard. The drilling diameter is 2" (50.8 mm). This innovation must be preceded by training courses in order to make the personnel aware of how to use this machine. The aspects related to the work safety must not be overlooked. It is also important to perform a geometric survey of all the tunnels in the mine, in order to ascertain that the boomer can move without problems. Where the tunnels turn out to be too narrow or low, the walls must be widened, and the roofs raised with localized interventions. The maintenance of the machine must also be foreseen, as well as the training of the personnel responsible for repairs and programmed checks.

The explosives used for cut, stoping (production) and contour (floor) holes are different compared to the previous stages. The bulk ANFO is abandoned in favour of cartridged charges. As booster, a semi-gelatin dynamite (Tronex® Plus) is used in 1 "1/2 x 10" cartridges and the Emultex CN® emulsion is used in cartridges of 1 "1/2 x 10". Modifying the bottom hole charge guarantees greater power and fragmentation capacity of the rock, that is necessary as the length of the holes has been doubled. The change in the column charge leads to several improvements. First, there are no problems related to the presence of water in the hole. The emulsion has a better water resistance and the cartridges guarantee a better protection; it is also more powerful than ANFO. This change also has a logic motivation: by increasing the length and the diameter of the holes, the amount of ANFO to be loaded into the hole would increase too. The ANFO is loaded pneumatically directly from the bags where it is transported and stored, and in this case large amounts of charge would be a practical problem. For this reason, the choice is to use only cartridged explosives. The use of new explosives should be discussed beforehand with the mine owners and the explosives suppliers so that availability, costs and purchase frequencies could be agreed. As for the contour holes (walls, roof), the detonating cord (Pentacord® special 80P) with charge density of 80 g/m is used, as introduced in step 2.

The ignition system is non-electric, as introduced in the first phase of improvement. The drilling scheme is designed according to the Olofsson (1991) procedure regarding the cut, the production holes and the floor holes. As for the contour holes (walls and roof), the empirical rules of smooth blasting are followed. By increasing the quantities of explosives, it has also to be considered that the widening of the explosive storage area is necessary, according to the strict national regulations. When the third phase is implemented, the storage room must have already been expanded, the drilling machine already tested and maintained, the explosives already tested and approved, the operators correctly trained.

The three steps of improvement are interrelated and each change, as already mentioned, is the result of precautions, decisions and preparatory actions that have to be implemented in advance. All the solutions proposed must be discussed with the mine personnel, at all levels of employment, in order to promote cooperation and mutual respect. It should also be noted that these changes must be effectively tested to the real conditions of the mine, on a real scale in order to adapt them to actual conditions. Therefore, the proposed scheme is not rigid and fixed, but can be easily modified with corrections and implementations if necessary. Each phase must be followed by a check of the results obtained that will be discussed and shared. The presence in the mine of a professional who checks the evolution of each phase is therefore fundamental.

Any change, anticipations or postponement on the times and the methods of intervention may depend on different reasons: lack of willingness on behalf of the personnel or managers of the mine, unavailability of materials or equipment, impossibility of finding the explosives by the supplier company and delays due to poor organization. For this reason, the scheme should not be bound, but always modifiable, open to suggestions and implementations.

For each phase, three variants have been proposed for the size of the excavation section: 2.5 x 2.5 m, 3.0 x 3.0 m, 3.5 x 3.5 m. These scenarios were adopted in order to organize every possible situation linked to the different operational possibilities. In fact, due to the mine exploitation technique, the size of the excavation cross section is linked to the thickness of the orebody. By proposing three different excavation sizes, all the ranges are covered, in order to create operational schemes that do not cause misunderstanding on behalf of the operators.

Table 2.14 shows a summary of each phase, together with the main improvement purposes introduced. For each phase, the following details are specified: improvements introduced, shape of the sections and holes diameter, explosives used, ignition system, preliminary actions, possible problems, solutions that need to be implemented in the subsequent phases.

Table 2.14 - Summary of the main characteristics of the interventions implemented at each stage

Phase	Improvements introduced	Shape of the section and holes dimensions	Explosive used			Ignition system	Preliminary actions	Possible problems	Solution that need to be implemented in the subsequent phases
			Bottom charge	Column charge	Contour (walls,roof)				
1	Excavation section with defined shape and dimensions Project of the drilling scheme according to the Olofsson (1991) procedure Introduction of the parallel holes cut Implementation of non-electric ignition system (blast timing)	Shape: square Holes length: 1.5 m Holes diameter: 1" 1/2 (38.1 mm)	Emultex CN® in cartridges (1" x 8") (emulsion explosive)	ANFO Premium® (Bulk ANFO)	As for the cut, production and floor holes	Non electric dual detonator and non electric detonator (nonel system)	Staff training for the use of new techniques Agreements with the company supplying explosives for the procurements of materials Lessons to the miners on the use of nonel technology	Unavailability of the materials to be used Lack of willingness on behalf of the personnel or managers of the mine Possible not regular profile of the tunnel contour with consequent localized collapses Learning difficulties on the part of miners	Use of the arched roof of the tunnels Use of more suitable explosives Decoupling of the charge in the contour holes (walls, roof) Production improvement
2	Improvement of the shape of the excavation section Decoupled explosive for the contour holes (walls, roof) using a different type of explosive	Shape: arched roof and vertical walls Holes length: 1.5 m Holes diameter: 1" 1/2 (38.1 mm)	Emultex CN® in cartridges (1" x 8") (emulsion explosive)	ANFO Premium® (Bulk ANFO)	Pentacord® Special 80P (detonating cord with charge concentration 80 g/m)	Non electric dual detonator and non electric detonator (nonel system)	Staff training for the use of new techniques Agreements with the company supplying explosives for the procurements of materials	Unavailability of the materials to be used Lack of willingness on behalf of the personnel or managers of the mine Learning difficulties on the part of miners	Use of more suitable explosives Production improvement
3	Doubling of the design pull Drilling performed using a Jumbo Use of more efficient explosives Increased production	Shape: arched roof and vertical walls Holes length: 3 m Holes diameter: 2" (50.8 mm)	Tronex® Plus in cartridges (1" 1/2 x 10") (semi-geltin type dinamite)	Emultex CN® in cartridges (1" 1/2 x 10") (emulsion explosive)	Pentacord® Special 80P (detonating cord with charge concentration 80 g/m)	Non electric dual detonator and non electric detonator (nonel system)	Perform a geometric survey of all the tunnels of the mine to ascertain that the Jumbo can move without problems Jumbo maintenance Training of operators in the use of jumbo Staff training for the use of new techniques Agreements with the company supplying explosives for the procurements of materials	Unavailability of the materials to be used Lack of willingness on behalf of the personnel or managers of the mine Learning difficulties on the part of miners Execution of many preparatory works for the adaptation of the dimensions of the tunnels, with a consequent great waste of time	-

2.3 - General design of an underground blast

In the following, the calculations and the rules used to design the blasts for each phase and for the three cross sections considered are reported.

The project is performed according to the Olofsson (1991) procedure. All the graphs that will be shown in order to carry out the project procedure are taken from: "*Applied explosive technology for construction and mining*". Olofsson, S.O., (1991).

A parallel holes cut was chosen, for the following main reasons:

- The excavation section required by the mine is rather narrow;
- Accuracy and manual skills of the operators managing the servo-sustained pneumatic drills and also Jumbo are poor;
- High simplicity of the blast organization, as it isn't necessary to perform inclined holes (which, in any case, would not allow to reach high pulls).

When designing a parallel holes cut, the following parameters have to be considered as fundamental:

- Diameter (or fictitious diameter) of the empty hole;
- Burden;
- Charge/hole.

Furthermore, drilling accuracy is essential, especially with regard to the holes near the empty hole: even the smallest deviation could cause the intersection between the blast-hole and the empty hole, or a too high burden, that would entail limited cracking of the rock or plastic deformations, determining a poor efficiency as well as too coarse fragmentation.

2.3.1 - Design of the cut blast-holes

Being not possible using big diameter bits or a reamer, it is necessary to resort to a number of close holes, having the same diameter as those used for the whole blast. A fictitious diameter is then calculated, by the formula (2.1):

$$D = \sqrt{n} \cdot d \tag{2.1}$$

Where: D is the fictitious diameter and n is the number of holes having diameter d.

The general criterion for designing the number (and hence the fictitious diameter) or the diameter of the empty hole is related to the volume of rock broken by the first blast-hole that will detonate: this volume must be less or equal to the volume of the empty hole (calculated considering the fictitious diameter). This criterion is summarized in the following expression (2.2):

$$V_{rock1^{st}} < V_{empty\ hole} \tag{2.2}$$

This concept is represented in Figure 2.52, which shows the volume of the rock that detonates at the first phase of the sequence and the empty hole.

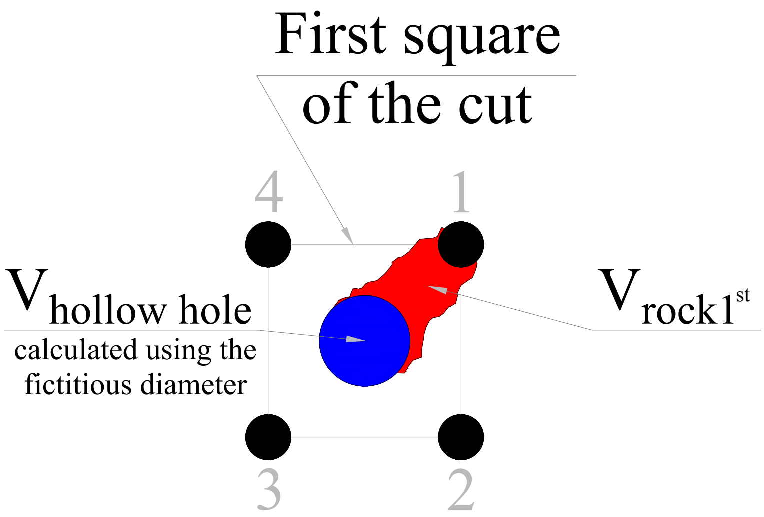

Figure 2.52 - Representation of the factors necessary for the dimensioning of the empty holes

Using an empirical graph, the theoretical pull of the blast can be evaluated. Figure 2.53 shows the length of blast-holes (horizontal axis) vs. the % advance per round (vertical axis), i.e. the ratio between the actual pull and the design pull. The curves represent the diameter of the empty hole. It can easily be seen that by increasing the length of the holes, the overall yield decreases. It can also be noted that, with the same length of blast-holes, the advance is greater by increasing the diameter of the empty hole. The length of the holes in the blast and the diameter of the empty hole (in this case the fictitious diameter) being known, it's possible to determine the general advance/blast.

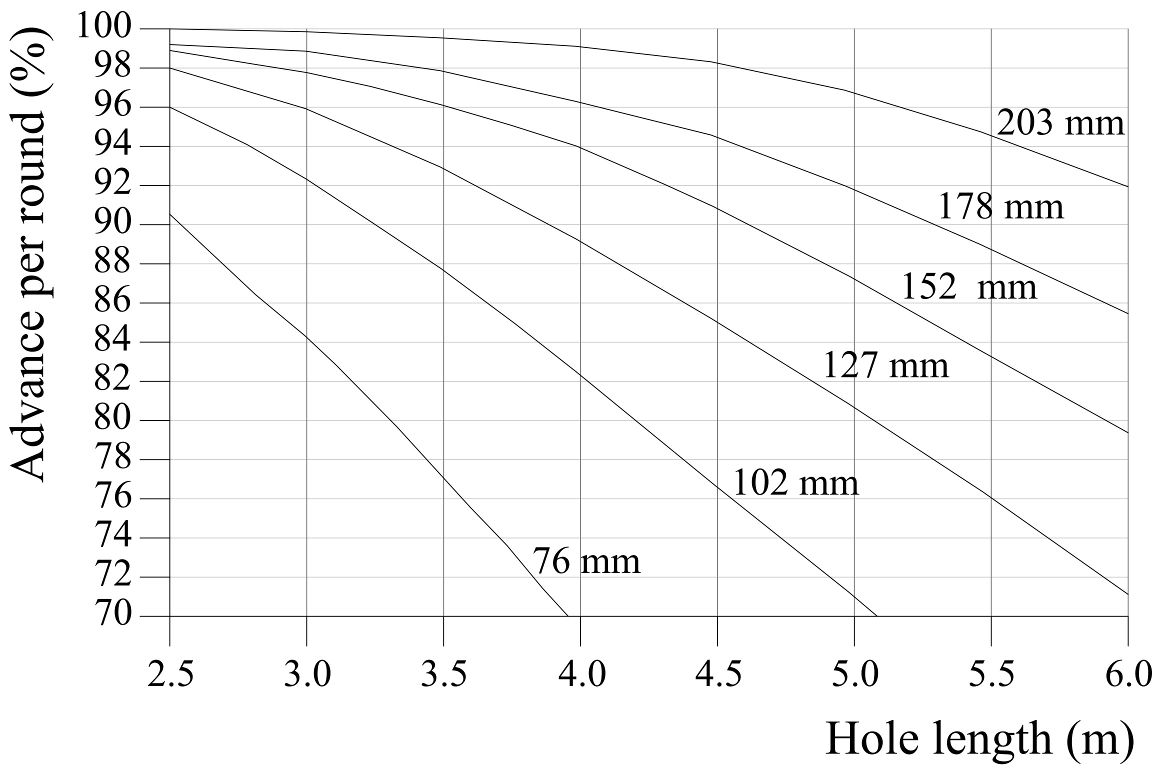

Figure 2.53 - Empirical graph used to obtain the advance of the blast knowing the length of the holes and the diameter of the empty hole of the cut

The position of the cut within the cross section will subsequently be determined. This is generally positioned at the centre for practical reasons, but in some cases, it can be placed in other positions (for example in areas where the presence of more compact or hard rock has been detected or for practical reasons related to an easier drilling and charging).

The first phase of blast design consists in the calculation of the first square. The distance between the blast-hole and the empty hole must not be more than 1.5 times the diameter (or fictitious diameter) of the empty hole, to obtain an effective opening of the excavation face. If the blast-hole is farther away, the detonation will merely break and crack the rock (involving rock plasticization); if it is closer, the holes and the empty hole could cross each other (in addition to the fact that, in this case, the volume of blasted rock will be less than the optimal one).

Therefore, the position of the blast-holes in the first square is expressed by the formula (2.3):

$$a = 1.5 \cdot \Phi \tag{2.3}$$

Where: a is the distance between the centers of the large empty hole and the blast-hole, Φ is the diameter of the large empty hole.

In case of several empty holes, the relation is expressed as (2.4):

$$a = 1.5 \cdot \mathrm{D} \tag{2.4}$$

Where: a is the distance between the center point of the empty holes and the center of the blast-hole and D is the fictitious diameter.

Figure 2.54, strictly empirical, shows the aforementioned correlation between the distance between the blast-holes and the diameter or fictitious diameter of the empty hole. The inclined straight lines identify the range of acceptability of the distance which turns out to be, as already seen, optimal for the value equal to 1.5 times the diameter of the empty hole. The curves represent the fictitious diameter or diameter of the empty hole. The distance between the centers of the holes is shown in horizontal axis while the linear concentration of charge, expressed in kg/m, can be read on the vertical axis. The central inclined line is intercepted corresponding to a distance between the blast-hole and the empty hole equal to 1.5 times the diameter or fictitious diameter of the empty hole. Once the point of intersection is known, the distance between the holes can be read on the horizontal axis and the linear concentration of charge on the vertical axis.

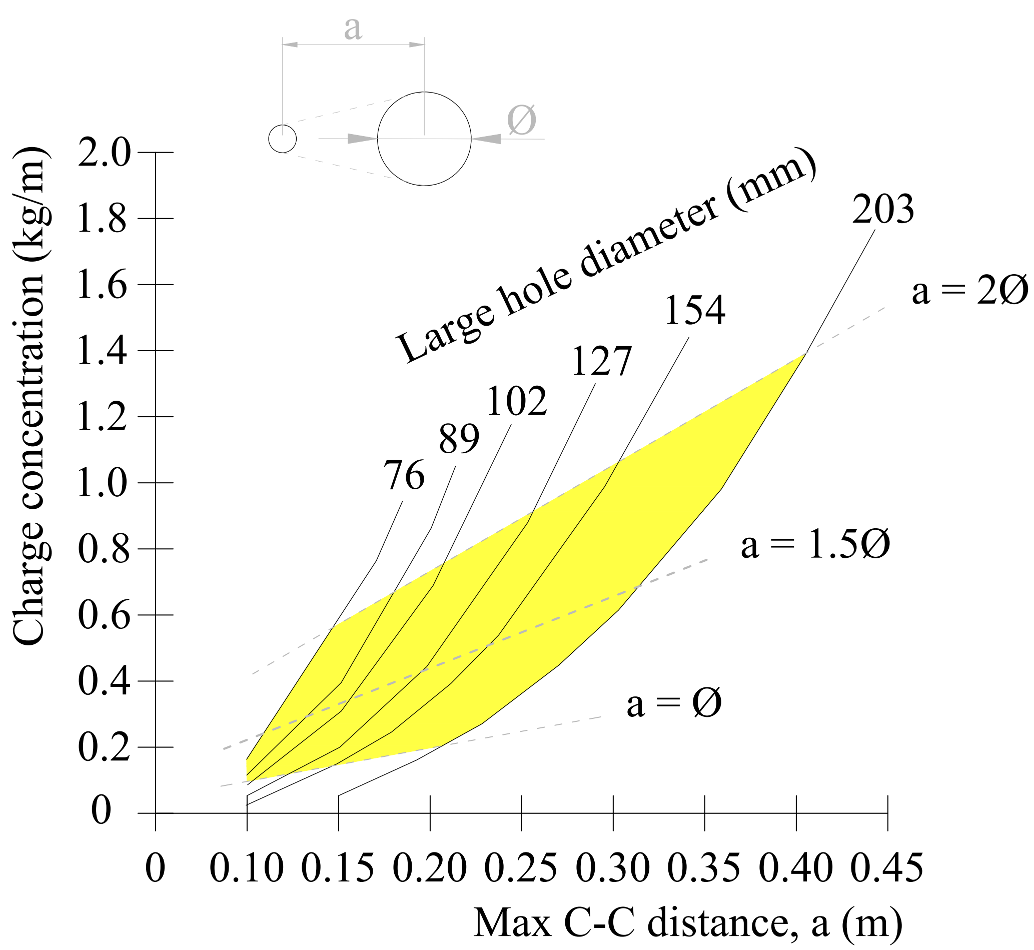

Figure 2.54 – Empirical graph used to determine the distance between the centre point of the empty holes and the centre of the blast-hole and the linear charge concentration knowing the value of the fictitious diameter of the large empty hole

The design continues by dimensioning the remaining squares of the cut. W is defined as the length of the side of the square hole produced by the detonation of the previous square and B is defined as the distance of the blast-hole from the previous square.

Experimentally it has been found how the optimal value of the distance between the blast-hole and the empty volume produced by the detonation of the first square of the cut (B) is equal to the side (W) of the aforementioned square.

It should be remembered that values greater than W = B value can lead to an inefficient opening of the excavation face with cracking of the rock but without fragmentation. A lower distance will cause an overcharge with consequent problems related to excess of vibrations, fly-rocks as well as a useless waste of resources (time, energy, drilling materials and explosives).

The empirical graph in Figure 2.55 shows the aforementioned correlation. The inclined and dashed lines identify the range of acceptability of the distance between different squares. The curves represent the value of the length of the side of the previous square(s). In the horizontal axis, the burden (B), while the linear charge concentration in kg/m is given on the vertical axis.

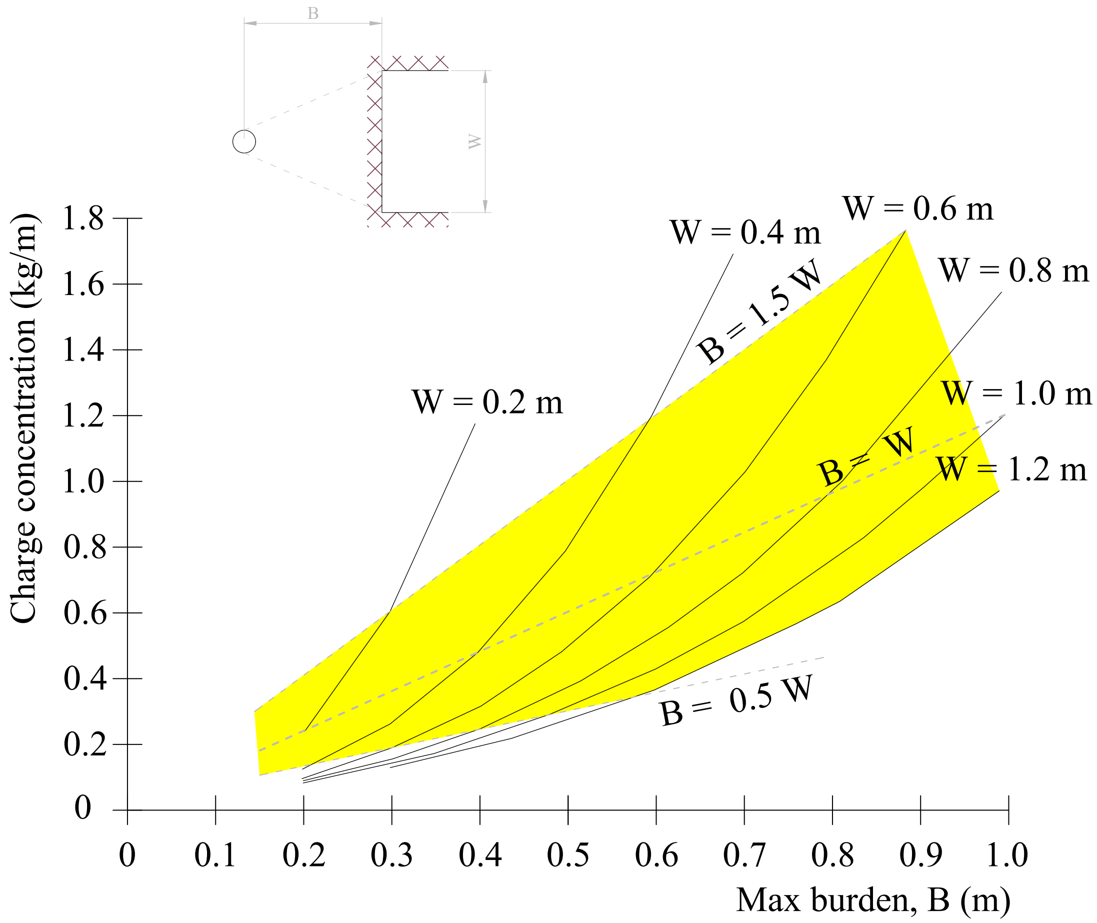

Figure 2.55 – Empirical graph used to determine the distance between squares of the cut knowing the value of the length of the side of the previous square of the cut

The necessary number of squares must guarantee a good opening of the excavation face without an excess of charge or, on the contrary, without inducing fragmentation. It is therefore evident that the size of the cut is proportional to the size of the excavation section: the larger the excavation section is, the greater the number of squares that can be made.

2.3.2 - Design of the production and contour blast-holes

Once the cut has been dimensioned, the design sequence continues with the next step, that is the positioning of the production blast-holes (stoping holes, holes surrounding the cut). Usually the production holes are positioned around the cut following a uniformly distributed scheme, using a space/burden ratio of 1: 1.1. The design of the pattern related to the stoping holes can be carried out through an empirical graph (Figure 2.56), where the linear charge concentration (kg/m) is given on horizontal axis, and the burden (m) on the vertical axis. The segments below the graph take into account various types of explosive, and the blast-holes diameter.

The type of charge and the diameter of the holes being known, the horizontal axis and the curve are intercepted. By this way, both the linear charge concentration and the burden can be easily read on the graph.

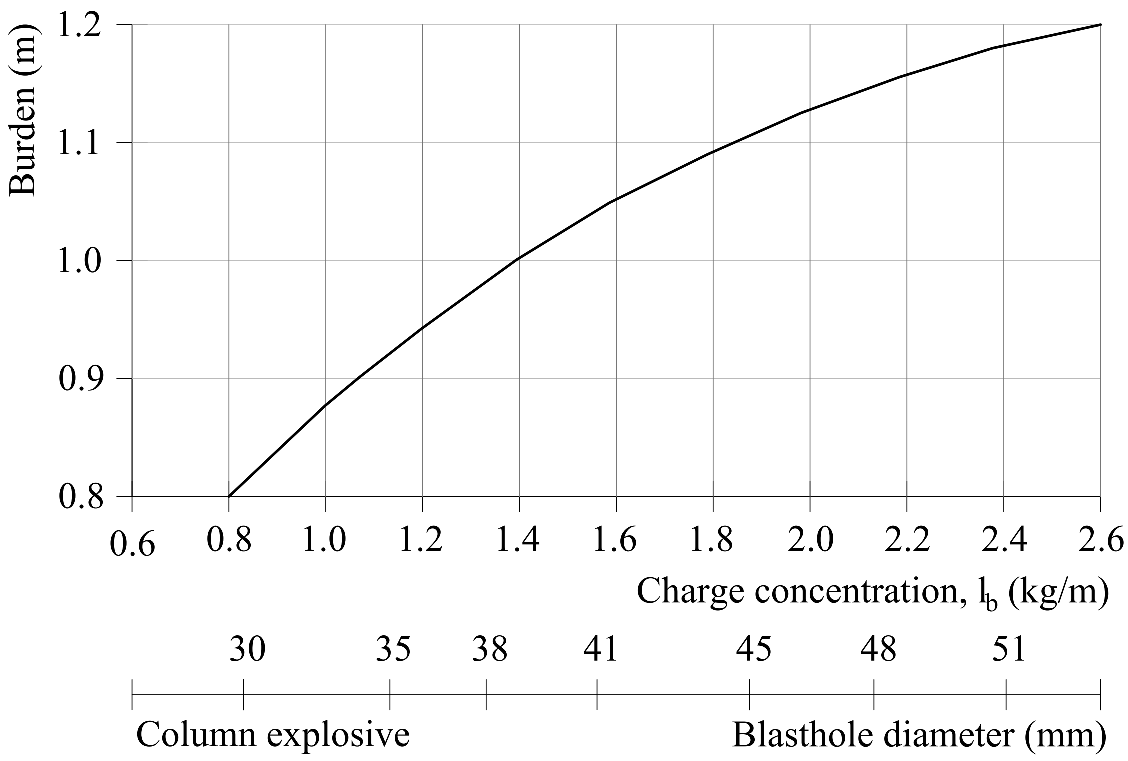

Figure 2.56 – Empirical graph used to determine the linear charge concentration ant the burden for production and contour holes knowing the column explosive type and the blast-hole diameter

These data are necessary for the dimensioning of the production and contour blast-holes scheme, according to Table 2.15, which shows the values of burden (m), spacing (m), height of the bottom charge (m), charge concentration for the bottom charge and for the column charge (kg/m) for the stoping blast-holes (upwards, horizontal, downwards) and for the contour holes (floor, walls and roof) of the tunnel section.

Table 2.15 - Empirical table used for the dimensioning of the production (stoping) and contour blast-holes knowing the values of burden (B) and of linear charge concentration (l_b) obtained from the graph in Figure 2.56

	Part of the round:	Burden (m)	Spacing (m)	Height of the bottom charge (m)	Charge concentration		Stemming (m)
					Bottom (kg/m)	Column (kg/m)	
Contour:	Floor	1.0 x B	1.1 x B	1/3 x H	l_b	1.0 x l_b	0.2 x B
	Walls	0.9 x B	1.1 x B	1/6 x H	l_b	0.4 x l_b	0.5 x B
	Roof	0.9 x B	1.1 x B	1/6 x H	l_b	0.3 x l_b	0.5 x B
Stoping:	Upwards	1.0 x B	1.1 x B	1/3 x H	l_b	0.5 x l_b	0.5 x B
	Horizontal	1.0 x B	1.1 x B	1/3 x H	l_b	0.5 x l_b	0.5 x B
	Downwards	1.0 x B	1.2 x B	1/3 x H	l_b	0.5 x l_b	0.5 x B
Where: B is the burden and l_b is the linear charge concentration determined with the graph shown in Figure 2.56. H is the length of the blast-hole.							

2.3.3 - Trigger sequence

Is then necessary to define the trigger sequence. The initiation system selected for the project is non-electric. This system has many advantages, among which: high ease and speed of use and installation, excellent safety and efficacy of use, good accuracy of the detonation sequence. It should also be considered the possible easy availability of non-electric detonators on the Chilean explosives market.

The trigger sequence has to respect the principle according to which at least two free faces are available. For this reason, the first blast-holes that detonate are those belonging to the cut. They are detonated one by one, in series. As for the production, the blast proceeds with a series-parallel scheme: it can be observed a branching on the two sides of the section. The order of detonation still continues in series-parallel for the contour holes: first the walls are detonated and then the roof and the holes at the corner of the floor.

Figure 2.57 provides an example of a detonation sequence. The arrows represent the propagation of the detonation. The sequence, as it is clearly visible, starts from the cut holes, and evolves according to the scheme shown in the same figure.

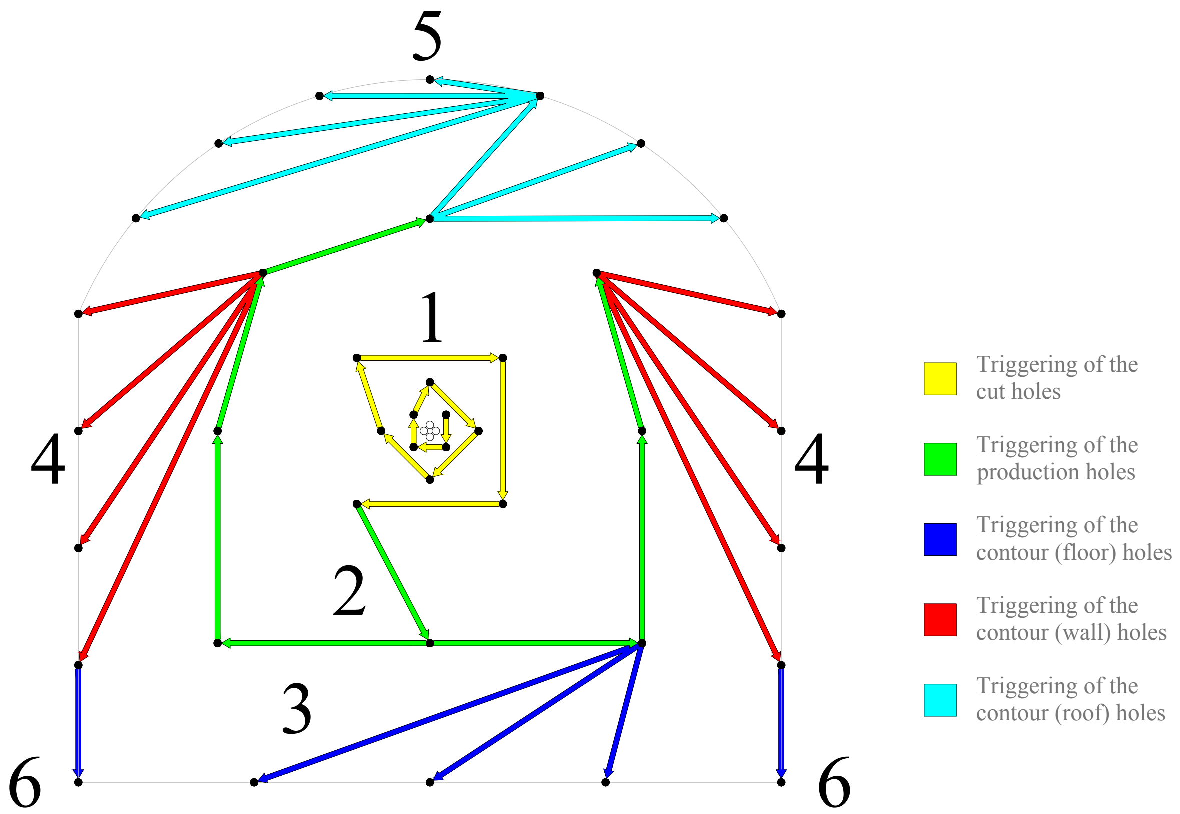

Figure 2.57 – Example of a detonation order in an excavation section. The arrows indicate the sequence of initiation of the holes

2.4 - Design of the blast related to phase 1

The calculations are reported for the three dimensions of the section, i.e.: 2.5 x 2.5 m, 3.0 x 3.0 m, 3.5 x 3.5 m.

As already discussed above, the improvement provided in phase 1 consists in: square-shaped section, holes with a length of 1.5 m and a diameter of 1"1/2 (38.1 mm). The bottom hole charge consists of Emultex® CN, in cartridges 1" x 8" in size. The column charge consists of ANFO premium® (pneumatically charged bulk ANFO). The ignition system is non-electric.

The cut, for the 3 sections, is the same. For this reason, the calculations and schemes of the cut will be shown only once, while for the production and contour blast-holes all cases will be described. The cut is placed in the middle of the section.

By evaluating the blasted volume at the first instant of detonation, the equivalent diameter of the central empty hole has been determined. It is made with 4 holes having a diameter of 38.1 mm and spaced 6 cm with respect to the axis. The equivalent diameter, according to the formula is calculated trough the equation (2.5):

$$D = \sqrt{4} \cdot 38.1 = 76.2\ mm \tag{2.5}$$

A graphic representation of the empty holes and of the equivalent diameter is shown in Figure 2.58:

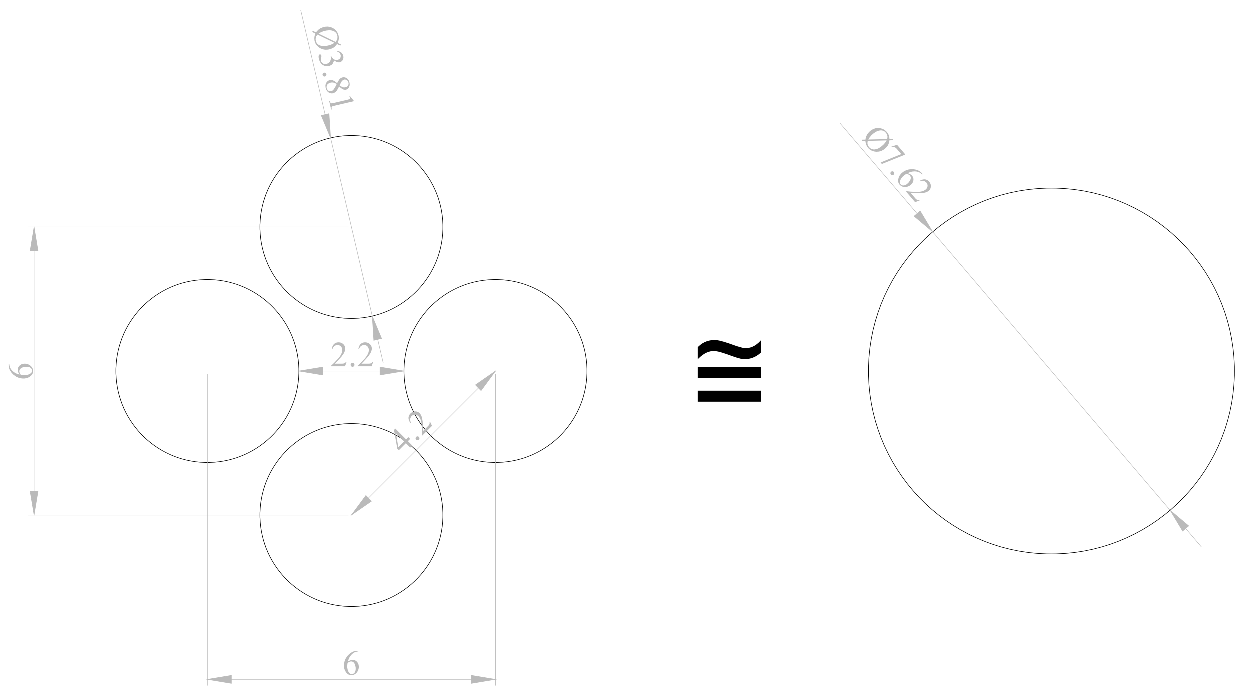

Figure 2.58 – Representation of the empty holes of the cut and of the corresponding equivalent diameter

The following step consists in positioning the holes of the first square, according to Figure 2.59. The distance between the center of each blast-hole and the center of the fictitious hole is equal to 11 cm. Therefore, the first square of the cut will have a side of (2.6):

$$L_1 = \sqrt{2} \cdot 11 = 16\ cm \tag{2.6}$$

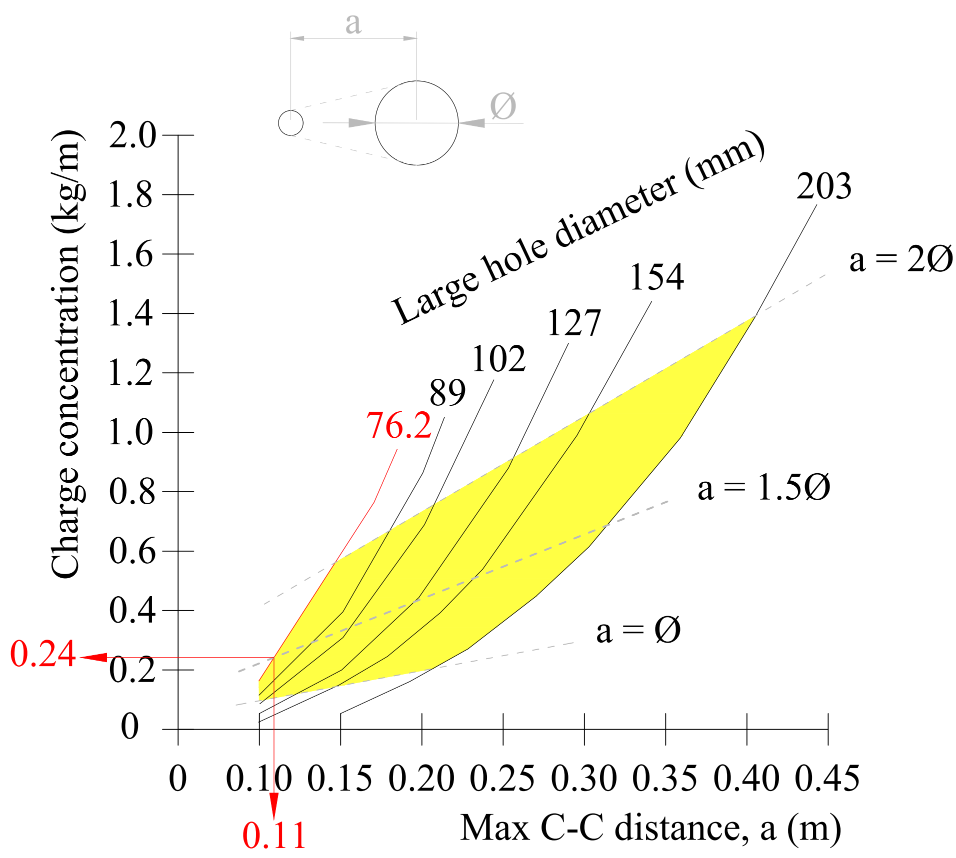

Figure 2.59 – Empirical graph used to determine the distance between the blast-holes of the first square of the cut from the centre point of the empty holes and the linear charge concentration knowing the fictitious diameter of the empty hole

The graphical representation of the first square of the cut is shown in Figure 2.60.

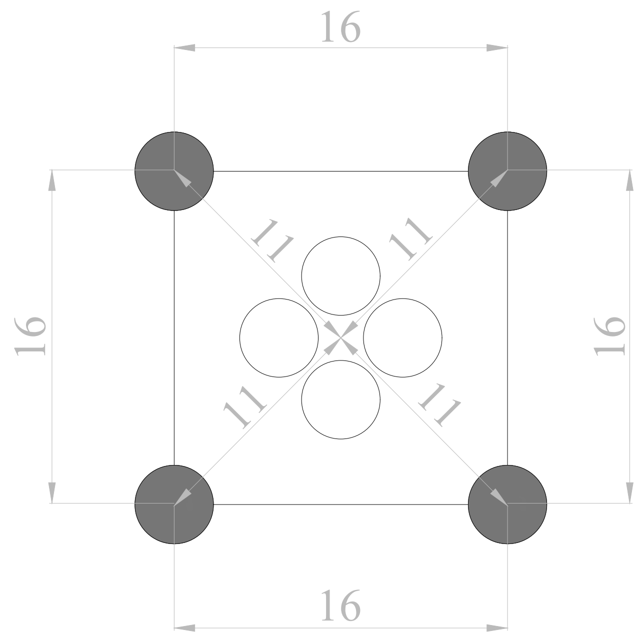

Figure 2.60 - Quoted scheme of the first square of the cut

It is then possible to proceed with the positioning of the second square. Once the side of the previous square is known, the burden of the next is determined, which in the specific case turns out to be equal to 16 cm. The second square of the cut will have a side of 34 cm:

$$L_2 = \sqrt{2} \cdot \left(16 + \frac{16}{2}\right) = 34\ cm \tag{2.7}$$

The empirical graph shown in Figure 2.61 was used to evaluate the burden of the second square and the linear charge concentration, knowing the side length of the first square of the cut.

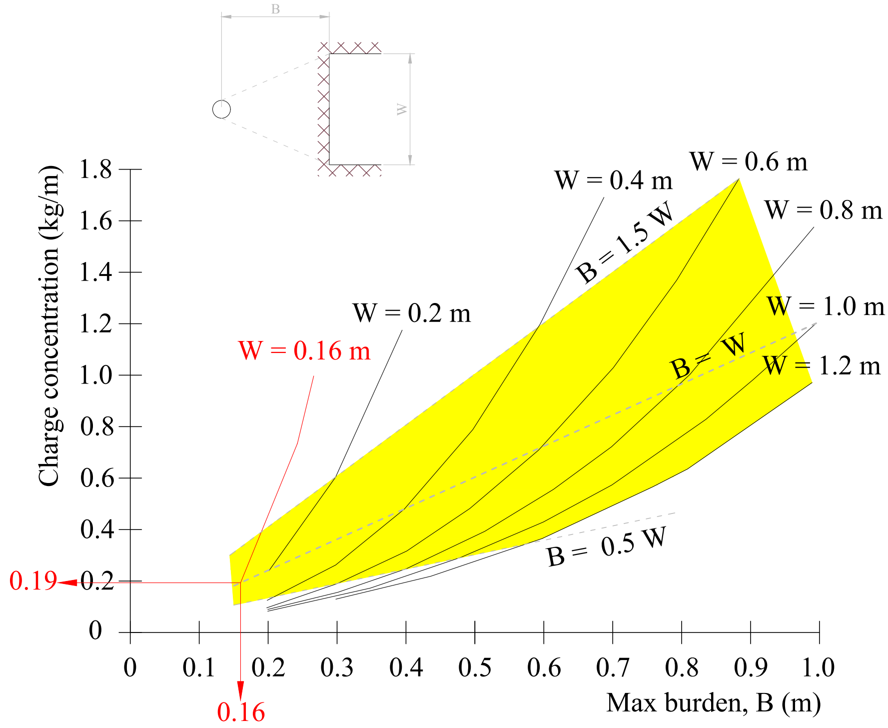

Figure 2.61 - Empirical graph used to determine the distance between the second square of the cut and the first (burden) and the linear charge concentration, knowing the value of the side of the first square of the cut

The scheme representing the second square of the cut and the relative dimensions is given in Figure 2.62.

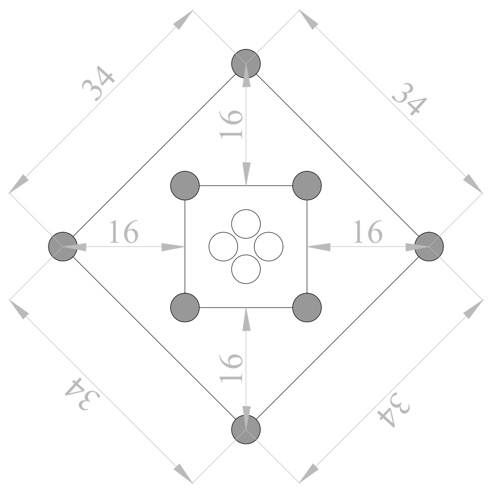

Figure 2.62 - Quoted scheme of the second square of the cut

At this point the insertion of a further square was considered, given the extent of the burden with respect to the section boundary. Being the burden very high, it was chosen to add another square: by knowing the length of the previous side, it was determined the burden of the next, which in the specific case turns out to be equal to 34 mm. The third square of the cut will have a side of 72 cm, as visible in the equation (2.8):

$$L_3 = \sqrt{2} \cdot \left(34 + \frac{34}{2}\right) = 72\ cm \tag{2.8}$$

The empirical graph shown in Figure 2.63 was used to find the burden between the blast-holes of the third square and those of the second, and the linear charge concentration:

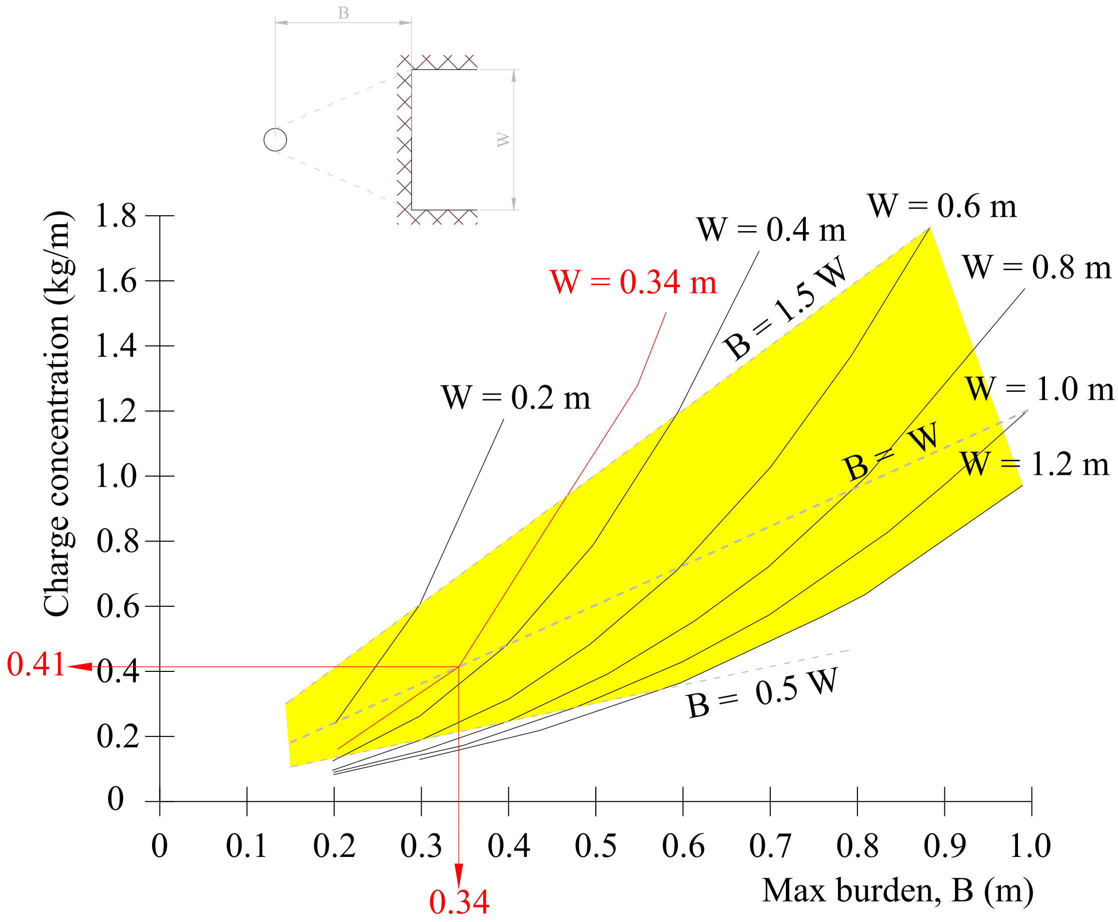

Figure 2.63 - Empirical graph used to determine the burden between the third square of the cut and the second and the linear charge concentration, knowing the value of the side of the second square of the cut.

The scheme representing the third scheme of the cut and the relative dimensions is given in Figure 2.64.

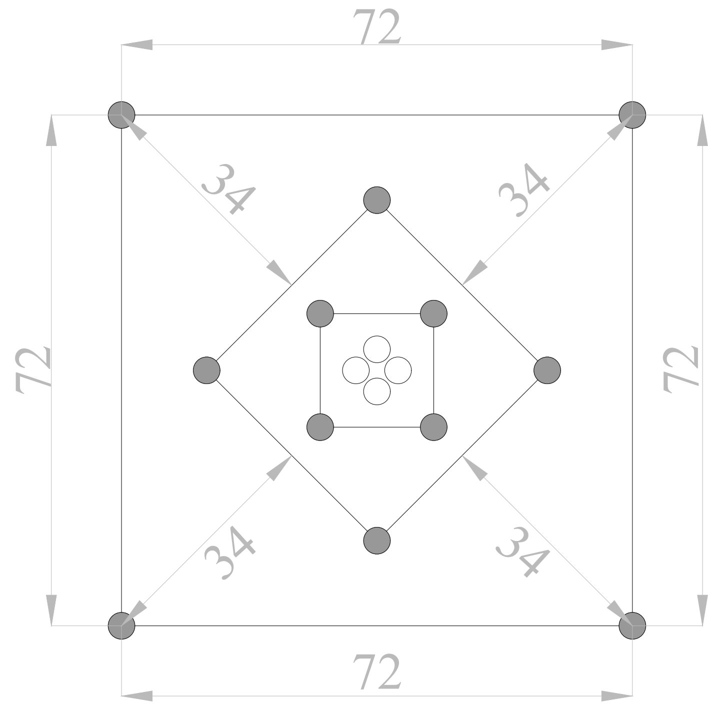

Figure 2.64 - Quoted scheme of the third square of the cut

From the empirical diagrams, the linear charge concentration obtained was respectively: 0.24 kg/m for the first square, 0.19 kg/m for the second and 0.41 kg/m for the third. The average linear concentration of charge for the cut turns out to be equal to 0.28 kg/m. Regarding the stemming, for the first square, its length is equal to the distance between the blast-holes and the central point of the empty holes.

For simplicity sake, it was decided to proceed with the same loading for all the blast-holes composing the cut. The explosive is Emultex CN®, in 1" x 8" cartridges placed at the bottom of the hole, and bulk ANFO premium® pneumatically loaded as column charge.

In the cut-holes, a cartridge of Emultex CN® is placed at the bottom hole, then the blast-holes are loaded for a length of 130 cm, leaving a stemming of 20 cm. 1.062 kg of ANFO and a cartridge of Emultex CN® weighing 0.119 kg are then introduced in each blast-hole: the total charge per hole, then, amounts to 1.181 kg. The charge concentration is equal to 0.79 kg/m, higher than that required and graphically determined. However, albeit apparently excessive, this charge is chosen, because:

- The perfect detonation of the ANFO is not guaranteed in such a small diameter (less than or equal to the critical diameter);
- ANFO does not work well in humid conditions;
- This procedure allows to avoid the intermediate stemming, which would have caused complexity and slowdown of loading.

A scheme of a charged cut-hole is shown in Figure 2.65.

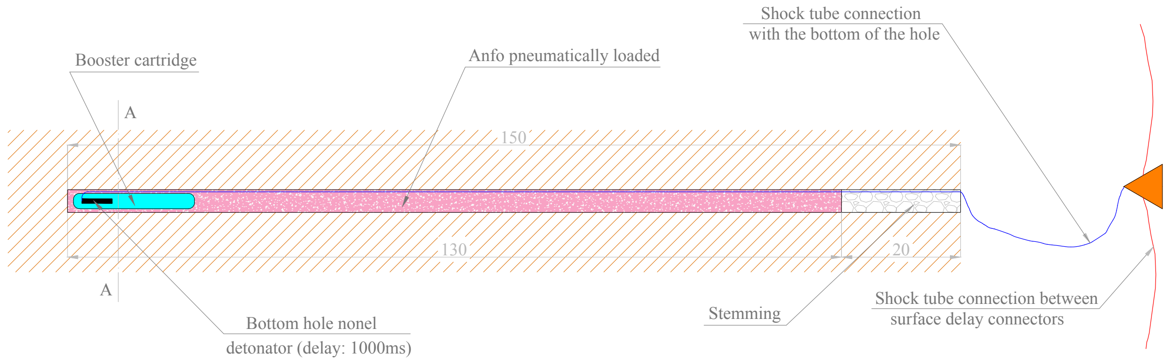

Figure 2.65 - Detail of a cut blast-hole with charge and trigger system

The next step pertains to the design of production holes: the linear charge concentration and the burden are defined for all production and contour blast-holes. In this case, the design of the blast is done considering the same explosive already used in the production cycle of the mine and employed for the cut-holes too.

Being known the diameter of the hole (38.1 mm), the linear charge concentration l_b (0.88 kg/m) and the burden B (0.83 m) can be easily found (Figure 2.66). These values will be used as input data for Table 2.15, which is needed for sizing both production and contour holes.

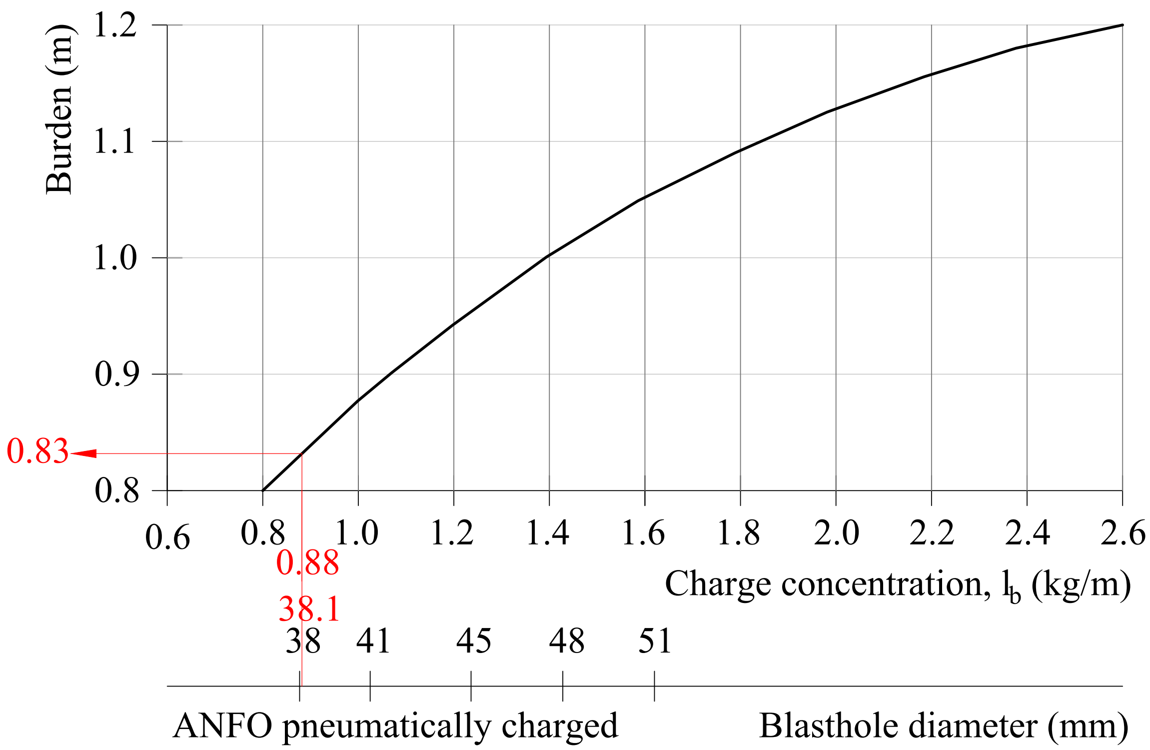

Figure 2.66 - Empirical graph used to determine the linear charge concentration ant the burden for production and contour holes for a 38.1 mm diameter hole charged with ANFO

As for the production blast-holes (upwards, horizontal, downwards), Table 2.15 shows that the burden is equal to B, previously determined on the experimental graph and therefore equal to 0.83 m. The spacing is 1.1 B for the upwards and horizontal holes (therefore, 0.91 cm), and 1.2 B (therefore, 1 m) for the downwards holes.

For simplicity sake, the design of the stoping holes has been defined by placing the holes around the cut with the same spacing.

When the cross-section is 2.5 m x 2.5 m, the distance between the cut and the contour of the tunnel is 88 cm. It was therefore decided to place the production holes exactly in the middle of this distance, so that the burden is 44 cm. The spacing, instead, is 81 cm.

The position of the production holes with respect to the cut is given in Figure 2.67.

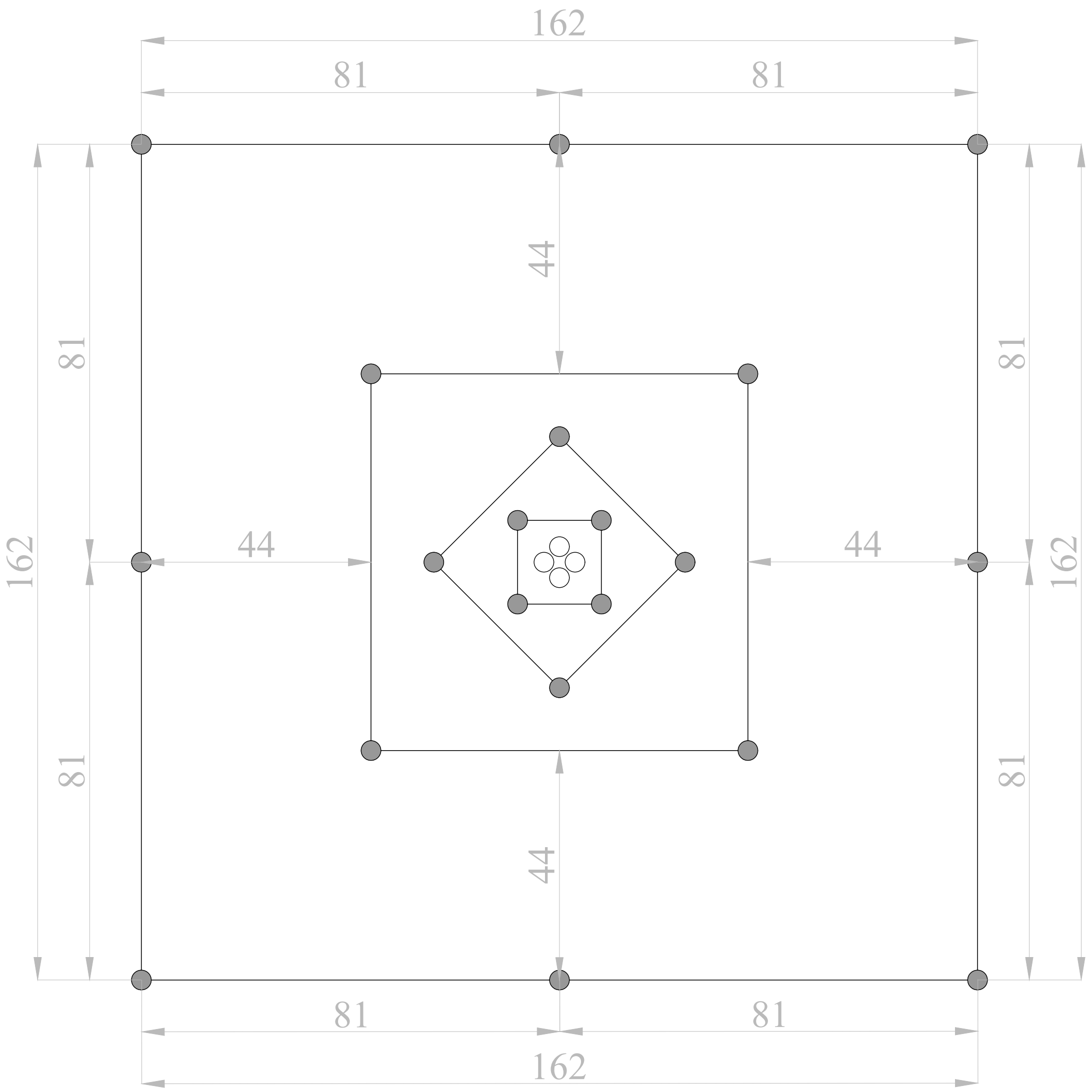

Figure 2.67 - Quoted scheme of the production holes for the section with dimension of 2.5 x 2.5 m

When the cross-section is 3.0 x 3.0 m, the distance between the cut and the tunnel contour is 114 cm. The production holes were placed at the centre of this distance, so that the burden is 57 cm, as represented in Figure 2.68. The spacing between the holes is 93 cm.

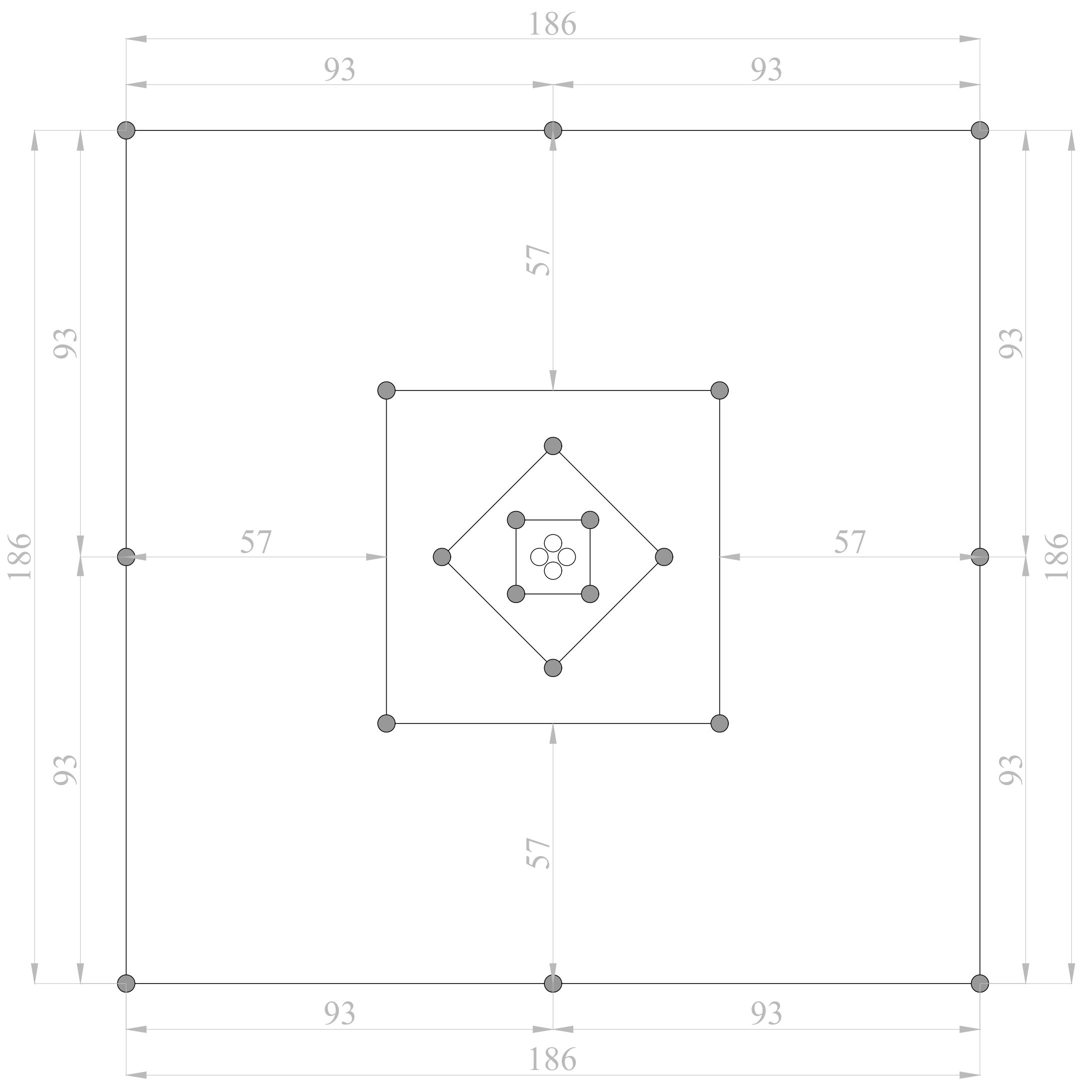

Figure 2.68 - Quoted scheme of the production holes for the section with dimension of 3.0 x 3.0 m

As for the third case (cross-section 3.5 x 3.5 m), the distance between the cut and the outline of the section is 138 cm. The production holes, located at the centre of this distance, have a burden of 69 cm. The spacing is 106 cm. This value is higher than that suggested in Table 2.15, but it is acceptable because the measure of the burden is lower than the data suggested by the same table.

The position of the production holes is shown in Figure 2.69.

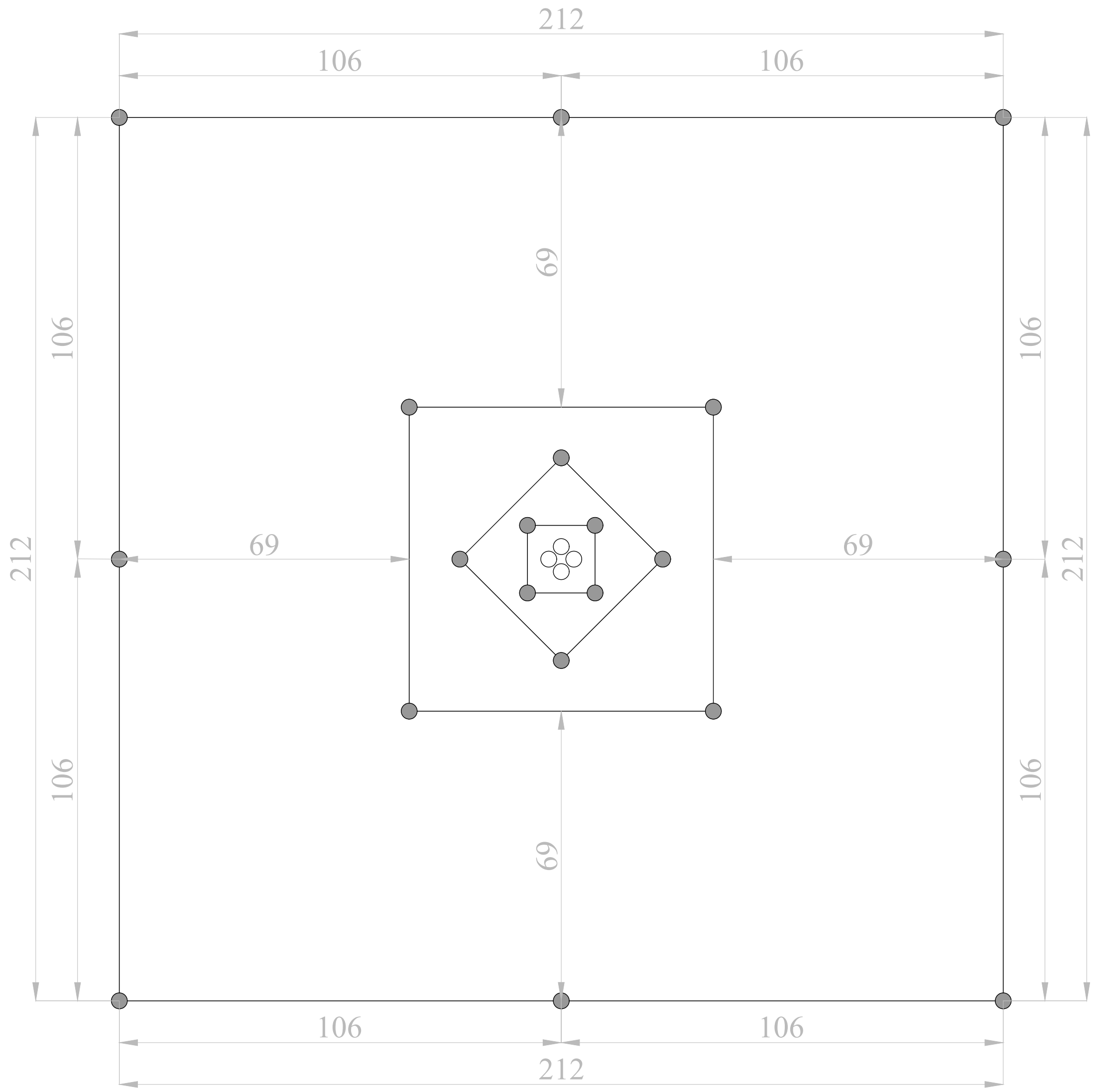

Figure 2.69 - Quoted scheme of the production holes for the section with dimension of 3.5 x 3.5 m

The charging of the stoping holes is the same for all positions with respect to the cut (upwards, horizontal, downwards). This was found based on Table 2.15. The length of the bottom charge is roughly 1/3 of the hole length (in this case 0.5 m) with a linear charge concentration l_b of 0.88 kg/m. The column charge is instead 0.5 l_b, hence 0.44 kg/m. The length of the stemming is 0.5 times the burden, and therefore 0.41 m.

The charging of the production holes was dimensioned by placing two Emultex CN® cartridges at the bottom hole (dimension 1" x 8"), pneumatically placing 110 cm of ANFO premium® and leaving the stemming with a length of 40 cm. In total, 0.238 kg of Emultex CN® (linear concentration of the bottom charge: 0.585 kg/m) and 0.807 kg of ANFO premium® (linear concentration of the column charge: 0.734 kg/m) are used. In total, therefore, 1.045 kg of explosives are employed, with a linear charge concentration which is therefore of about 0.7 kg/m.

A scheme of a charged blast-hole pertaining to the stoping group is shown in Figure 2.70.

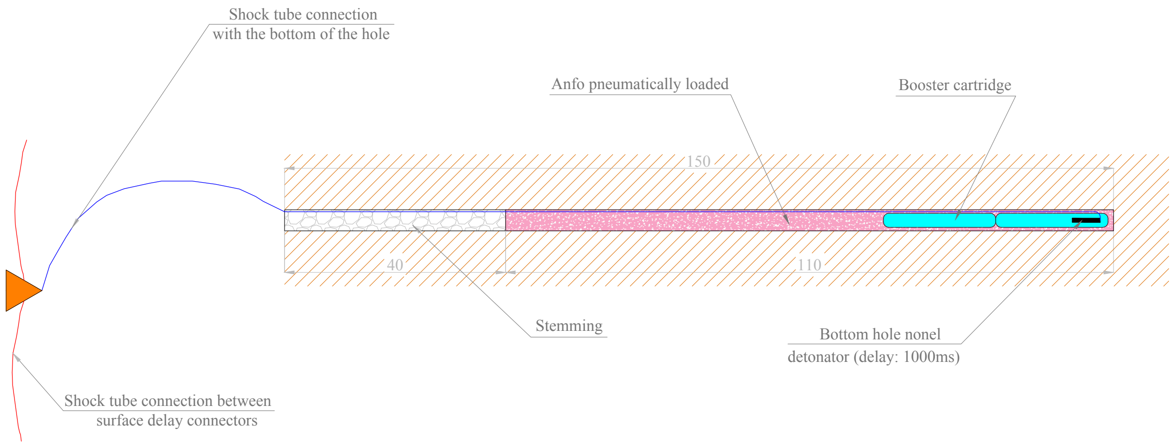

Figure 2.70 - Detail of a production blast-hole with charge and trigger system

As for the contour holes, the same explosive already used for the other functional groups is employed, such as Emultex CN® at the bottom hole and ANFO premium® in the column. Using the same Table 2.15, with the input data derived from the empirical graph, the burden and the data related to charging are determined. The contour is divided into 3 functional groups of blast-holes: floor, with burden equal to B (0.83 m), walls and roof, with burden equal to 0.9 B (0.75 m). Spacing is the same for the three groups and is 1.1 B (0.91 m).

When the cross section is 2.5 x 2.5 m, 4 blast-holes have been placed along the contour (walls and roof) and the floor: the spacing will therefore be equal to 83 cm while the burden is equal to 44 cm.

The position of the contour holes is shown in Figure 2.71.

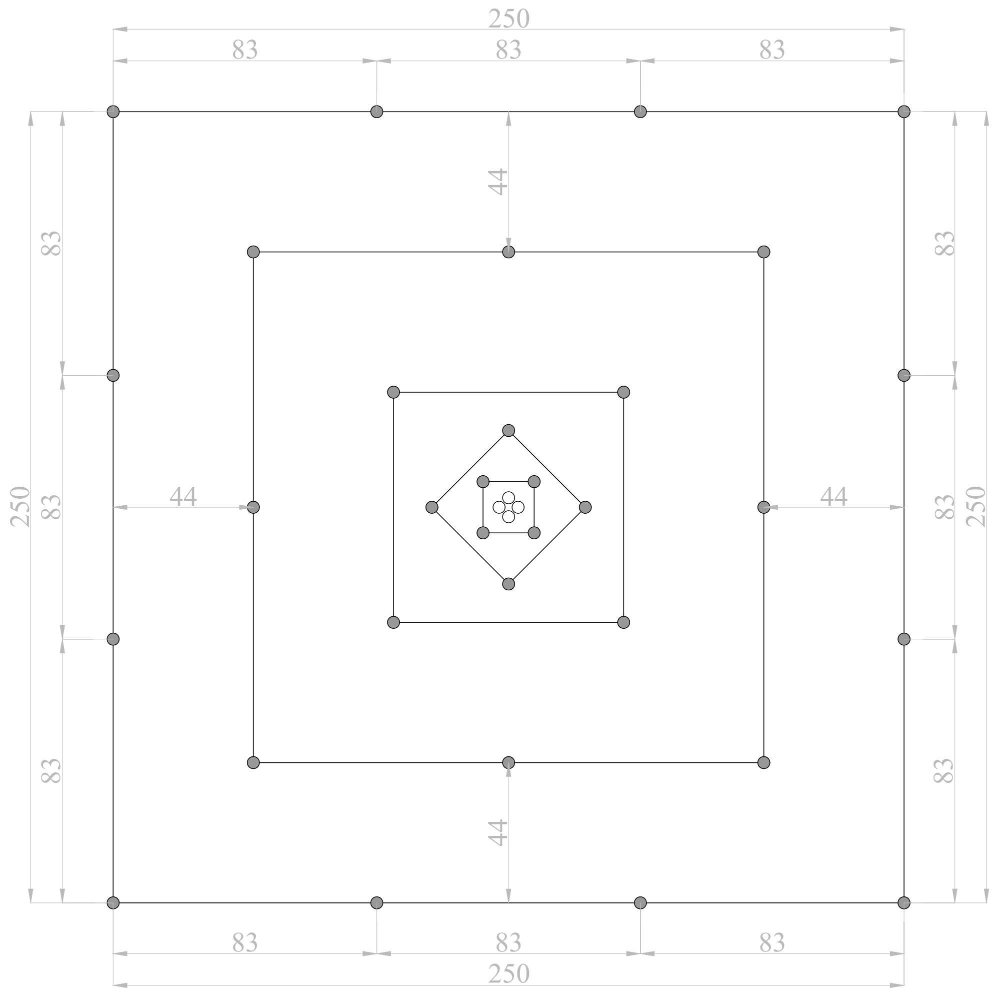

Figure 2.71 - Quoted scheme of the contour holes for the section with dimension of 2.5 x 2.5 m

When the cross section is 3.0 x 3.0 m, as shown in Figure 2.72, 5 blast-holes were positioned, with a spacing of 75 cm and a burden of 57 cm.

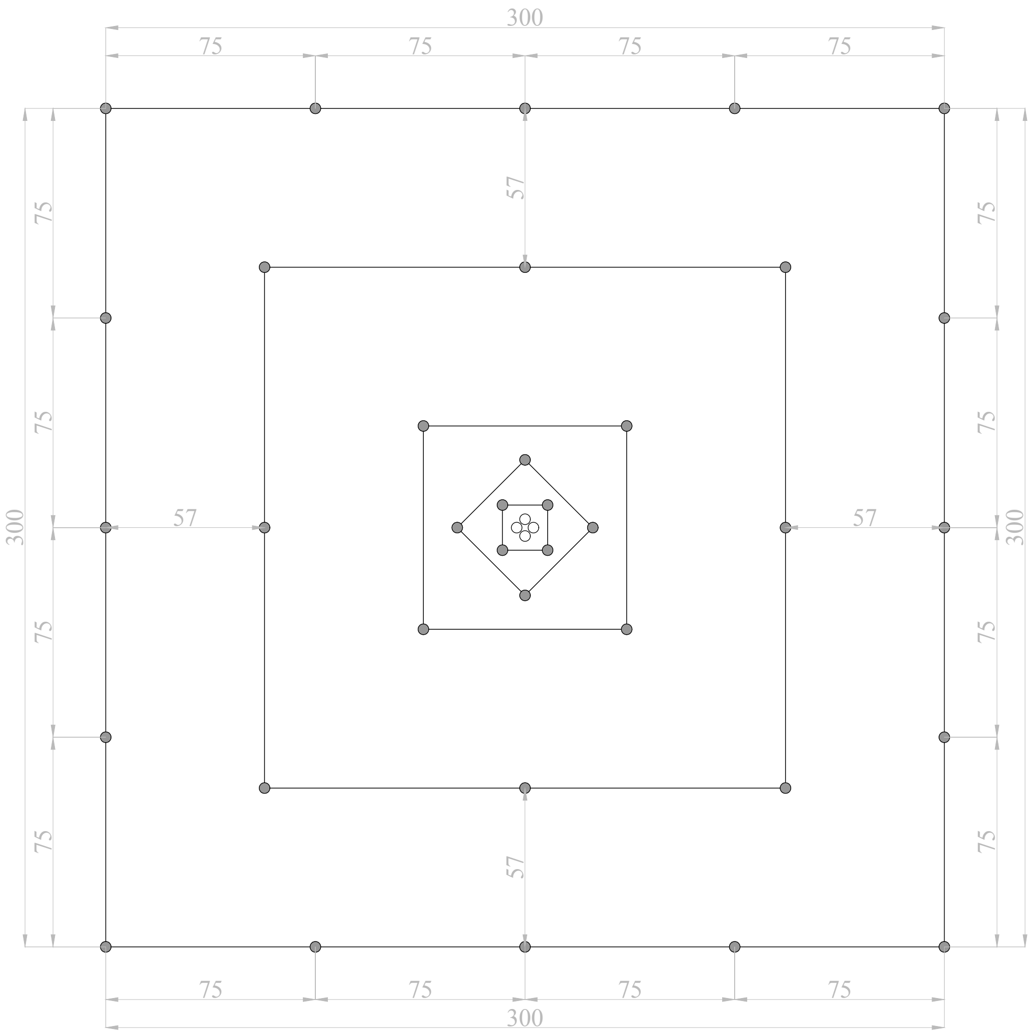

Figure 2.72 - Quoted scheme of the contour holes for the section with dimension of 3.0 x 3.0 m

As for the 3.5 x 3.5 m cross-section, 5 blast-holes were positioned, with a spacing of 87 cm and a burden of 69 cm. The scheme of the contour holes is shown in Figure 2.73.

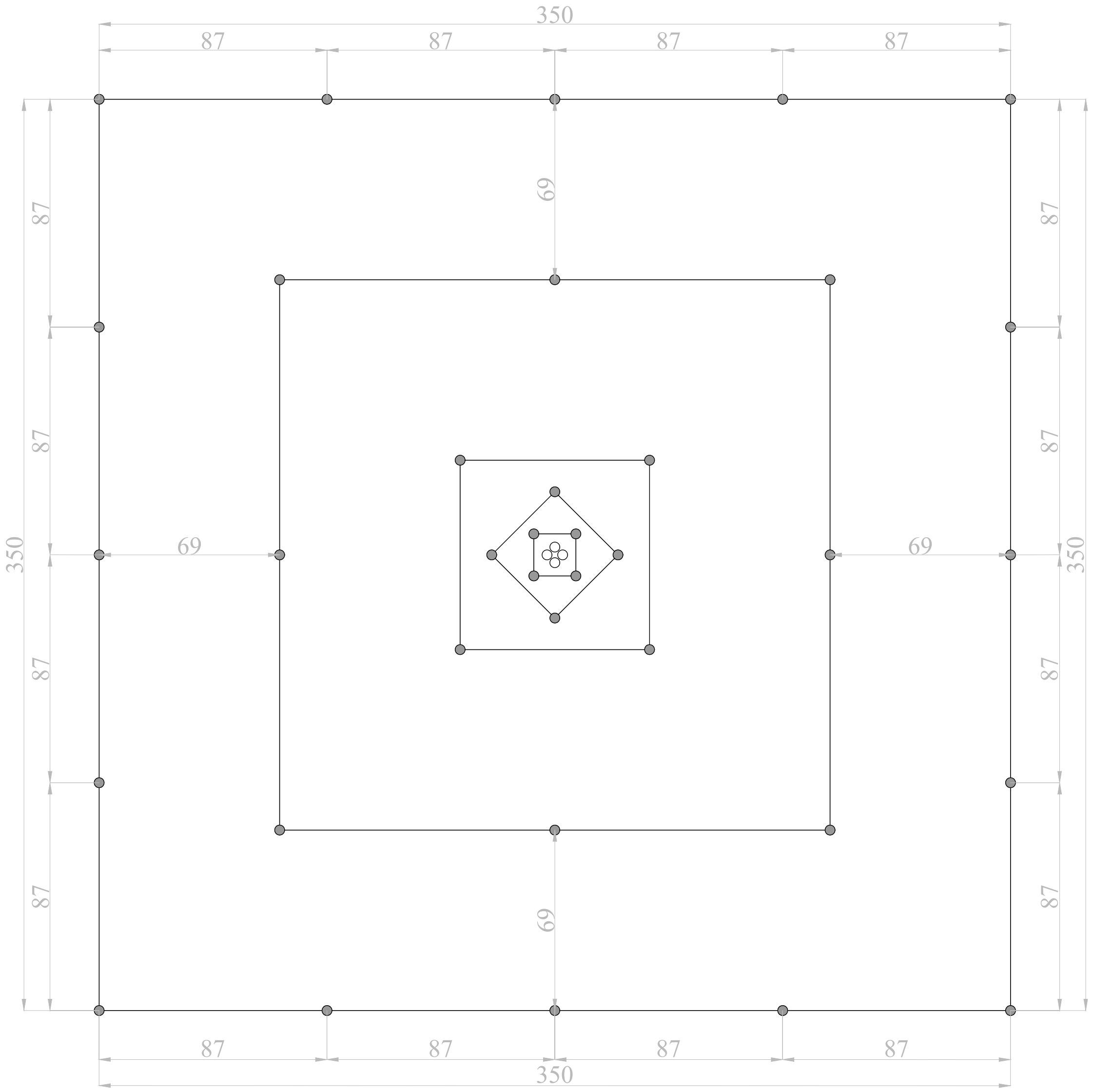

Figure 2.73 - Quoted scheme of the contour holes for the section with dimension of 3.5 x 3.5 m

As for the charging, Table 2.15 has been used again. Within the functional group of contour blast-holes, three distinct subgroups are identified: floor, walls and roof. The length of the bottom charge for the contour holes in the floor is about 1/3 of the total length of the hole (therefore 0.5 m). The concentration of the bottom charge is l_b (0.88 kg/m), equal to that of the column. The stemming length is 0.2 times the burden, i.e. 0.17 m.

Finally, as for the roof, the bottom charge is 1/6 of the hole length (0.25 m), with a linear charge concentration l_b of 0.88 kg/m. The column charge has a linear concentration of 0.3 l_b, therefore 0.26 kg/m. The stemming is equal to half the burden and therefore 0.41 cm.

Basically, for design purposes, the contour holes were divided into two groups depending on the loading: blast-holes related to the floor and blast-holes related to walls and roof.

The loading of the floor was defined by placing two cartridges of Emultex CN® (size 1" x 8") at the bottom of the hole and loading the hole with ANFO premium® for a length of 130 cm. The stemming is 20 cm. Totally, 0.238 kg of Emultex CN® (linear charge concentration of 0.59 kg/m) and 1.141 kg of ANFO premium® (linear charge concentration of 0.88 kg/m) are placed in the hole. The amount of charge that was globally inserted into the hole is then 1.379 kg, with a linear charge concentration of 0.92 kg/m.

A scheme of a floor hole and its loading is shown in the Figure 2.74.

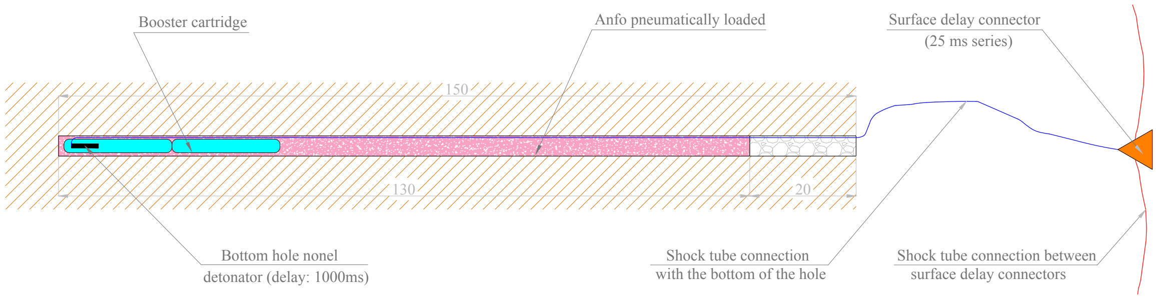

Figure 2.74 - Detail of a contour (floor) blast-hole with charge and trigger system

The charging of the blast-holes of the walls and the roof has been defined differently compared to the floor-holes. Only 1 Emultex CN® cartridge was inserted at the bottom of the hole (0.119 kg, with a charge concentration of 0.59 kg/m). The charging is completed with 110 cm of ANFO premium® loaded pneumatically and with 40 cm of stemming. An amount of ANFO of 0.886 kg (charge concentration 0.80 kg/m) is introduced into the hole. The total charge placed in the hole is then 1.00 kg with a linear charge concentration of 0.67 kg/m.

The scheme of a contour hole (walls and roof) and its loading is shown in Figure 2.75.

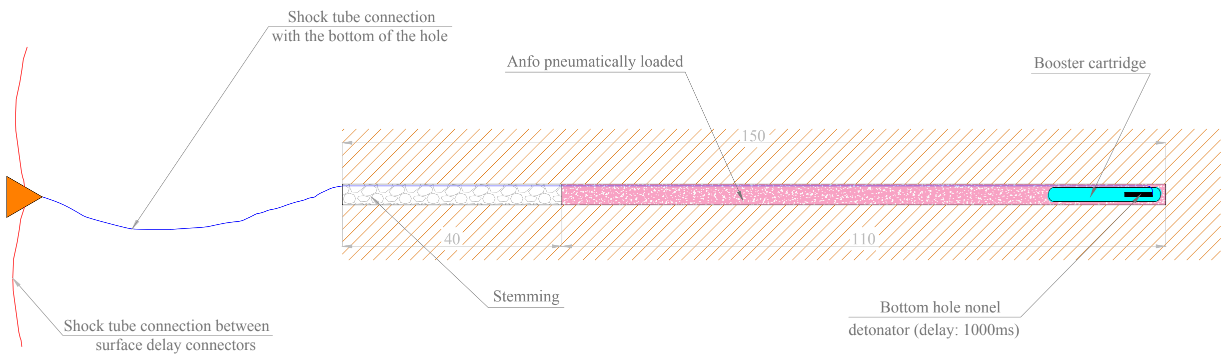

Figure 2.75 - Detail of a contour (walls and roof) blast-hole with charge and trigger system

Figure 2.76, Figure 2.77, Figure 2.78 show the complete drilling schemes for the three excavation cross-section considered in phase 1.

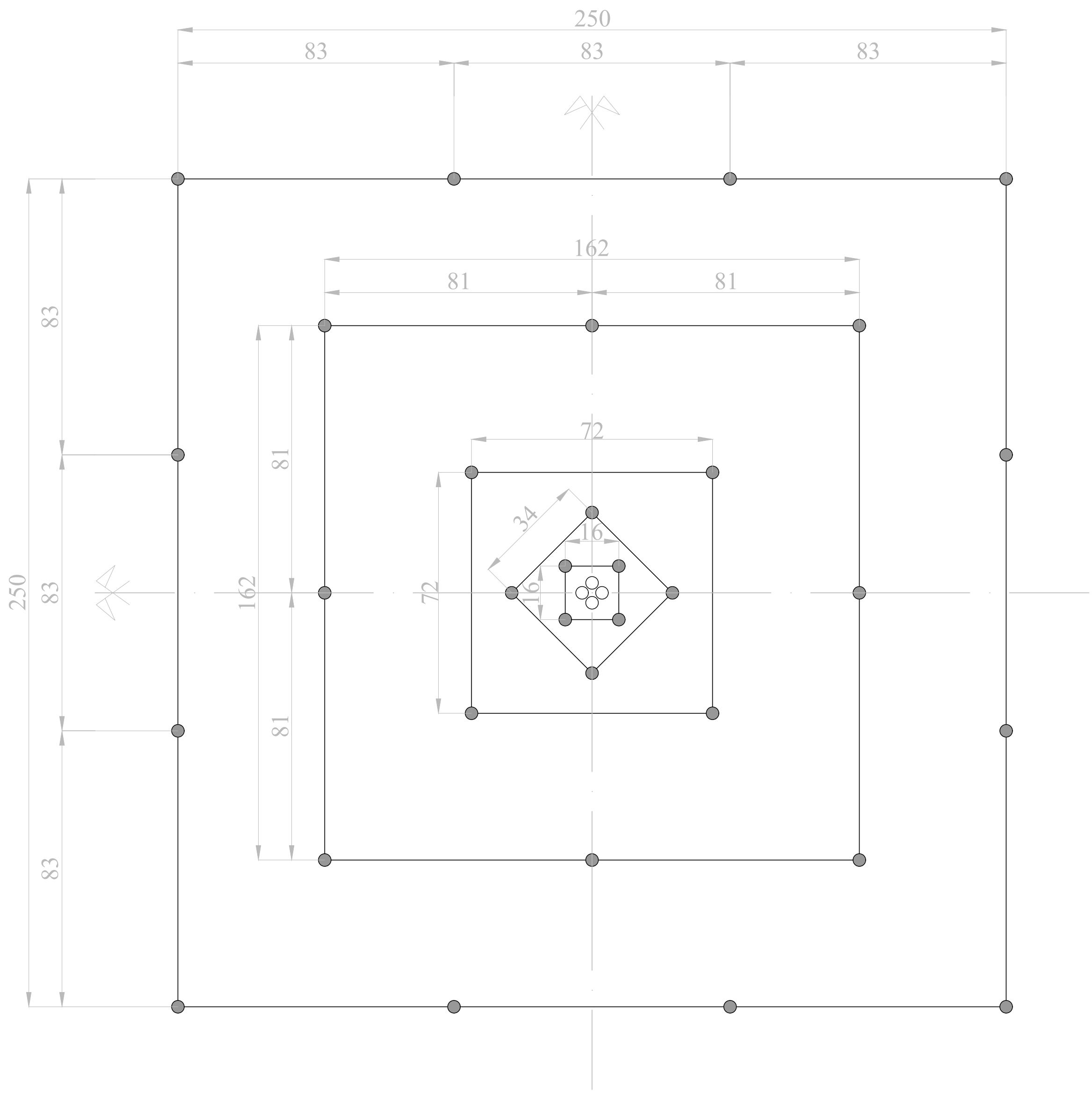

Figure 2.76 – Complete drilling scheme for the excavation section of 2.5 m x 2.5 m

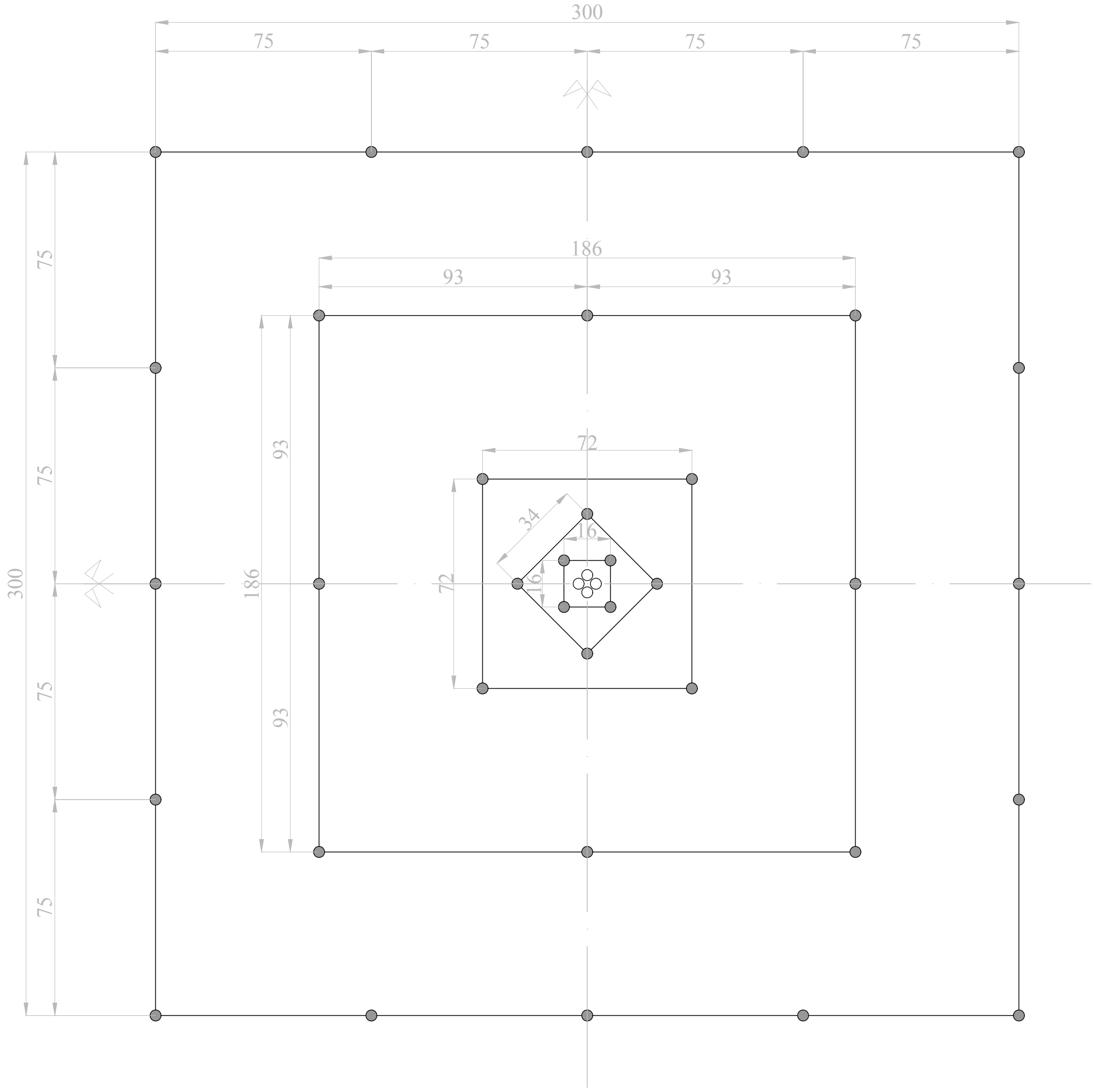

Figure 2.77 - Complete drilling scheme for the excavation section of 3.0 m x 3.0 m

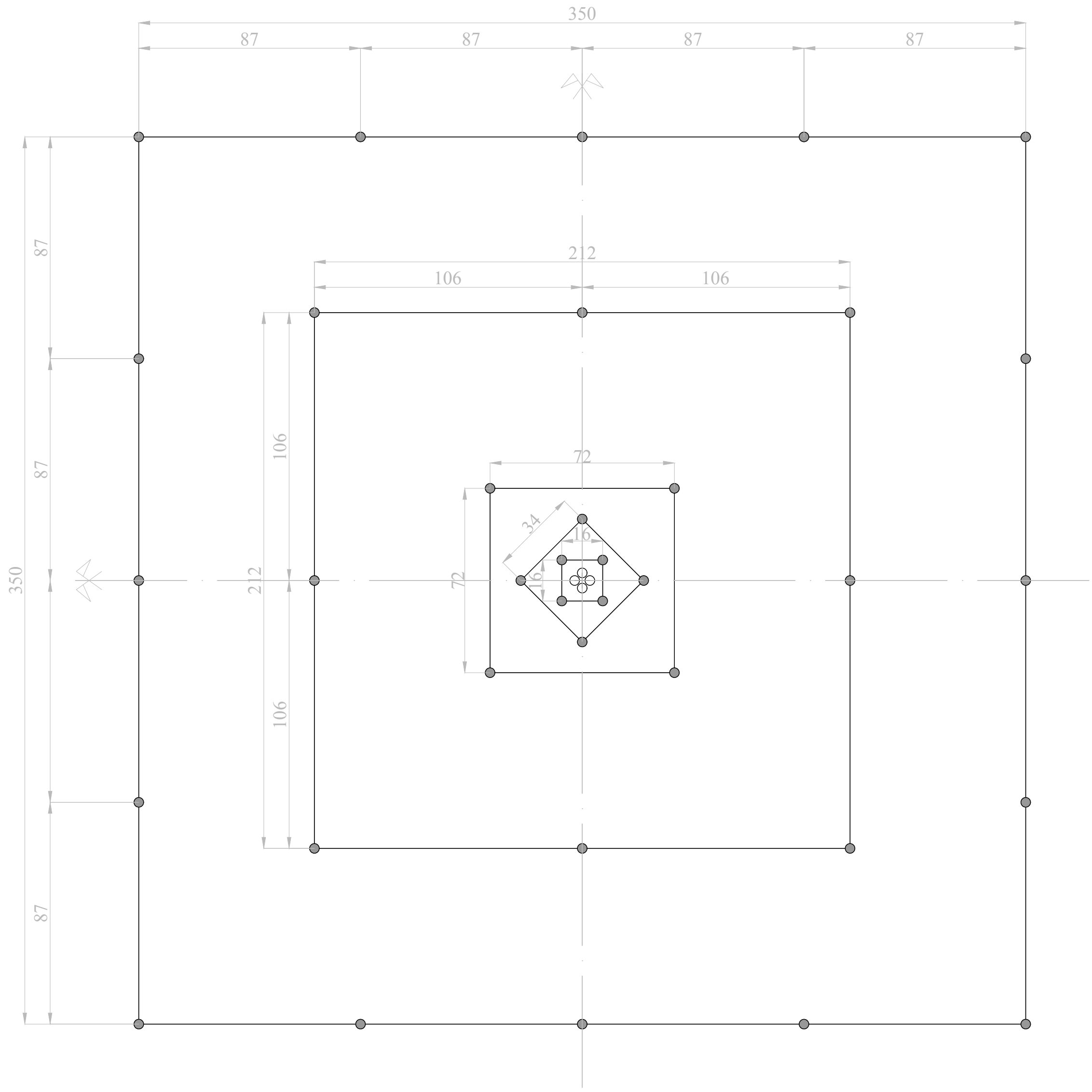

Figure 2.78 - Complete drilling scheme for the excavation section of 3.5 m x 3.5 m

Once the drilling scheme and the holes charging have been defined, the triggering order and the length of the nonel cables must be defined.

The sequence of triggering was defined following the concepts shown in Figure 2.57. The cut detonates first, then the production and the contour holes (first those of the floor, then the walls and the roof and finally the corners at the bottom).

Considering the maximum distances that nonel cables must cover in different cases and for the various cross-sections adopted, considering also a safety margin to take into account possible inaccuracies due to drilling and the fact that the nonel tubes must not be taut but loosened, a cable length of the detonators (non-electric detonators and non-electric dual detonators) of 3.6 m was choosen.

In the following schemes, the detonation orders are shown: Figures 2.28, 2.30 and 2.32 show the triggering order of the surface delays connectors. It is also possible to observe the connection system between the various holes and measure the length of the connection tubes. Figures 2.29, 2.31 and 2.33 show the initiation of the in-hole detonators. The yellow arrows are a graphic aid that facilitates the reading of the scheme, representing the holes that detonate in succession.

Both Figure 2.79 and Figure 2.80 are related to the 2.5 m x 2.5 m cross-section, Figure 2.81 and Figure 2.82 to the 3.0 x 3.0 m cross-section and, finally, Figure 2.83 and Figure 2.84 to the 3.5 x 3.5 m cross-section.

For each dimension of the excavation section a table containing the most significant data concerning the initiation system is presented (for each phase of the detonation order are reported the delay of the surface connector, the delay of the in-hole detonator, the number of the surface connectors and the number of the in-in hole detonators).

Table 2.16 is related to the 2.5 x 2.5 section, Table 2.17 to the 3.0 x 3.0 m section and, finally, Table 2.18 to the 3.5 x 3.5 m section.

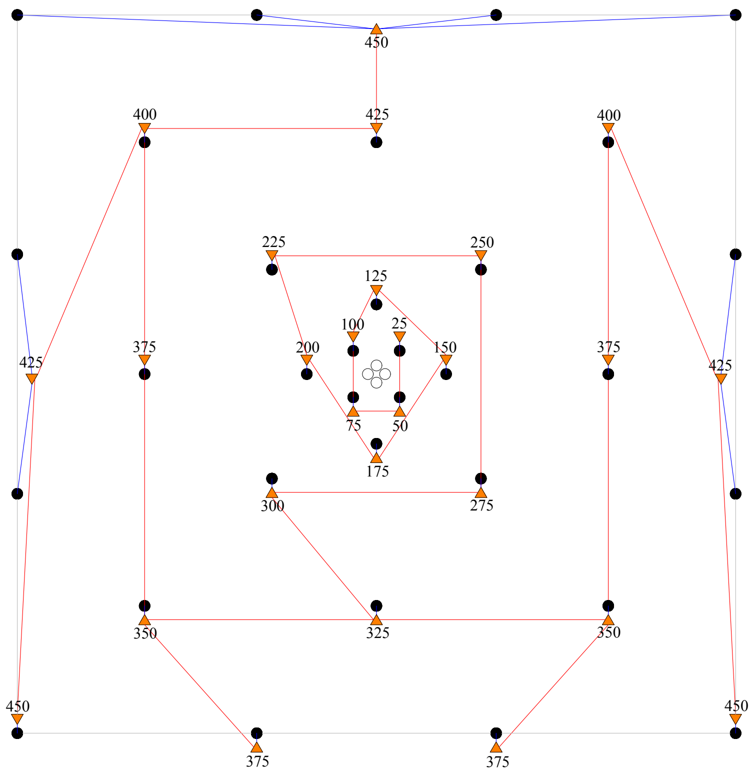

Figure 2.79 – Order of detonation of the surface delay connectors related to the 2.5 x 2.5 m section

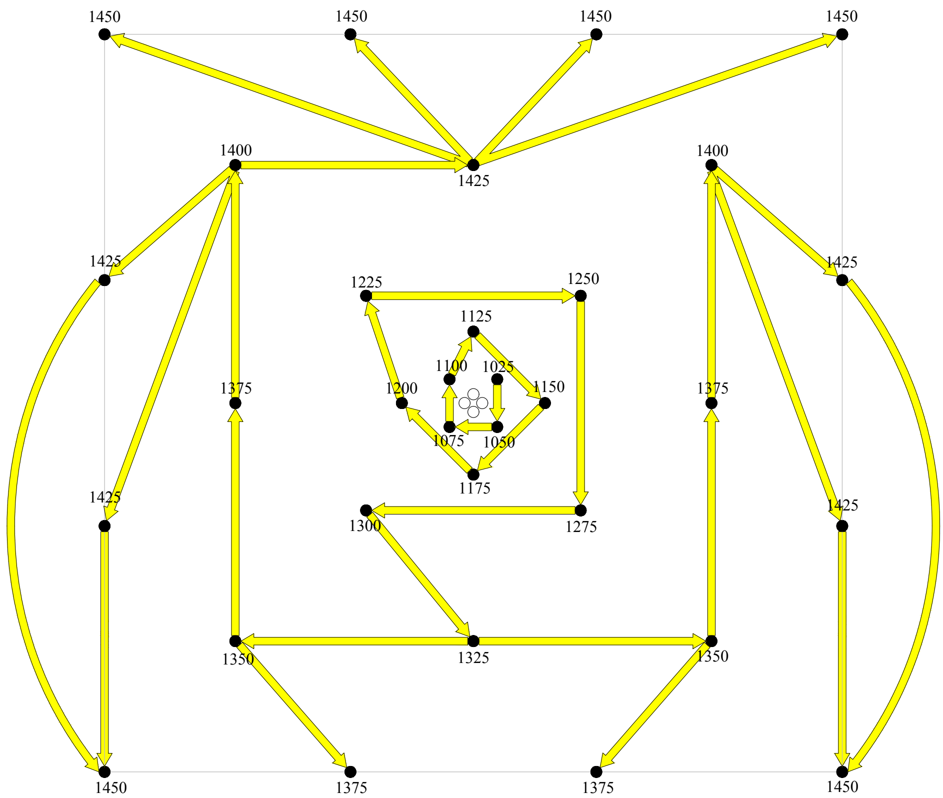

Figure 2.80 - Order of detonation of the in-hole detonators related to the 2.5 x 2.5 m section

Table 2.16 - Summary table of the main data related to the initiation system for the 2.5 x 2.5 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	1	1
11	25 ms	1000 ms	1275 ms	1	1
12	25 ms	1000 ms	1300 ms	1	1
13	25 ms	1000 ms	1325 ms	1	1
14	25 ms	1000 ms	1350 ms	2	2
15	25 ms	1000 ms	1375 ms	4	4
16	25 ms	1000 ms	1400 ms	2	2
17	25 ms	1000 ms	1425 ms	3	5
18	25 ms	1000 ms	1450 ms	3	6

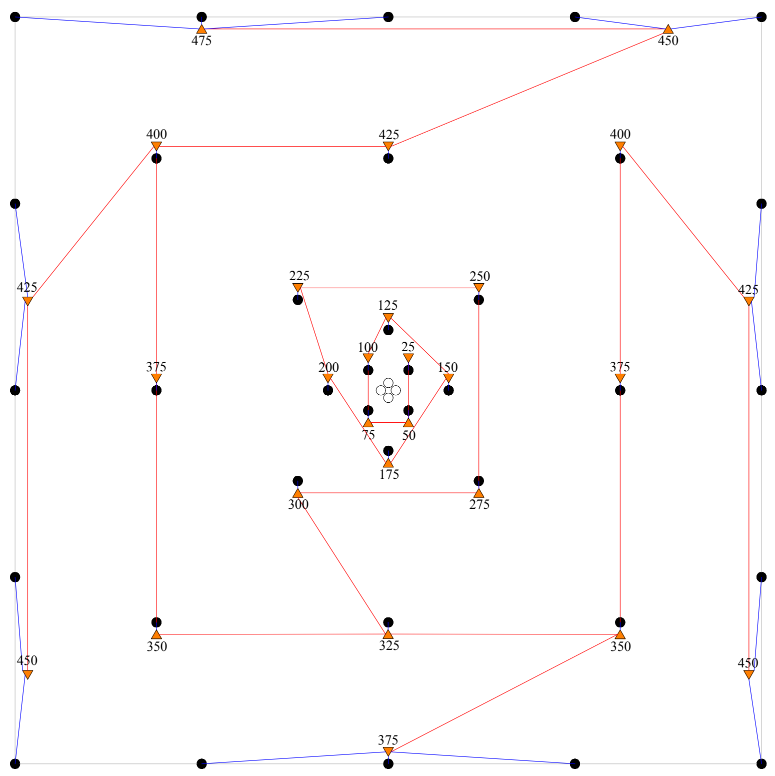

Figure 2.81 - Order of detonation of the surface delay connectors related to the 3.0 x 3.0 m section

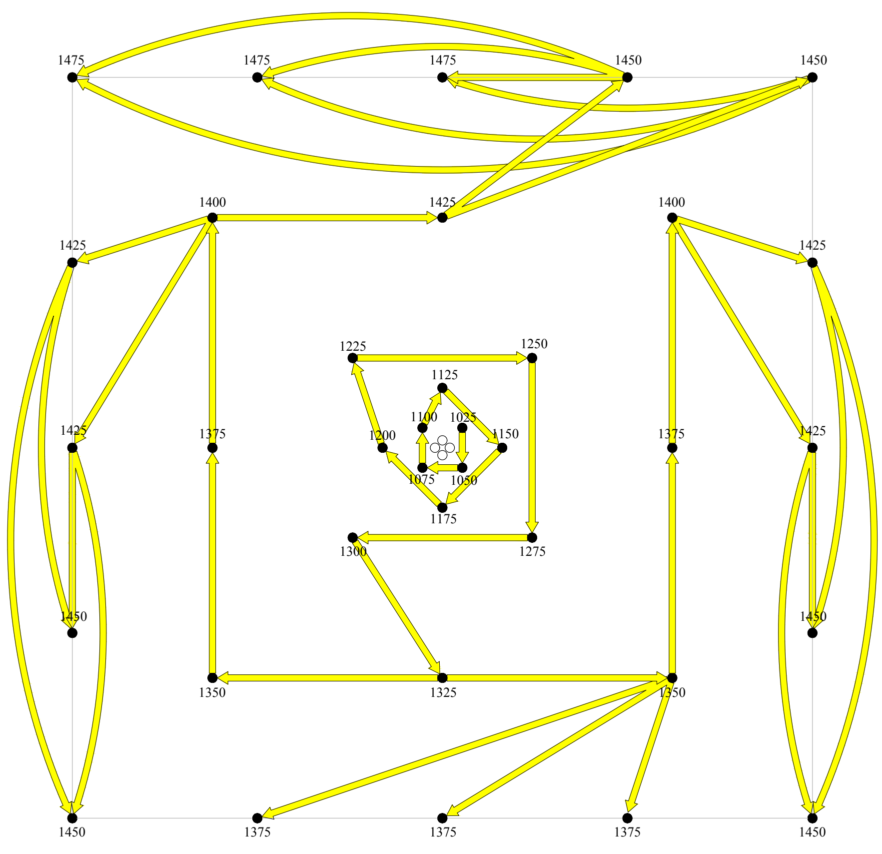

Figure 2.82 - Order of detonation of the in-hole detonators related to the 3.0 x 3.0 m section

Table 2.17 - Summary table of the main data related to the initiation system for the 3.0 x 3.0 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	1	1
11	25 ms	1000 ms	1275 ms	1	1
12	25 ms	1000 ms	1300 ms	1	1
13	25 ms	1000 ms	1325 ms	1	1
14	25 ms	1000 ms	1350 ms	2	2
15	25 ms	1000 ms	1375 ms	3	5
16	25 ms	1000 ms	1400 ms	2	2
17	25 ms	1000 ms	1425 ms	3	5
18	25 ms	1000 ms	1450 ms	3	6
19	25 ms	1000 ms	1475 ms	1	3

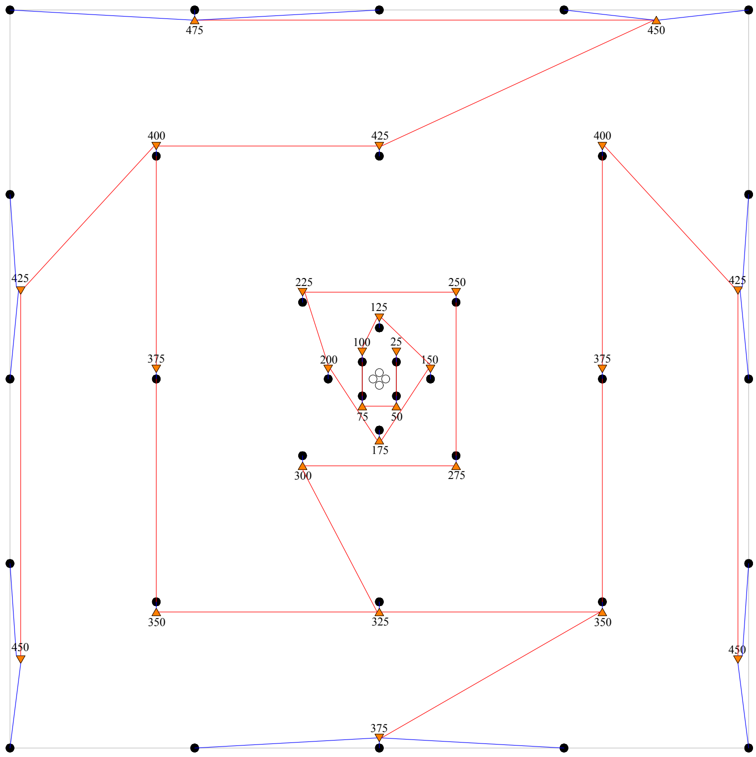

Figure 2.83 - Order of detonation of the surface delay connectors related to the 3.5 x 3.5 m section

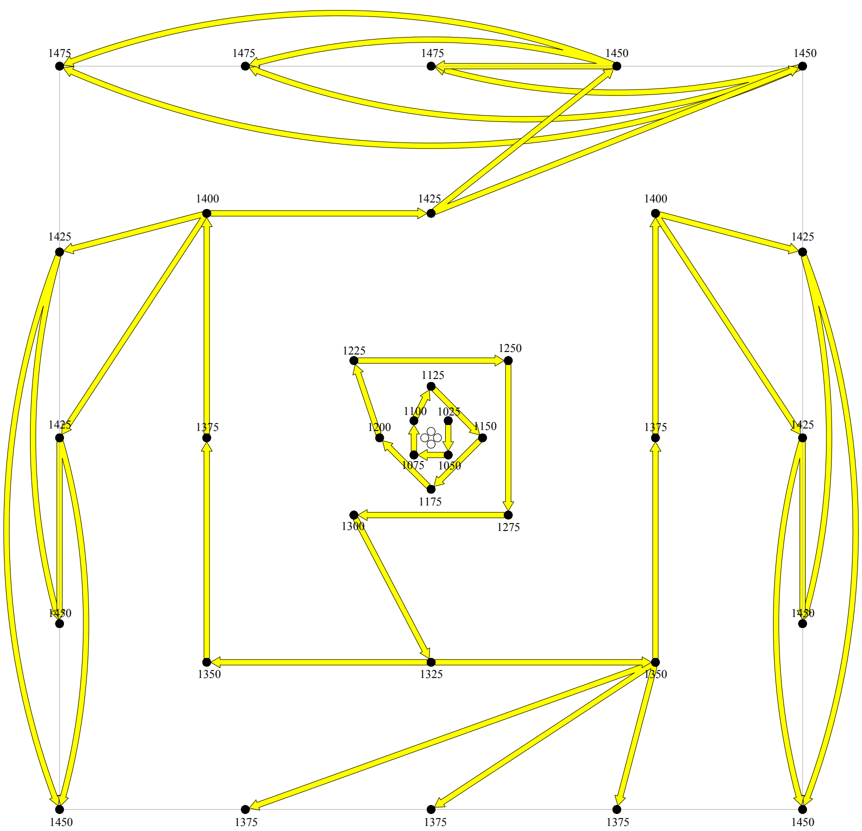

Figure 2.84 - Order of detonation of the in-hole detonators related to the 3.5 x 3.5 m section

Table 2.18 - Summary table of the main data related to the initiation system for the 3.5 x 3.5 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	1	1
11	25 ms	1000 ms	1275 ms	1	1
12	25 ms	1000 ms	1300 ms	1	1
13	25 ms	1000 ms	1325 ms	1	1
14	25 ms	1000 ms	1350 ms	2	2
15	25 ms	1000 ms	1375 ms	3	5
16	25 ms	1000 ms	1400 ms	2	2
17	25 ms	1000 ms	1425 ms	3	5
18	25 ms	1000 ms	1450 ms	3	6
19	25 ms	1000 ms	1475 ms	1	3

The following tables (Table 2.19 for the 2.5 x 2.5 m cross-section, Table 2.20 for the 3.0 x 3.0m cross-section and, finally, Table 2.21 for the 3.5 x 3.5 m cross-section) summarize the main data concerning the blast for each section size. The calculated data are relative to the whole blast and also to the individual functional groups of blast-holes. Specifically, the most important data are: volume blasted, amount of explosive, powder factor, volume to be removed (using a bulking factor equal to 1.4), holes density, total drilled length, specific drilling, detonators specific consumption. The volumes of competence of each functional group have been highlighted, for each dimension of the cross-section, in the following figures: Figure 2.85 (2.5 x 2.5 m cross-section), Figure 2.86 (3.0 x 3.0 m cross-section), Figure 2.87 (3.5 x 3.5 m cross-section).

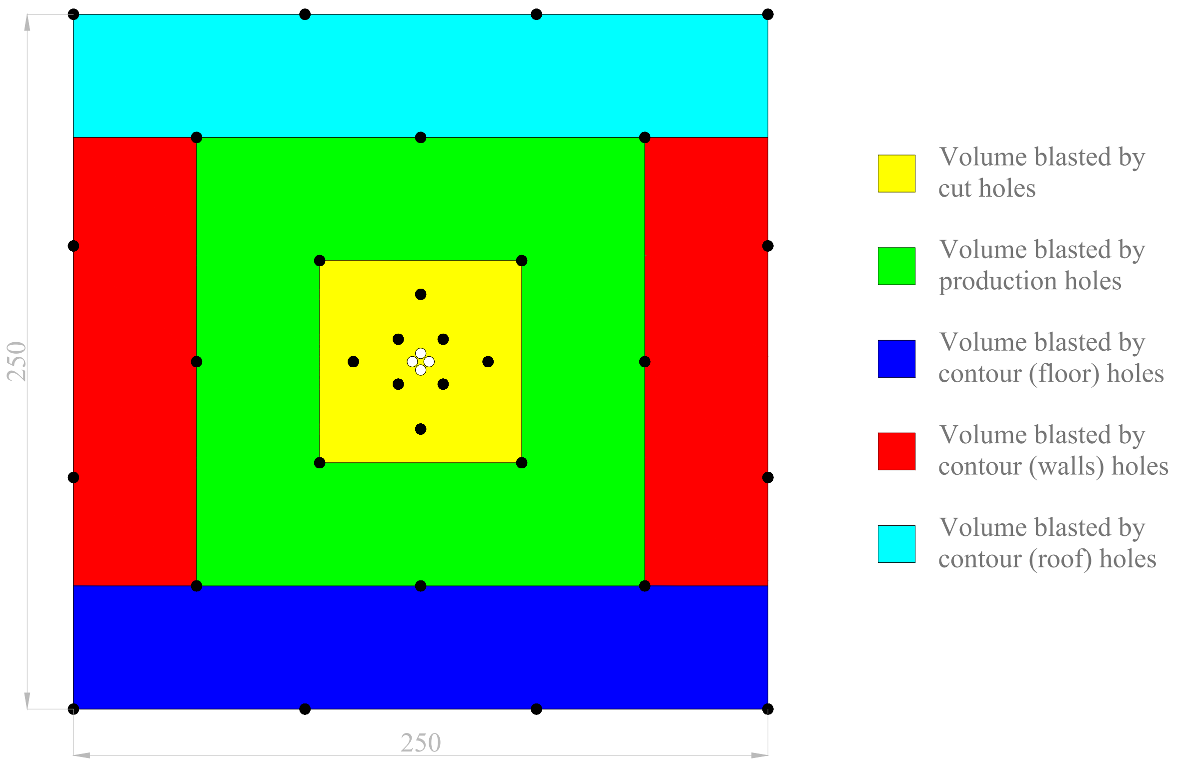

Figure 2.85 - Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 2.5 x 2.5 m section

Table 2.19 - Summary of the main data relating to the excavation section of 2.5 x 2.5 m

Cross section (m^2)		6.25
Design pull (m)		1.5
Hole diameter (m)		38.1
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.771
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	3.102
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	2.899
Contour (floor) holes	Volume blasted (m^3)	1.657
	Amount of explosive (kg)	5.200
	Powder factor (P.F.) (kg/m^3)	3.139
Contour (walls) holes	Volume blasted (m^3)	2.138
	Amount of explosive (kg)	4.020
	Powder factor (P.F.) (kg/m^3)	1.880
Contour (roof) holes	Volume blasted (m^3)	1.657
	Amount of explosive (kg)	4.020
	Powder factor (P.F.) (kg/m^3)	2.427
Volume blasted by contour holes (walls + roof) (m^3)		3.795
Amount of explosive for the contour holes (walls + roof) (kg)		8.040
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		2.118
Volume blasted by contour holes (floor + walls + roof) (m^3)		5.452
Amount of explosive for the contour holes (floor + walls + roof) (kg)		13.240
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		2.429
Total blasted volume (m^3)		9.325
Volume to be removed (m^3)		13.055
Total amount of explosive (kg)		36.404
Overall powder Factor (kg/m^3)		3.904
Holes number (-)		36
Holes density ($holes/m^2$)		5.76
Total drilled length (m)		54
Specific drilling (m/m^3)		5.791
Total number of detonators (-)		59
Detonators specific consumption (D.C.) ($detonators/m^3$)		6.33

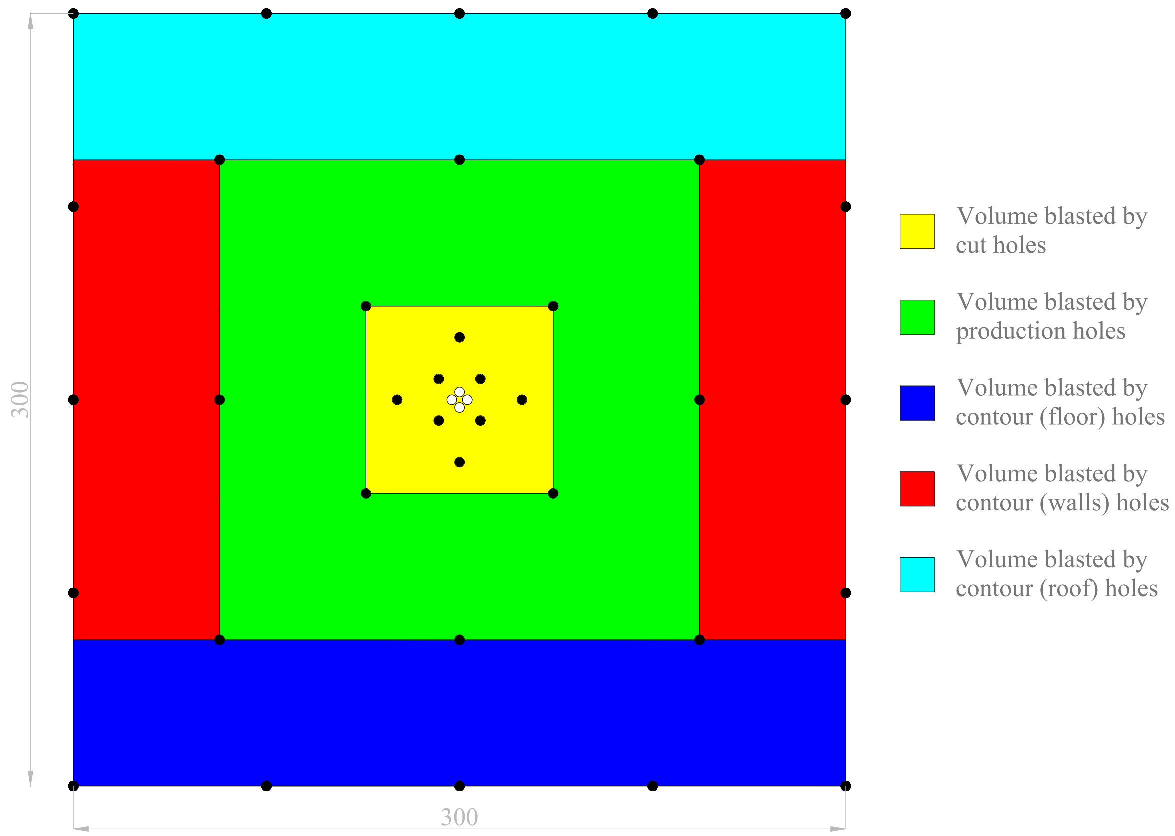

Figure 2.86 - Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 3.0 x 3.0 m section

Table 2.20 - Summary of the main data relating to the excavation section of 3.0 x 3.0 m

Cross section (m^2)		9.00
Design pull (m)		1.5
Hole diameter (m)		38.1
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.771
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	4.406
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	2.041
Contour (floor) holes	Volume blasted (m^3)	2.564
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	2.535
Contour (walls) holes	Volume blasted (m^3)	3.891
	Amount of explosive (kg)	6.030
	Powder factor (P.F.) (kg/m^3)	1.891
Contour (roof) holes	Volume blasted (m^3)	2.551
	Amount of explosive (kg)	5.025
	Powder factor (P.F.) (kg/m^3)	1.970
Volume blasted by contour holes (walls + roof) (m^3)		5.739
Amount of explosive for the contour holes (walls + roof) (kg)		11.055
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		1.970
Volume blasted by contour holes (floor + walls + roof) (m^3)		8.304
Amount of explosive for the contour holes (floor + walls + roof) (kg)		17.555
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		2.114
Total blasted volume (m^3)		13.481
Volume to be removed (m^3)		20.222
Total amount of explosive (kg)		40.719
Overall powder Factor (kg/m^3)		3.020
Holes number (-)		40
Holes density (holes/m^2)		4.44
Total drilled length (m)		60
Specific drilling (m/m^3)		4.451
Total number of detonators (-)		63
Detonators specific consumption (D.C.) (detonators/m^3)		4.67

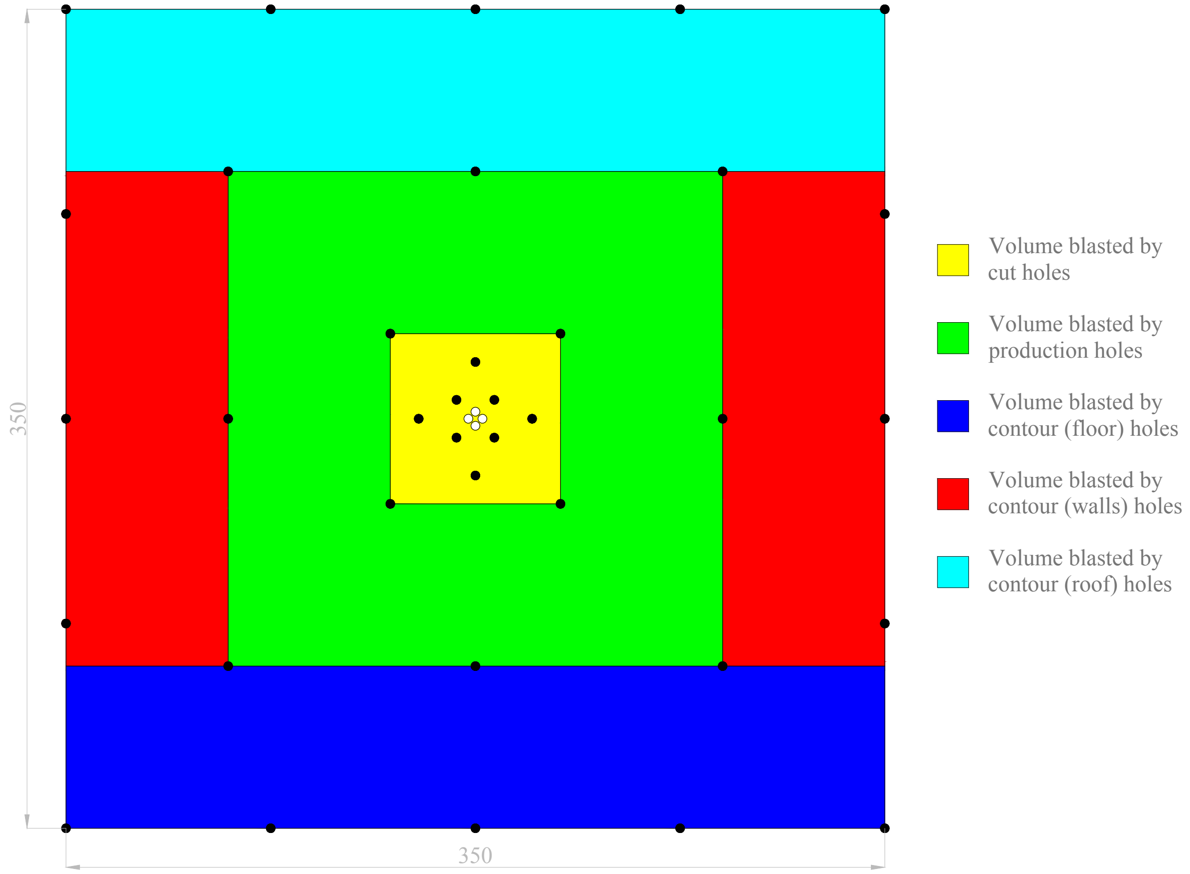

Figure 2.87 - Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 3.5 x 3.5 m section

Table 2.21 - Summary of the main data relating to the excavation section of 3.5 x 3.5 m

Cross section (m^2)		12.25
Design pull (m)		1.5
Hole diameter (m)		38.1
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	5.898
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	1.525
Contour (floor) holes	Volume blasted (m^3)	3.633
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	1.789
Contour (walls) holes	Volume blasted (m^3)	4.397
	Amount of explosive (kg)	6.030
	Powder factor (P.F.) (kg/m^3)	1.371
Contour (roof) holes	Volume blasted (m^3)	3.633
	Amount of explosive (kg)	5.025
	Powder factor (P.F.) (kg/m^3)	1.383
Volume blasted by contour holes (walls + roof) (m^3)		8.030
Amount of explosive for the contour holes (walls + roof) (kg)		11.055
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		1.377
Volume blasted by contour holes (floor + walls + roof) (m^3)		11.663
Amount of explosive for the contour holes (floor + walls + roof) (kg)		17.555
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.505
Total blasted volume (m^3)		18.332
Volume to be removed (m^3)		25.665
Total amount of explosive (kg)		40.719
Overall powder Factor (kg/m^3)		2.221
Holes number (-)		40
Holes density (holes/m^2)		3.26
Total drilled length (m)		60
Specific drilling (m/m^3)		3.273
Total number of detonators (-)		63
Detonators specific consumption (D.C.) (detonators/m^3)		3.44

The following Table 2.22 shows, for each instant of the detonation, the blasted volumes. The figures (Figure 2.88, Figure 2.89, Figure 2.90, Figure 2.91, Figure 2.92) illustrate the progression of the detonation. These data refer to the section with dimensions 3.0 x 3.0 m.

Table 2.22 - Volumes blasted at every instant of detonation

Instant of the detonation sequence (-)	Time (ms)	Volume of rock blasted at each step (m^3)	Overall volume of rock blasted (m^3)	Charge per delay (C.P.D.) (kg)
1	1000	0.000	0.000	0.000
2	1025	0.007	0.007	1.181
3	1050	0.014	0.021	1.181
4	1075	0.014	0.035	1.181
5	1100	0.018	0.053	1.181
6	1125	0.023	0.076	1.181
7	1150	0.031	0.107	1.181
8	1175	0.042	0.149	1.181
9	1200	0.055	0.204	1.181
10	1225	0.099	0.303	1.181
11	1250	0.177	0.480	1.181
12	1275	0.168	0.648	1.181
13	1300	0.228	0.876	1.181
14	1325	0.341	1.217	1.124
15	1350	0.820	2.037	2.248
16	1375	2.863	4.900	6.148
17	1400	1.165	6.065	2.248
18	1425	2.410	8.475	5.144
19	1450	3.438	11.913	6.620
20	1475	1.580	13.493	3.015

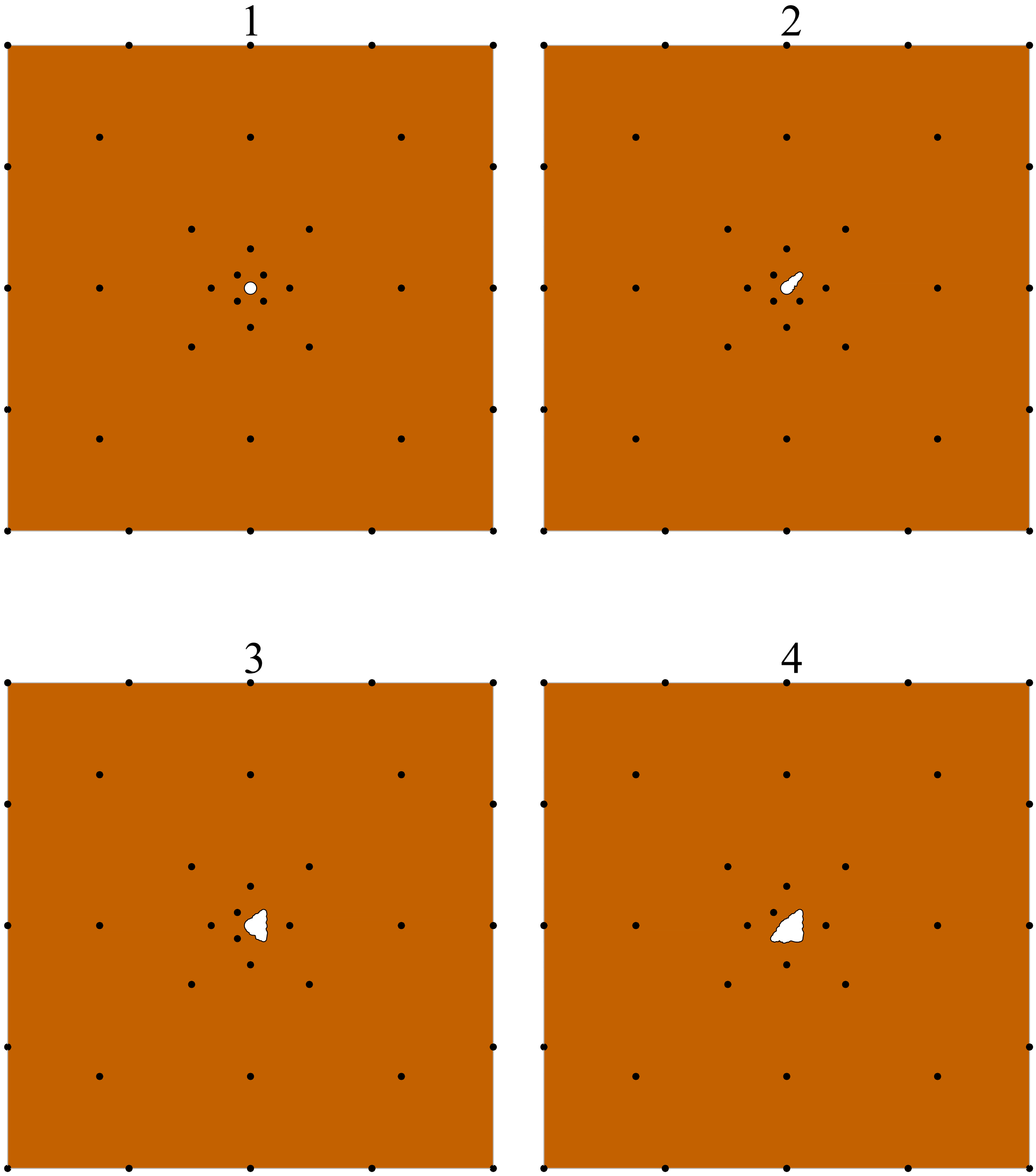

Figure 2.88 - Graphical representation of the detonation instants 1 - 4

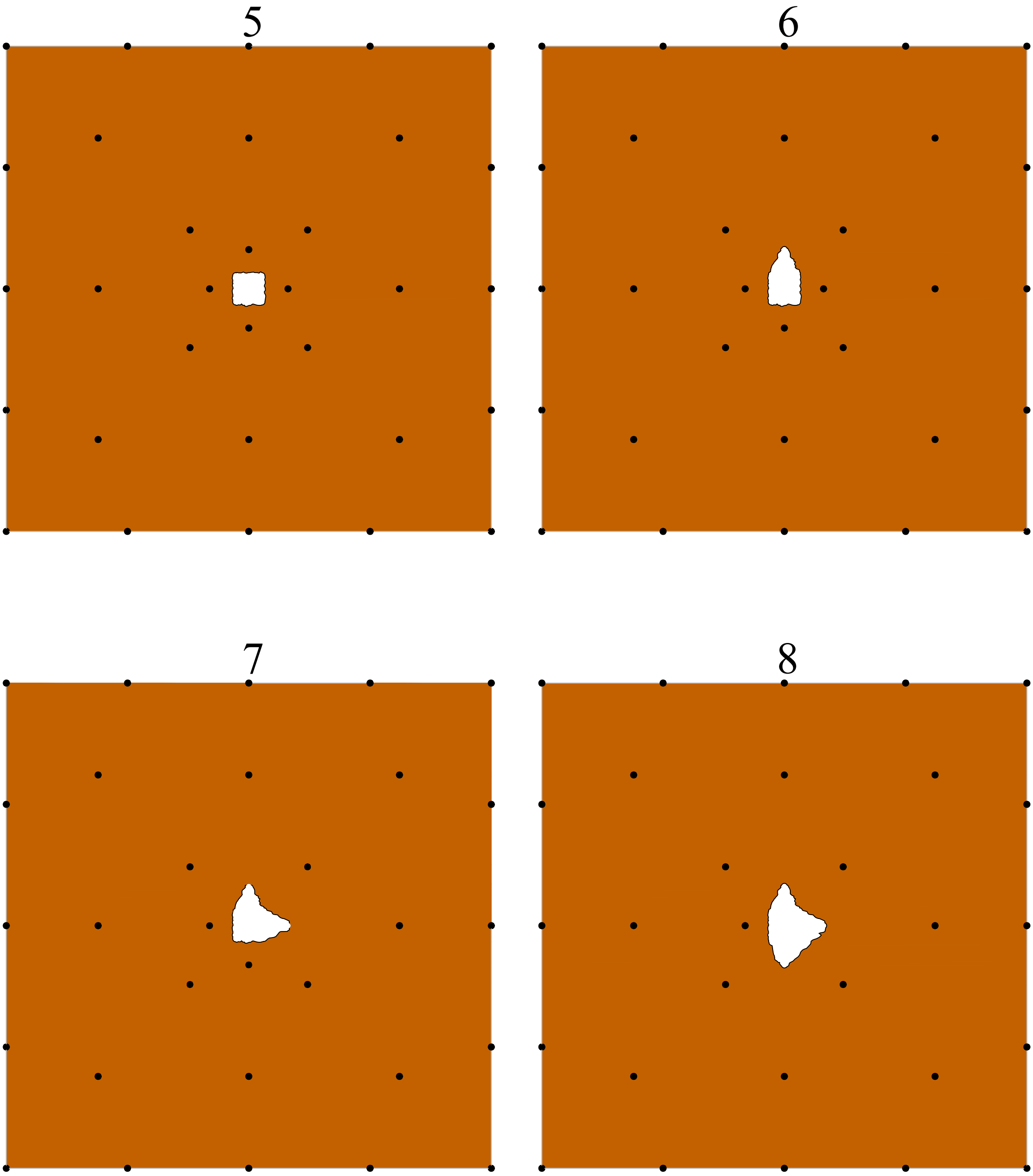

Figure 2.89 - Graphical representation of the detonation instants 5 - 8

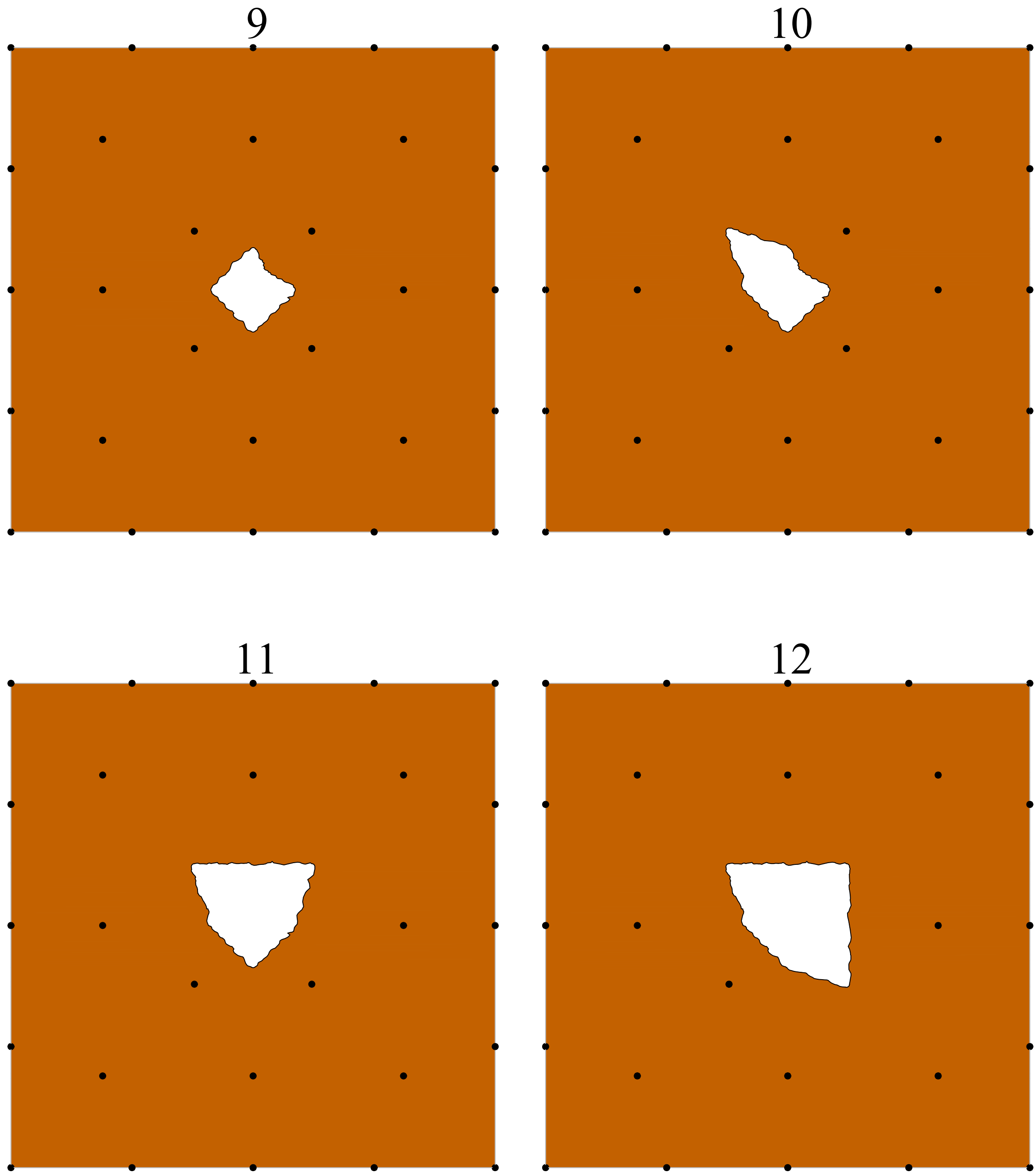

Figure 2.90 - Graphical representation of the detonation instants 9 - 12

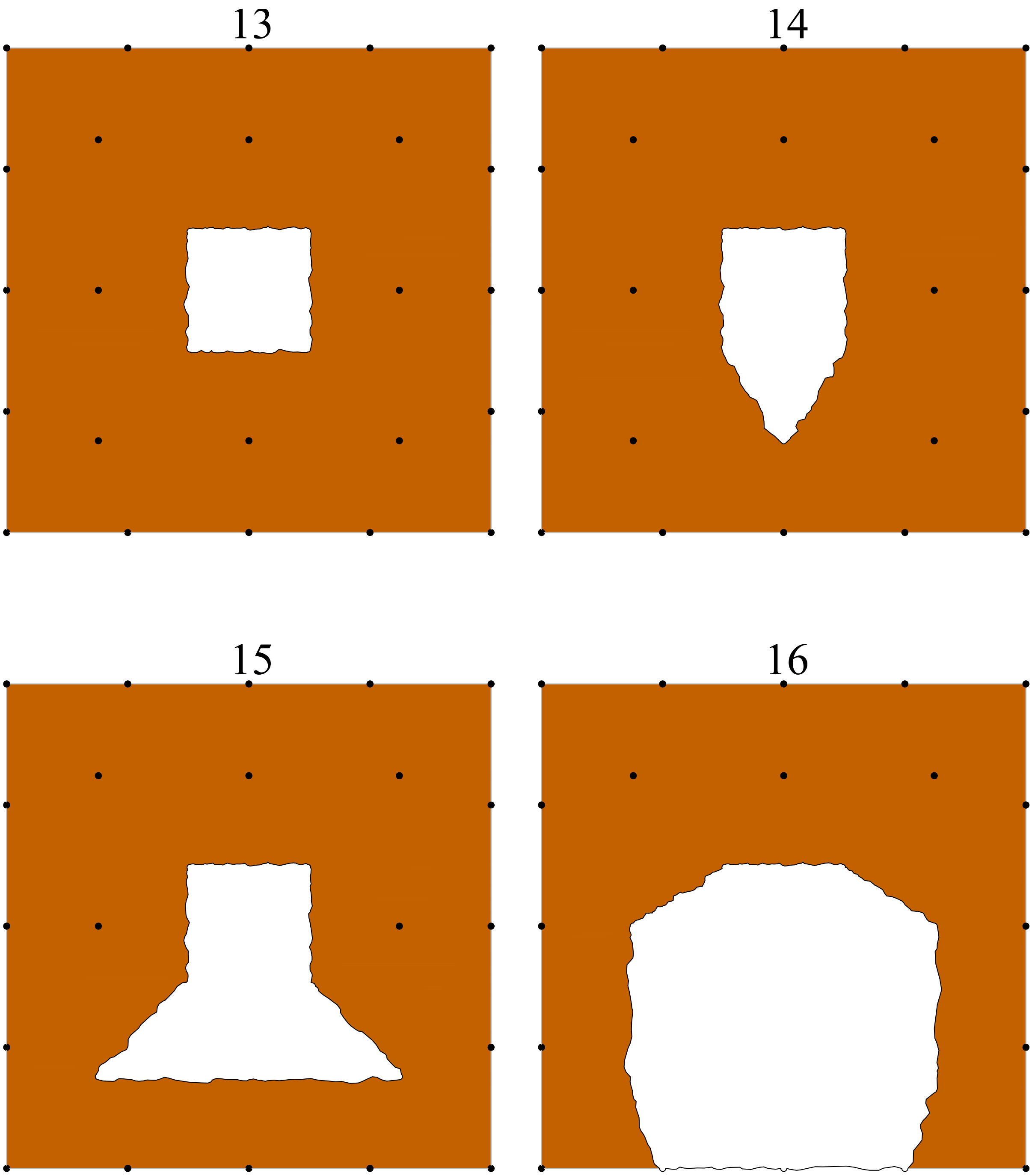

Figure 2.91 - Graphical representation of the detonation instants 13 - 16

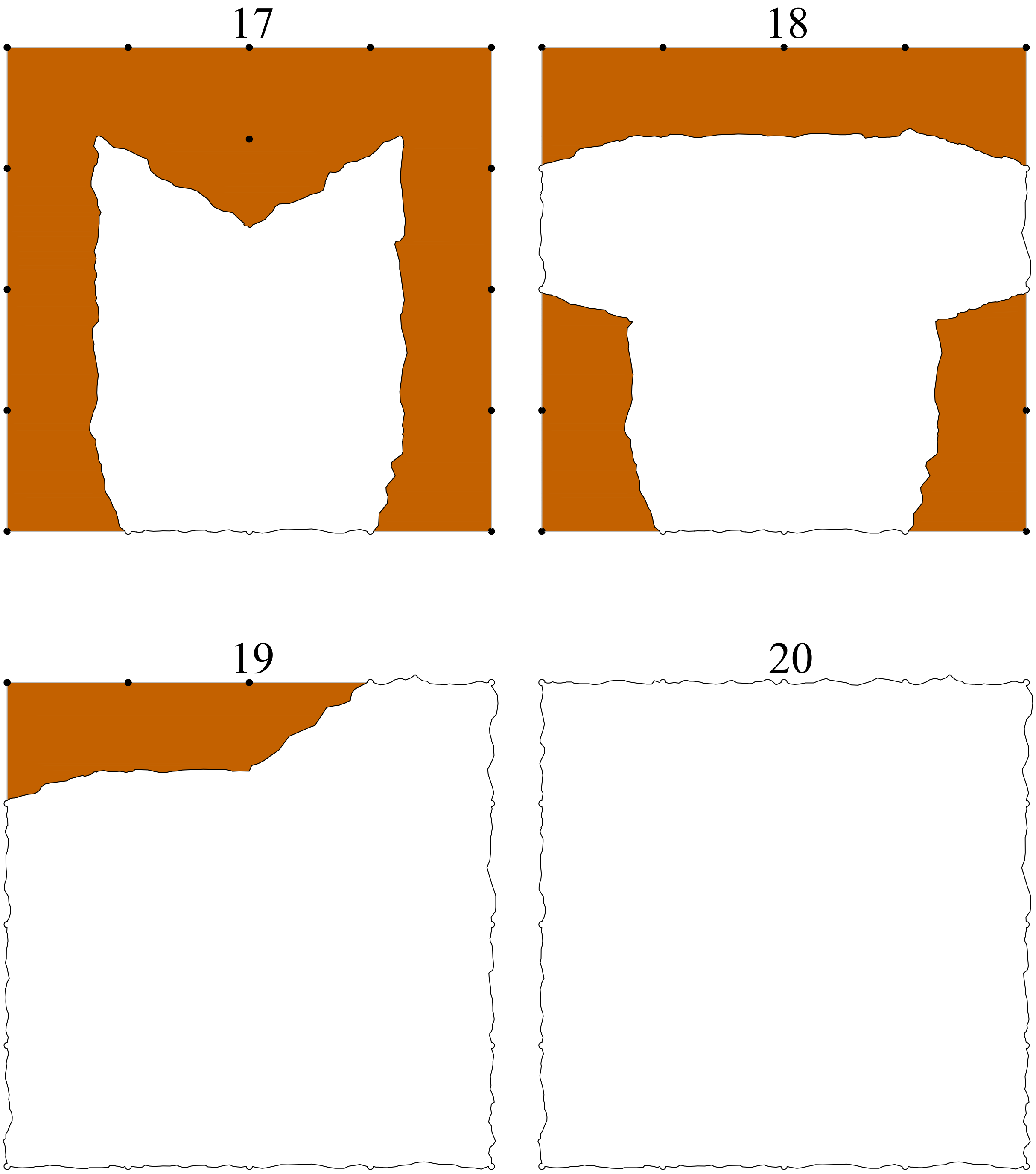

Figure 2.92 - Graphical representation of the detonation instants 17 - 20

2.5 - Design of the blast related to phase 2

The calculations are given for the three cross-section analysed, i.e.: 2.5 x 2.5 m, 3.0 x 3.0 m, 3.5 x 3.5 m.

As already mentioned, as for phase 2, the changes pertain to the shape of the cross section (which goes from being square to being with arched roof and vertical walls) and the explosive used for the contour blast-holes (considering the introduction of decoupling).

The diameter of the holes used is the same used in step 1. The empty holes, placed at the centre of the cut, have the same geometry as those adopted in phase 1: the whole cut is therefore equal to that used in phase 1. Therefore, only the schemes, calculations and considerations related to the production and contour blast-holes are reported. In case no additional information is mentioned, this means that there are no changes compared to phase 1.

When the cross section is 2.5 x 2.5 m, the spacing among the production blast-holes is 81 cm in the linear sides and 67 in the arch. The burden varies between 44 and 49 cm. The scheme is shown in Figure 2.93.

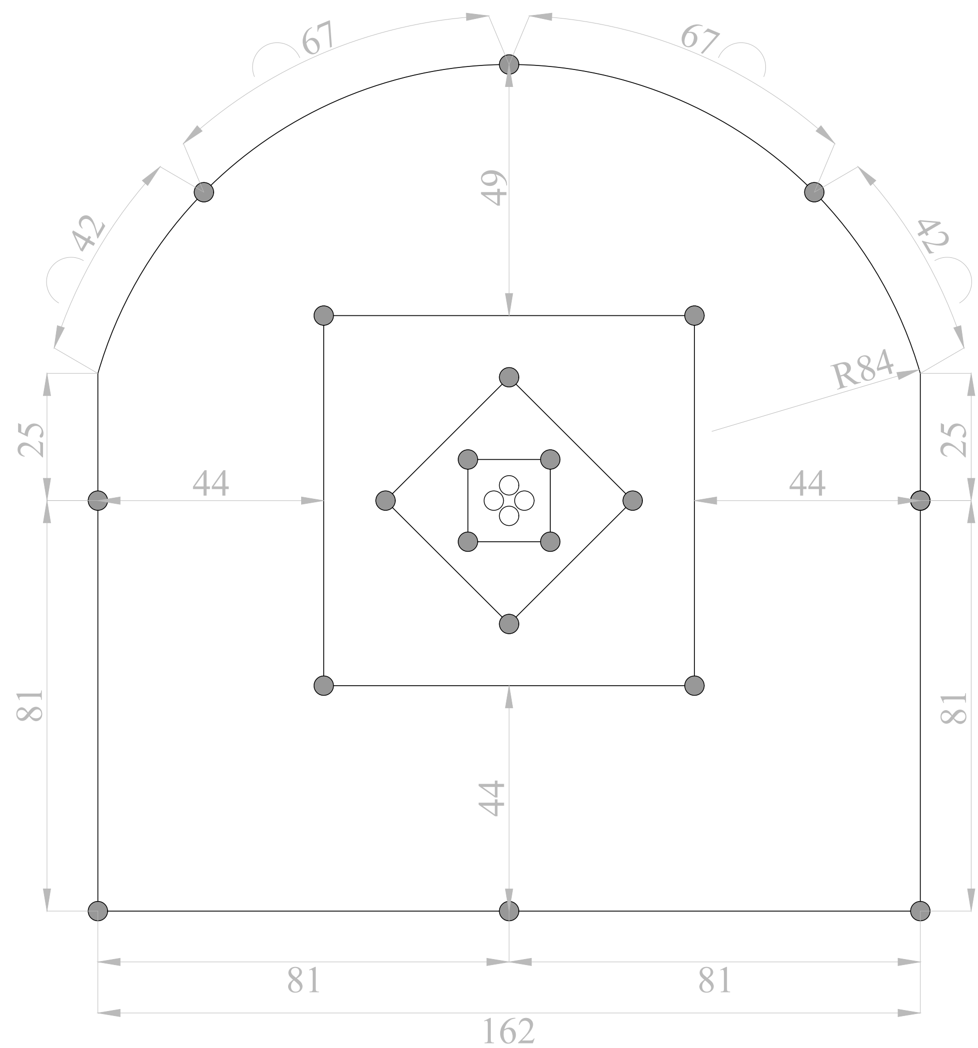

Figure 2.93 - Quoted scheme of the production holes for the section with dimension of 2.5 x 2.5 m

As for the 3 x 3 m section, the burden is 57 cm, the spacing is 93 cm for the linear side, 75 cm in the arched section. The quoted scheme is shown in Figure 2.94.

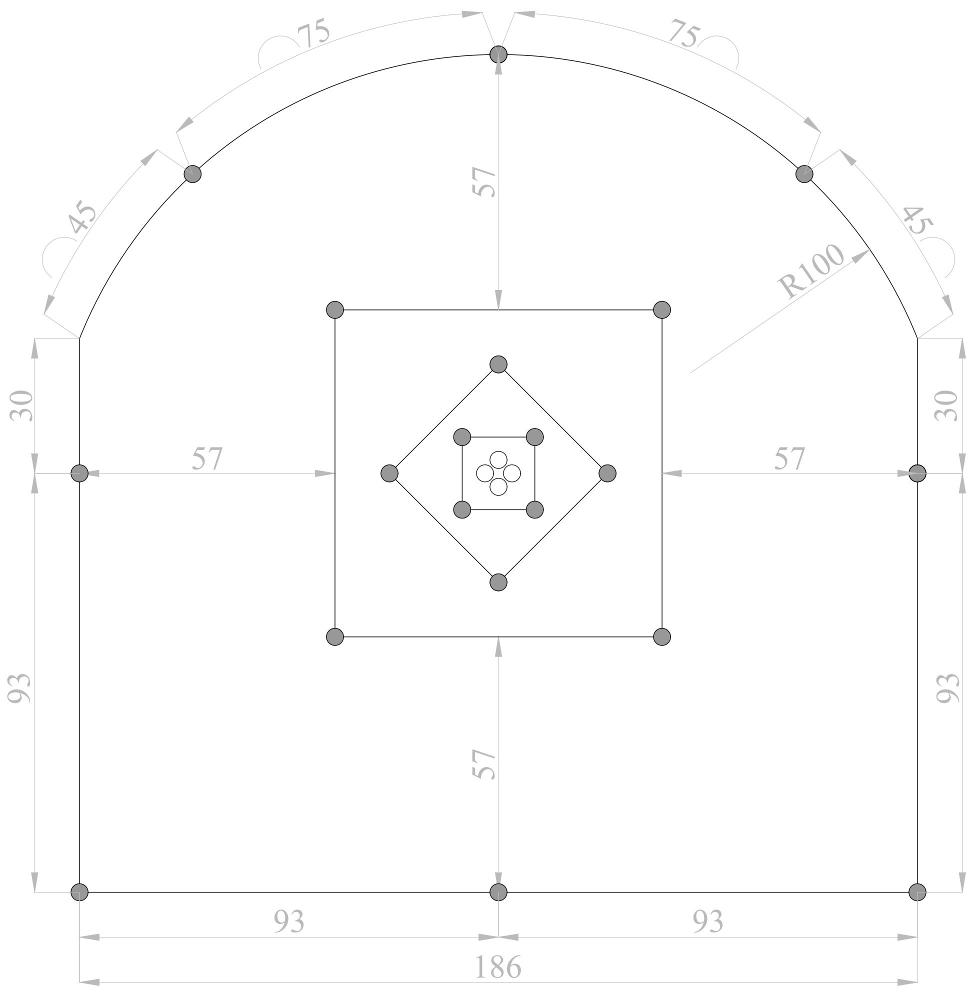

Figure 2.94 - Quoted scheme of the production holes for the section with dimension of 3.0 x 3.0 m

The blast-holes of the 3.5 x 3.5 m section have a spacing equal to 106 cm in the linear side, equal to 89 cm in the arched roof. The burden is 69 cm. The quoted scheme is shown in Figure 2.95.

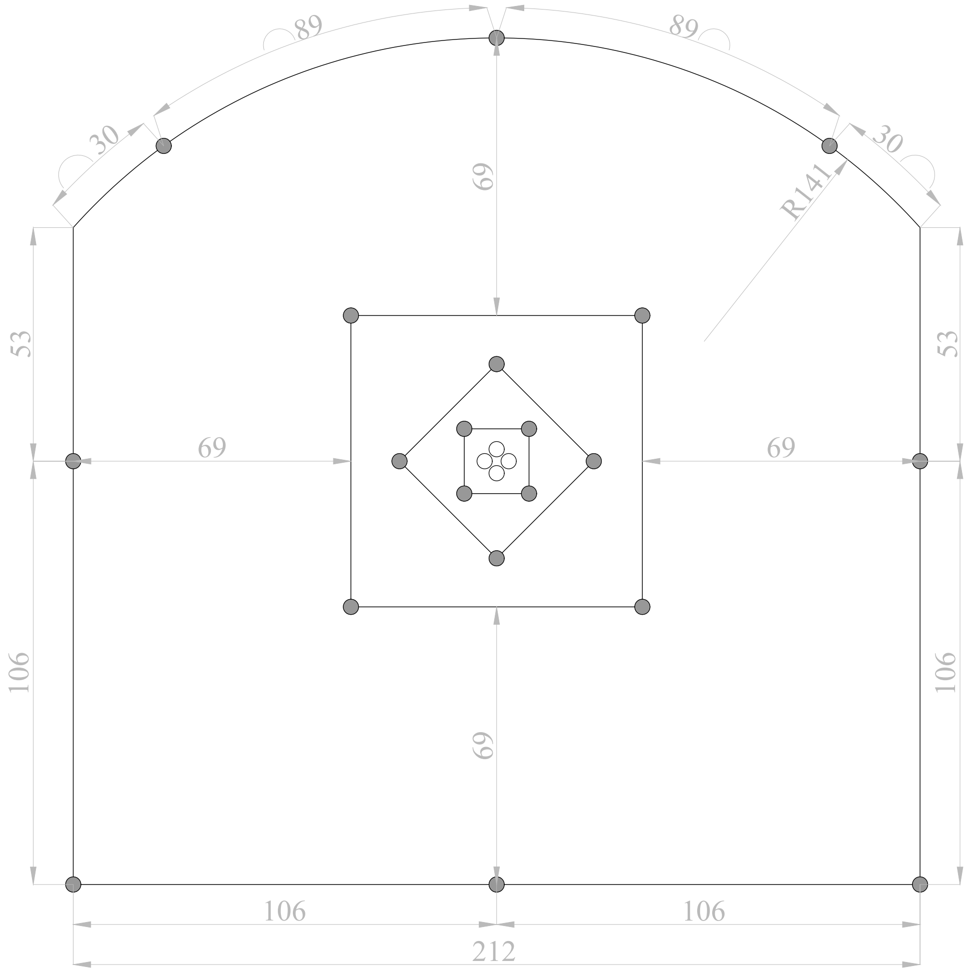

Figure 2.95 - Quoted scheme of the production holes for the section with dimension of 3.5 x 3.5 m

The charge of the production holes, as for those of the cut, is the same as that of phase 1.

As for the holes in the contour, the calculation procedure used for the design of the floor's blast-holes is the same as that adopted in step 1 (using Table 2.15). The contour holes in the walls and the roof were designed by applying the empirical rules of the smooth blasting (E. I. du Pont de Nemours & Co., 1978). The following table (Table 2.23) shows the empirical data used for the design of contour holes with the rules of smooth blasting. The recommended values for hole diameter, burden, spacing and linear charge concentration are shown.

In this case, being the diameter of the holes equal to 38.1 mm, the recommended spacing is 60 cm and the burden 100 cm.

Table 2.23 - Parameters for smooth blasting

Hole diameter		Spacing		Burden		Explosive charge	
(inch)	(mm)	(feet)	(m)	(feet)	(m)	(lb/ft)	(kg/m)
1.50-1.75	38-44	2.00	0.6	3.00	1.00	0.12-0.25	0.05-0.55
2.0	50	2.50	0.75	3.50	1.06	0.12-0.25	0.05-0.55

As for the 2.5 x 2.5 m section, the burden of the contour holes is 44 cm. The spacing is 83 cm for the floor holes. The spacing for the holes placed on the vertical walls is 50 cm, for those on the arched roof it is 58 cm. The geometric outline of the contour holes is shown in Figure 2.96.

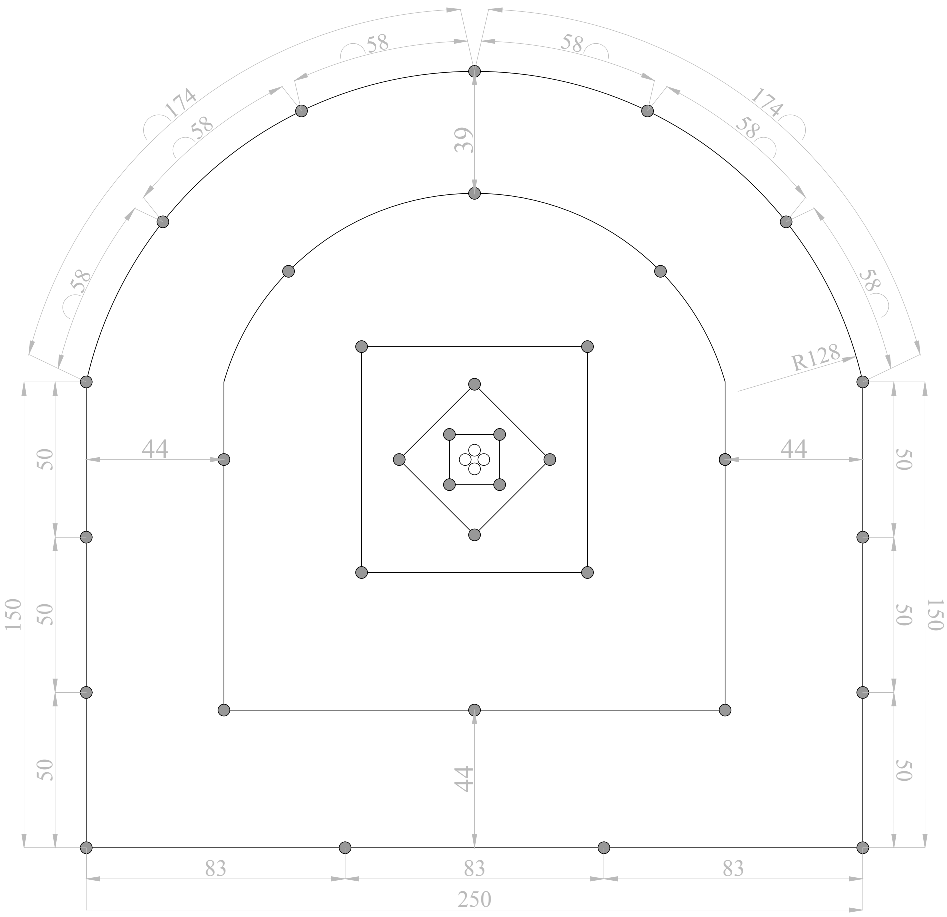

Figure 2.96 - Quoted scheme of the contour holes for the section with dimension of 2.5 x 2.5 m

The excavation section with dimensions 3.0 x 3.0 m has contour holes with a burden of 57 cm. The spacing of the floor holes is equal to 75 cm, for the holes on the vertical walls it is equal to 60 cm, while on the arched roof the spacing is equal to 57 cm. The geometry of the contour holes is shown in the Figure 2.97.

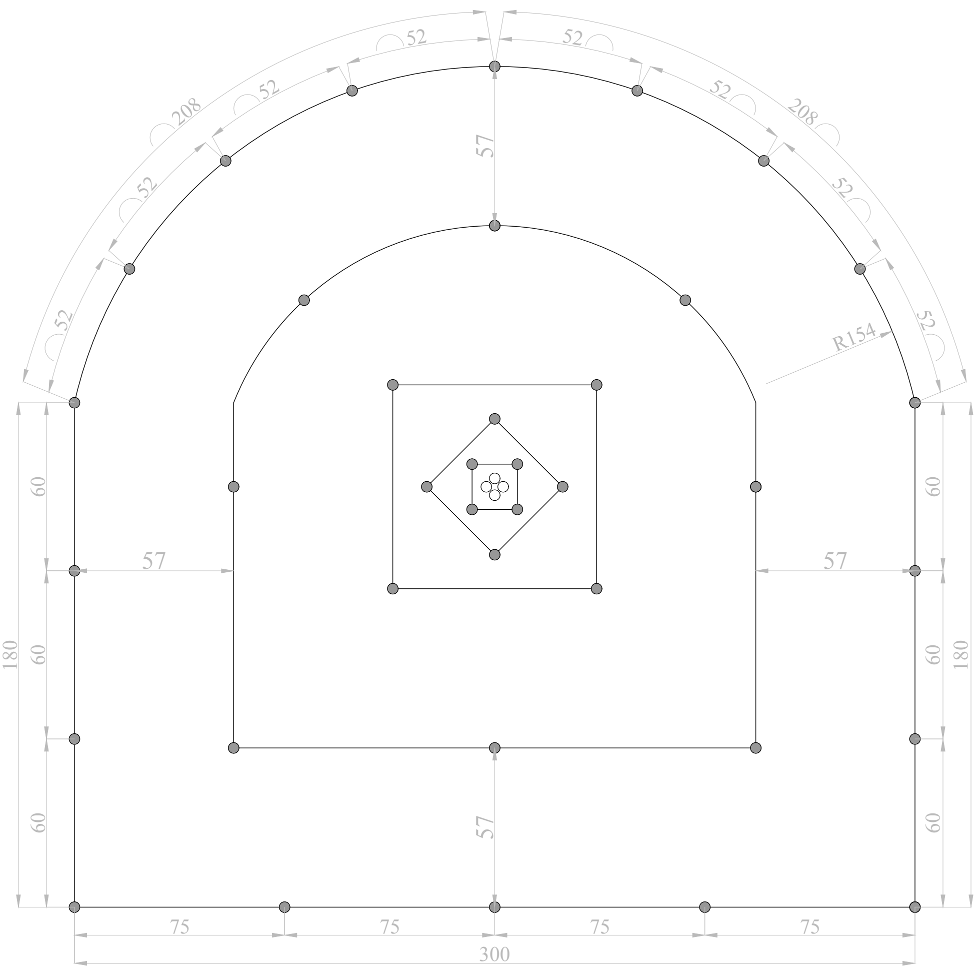

Figure 2.97 - Quoted scheme of the contour holes for the section with dimension of 3.0 x 3.0 m

Regarding the 3.5 x 3.5 m section, the burden is 69 cm. The spacing of the blast-holes of the floor is 87 cm, that of the holes in the walls is 58 cm, that of the arched roof is 56 cm.

The position of the contour holes is visible in Figure 2.98.

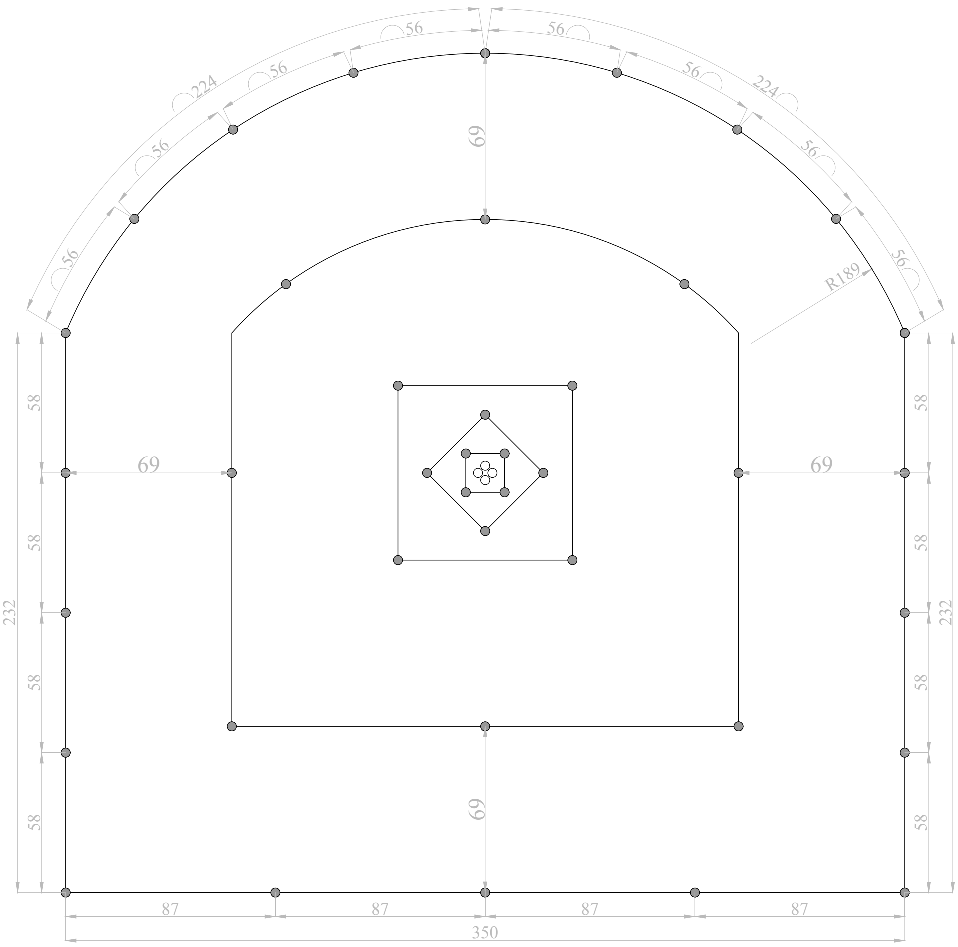

Figure 2.98 - Quoted scheme of the contour holes for the section with dimension of 3.5 x 3.5 m

The charge of the floor holes is the same as that used in step 1. As for the holes in the walls and the roof, the charge is different: the holes are loaded with a detonating cord (Pentacord® special 80P), having a linear charge concentration of 80 g/m. The diameter of the detonating cord is 10.3 mm. The decoupling of the charge (ratio between the diameter of the hole and that of the charge) is 3.7.

Figure 2.99 shows the scheme of a contour hole of the walls and of the roof charged with the detonating cord. Figure 2.100 shows a section of a contour hole representing this group: the decoupling is noticeable.

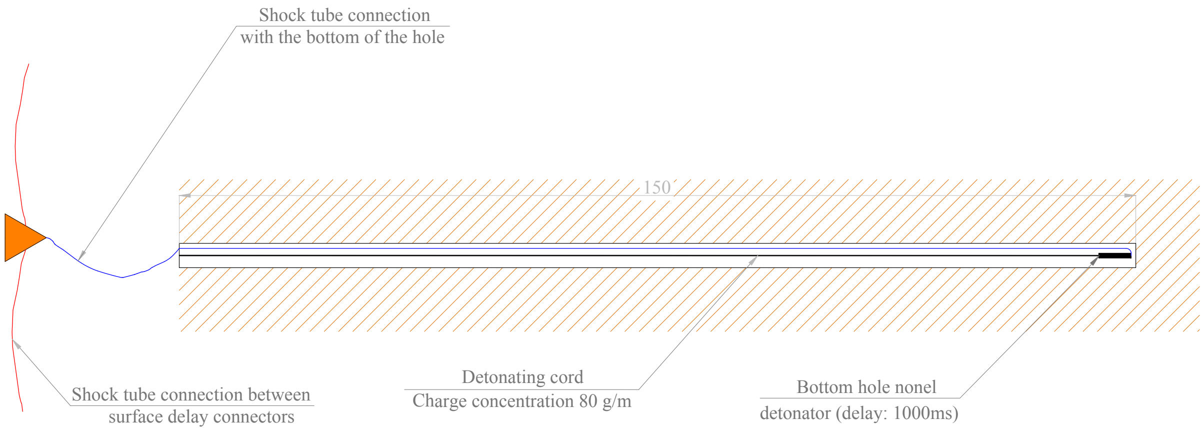

Figure 2.99 - Detail of a contour (walls and roof) blast-hole with charge and initiation system

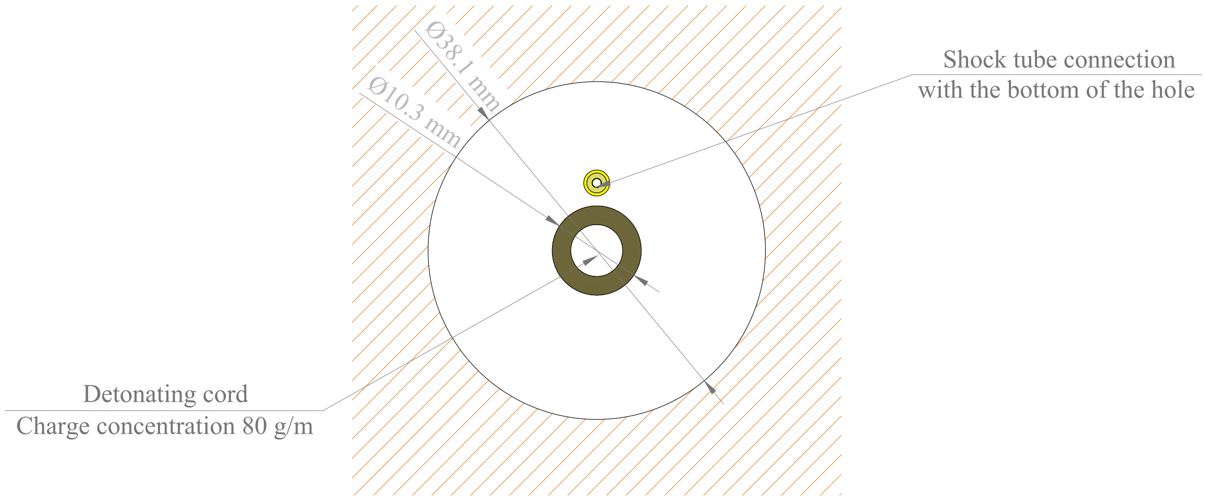

Figure 2.100 – Section of a charged contour hole (walls, roof)

Figure 2.101, Figure 2.102, Figure 2.103, show the complete drilling schemes for the three excavation cross-sections considered in phase 2.

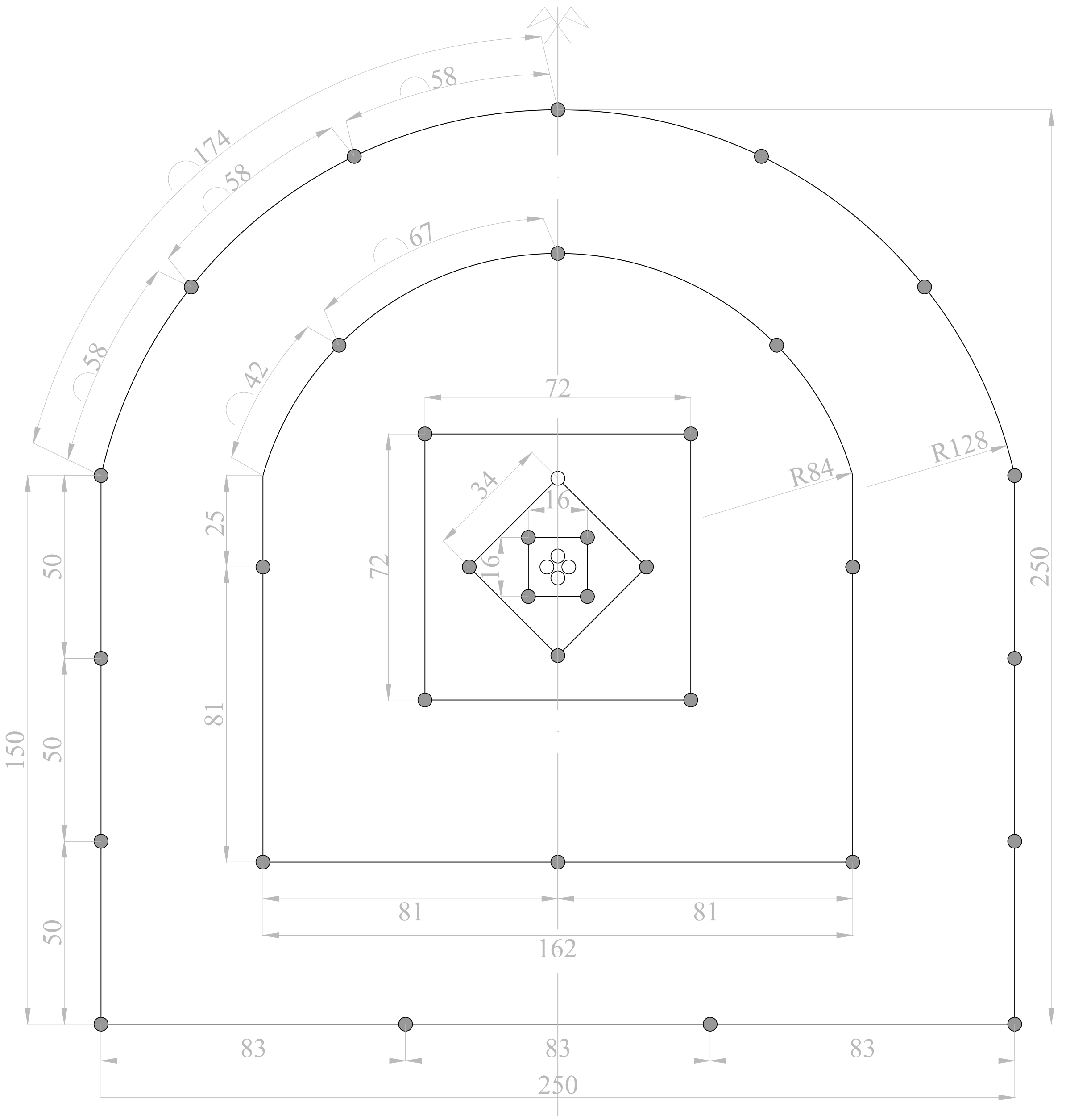

Figure 2.101 - Complete drilling scheme for the excavation section of 2.5 m x 2.5 m

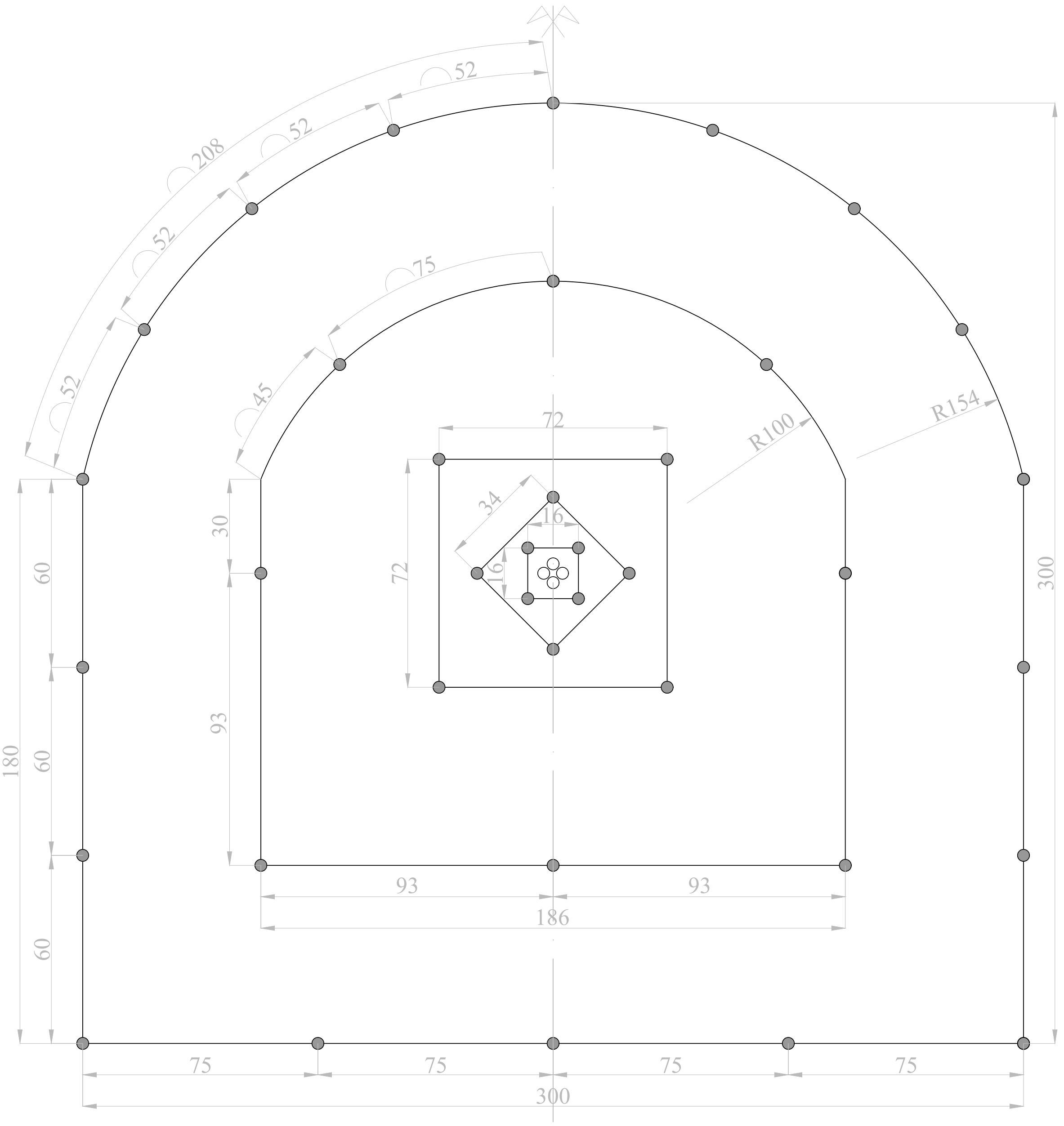

Figure 2.102 – Complete drilling scheme for the excavation section of 3.0 m x 3.0 m

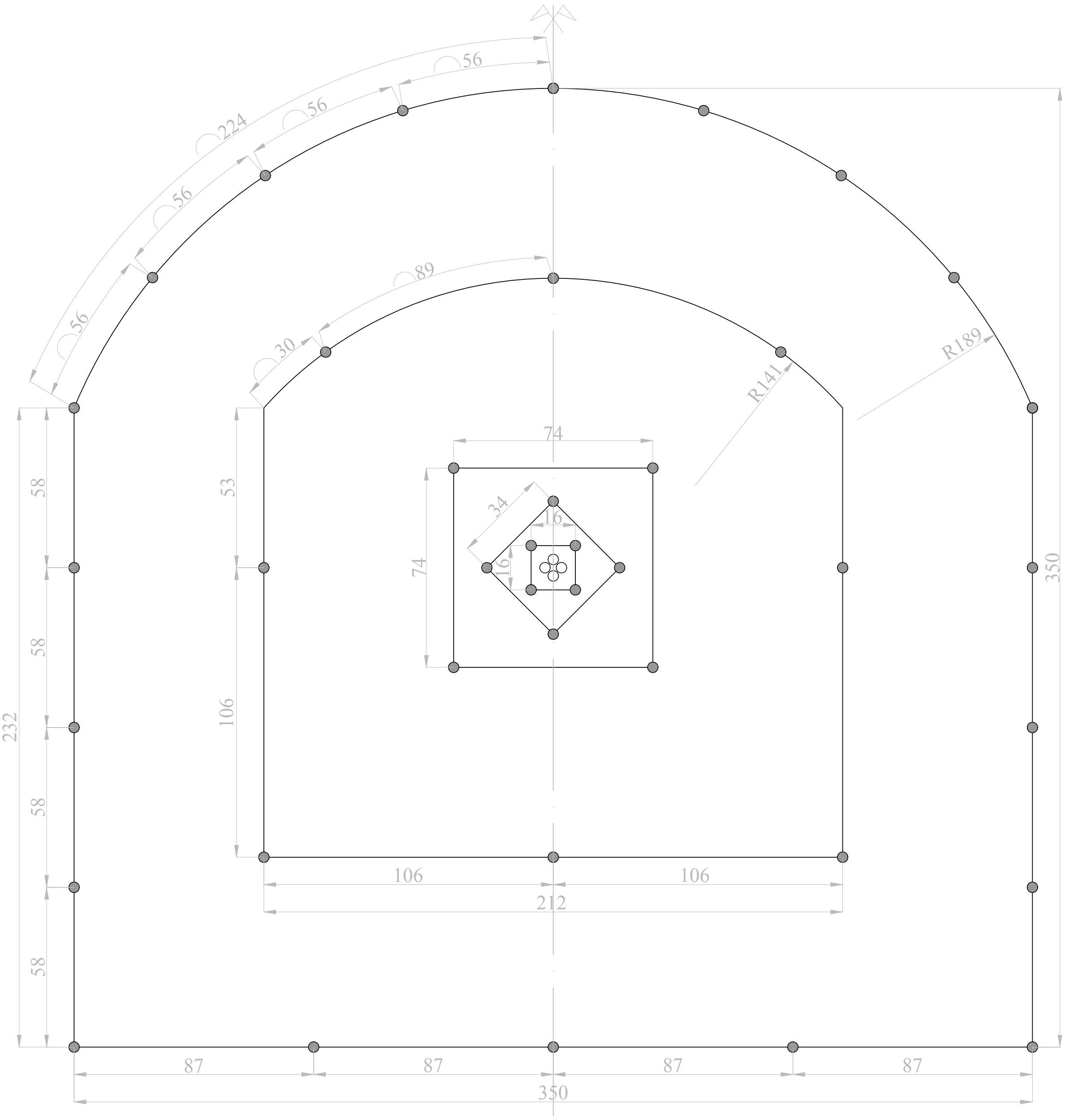

Figure 2.103 - Complete drilling scheme for the excavation section of 3.5 m x 3.5 m

Considering the maximum distances that nonel cables must cover for the various cross-sections adopted, as well as a safety margin to take into account possible inaccuracies due to drilling, and the fact that the nonel tubes must not be taut but loosened, a 3.6 m cable length of the detonators (non-electric detonators and non-electric dual detonators) was choosen.

In the following schemes, the detonation orders are shown: Figures 53, 55 and 57 show the triggering order of the surface delays connectors. It is also possible to observe the connection system between the various holes and measure the length of the connection tubes. Figures 54, 56 and 58 show the initiation of the in-the-hole detonators. The yellow arrows represent a graphic aid that facilitates the reading of the scheme, showing the holes that detonate in succession.

Both Figure 2.104 and Figure 2.105 are related to the 2.5 m x 2.5 m cross-section, Figure 2.106 and Figure 2.107 to the 3.0 x 3.0 m cross-section and, finally, Figure 2.108 and Figure 2.109 to the 3.5 x 3.5 m cross-section.

For each size of the excavation section, a table containing the most significant data concerning the initiation system is presented (containing the delay of the surface connector, the delay of the in-hole detonator, the number of the surface connectors and the number of the in-the-hole detonators).

Table 2.24 is related to the 2.5 x 2.5 section, Table 2.25 to the 3.0 x 3.0 m section and, finally, Table 2.26 to the 3.5 x 3.5 m section.

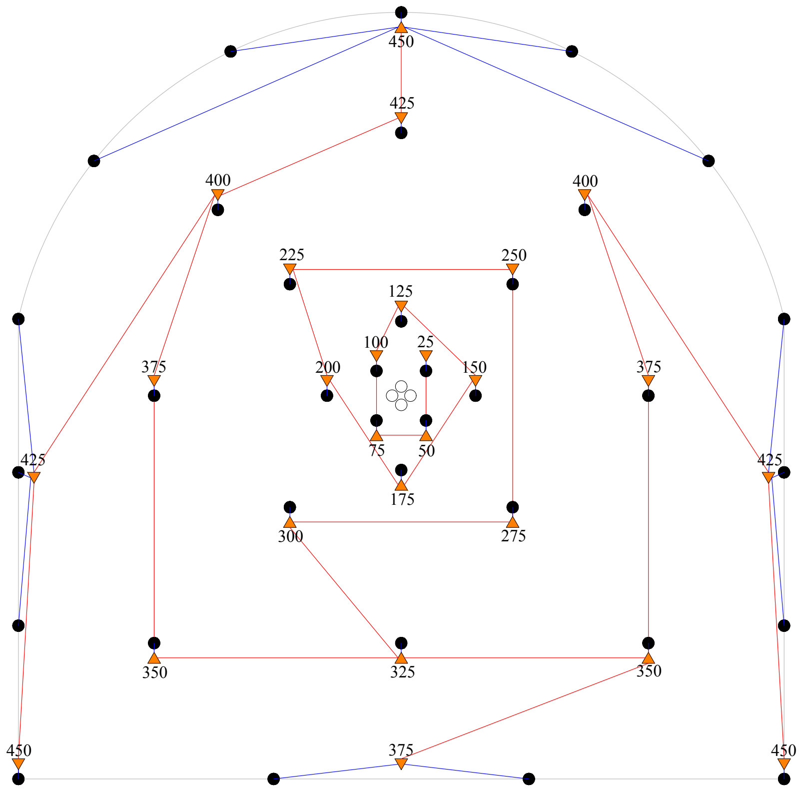

Figure 2.104 - Order of detonation of the surface delay connectors related to the 2.5 x 2.5 m section

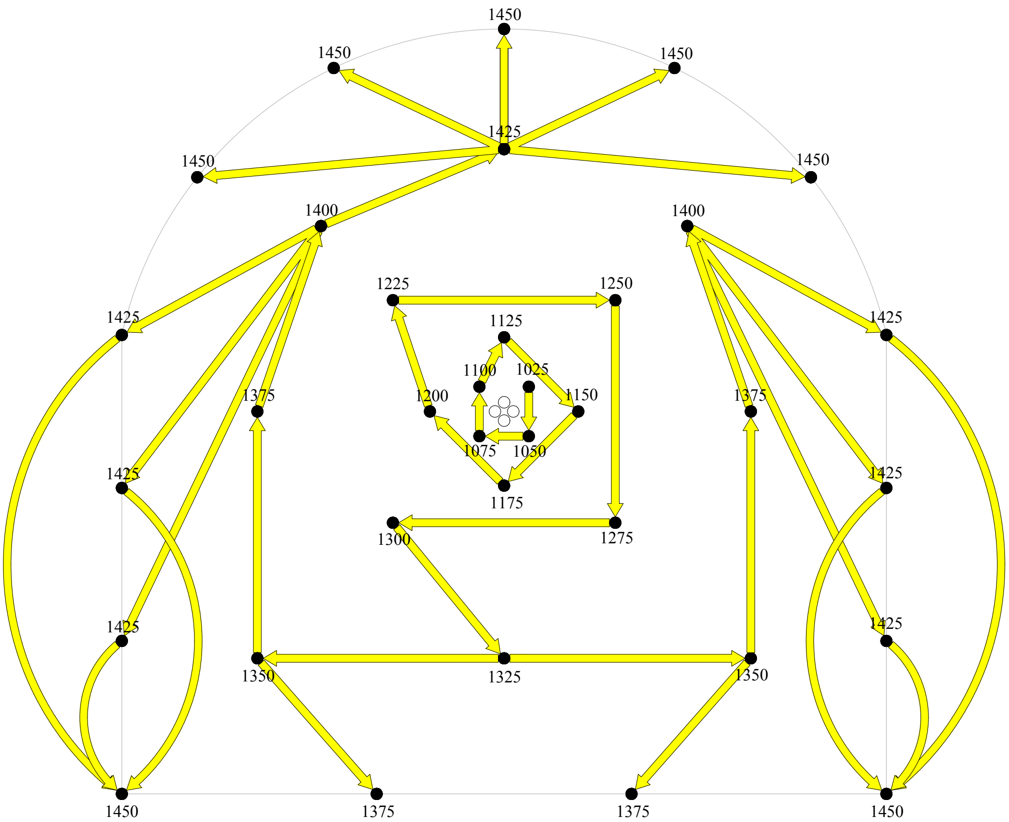

Figure 2.105 - Order of detonation of the in-hole detonators related to the 2.5 x 2.5 m section

Table 2.24 - Summary table of the main data related to the initiation system for the 2.5 x 2.5 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	1	1
11	25 ms	1000 ms	1275 ms	1	1
12	25 ms	1000 ms	1300 ms	1	1
13	25 ms	1000 ms	1325 ms	1	1
14	25 ms	1000 ms	1350 ms	2	2
15	25 ms	1000 ms	1375 ms	3	4
16	25 ms	1000 ms	1400 ms	2	2
17	25 ms	1000 ms	1425 ms	3	7
18	25 ms	1000 ms	1450 ms	3	7

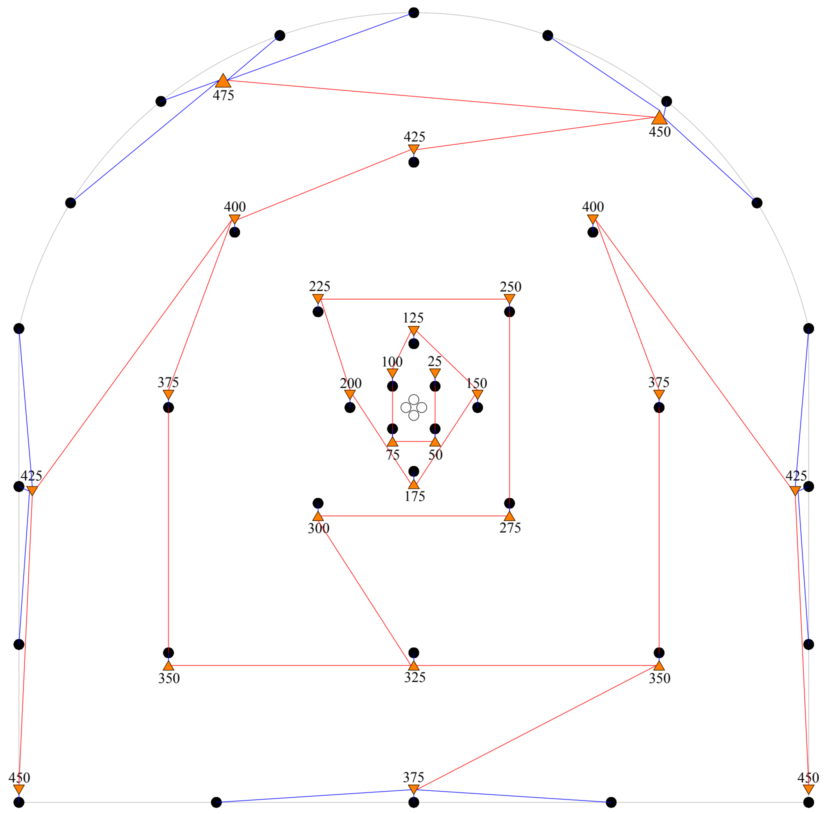

Figure 2.106 - Order of detonation of the surface delay connectors related to the 3.0 x 3.0 m section

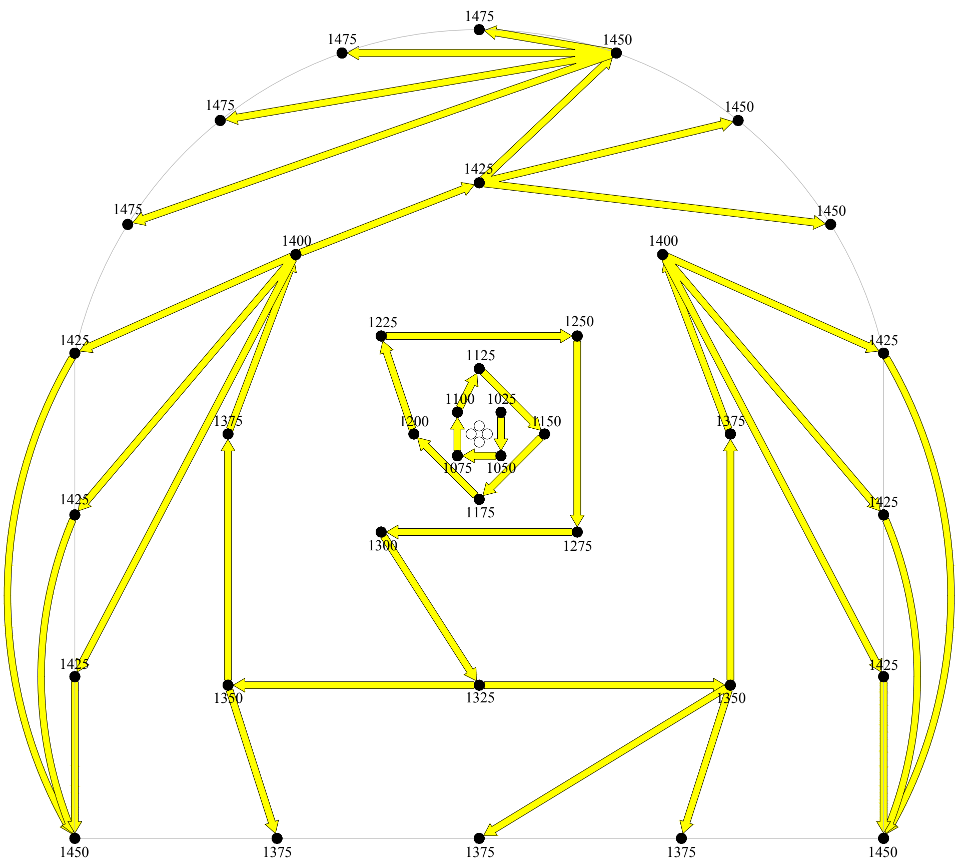

Figure 2.107 - Order of detonation of the in-hole detonators related to the 3.0 x 3.0 m section

Table 2.25 - Summary table of the main data related to the initiation system for the 3.0 x 3.0 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	1	1
11	25 ms	1000 ms	1275 ms	1	1
12	25 ms	1000 ms	1300 ms	1	1
13	25 ms	1000 ms	1325 ms	1	1
14	25 ms	1000 ms	1350 ms	2	2
15	25 ms	1000 ms	1375 ms	3	5
16	25 ms	1000 ms	1400 ms	2	2
17	25 ms	1000 ms	1425 ms	3	7
18	25 ms	1000 ms	1450 ms	3	5
19	25 ms	1000 ms	1475 ms	1	4

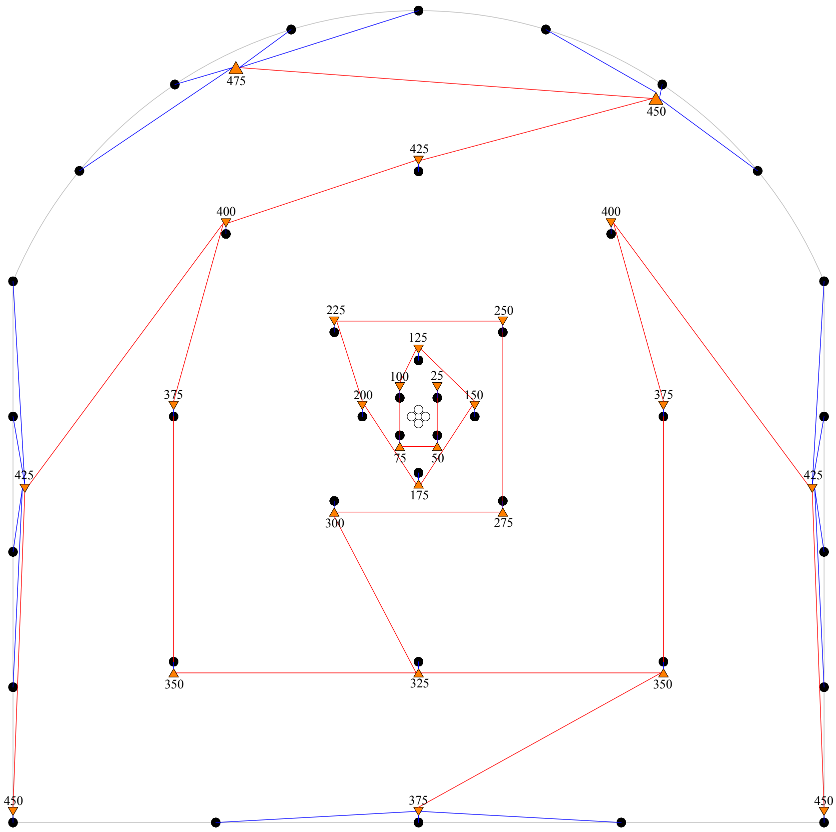

Figure 2.108 - Order of detonation of the surface delay connectors related to the 3.5 x 3.5 m section

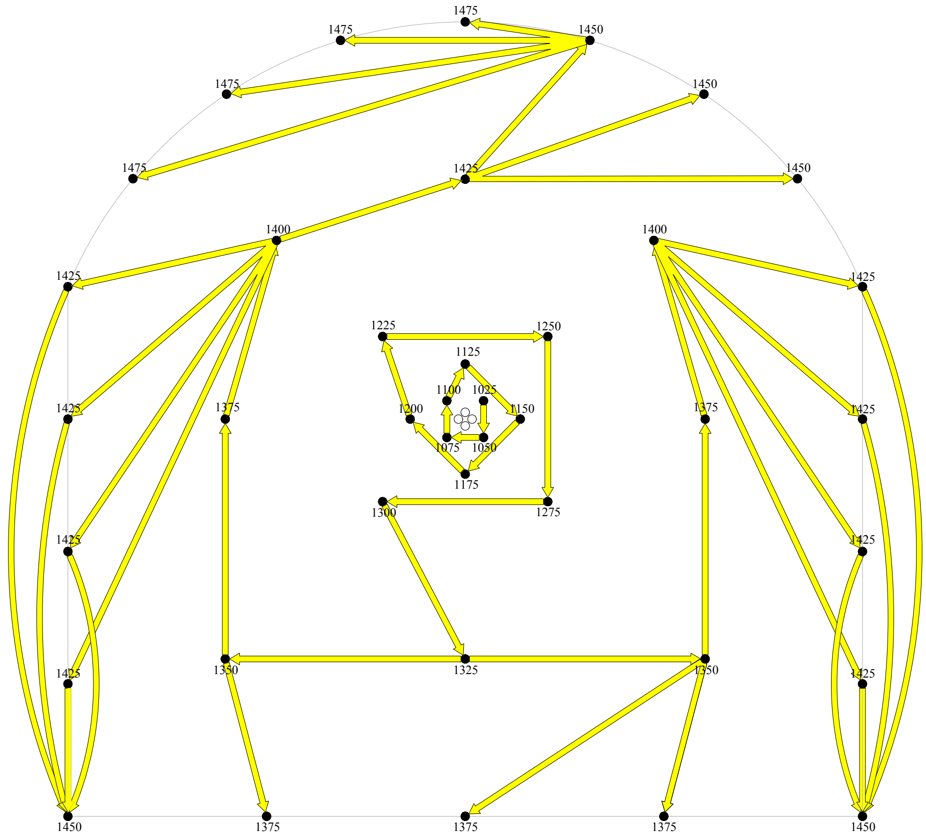

Figure 2.109 - Order of detonation of the in-hole detonators related to the 3.5 x 3.5 m section

Table 2.26 - Summary table of the main data related to the initiation system for the 3.5 x 3.5 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	1	1
11	25 ms	1000 ms	1275 ms	1	1
12	25 ms	1000 ms	1300 ms	1	1
13	25 ms	1000 ms	1325 ms	1	1
14	25 ms	1000 ms	1350 ms	2	2
15	25 ms	1000 ms	1375 ms	3	5
16	25 ms	1000 ms	1400 ms	2	2
17	25 ms	1000 ms	1425 ms	3	9
18	25 ms	1000 ms	1450 ms	3	5
19	25 ms	1000 ms	1475 ms	1	4

The following tables (Table 2.27 for the 2.5 x 2.5 m cross-section, Table 2.28 for the 3.0 x 3.0m cross-section and, finally, Table 2.29 for the 3.5 x 3.5 m cross-section) summarize the main data about the blasts for each cross section adopted. Specifically, the most important data are: volume blasted, amount of explosive, powder factor, volume to be removed (using a bulking factor equal to 1.4), holes density, total drilled length, specific drilling, detonators specific consumption. The volumes of competence of each functional group have been highlighted, for each dimension of the cross-section, in the following figures: Figure 2.110 (2.5 x 2.5 m cross-section), Figure 2.111 (3.0 x 3.0 m cross-section), Figure 2.112 (3.5 x 3.5 m cross-section).

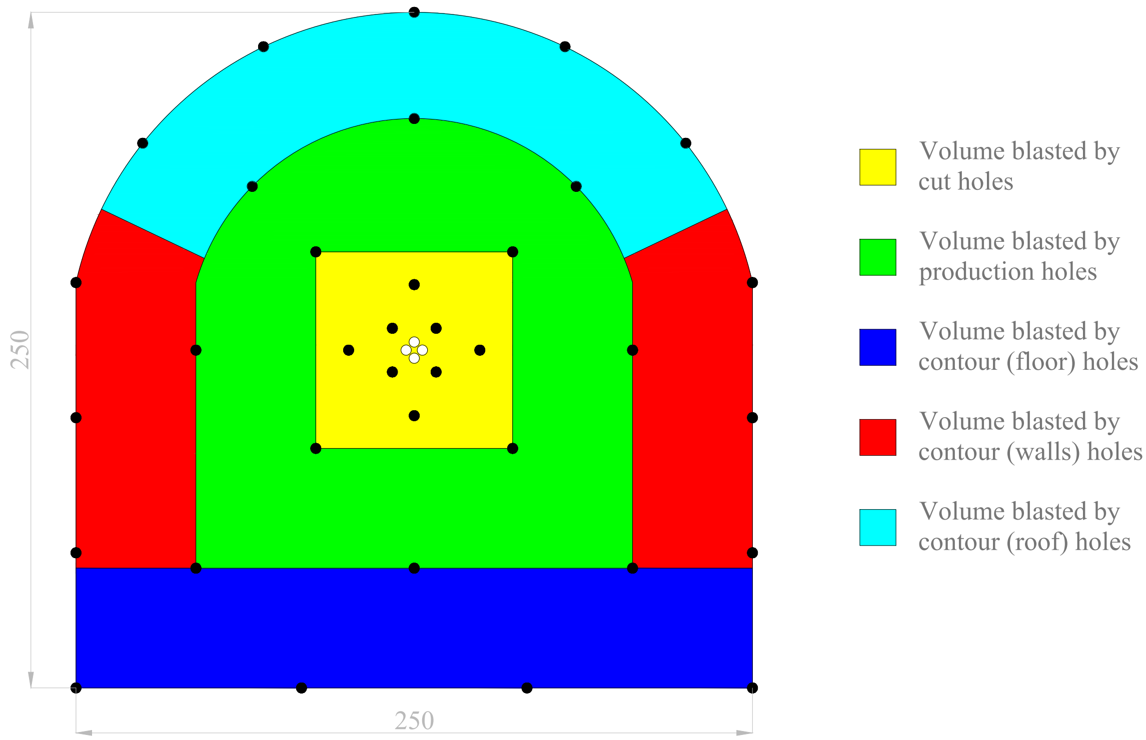

Figure 2.110 – Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 2.5 x 2.5 m section

Table 2.27 - Summary of the main data relating to the excavation section of 2.5 x 2.5 m

Cross section (m^2)		5.58
Design pull (m)		1.5
Hole diameter (m)		38.1
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	2.863
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	3.170
Contour (floor) holes	Volume blasted (m^3)	1.657
	Amount of explosive (kg)	5.200
	Powder factor (P.F.) (kg/m^3)	3.134
Contour (walls) holes	Volume blasted (m^3)	1.642
	Amount of explosive (kg)	0.720
	Powder factor (P.F.) (kg/m^3)	0.438
Contour (roof) holes	Volume blasted (m^3)	1.461
	Amount of explosive (kg)	0.600
	Powder factor (P.F.) (kg/m^3)	0.411
Volume blasted by contour holes (walls + roof) (m^3)		3.103
Amount of explosive for the contour holes (walls + roof) (kg)		1.320
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.425
Volume blasted by contour holes (floor + walls + roof) (m^3)		4.760
Amount of explosive for the contour holes (floor + walls + roof) (kg)		6.520
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.370
Total blasted volume (m^3)		8.368
Volume to be removed (m^3)		11.715
Total amount of explosive (kg)		29.684
Overall powder Factor (kg/m^3)		3.547
Holes number (-)		41
Holes density (holes/m^2)		7.35
Total drilled length (m)		61.5
Specific drilling (m/m^3)		7.349
Total number of detonators (-)		61
Detonators specific consumption (D.C.) (detonators/m^3)		7.289

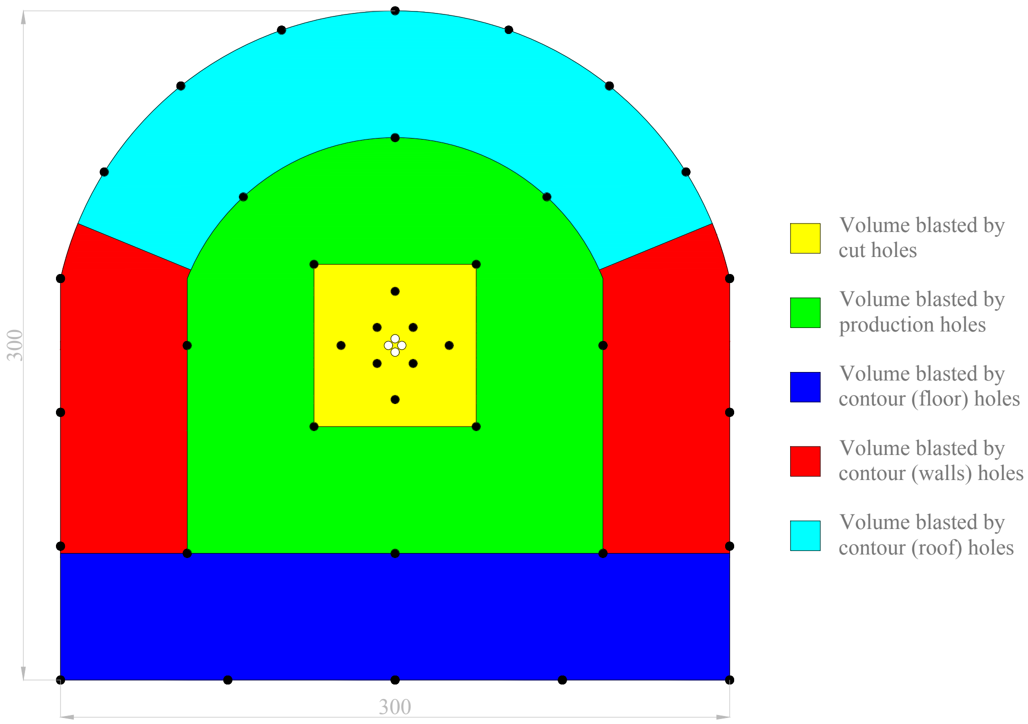

Figure 2.111 – Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 3.0 x 3.0 m section

Table 2.28 - Summary of the main data relating to the excavation section of 3.0 x 3.0 m

Cross section (m^2)		8.05
Design pull (m)		1.5
Hole diameter (m)		38.1
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	3.919
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	2.295
Contour (floor) holes	Volume blasted (m^3)	2.551
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	2.548
Contour (walls) holes	Volume blasted (m^3)	2.340
	Amount of explosive (kg)	0.720
	Powder factor (P.F.) (kg/m^3)	0.308
Contour (roof) holes	Volume blasted (m^3)	2.490
	Amount of explosive (kg)	0.840
	Powder factor (P.F.) (kg/m^3)	0.337
Volume blasted by contour holes (walls + roof) (m^3)		4.830
Amount of explosive for the contour holes (walls + roof) (kg)		1.560
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.323
Volume blasted by contour holes (floor + walls + roof) (m^3)		7.380
Amount of explosive for the contour holes (floor + walls + roof) (kg)		8.060
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.092
Total blasted volume (m^3)		12.070
Volume to be removed (m^3)		16.898
Total amount of explosive (kg)		31.224
Overall powder Factor (kg/m^3)		2.587
Holes number (-)		42
Holes density (holes/m^2)		5.22
Total drilled length (m)		63
Specific drilling (m/m^3)		5.219
Total number of detonators (-)		65
Detonators specific consumption (D.C.) (detonators/m^3)		5.385

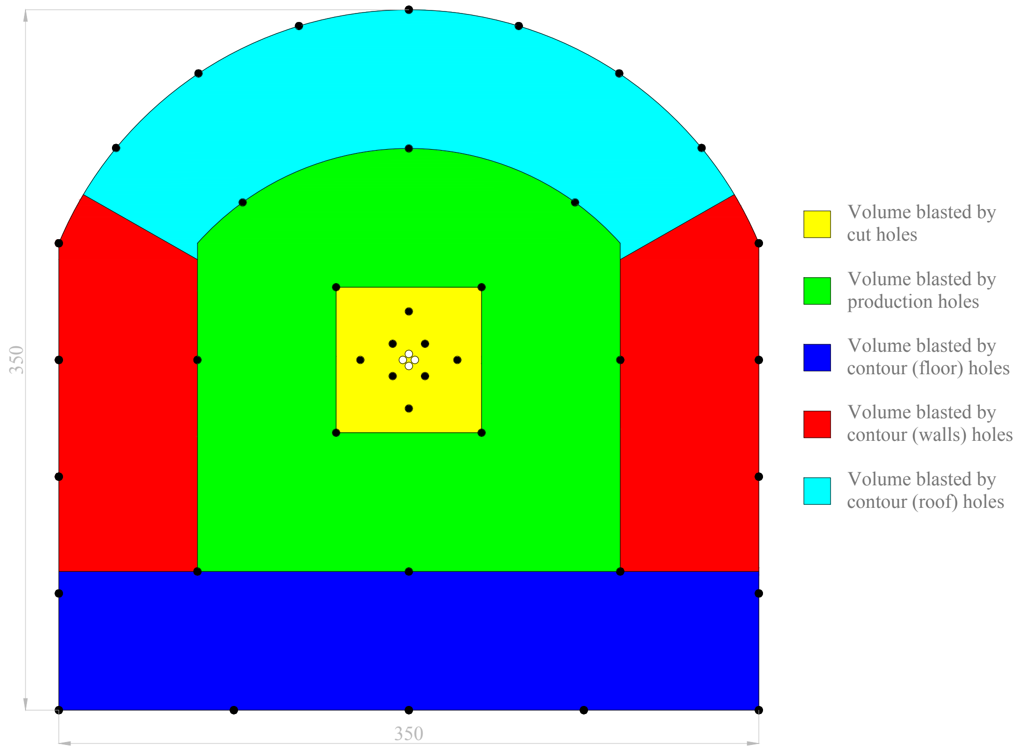

Figure 2.112 – Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 3.5 x 3.5 m section

Table 2.29 - Summary of the main data relating to the excavation section of 3.5 x 3.5 m

Cross section (m^2)		11.08
Design pull (m)		1.5
Hole diameter (m)		38.1
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	5.435
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	1.654
Contour (floor) holes	Volume blasted (m^3)	3.631
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	1.790
Contour (walls) holes	Volume blasted (m^3)	3.598
	Amount of explosive (kg)	0.720
	Powder factor (P.F.) (kg/m^3)	0.200
Contour (roof) holes	Volume blasted (m^3)	3.192
	Amount of explosive (kg)	0.840
	Powder factor (P.F.) (kg/m^3)	0.263
Volume blasted by contour holes (walls + roof) (m^3)		6.789
Amount of explosive for the contour holes (walls + roof) (kg)		1.560
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.263
Volume blasted by contour holes (floor + walls + roof) (m^3)		10.421
Amount of explosive for the contour holes (floor + walls + roof) (kg)		8.060
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		0.773
Total blasted volume (m^3)		16.682
Volume to be removed (m^3)		23.279
Total amount of explosive (kg)		31.224
Overall powder Factor (kg/m^3)		1.878
Holes number (-)		44
Holes density (holes/m^2)		3.97
Total drilled length (m)		66
Specific drilling (m/m^3)		3.969
Total number of detonators (-)		67
Detonators specific consumption (D.C.) (detonators/m^3)		4.029

The following Table 2.30 shows, for each delay time, the blasted volumes. The figures (Figure 2.113, Figure 2.114, Figure 2.115, Figure 2.116, Figure 2.117) clarify the progression of the detonation. These data refer to the 3.0 x 3.0 m cross section.

Table 2.30 - Volumes blasted at every instant of detonation

Detonation sequence (-)	Time (ms)	Volume of rock blasted at each step (m^3)	Overall volume of rock blasted (m^3)	Charge per delay (C.P.D.) (kg)
1	1000	0.000	0.000	0.000
2	1025	0.007	0.007	1.181
3	1050	0.014	0.021	1.181
4	1075	0.013	0.034	1.181
5	1100	0.019	0.053	1.181
6	1125	0.025	0.078	1.181
7	1150	0.042	0.120	1.181
8	1175	0.044	0.164	1.181
9	1200	0.056	0.220	1.181
10	1225	0.103	0.323	1.181
11	1250	0.167	0.490	1.181
12	1275	0.166	0.656	1.181
13	1300	0.231	0.887	1.181
14	1325	0.332	1.219	1.124
15	1350	0.857	2.076	2.248
16	1375	2.827	4.903	6.148
17	1400	0.643	5.546	2.248
18	1425	3.029	8.575	1.844
19	1450	2.126	10.701	2.960
20	1475	1.426	12.127	0.480

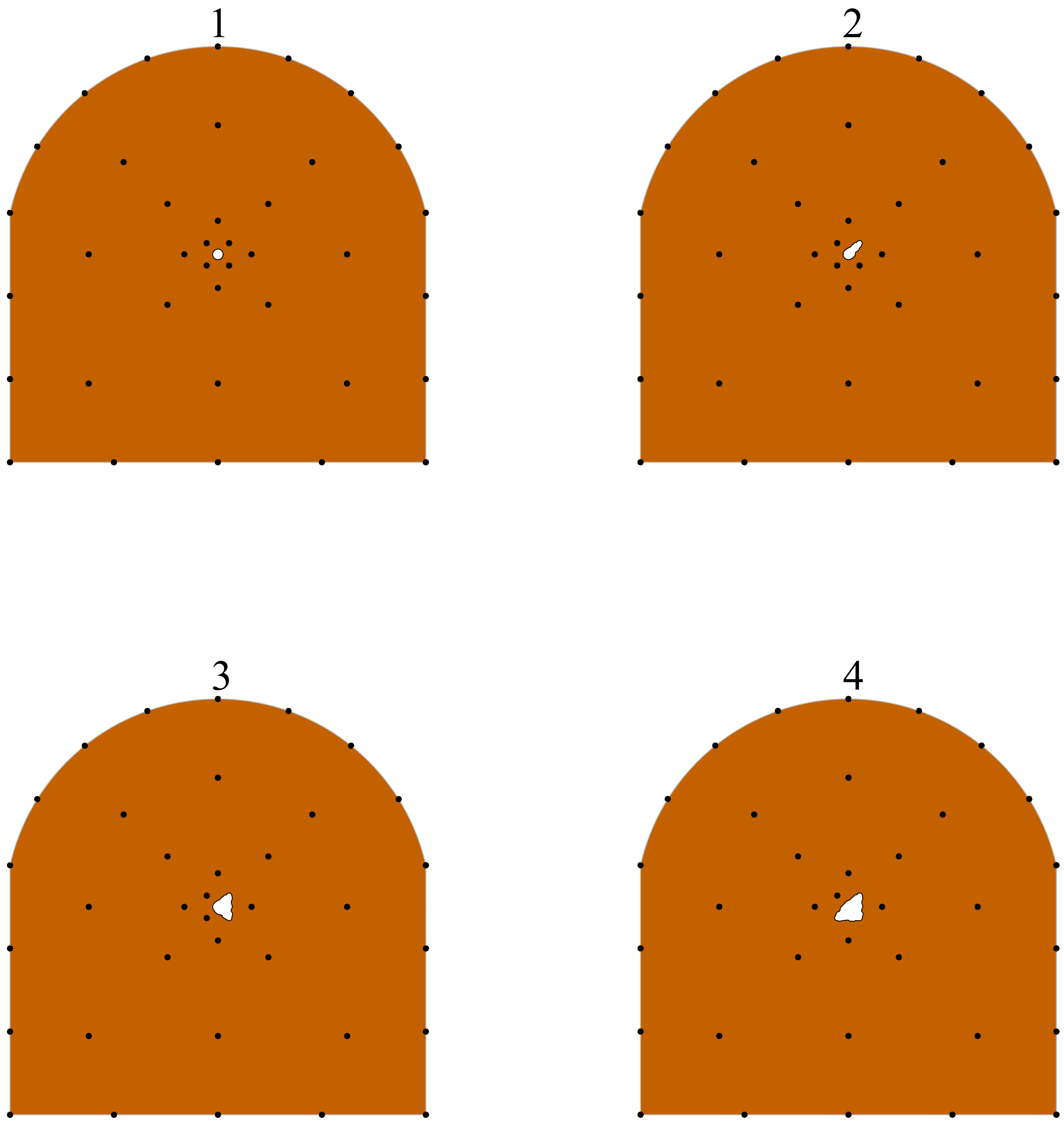

Figure 2.113 - Graphical representation of the detonation instants 1 - 4

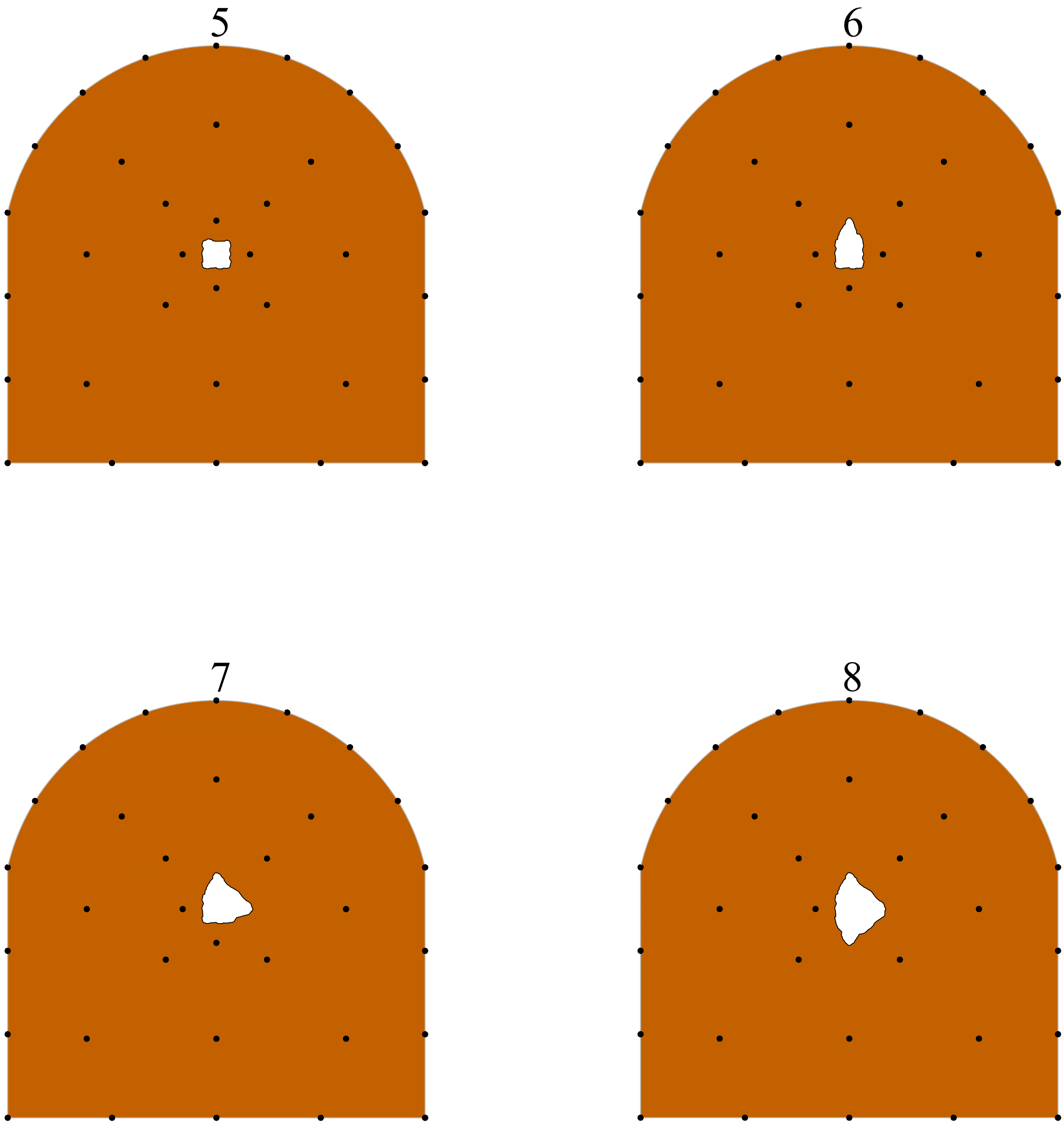

Figure 2.114 - Graphical representation of the detonation instants 5 - 8

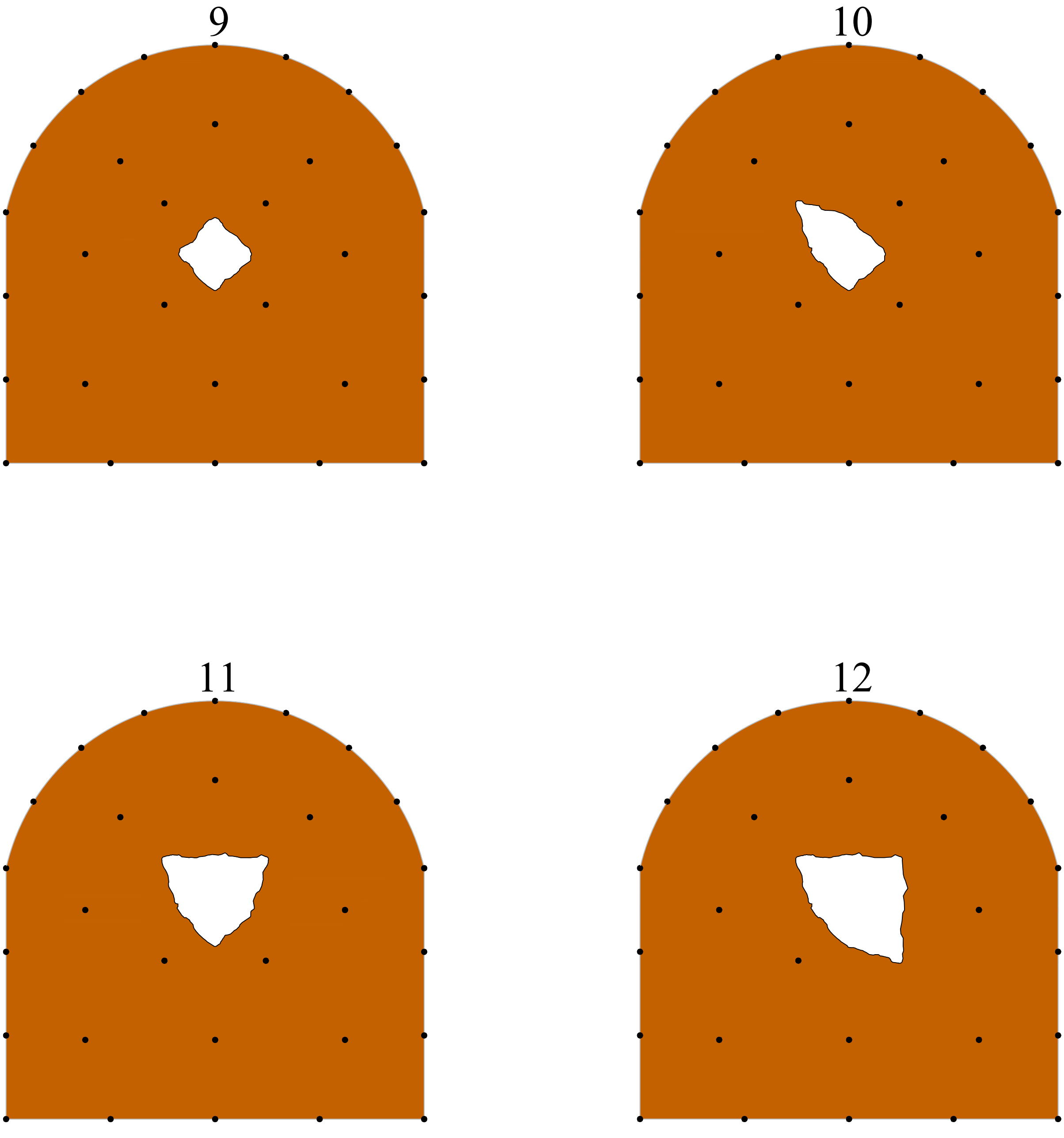

Figure 2.115 - Graphical representation of the detonation instants 9 - 12

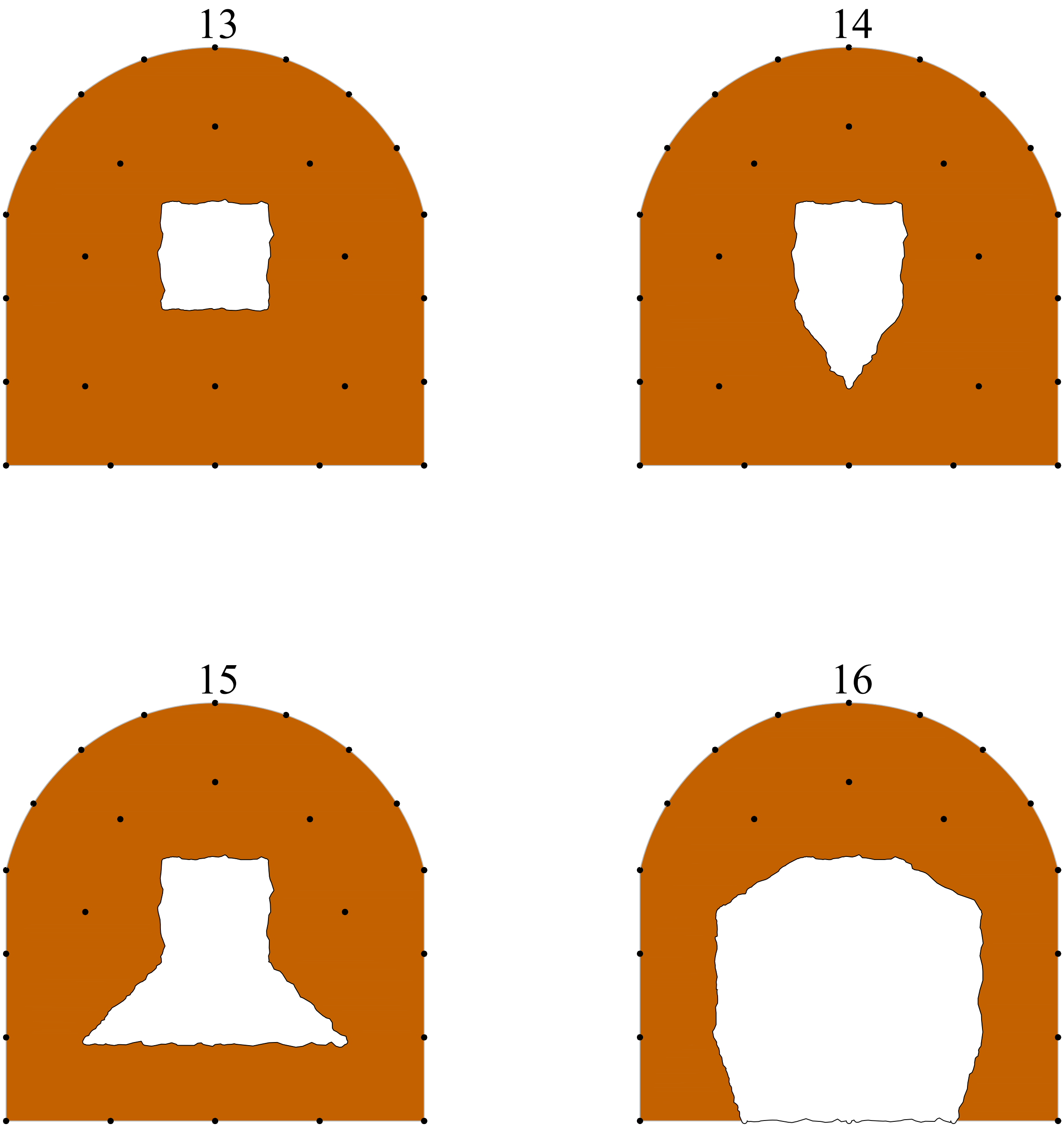

Figure 2.116 - Graphical representation of the detonation instants 13 - 16

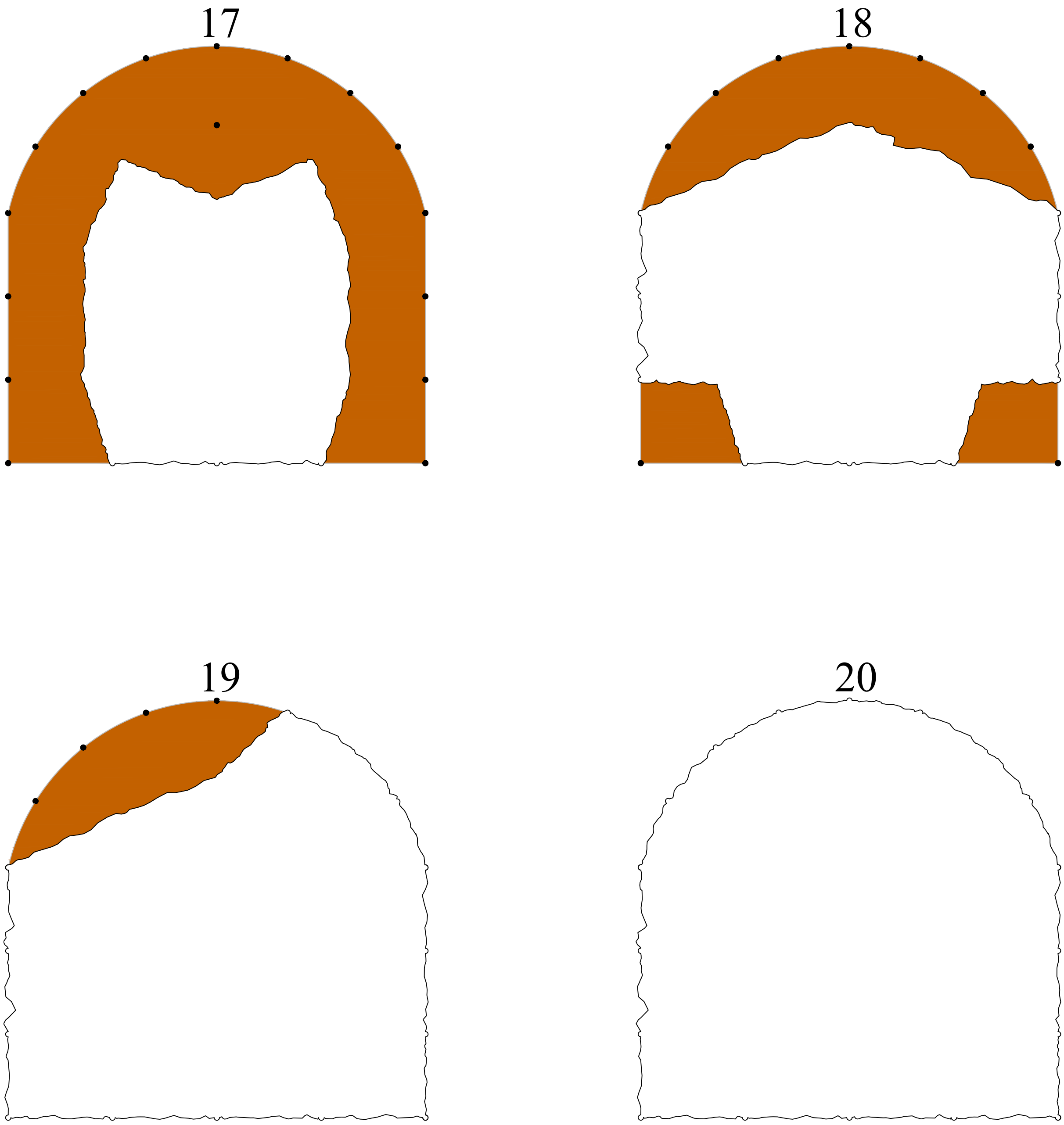

Figure 2.117 - Graphical representation of the detonation instants 17 - 20

2.6 - Design of the blast related to phase 3

The calculations are reported for the three cross-sections, i.e.: 2.5 x 2.5 m, 3.0 x 3.0 m, 3.5 x 3.5 m.

Phase 3, as already mentioned, provides the biggest changes on the point of view of both the organization and execution of the work. Drilling is made with a Jumbo (Boomer 104®) able to drill holes whose diameter is 50.8 mm and whose length is 3 m. The explosives are cartridged: a semi-gelatin (Tronex® Plus) 1 "1/2 x 10" at the bottom hole and an Emultex CN® emulsion 1 "1/2 x 10" at the column are employed. Both the calculations and schemes used are shown below.

The cut is the same for the 3 cross sections. For this reason, the calculations and schemes of the cut will be shown only once, whereas, as for the production and contour blast-holes, every case will be described in detail. The cut is placed in the middle of the cross-section.

Since a larger diameter bit is not available and not even a reamer, more than one empty hole having the same diameter of the blast-holes are placed at the center of the cut.

By evaluating the blasted volume at the first instant of detonation, the equivalent diameter of the central empty hole has been determined. It is made by 4 holes having a diameter of 50.8 mm and spaced by 7.5 cm with respect to the axis. The equivalent diameter, according to the formula (2.9) is:

$$D = \sqrt{4} \cdot 50.8 = 101.6\ mm \tag{2.9}$$

A graphic representation of the empty holes and of the equivalent diameter are shown in Figure 2.118:

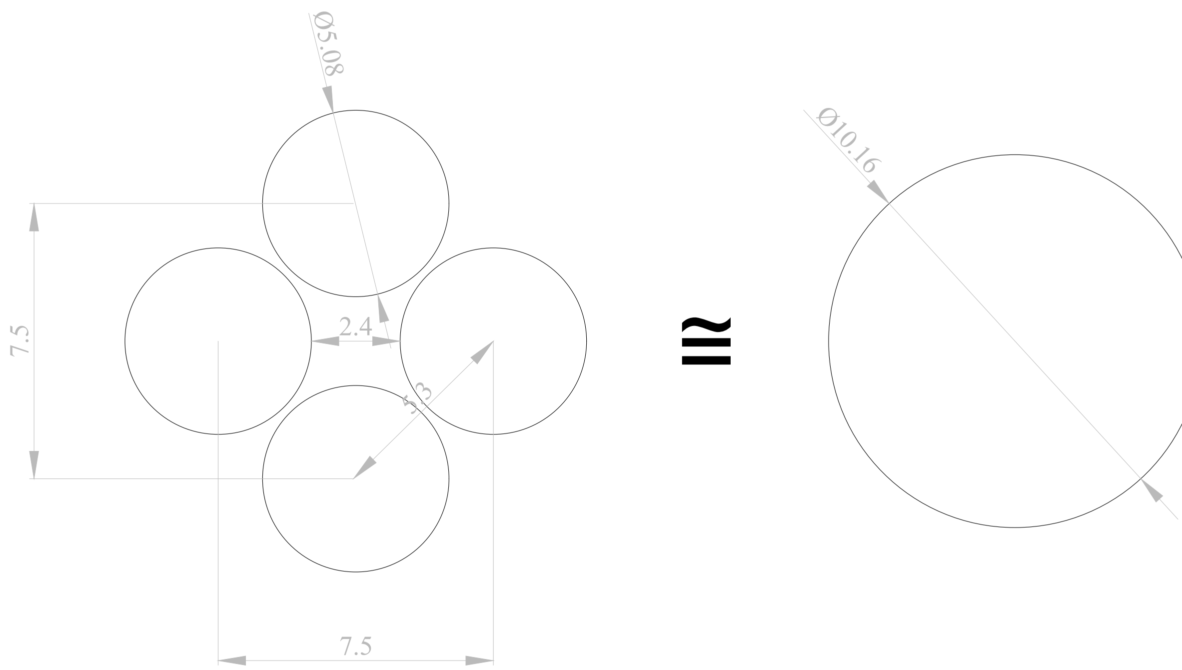

Figure 2.118 - Representation of the empty holes of the cut and of the corresponding equivalent diameter

Using the empirical graph shown in Figure 2.119, the theoretical pull of the blast was evaluated. Knowing the length of the holes (3m) and the fictitious diameter of the large empty hole (101.6 mm), a 92.2% round efficiency was estimated.

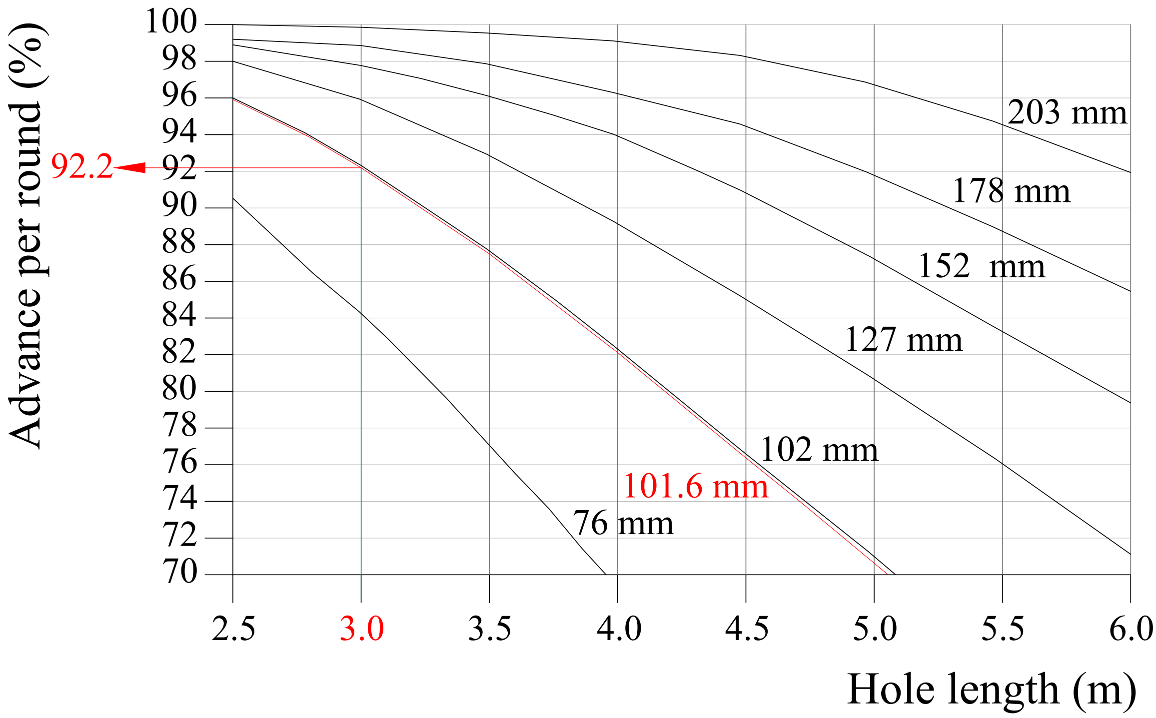

Figure 2.119 - Evaluation of the advance per round knowing the hole length and the fictitious diameter of the empty hole of the cut

The following step consists in positioning the holes of the first square, according to Figure 2.120. The distance between the centre of each blast-hole and the centre of the fictitious diameter is 15 cm. Therefore, the first square of the cut has a side of:

$$L_1 = \sqrt{2} \cdot 15 = 21\ cm \tag{2.10}$$

The graphical representation of the first square is shown in Figure 2.121.

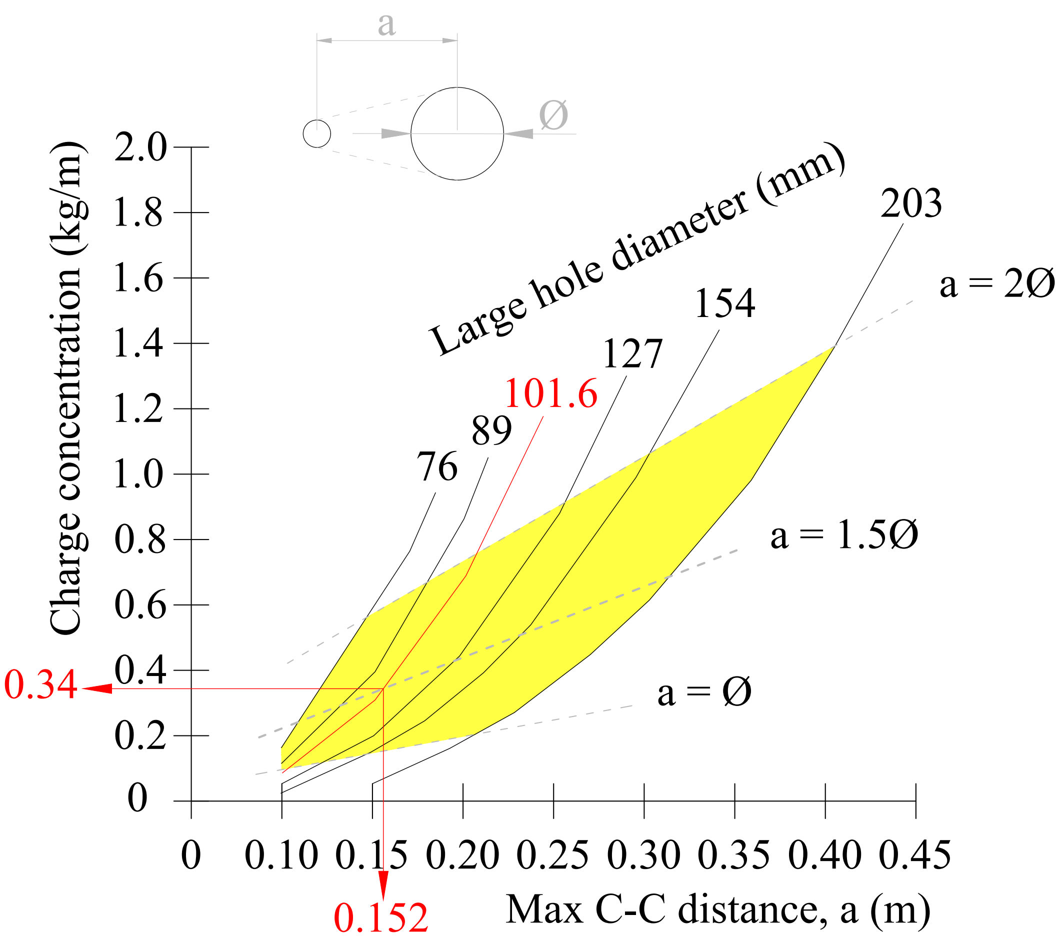

Figure 2.120 - Empirical graph used to determine the distance between the blast-holes of the first square of the cut from the center point of the empty holes and the linear charge concentration knowing the fictitious diameter of the empty hole

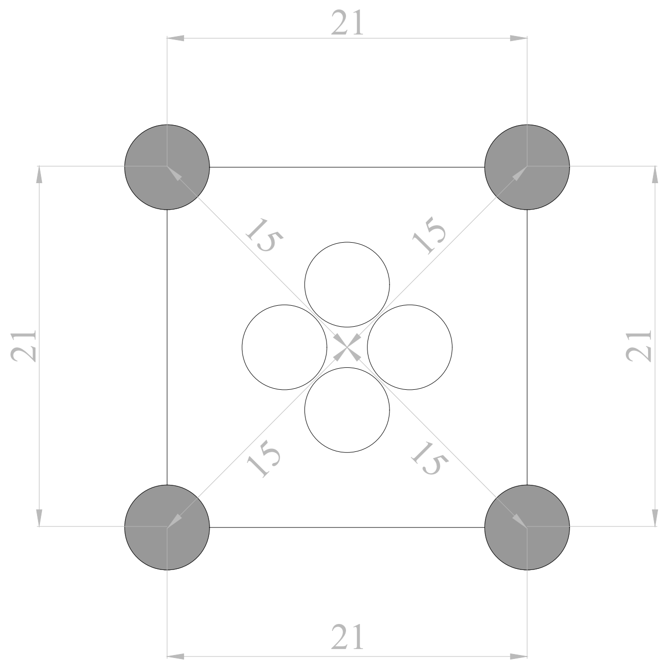

Figure 2.121 - Quoted scheme of the first square of the cut

It is then possible to proceed with the positioning of the second square, using the empirical graph shown in Figure 2.122. Once the side of the previous square is known, the burden of the next is determined, which in this case turns out to be equal to 21 cm. The scheme representing the second square of the cut and the relative dimensions is given in Figure 2.123. The second square of the has a side of 44 cm:

$$L_2 = \sqrt{2} \cdot \left(21 + \frac{21}{2}\right) = 44\ cm \tag{2.11}$$

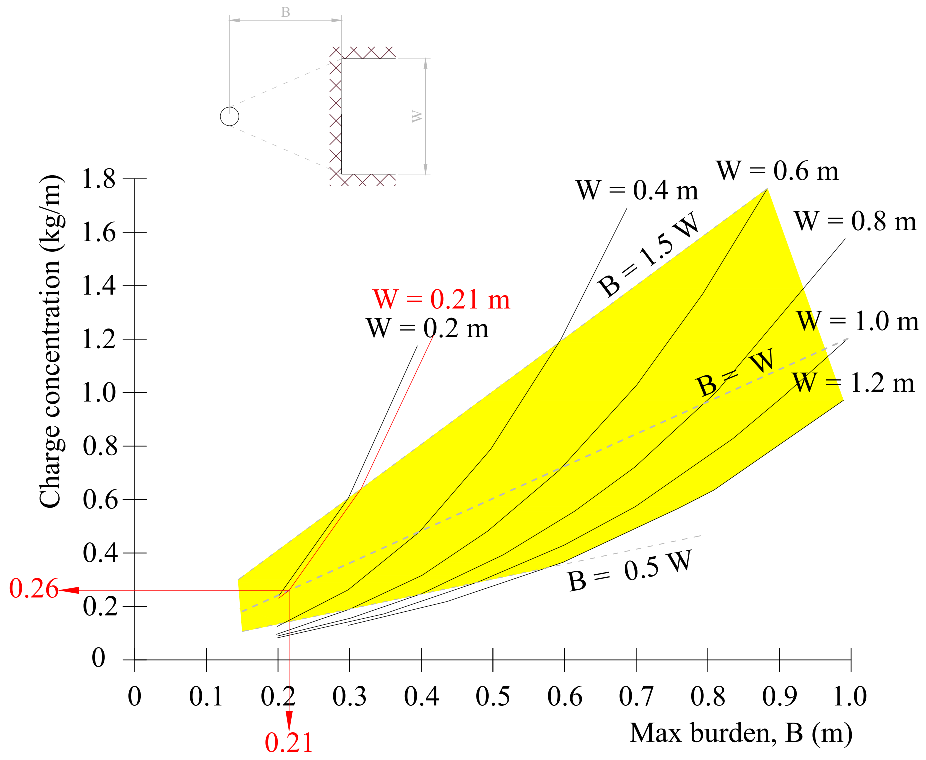

Figure 2.122 – Empirical graph used to determine the distance between the second square of the cut and the first (burden) and the linear charge concentration, knowing the value of the side of the first square of the cut

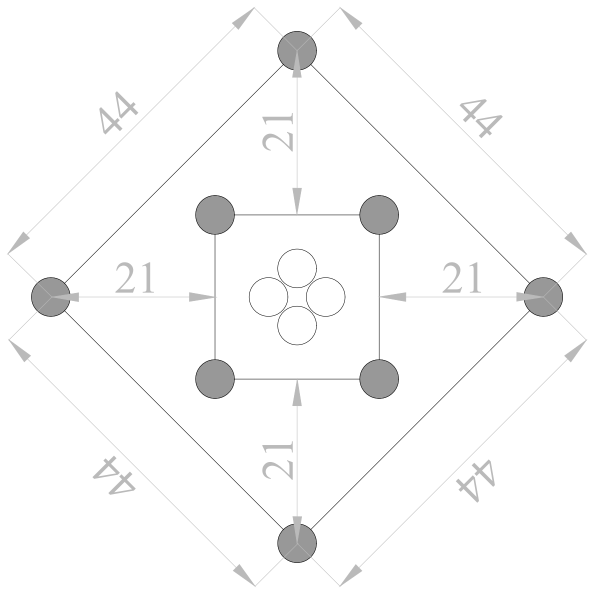

Figure 2.123 - Quoted scheme of the second square of the cut

At this point the insertion of a further square was considered, given the extent of the burden with respect to the section boundary. Considering the distance between the contour of the tunnel and the second square of the cut, it was decided not to add further squares. Therefore, the cut in phase 3 has only two squares.

From the empirical diagrams, the linear charge concentration obtained was respectively: 0.34 kg/m for the first square and 0.26 kg/m for the second. The average linear charge concentration for the cut turns out to be 0.30 kg/m. As for the stemming of the first square, its length is equal to the distance between the blast-holes and the central point of the empty holes.

For simplicity sake, it was decided to proceed with the same charge for the blast-holes composing the cut. The explosive Tronex® Plus in 1 "1/2 x 10" cartridges is placed at the bottom, and Emultex CN® emulsion in 1 "1/2 x 10" cartridges is used as column charge.

In the cut-holes, a cartridge of Tronex® Plus is placed at the bottom hole, then the blast-holes are loaded with 3 cartridges of Emultex CN®, leaving a stemming of 15 cm. Among the cartridges, 4 intermediate stemming were positioned, two equals to 46 cm and two equals to 34 cm. Due to the presence of intermediate stemming, a 10 g/m strand of donating cord (Britacord® NP 10R) is used, in order to ensure the propagation of the detonation along the holes.

A Tronex® Plus cartridge weighing 0.348 kg and 4 Emultex CN® cartridges with a total weight of 0.999 kg are then introduced in each blast-hole: the total charge per hole, then, amounts to 1.680 kg. The linear charge concentration is 0.56 kg/m, which is slightly higher than that found through the empirical graphs.

A scheme of a cut-hole is shown in Figure 2.124.

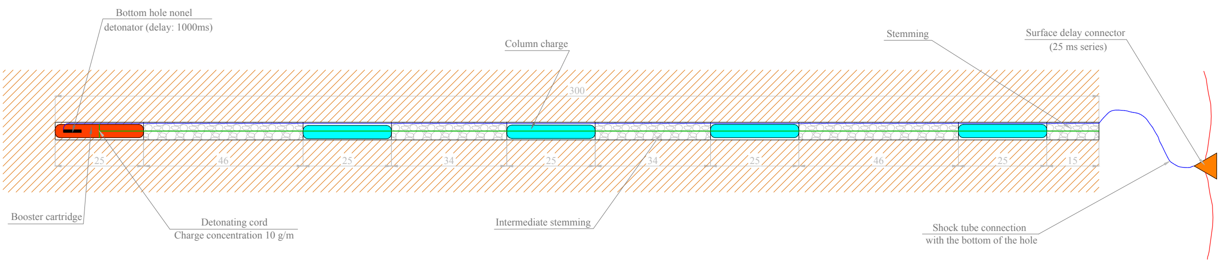

Figure 2.124 - Detail of a cut blast-hole with charge and initiation system

The next step consists in designing the production holes: the linear charge concentration and the burden are defined for both production and contour blast-holes.

The diameter of the hole (50.8 mm) being known, the linear charge concentration l_b (2.38 kg/m) and the burden B (1.18 m) are easily found (Figure 2.125). These values will be used as input data for Table 2.15, which is needed for sizing both production and contour holes.

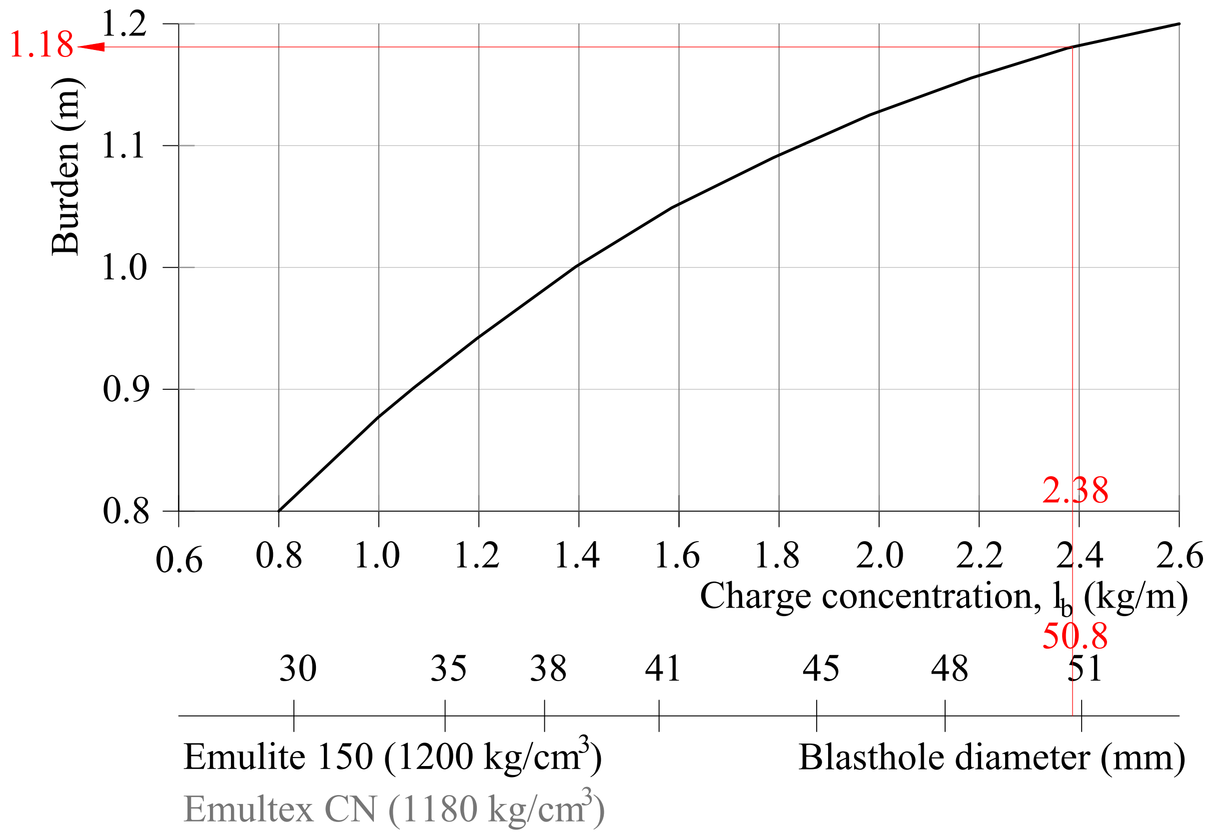

Figure 2.125 - Empirical graph used to determine the linear charge concentration ant the burden for production and contour holes for a 50.8 mm diameter hole charged with an emulsion

As for the production blast-holes (upwards, horizontal, downwards), Table 2.15 shows that the burden is equal to B, previously determined on the experimental graph and therefore equal to 1.18 m. The spacing is 1.1 B for the upwards and horizontal holes (therefore, 1.30 cm), and 1.2 B (therefore, 1.42 m) for the downwards holes.

For simplicity sake, the design of the stoping holes has been defined by placing the holes around the cut with the same spacing.

When the cross-section is 2.5 m x 2.5 m, the distance between the cut and the contour of the tunnel is 92 cm. It was therefore decided to place the production holes exactly in the middle of this distance, so that the burden is 46 cm. The spacing, instead, is equal to 79 cm on the linear side, and to 67 cm on the crown.

The position of the production holes with respect to the cut is given in Figure 2.126.

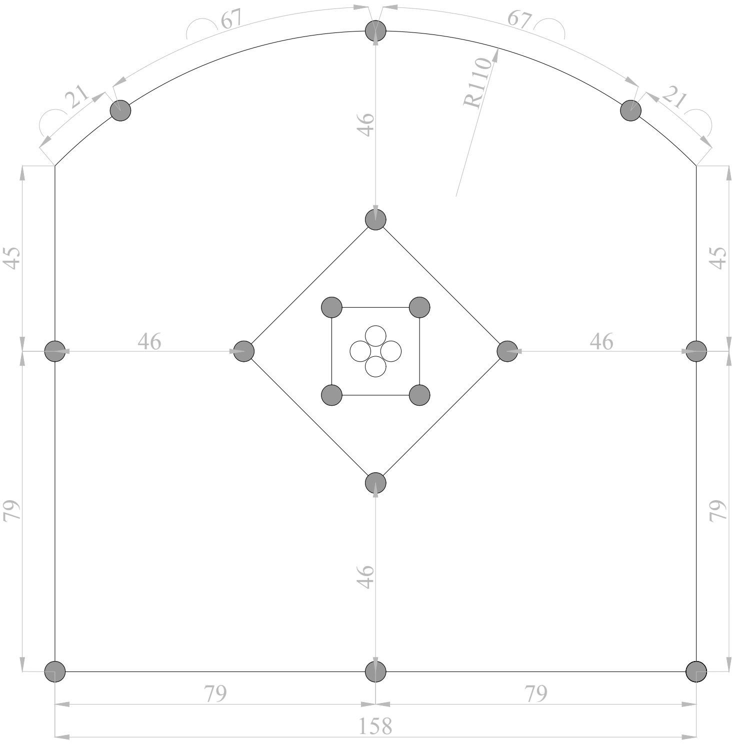

Figure 2.126 - Quoted scheme of the production holes for the section with dimension of 2.5 x 2.5 m

When the cross-section is 3.0 x 3.0 m, the distance between the cut and the tunnel contour is 118 cm. The production holes were placed at the center of this distance, so that the burden is 59 cm, as represented in Figure 2.127. The spacing between the holes in the linear side is 91 cm, is equal to 77 cm in the arched stretch.

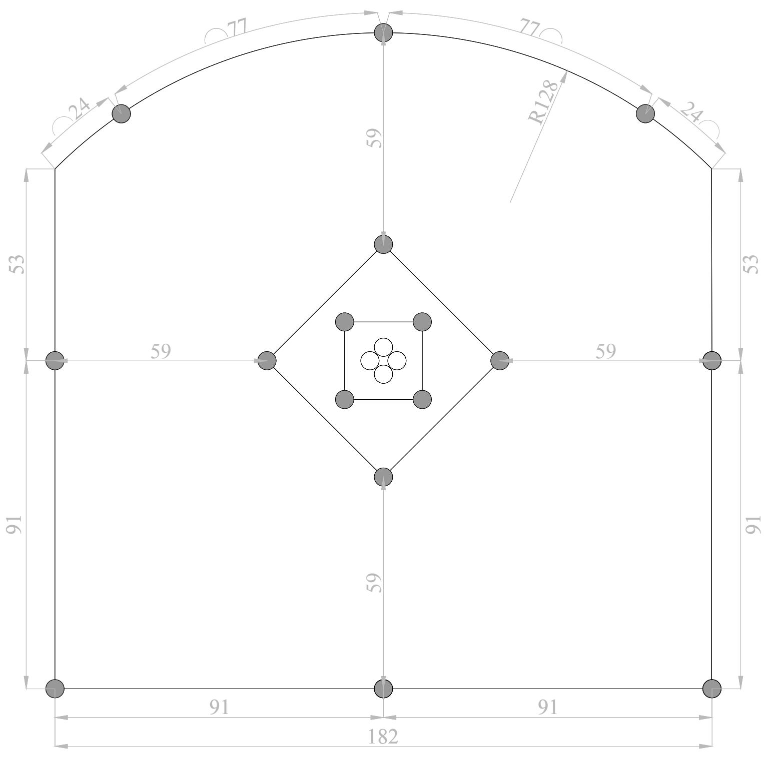

Figure 2.127 - Quoted scheme of the production holes for the section with dimension of 3.0 x 3.0 m

As for the third case (cross-section 3.5 x 3.5 m), the distance between the cut and the outline of the section is 142 cm. The production holes, located at the center of this distance, have a burden of 71 cm. The spacing is 104 cm in the linear side and 82 cm in the arched stretch. The position of the production holes is shown in Figure 2.128.

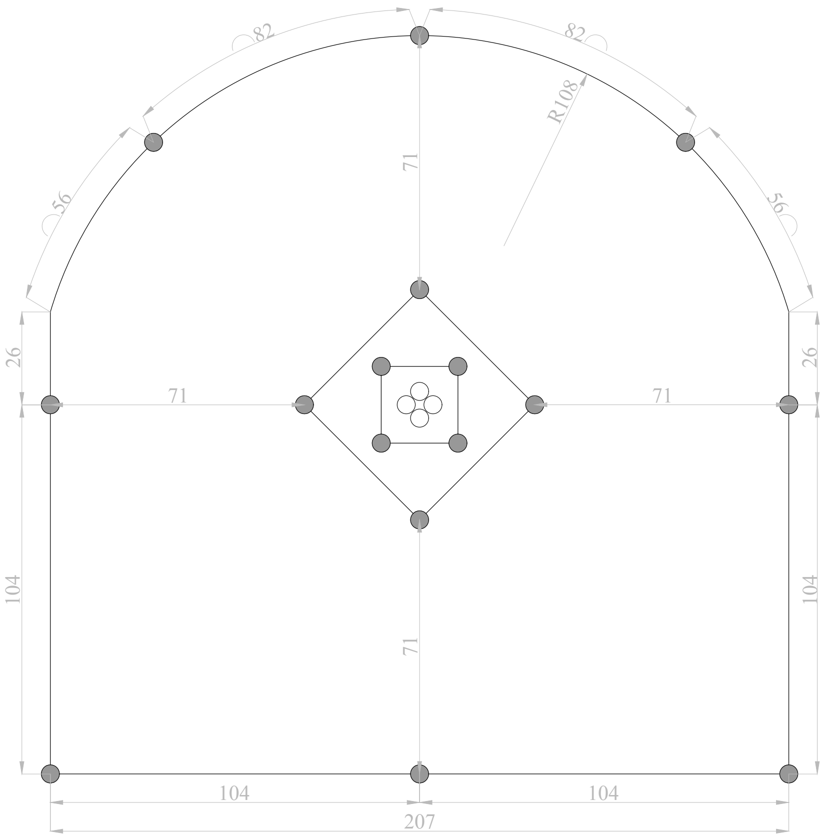

Figure 2.128 - Quoted scheme of the production holes for the section with dimension of 3.5 x 3.5 m

The charge of the stoping holes is the same for all positions with respect to the cut (upwards, horizontal, downwards). The length of the bottom charge is roughly 1/3 of the hole length (in this case 1 m) with a linear charge concentration l_b of 2.38 kg/m. The column charge is instead 0.5 l_b, hence 1.19 kg/m. The length of the stemming is 0.5 times the burden, and therefore 0.59 m.

The charge of the production holes was dimensioned by placing four Tronex® Plus cartridges at the bottom hole (dimension 1 "1/2 x 10"), six Emultex CN® 1 "1/2 x 10" cartridges and leaving a 46 cm stemming. Totally, 1.392 kg of Tronex® Plus (linear concentration of the bottom charge: 1.365 kg/m) and 1.998 kg of Emultex CN® (linear concentration of the column charge: 1.314 kg/m) were used. Therefore, an amount of 3.390 kg of explosive were employed, with a linear charge concentration which is about 1.13 kg/m. The linear charge concentration at the bottom is considerably lower than that recommended by the empirical table. However, this solution is acceptable as the burden is lower than that suggested by the table.

A scheme of a charged blast-hole pertaining to the stoping group is shown in Figure 2.129.

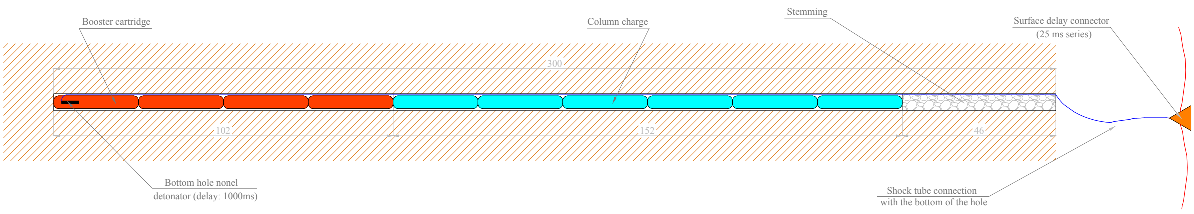

Figure 2.129 - Detail of a production blast-hole with charge and initiation system

As for the contour holes, the calculation for the floor's blast-holes is the same that was adopted in step 1, and 2 for the same functional group (using Table 2.15). The contour holes in the walls and the roof were designed by applying the empirical rules of the smooth blasting (E. I. du Pont de Nemours & Co., 1978) showed in Table 2.23. In this case, being the diameter of the holes 50.8 mm, the recommended spacing is 75 cm and the burden 106 cm.

As for the 2.5 x 2.5 m section, the burden of the holes in the contour is 46 cm. The spacing is respectively 83 cm as for the floor holes, 62 cm on the vertical walls, and 72 cm on the crown. The outline of the contour holes is shown in the Figure 2.130.

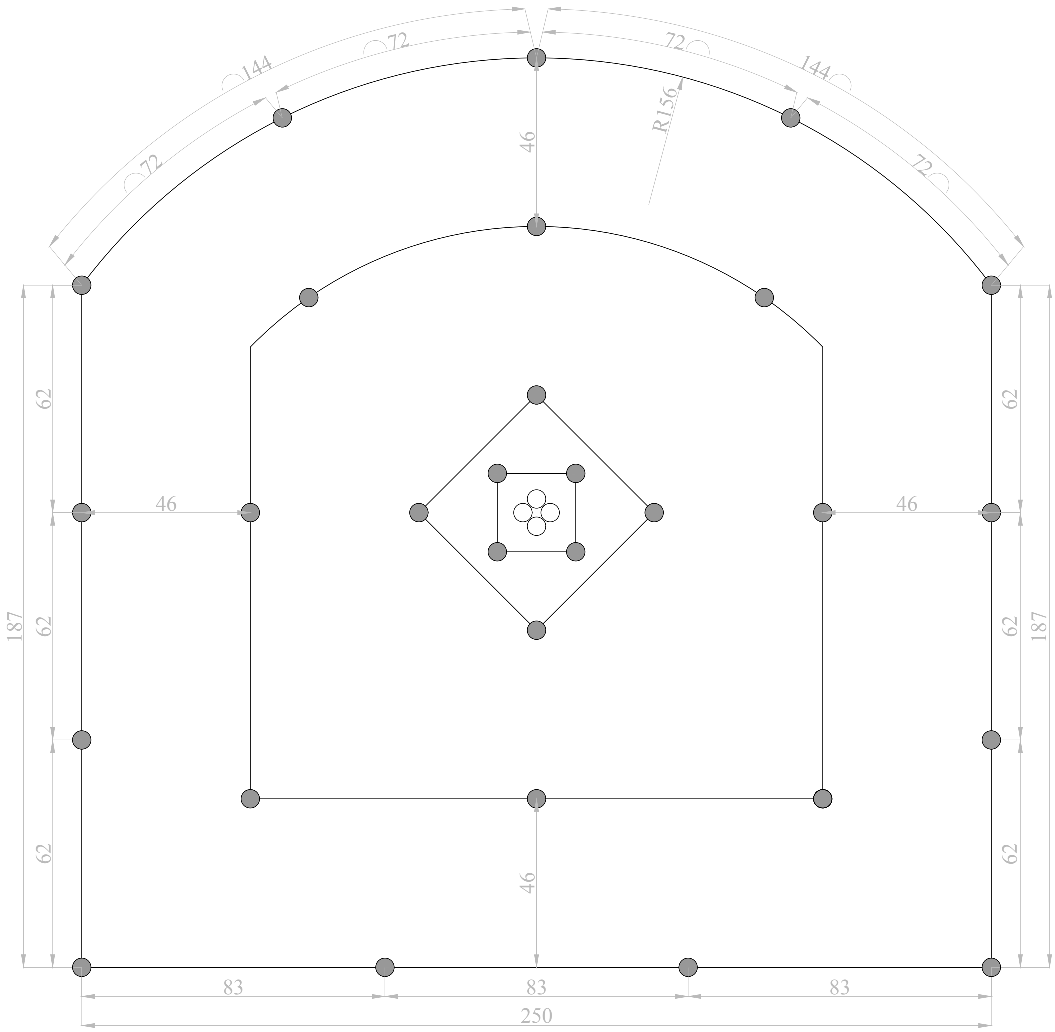

Figure 2.130 - Quoted scheme of the contour holes for the section with dimension of 2.5 x 2.5 m

As for the 3.0 x 3.0 m excavation section, contour holes have a 59 cm burden. The spacing is 100 cm (floor holes), 75 cm (vertical walls), and 58 cm (crown). The outline of the contour holes is shown in the Figure 2.131.

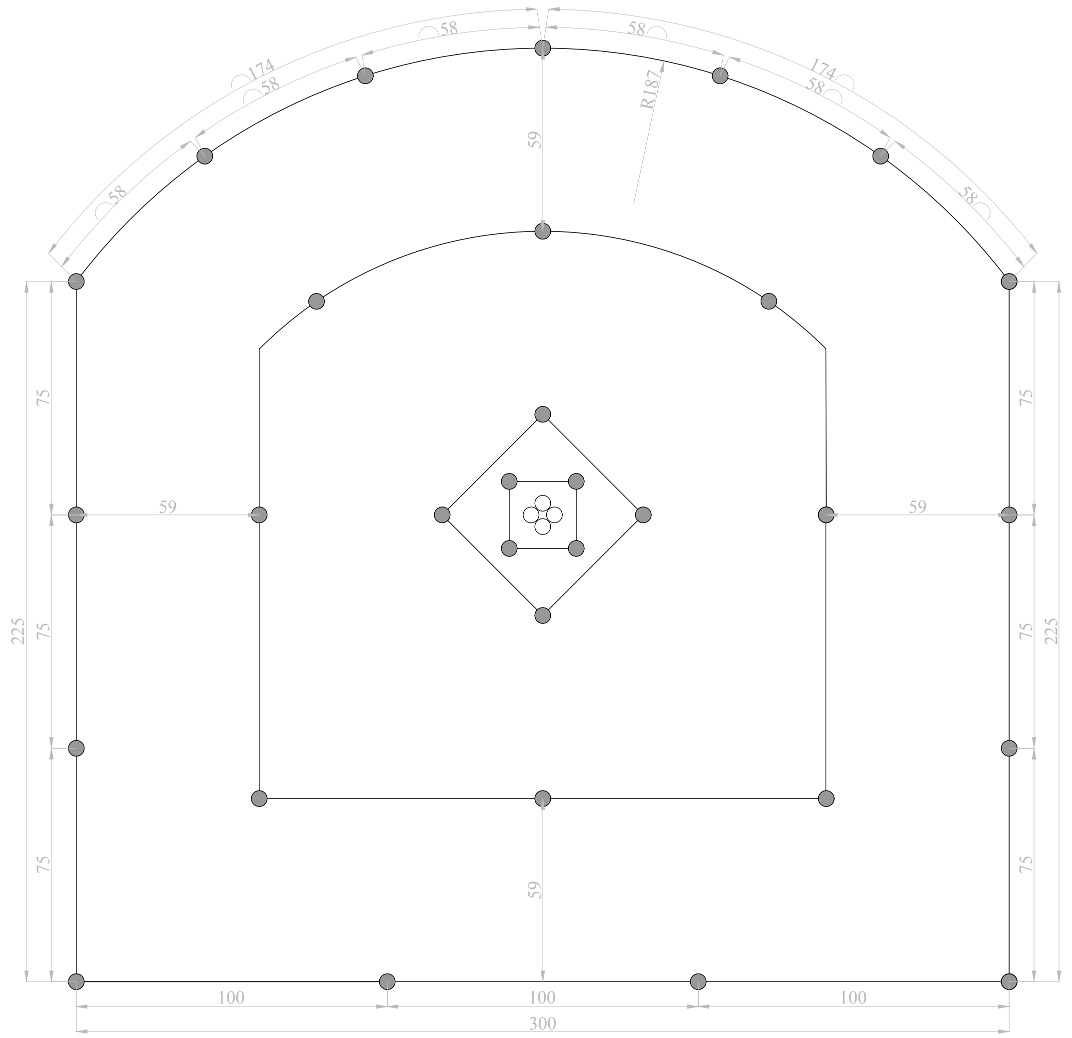

Figure 2.131 - Quoted scheme of the contour holes for the section with dimension of 3.0 x 3.0 m

As for the 3.5 x 3.5 m excavation section, the burden is 71 cm. The spacing between the blast-holes is 117 cm on the floor, 70 cm on the walls, and 60 cm on the crown.

The outline of the contour holes is given in Figure 2.132.

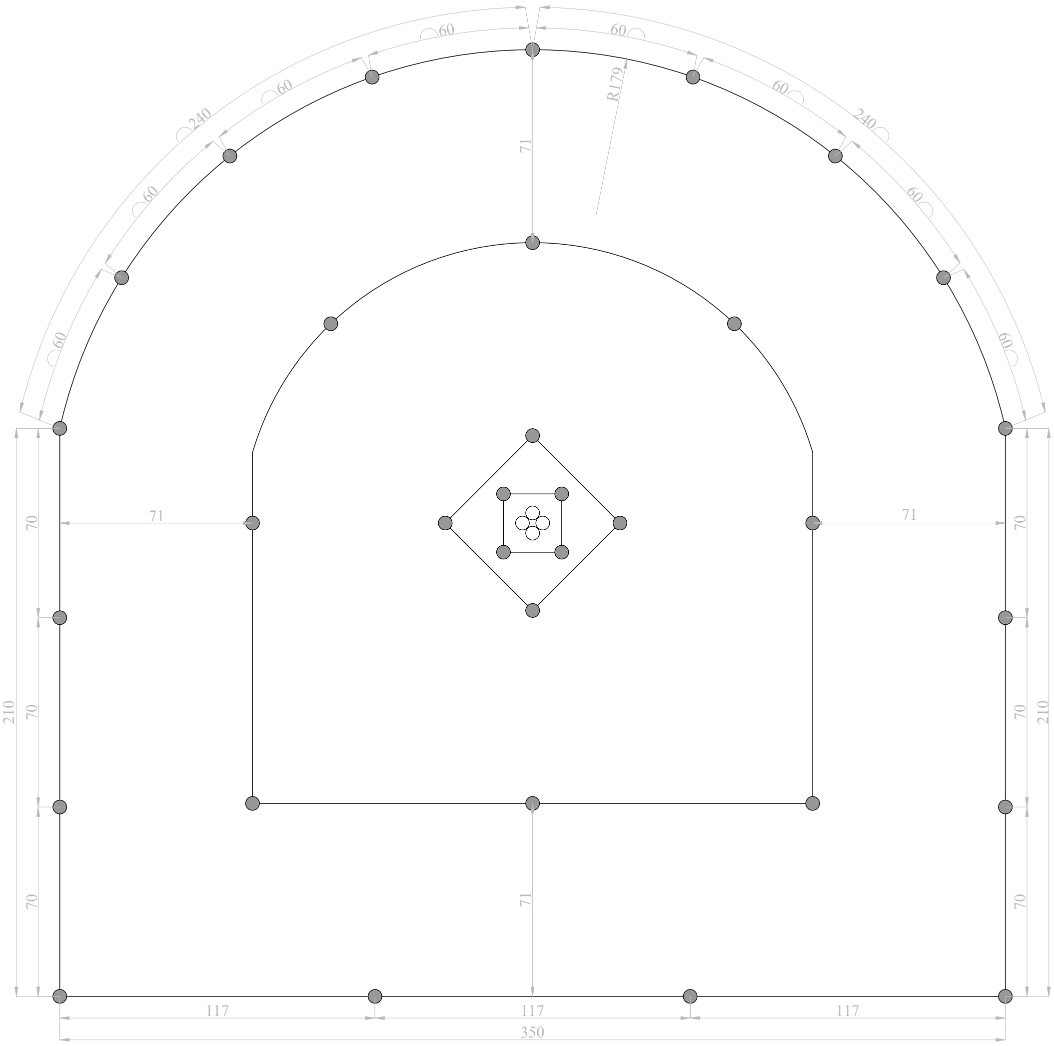

Figure 2.132 - Quoted scheme of the contour holes for the section with dimension of 3.5 x 3.5 m

The charge of the contour holes of the floor was dimensioned by placing four Tronex® Plus cartridges at the bottom (1"1/2 x 10"), seven 1"1/2 x 10" Emultex CN® cartridges and leaving a 20 cm stemming. Totally, 1.392 kg of Tronex® Plus (linear concentration of the bottom charge: 1.365 kg/m) and 2.331 kg of Emultex CN® (linear concentration of the column charge: 1.310 kg/m) were used, giving rise to a 3.723 kg charge/hole, with a linear charge concentration of about 1.33 kg/m. The linear charge concentration for the bottom hole is considerably lower than that recommended by the empirical table. However, it is acceptable, as the burden is lower than that suggested in the same table.

As for the holes in the walls and the roof, the charge is different: the holes are loaded with a detonating cord (Pentacord® special 80P), having a linear charge concentration of 80 g/m. The diameter of the detonating cord is 10.3 mm. The decoupling is then 4.9.

Figure 2.133 shows the scheme of a charged floor hole. Figure 2.134 shows the scheme of a contour hole on the walls and the roof charged with detonating cord. Figure 2.135 shows a section of a contour hole on the walls and the roof charged with the detonating cord, where the decoupling is noticeable.

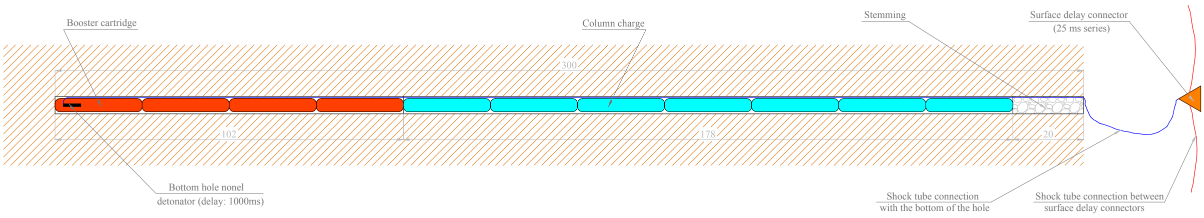

Figure 2.133 – Detail of a contour (floor) blast-hole with charge and initiation system

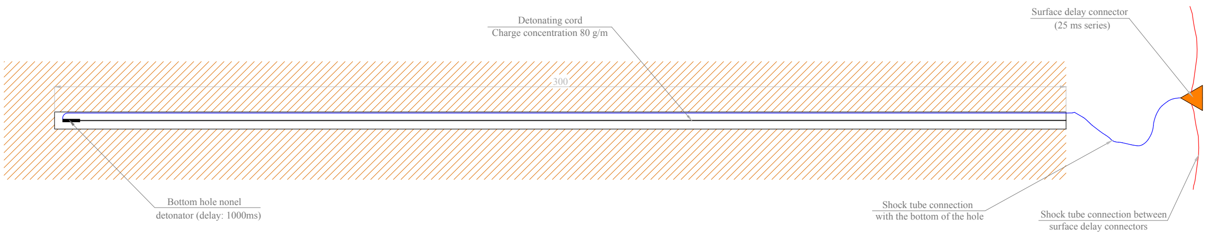

Figure 2.134 – Detail of a contour (walls and roof) blast-hole with charge and initiation system

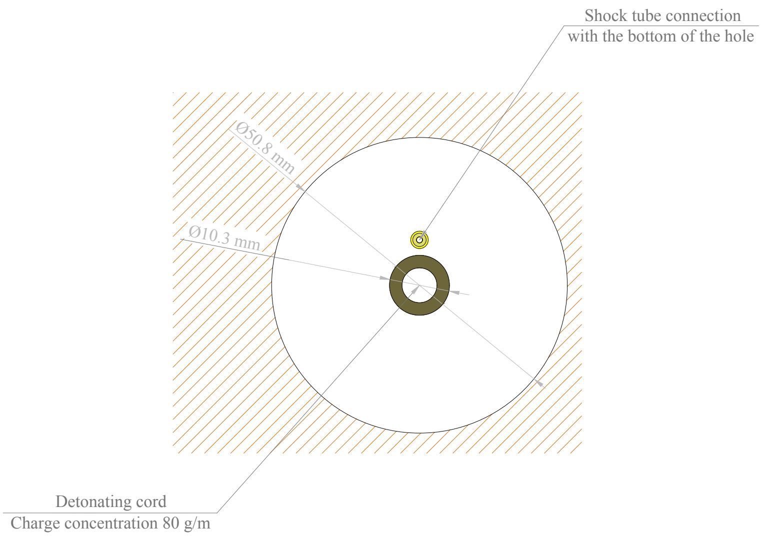

Figure 2.135 - Section of a charged contour hole (walls, roof)

Figure 2.136, Figure 2.137, Figure 2.138 show the drilling schemes for the three cross-sections examined in phase 3.

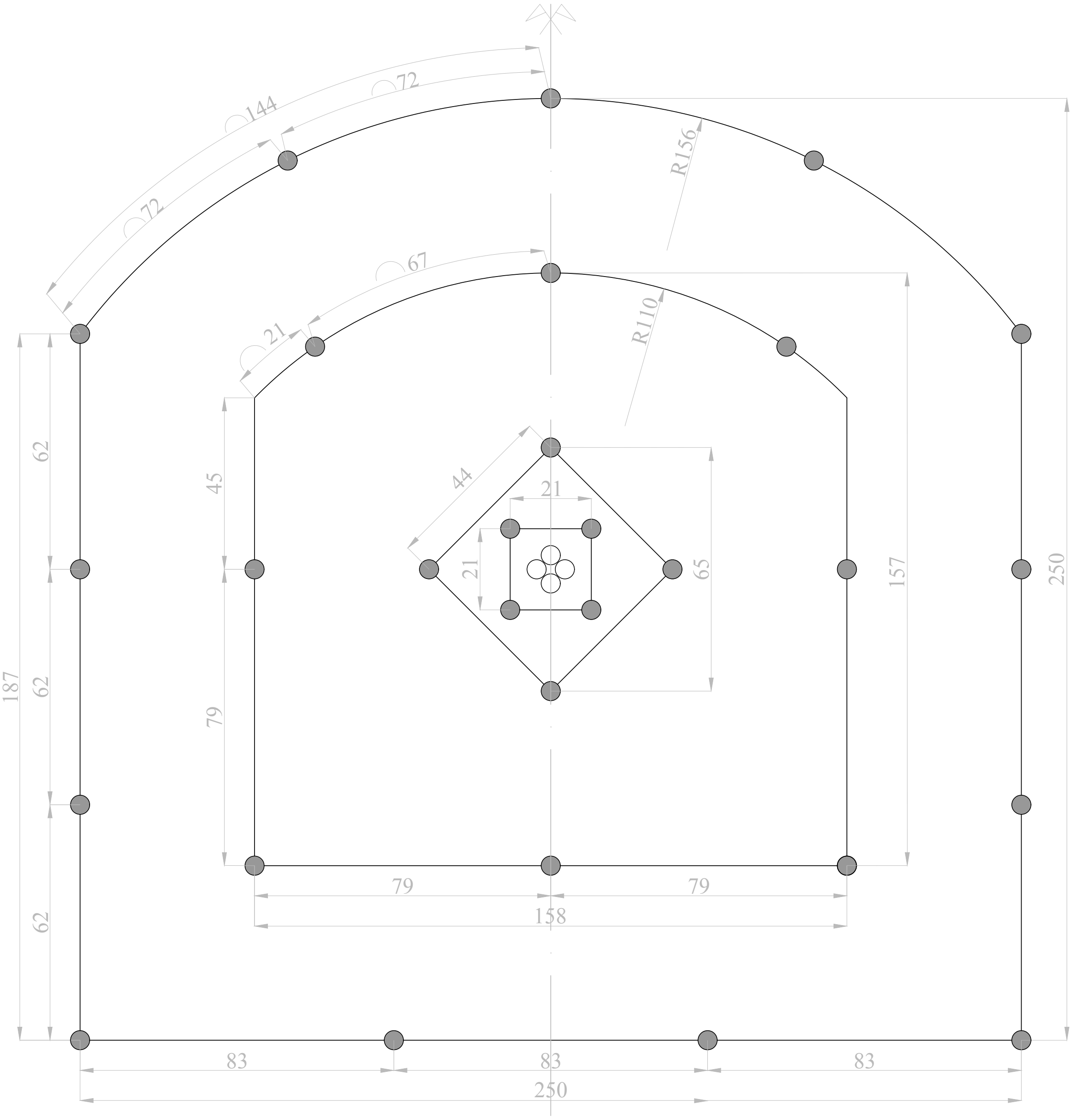

Figure 2.136 - Complete drilling scheme for the excavation section of 2.5 m x 2.5 m

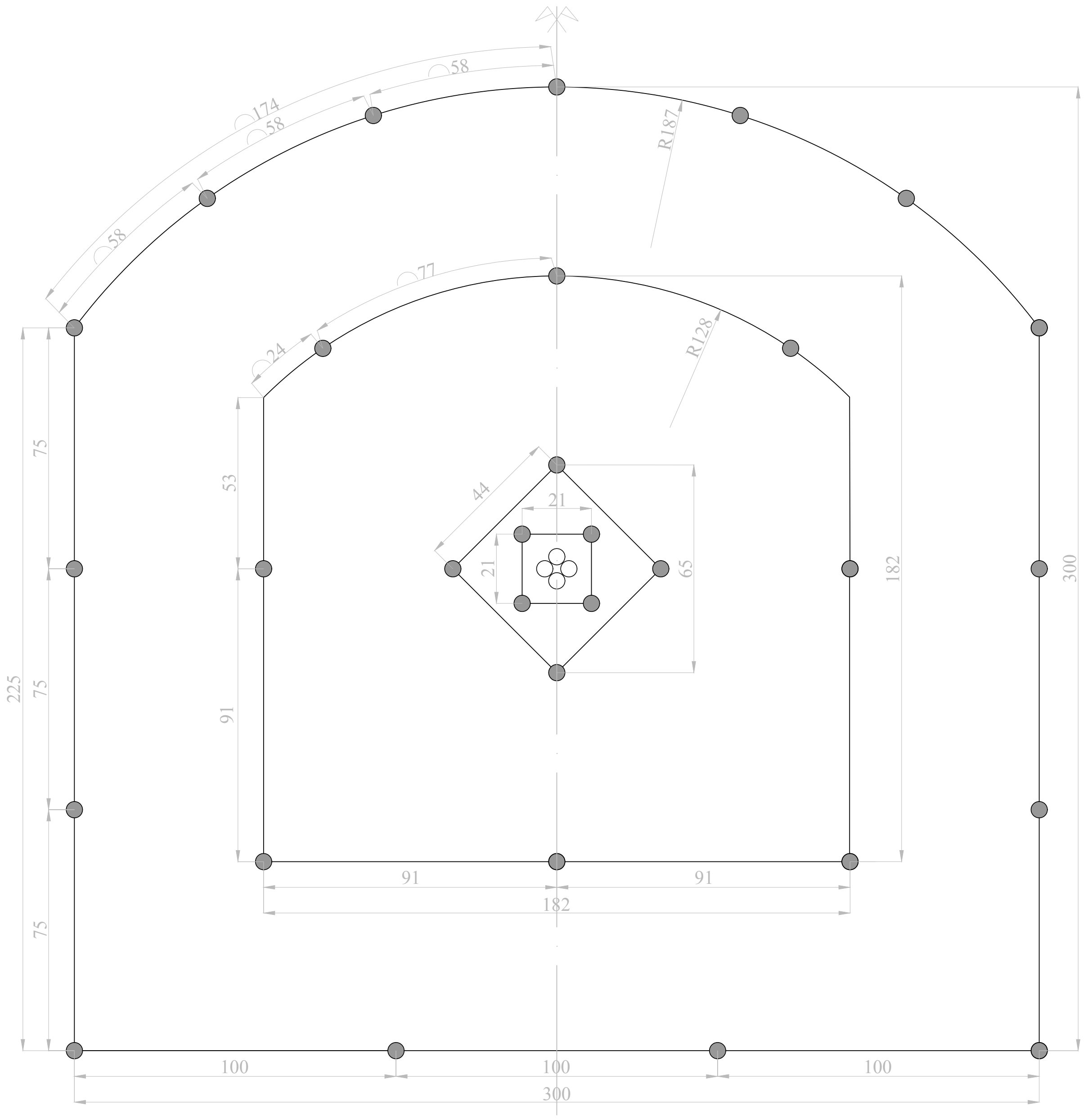

Figure 2.137 - Complete drilling scheme for the excavation section of 3.0 m x 3.0 m

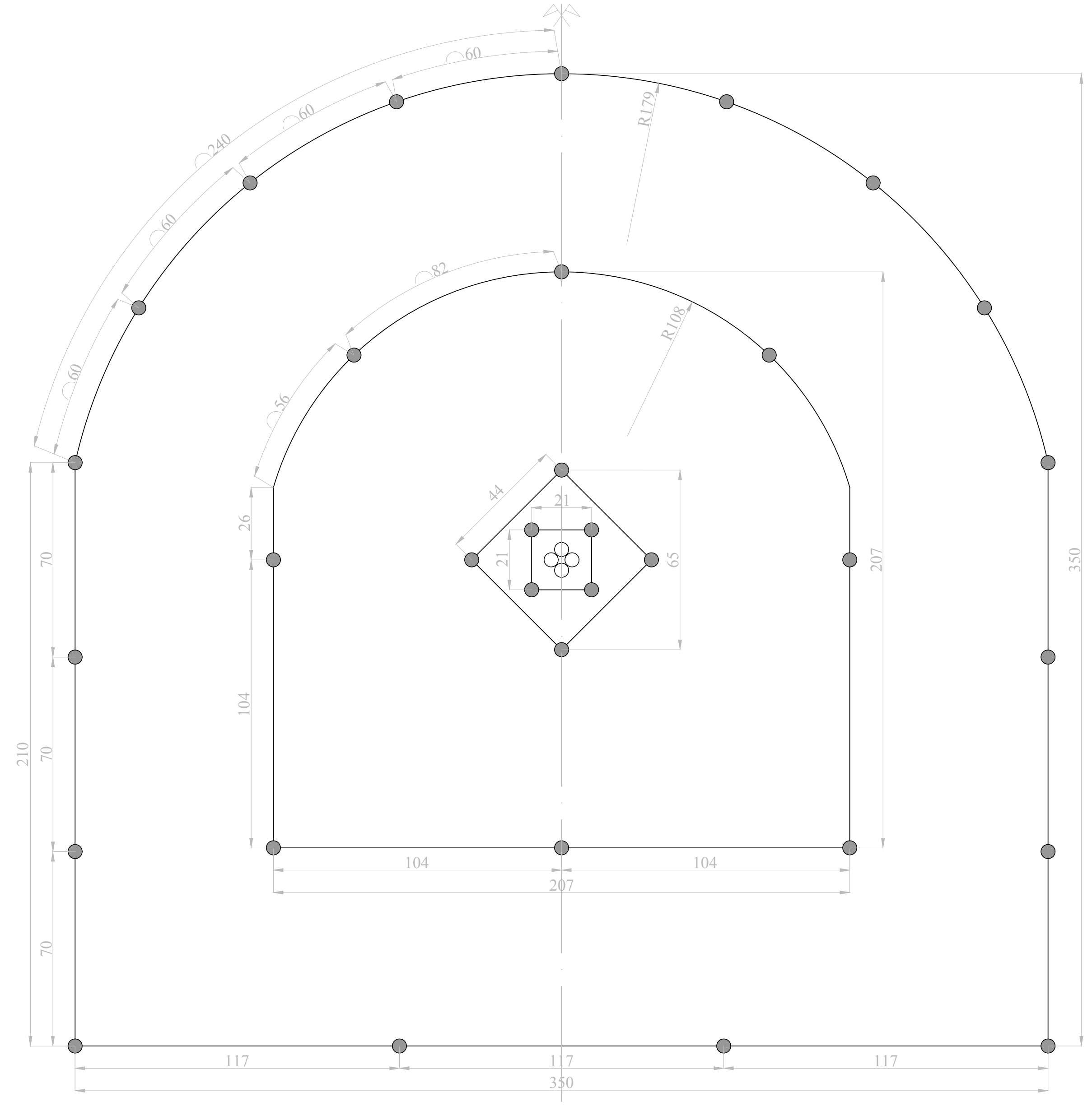

Figure 2.138 - Complete drilling scheme for the excavation section of 3.5 m x 3.5 m

Considering the maximum distances that Nonel cables must cover for the various cross-sections adopted, taking into account a safety margin for possible inaccuracies due to drilling, and the fact that the Nonel tubes must not be taut but loosened, a 4.8 m cable length of the detonators (non-electric detonators and non-electric dual detonators) was chosen.

In the following schemes, the detonation orders are shown:Figure 2.139, 90 and 92 show the triggering order of the surface delays connectors. The connection system between the various holes is also noticeable, together with the length of the connection tubes. Figure 2.140, 91 and 93 show the initiation of the in-hole detonators. The yellow arrows are a graphic aid to make easier the reading of the scheme, representing the detonation sequence.

Both Figure 2.139 and Figure 2.140 are related to the 2.5 m x 2.5 m cross-section, Figure 2.141 and Figure 2.142 to the 3.0 x 3.0 m cross-section and, finally, Figure 2.143 and Figure 2.144 to the 3.5 x 3.5 m cross-section.

For each cross-section a table containing the most significant data concerning the initiation system is presented (for each phase of the detonation order are reported the delay of the surface connector, the delay of the in-hole detonator, the number of the surface connectors and the number of the in-in hole detonators).

Table 2.31 is related to the 2.5 x 2.5 section, Table 2.32 to the 3.0 x 3.0 m section and, finally, Table 2.33 to the 3.5 x 3.5 m section.

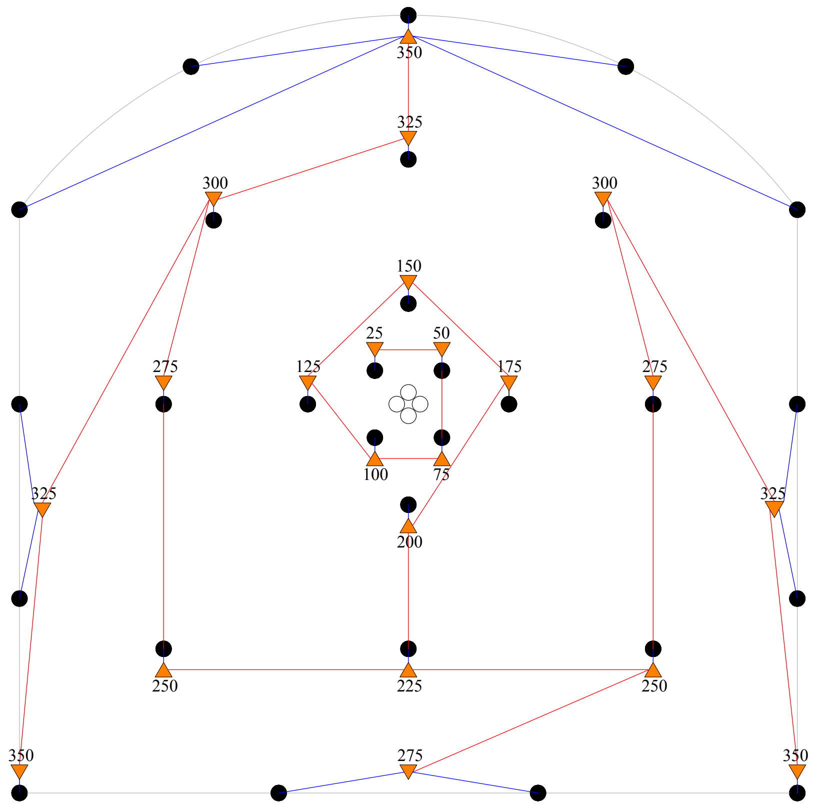

Figure 2.139 - Order of detonation of the surface delay connectors related to the 2.5 x 2.5 m section

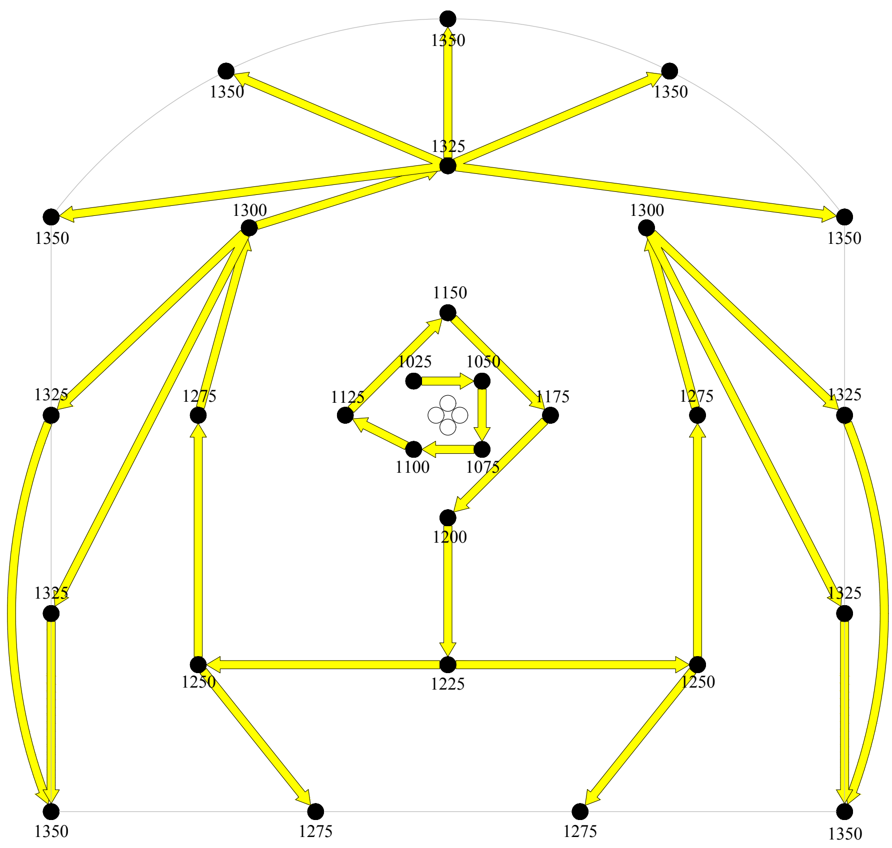

Figure 2.140 – Order of detonation of the in-hole detonators related to the 2.5 x 2.5 m section

Table 2.31 - Summary table of the main data related to the initiation system for the 2.5 x 2.5 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	2	2
11	25 ms	1000 ms	1275 ms	3	4
12	25 ms	1000 ms	1300 ms	2	2
13	25 ms	1000 ms	1325 ms	3	5
14	25 ms	1000 ms	1350 ms	3	7

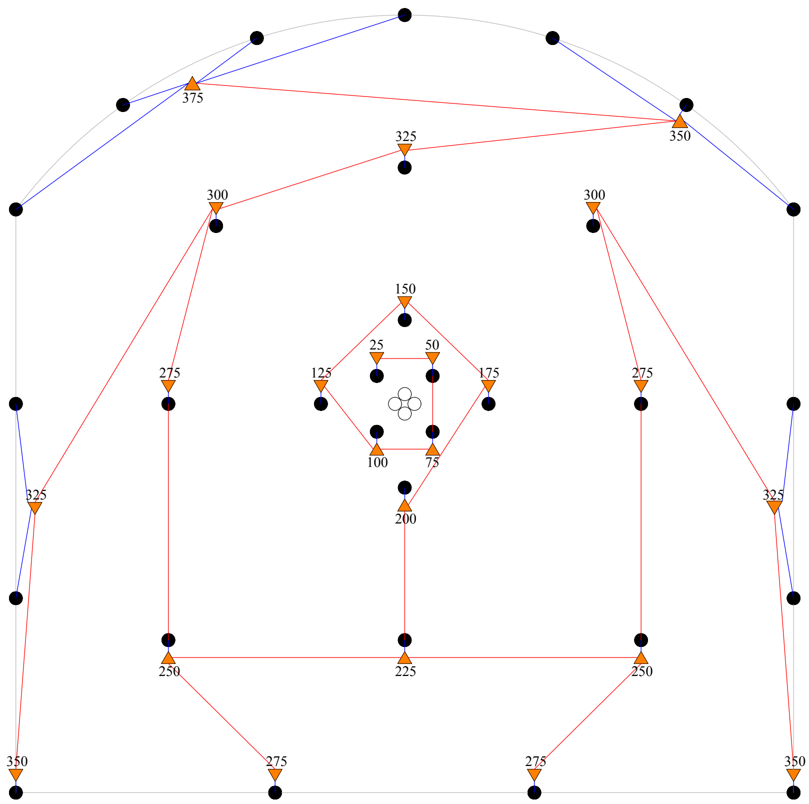

Figure 2.141 - Order of detonation of the surface delay connectors related to the 3.0 x 3.0 m section

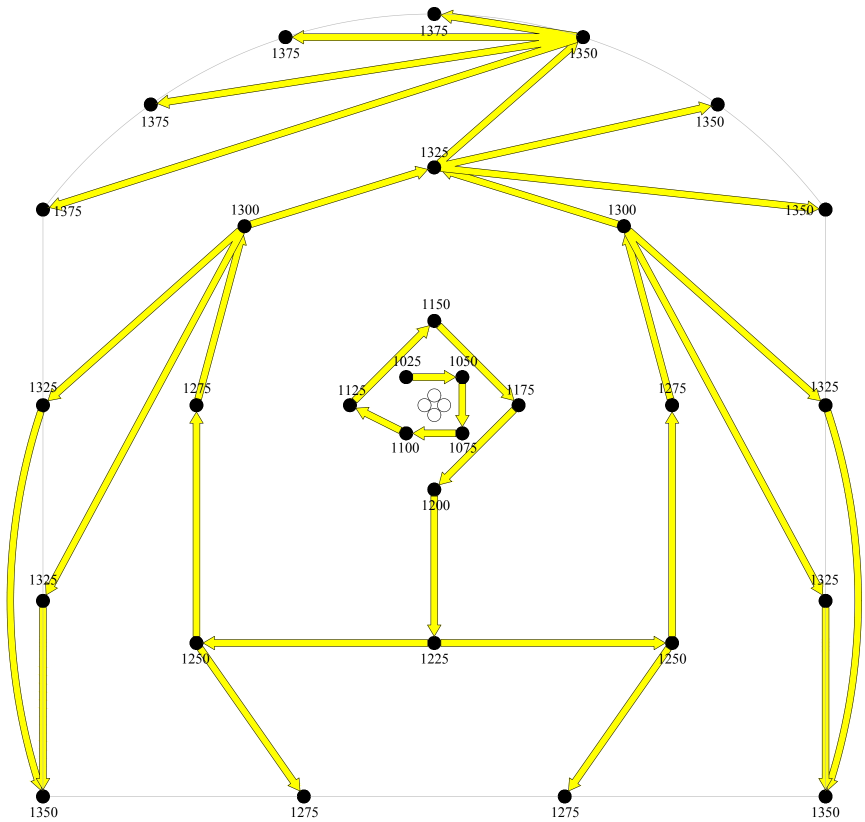

Figure 2.142 – Order of detonation of the in-hole detonators related to the 3.0 x 3.0 m section

Table 2.32 - Summary table of the main data related to the initiation system for the 3.0 x 3.0 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	2	2
11	25 ms	1000 ms	1275 ms	4	4
12	25 ms	1000 ms	1300 ms	2	2
13	25 ms	1000 ms	1325 ms	3	5
14	25 ms	1000 ms	1350 ms	3	5
15	25 ms	1000 ms	1375 ms	1	4

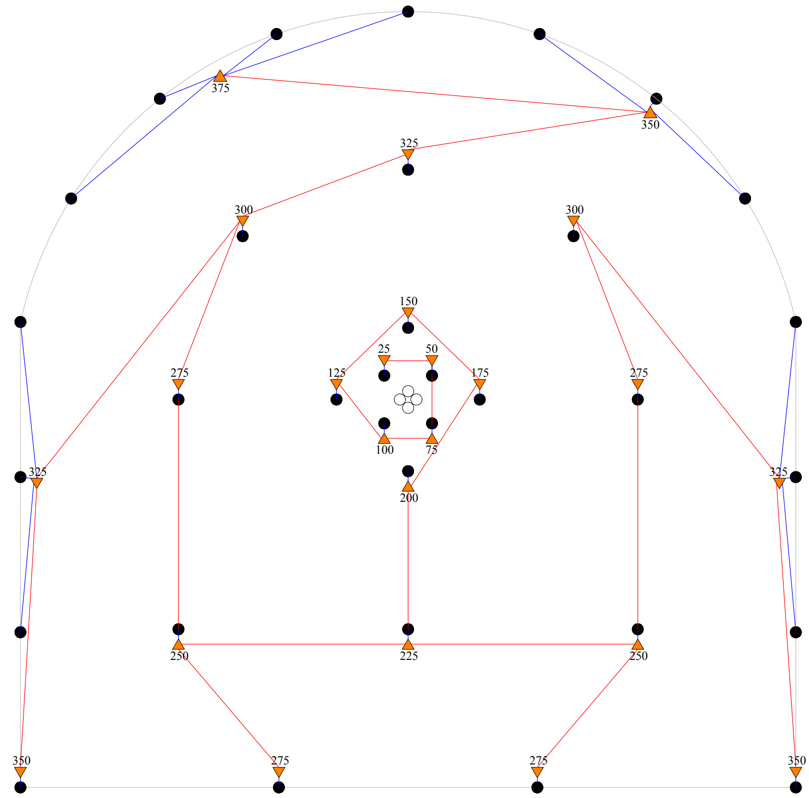

Figure 2.143 - Order of detonation of the surface delay connectors related to the 3.5 x 3.5 m section

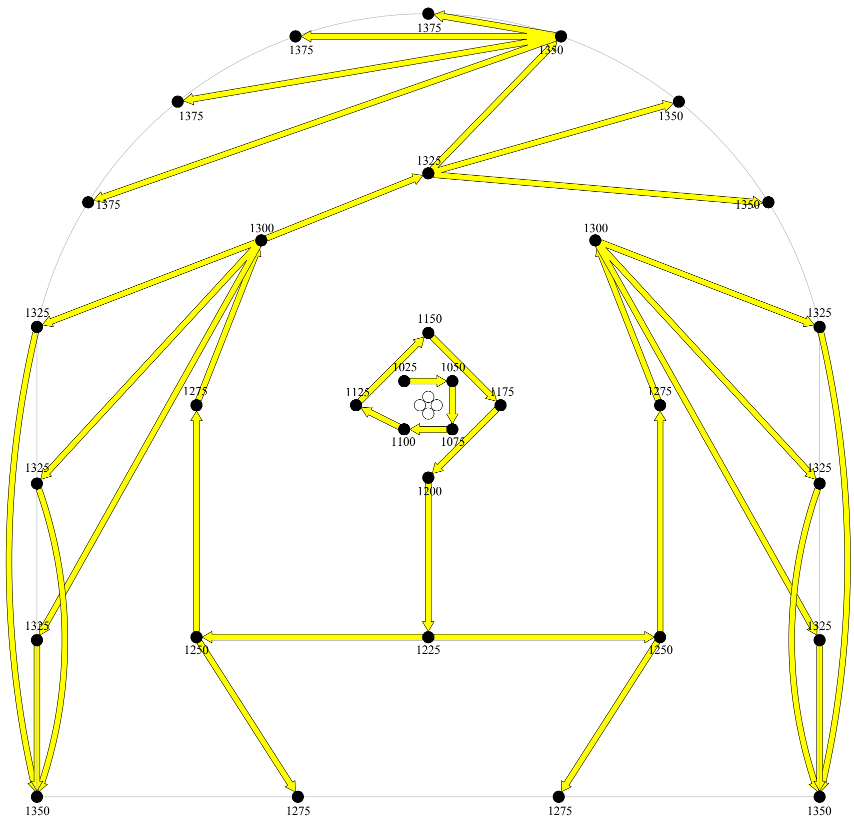

Figure 2.144 – Order of detonation of the in-hole detonators related to the 3.5 x 3.5 m section

Table 2.33 - Summary table of the main data related to the initiation system for the 3.5 x 3.5 m section

Detonation order	Delay of the surface connector	Delay of the in-hole detonator	Time of detonation of the explosive	No. of the surface connectors	No. of the bottom hole detonators
1	25 ms	1000 ms	1025 ms	1	1
2	25ms	1000 ms	1050 ms	1	1
3	25 ms	1000 ms	1075 ms	1	1
4	25 ms	1000 ms	1100 ms	1	1
5	25 ms	1000 ms	1125 ms	1	1
6	25 ms	1000 ms	1150 ms	1	1
7	25 ms	1000 ms	1175 ms	1	1
8	25 ms	1000 ms	1200 ms	1	1
9	25 ms	1000 ms	1225 ms	1	1
10	25 ms	1000 ms	1250 ms	2	2
11	25 ms	1000 ms	1275 ms	4	4
12	25 ms	1000 ms	1300 ms	2	2
13	25 ms	1000 ms	1325 ms	3	7
14	25 ms	1000 ms	1350 ms	3	5
15	25 ms	1000 ms	1375 ms	1	4

The following tables (Table 2.34 for the 2.5 x 2.5 m cross-section, Table 2.35 for the 3.0 x 3.0m cross-section and, finally, Table 2.36 for the 3.5 x 3.5 m cross-section) summarize the main data concerning the blast for each cross-section examined. The calculations are related to the whole blast and also to the individual functional groups of blast-holes. Specifically, the most important data are: volume blasted, amount of explosive, powder factor, volume to remove (using a 1.4 bulking factor), holes density, total drilled length, specific drilling, specific consumption of detonators. The volumes pertaining to each functional group were highlighted, for each cross-section examined, in Figure 2.145 (2.5 x 2.5 m cross-section), Figure 2.146 (3.0 x 3.0 m cross-section), and Figure 2.147 (3.5 x 3.5 m cross-section).

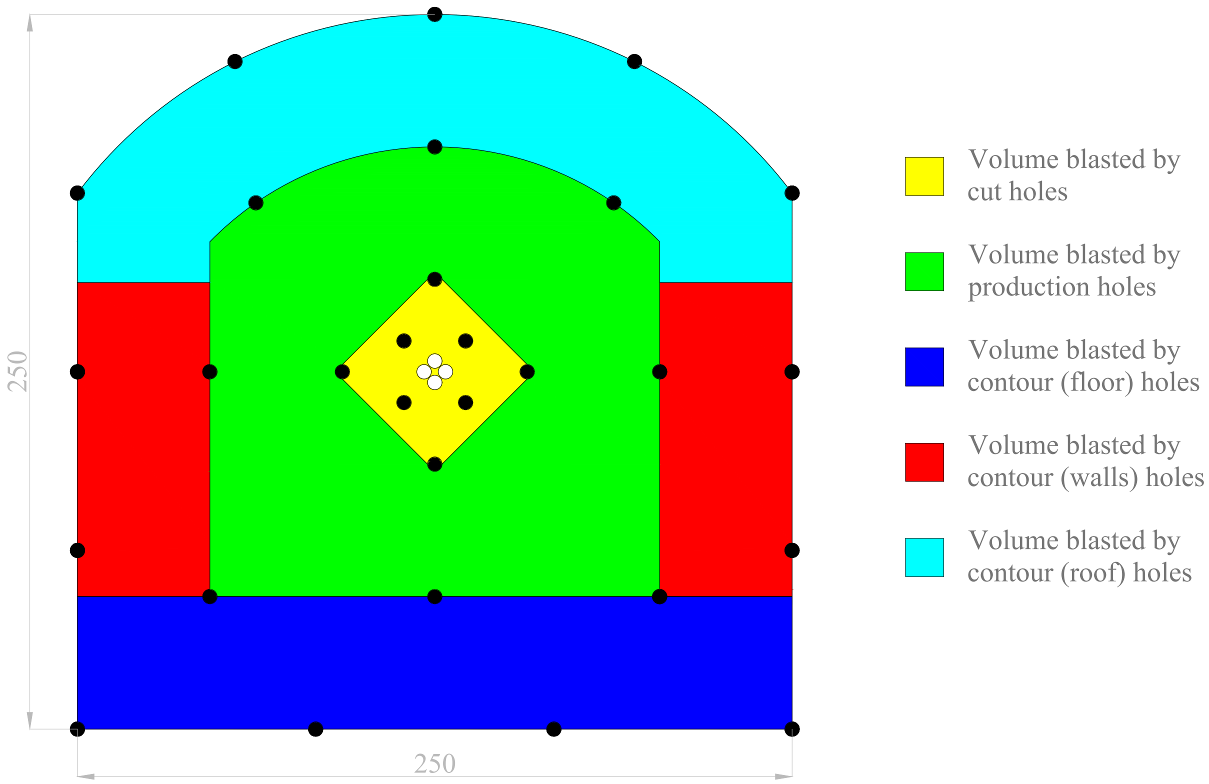

Figure 2.145 – Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 2.5 x 2.5 m section

Table 2.34 - Summary of the main data relating to the excavation section of 2.5 x 2.5 m

Cross section (m^2)		5.73
Design pull (m)		3.0
Actual pull (m)		2.77
Hole diameter (m)		50.8
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.661
	Amount of explosive (kg)	13.440
	Powder factor (P.F.) (kg/m^3)	20.334
Production holes	Volume blasted (m^3)	6.185
	Amount of explosive (kg)	27.120
	Powder factor (P.F.) (kg/m^3)	4.385
Contour (floor) holes	Volume blasted (m^3)	3.457
	Amount of explosive (kg)	14.892
	Powder factor (P.F.) (kg/m^3)	4.308
Contour (walls) holes	Volume blasted (m^3)	3.034
	Amount of explosive (kg)	0.960
	Powder factor (P.F.) (kg/m^3)	0.316
Contour (roof) holes	Volume blasted (m^3)	3.844
	Amount of explosive (kg)	1.200
	Powder factor (P.F.) (kg/m^3)	0.312
Volume blasted by contour holes (walls + roof) (m^3)		6.879
Amount of explosive for the contour holes (walls + roof) (kg)		2.160
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.314
Volume blasted by contour holes (floor + walls + roof) (m^3)		10.336
Amount of explosive for the contour holes (floor + walls + roof) (kg)		17.052
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.650
Total blasted volume (m^3)		17.182
Volume to be removed (m^3)		24.055
Total amount of explosive (kg)		57.612
Overall powder Factor (kg/m^3)		3.353
Holes number (-)		33
Holes density (holes/m^2)		5.76
Total drilled length (m)		99
Specific drilling (m/m^3)		5.762
Total number of detonators (-)		51
Detonators specific consumption (D.C.) (detonators/m^3)		2.968

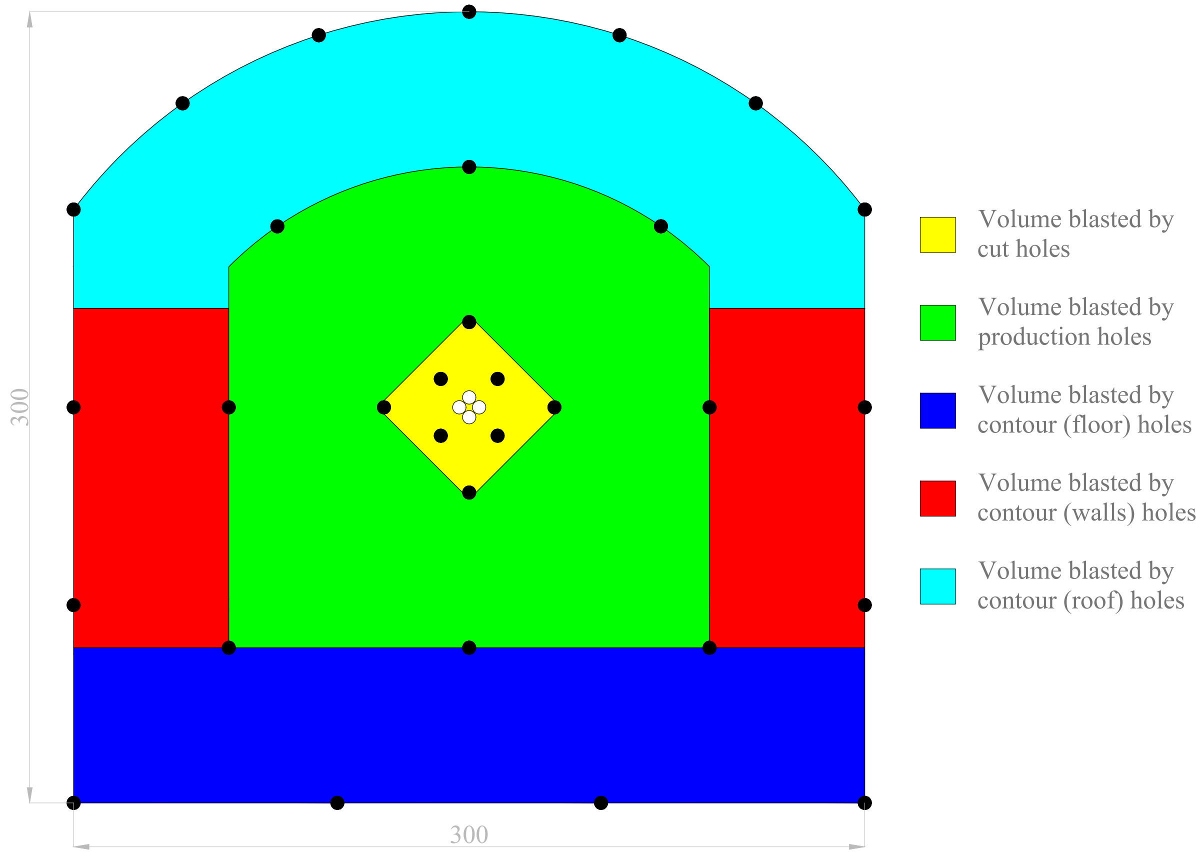

Figure 2.146 – Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 3.0 x 3.0 m section

Table 2.35 - Summary of the main data relating to the excavation section of 3.0 x 3.0 m

Cross section (m^2)		8.27
Design pull (m)		3.0
Actual pull (m)		2.77
Hole diameter (m)		50.8
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.661
	Amount of explosive (kg)	13.440
	Powder factor (P.F.) (kg/m^3)	20.334
Production holes	Volume blasted (m^3)	8.573
	Amount of explosive (kg)	27.120
	Powder factor (P.F.) (kg/m^3)	14.892
Contour (floor) holes	Volume blasted (m^3)	5.277
	Amount of explosive (kg)	14.892
	Powder factor (P.F.) (kg/m^3)	2.822
Contour (walls) holes	Volume blasted (m^3)	4.521
	Amount of explosive (kg)	0.960
	Powder factor (P.F.) (kg/m^3)	0.212
Contour (roof) holes	Volume blasted (m^3)	5.772
	Amount of explosive (kg)	1.680
	Powder factor (P.F.) (kg/m^3)	0.291
Volume blasted by contour holes (walls + roof) (m^3)		10.294
Amount of explosive for the contour holes (walls + roof) (kg)		2.640
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.256
Volume blasted by contour holes (floor + walls + roof) (m^3)		15.571
Amount of explosive for the contour holes (floor + walls + roof) (kg)		17.523
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.126
Total blasted volume (m^3)		24.805
Volume to be removed (m^3)		34.727
Total amount of explosive (kg)		58.092
Overall powder Factor (kg/m^3)		2.342
Holes number (-)		35
Holes density ($holes/m^2$)		4.23
Total drilled length (m)		105
Specific drilling (m/m^3)		4.233
Total number of detonators (-)		55
Detonators specific consumption (D.C.) ($detonators/m^3$)		2.217

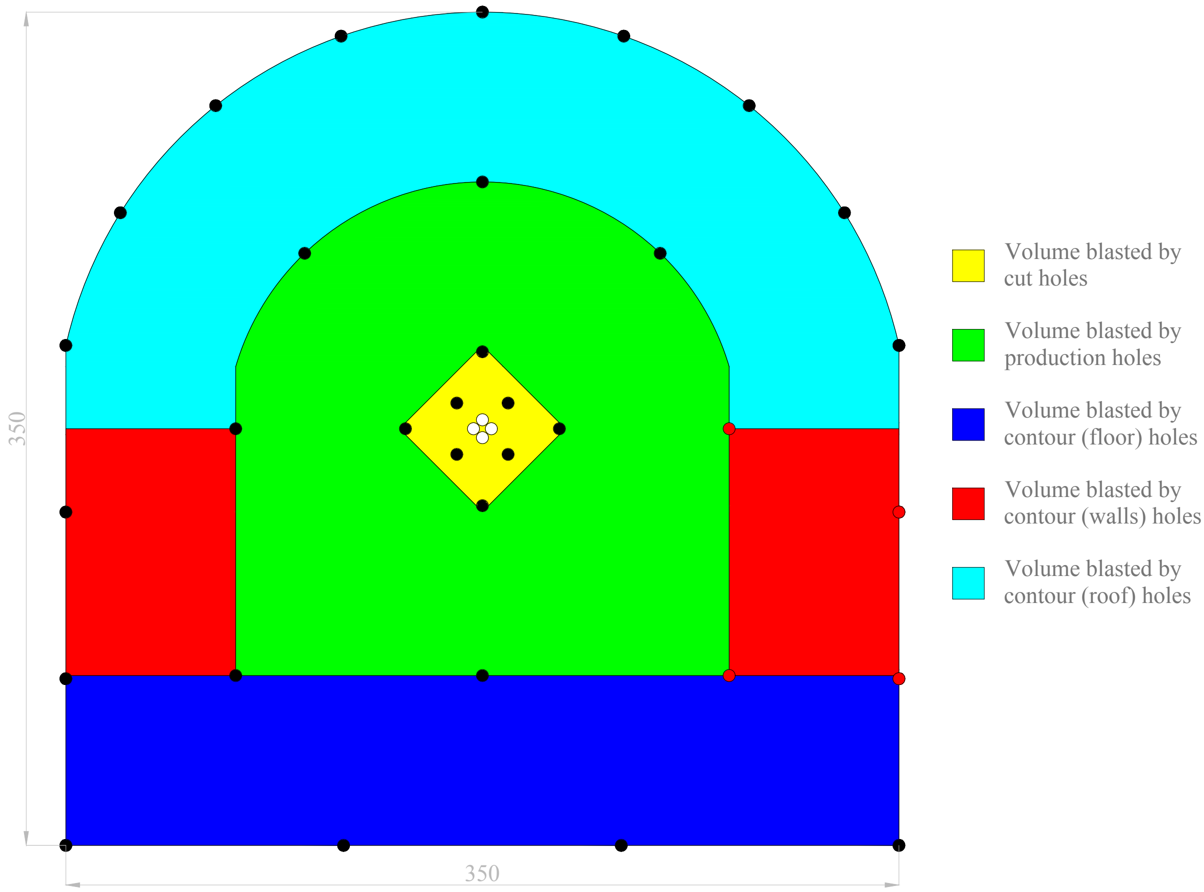

Figure 2.147 – Subdivision of the volumes of competence of each functional group of blast-holes in the excavation section for the 3.5 x 3.5 m section

Table 2.36 - Summary of the main data relating to the excavation section of 3.5 x 3.5 m

Cross section (m^2)		10.95
Design pull (m)		3.0
Actual pull (m)		2.77
Hole diameter (m)		50.8
Look - out angle (°)		5
Cut holes	Volume blasted (m^3)	0.661
	Amount of explosive (kg)	13.440
	Powder factor (P.F.) (kg/m^3)	20.334
Production holes	Volume blasted (m^3)	10.686
	Amount of explosive (kg)	27.120
	Powder factor (P.F.) (kg/m^3)	2.495
Contour (floor) holes	Volume blasted (m^3)	7.467
	Amount of explosive (kg)	14.892
	Powder factor (P.F.) (kg/m^3)	1.994
Contour (walls) holes	Volume blasted (m^3)	4.424
	Amount of explosive (kg)	0.960
	Powder factor (P.F.) (kg/m^3)	0.217
Contour (roof) holes	Volume blasted (m^3)	9.420
	Amount of explosive (kg)	2.160
	Powder factor (P.F.) (kg/m^3)	0.229
Volume blasted by contour holes (walls + roof) (m^3)		13.844
Amount of explosive for the contour holes (walls + roof) (kg)		3.120
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.225
Volume blasted by contour holes (floor + walls + roof) (m^3)		21.331
Amount of explosive for the contour holes (floor + walls + roof) (kg)		18.012
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		0.845
Total blasted volume (m^3)		32.839
Volume to be removed (m^3)		45.975
Total amount of explosive (kg)		58.572
Overall powder Factor (kg/m^3)		1.784
Holes number (-)		37
Holes density (holes/m^2)		3.38
Total drilled length (m)		111
Specific drilling (m/m^3)		3.380
Total number of detonators (-)		57
Detonators specific consumption (D.C.) (detonators/m^3)		1.736

Table 2.37 shows, the blasted volumes according to the detonation sequence. Figure 2.148, Figure 2.149, Figure 2.150, Figure 2.151 depict the progression of the detonation. Data refer to the 3.0 x 3.0 m cross-section.

Table 2.37 - Volumes blasted at every instant of detonation

Detonation sequence (-)	Time (ms)	Volume of rock blasted at each step (m^3)	Overall volume of rock blasted (m^3)	Charge per delay (C.P.D.) (kg)
1	1000	0.000	0.000	0.000
2	1025	0.024	0.024	1.680
3	1050	0.051	0.075	1.680
4	1075	0.047	0.122	1.680
5	1100	0.068	0.190	1.680
6	1125	0.090	0.280	1.680
7	1150	0.144	0.424	1.680
8	1175	0.139	0.563	1.680
9	1200	0.202	0.765	1.680
10	1225	0.653	1.418	3.390
11	1250	2.642	4.060	6.780
12	1275	4.733	8.793	14.226
13	1300	1.776	10.569	6.780
14	1325	5.416	15.985	4.350
15	1350	5.500	21.485	8.166
16	1375	3.524	25.009	0.960

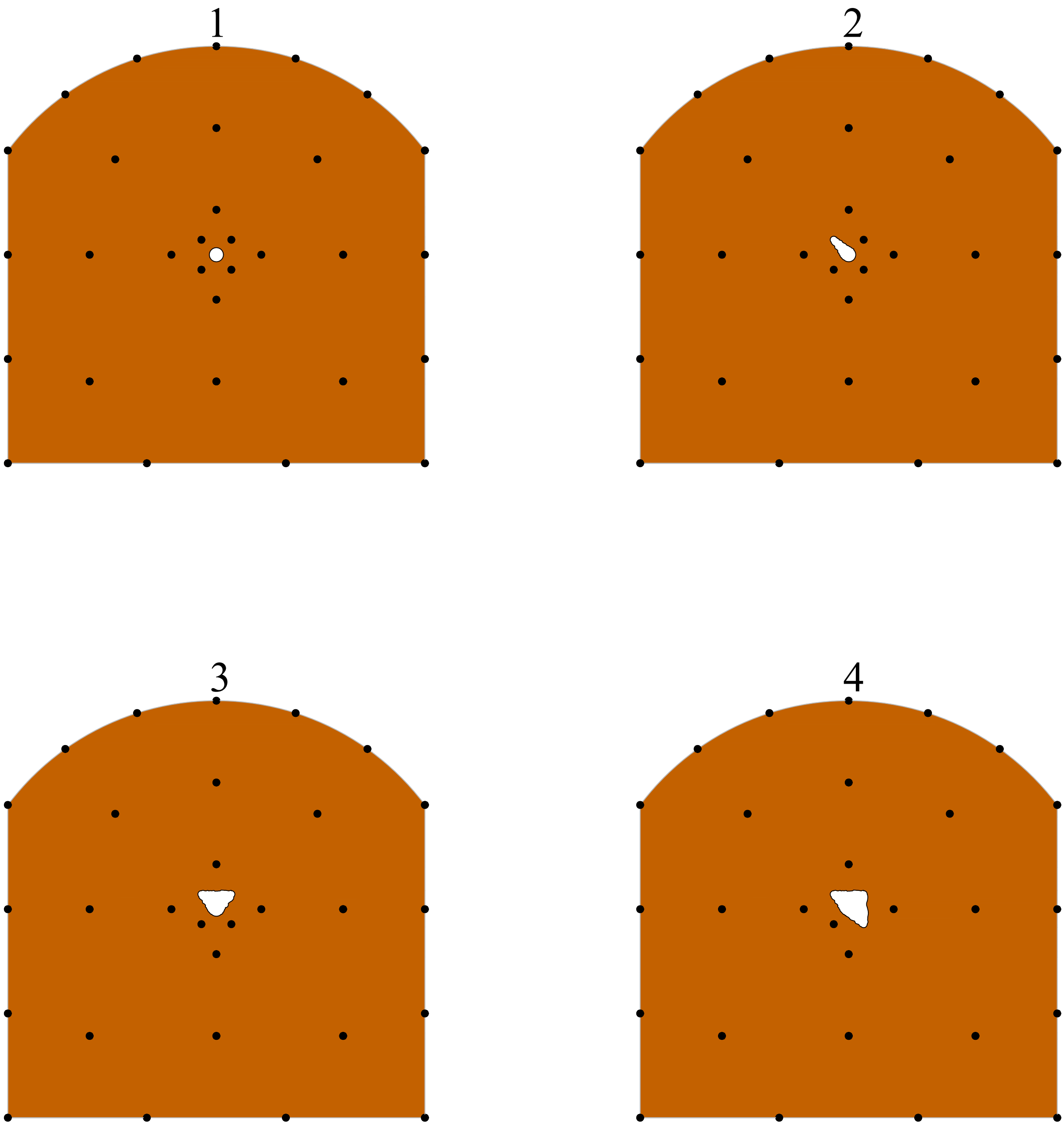

Figure 2.148 – Graphical representation of the detonation instants 1 - 4

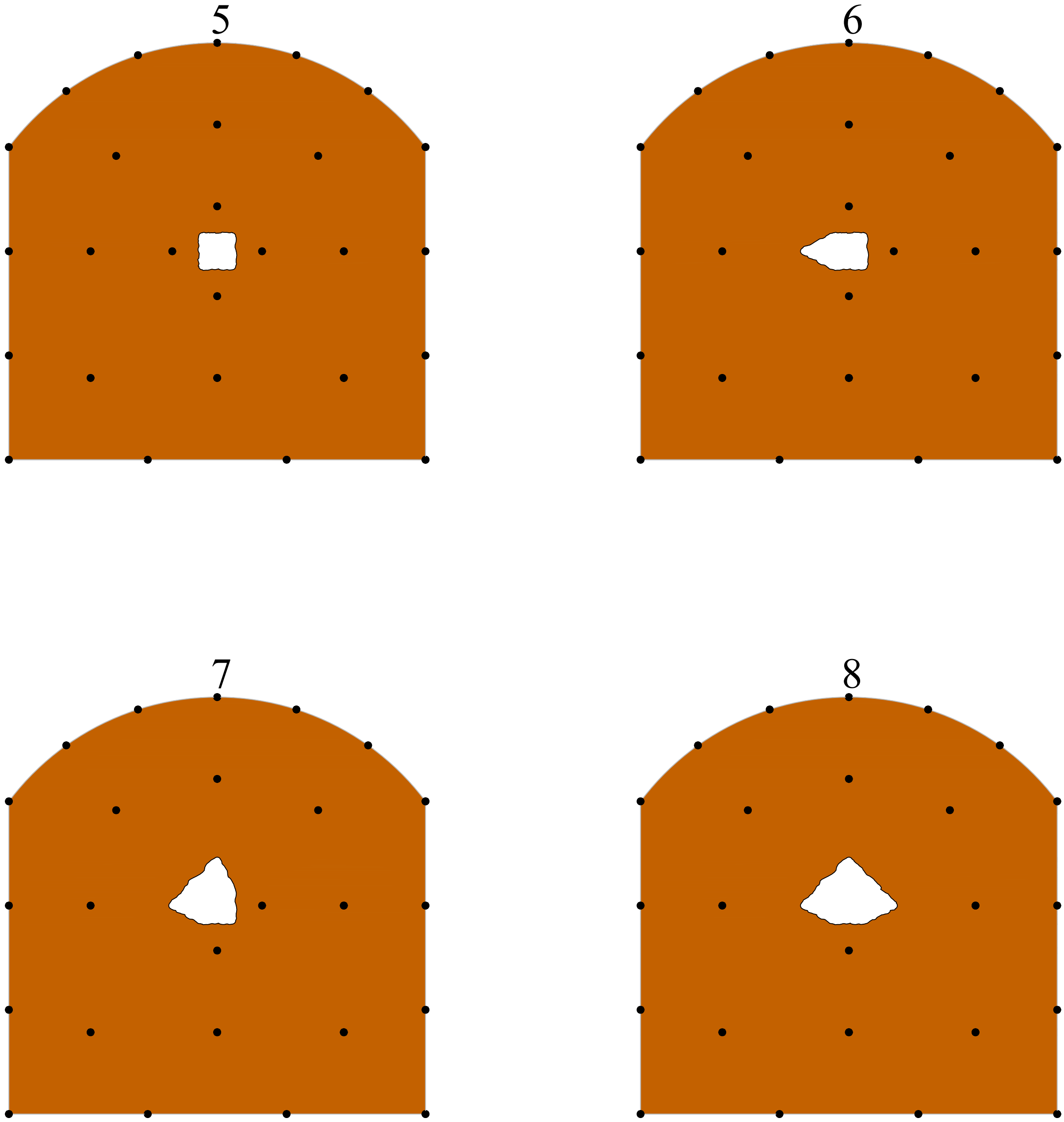

Figure 2.149 - Graphical representation of the detonation instants 5 - 8

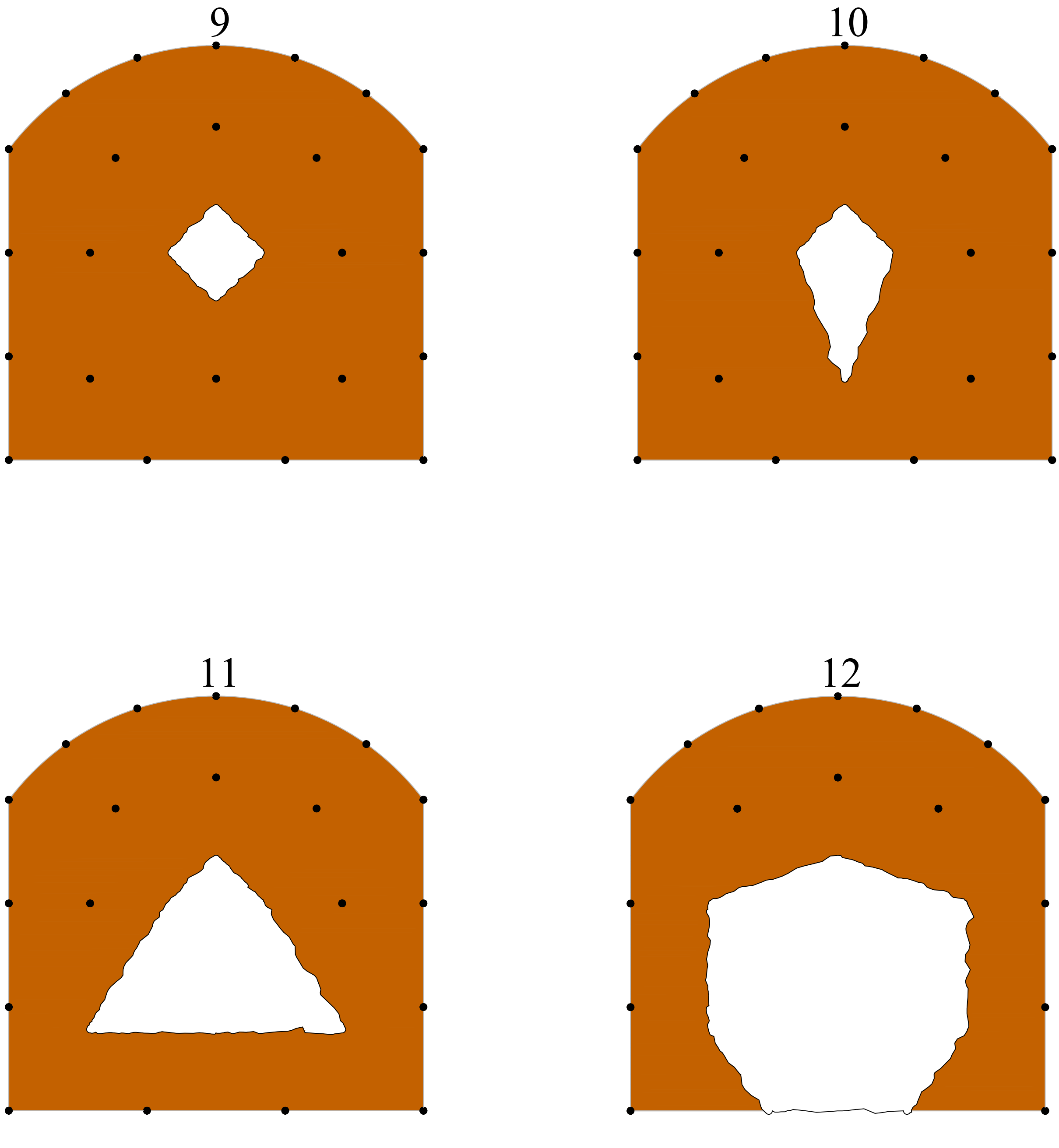

Figure 2.150 - Graphical representation of the detonation instants 9 - 12

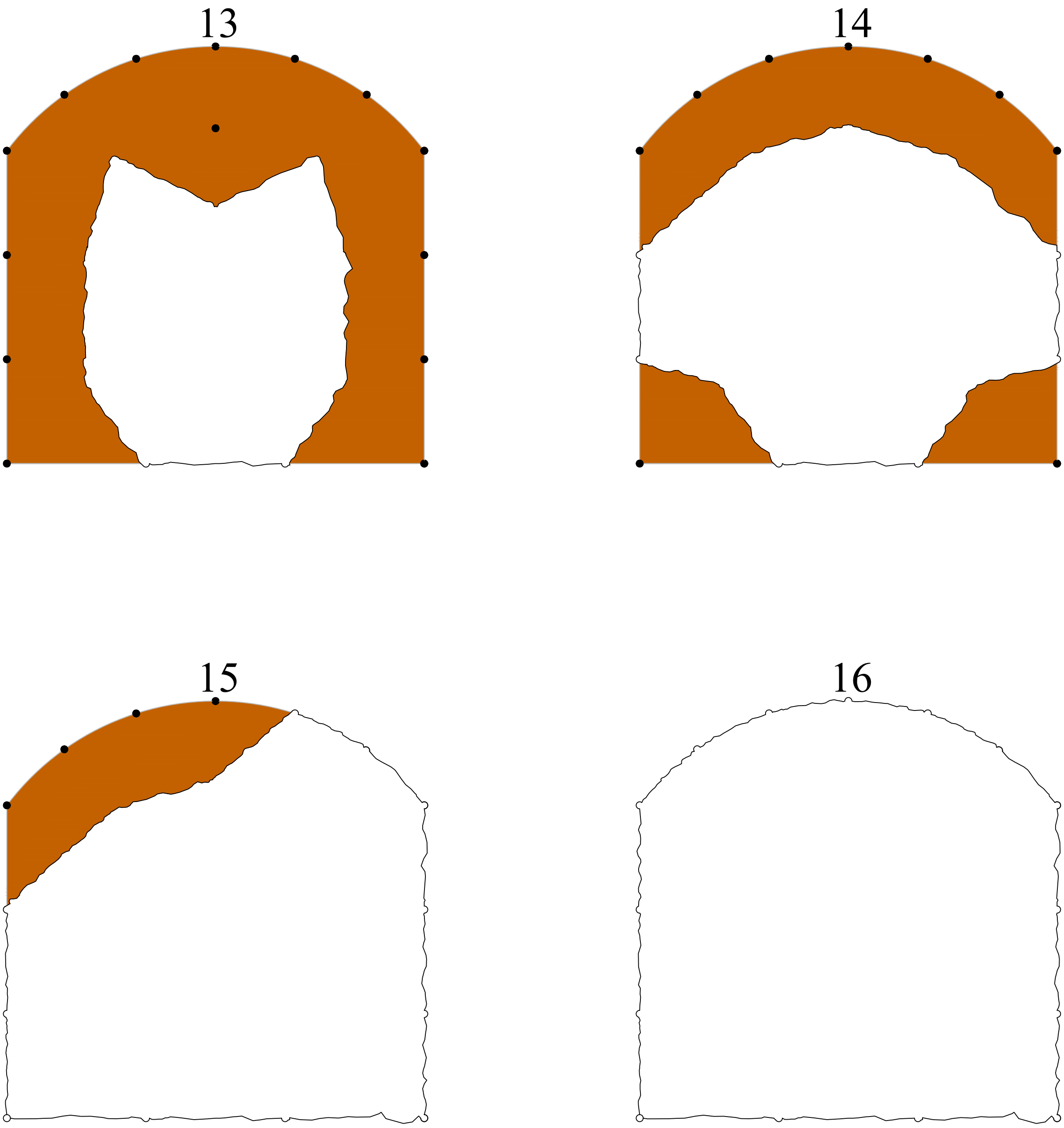

Figure 2.151 - Graphical representation of the detonation instants 13 - 16

CONCLUSIONS

The study that was presented in this thesis is aimed, as already mentioned, at the development of the project for the improvement of the drill and blast excavation technique at the artisanal gold mine "*La Palmera*" located in the village of Chancón, near Rancagua, in Chile. The procedures and technologies currently used in the mining process have been observed and critically analysed from a scientific point of view. During the numerous surveys carried out at the mine, many data were collected and then, subsequently processed and studied. A lot of drawbacks, both from a technical and organizational point of view, have been detected. These make the extraction process inefficient and unproductive, as well as dangerous. The problems and possible interventions to be implemented in order to increase both work safety and production have been identified. These interventions have also been designed taking into account the social, cultural, economic and logistic aspects present in the geographical context where the mine is located.

The interventions have been divided into three distinct and progressive phases, in order to avoid too radical and abrupt changes that would not be easily put into practice for logistical and cultural reasons. The first phase consists in the introduction of a new work process and a different blast triggering system (nonel system). The second involves the introduction of a different type of explosive for the contour blast-holes and a new geometry of the excavation cross-section. The third phase involves the introduction of a Jumbo for drilling, increasing the pull and changing the explosives used.

Since it has not been possible to await the implementation of the phases and observe the development of the improvement process, the results obtained have been compared with those found in the scientific literature. In particular, the reference is: "*Atlas of Blasting Rounds of Tunnel Driving*". Mancini, R., Gaj, F., Cardu, M. (1996).

A further development of the research work shown in this thesis will consist in practically observing the implementation of the proposed project and eventually highlighting problems and proposing further corrective measures during the development of the project.

The graphs shown in Figure C.152, built with the data obtained from various cases found in literature, show how the great majority of small-section tunnels (less than 10 m^2) have drilling diameters smaller or equal to 35 mm.

In the artisanal mine, a diameter of 38 mm is used. The same is used in the development of phase 1 and 2 for the improvement of the project. In phase 3, a larger diameter is used, equal to 50.8 mm. As for the intermediate size sections (area between 10 m^2 and 40 m^2), the most frequently used drilling diameter is still less than 35 mm. In the improvement project, in the 3.5 x 3.5 m sections, the drilling diameter of 38 mm was used in phases 1 and 2, whereas a 50.8 mm diameter was used in phase 3.

The choice of the drilling diameter was partly dictated by the drilling technique and partly by the type of explosive used. The use of servo-supported drilling machines does not allow excessively large diameters and very long drill-holes. Moreover, the use of bulk ANFO, an explosive having a rather high critical diameter, does not allow drilling with too small diameters. If a drilling diameter less than 35 mm was used (as in most of the cases analyzed in literature), the ANFO would not have detonated correctly and cartridged nitroglycerin-based explosives should have been used. It should also be noted that the choice of using the same drilling diameter already used in the mine allows not to purchase additional bits.

As for phase 3, a greater diameter has been chosen, because drilling is performed by means of a boomer. Using a larger diameter, it can be obtained a wider drilling pattern with fewer holes, optimizing drilling times and therefore the productivity. Since the drilling length is greater (double), using a larger diameter can avoid unpleasant deviations of the hole that could occur with smaller diameters, especially with regard to very close cut holes.

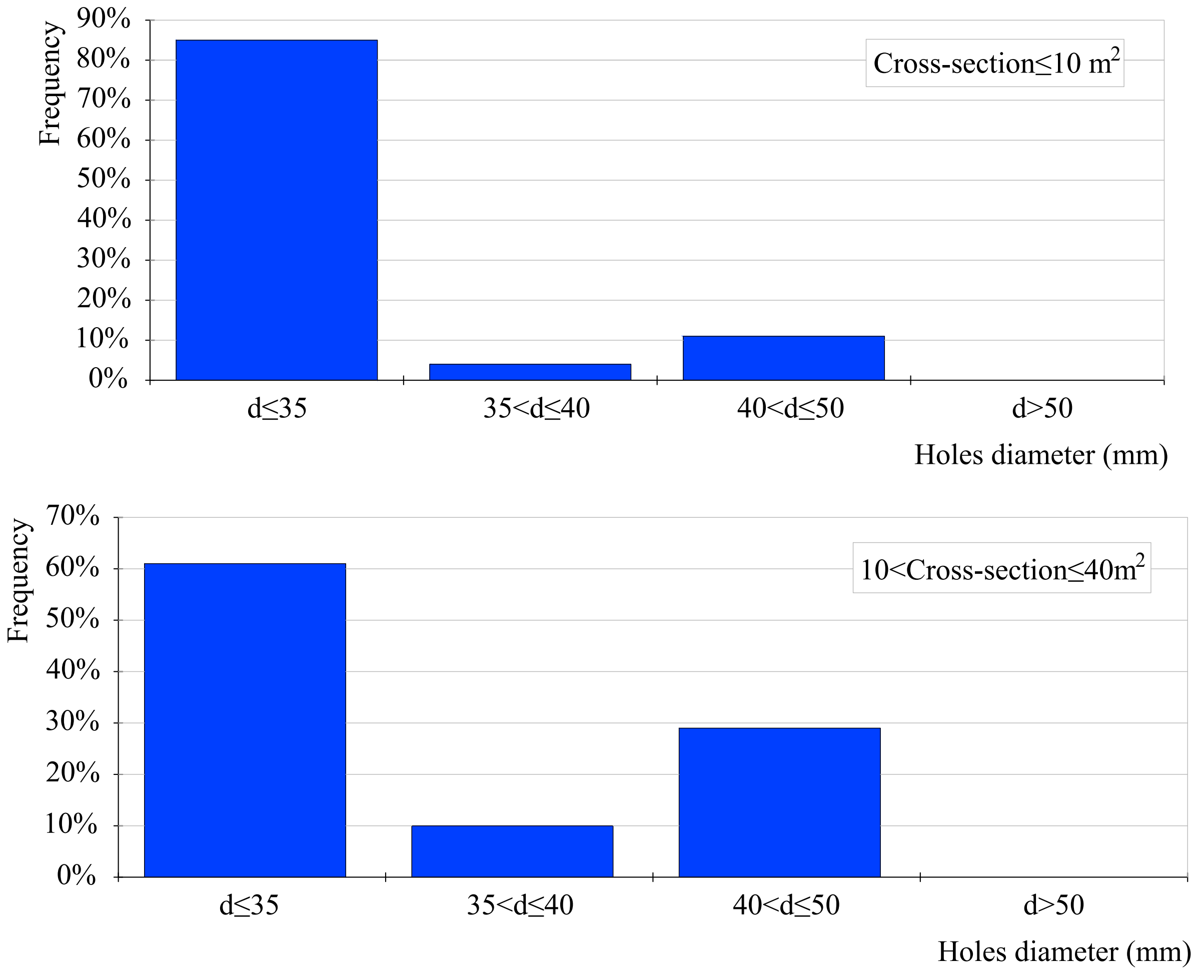

Figure C.152 - Frequency of the drilling diameters used in the tunnel excavation of small and medium cross-section

The choice of the type of cut was made taking into consideration many aspects: the size of the excavation section, the type of drilling machine used, the experience and the skill of the operators, etc.

The choice of using parallel holes cuts is optimal in small section tunnels, where the positioning of the drilling machine for the execution of inclined holes would be a problem. Furthermore, by choosing this cut type, a simplified drilling scheme is obtained. Observing data from literature shown in the graph in figure Figure C.153 it can be seen that for small excavation sections (smaller than 10 m^2 and between 10 e 20 m^2), the most popular option is that of the parallel holes cut.

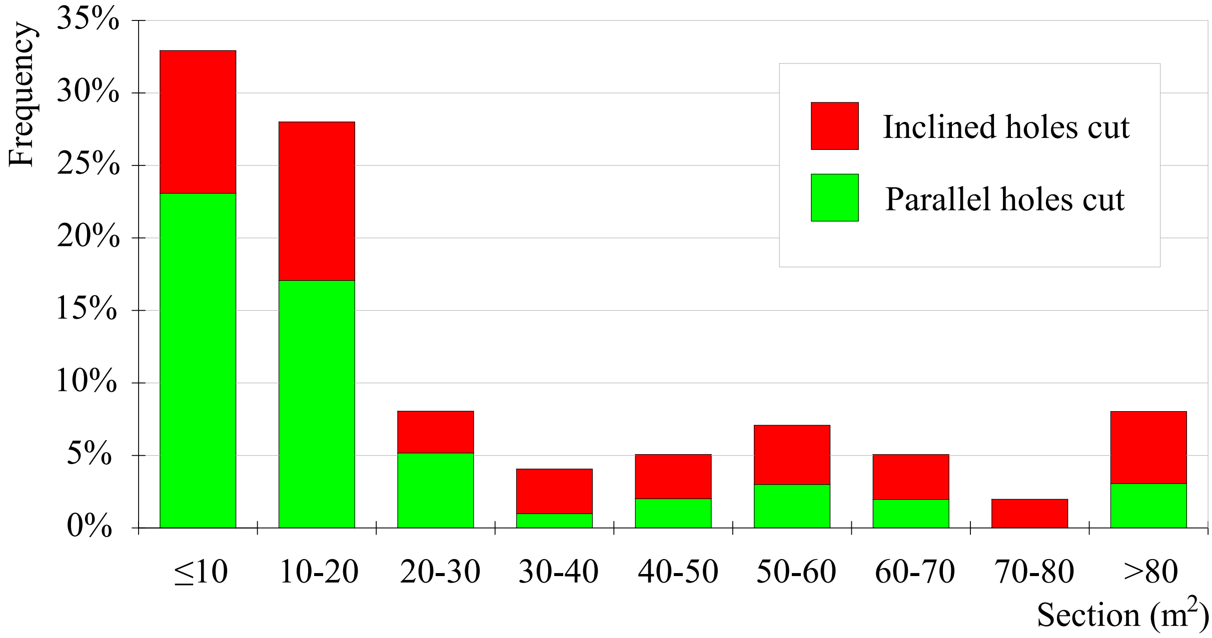

Figure C.153 - Frequency of use of the cut with parallel and inclined holes

The graph in Figure C.154 shows the frequency of blast efficiency for parallel and inclined holes cut blasts. The estimated efficiency for the blast studied in the artisanal mine, equal to 71%, is so low that it does not even appear in the ranges considered. It should however be noted that efficiencies below 90% are very infrequent and probably linked to exceptions and complex cases. This clarifies the importance of introducing urgently the necessary improvements.

The estimated efficiency for phase 3 of the project proposed in this thesis, equal to 92.2%, is perfectly in line with the literature data for parallel holes cuts.

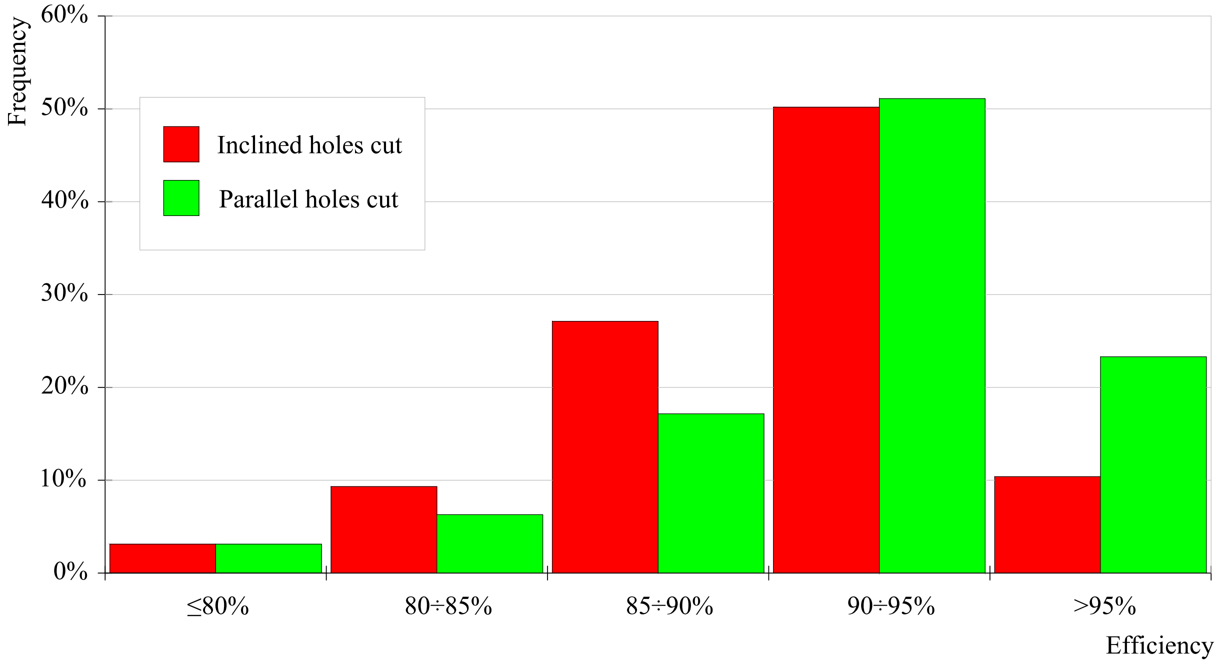

Figure C.154 - Frequency of blast efficiencies for cut with parallel holes and cut with inclined holes

At this point it is possible to analyze the data obtained from the design calculation and compare them with those obtained from the inspections carried out in the artisanal mine.

With regard to the data related to drilling, the values of the Specific Drilling (S.D.) are analyzed and compared. They are shown in the Table C.38.

The value of S.D. detected in the artisanal mine is much smaller than the one designed in phases 1, 2 and 3 for sections of similar size (2.5 x 2.5 m). This is due to the smaller number of holes made in lack of a drilling scheme.

Observing the graph shown in Figure C.155, it can be understood how, for each design phase, increasing the excavation section, S.D. decreases because the blasted volume increases, while the number of holes in the drilling scheme does not increase substantially.

The highest values of S.D. were obtained for phase 2. In this phase, in fact, using the same drilling diameter of phase 1 (38.1 mm), another design system was employed for the contour holes. Using the smooth blasting technique (following the empirical table proposed by E. I. du Pont de Nemours & Co., 1978) instead of the empirical method proposed by Olofsson (1991) used for the phase 1, smaller values of spacing between the contour holes of the walls and roof were obtained. With respect to phase 1, with the same blasted volume, the S.D. is greater.

The S.D. calculated for phase 3 is lower than that of phase 2 because holes with a larger diameter (50.8 mm) have been used. The drilling pattern is therefore wider and fewer holes are then made. The specific drilling obtained for phase 1 is roughly equal to that obtained for phase 3.

Table C.38 - Data relating to Specific Drilling

	Specific Drilling (S.D.) (m/m^3)
Artisanal Mine	2.481
Phase 1; 2.5 x 2.5 m	5.791
Phase 1; 3.0 x 3.0 m	4.451
Phase 1; 3.5 x 3.5 m	3.273
Phase 2; 2.5 x 2.5 m	7.349
Phase 2; 3.0 x 3.0 m	5.219
Phase 2; 3.5 x 3.5 m	3.969
Phase 3; 2.5 x 2.5 m	5.762
Phase 3; 3.0 x 3.0 m	4.233
Phase 3; 3.5 x 3.5 m	3.380

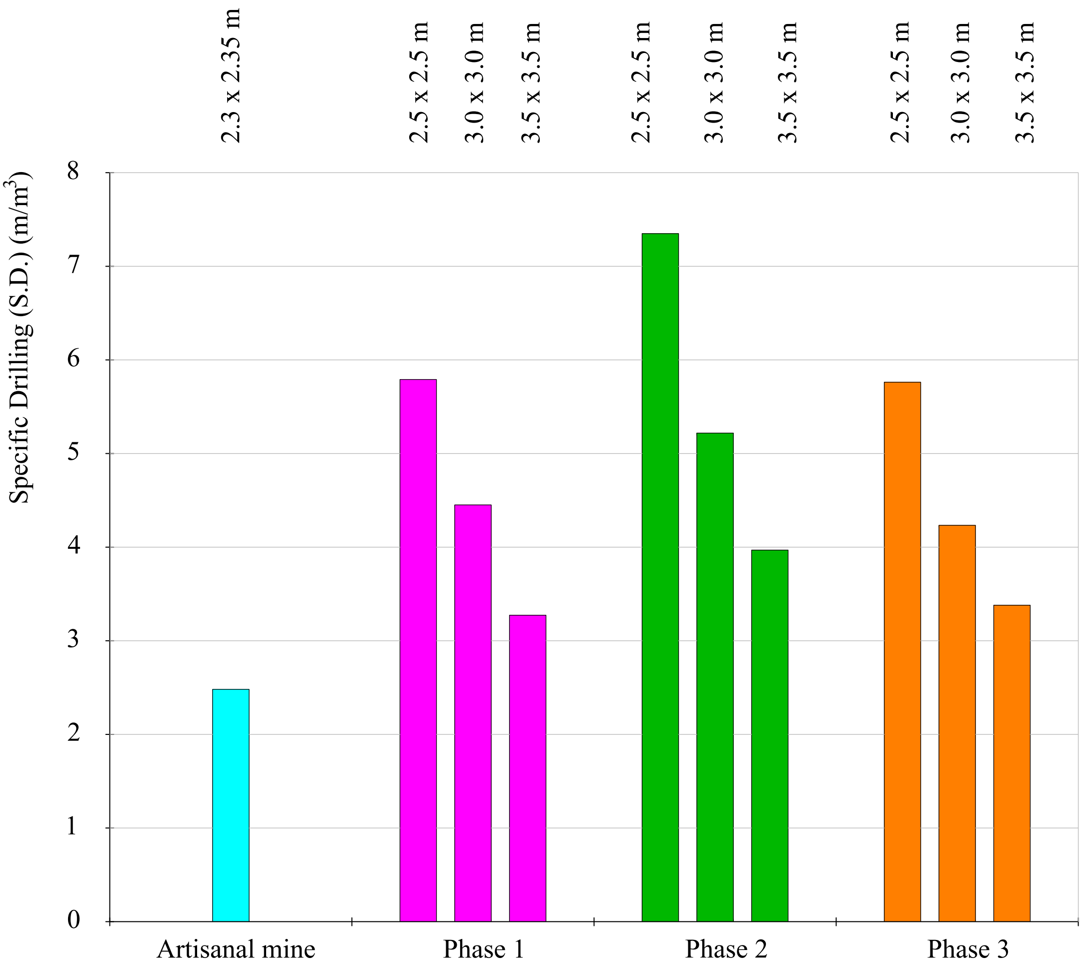

Figure C.155 - Comparison of data relating to specific drilling

As for the Powder Factor (P.F.), the value calculated from the data obtained in the artisanal mine is much lower than that determined in the design phases 1, 2 and 3 for similar cross sections. This is due to the lower number of holes and amount of explosives used. For this reason, the result of the blast is so unsatisfactory. The data related to the Powder Factor are shown in Table C.39.

Looking at the graph shown in Figure C.156, it can be understood how for each design phase, increasing the excavation section, P.F. decreases because the blasted volume increases while the total amount of explosive used in the blast does not increase substantially.

Phase 2 presents a P.F. lower than phase 1 because the detonating cord was used in the contour holes (walls and roof) instead of the explosive used for other blast-holes.

Phase 3 is that with the lower P.F. because the design pull has been doubled, the drilling diameter has been increased and a more disruptive explosive is used, suitable for the excavation conditions encountered in the mine.

Table C.39 - Data relating to Powder Factor (P.F.)

	Powder Factor (P.F.) (kg/m3)
Artisanal Mine	1.528
Phase 1; 2.5 x 2.5 m	3.904
Phase 1; 3.0 x 3.0 m	3.020
Phase 1; 3.5 x 3.5 m	2.221
Phase 2; 2.5 x 2.5 m	3.547
Phase 2; 3.0 x 3.0 m	2.587
Phase 2; 3.5 x 3.5 m	1.878
Phase 3; 2.5 x 2.5 m	3.353
Phase 3; 3.0 x 3.0 m	2.342
Phase 3; 3.5 x 3.5 m	1.784

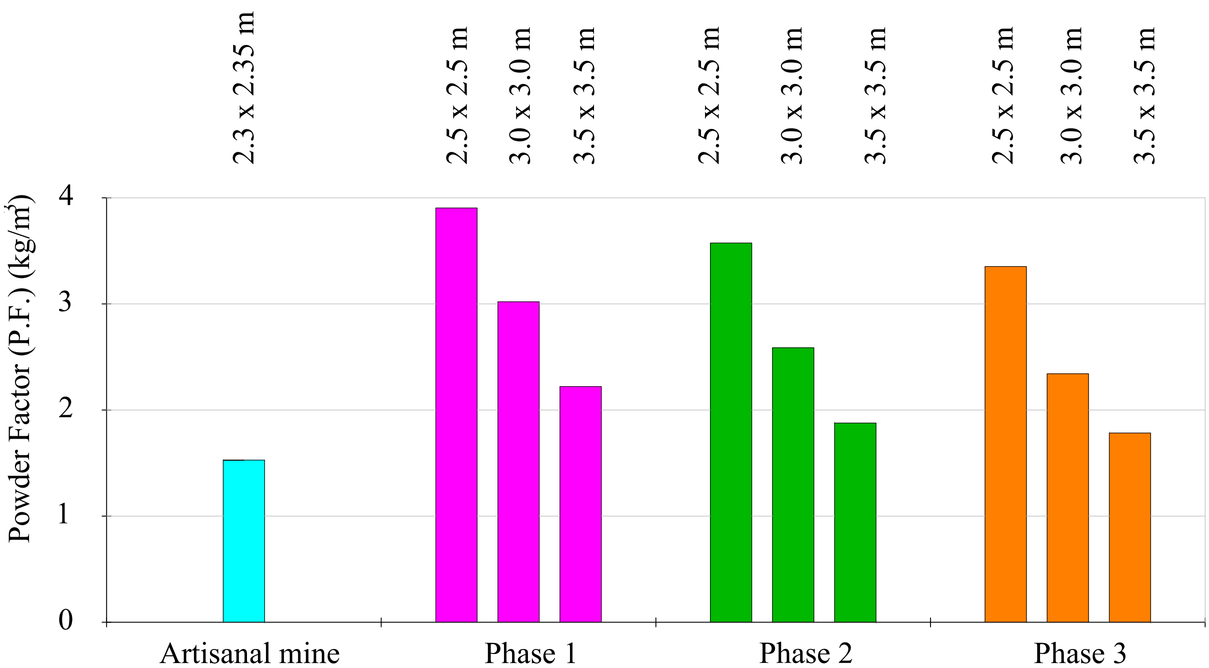

Figure C.156 - Comparison of data relating to Powder Factor

As for the specific consumption (D.C.) of detonators, the value calculated with the data collected in the mine is limited because there are few holes and for the triggering of the blast with ordinary detonators and safety fuse. In phases 1, 2 and 3 of the project, the number of detonators is greater as the nonel technique is proposed, that involves using both in-the-hole detonators for the booster triggering and connection units (surface connectors detonators). The data related to D.C. are given in Table C.40.

For each phase of the improvement project, it can be observed that D.C. decreases by increasing the area of the excavation section. This is because the blasted volume increases while the number of holes remains almost unchanged. This can be observed in the graph shown in Figure C.157.

Phase 2 is the one that presents the highest values of D.C. because it is the phase with the highest number of holes in the excavation face. Phase 3 is characterized by a lower D.C. because of a smaller number of holes and a higher blasted volume. It can be observed how the values of D.C. reached in this phase are similar to those calculated for the artisanal mine.

Table C.40 - Data relating to Detonators specific Consumption (D.C.)

	Detonators specific Consumption (detonators/m^3)
Artisanal Mine	2.370
Phase 1; 2.5 x 2.5 m	6.330
Phase 1; 3.0 x 3.0 m	4.670
Phase 1; 3.5 x 3.5 m	3.440
Phase 2; 2.5 x 2.5 m	7.289
Phase 2; 3.0 x 3.0 m	5.385
Phase 2; 3.5 x 3.5 m	4.029
Phase 3; 2.5 x 2.5 m	2.968
Phase 3; 3.0 x 3.0 m	2.217
Phase 3; 3.5 x 3.5 m	1.736

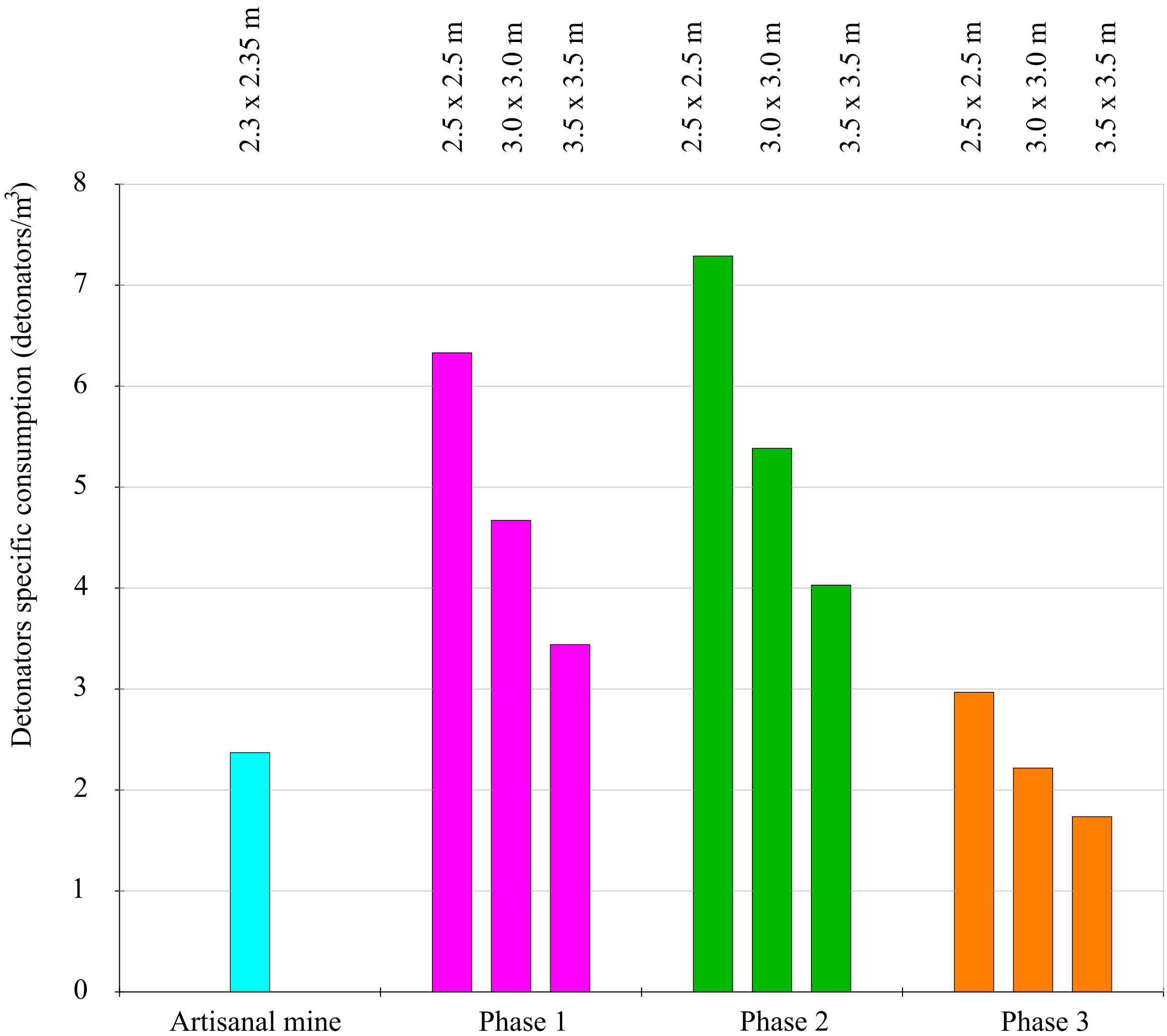

Figure C.157 - Comparison of data relating to Detonators Specific Consumption

In literature it is possible to find several graphs of an empirical nature that allow to observe the correspondence of the calculations obtained with the results actually observed and analyzed in real cases.

The empirical graph shown in Figure C.158, relates to the area of the excavation section with the number of holes required. The blue and red curves were obtained by interpolating the data collected in the study of multiple real cases. The blue curve is related to bad blastability conditions (for example very hard and compact rocks, presence of particular geological structures, etc.) and shows the need of a greater number of holes compared to the good blastability conditions represented by the red curve. Obviously, by increasing the size of the tunnel, the number of holes required for the blast increases.

The point representing the values obtained in the artisanal mine is positioned far below the red curve. This shows that the number of holes drilled on the excavation face is much lower than that required to obtain a good blast, being very far from the values found in the practical cases analyzed. This confirms the unsatisfactory results obtained by the blast carried out.

As for the points related to the various dimensions of the excavation sections for phases 1, 2 and 3, all the points are positioned close to the curve related to poor blast conditions, a situation that can actually be found in the mine, given the geological conditions and the lithology of the rock mass.

The data used for entering the points in the graph are shown in Table C.41.

Table C.41 - Data related to section area and number of holes

	Section (m^2)	Holes number (-)
Artisanal Mine	6.85	17
Phase 1; 2.5 x 2.5 m	6.25	36
Phase 1; 3.0 x 3.0 m	9.00	40
Phase 1; 3.5 x 3.5 m	12.25	40
Phase 2; 2.5 x 2.5 m	5.58	41
Phase 2; 3.0 x 3.0 m	8.05	42
Phase 2; 3.5 x 3.5 m	11.08	44
Phase 3; 2.5 x 2.5 m	5.73	33
Phase 3; 3.0 x 3.0 m	8.27	35
Phase 3; 3.5 x 3.5 m	10.95	37

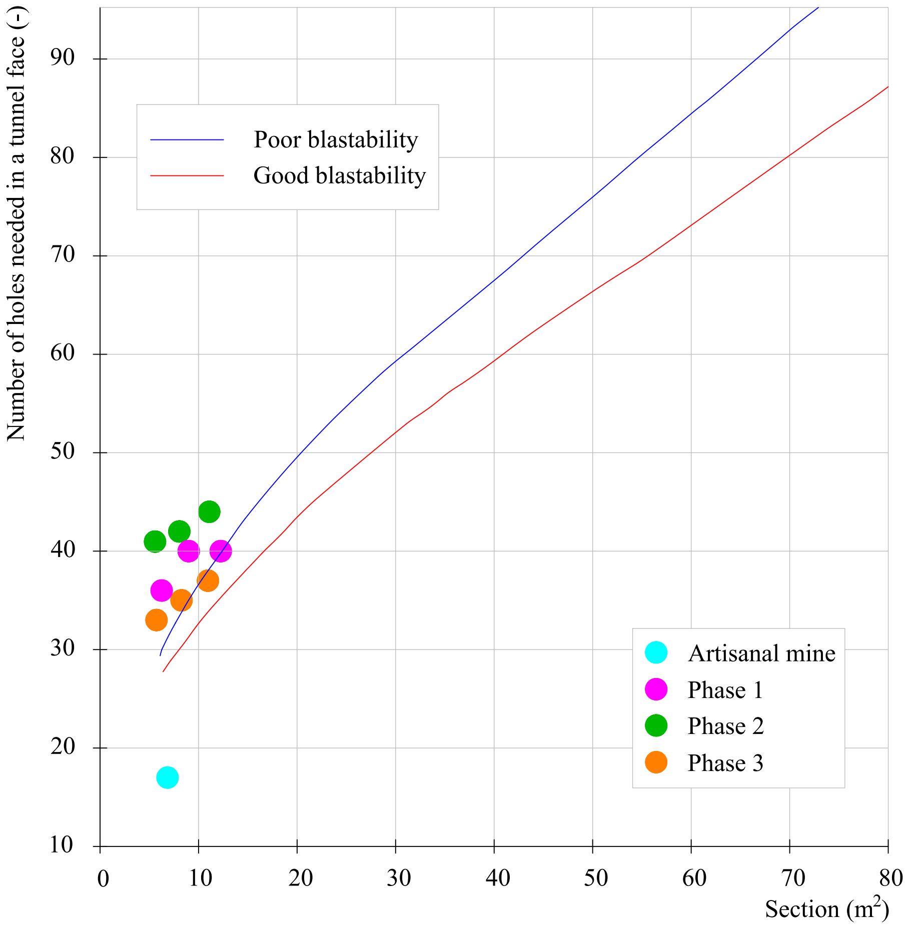

Figure C.158 - Empirical graph that relates section and number of holes needed in a tunnel face

The empirical graph shown in Figure C.159 relates to the area of the excavation section with the Specific Drilling (S.D.). The curve expressing the 38 mm drilling diameter of is blue, that relative to the 51 mm diameter is red. As it can be seen, by increasing the diameter, with the same excavation section, the S.D. decreases. This is because the drilling pattern is wider and therefore the number of holes decreases. By decreasing the excavation section, the specific Drilling increases: the cut is the functional group of blast-holes having the highest S.D. and is preponderant in smaller sections.

The point related to the artisanal mine differs very much from the empirical curves, showing a S.D. much lower than that found in common practice. The points related to steps 1 and 2 are very close to the 38 mm diameter curve, while those related to phase 3 are close to the 51 mm curve. This confirms the nearness of the results obtained with the cases observed in the reality and once again highlights the bad results of the blast performed in the artisanal mine.

The data used for entering the points in the graph are shown in Table C.42.

Table C.42 - Data related to section area and Specific Drilling

	Section (m2)	Specific Drilling (S.D.) (m/m3)
Artisanal Mine	6.85	2.481
Phase 1; 2.5 x 2.5 m	6.25	5.791
Phase 1; 3.0 x 3.0 m	9.00	4.451
Phase 1; 3.5 x 3.5 m	12.25	3.273
Phase 2; 2.5 x 2.5 m	5.58	7.349
Phase 2; 3.0 x 3.0 m	8.05	5.219
Phase 2; 3.5 x 3.5 m	11.08	3.969
Phase 3; 2.5 x 2.5 m	5.73	5.762
Phase 3; 3.0 x 3.0 m	8.27	4.233
Phase 3; 3.5 x 3.5 m	10.95	3.380

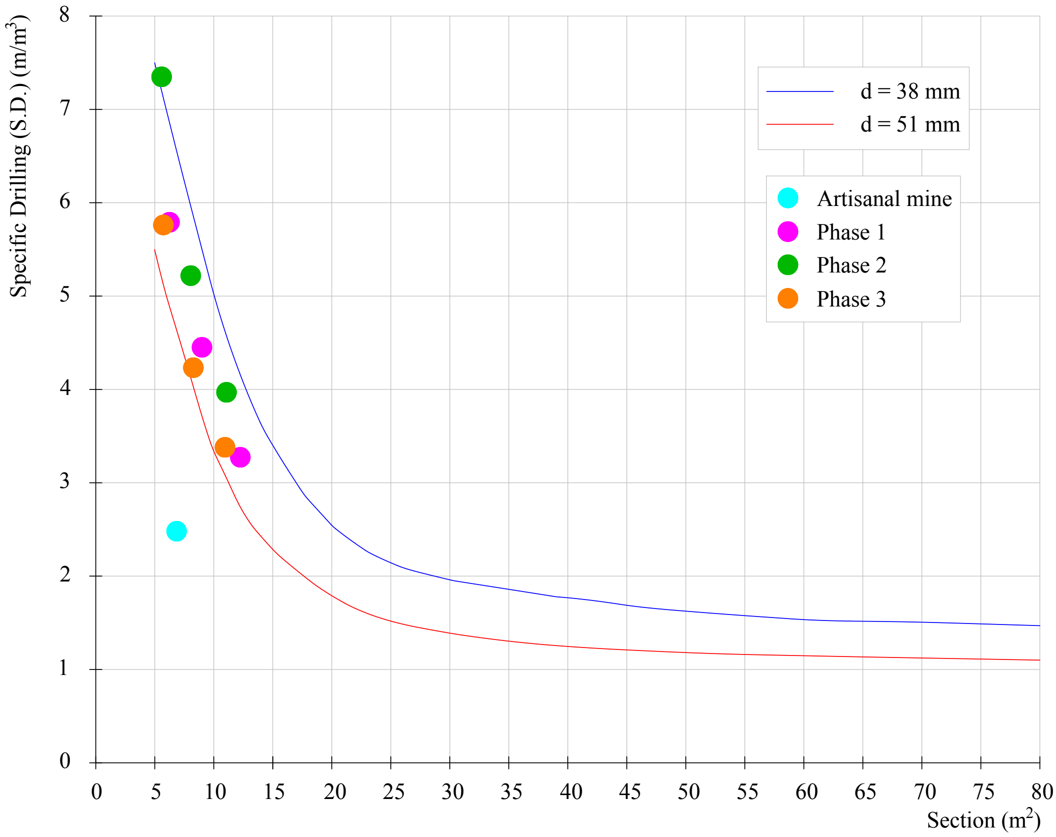

Figure C.159 - Empirical graph that relates section and the Specific Drilling

The empirical graph shown in Figure C.160, expresses the relationship between the excavation section and the Powder Factor (P.F.). The yellow band at the center of the graph represents the values that were most frequently found in the study of real cases. As it can be seen, by decreasing the excavation section, the value of P.F. increases. This is due to the fact that the cut (functional group of blast-holes characterized by high values of P.F.) is preponderant in smaller sections.

The value of P.F. detected in the artisanal mine is very far from the yellow band, indicating a too low value of P.F.. The values obtained for phase 1 fall perfectly within the yellow area. The values for phase 2 and 3 are slightly lower. This is due to the fact that the charge employed for wall and roof contour holes is the detonating cord, and this has reduced the overall P.F.

The data used to build the graph are shown in Table C.43.

Table C.43 - Data related to section area and Powder Factor

	Section (m^2)	Powder Factor (P.F.) (kg/m^3)
Artisanal Mine	6.85	1.528
Phase 1; 2.5 x 2.5 m	6.25	3.904
Phase 1; 3.0 x 3.0 m	9.00	3.020
Phase 1; 3.5 x 3.5 m	12.25	2.221
Phase 2; 2.5 x 2.5 m	5.58	3.547
Phase 2; 3.0 x 3.0 m	8.05	2.587
Phase 2; 3.5 x 3.5 m	11.08	1.878
Phase 3; 2.5 x 2.5 m	5.73	3.353
Phase 3; 3.0 x 3.0 m	8.27	2.342
Phase 3; 3.5 x 3.5 m	10.95	1.784

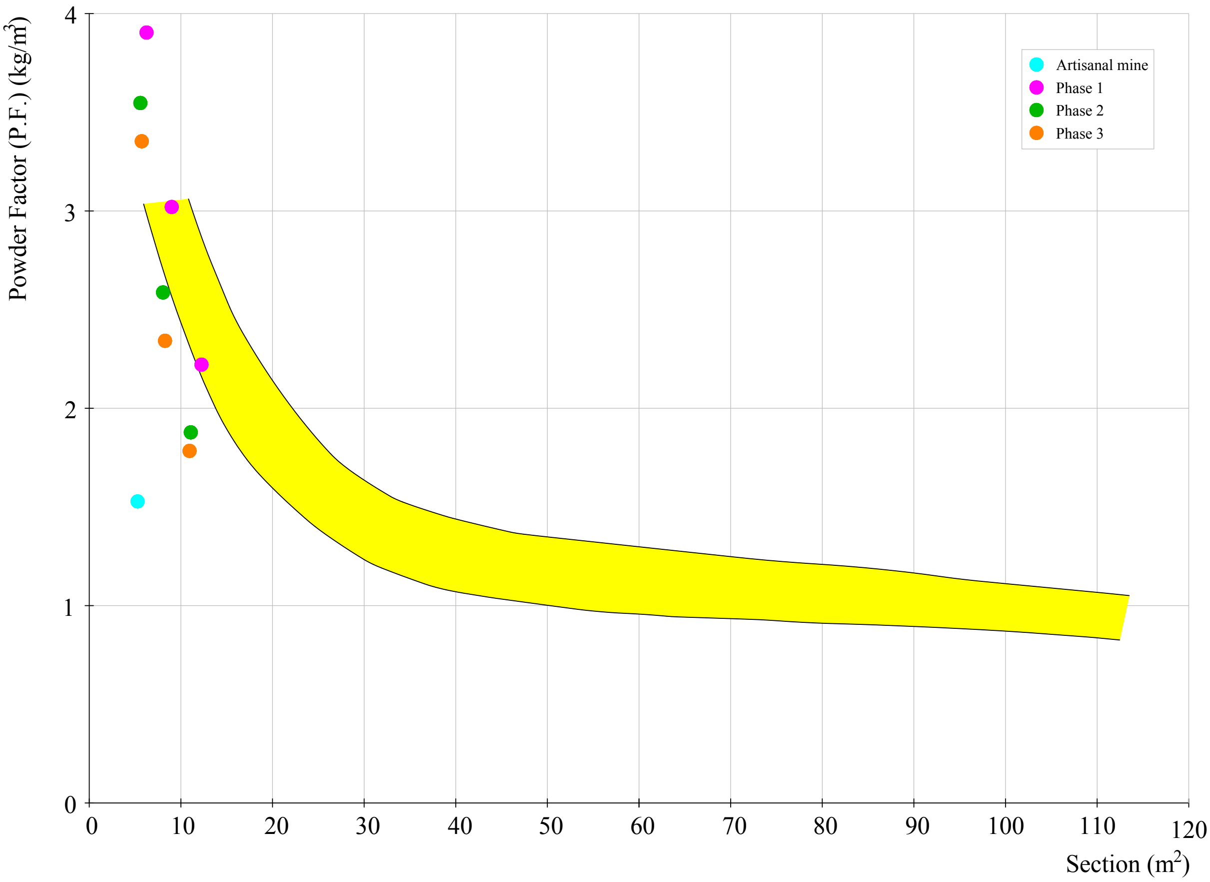

Figure C.160 - Empirical graph that relates section and Powder Factor

The empirical graph shown in Figure C.161 relates the holes density and the Powder Factor (P.F.). The points in the scatter plot represent the data obtained from the study of real cases. The gray points are related to blasts with parallel holes cut, the white points with a black border are related to blasts with inclined holes cut. Generally, with the same density of holes, the P.F. is greater for parallel cut holes. The data used for entering the points in the graph are shown in Table C.44.

Table C.44 - Data related to holes density and Powder Factor

	Holes density (holes/m^2)	Powder Factor (P.F.) (kg/m^3)
Artisanal Mine	2.481	1.528
Phase 1; 2.5 x 2.5 m	5.76	3.904
Phase 1; 3.0 x 3.0 m	4.44	3.020
Phase 1; 3.5 x 3.5 m	3.26	2.221
Phase 2; 2.5 x 2.5 m	7.35	3.547
Phase 2; 3.0 x 3.0 m	5.22	2.587
Phase 2; 3.5 x 3.5 m	3.97	1.878
Phase 3; 2.5 x 2.5 m	5.76	3.353
Phase 3; 3.0 x 3.0 m	9.23	2.342
Phase 3; 3.5 x 3.5 m	3.38	1.784

All the points, even those related to the mine, are not very far from the cloud of points related to the cases found in literature. For the three phases of the improvement project and for the three points related to the various dimensions of the sections, the points were interpolated following a linear interpolation. The comparison of the interpolating lines is shown in Figure C.162.

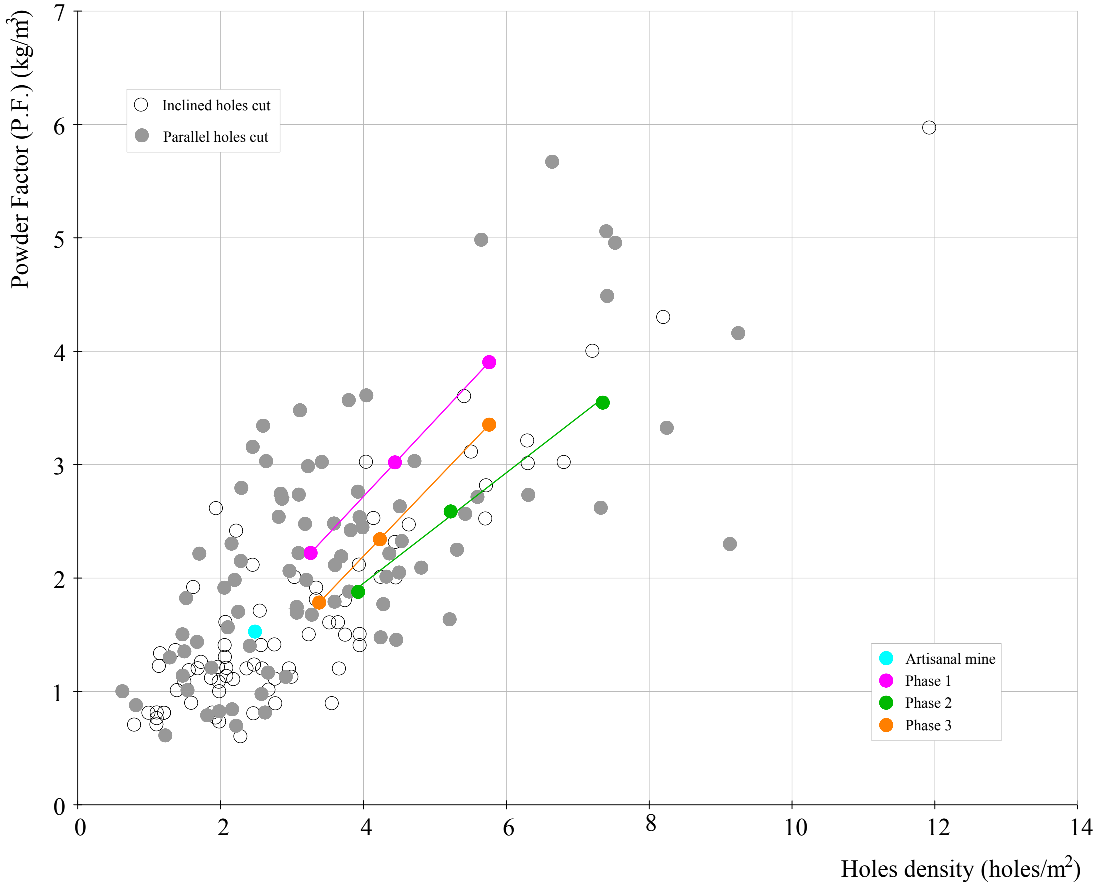

Figure C.161 - Empirical graph that relates holes density and Powder Factor

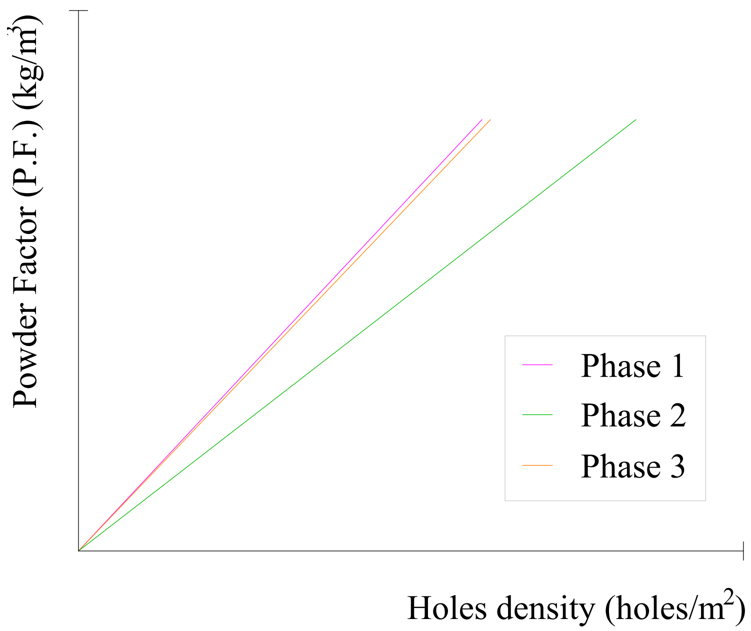

Figure C.162 - Comparison between interpolating lines

The comparison of the slopes can be made taking into account the following relation (C.1):

$$slope = \frac{y}{x} = \frac{P.F.}{holes\ density} = \frac{kg/_{m^3}}{n.holes/_{m^2}} = \frac{kg}{m \cdot n.holes} = \frac{Quantity\ of\ explosive}{Design\ pull\ \cdot n.holes} \quad (C.1)$$

The interpolating line related to phase 1 is more inclined than that of phase 2, because the amount of explosive is greater, and the number of holes is smaller. The interpolating line of the data related to phase 3 is more inclined, because the amount of explosive and the design pull is greater, and the number of holes is smaller.

The inclination of the interpolating lines related to phases 1 and 3 is very similar, because the number of holes is approximately the same, the explosive amount of phase 3 is slightly less than twice that of phase 1, the design pull of phase 1 is the half of that in phase 3.

Another analysis can be done using the empirical formulas of Mancini and Pelizza, (1969). These formulas have been determined by interpolating many data obtained from the study of tunnel excavations and are used to predict the Specific Drilling and the Powder Factor, knowing the boundary conditions, such as type of rock to be excavated, type of explosive used and cut.

The following formula (C.2) is used to estimate the Powder factor:

$$P.F. = \left(\frac{10}{S} + 0.6\right) \cdot A \cdot B \cdot C \quad (C.2)$$

where A, B, C are coefficients accounting, respectively, for the type of rock, of explosive and cut. The following Table C.45 shows the values of coefficient A, depending on the type of rock. The values of coefficient B, depending on the type of explosive used, are shown in Table C.46, those of the coefficient C, related to the type of cut used, are shown in Table C.47.

Table C.45 - Values of the coefficient A

Class	Protodyakonov class	Examples	A coefficient
1	0	Quartzites, sound porphyries	1.3
2	I	Sound granites and gneiss	1.0
3	II	Strong limestones	0.9
4	III	Strong schists and slates	0.8
5	IV	Soft limestones, marl, gypsum	0.5

Table C.46 - Values of the coefficient B

Class	Explosive type	B coefficient
1	Straight gelatin dynamite (γ>1.5)	0.95
2	Semigelatin dynamite (γ >1.4)	1.0
3	Other heavy explosives (γ >1.2)	1.1
4	N.A. based, powder explosives (γ <1.2)	1.2

Table C.47 - Values of the coefficient C

Class	Type of cut	C coefficient
1	Inclined holes cut (V cut)	1.0
2	Fan cut	0.9
3	Parallel holes cut	1.45

The empirical formula for calculating the Specific Drilling is (the values of S.D. calculated with this formula are valid for the range of diameters from 30 to 35 mm) is given in the following:

$$S.D. = 2.3 \cdot \left(\frac{10}{S} + 0.6\right) \cdot A \cdot B' \qquad (C.3)$$

where A is the coefficient determined by using Table C.45 and B' is the coefficient dependent on the explosive type, which is found by using the values shown in Table C.48.

Table C.48 - Values of the coefficient B'

Explosive class	B' coefficient
1	0.60
2	0.65
3	0.80
4	1.20

In the case considered, the value of the coefficient A is 1.3, the coefficient B is 1.2, the coefficient C is 1.45, and finally the value of the coefficient B' is 1.2. The comparison among the values calculated during the development of the improvement project and of those calculated using the empirical formulas of Mancini and Pelizza is reported in the Table C.49 (as for the Powder Factor) and in Table C.50 (as for the Specific Drilling).

Table C.49 - Comparison between the Powder Factor values obtained in the project and those obtained with the empirical formula of Mancini and Pelizza

	Section (m^2)	Powder factor (P.F.) (kg/m^3)	Powder Factor calculated with the formula of Mancini and Pelizza (kg/m^3)
Artisanal Mine	6.850	1.528	4.659
Phase 1; 2.5 x 2.5 m	6.250	3.904	4.976
Phase 1; 3.0 x 3.0 m	9.000	3.020	3.871
Phase 1; 3.5 x 3.5 m	12.250	2.221	3.204
Phase 2; 2.5 x 2.5 m	5.580	3.547	5.411
Phase 2; 3.0 x 3.0 m	8.050	2.587	4.167
Phase 2; 3.5 x 3.5 m	11.080	1.878	3.399
Phase 3; 2.5 x 2.5 m	5.730	3.353	5.305
Phase 3; 3.0 x 3.0 m	8.270	2.342	4.092
Phase 3; 3.5 x 3.5 m	10.950	1.784	3.423

Table C.50 - Comparison between the Specific Drilling values obtained in the project and those obtained with the empirical formula of Mancini and Pelizza

	Section (m^2)	Specific Drilling (S.D.) (m/m^3)	Specific Drilling calculated with the formula of Mancini and Pelizza (m/m^3)
Artisanal Mine	6.85	2.481	7.391
Phase 1; 2.5 x 2.5 m	6.25	5.791	7.894
Phase 1; 3.0 x 3.0 m	9.00	4.451	6.139
Phase 1; 3.5 x 3.5 m	12.25	3.273	5.082
Phase 2; 2.5 x 2.5 m	5.58	7.349	8.583
Phase 2; 3.0 x 3.0 m	8.05	5.219	6.610
Phase 2; 3.5 x 3.5 m	11.08	3.969	5.391
Phase 3; 2.5 x 2.5 m	5.73	5.762	8.415
Phase 3; 3.0 x 3.0 m	8.27	4.233	6.491
Phase 3; 3.5 x 3.5 m	10.95	3.380	5.430

The results obtained using the above quoted empirical formulas differ greatly from the values that have been calculated with the data obtained during the inspections carried out in the artisanal mine. This difference is very evident above all with regard to the value of Specific Drilling. It should be noted that, as for the calculated value of P.F., it is not possible to define the value of the coefficient C of the empirical formula, since no drilling scheme is used.

Regarding the data of P.F. and S.D. calculated for steps 1, 2 and 3 of the improvement project, the data are quite similar to those obtained with the empirical formulas. It should be noted that in all cases, the results are slightly lower.

Above all, as for the P.F. of phases 2 and 3, the result of the project is lower than the empirical one, and this is due to the use of detonating cord in the contour holes (walls and roof).

The values of S.D. are generally lower than the empirical ones because the diameter of the holes is wider than that to which the formula refers (38.1 mm for phases 1 and 2, 50.8 mm for phase 3, where the discrepancy between the results is more evident).

As for the improvement of the conditions of work safety, carrying out a quantitative analysis is impossible due to the absence of data and the lack of observation of the results obtained by the improvement process. However, a qualitative analysis can be performed using the risk rating matrix method. This technique is based on the estimation of the consequences and frequency of occurrence of an accident. An example of a risk matrix is shown in the following Figure C.163.

ACTUAL RISK OUTCOME			
LOW	MODERATE	SIGNIFICANT	HIGH

		CONSEQUENCE				
		INSIGNIFICANT 1	MINOR 2	MODERATE 3	MAJOR 4	CATASTROPHIC 5
LIKELIHOOD	ALMOST CERTAIN 5	5	10	15	20	25
	LIKELY 4	4	8	12	16	20
	POSSIBLE 3	3	6	9	12	15
	UNLIKELY 2	2	4	6	8	10
	RARE 1	1	2	3	4	5

RISK LIKELIHOOD TABLE - Guidance					
DESCRIPTOR	1	2	3	4	5
	RARE	UNLIKELY	POSSIBLE	LIKELY	ALMOST CERTAIN
FREQUENCY	Not expected to occur for years	Expected to occur at least annually	Expected to occur at least monthly	Expected to occur at least weekly	Expected to occur at least daily
PROBABILITY	<1%	1-5%	6-20%	21-50%	>50%
	Will only occur in exceptional cases	Unlikely to occur	Reasonable chance to occur	Likely to occur	More likely to occur than not

Figure C.163 - Example of a risk rating matrix

In general, two aspects of work safety that can be improved in the application of the proposed project can be assessed: safety in preventing rock falls from the tunnel contour and safety in the use and handling of explosives.

As for the safety in using explosives, the actual risk outcome is high: the occurrence of an accident is high (5 - almost certain) and the possible consequences are serious (4- majors). The calculated risk is therefore equal to 20. Implementing training courses for the correct use of explosive in safety for the workers can reduce the expected frequency of occurrence and the consequences. In particular, the likelihood can be reduced to be rare (value 1) and the consequences to be minor consequences (value 2). The estimated value of the risk after the application of the safety procedures and the elimination of incorrect behavior is 2.

The risk of rock falls from the tunnel contour is currently high. The rock that is very disturbed by incorrect blasts can break down in portions of metric dimensions. The consequences can be catastrophic (value 5) and the expected frequency of occurrence is monthly (according to what was observed during the inspections and according to what the workers told during the interviews, so the value is equal to 3). The estimated risk value is 15. By adopting new excavation techniques, a more regular contour can be obtained, by significantly reducing the damage to the boundary rock, which will therefore show fewer discontinuities and fractures. The risk is further reduced with the application of monitoring plans, supports (rock bolts, electro-welded mesh, projected concrete, etc.). The estimate of the frequency is reduced to rare (value 1) and the consequences are reduced to be minor (value 2). The estimated risk value of the implementation of the improvement project is reduced to 2.

It should be remembered with the utmost attention that these results are purely qualitative and are based on evaluations without certain data.

The analysis of the results achievable thanks to the implementation of the proposed project inside the artisanal mine is sufficiently completed. It is therefore clear that the suggested improvements are congruent with the objective set to increase the safety and productivity of the excavation techniques. Finally, it should be said that the study could be completed with the execution of an economic analysis, studying the variation in production costs compared to the conditions prior to the implementation of the improvement design phases.

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WEB REFERENCES

References to the web pages from which the figures were taken:

[1] http://www.geocurrents.info/cartography/customizable-maps-mexico-argentina-chile-peru-ecuador/attachment/chile-regions-map.

[2] https://pt.wikiloc.com/trilhas-mountain-bike/chancon-mina-el-ingles-6344956/photo-3489500.

ATTACHMENTS

The attachments of the thesis consist of 55 graphic projects, in A3 format, realized with the AutoCAD® technical drawing program. These projects were designed and realized with the aim of providing the mine personnel with a technical basis to illustrate the current methodology and technique of work, highlighting the problems and critical aspects that need to be improved and the new improvement techniques technologies proposed in all three phases in which the development of the improvement process explained in the thesis is articulated.

The projects concerning the explication of the current working method analyze the explosives used, the connection with the trigger system, the drilling and charge schemes, the analysis of working times and the blast results.

The projects concerning the explanation of the new technologies and working methodology to be adopted in the improvement of the excavation technique for the three phases of the process of development are: the explosives used and the priming system with the relative junctions, the boomer scheme, the drilling and charge schemes, the detonation sequence and the main blast data.

All the projects have been realized taking into account the cultural and educational level of the subjects who will have to read, interpret and put into practice them. Therefore, they were made as easy as possible, using easy graphical diagrams of all the details.

The catalogs of all the explosives and priming systems currently used in the mine and those that will be used in the improvement project have also been attached. Finally, the catalog of the available boomer was also reported.

A.1 - Index of the attachments

A.1.1 - Projects related to the illustration of the current situation

1- Geometric relief of the blast – Horizontal sections;
2- Geometric relief of the blast – Vertical sections;
3- Details related to the explosives used in the blast;
4- Geometric relief of the charge of the holes (1);
5- Geometric relief of the charge of the holes (2);
6- Geometric relief of the charge of the holes (3);
7- Working time analysis;
8- Relief of the face before and after the blast (1);
9- Relief of the face before and after the blast (2);

A.1.2 - Projects related to the illustration of the improvement solutions proposed

10- Detail of the nonel equipment and use;
11- Phase 1: Drilling and charge schemes for section 2.5 x 2.5 m;
12- Phase 1: Trigger times for section 2.5 x 2.5 m;
13- Phase 1: Drilling and charge schemes for section 3.0 x 3.0 m;
14- Phase 1: Trigger times for section 3.0 x 3.0 m;

A.2 - Technical catalogs

1- Riocap® - Cápsula detonante;
2- Riofuse® – Mecha de seguridad;
3- Anfo Premium® – Agente de voladura de alta calidad;
4- Emultex® CN – Emulsión envasada de diámetro pequeño;
5- Detonador No Eléctrico;
6- Detonador No Eléctrico Dual;
7- Pentacord® Especial 80P – Cordón detonante;
8- Britacord® – Cordón detonante de bajo gramaje;
9- Boomer 104 – Face drilling rig.

Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:

Simone BEZ

Badge Number: 236349

TECHNICAL DRAWINGS

POLITECNICO DI TORINO

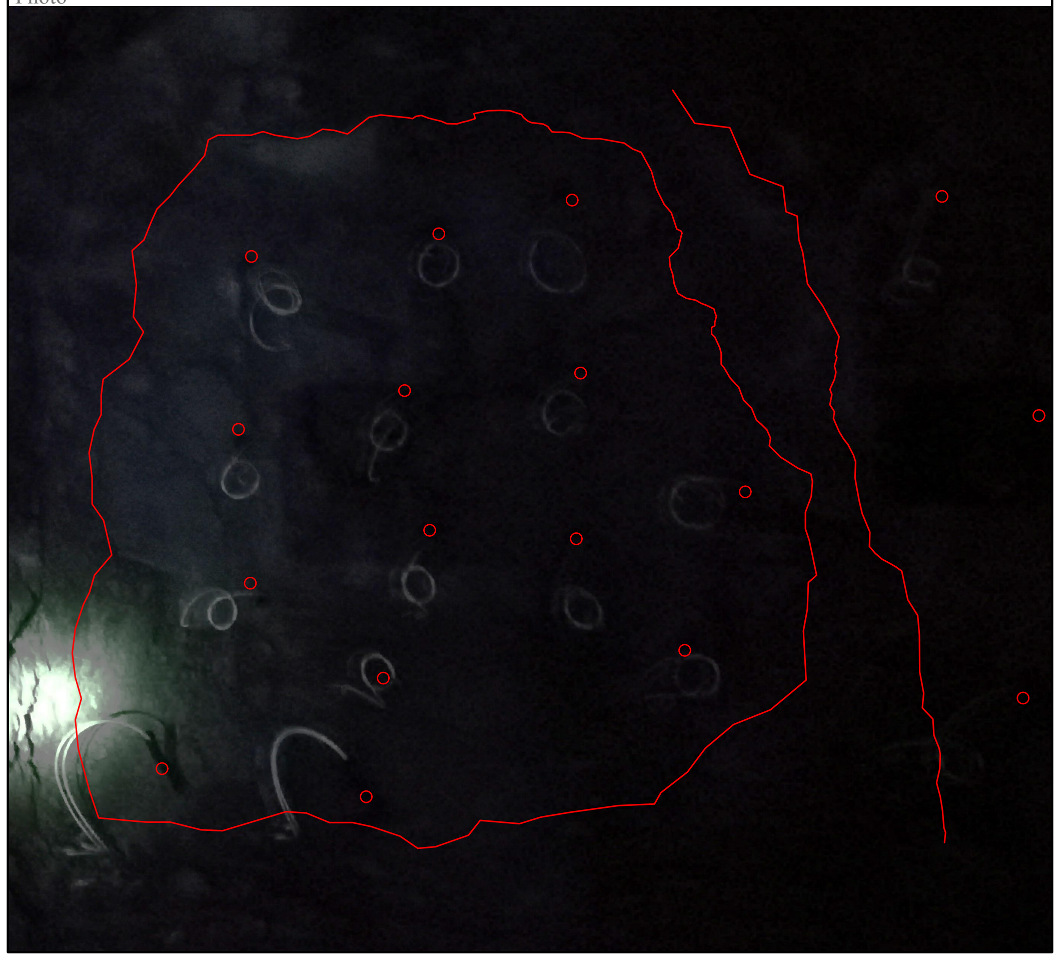
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:

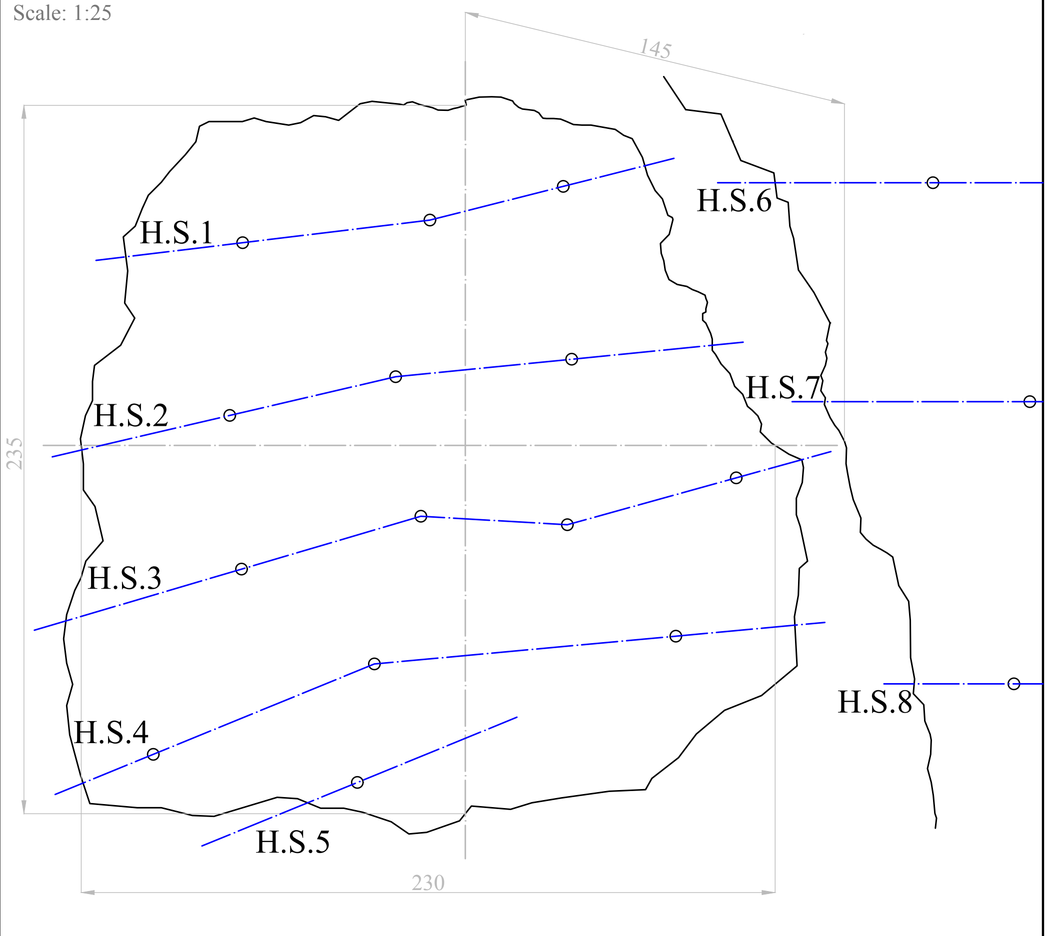
Prof. Marilena Cardu

A.Y. 2018 - 2019

Photo



Front view
Scale: 1:25



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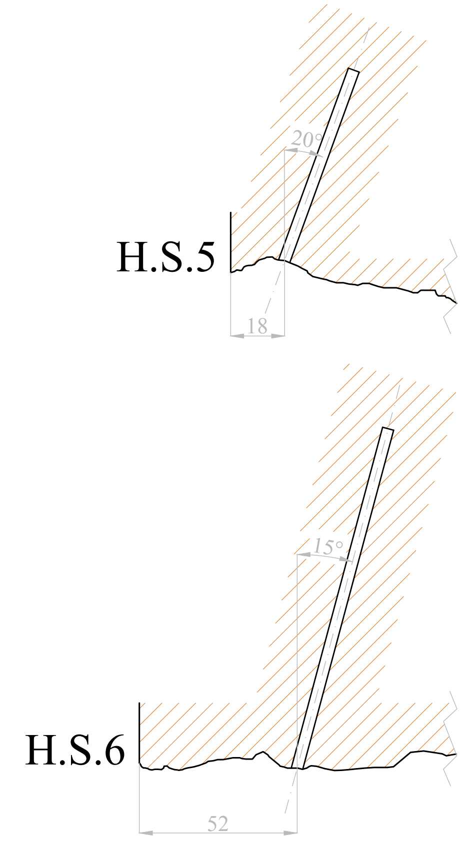

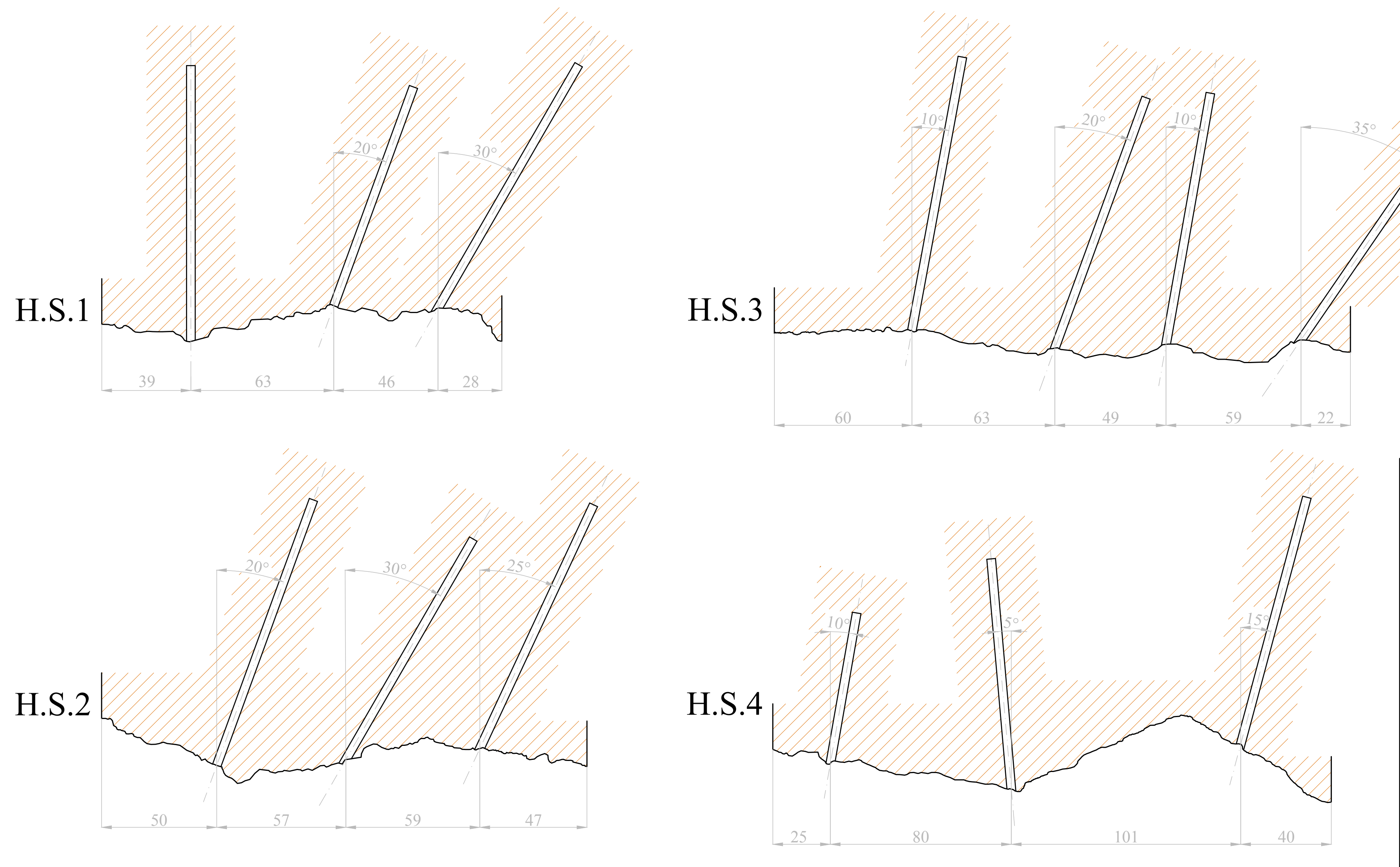

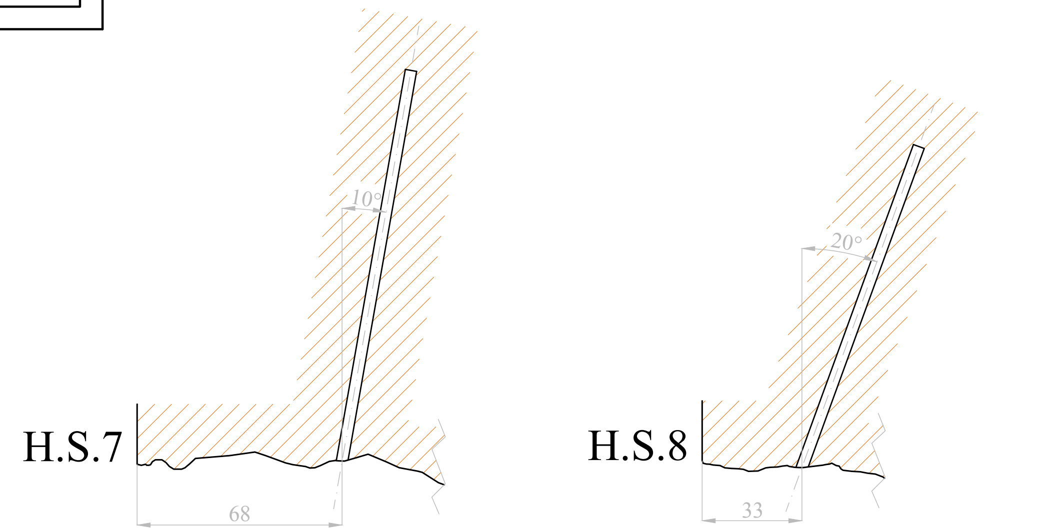

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

Technical drawing:

1

GEOMETRIC RELIEF OF THE BLAST HORIZONTAL SECTIONS

Scale: 1:25

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

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Prof. Marilena Cardu

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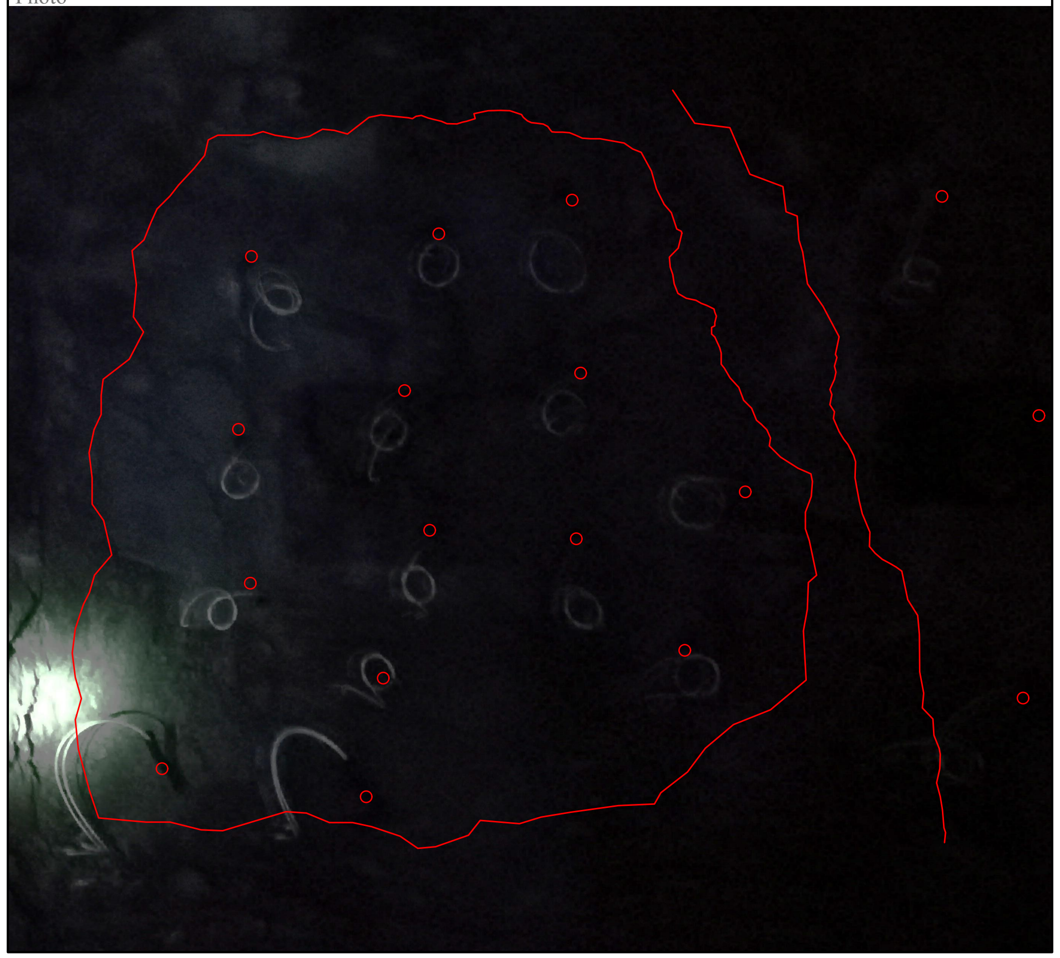
Photo



Front view

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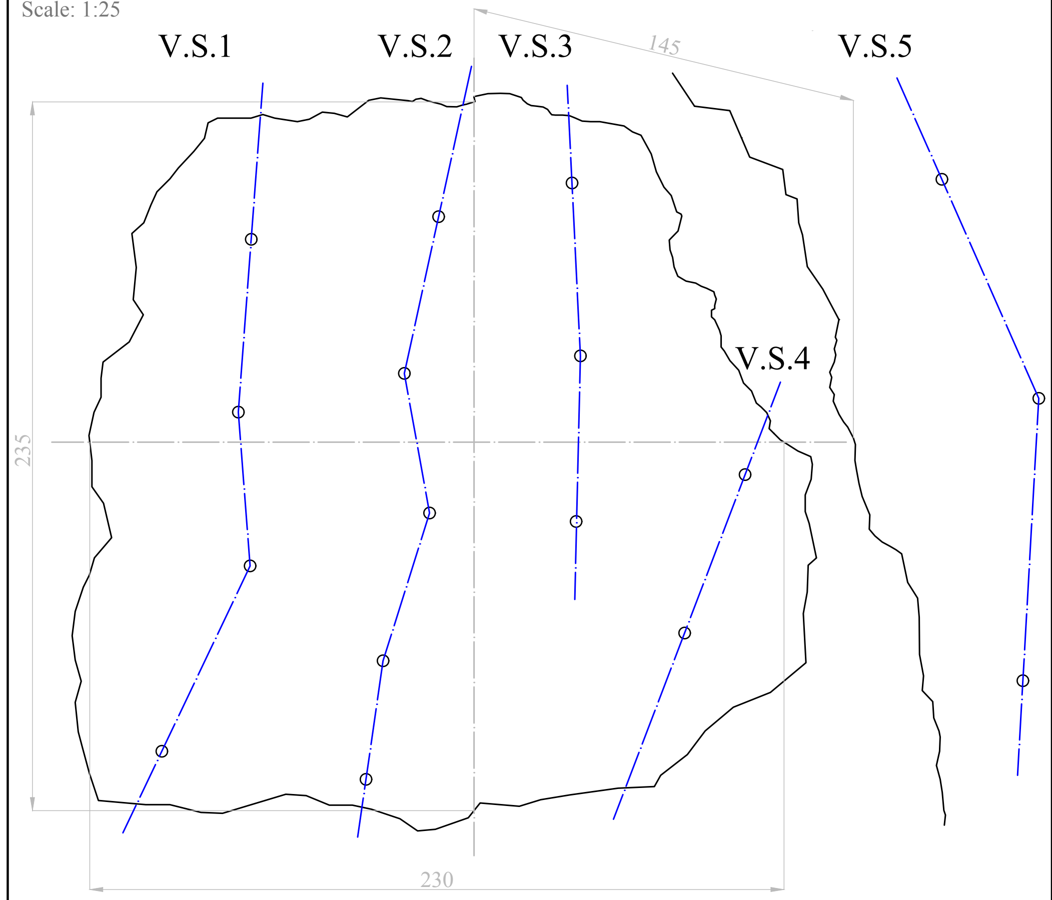
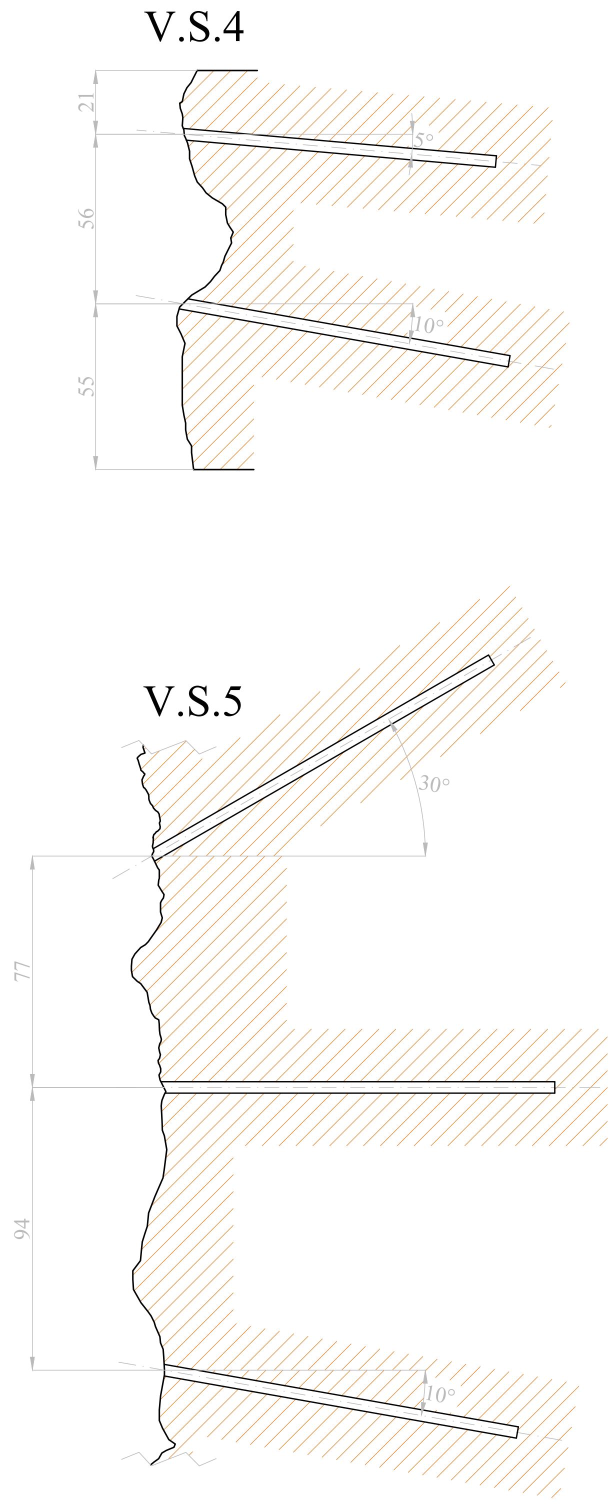
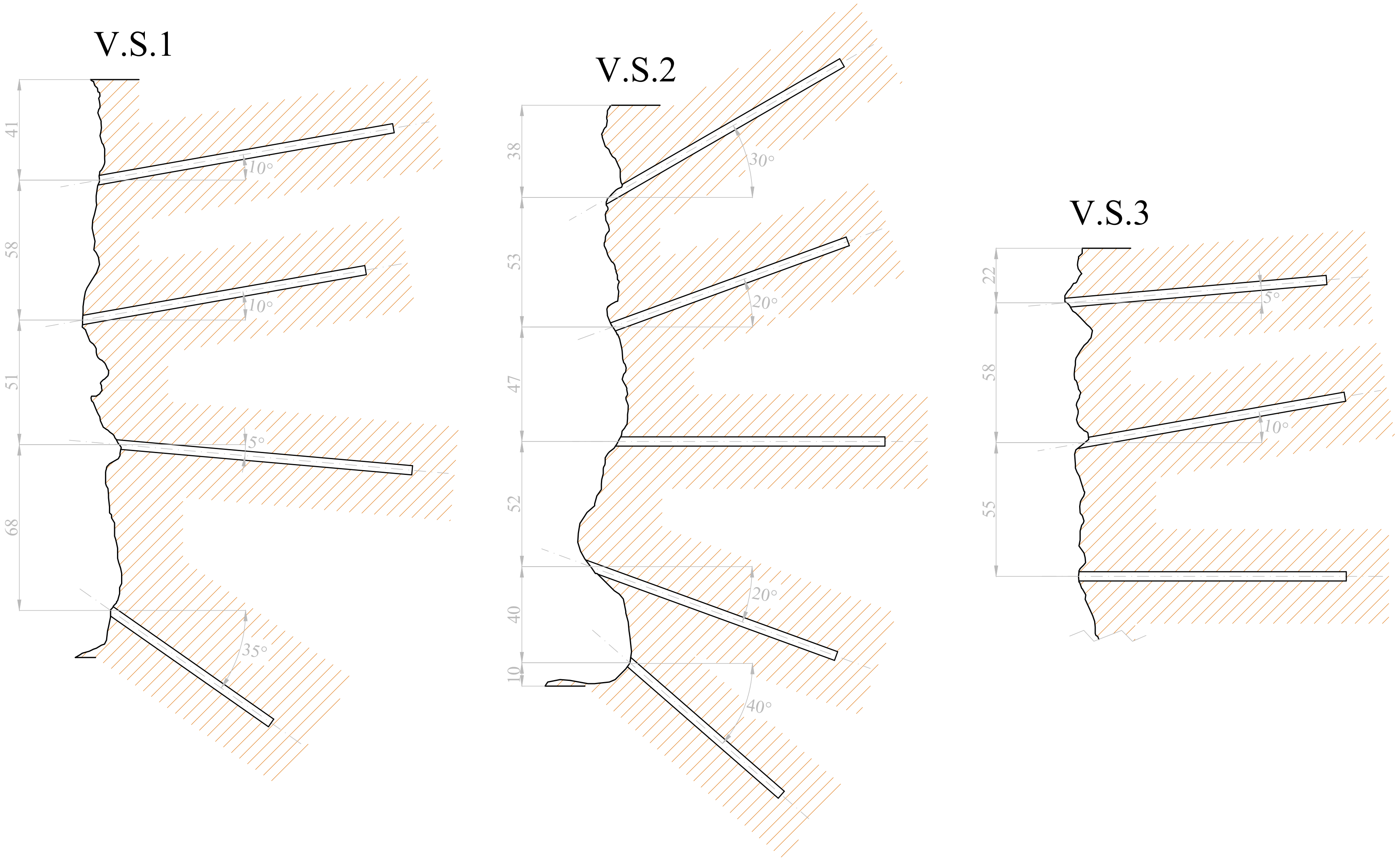


Vertical sections

Scale: 1:25





Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
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Badge Number: 236349

2

Technical drawing:
GEOMETRIC RELIEF OF THE BLAST VERTICAL SECTIONS

Scale: 1:25

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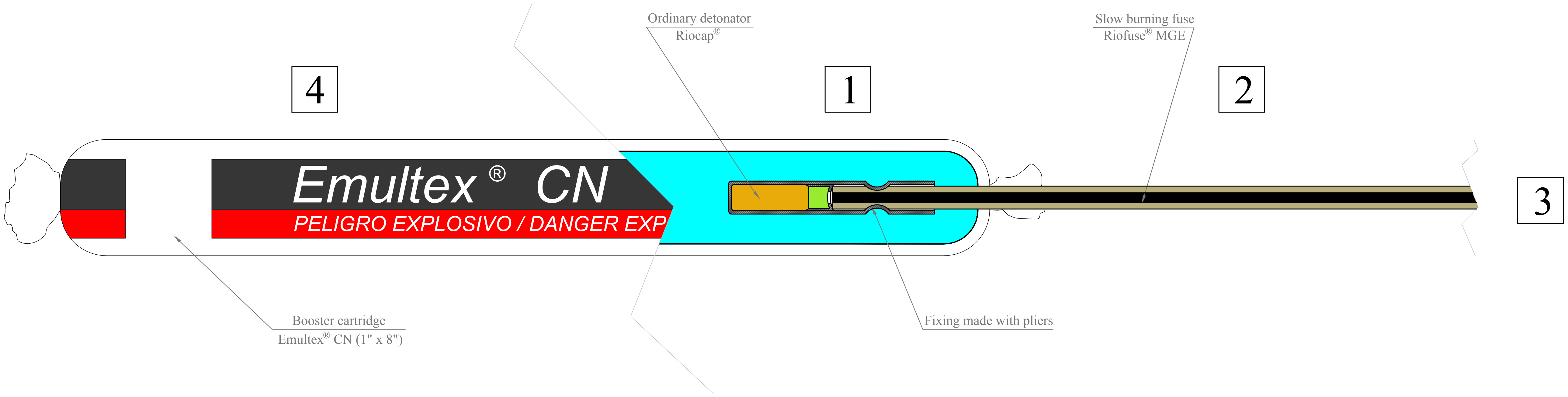
Detail of the junction booster - detonator - slow burning fuse
Scale: 1:1



Detail of the ordinary detonator
Scale: 2:1

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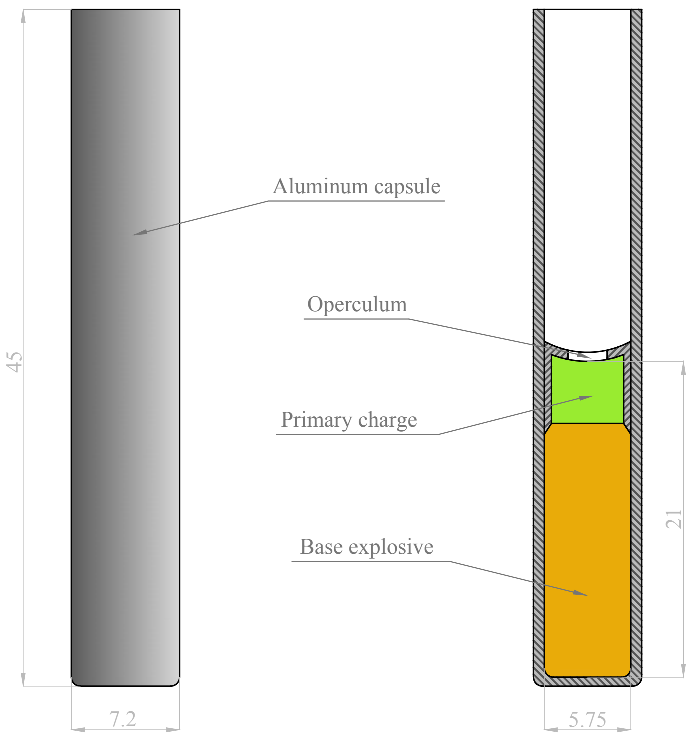

Detail of the slow burning fuse
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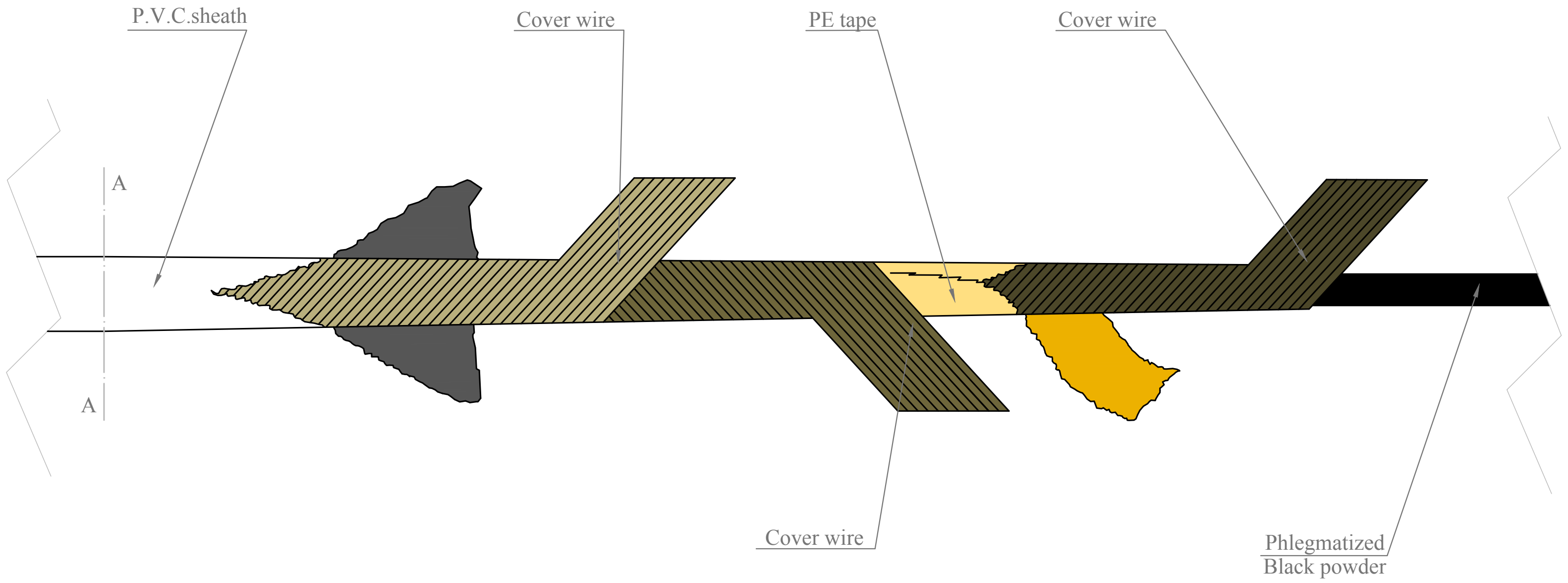
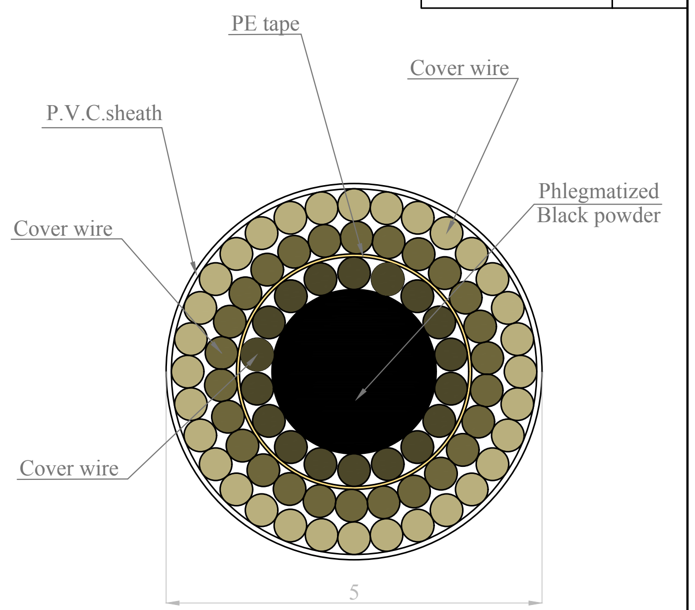
MAXAM 2



Section A - A of the slow burning fuse
Scale: 10:1

MAXAM 3



Detail of the booster cartridge
Scale: 1:1

Enaex 4

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

3

Technical drawing:
DETAILS RELATED TO THE EXPLOSIVES USED IN THE BLAST

POLITECNICO DI TORINO

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Supervisor:
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Firing order
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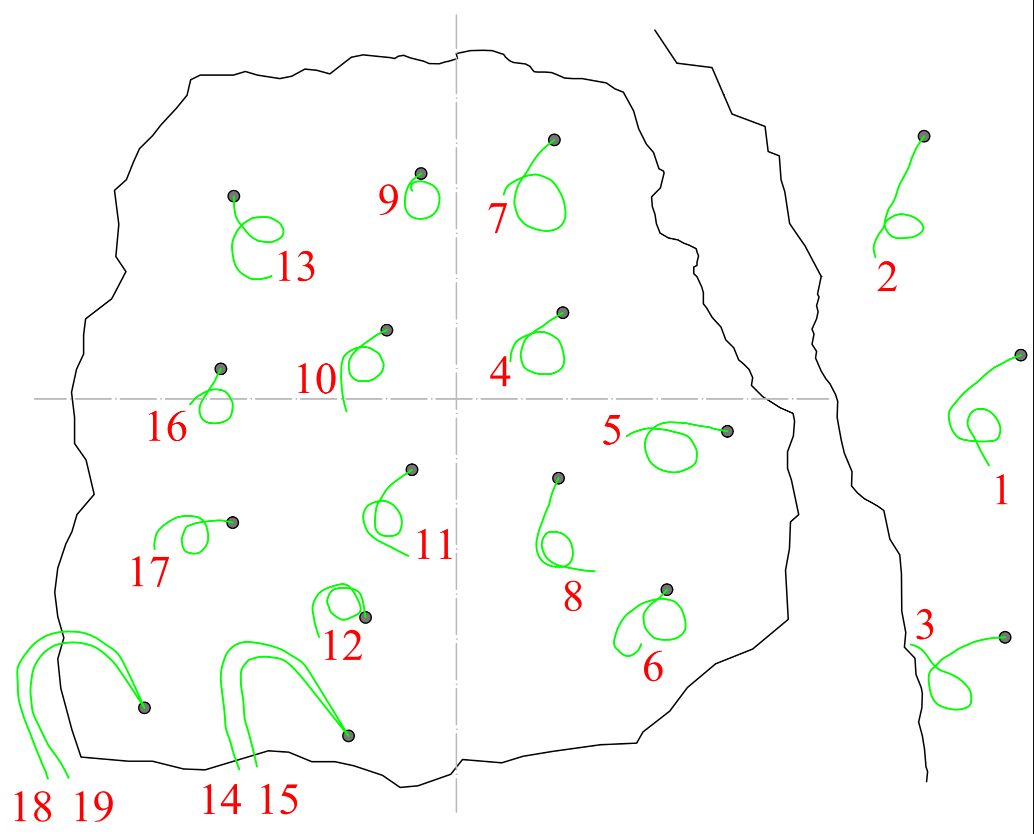

Detail of the charge
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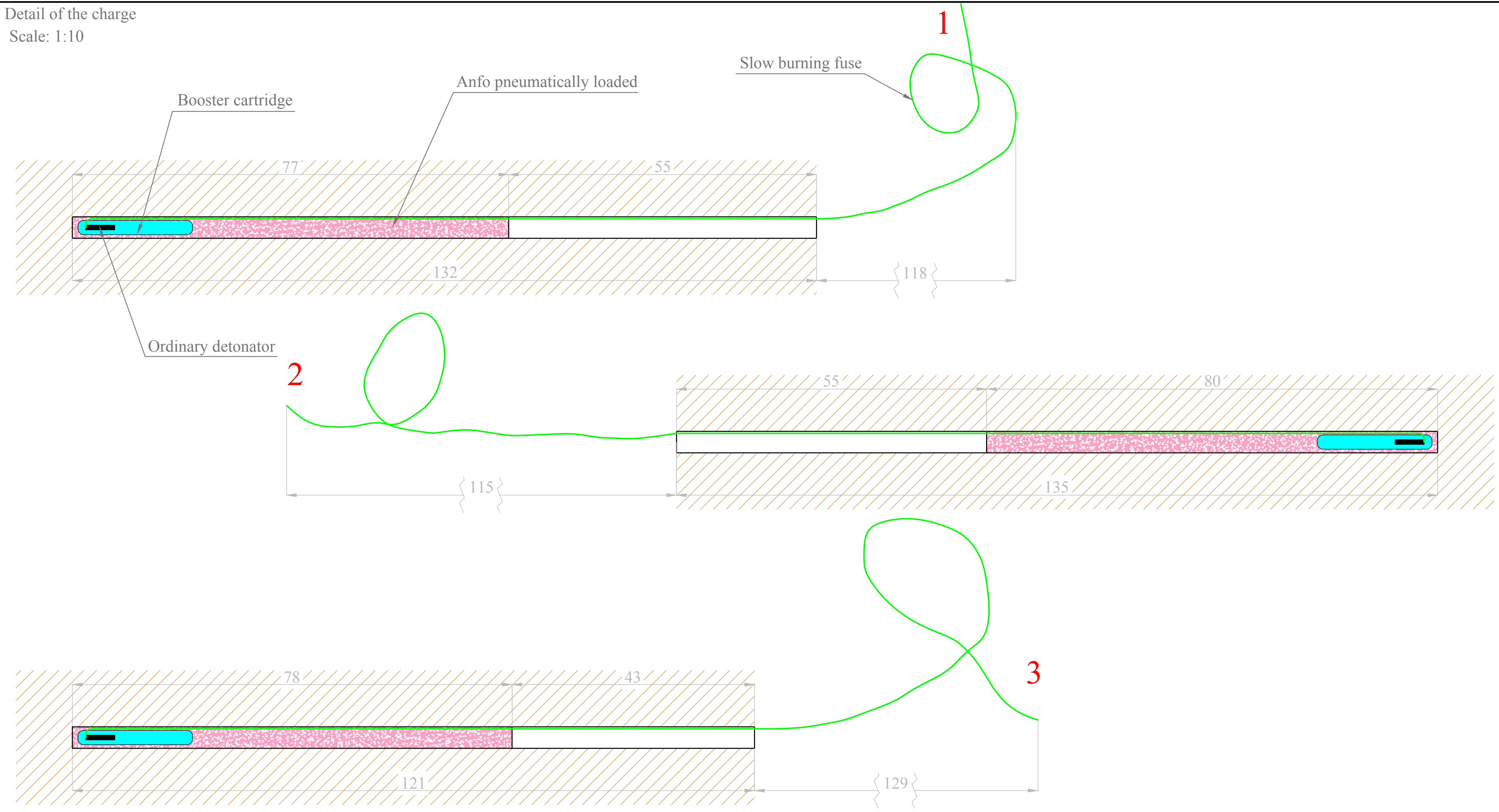

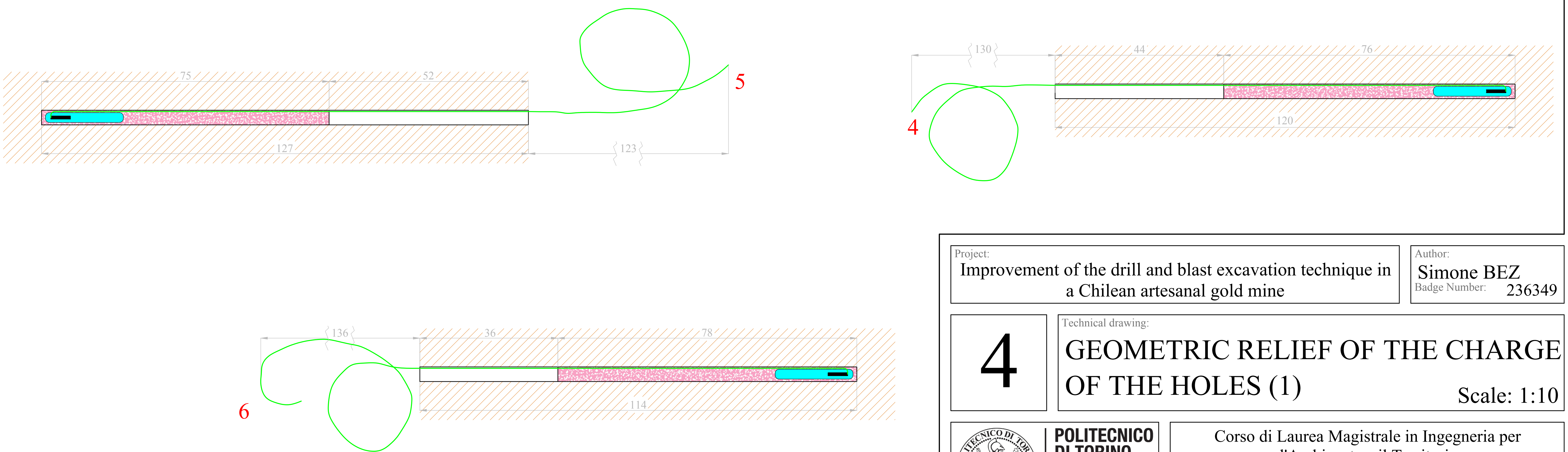

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
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Badge Number: 236349

4

Technical drawing:
GEOMETRIC RELIEF OF THE CHARGE OF THE HOLES (1)
Scale: 1:10

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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A.Y. 2018 - 2019

Firing order
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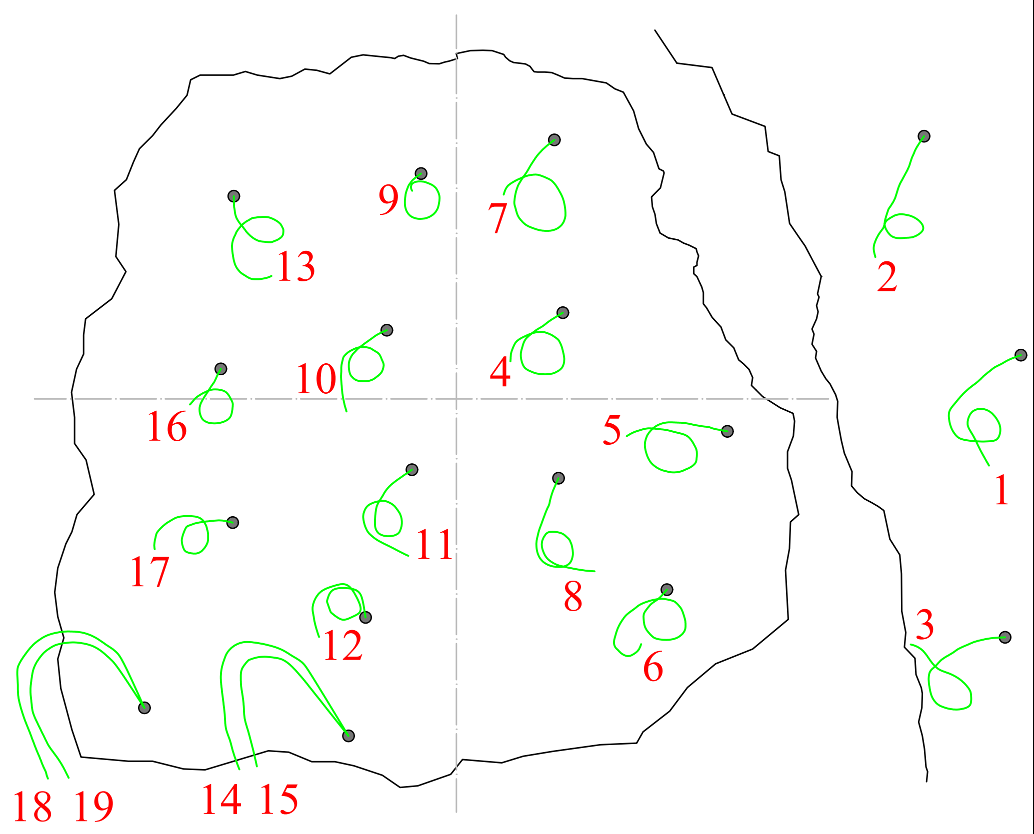
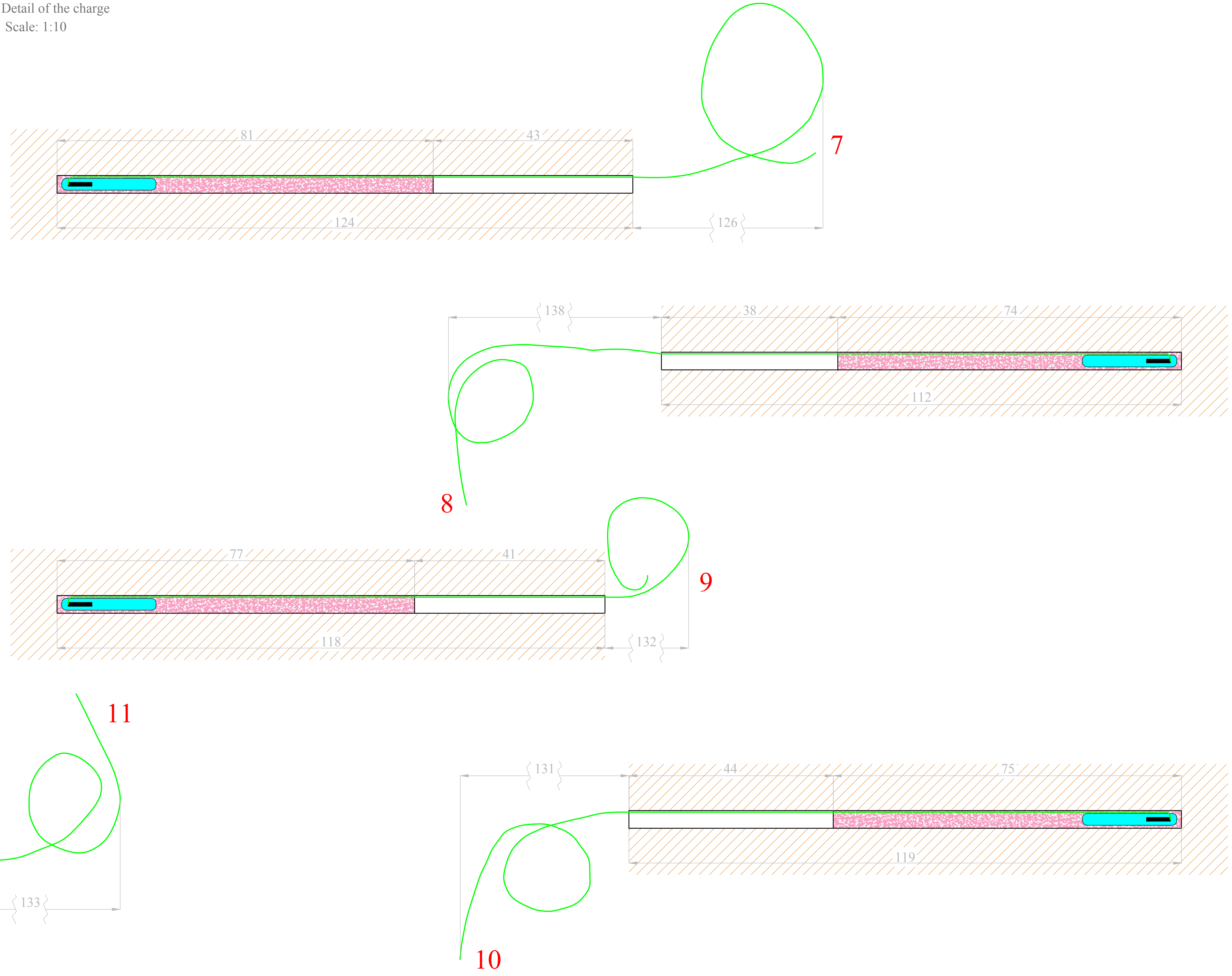
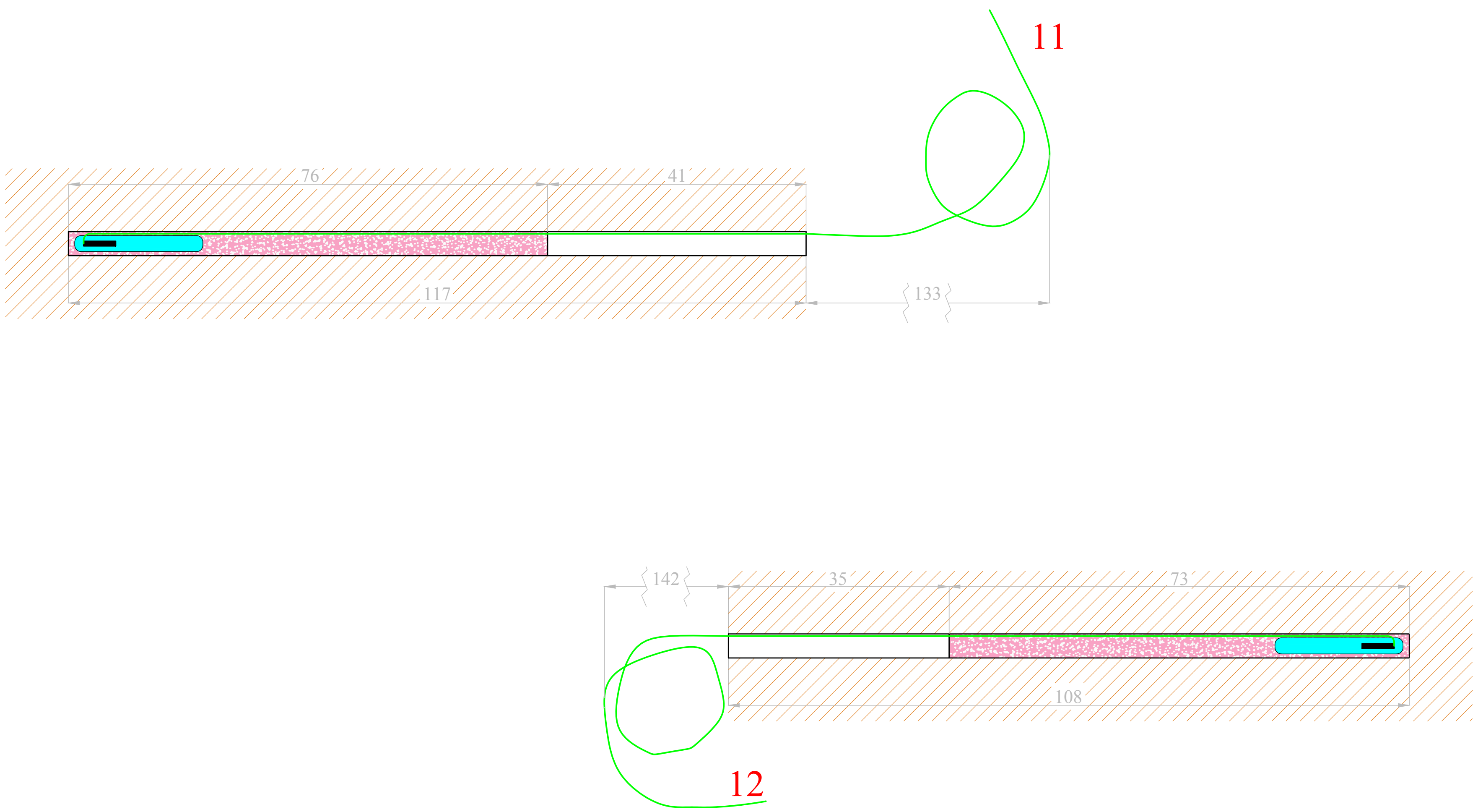


Detail of the charge
Scale: 1:10





Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
GEOMETRIC RELIEF OF THE CHARGE OF THE HOLES (2)

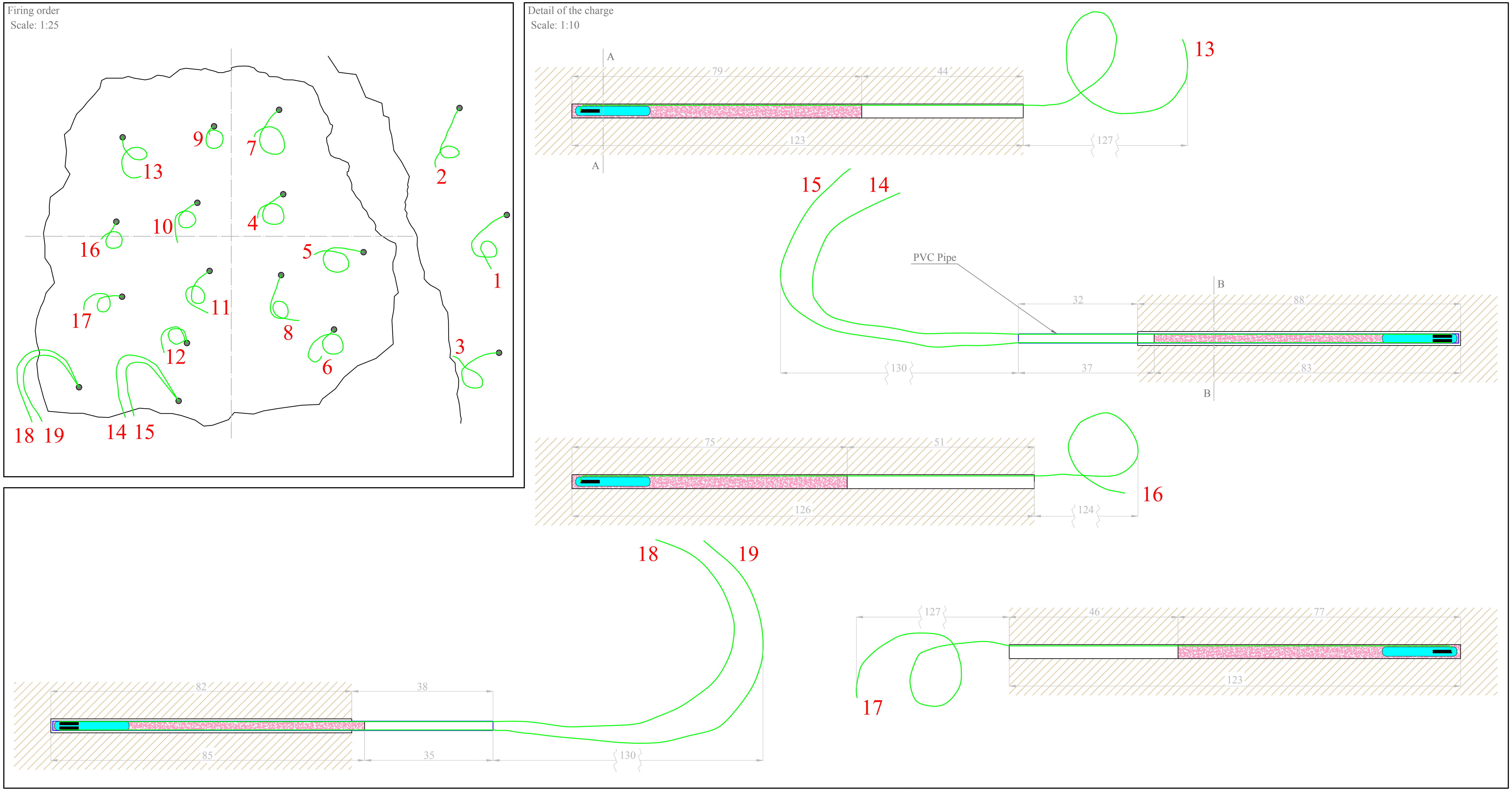
Scale: 1:10

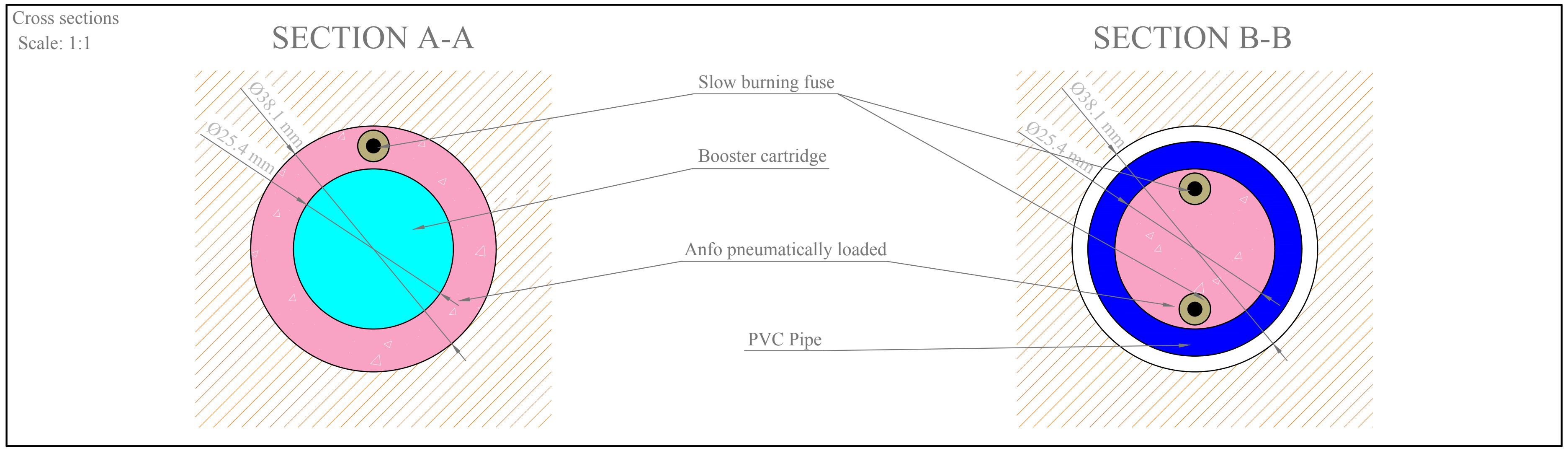
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019





Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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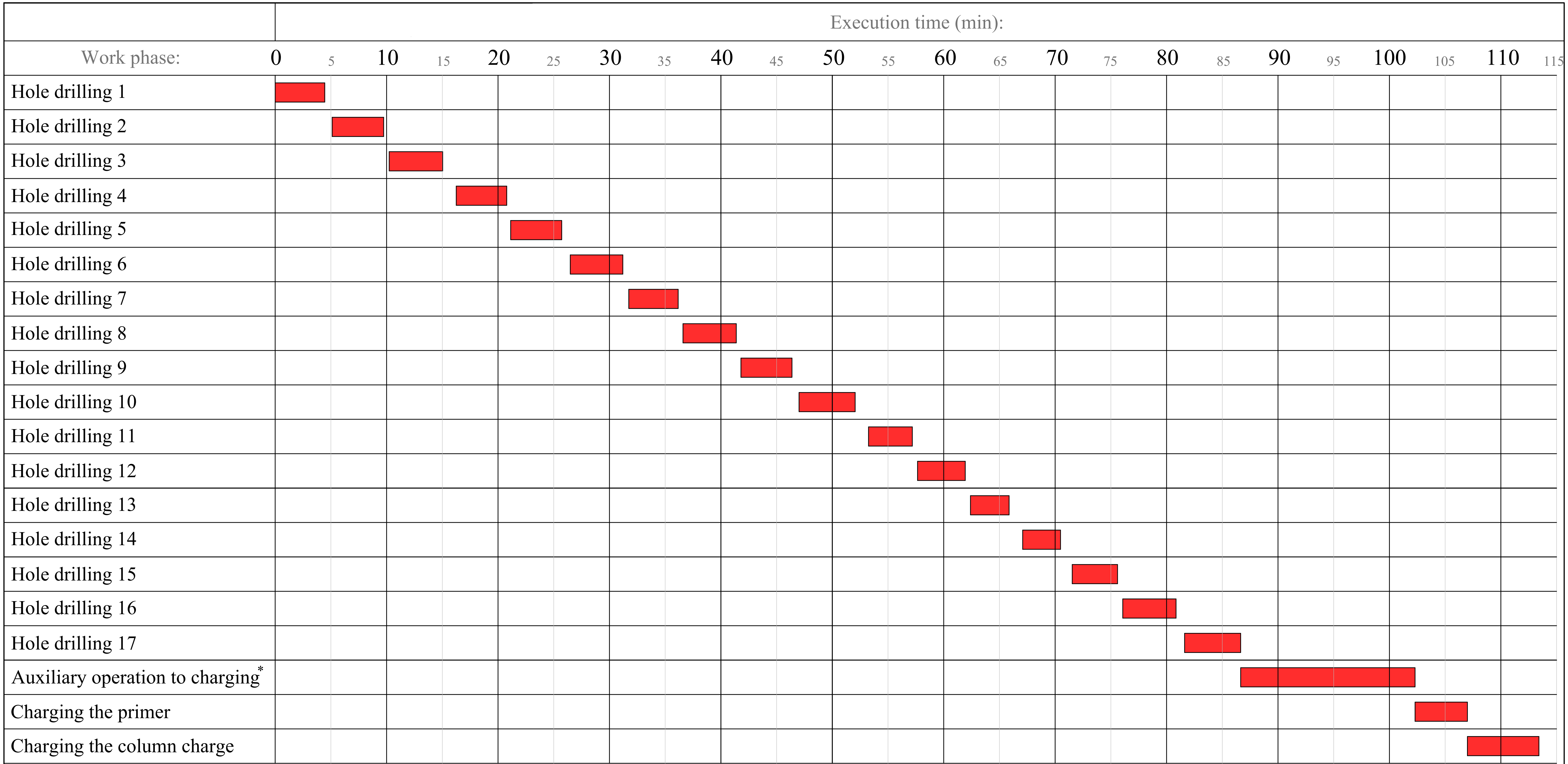
Technical drawing:
GEOMETRIC RELIEF OF THE CHARGE OF THE HOLES (3)
Scale: 1:10

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

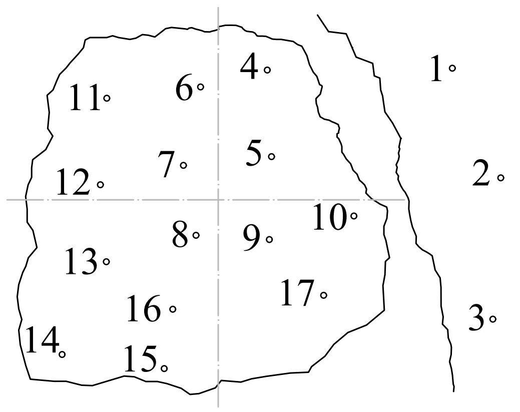
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019



* Cleaning holes with compressed air, explosive supply (booster and column charge) and primer preparation

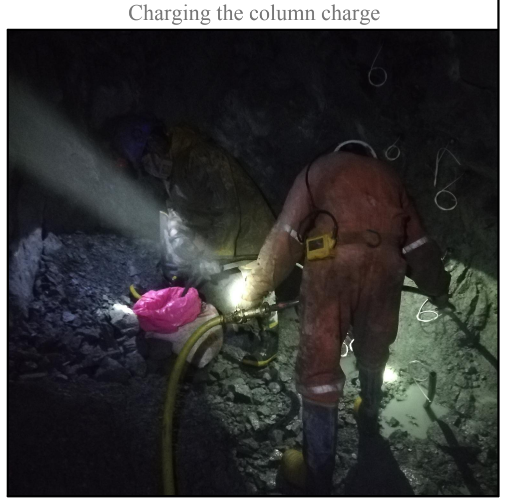
Drilling order
Scale: 1:50



Photos of work operations


Drilling


Charging the primer


Charging the column charge

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
WORKING TIME ANALYSIS

POLITECNICO DI TORINO

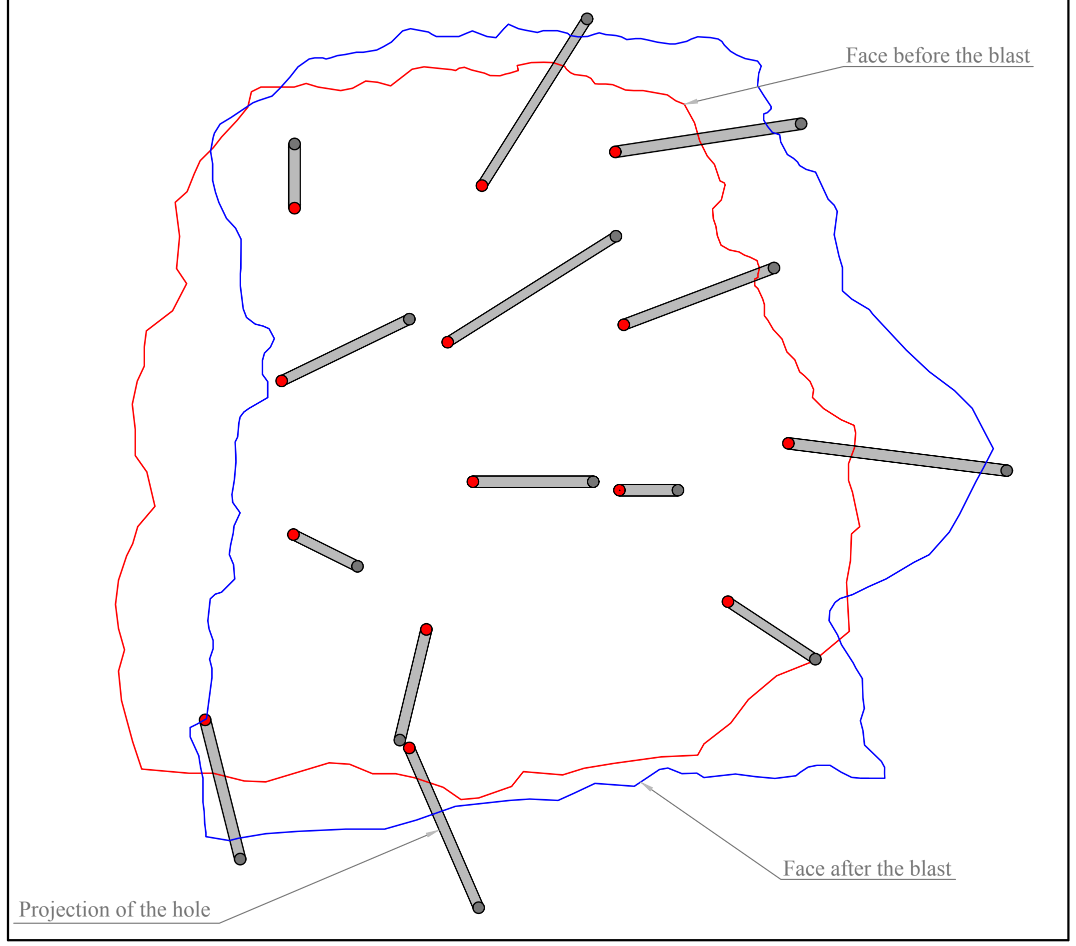
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

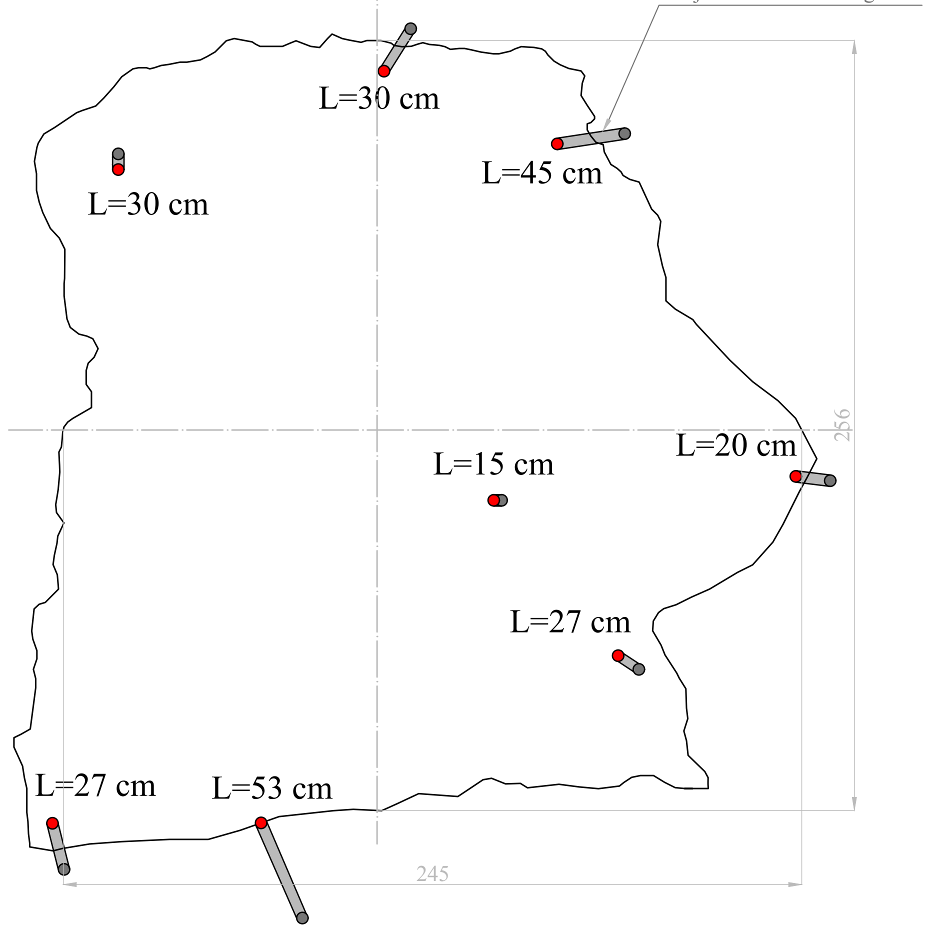
A.Y. 2018 - 2019

Photo

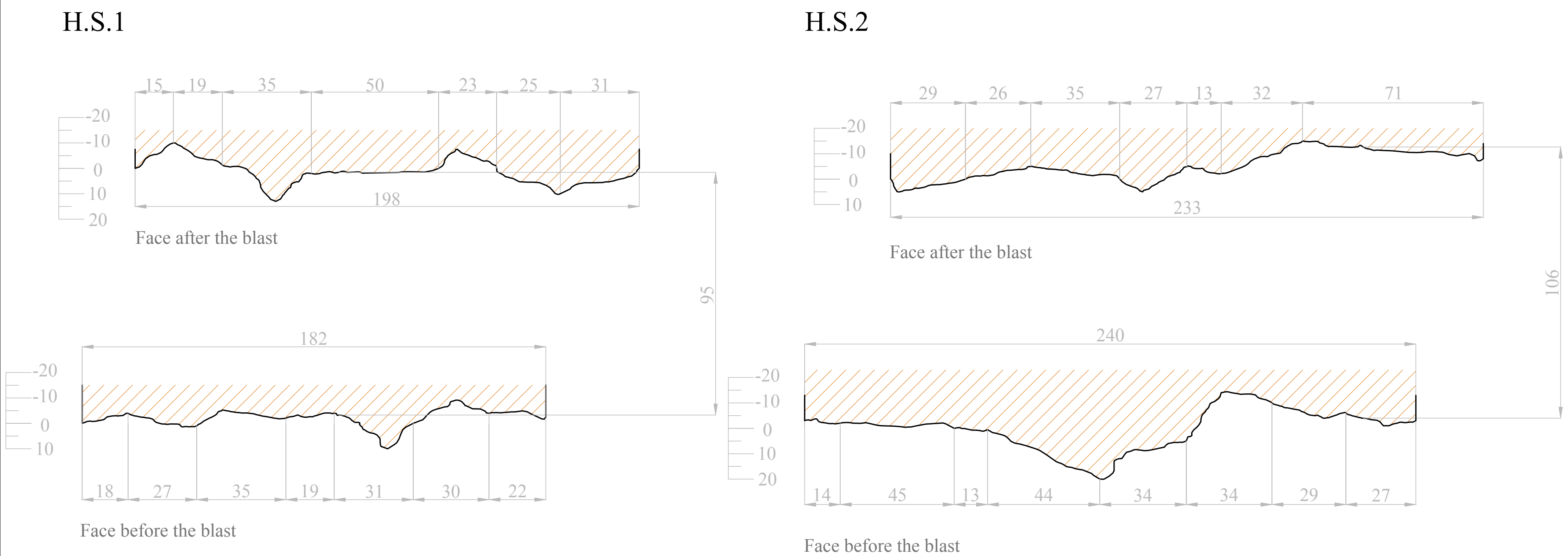
Comparison between the face before and after the blast
Scale: 1:25



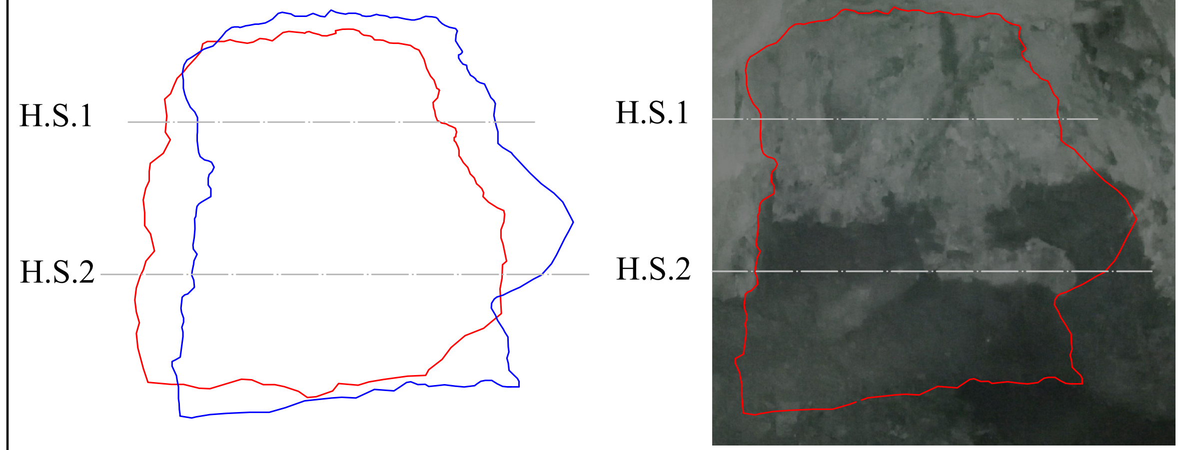
Face after the blast
Scale: 1:25



Horizontal sections
Scale: 1:25



Position of the sections
Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
RELIEF OF THE FACE BEFORE AND AFTER THE BLAST (1)
Scale: 1:25

POLITECNICO DI TORINO

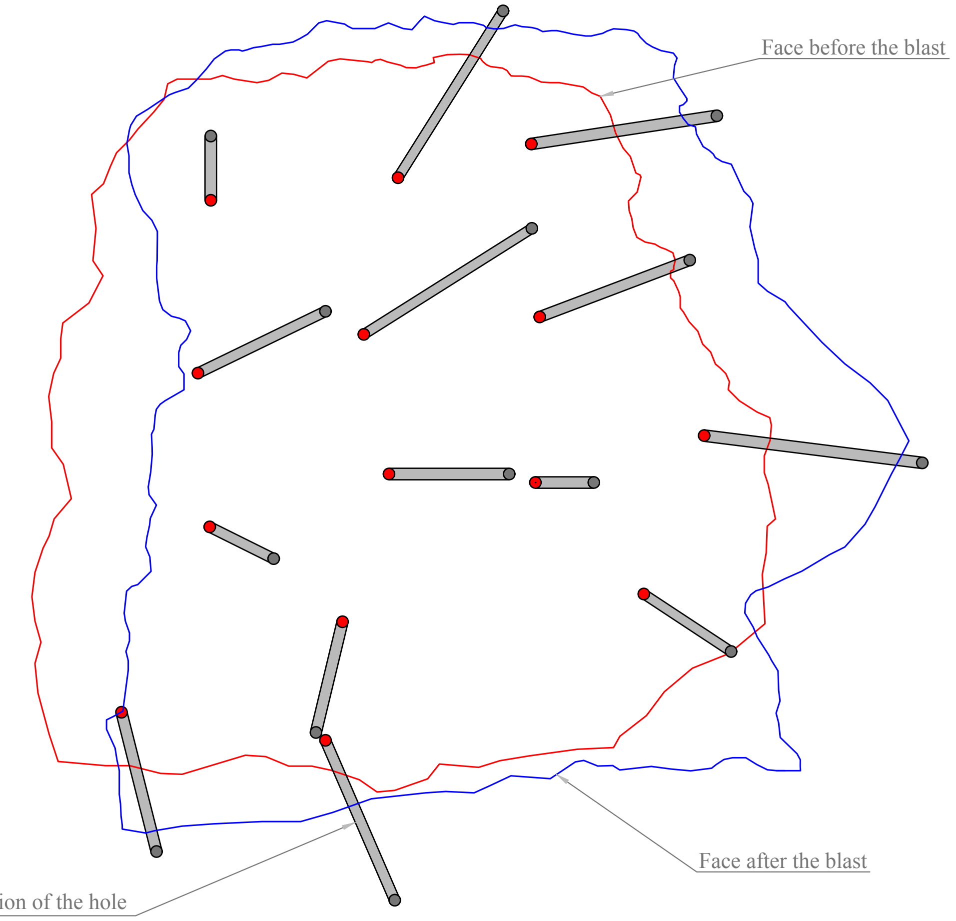
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

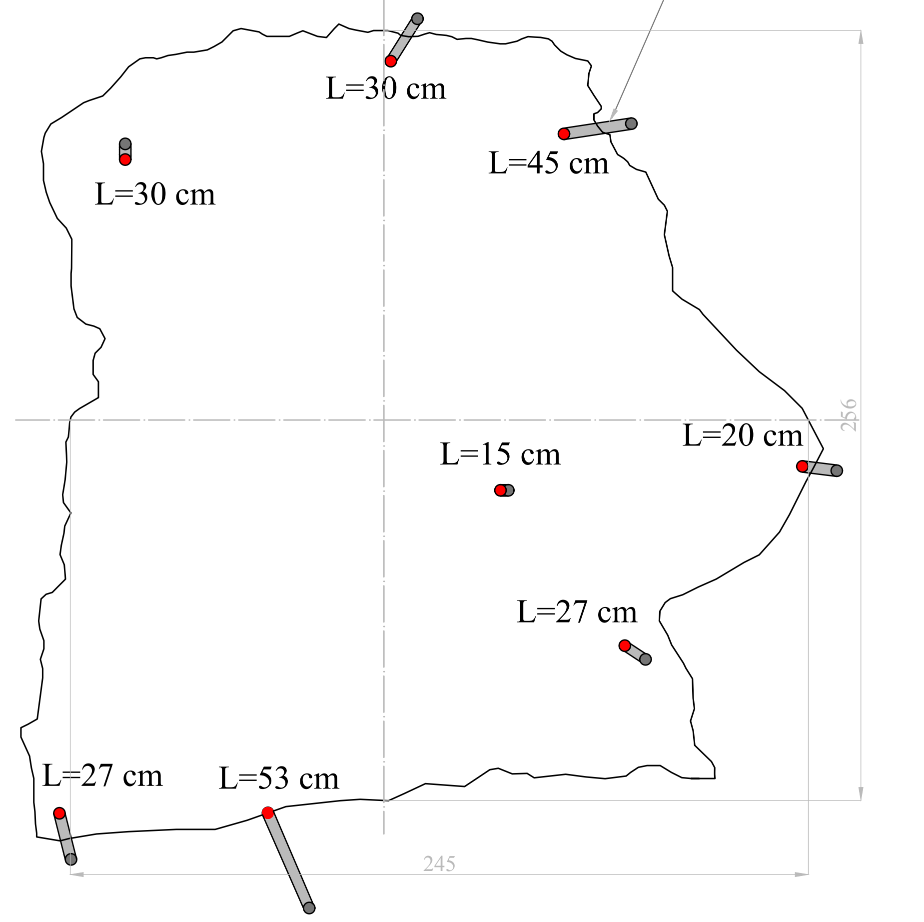
A.Y. 2018 - 2019

Photo

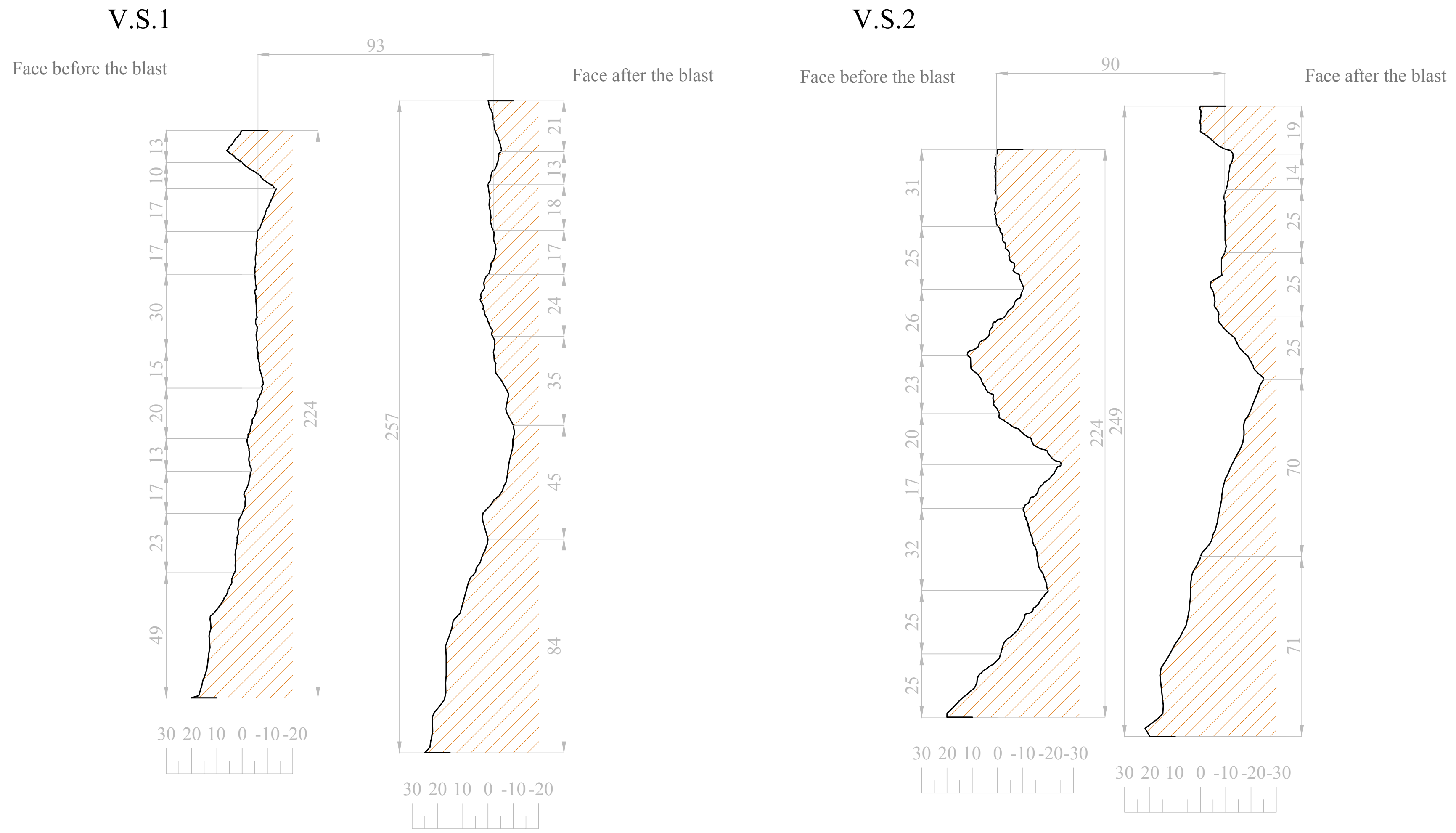
Comparison between the face before and after the blast
Scale: 1:25



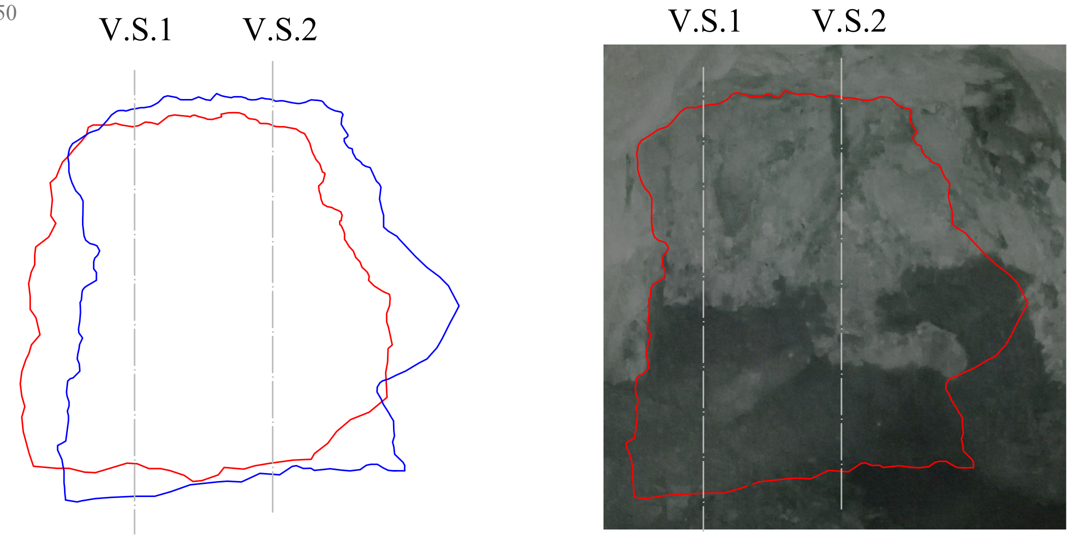
Face after the blast
Scale: 1:25



Vertical sections
Scale: 1:25



Position of the sections
Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
RELIEF OF THE FACE BEFORE AND AFTER THE BLAST (2)

Scale: 1:25

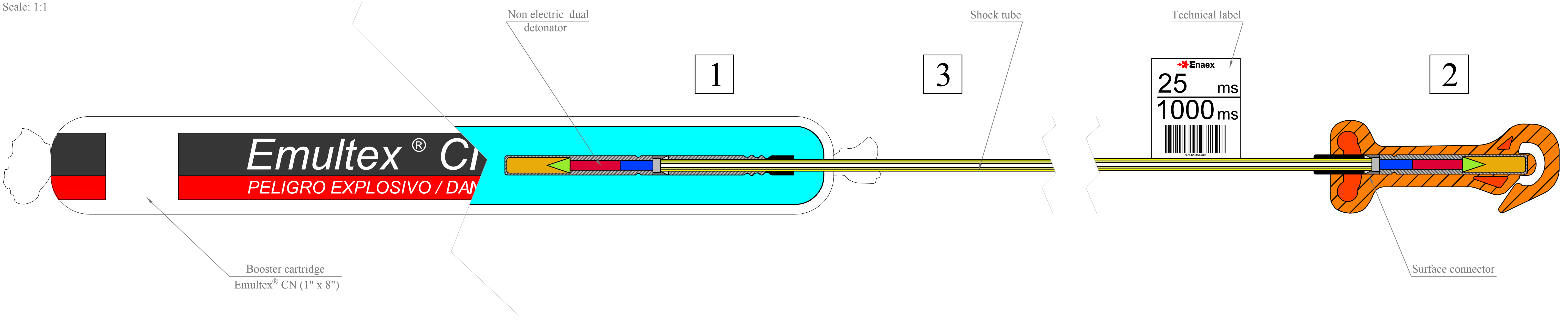
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

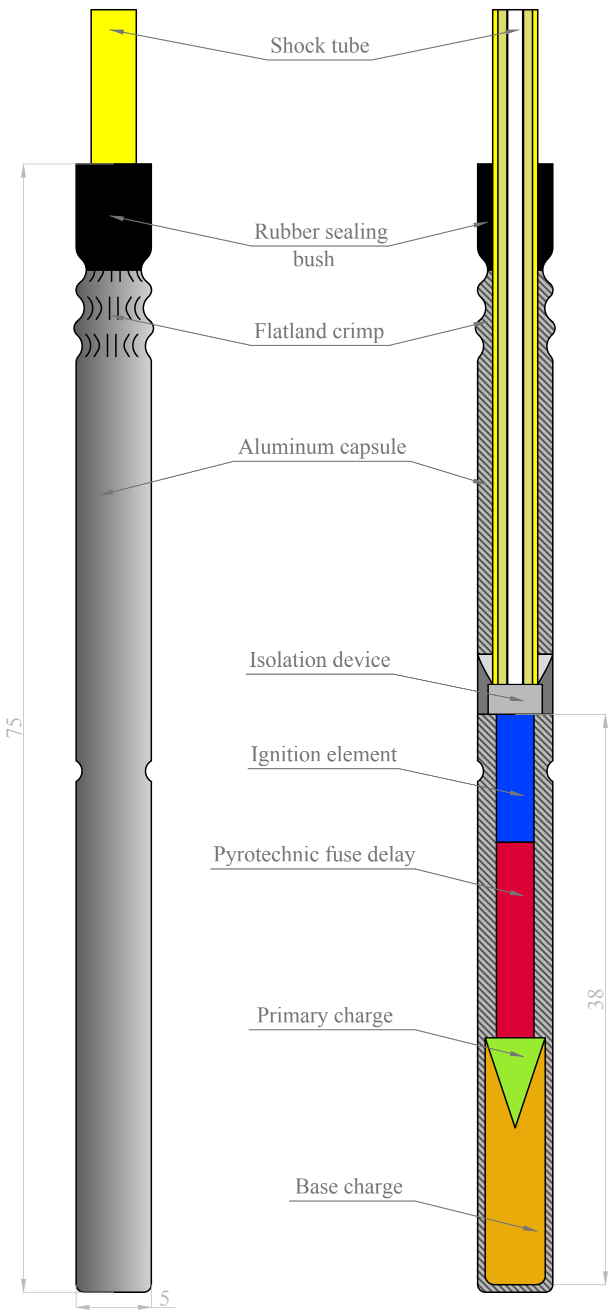
A.Y. 2018 - 2019

Detail of the junction booster - bottom hole detonator - surface connector (dual detonator system)
Scale: 1:1



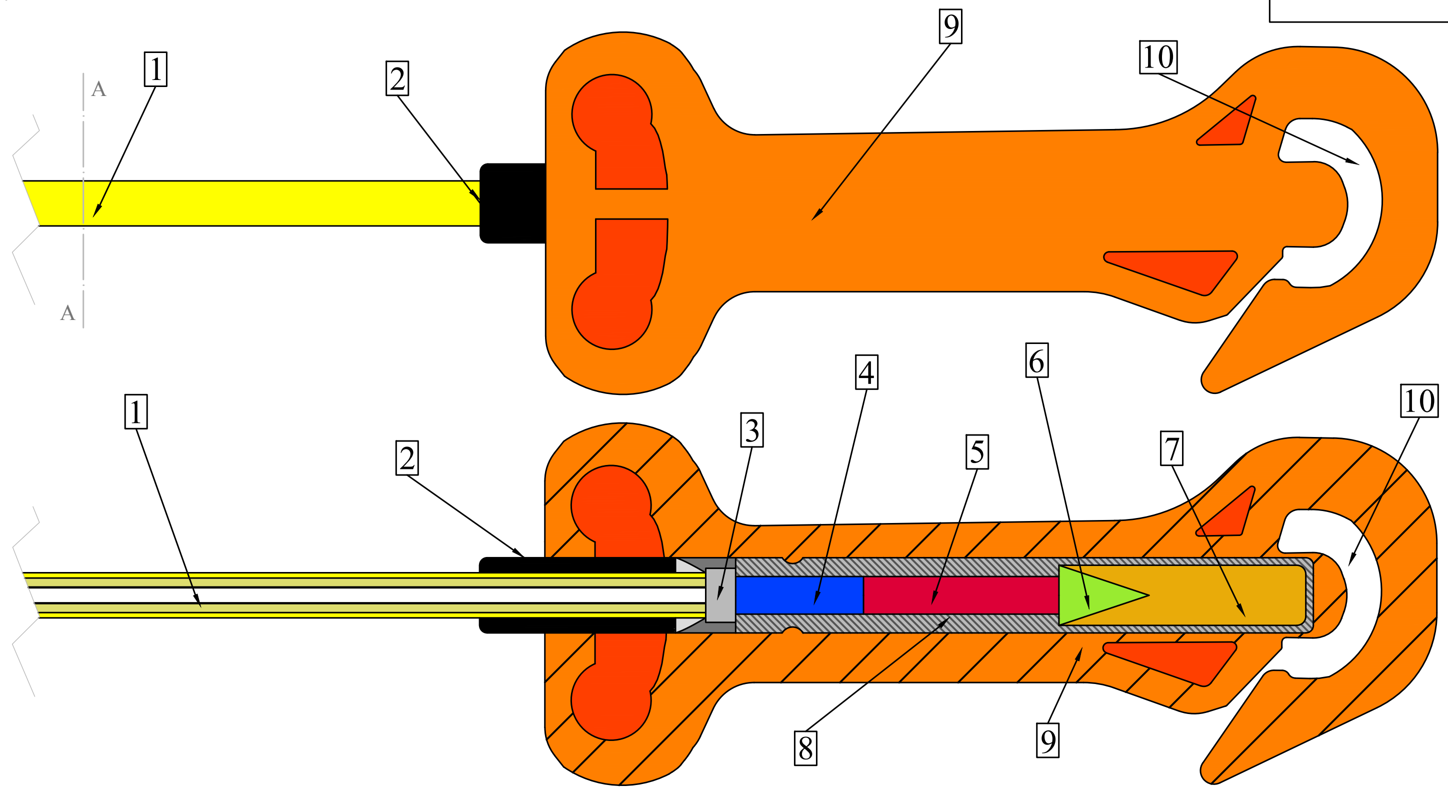
Detail of the nonel bottom hole detonator
Scale: 2:1

Enaex 1



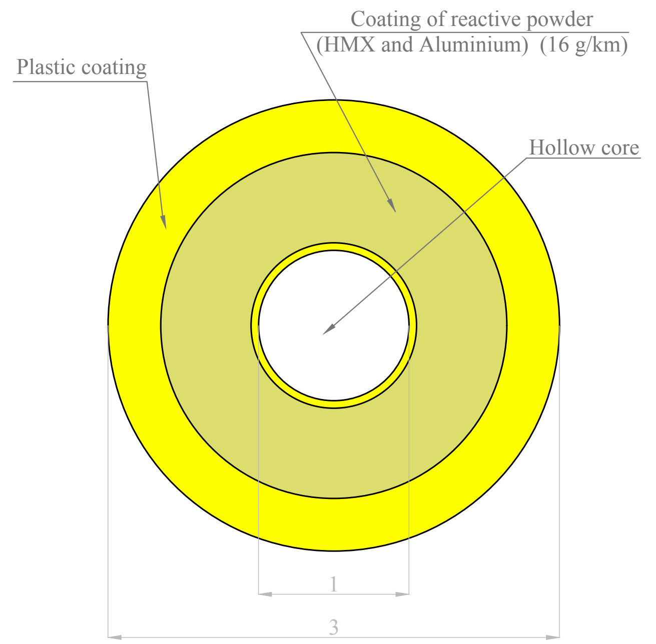
Detail of the surface connector
Scale: 2:1

Enaex 2



Section A - A of the shock tube
Scale: 20:1

Enaex 3



Number	Element indicated in the figure
1	Shock tube
2	Rubber sealing bush
3	Isolation device
4	Ignition element
5	Pyrotechnic fuse delay
6	Primary charge
7	Base charge
8	Aluminium capsule
9	Plastic surface of the connector
10	Space for interlocking of the nonel tubes

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
DETAIL OF THE NONEL EQUIPMENT AND USE

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

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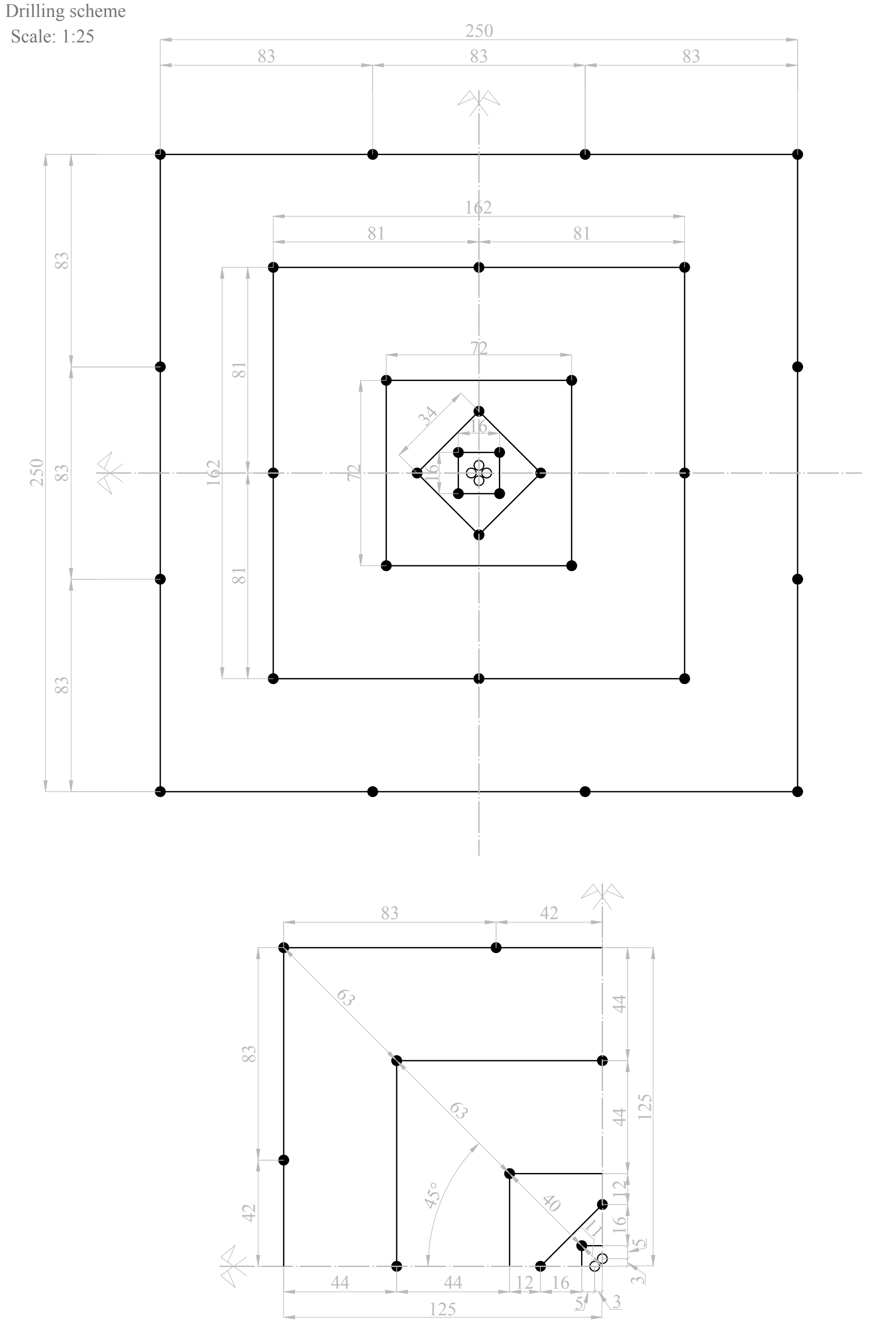
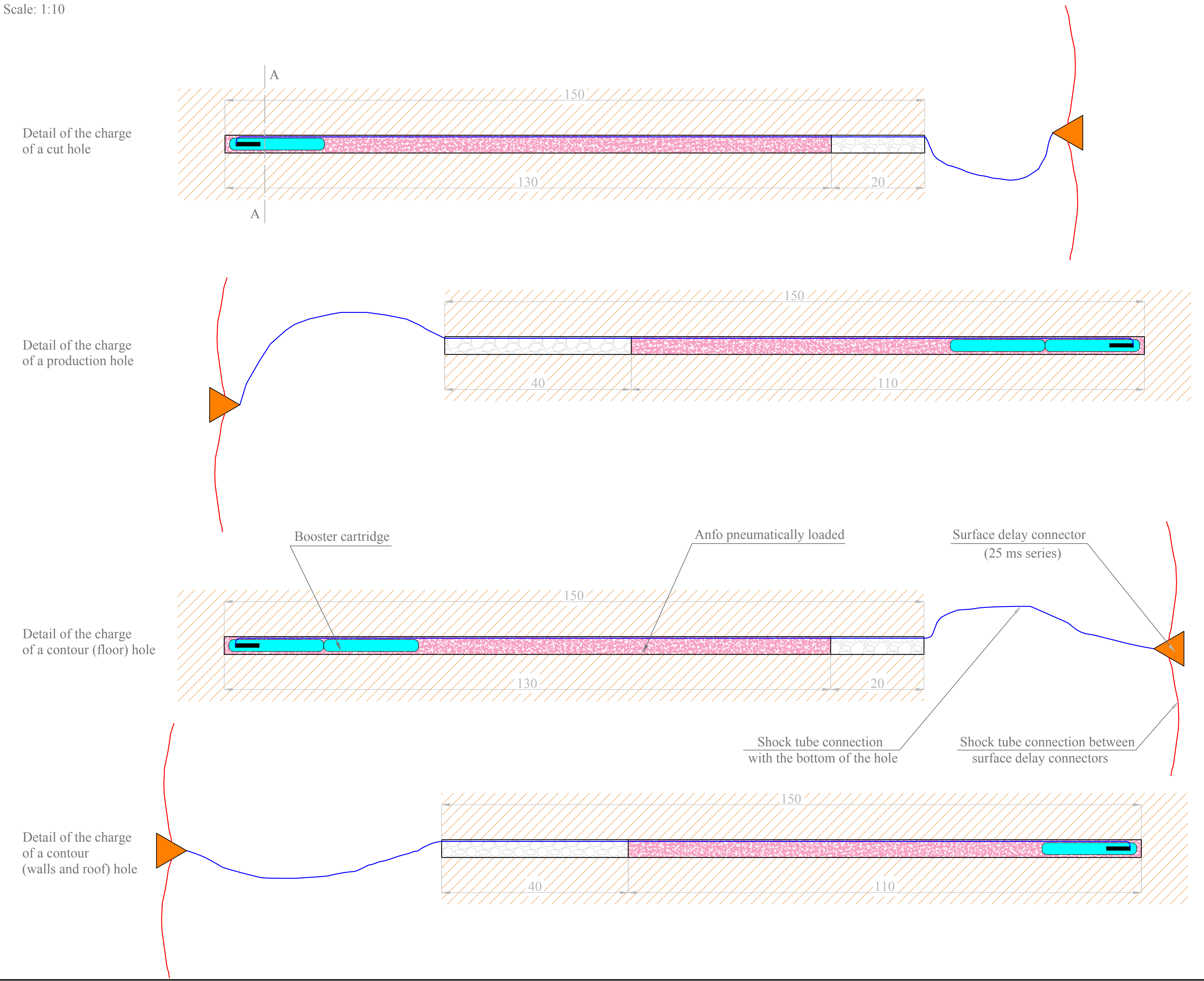
Drilling scheme
Scale: 1:25

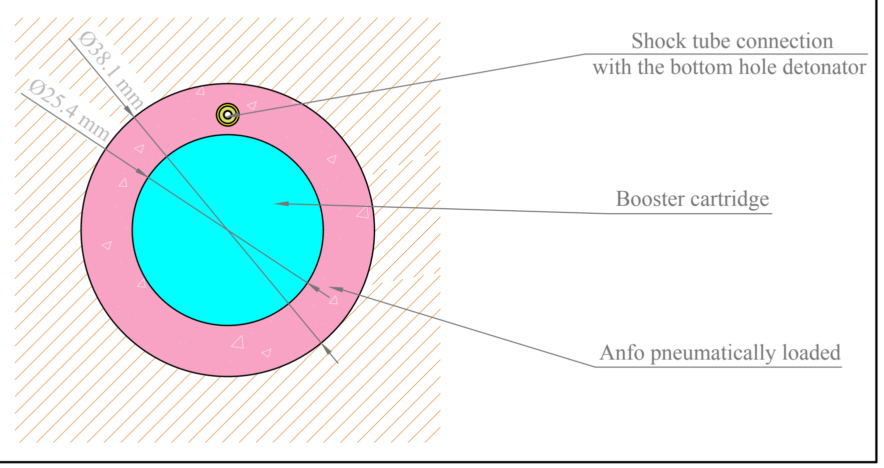


Detail of the charge
Scale: 1:10

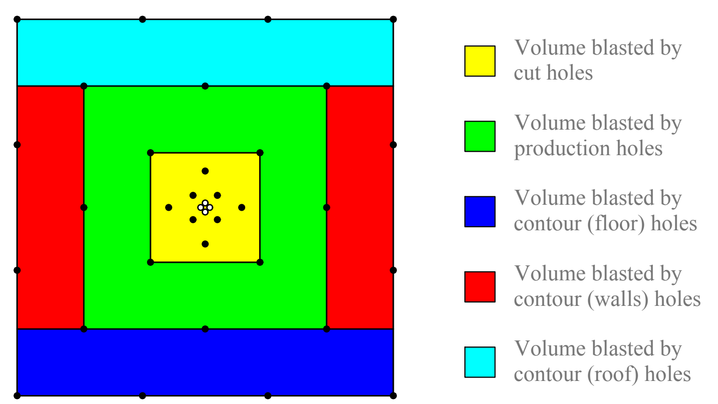


Cross sections of the charge
Scale: 1:1

SECTION A-A



Classification of the blasted volumes
Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: DRILLING AND CHARGE SCHEMES FOR SECTION 2.5 x 2.5 m

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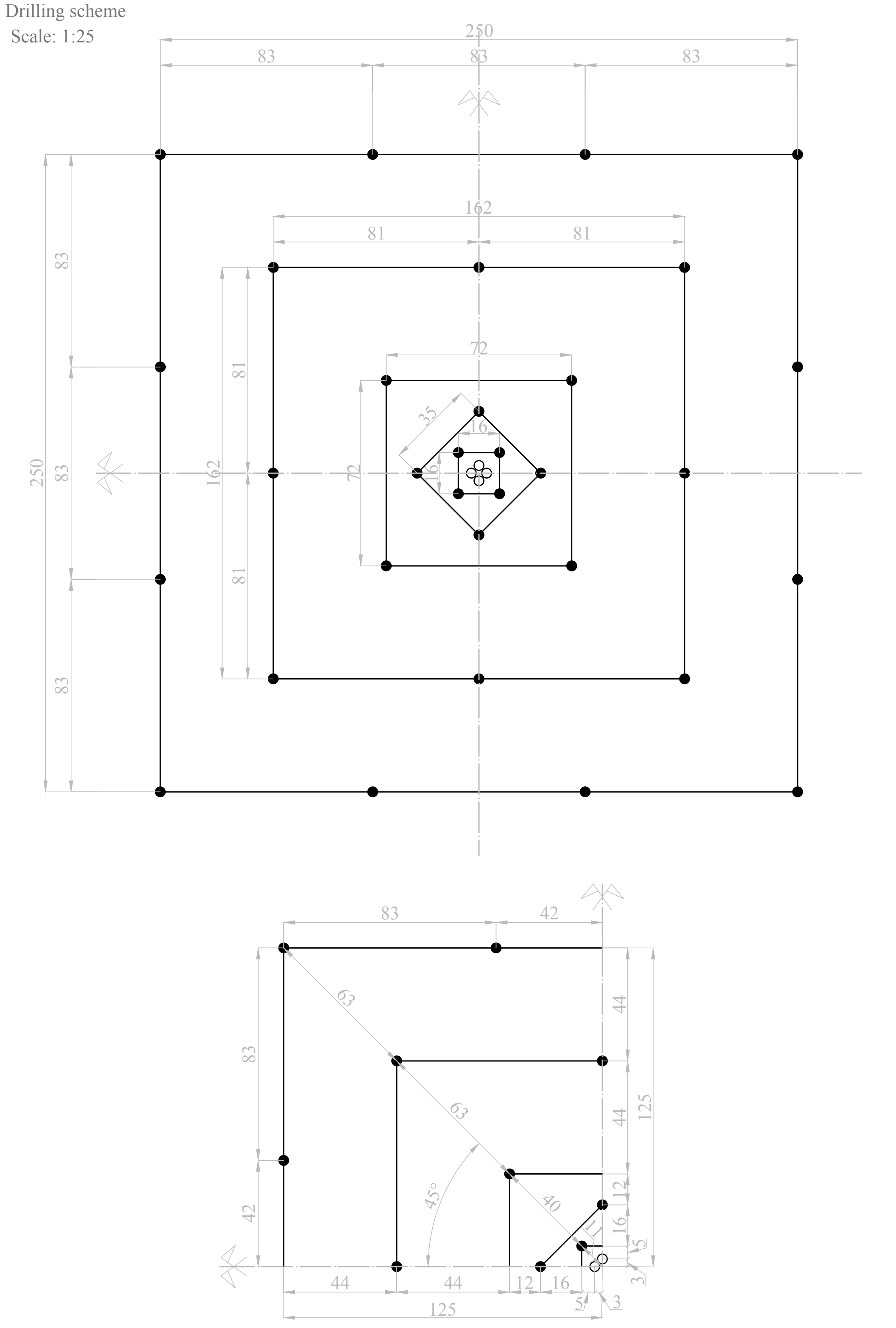
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

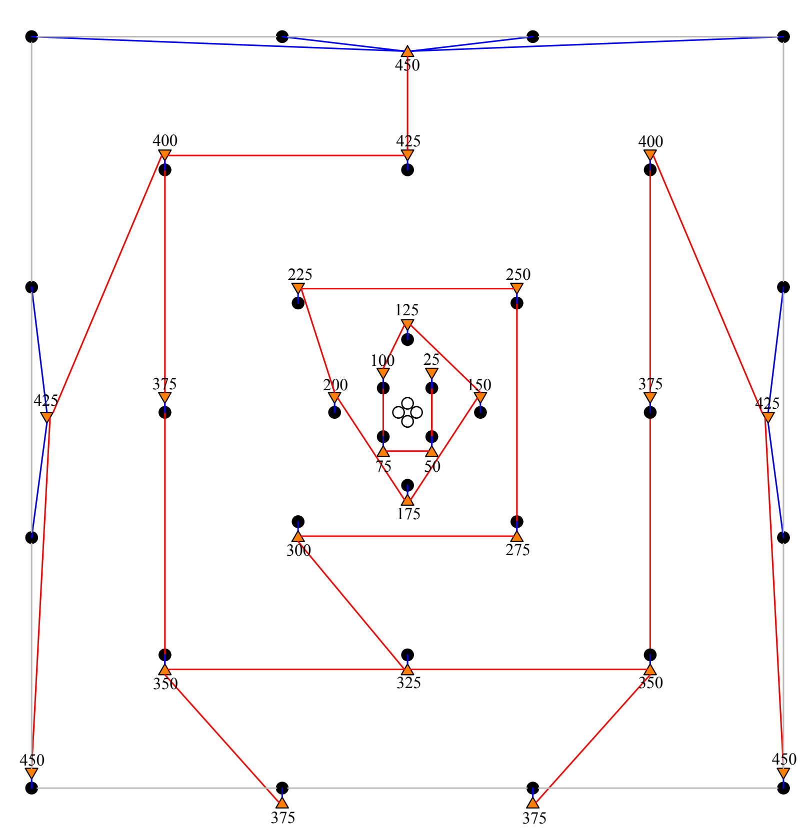
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

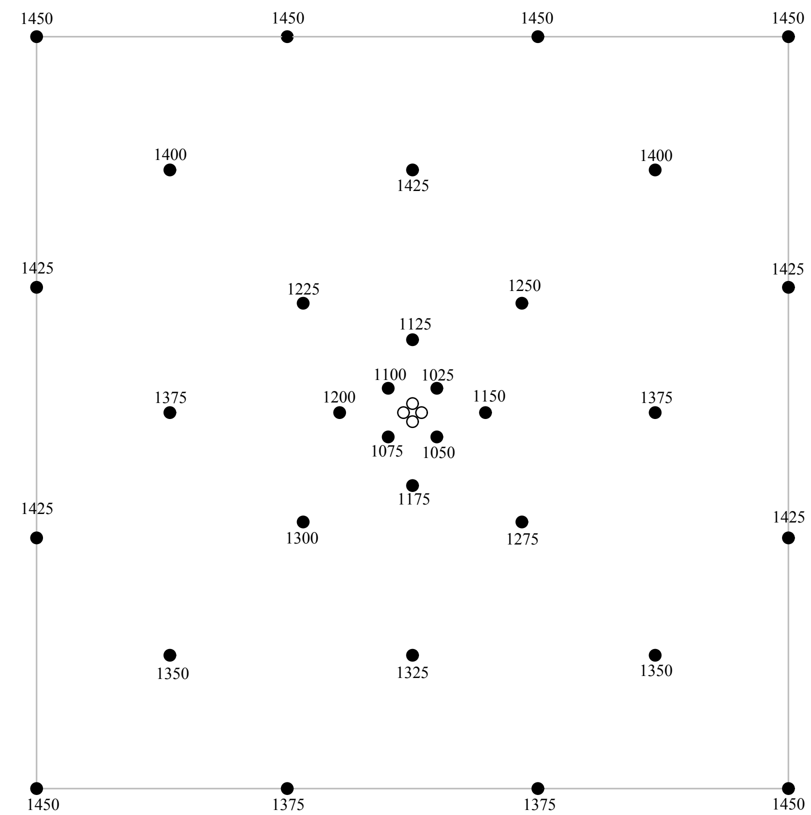
Drilling scheme
Scale: 1:25



Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
10 (1250 ms)	25 ms	1000 ms	1	1
11 (1275 ms)	25 ms	1000 ms	1	1
12 (1300 ms)	25 ms	1000 ms	1	1
13 (1325 ms)	25 ms	1000 ms	1	1
14 (1350 ms)	25 ms	1000 ms	2	2
15 (1375 ms)	25 ms	1000 ms	4	4
16 (1400 ms)	25 ms	1000 ms	2	2
17 (1425 ms)	25 ms	1000 ms	3	5
18 (1450 ms)	25 ms	1000 ms	3	6

Classification of the blasted volumes
Scale: 1:50

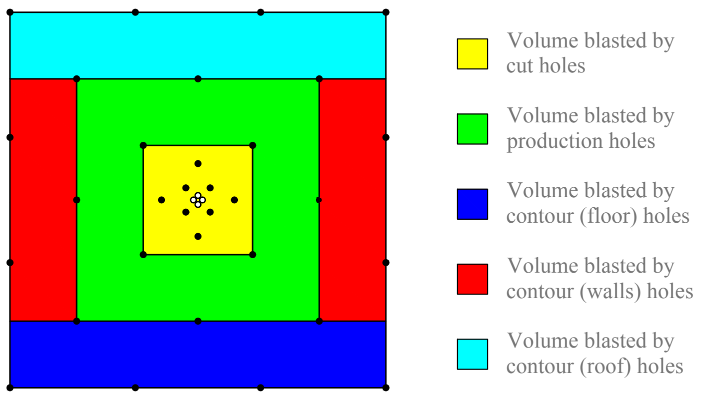
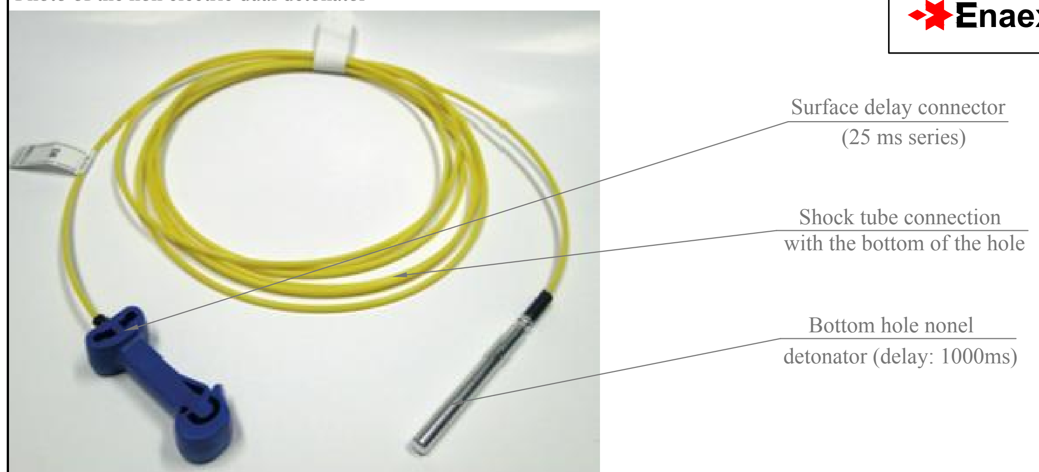


Photo of the non electric dual detonator

Enaex



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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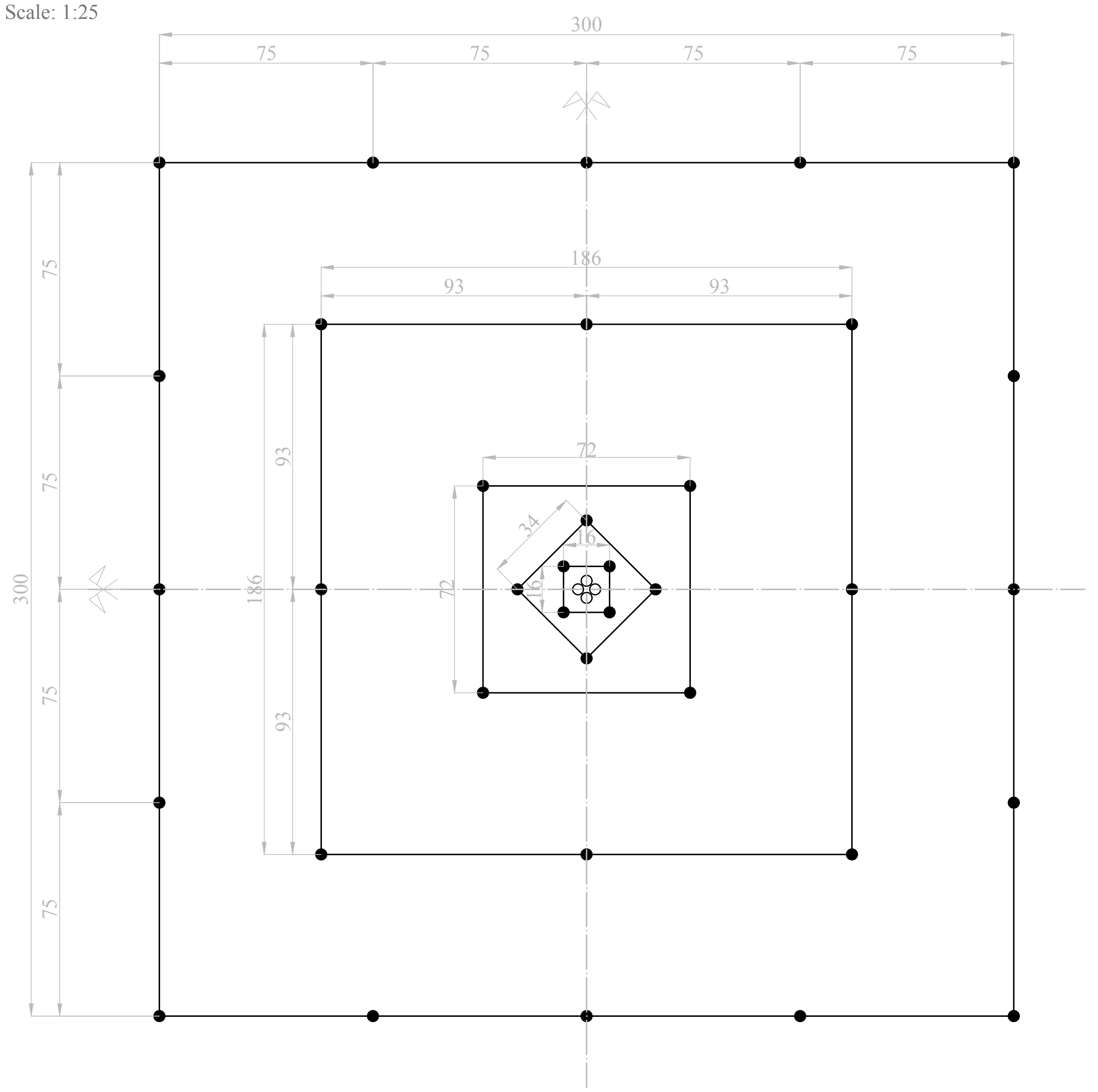
Technical drawing:
PHASE 1: TRIGGER TIMES FOR SECTION 2.5 x 2.5 m
Scale: 1:25

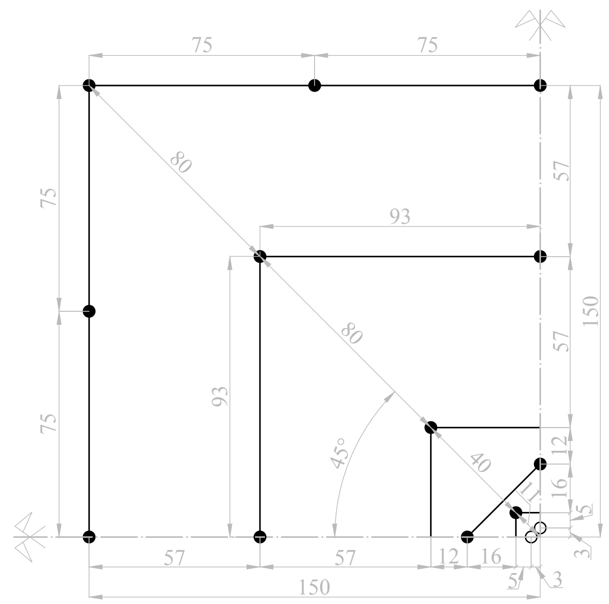
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu
A.Y. 2018 - 2019

Drilling scheme

Scale: 1:25





Detail of the charge

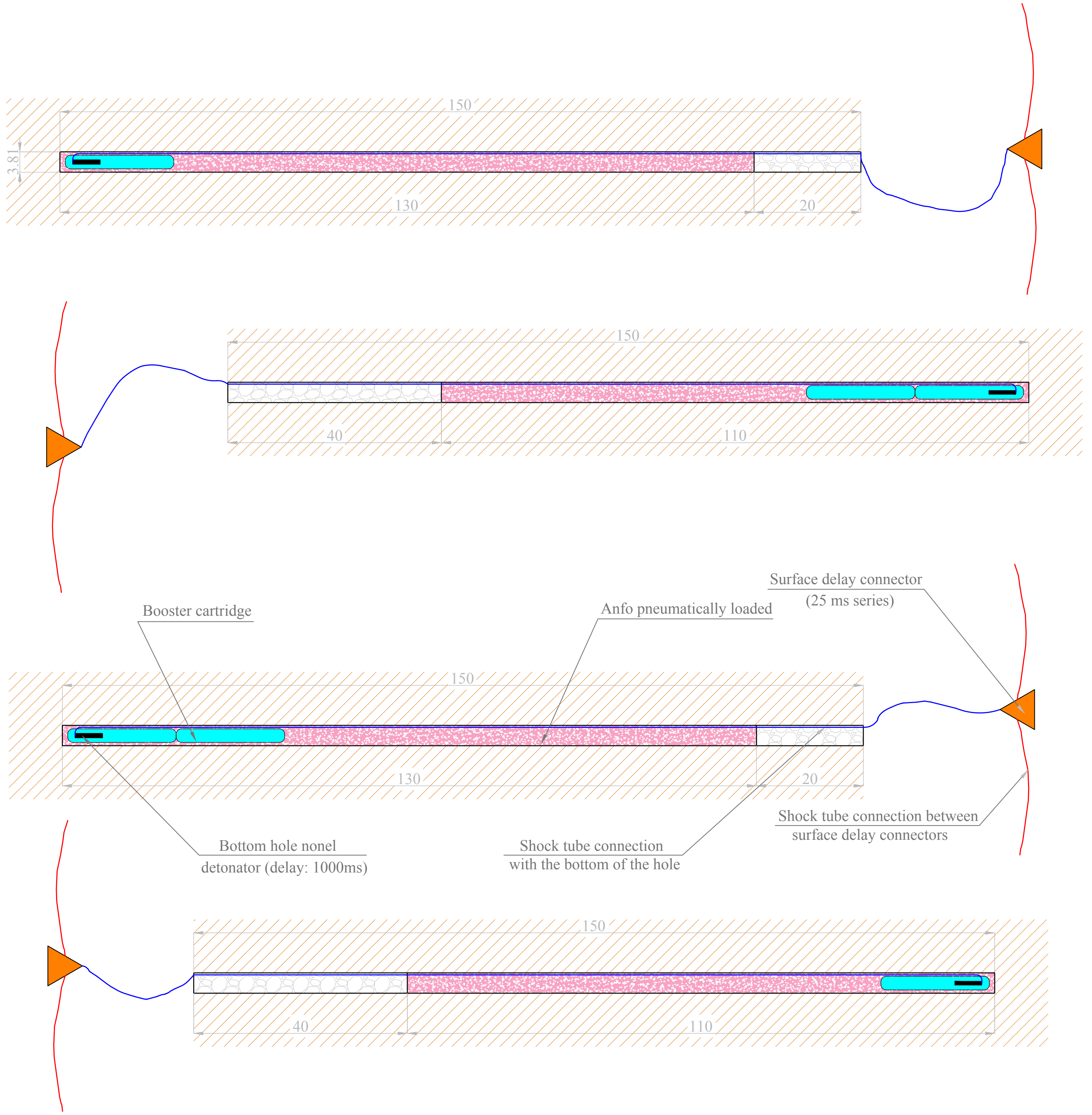
Scale: 1:10

Detail of the charge of a cut hole

Detail of the charge of a production hole

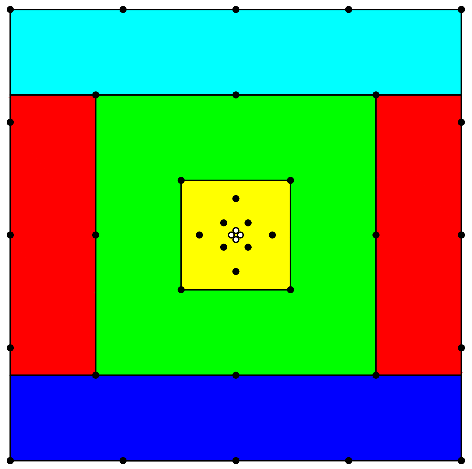
Detail of the charge of a contour (floor) hole

Detail of the charge of a contour (walls and roof) hole



Classification of the blasted volumes

Scale: 1:50



Volume blasted by cut holes

Volume blasted by production holes

Volume blasted by contour (floor) holes

Volume blasted by contour (walls) holes

Volume blasted by contour (roof) holes

Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:

Simone BEZ

Badge Number: 236349

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Technical drawing:

PHASE 1: DRILLING AND CHARGE SCHEMES FOR SECTION 3.0 x 3.0 m

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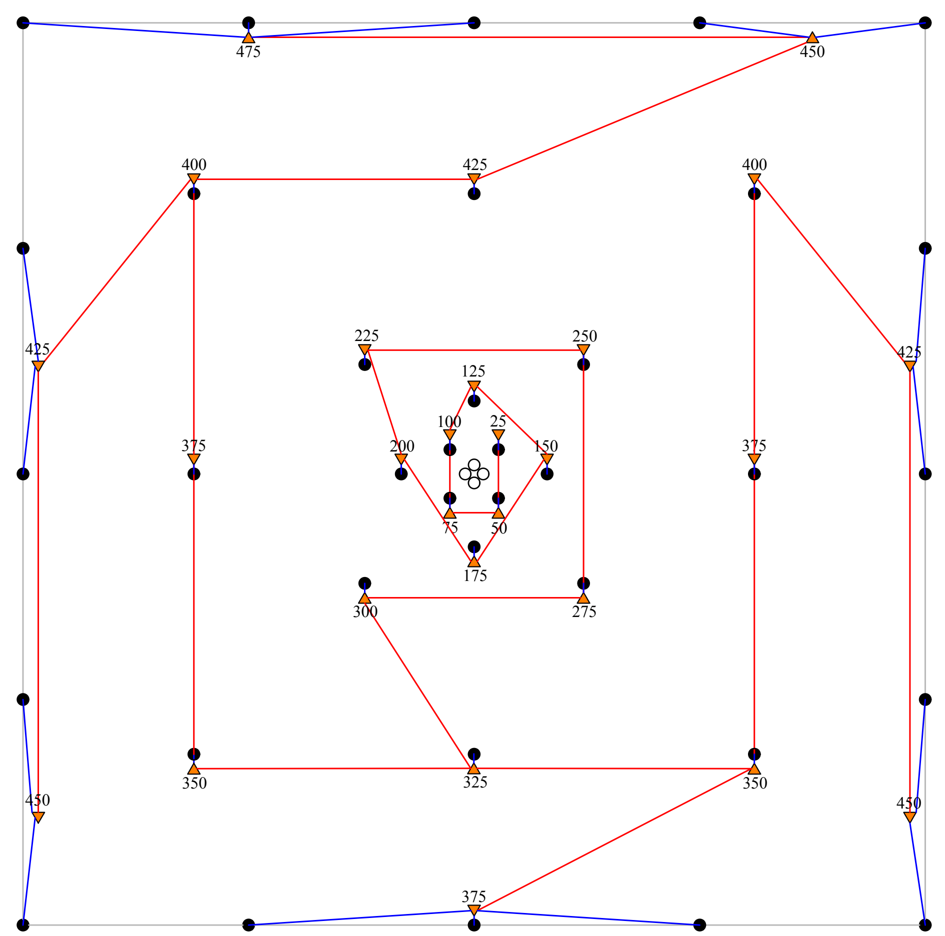
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:

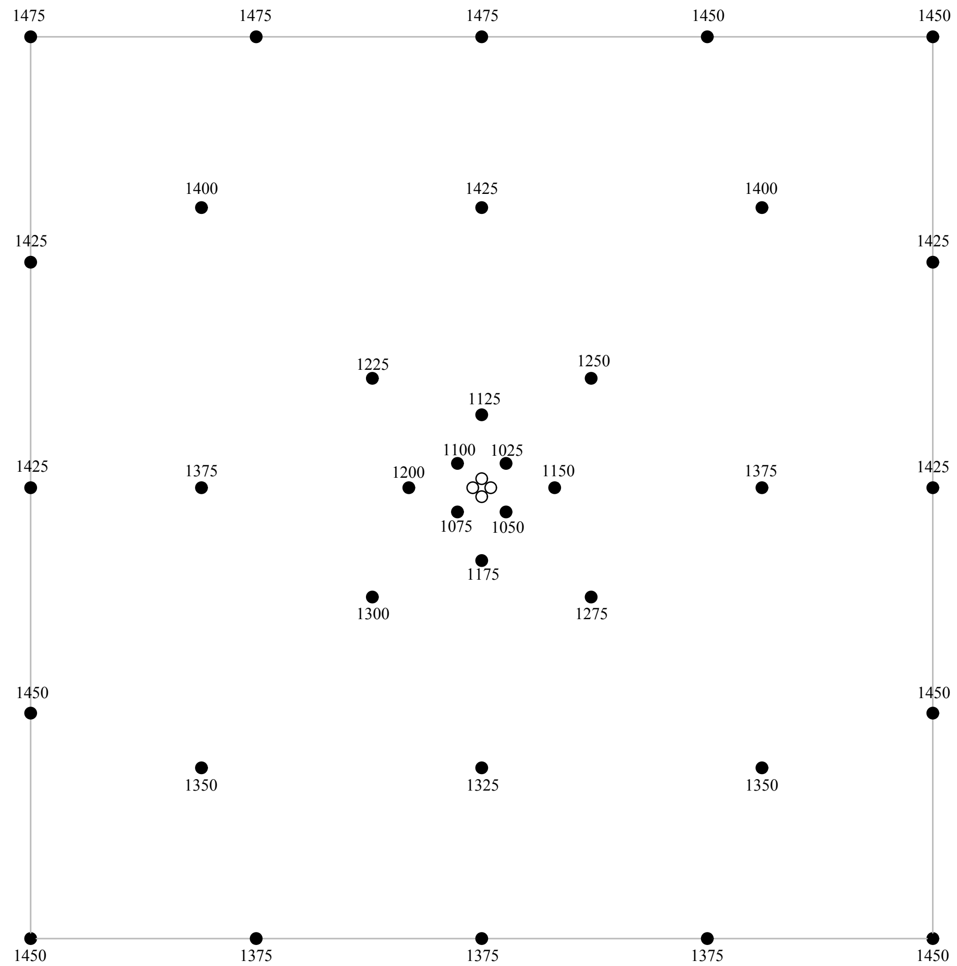
Prof. Marilena Cardu

A.Y. 2018 - 2019

Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
11 (1275 ms)	25 ms	1000 ms	1	1
12 (1300 ms)	25 ms	1000 ms	1	1
13 (1325 ms)	25 ms	1000 ms	1	1
14 (1350 ms)	25 ms	1000 ms	2	2
15 (1375 ms)	25 ms	1000 ms	3	5
16 (1400 ms)	25 ms	1000 ms	2	2
17 (1425 ms)	25 ms	1000 ms	3	5
18 (1450 ms)	25 ms	1000 ms	3	6
19 (1475 ms)	25 ms	1000 ms	1	3

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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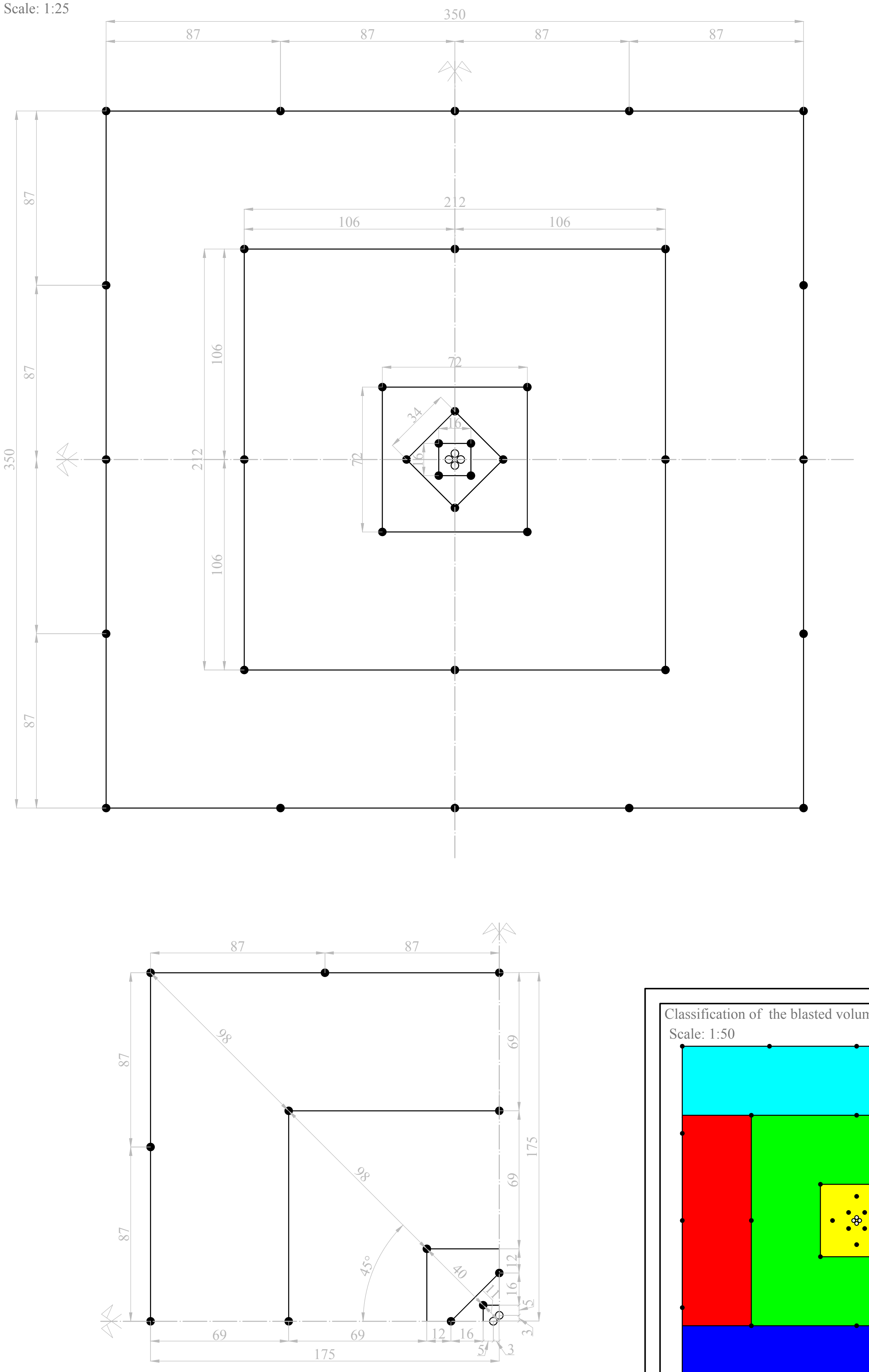
Technical drawing:
PHASE 1: TRIGGER TIMES FOR SECTION 3.0 x 3.0 m
Scale: 1:25

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu
A.Y. 2018 - 2019

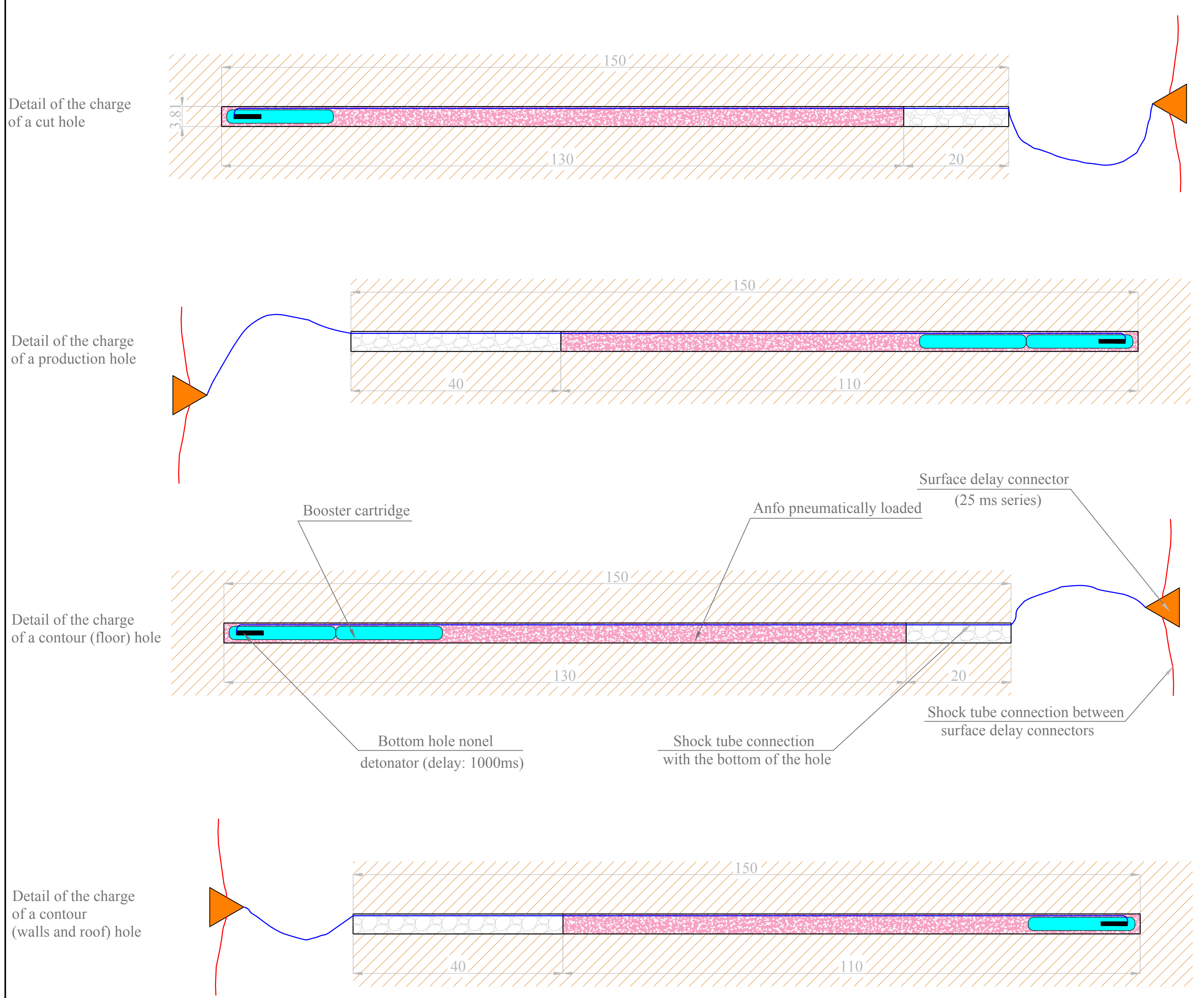
Drilling scheme

Scale: 1:25



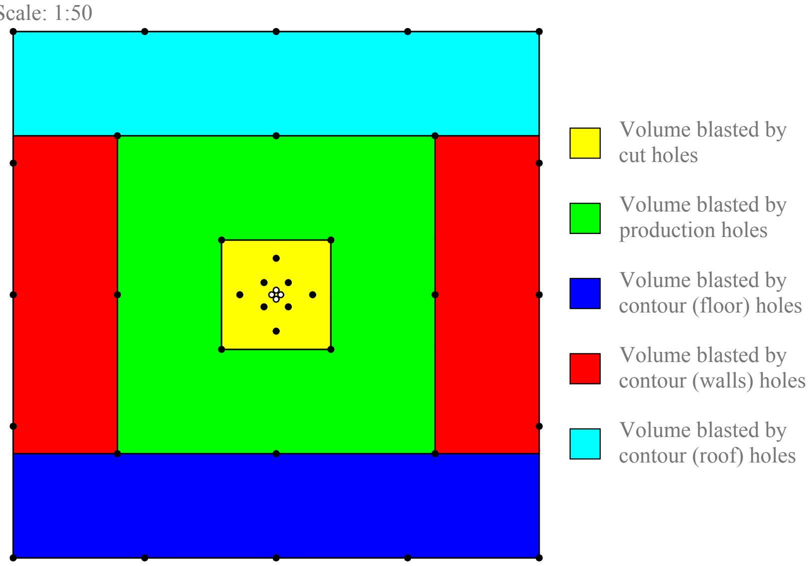
Detail of the charge

Scale: 1:10



Classification of the blasted volumes

Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: DRILLING AND CHARGE SCHEMES FOR SECTION 3.5 x 3.5 m

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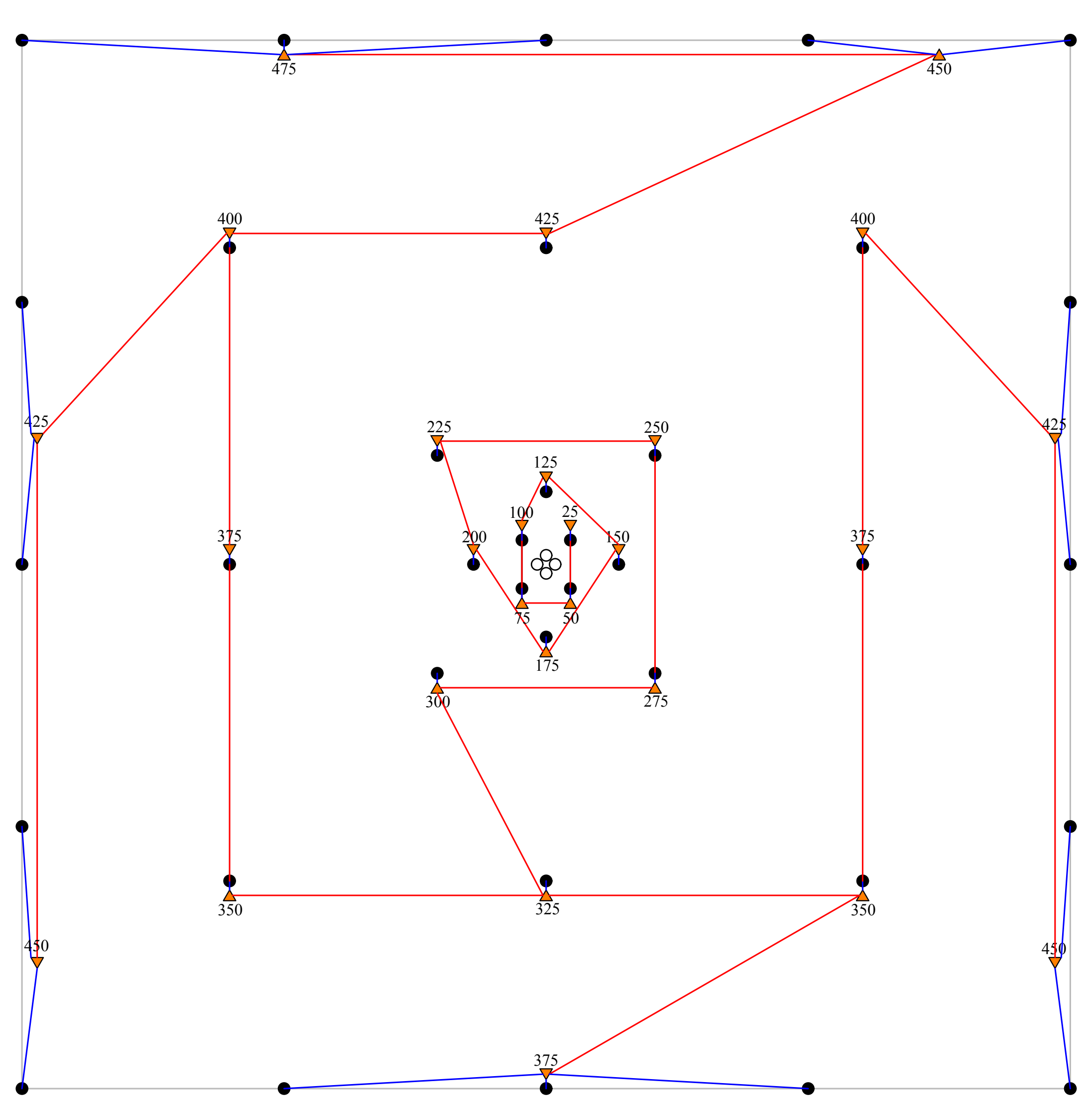
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

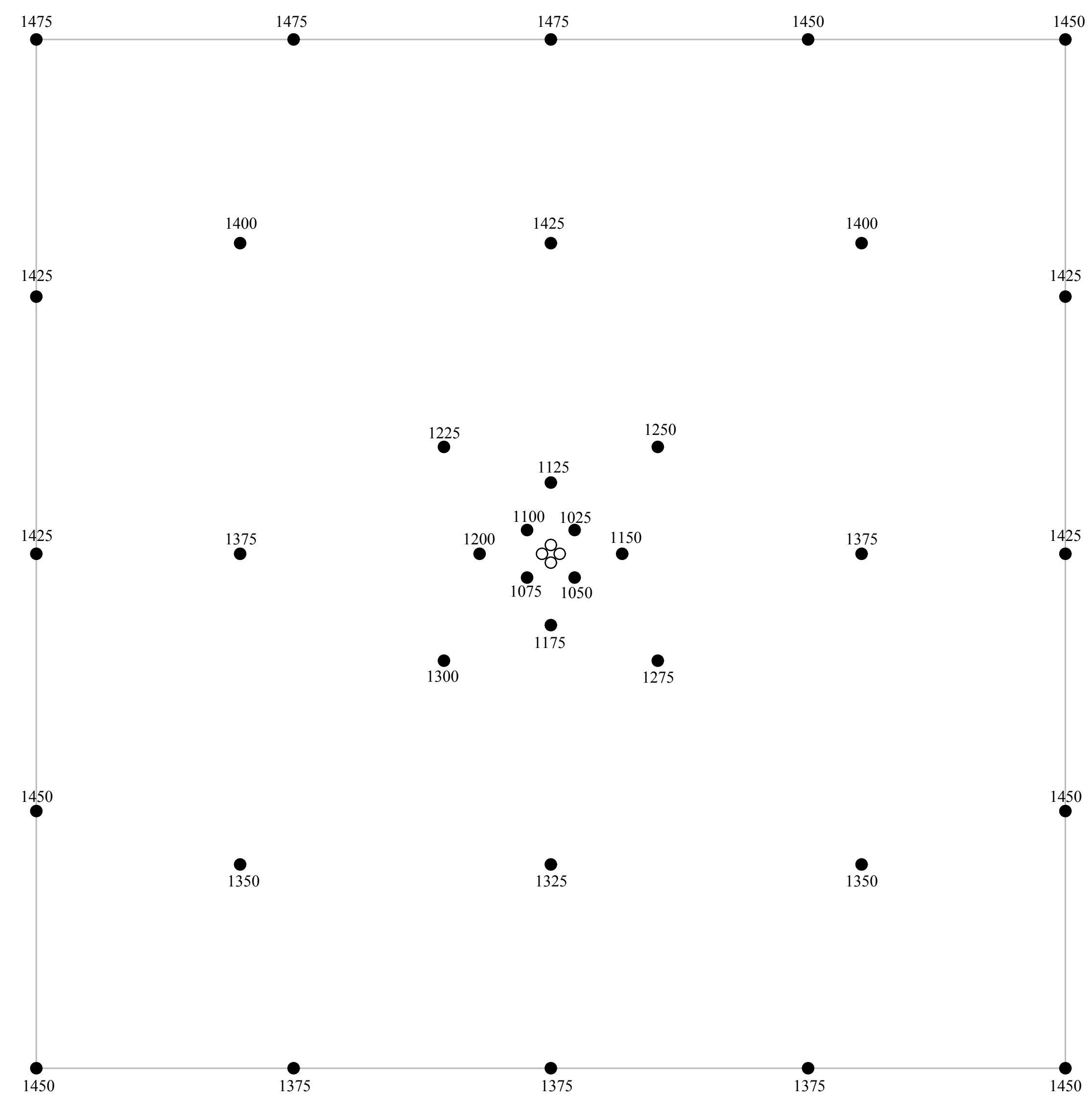
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
11 (1275 ms)	25 ms	1000 ms	1	1
12 (1300 ms)	25 ms	1000 ms	1	1
13 (1325 ms)	25 ms	1000 ms	1	1
14 (1350 ms)	25 ms	1000 ms	2	2
15 (1375 ms)	25 ms	1000 ms	3	5
16 (1400 ms)	25 ms	1000 ms	2	2
17 (1425 ms)	25 ms	1000 ms	3	5
18 (1450 ms)	25 ms	1000 ms	3	6
19 (1475 ms)	25 ms	1000 ms	1	3

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: TRIGGER TIMES FOR SECTION 3.5 x 3.5 m

Scale: 1:25

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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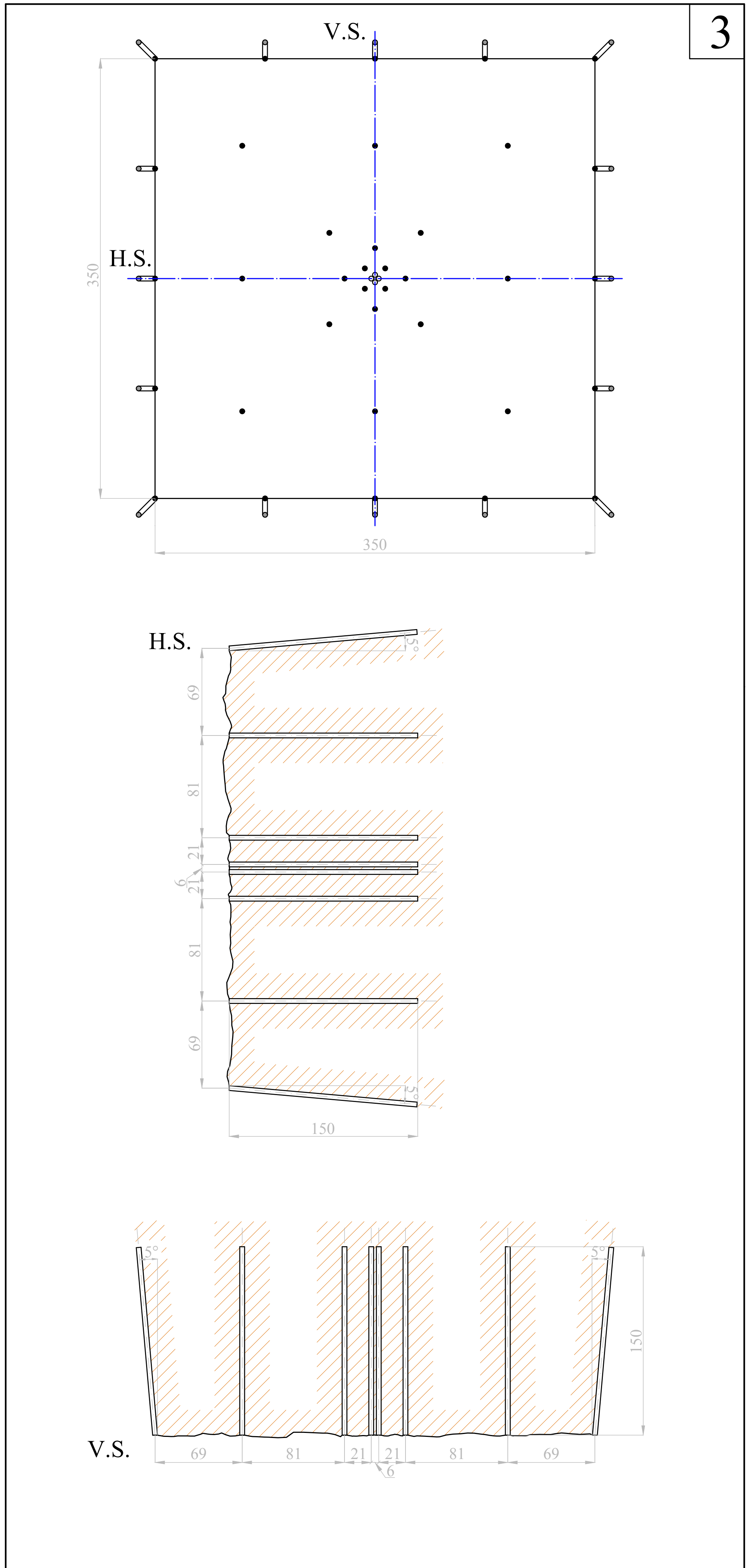
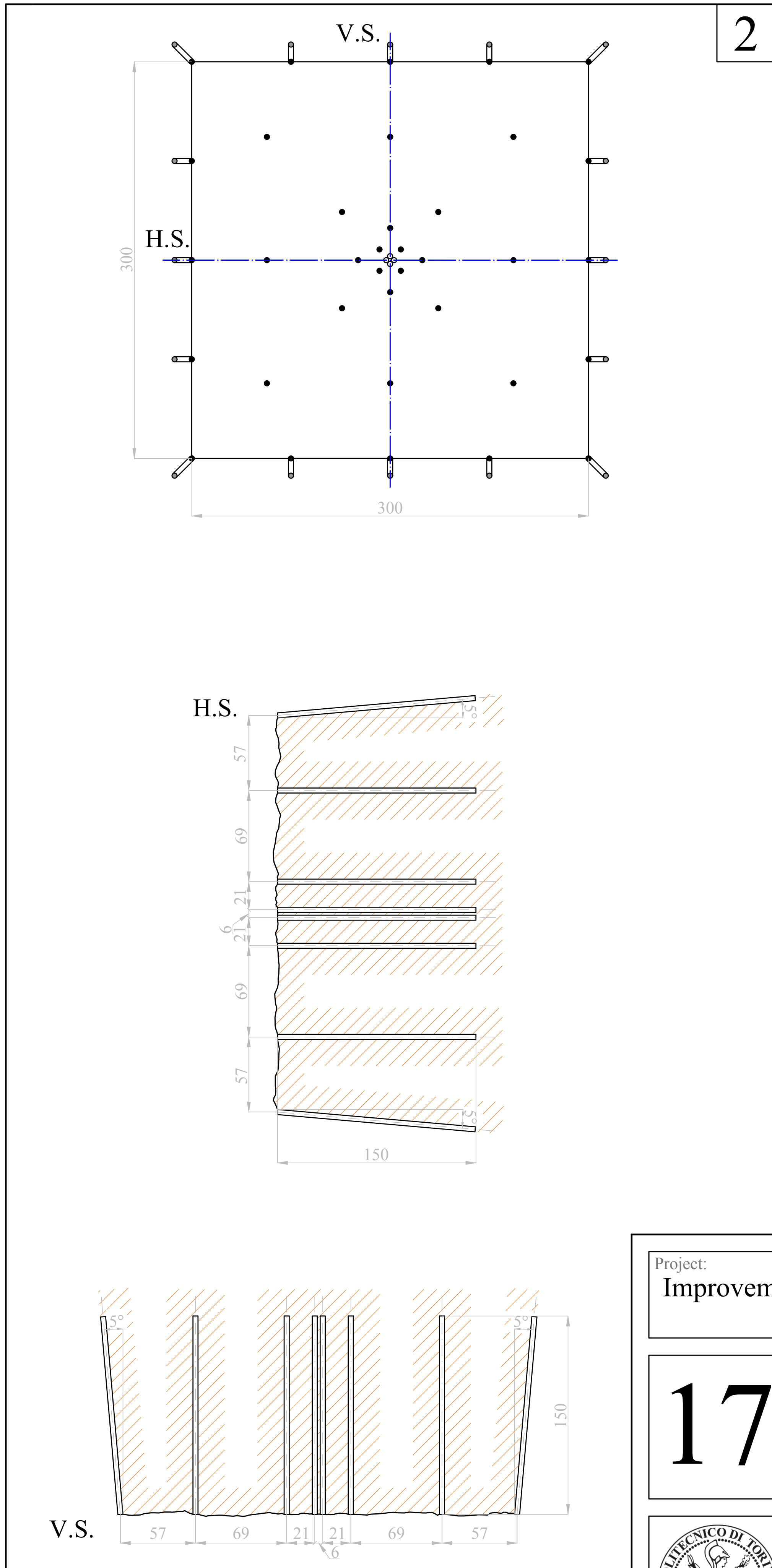
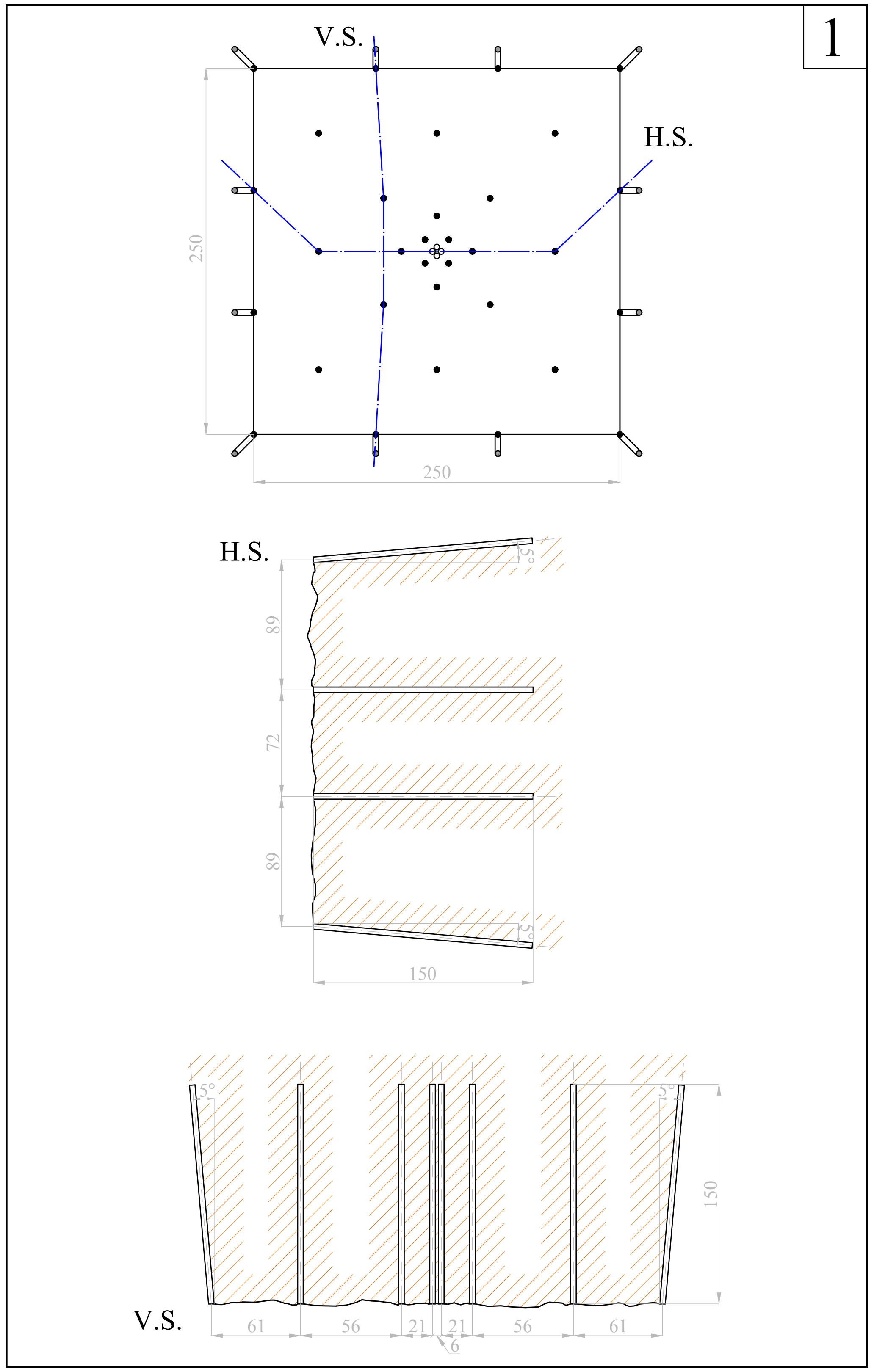
A.Y. 2018 - 2019







Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:

PHASE 1: SECTIONS AND DEFINITION OF THE LOOK-OUT ANGLE

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

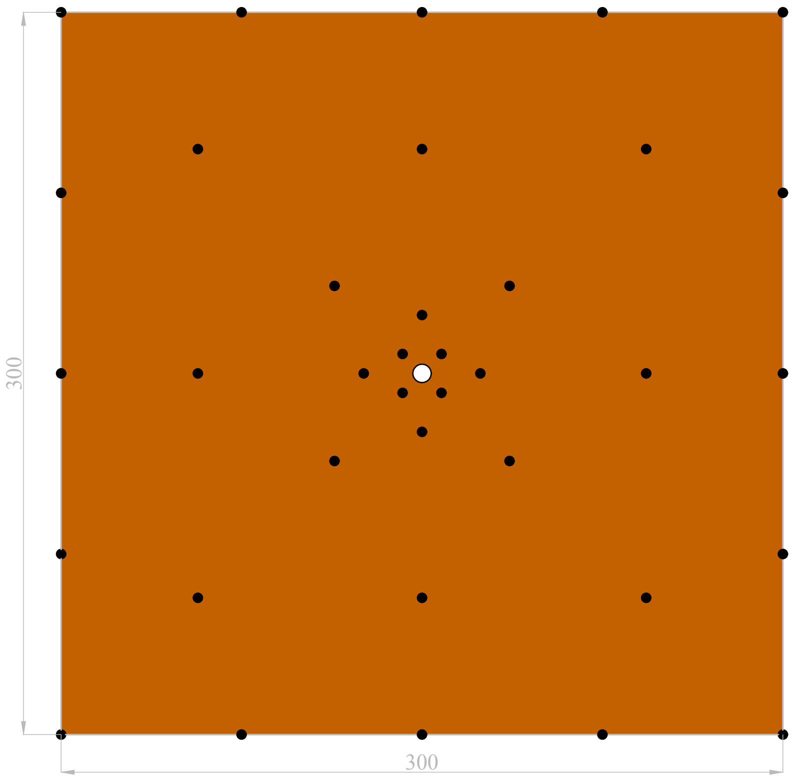
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

1

Instant of the detonation sequence at 1000 ms

Empty central hole volume (considering the fictitious diameter): 0.007 m^3
Rock volume blasted at this stage: 0 m^3
Rock volume blasted until now: 0 m^3

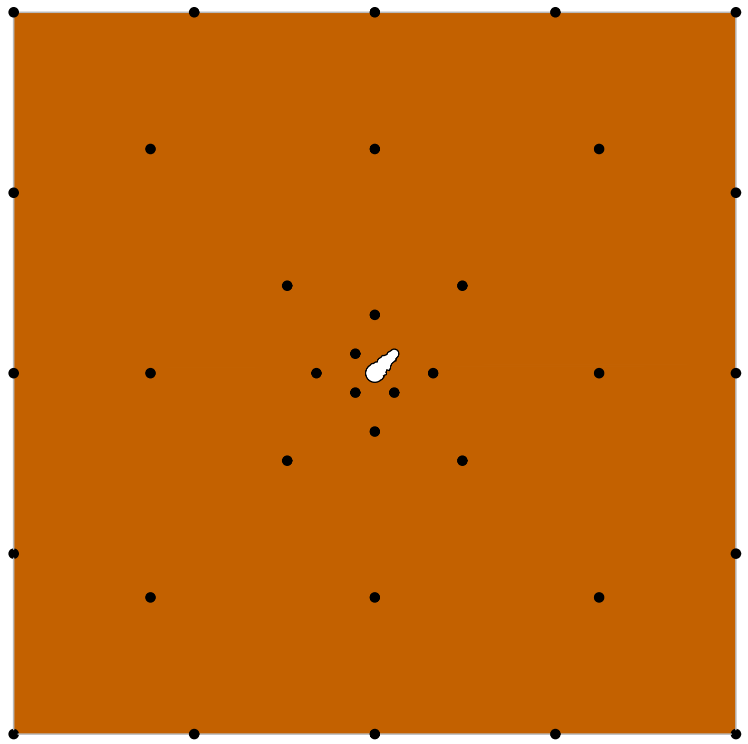


Charge per delay: 0 kg

2

Instant of the detonation sequence at 1025 ms

Rock volume blasted at this stage: 0.007 m^3
Rock volume blasted until now: 0.007 m^3

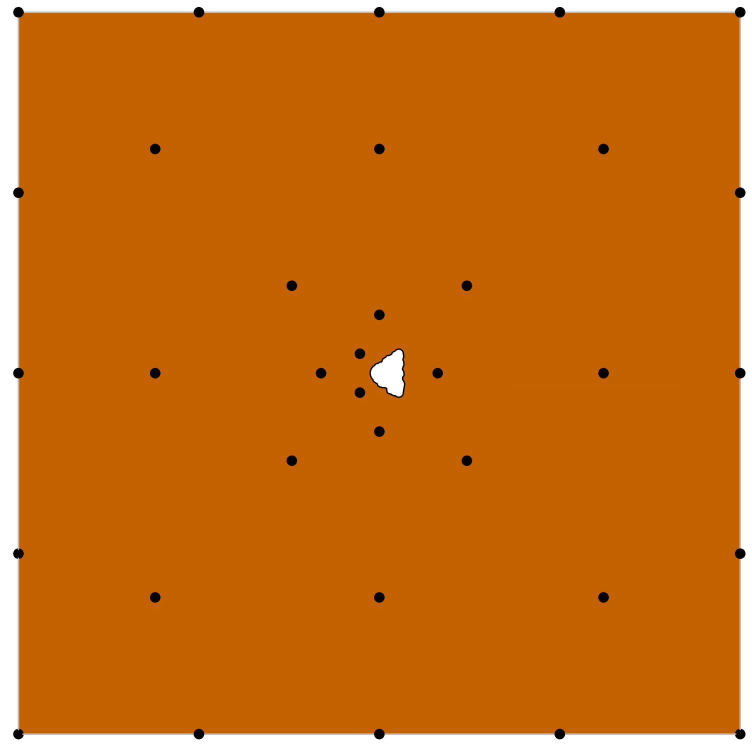


Charge per delay: 1.181 kg

3

Instant of the detonation sequence at 1050 ms

Rock volume blasted at this stage: 0.014 m^3
Rock volume blasted until now: 0.021 m^3

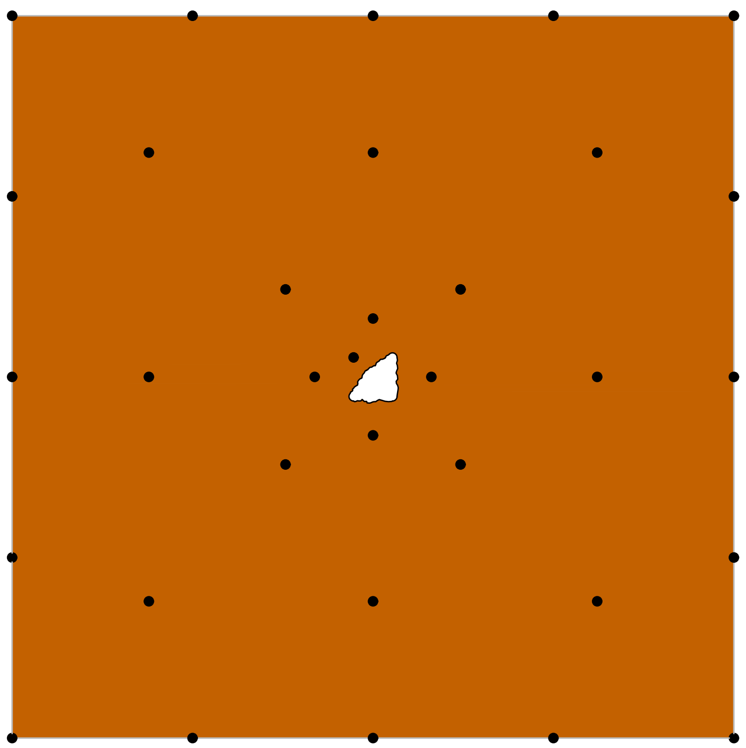


Charge per delay: 1.181 kg

4

Instant of the detonation sequence at 1075 ms

Rock volume blasted at this stage: 0.014 m^3
Rock volume blasted until now: 0.035 m^3

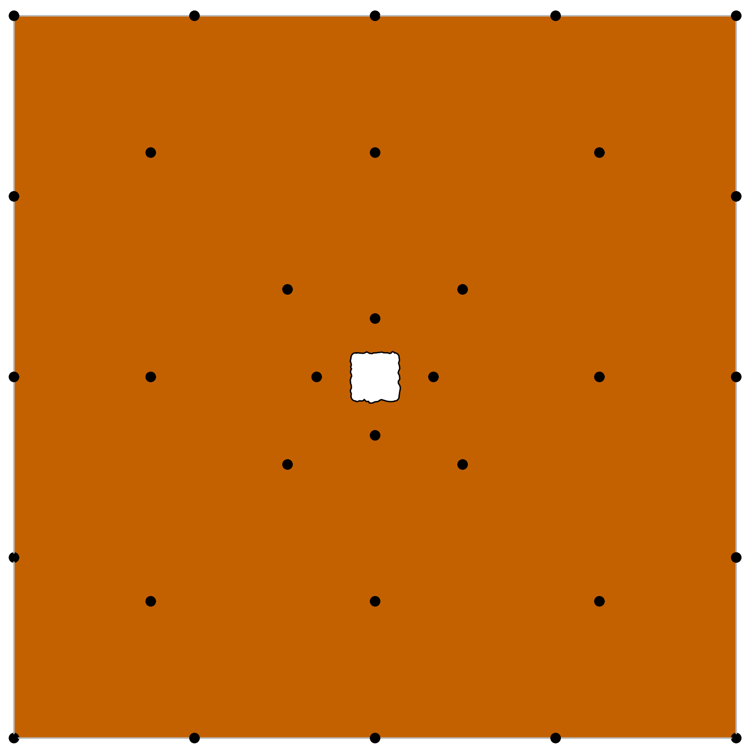


Charge per delay: 1.181 kg

5

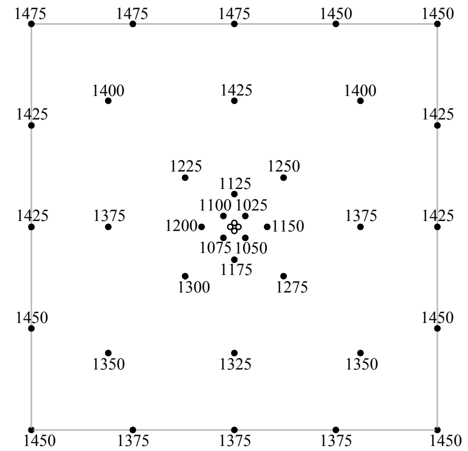
Instant of the detonation sequence at 1100 ms

Rock volume blasted at this stage: 0.018 m^3
Rock volume blasted until now: 0.053 m^3



Charge per delay: 1.181 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (1)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

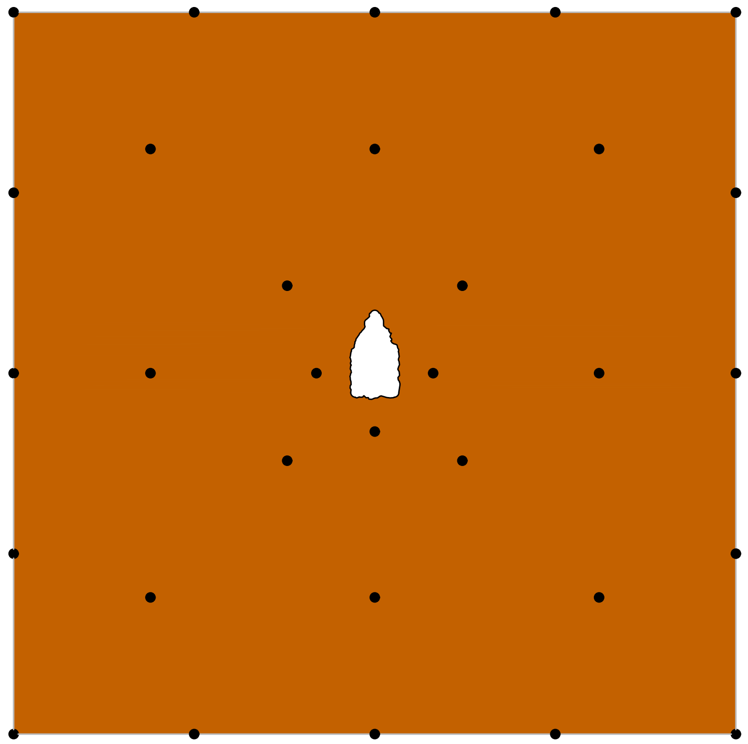
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Instant of the detonation sequence at 1125 ms

6

Rock volume blasted at this stage: 0.023 m^3
Rock volume blasted until now: 0.076 m^3

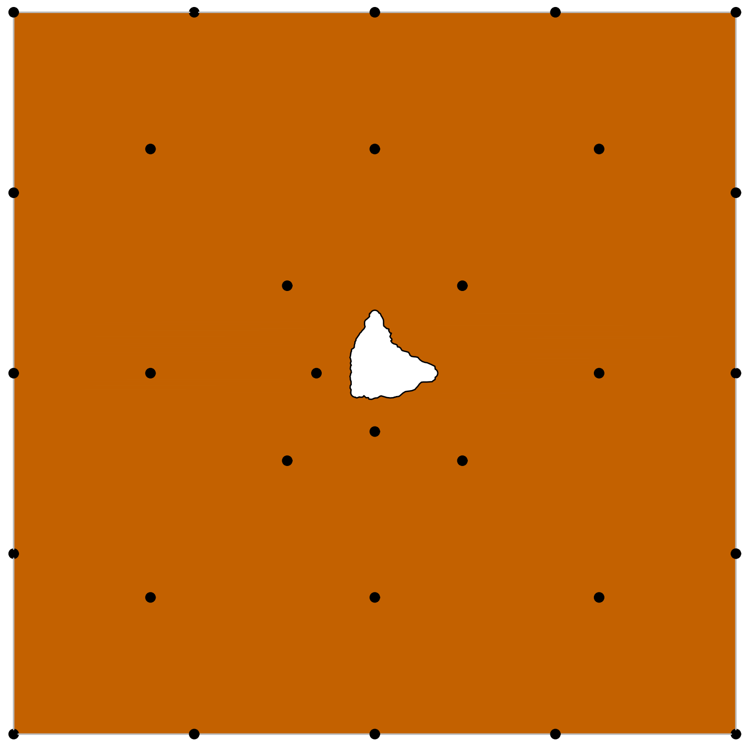


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1150 ms

7

Rock volume blasted at this stage: 0.031 m^3
Rock volume blasted until now: 0.107 m^3

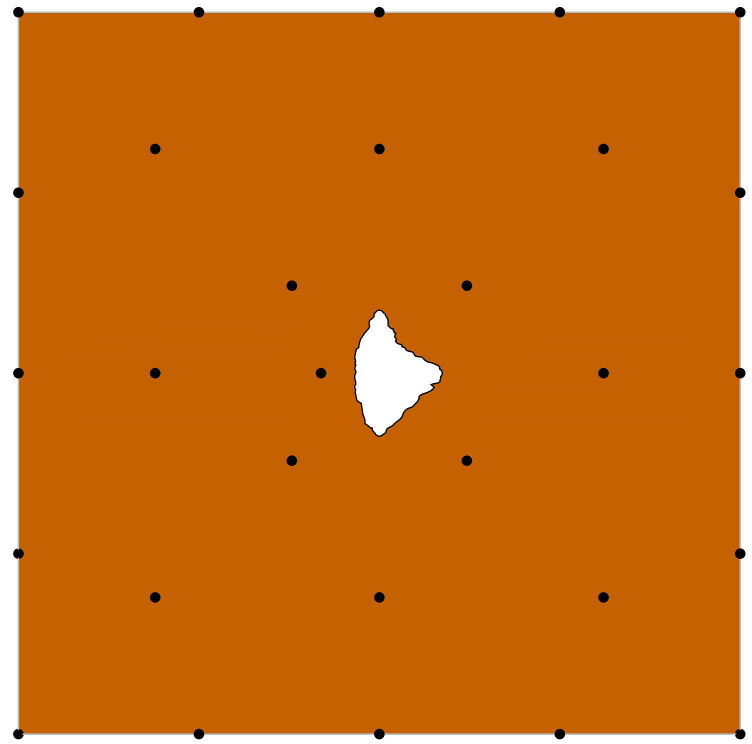


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1175 ms

8

Rock volume blasted at this stage: 0.042 m^3
Rock volume blasted until now: 0.149 m^3

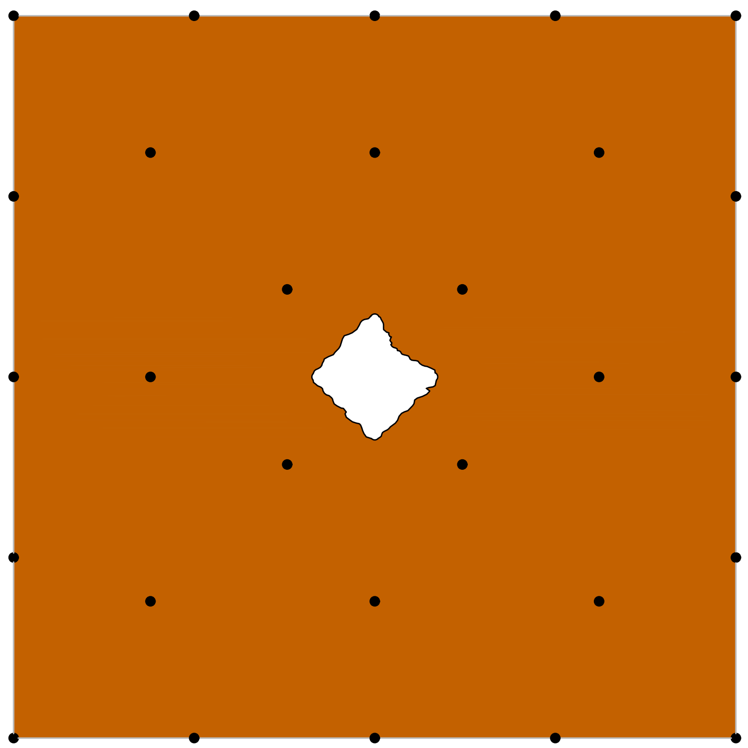


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1200 ms

9

Rock volume blasted at this stage: 0.055 m^3
Rock volume blasted until now: 0.204 m^3

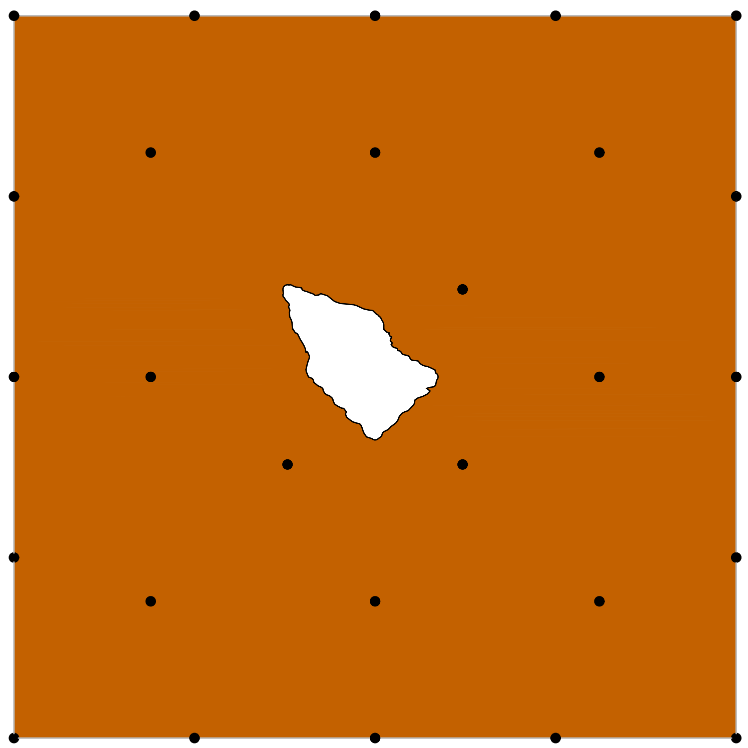


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1225 ms

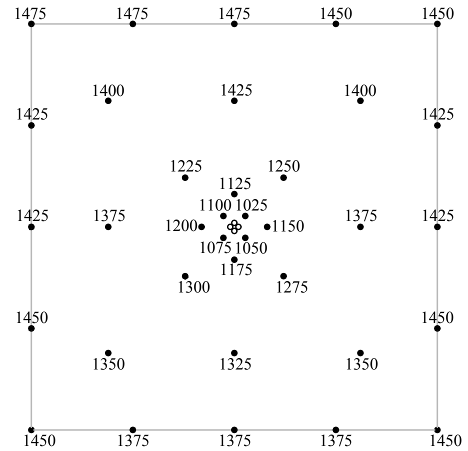
10

Rock volume blasted at this stage: 0.099 m^3
Rock volume blasted until now: 0.303 m^3



Charge per delay: 1.181 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (2)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

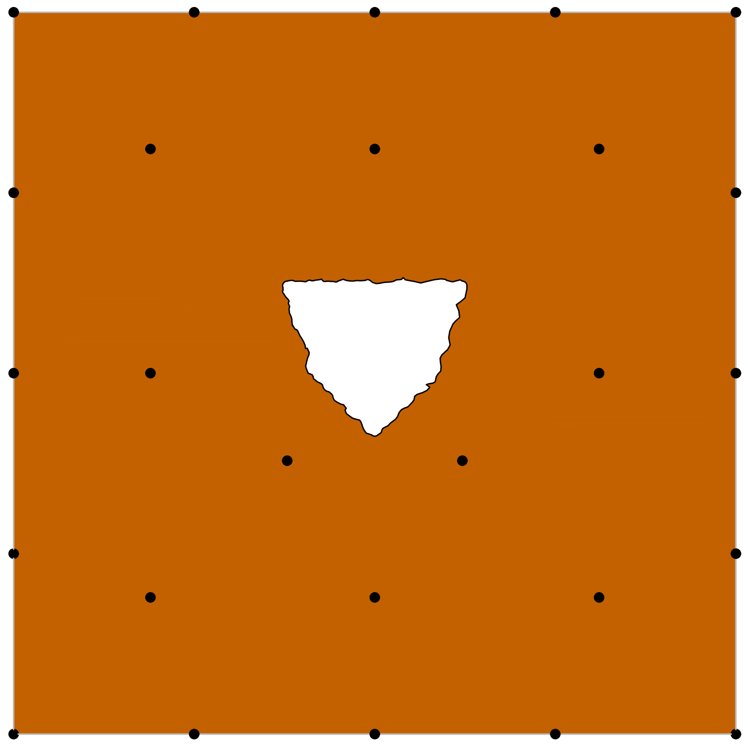
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Instant of the detonation sequence at 1250 ms

11

Rock volume blasted at this stage: 0.177 m^3
Rock volume blasted until now: 0.480 m^3

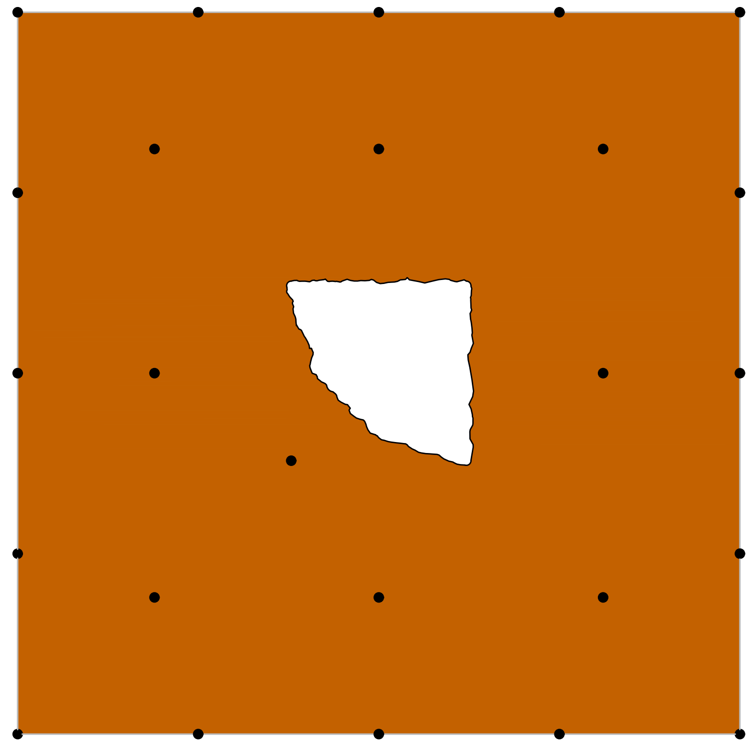


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1275 ms

12

Rock volume blasted at this stage: 0.168 m^3
Rock volume blasted until now: 0.648 m^3

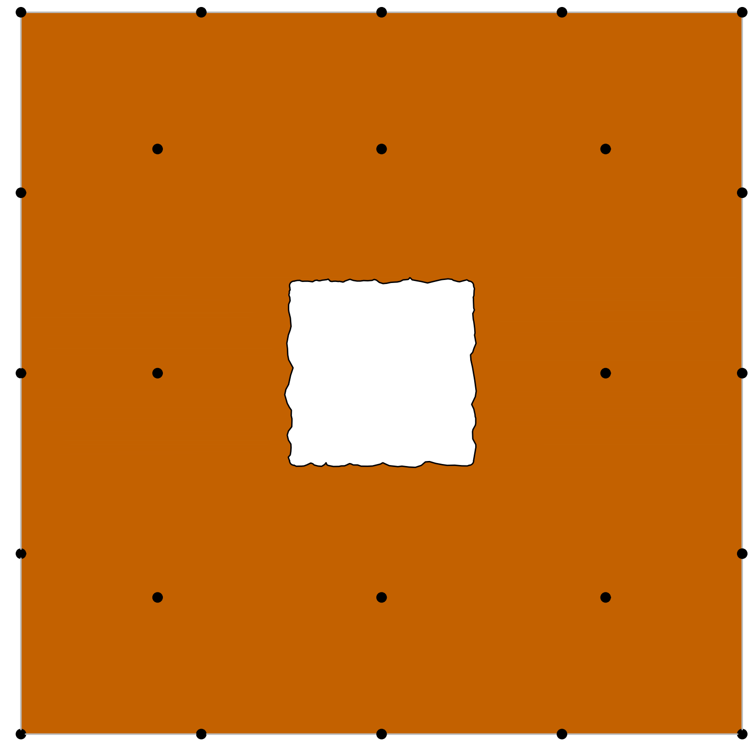


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1300 ms

13

Rock volume blasted at this stage: 0.228 m^3
Rock volume blasted until now: 0.876 m^3

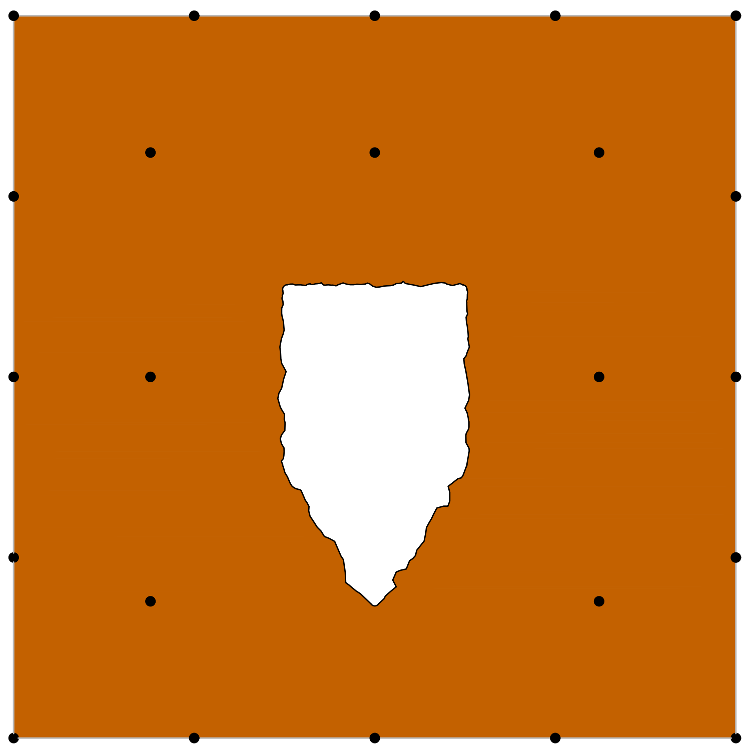


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1325 ms

14

Rock volume blasted at this stage: 0.341 m^3
Rock volume blasted until now: 1.217 m^3

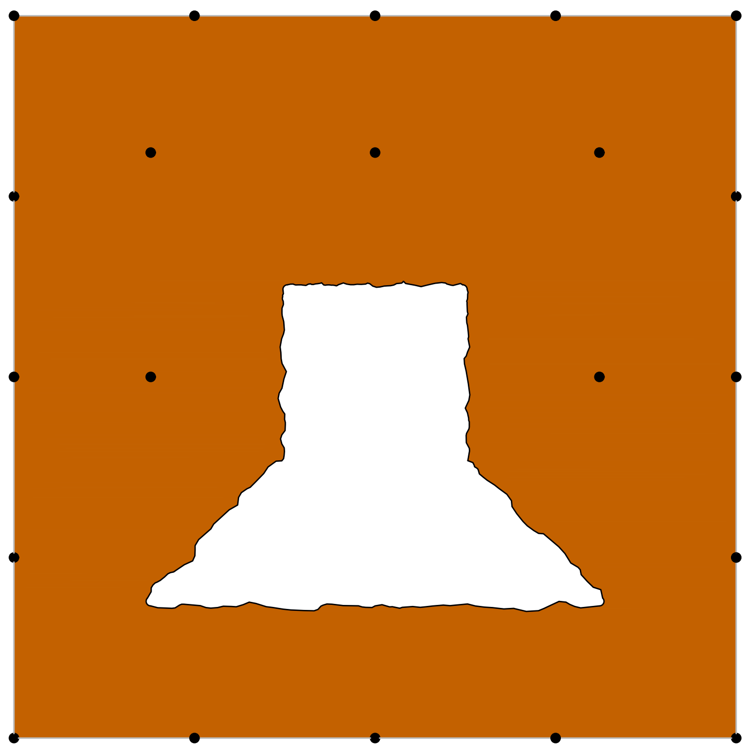


Charge per delay: 1.124 kg

Instant of the detonation sequence at 1350 ms

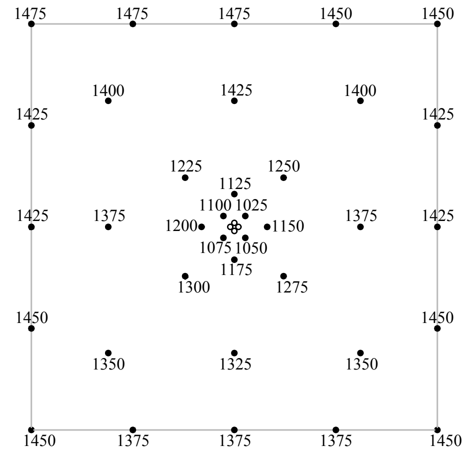
15

Rock volume blasted at this stage: 0.820 m^3
Rock volume blasted until now: 2.037 m^3



Charge per delay: 2.248 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (3)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

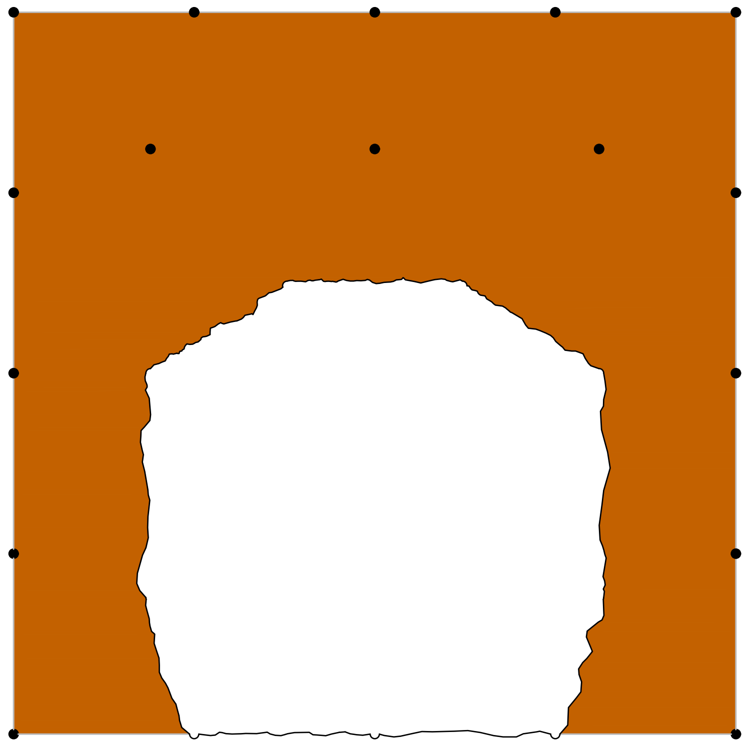
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

16

Instant of the detonation sequence at 1375 ms

Rock volume blasted at this stage: 2.863 m^3
Rock volume blasted until now: 4.900 m^3

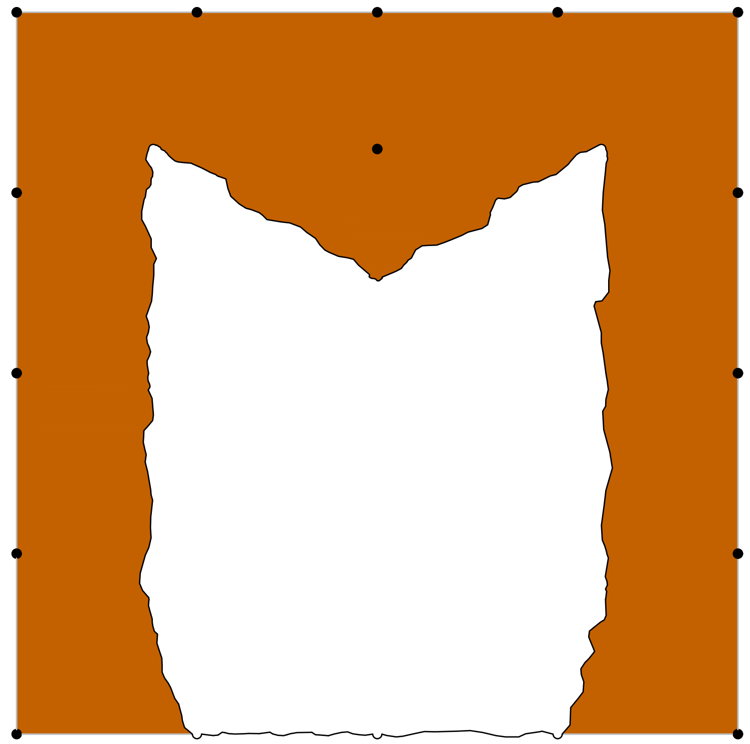


Charge per delay: 6.148 kg

17

Instant of the detonation sequence at 1400 ms

Rock volume blasted at this stage: 1.165 m^3
Rock volume blasted until now: 6.065 m^3

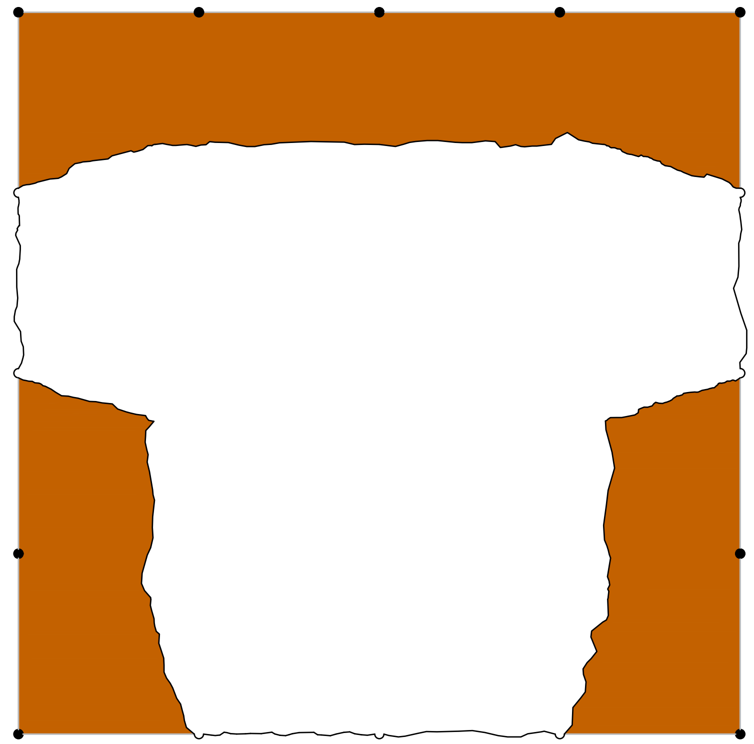


Charge per delay: 2.248 kg

18

Instant of the detonation sequence at 1425 ms

Rock volume blasted at this stage: 2.410 m^3
Rock volume blasted until now: 8.475 m^3

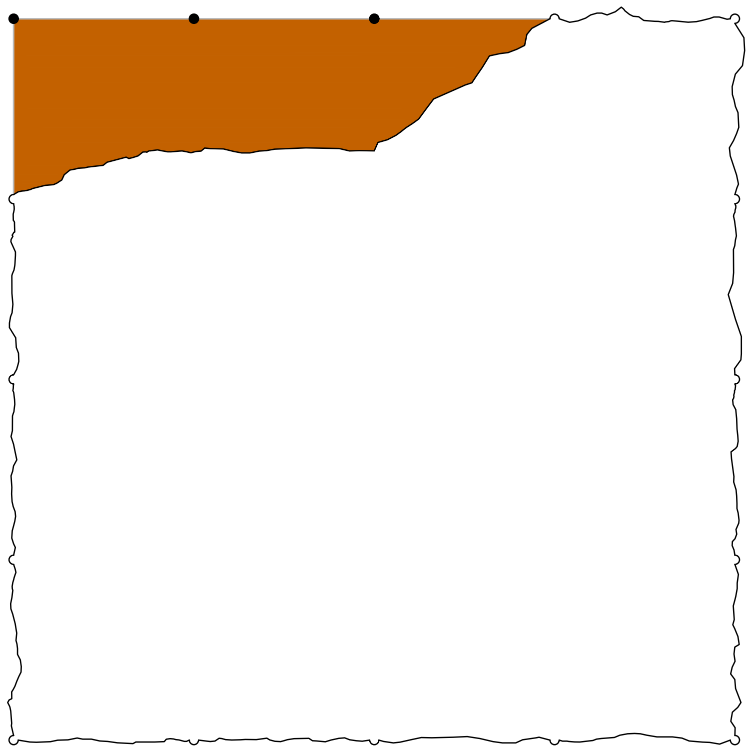


Charge per delay: 5.144 kg

19

Instant of the detonation sequence at 1450 ms

Rock volume blasted at this stage: 3.438 m^3
Rock volume blasted until now: 11.913 m^3

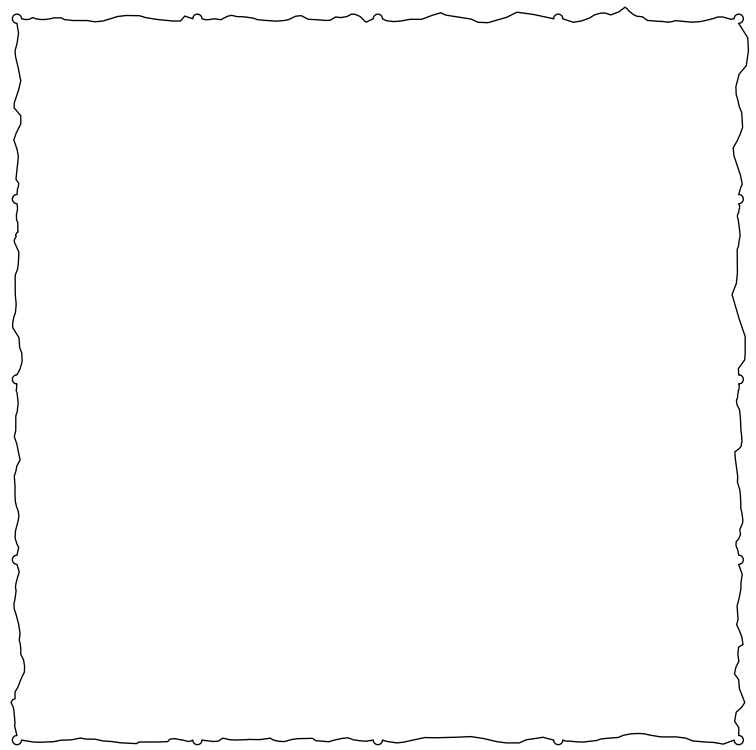


Charge per delay: 6.620 kg

20

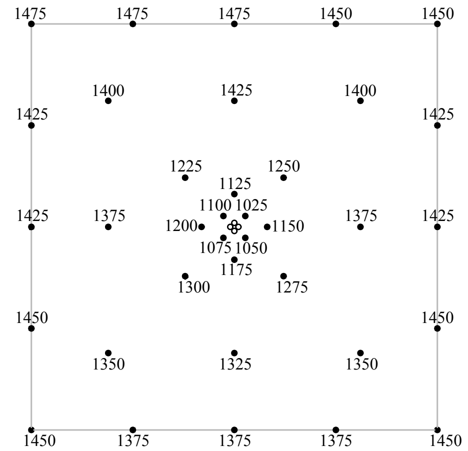
Instant of the detonation sequence at 1475 ms

Rock volume blasted at this stage: 1.580 m^3
Rock volume blasted until now: 13.493 m^3



Charge per delay: 3.015 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 1: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (4)

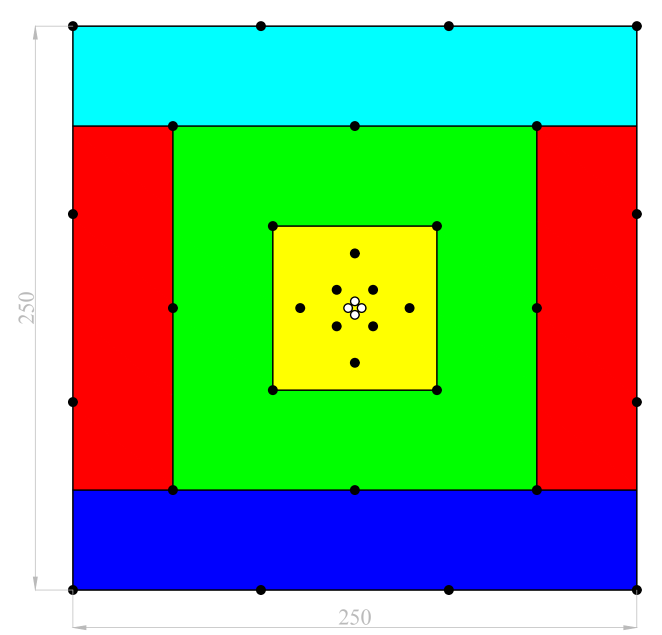
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

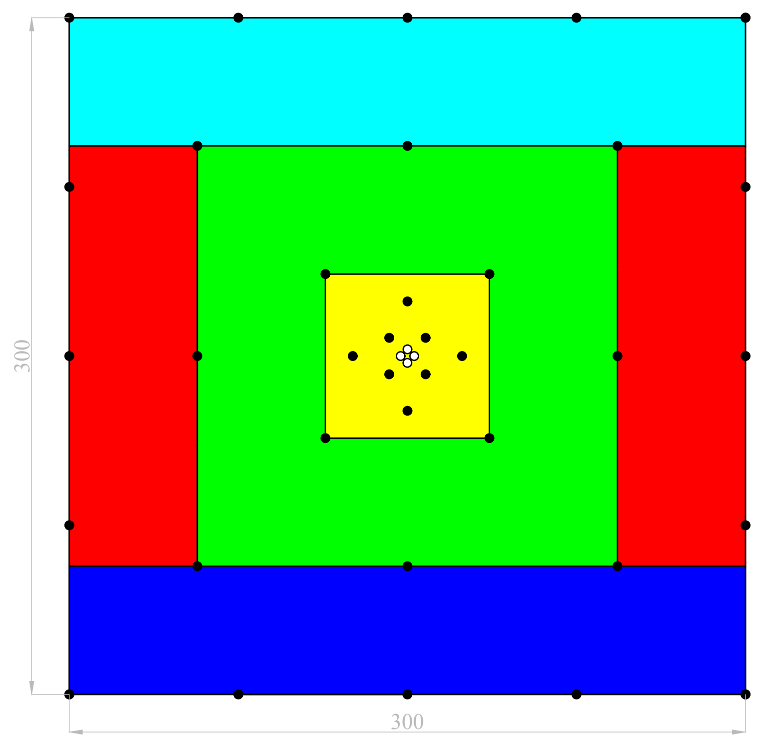
A.Y. 2018 - 2019

1



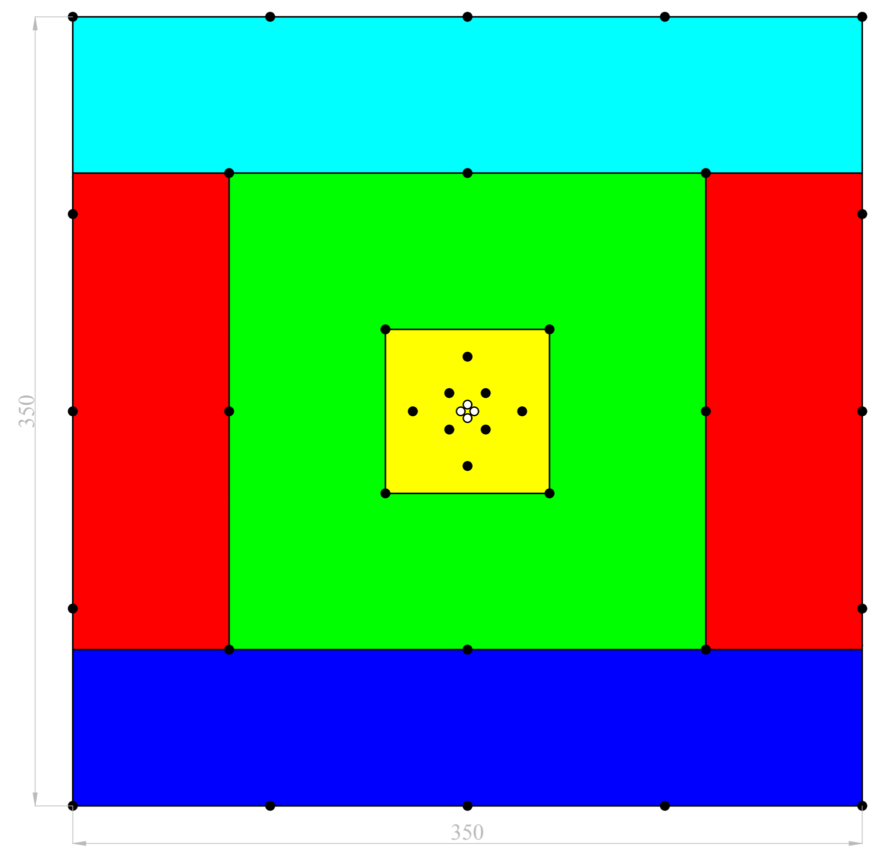
Cut holes	Volume blasted (m^3)	0.771
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	3.102
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	2.899
Contour (floor) holes	Volume blasted (m^3)	1.657
	Amount of explosive (kg)	5.200
	Powder factor (P.F.) (kg/m^3)	3.139
Contour (walls) holes	Volume blasted (m^3)	2.138
	Amount of explosive (kg)	4.020
	Powder factor (P.F.) (kg/m^3)	1.880
Contour (roof) holes	Volume blasted (m^3)	1.657
	Amount of explosive (kg)	4.020
	Powder factor (P.F.) (kg/m^3)	2.427
Volume blasted by contour holes (walls + roof) (m^3)		3.795
Amount of explosive for the contour holes (walls + roof) (kg)		8.040
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		2.118
Volume blasted by contour holes (floor + walls + roof) (m^3)		5.452
Amount of explosive for the contour holes (floor + walls + roof) (kg)		13.240
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		2.429

2



Cut holes	Volume blasted (m^3)	0.771
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	4.406
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	2.041
Contour (floor) holes	Volume blasted (m^3)	2.564
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	2.535
Contour (walls) holes	Volume blasted (m^3)	3.189
	Amount of explosive (kg)	6.030
	Powder factor (P.F.) (kg/m^3)	1.891
Contour (roof) holes	Volume blasted (m^3)	2.551
	Amount of explosive (kg)	5.025
	Powder factor (P.F.) (kg/m^3)	1.970
Volume blasted by contour holes (walls + roof) (m^3)		5.739
Amount of explosive for the contour holes (walls + roof) (kg)		11.055
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		1.926
Volume blasted by contour holes (floor + walls + roof) (m^3)		8.304
Amount of explosive for the contour holes (walls + roof) (kg)		17.555
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		2.114

3



Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	5.898
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	1.525
Contour (floor) holes	Volume blasted (m^3)	3.633
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	1.789
Contour (walls) holes	Volume blasted (m^3)	4.397
	Amount of explosive (kg)	6.030
	Powder factor (P.F.) (kg/m^3)	1.371
Contour (roof) holes	Volume blasted (m^3)	3.633
	Amount of explosive (kg)	5.025
	Powder factor (P.F.) (kg/m^3)	1.383
Volume blasted by contour holes (walls + roof) (m^3)		8.030
Amount of explosive for the contour holes (walls + roof) (kg)		11.055
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		1.377
Volume blasted by contour holes (floor + walls + roof) (m^3)		11.663
Amount of explosive for the contour holes (walls + roof) (kg)		17.555
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.505

	1	2	3
Cross section (m^2)	6.25	9.00	12.25
Design pull (m)	1.5	1.5	1.5
Blasted volume (m^3)	9.325	13.481	18.332
Volume to be removed (m^3)	13.055	20.222	25.665
Total amount of explosive (kg)	36.404	40.719	40.719
Overall Powder Factor (P.F.) (kg/m^3)	3.904	3.020	2.221
Holes diameter (mm)	38.1	38.1	38.1

	1	2	3
Holes number (-)	36	40	40
Holes density (holes/m^2)	5.76	4.44	3.26
Total drilled length (m)	54	60	60
Look - out angle (°)	5	5	5
Specific drilling (S.D.) (m/m^3)	5.791	4.451	3.273
Total number of detonators (-)	59	63	63
Detonators specific consumption (D.C.) (detonators/m^3)	6.33	4.67	3.44

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
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Badge Number: 236349

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Technical drawing:
MAIN DATA ON THE PHASE 1 SCHEMES

POLITECNICO DI TORINO

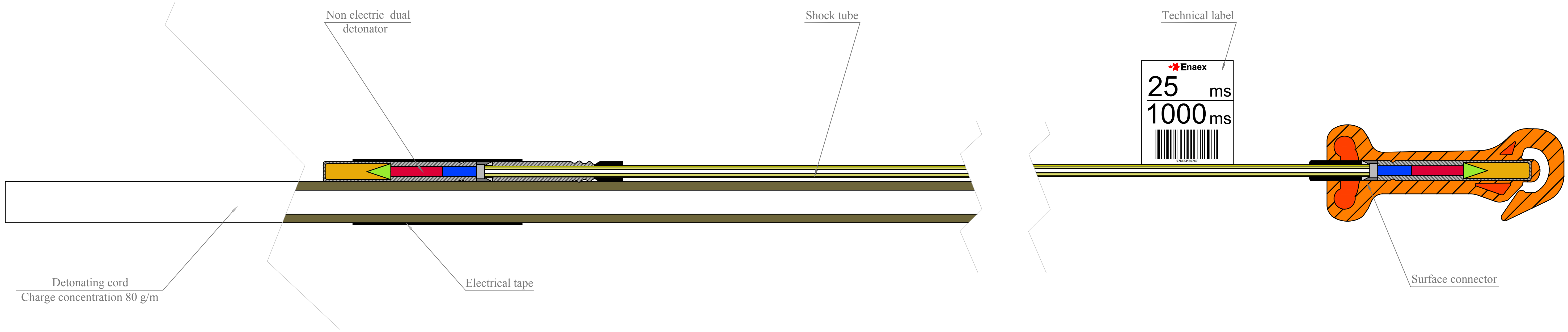
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Detail of the junction detonating cord- bottom hole detonator - surface connector (dual detonator system)

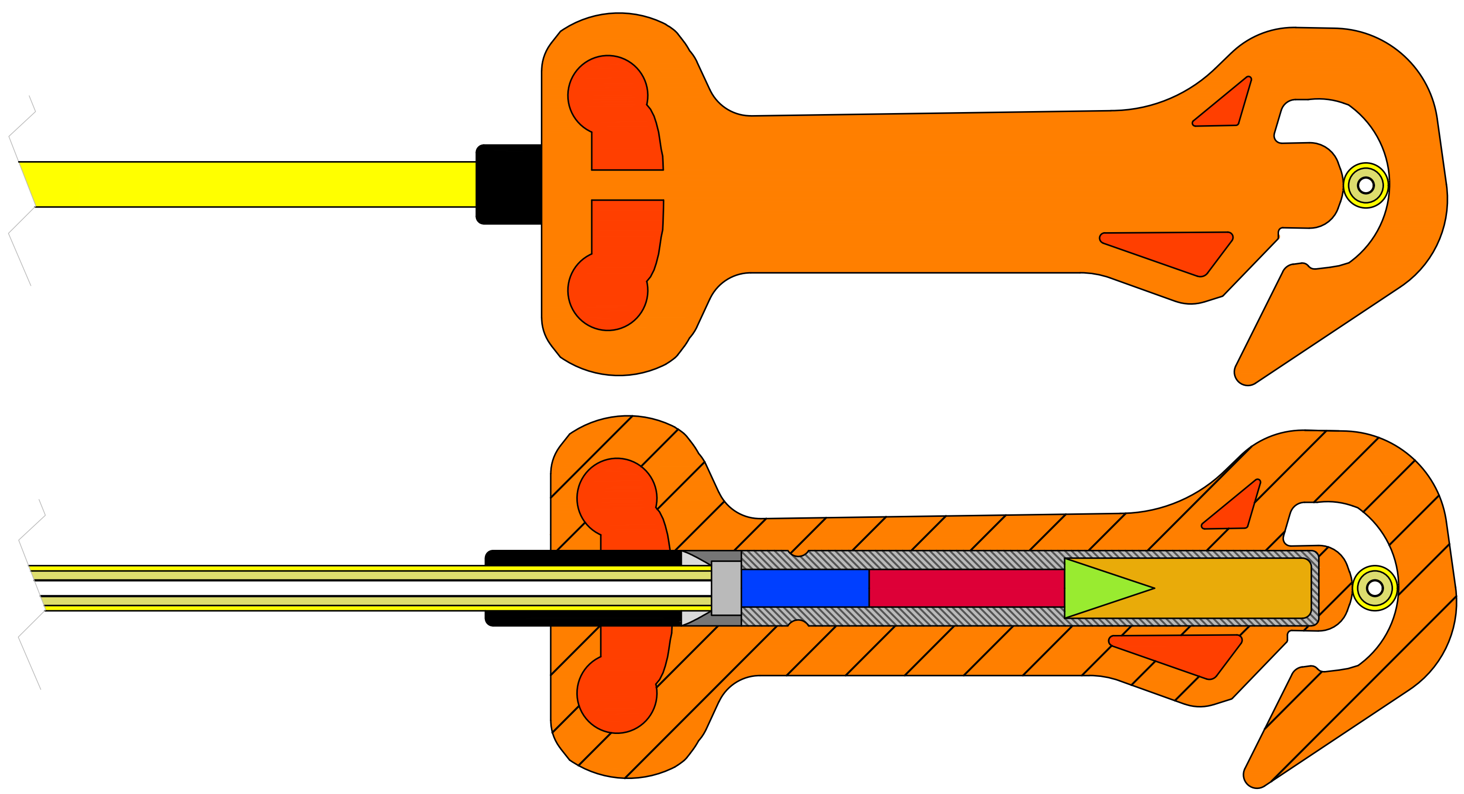
Scale: 1:1



Detail of the surface connector: top view

Scale: 2:1

Enaex



Detail of the surface connector: side view

Scale: 2:1

Enaex

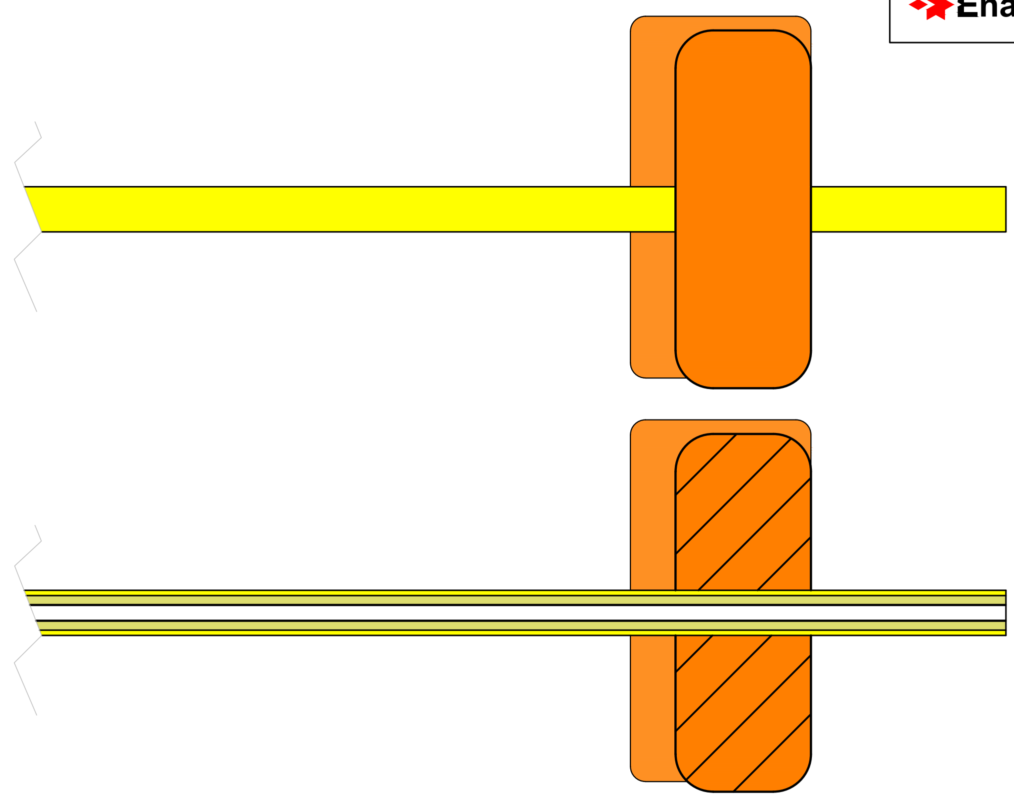
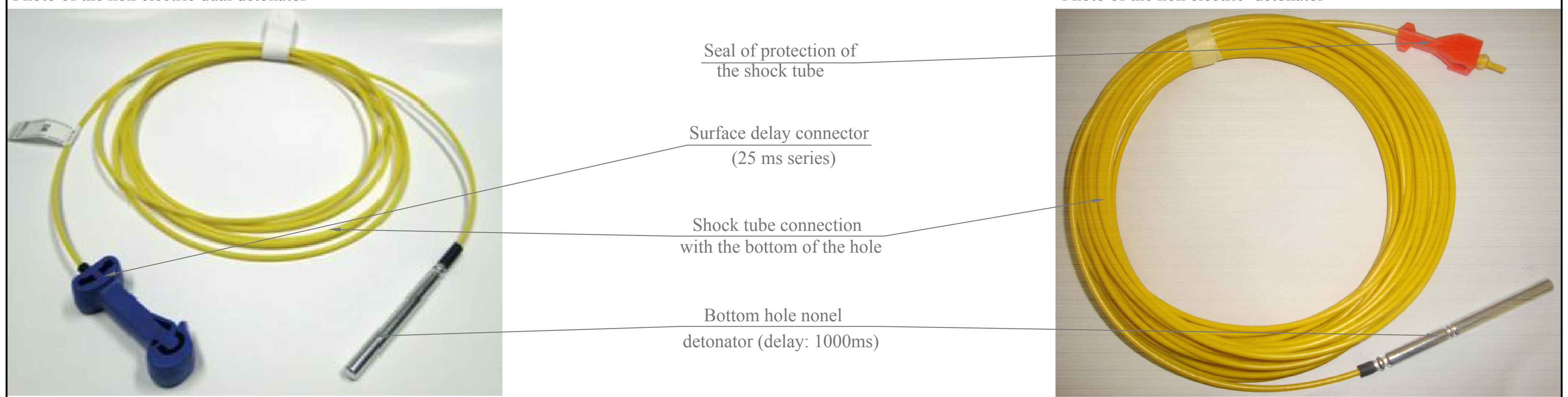


Photo of the non electric dual detonator

Photo of the non electric detonator



Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:

DETAIL OF DETONATING CORD AND DETONATOR JUNCTION AND NONEL EQUIPMENT (1)

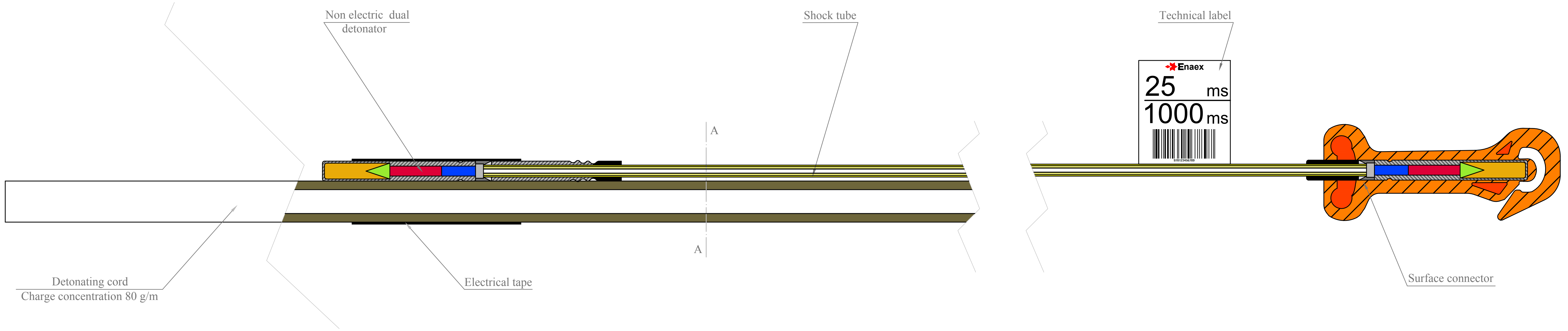
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

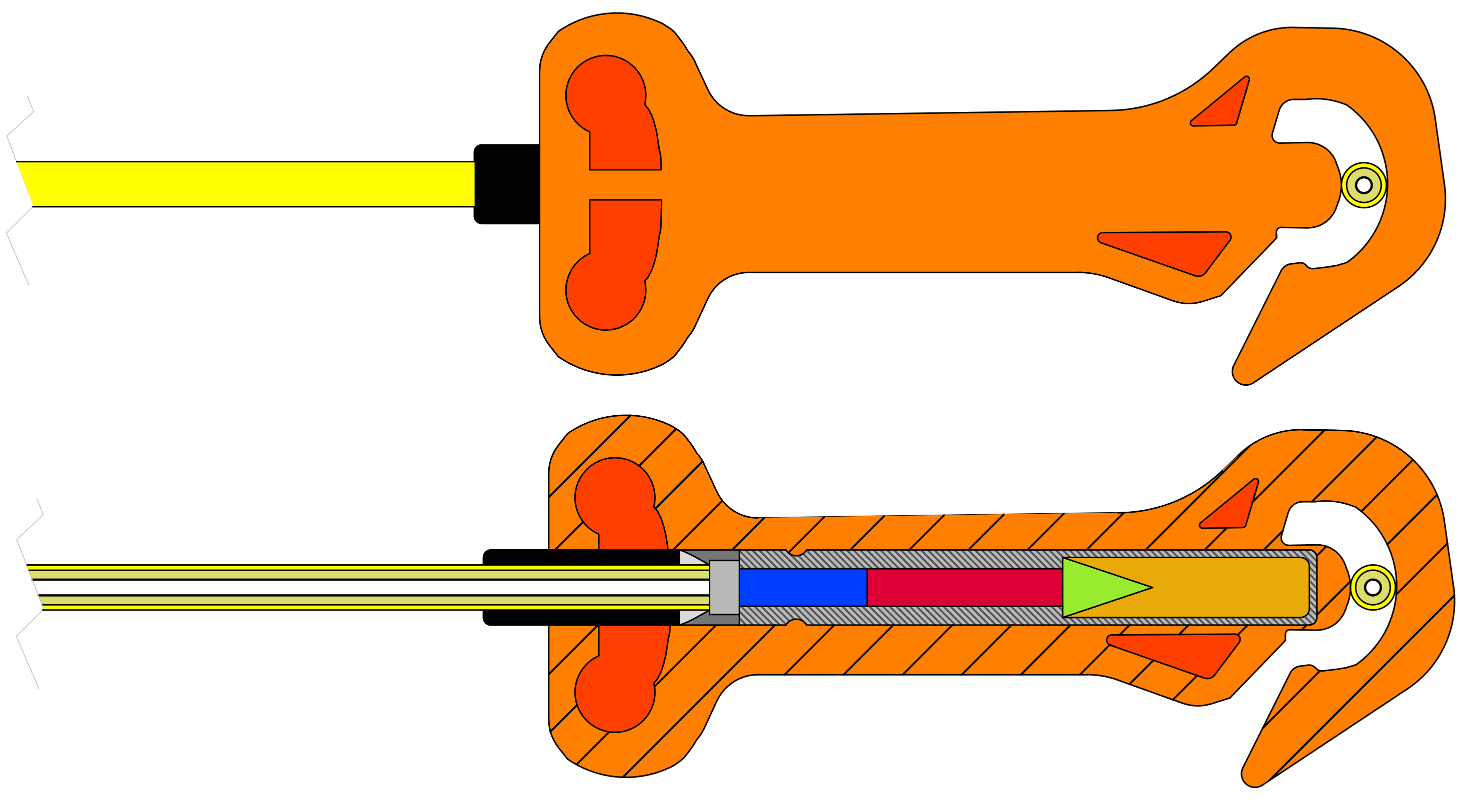
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

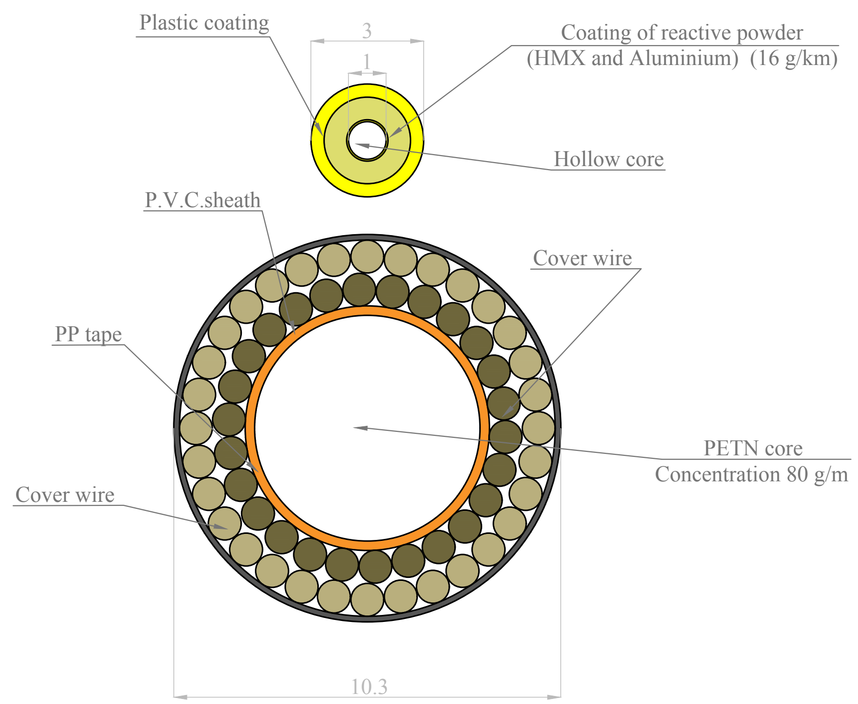
Detail of the junction detonating cord- bottom hole detonator - surface connector (dual detonator system)
Scale: 1:1



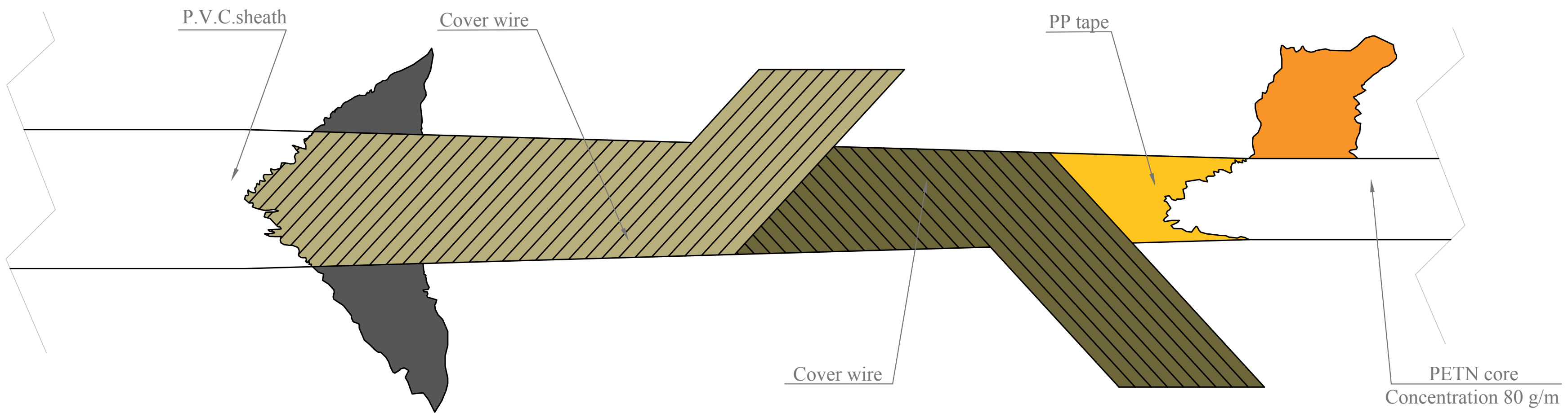
Detail of the surface connector: top view
Scale: 2:1



Section A - A of the junction between shock tube and detonating cord
Scale: 5:1



Detail of the detonating cord used in contour (walls and roof) holes
Scale: 2:1



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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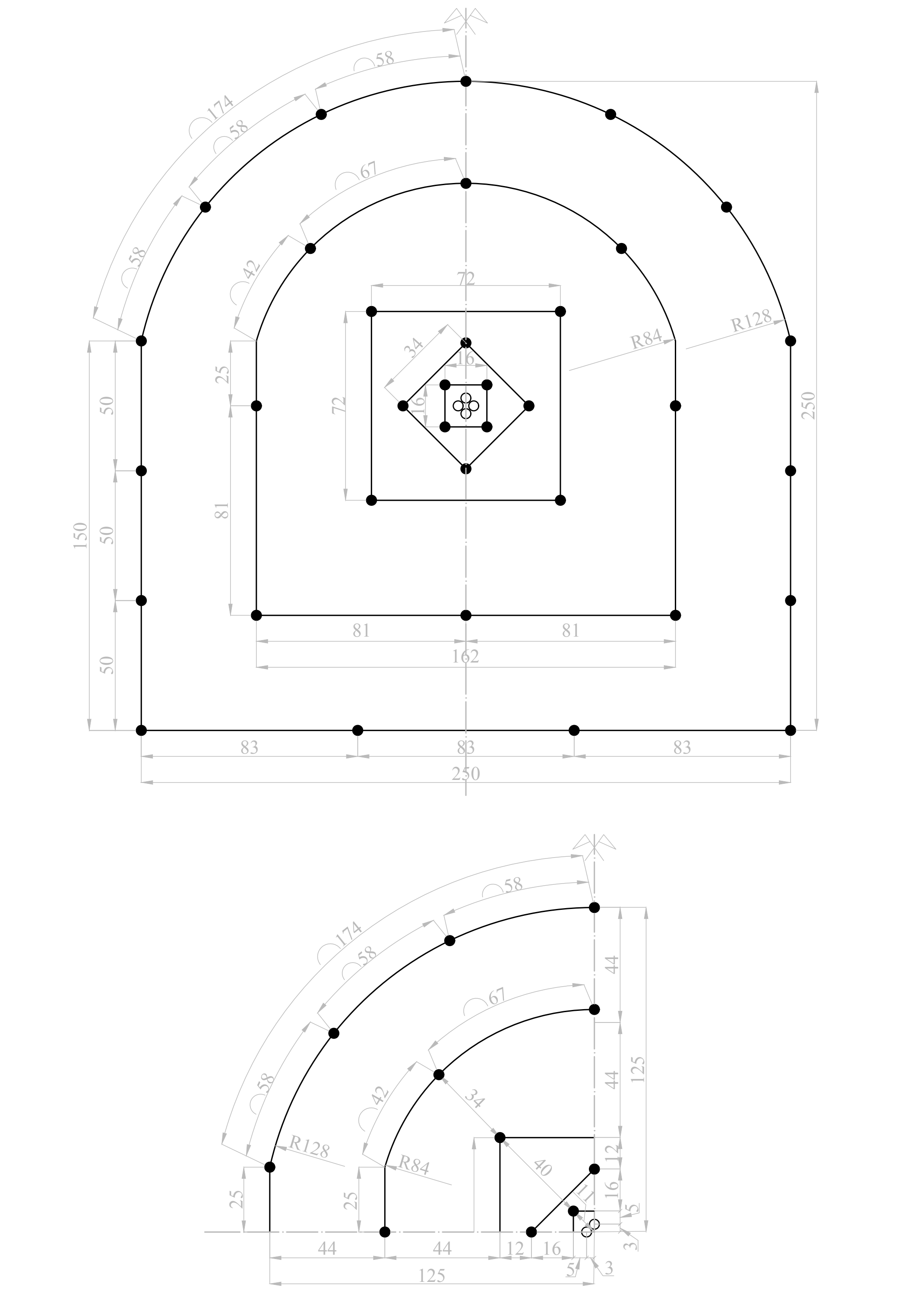
Technical drawing:
DETAIL OF DETONATING CORD AND DETONATOR JUNCTION AND NONEL EQUIPMENT (2)

POLITECNICO DI TORINO

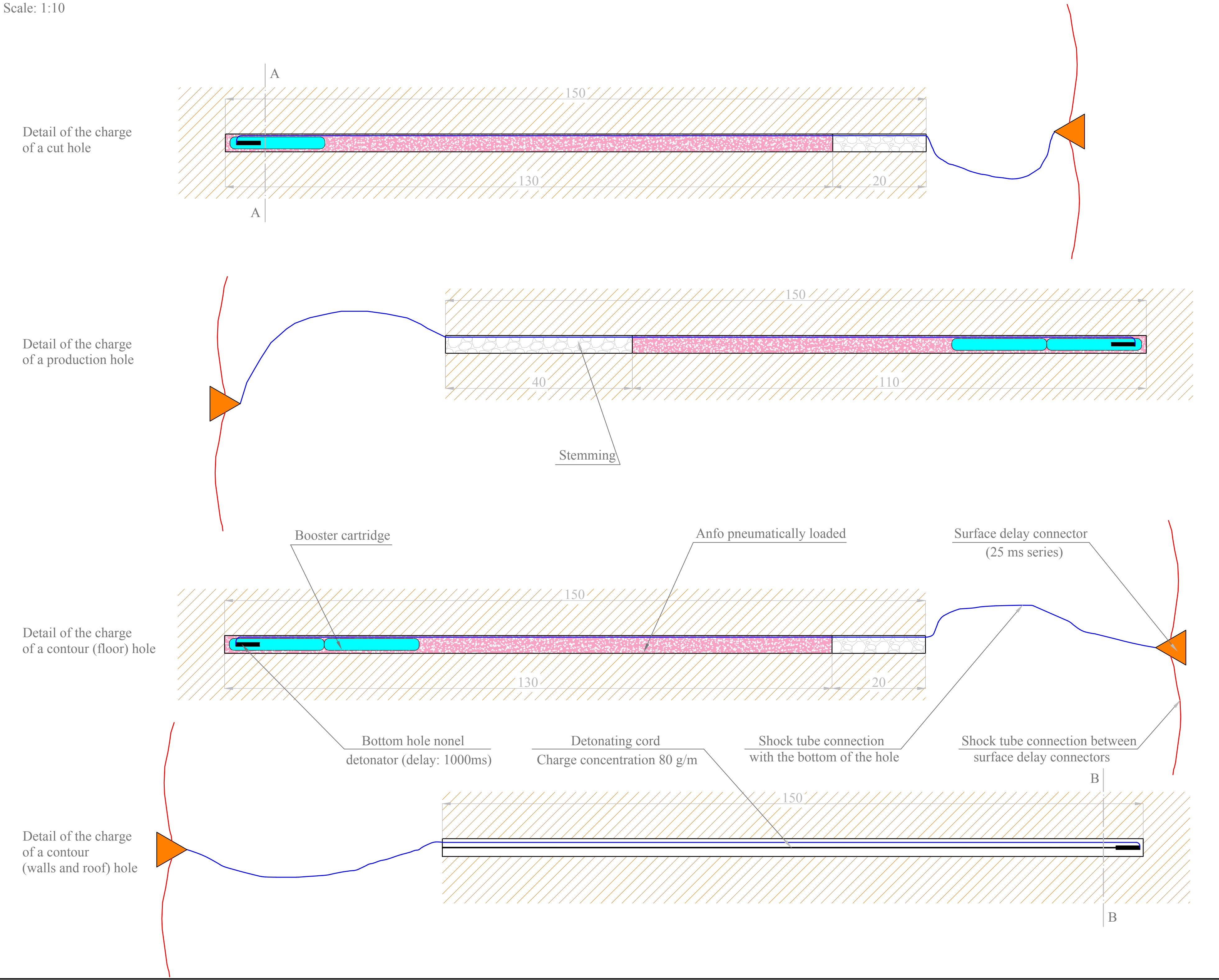
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

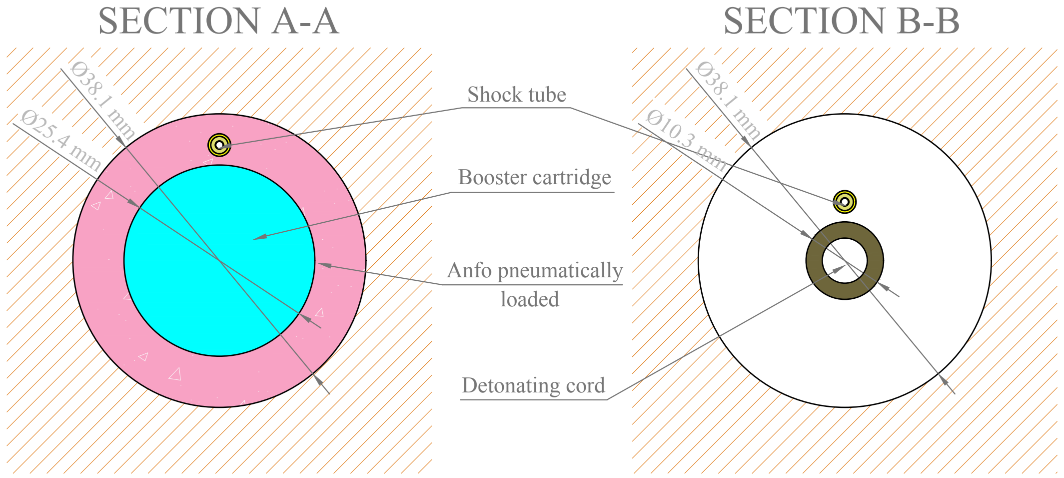
Drilling scheme
Scale: 1:25



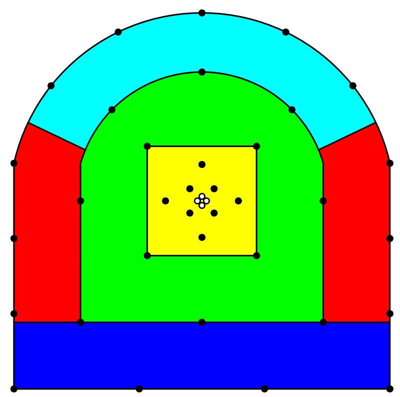
Detail of the charge
Scale: 1:10



Cross sections of the charge; Scale 1:1



Classification of the blasted volumes
Scale: 1:50



Volume blasted by cut holes

Volume blasted by production holes

Volume blasted by contour (floor) holes

Volume blasted by contour (walls) holes

Volume blasted by contour (roof) holes

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:

PHASE 2: DRILLING AND CHARGE SCHEMES FOR SECTION 2.5 x 2.5 m

POLITECNICO DI TORINO

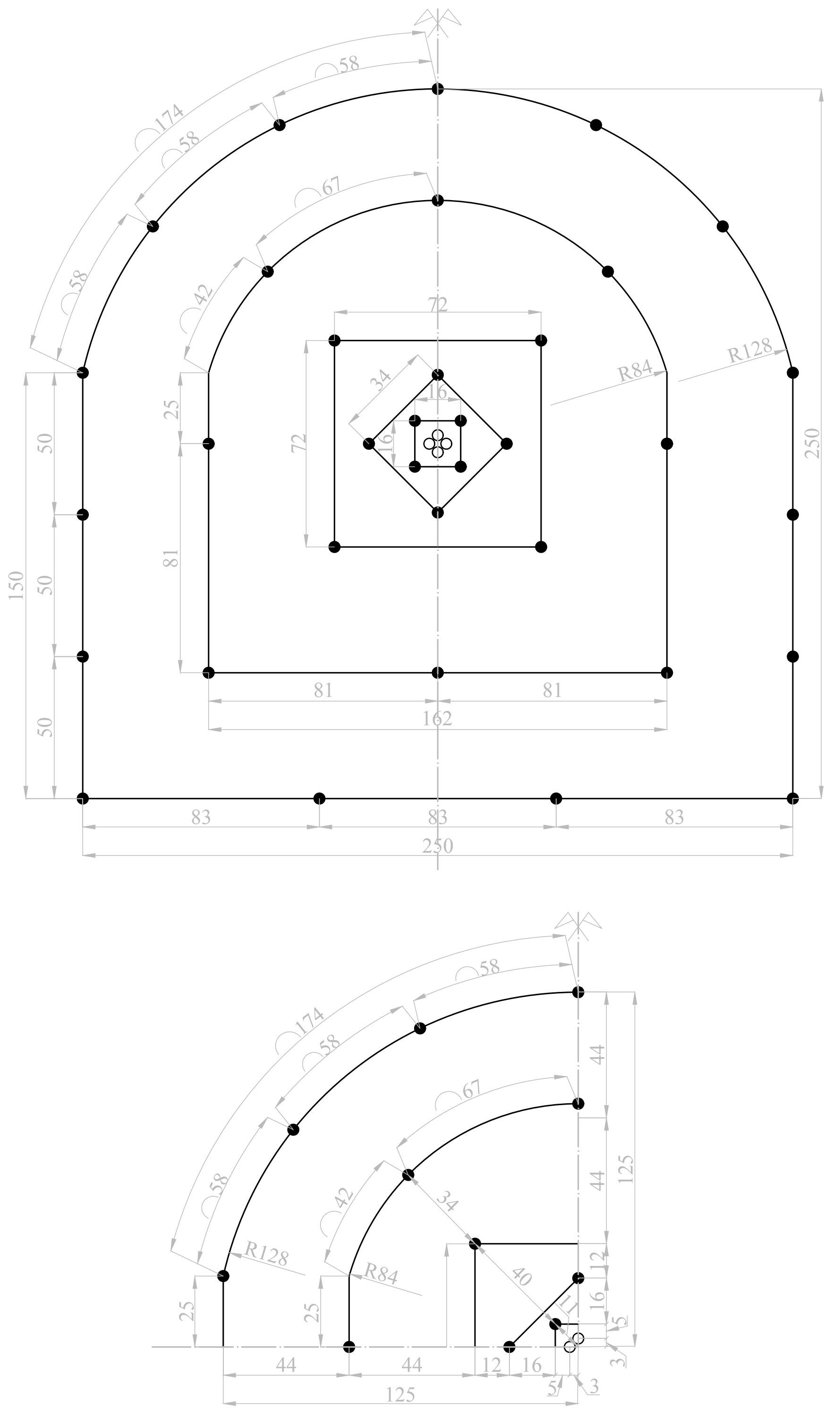
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

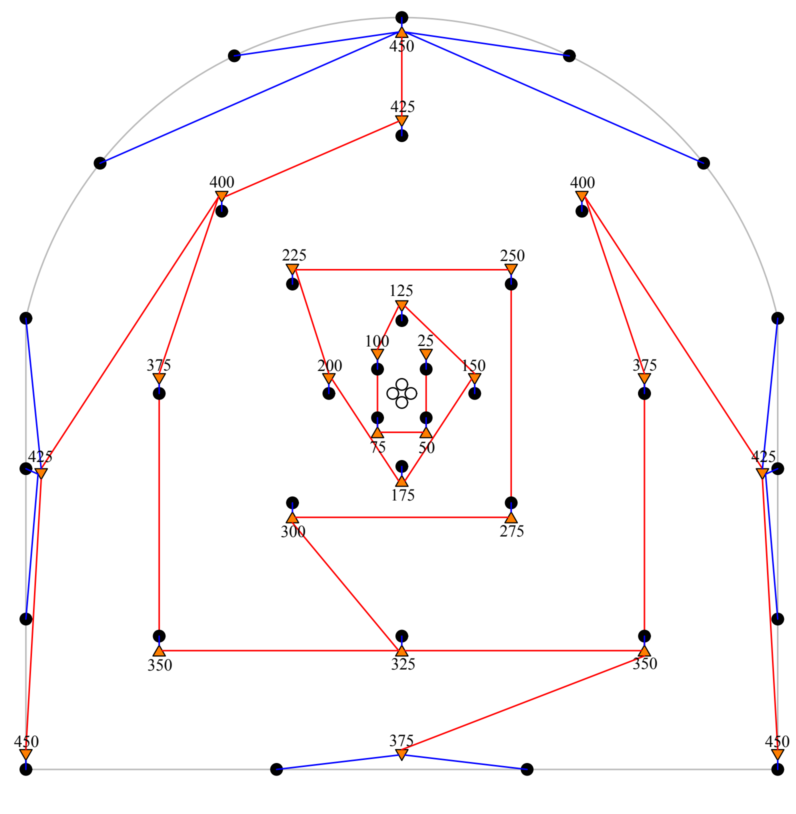
Drilling scheme

Scale: 1:25



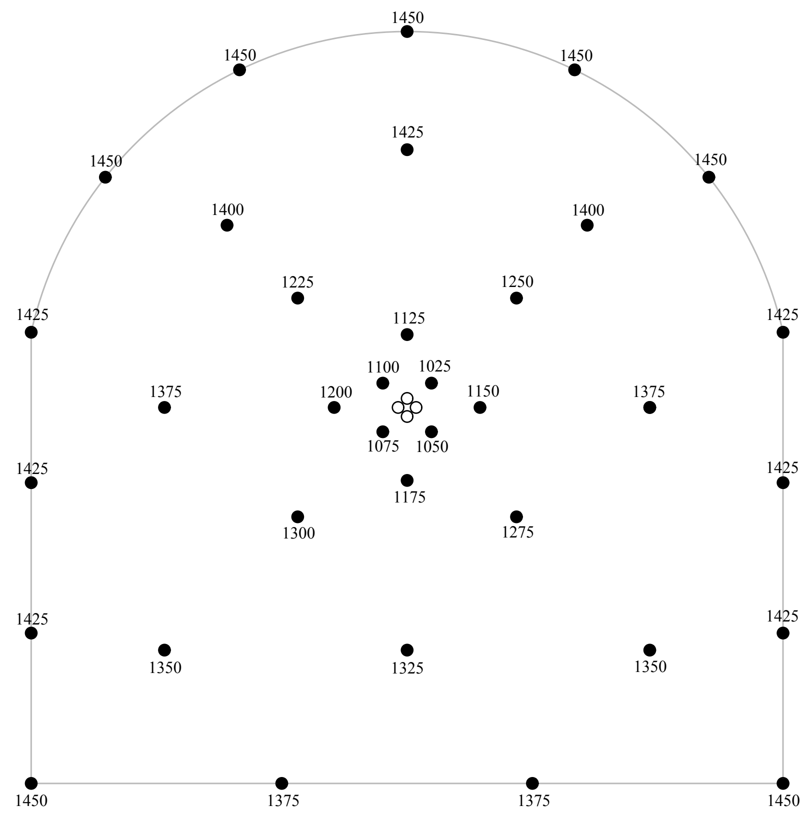
Trigger times of the surface delay connectors

Scale: 1:25



Trigger times of the in-hole detonators

Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
10 (1250 ms)	25 ms	1000 ms	1	1
11 (1275 ms)	25 ms	1000 ms	1	1
12 (1300 ms)	25 ms	1000 ms	1	1
13 (1325 ms)	25 ms	1000 ms	1	1
14 (1350 ms)	25 ms	1000 ms	2	2
15 (1375 ms)	25 ms	1000 ms	3	4
16 (1400 ms)	25 ms	1000 ms	2	2
17 (1425 ms)	25 ms	1000 ms	3	7
18 (1450 ms)	25 ms	1000 ms	3	7

Classification of the blasted volumes

Scale: 1:50

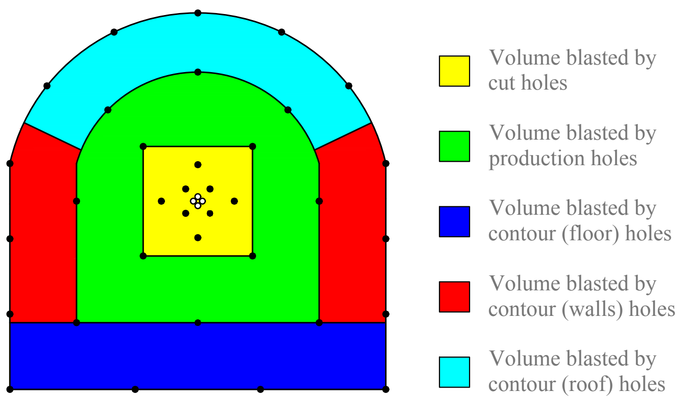
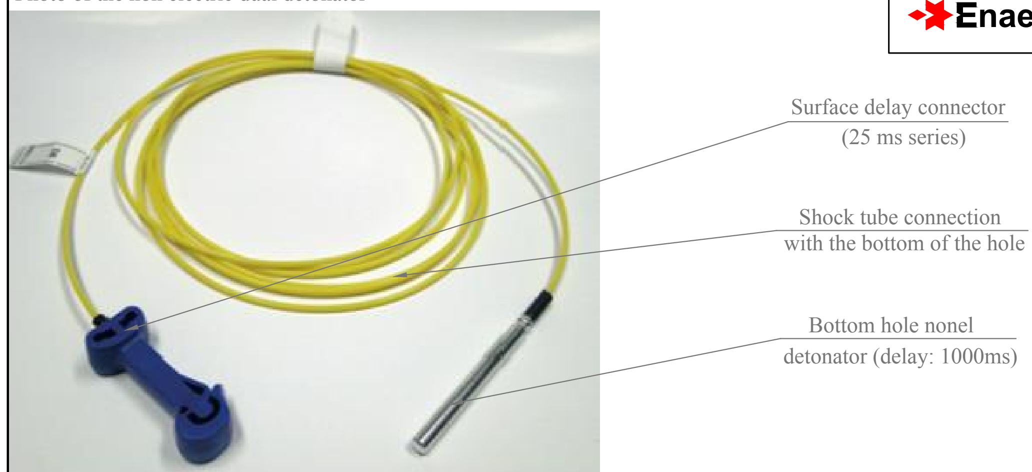


Photo of the non electric dual detonator

Enaex



Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:

Simone BEZ

Badge Number: 236349

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Technical drawing:

PHASE 2: TRIGGER TIMES FOR SECTION 2.5 x 2.5 m

Scale: 1:25

POLITECNICO DI TORINO

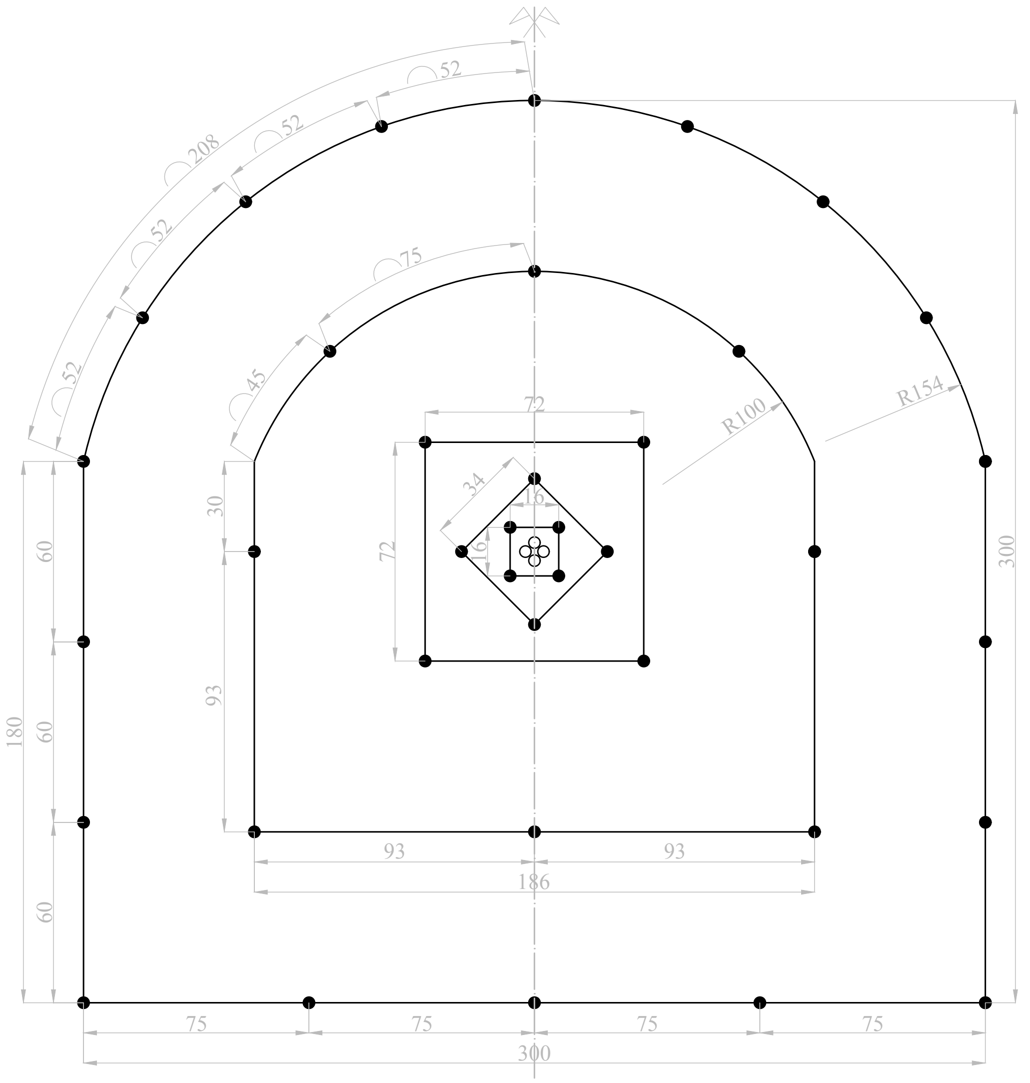
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

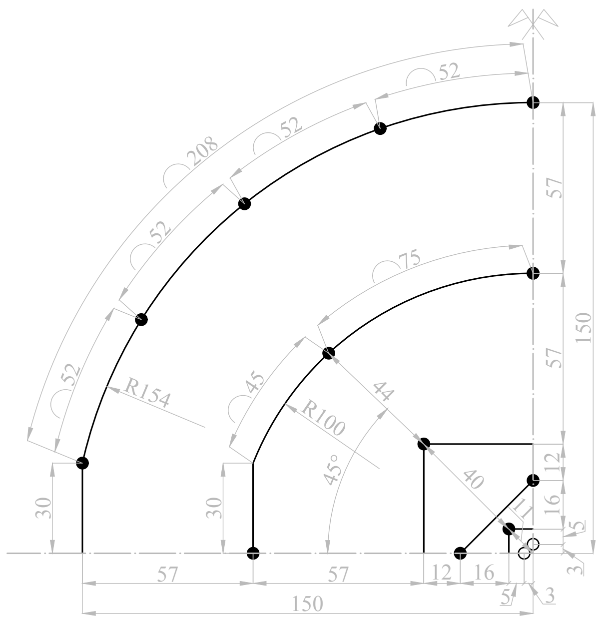
Supervisor:

Prof. Marilena Cardu

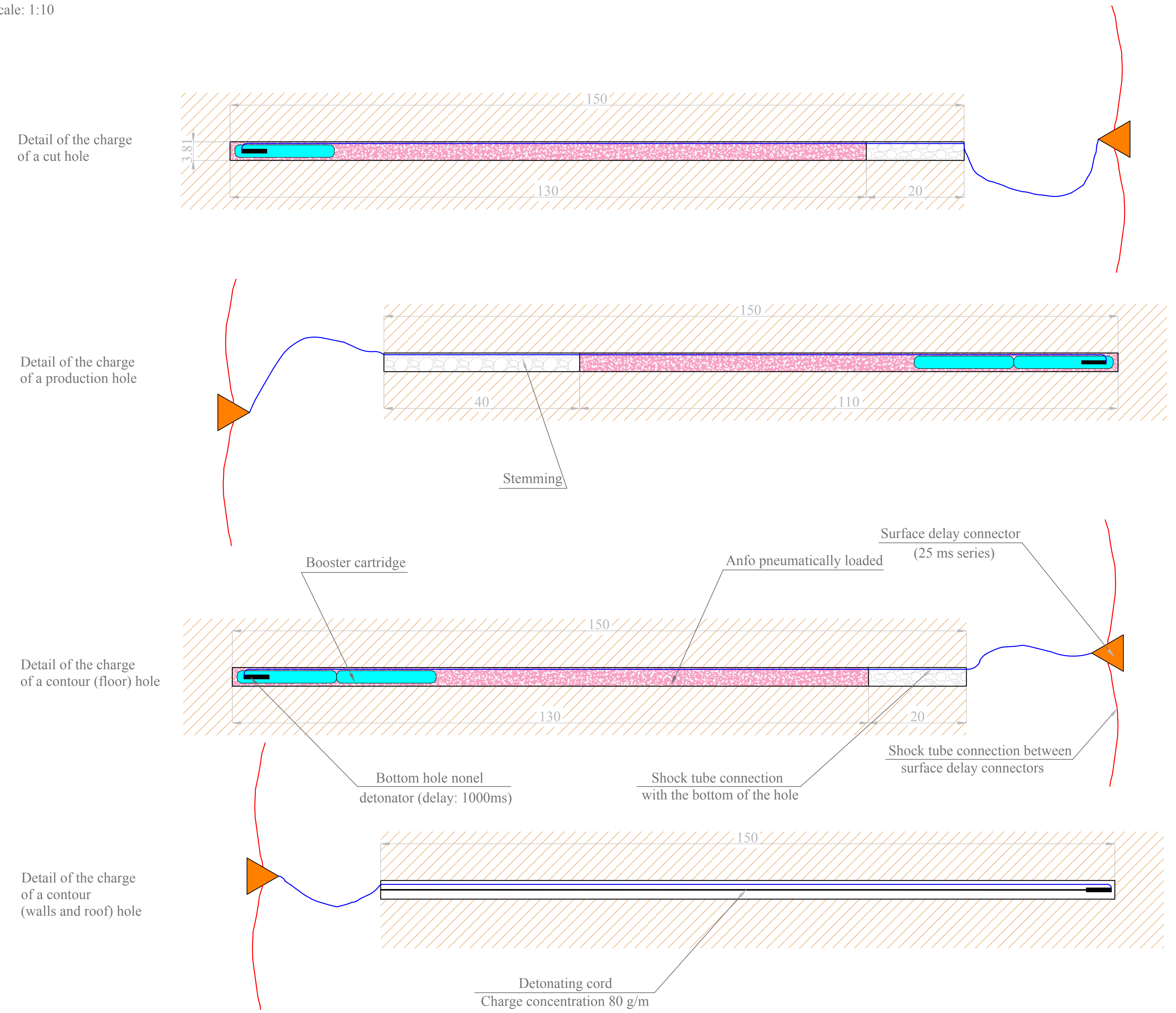
A.Y. 2018 - 2019

Drilling scheme
Scale: 1:25

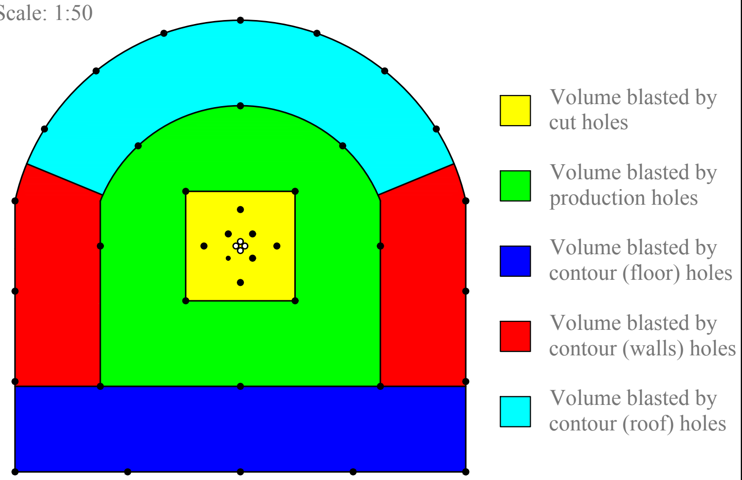




Detail of the charge
Scale: 1:10



Classification of the blasted volumes
Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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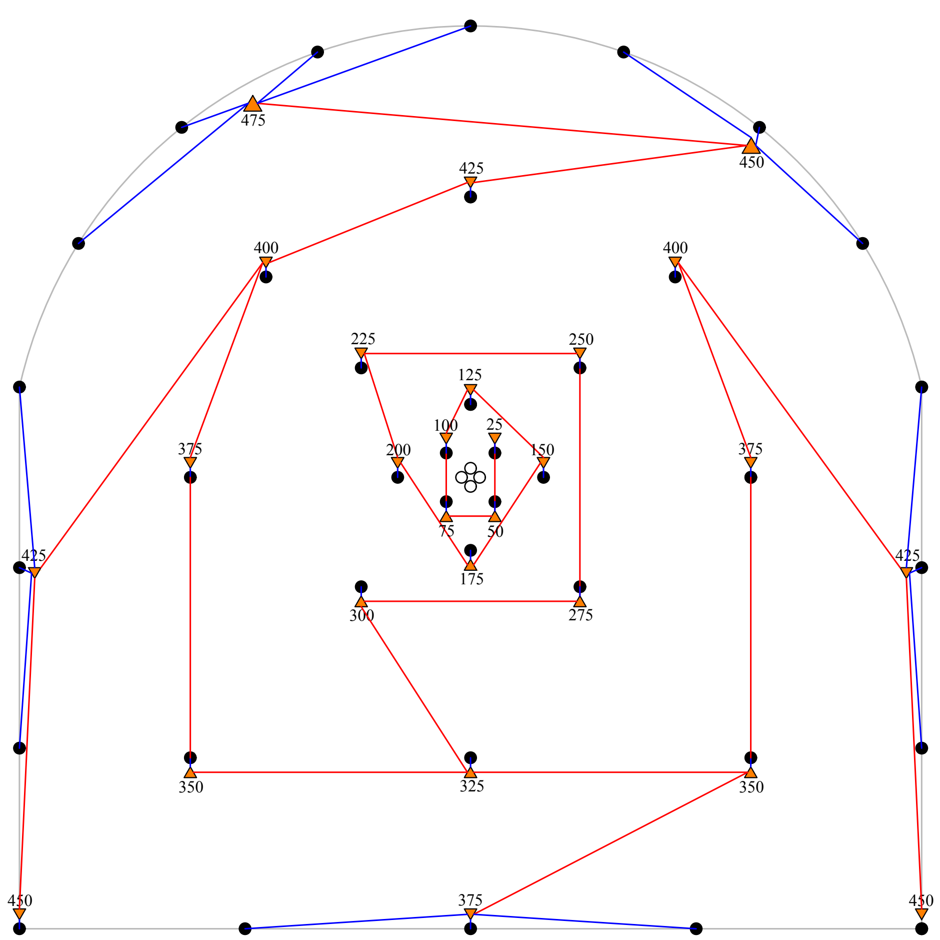
Technical drawing:
PHASE 2: DRILLING AND CHARGE SCHEMES FOR SECTION 3.0 x 3.0 m

POLITECNICO DI TORINO

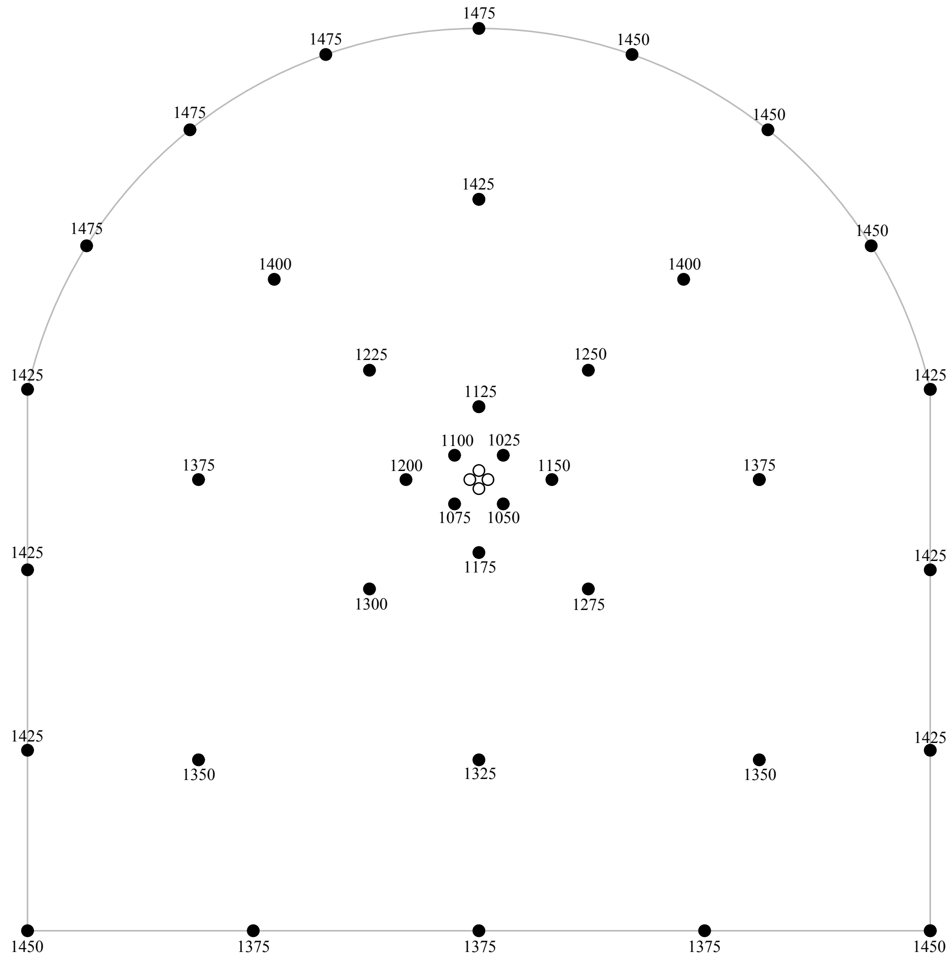
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
11 (1275 ms)	25 ms	1000 ms	1	1
12 (1300 ms)	25 ms	1000 ms	1	1
13 (1325 ms)	25 ms	1000 ms	1	1
14 (1350 ms)	25 ms	1000 ms	2	2
15 (1375 ms)	25 ms	1000 ms	3	5
16 (1400 ms)	25 ms	1000 ms	2	2
17 (1425 ms)	25 ms	1000 ms	3	7
18 (1450 ms)	25 ms	1000 ms	3	5
19 (1475 ms)	25 ms	1000 ms	1	4

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: TRIGGER TIMES FOR SECTION 3.0 x 3.0 m

Scale: 1:25

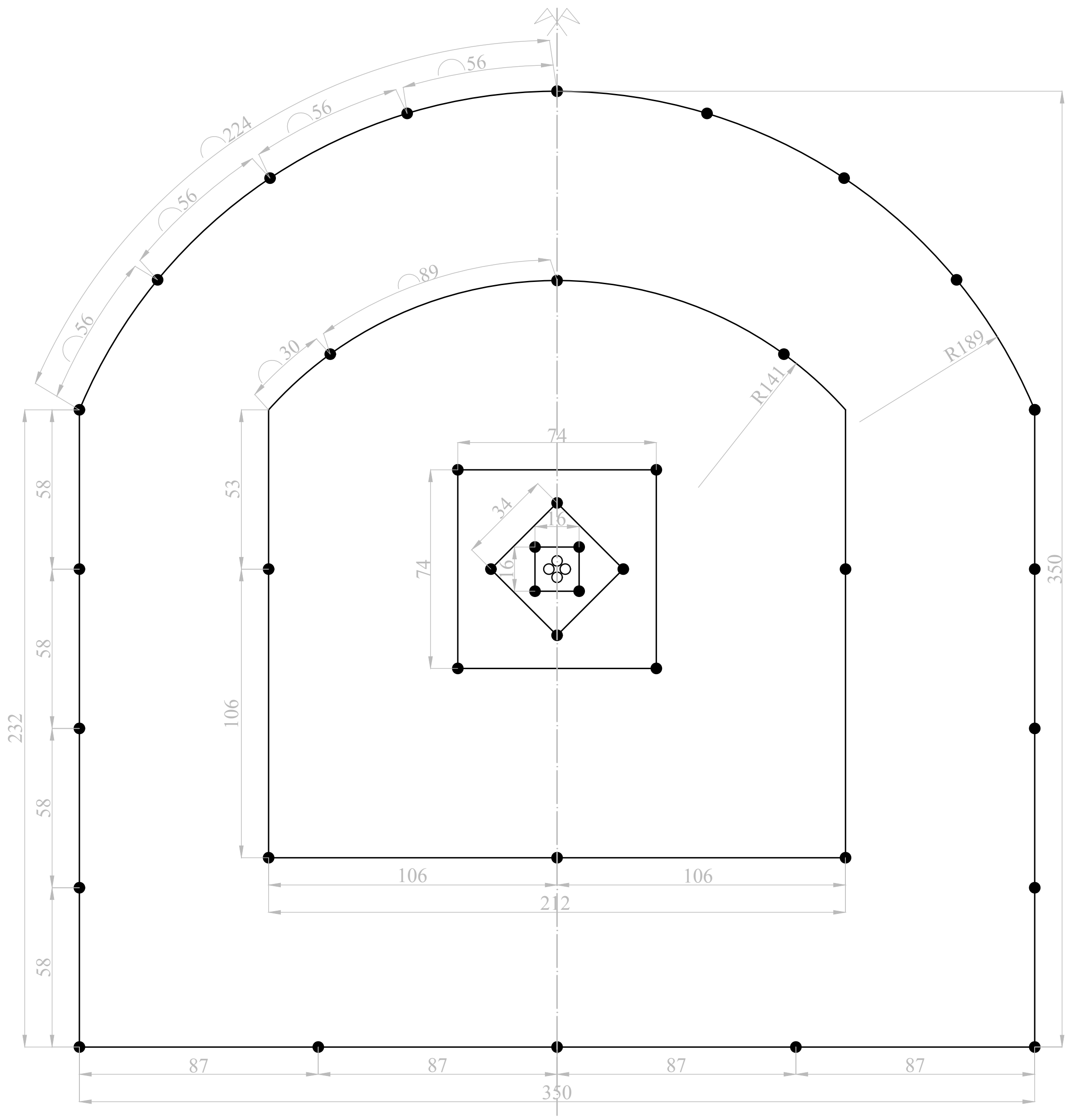
POLITECNICO DI TORINO

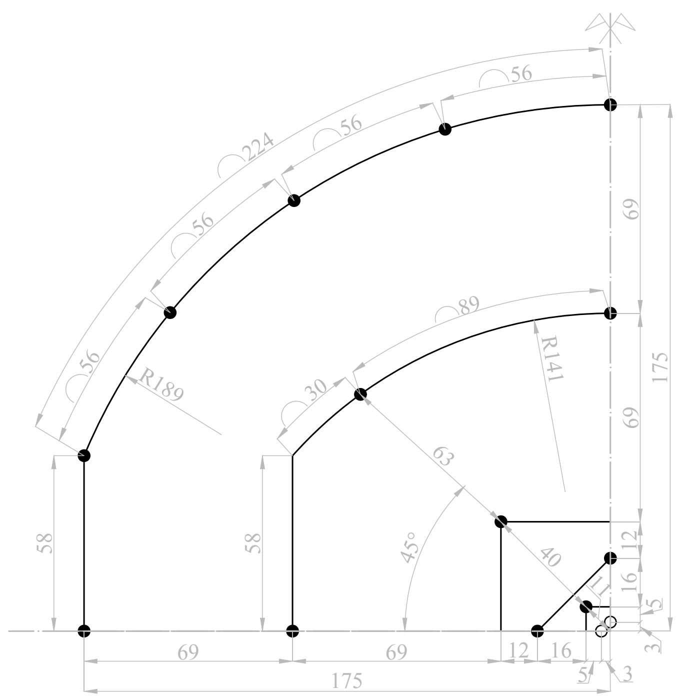
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Drilling scheme
Scale: 1:25





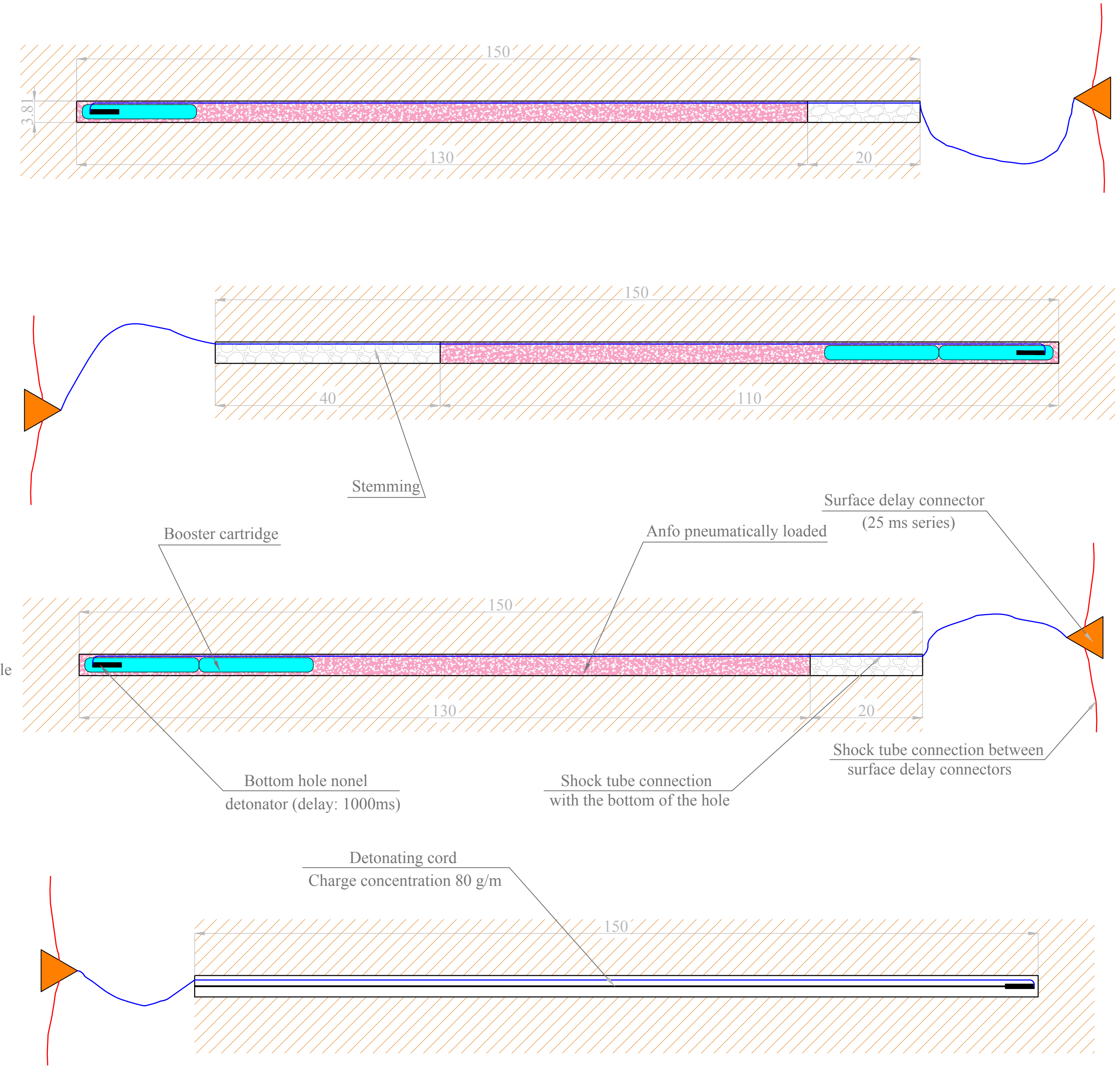
Detail of the charge
Scale: 1:10

Detail of the charge of a cut hole

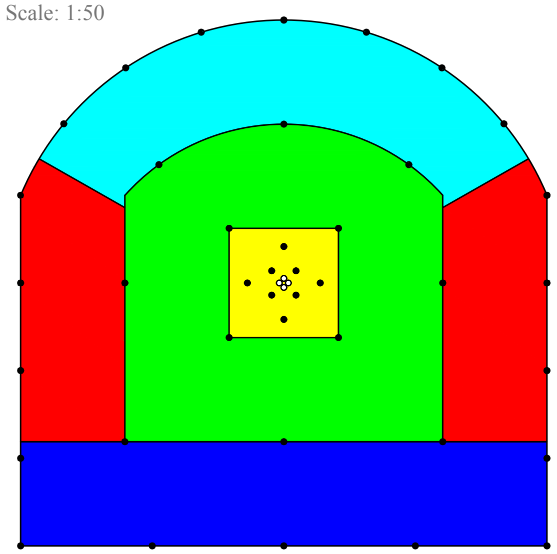
Detail of the charge of a production hole

Detail of the charge of a contour (floor) hole

Detail of the charge of a contour (walls and roof) hole



Classification of the blasted volumes
Scale: 1:50



Volume blasted by cut holes

Volume blasted by production holes

Volume blasted by contour (floor) holes

Volume blasted by contour (walls) holes

Volume blasted by contour (roof) holes

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: DRILLING AND CHARGE SCHEMES FOR SECTION 3.5 x 3.5 m

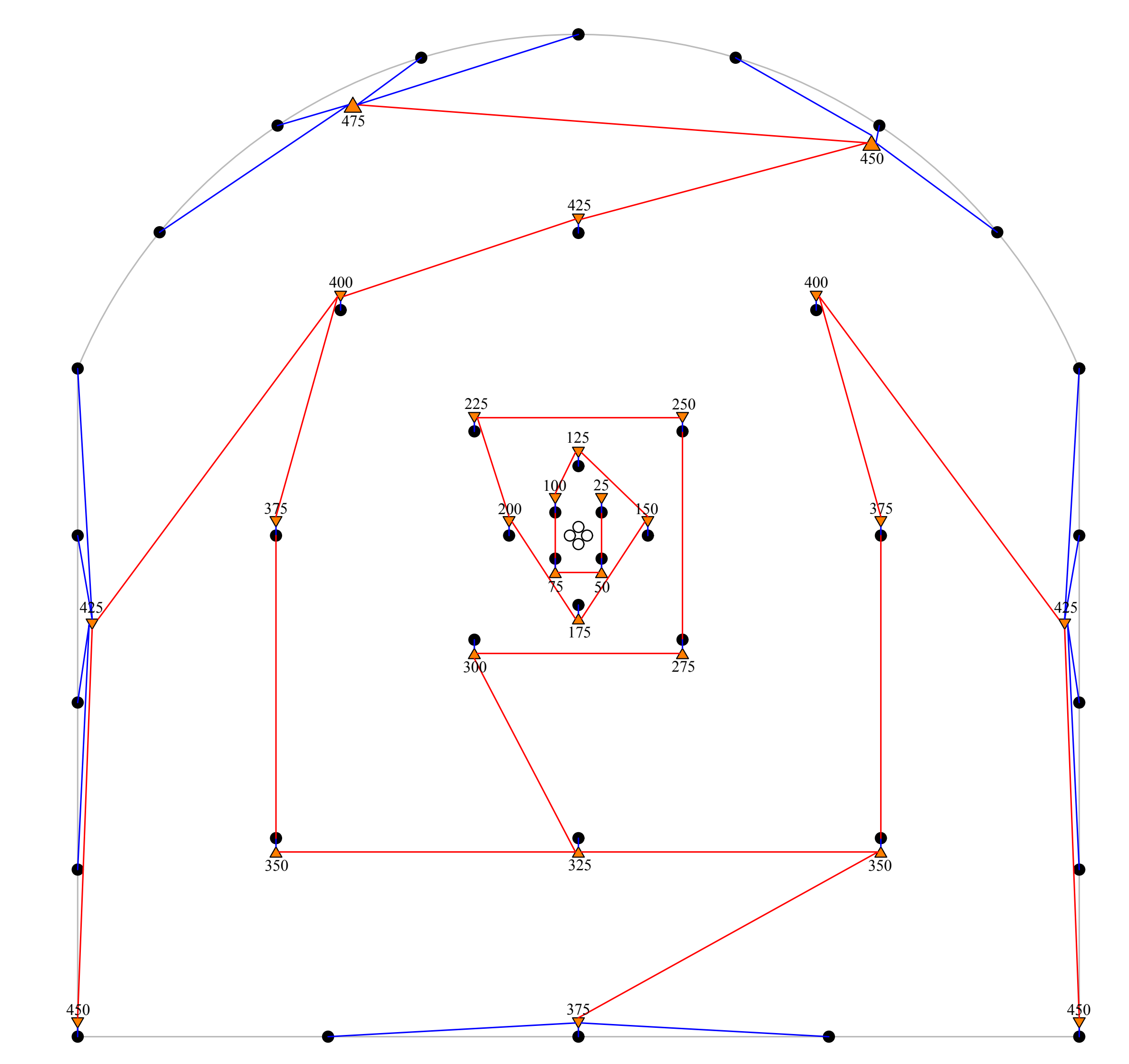
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

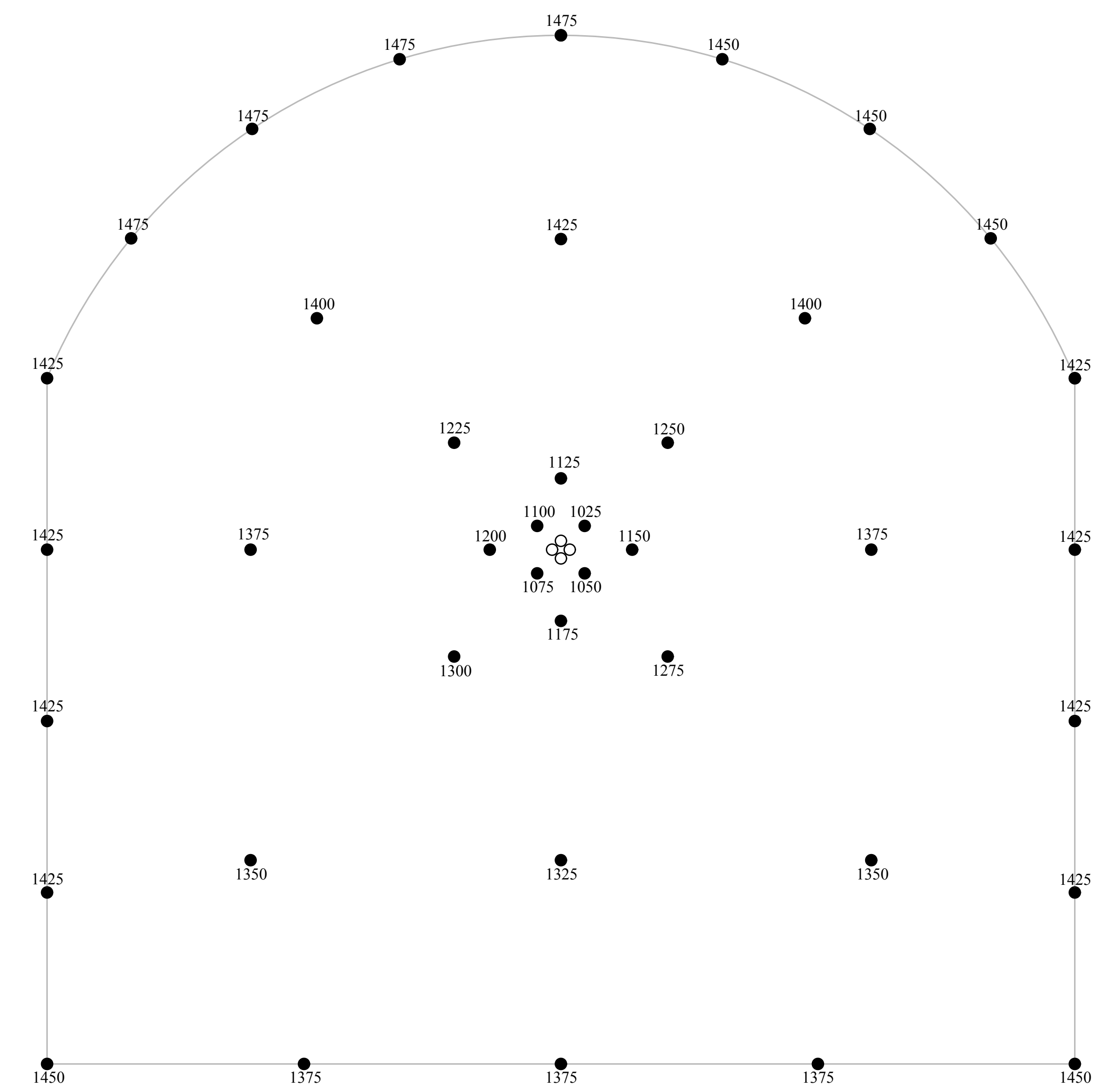
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
11 (1275 ms)	25 ms	1000 ms	1	1
12 (1300 ms)	25 ms	1000 ms	1	1
13 (1325 ms)	25 ms	1000 ms	1	1
14 (1350 ms)	25 ms	1000 ms	2	2
15 (1375 ms)	25 ms	1000 ms	3	5
16 (1400 ms)	25 ms	1000 ms	2	2
17 (1425 ms)	25 ms	1000 ms	3	9
18 (1450 ms)	25 ms	1000 ms	3	5
19 (1475 ms)	25 ms	1000 ms	1	4

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: TRIGGER TIMES FOR SECTION 3.5 x 3.5 m
Scale: 1:25

POLITECNICO DI TORINO

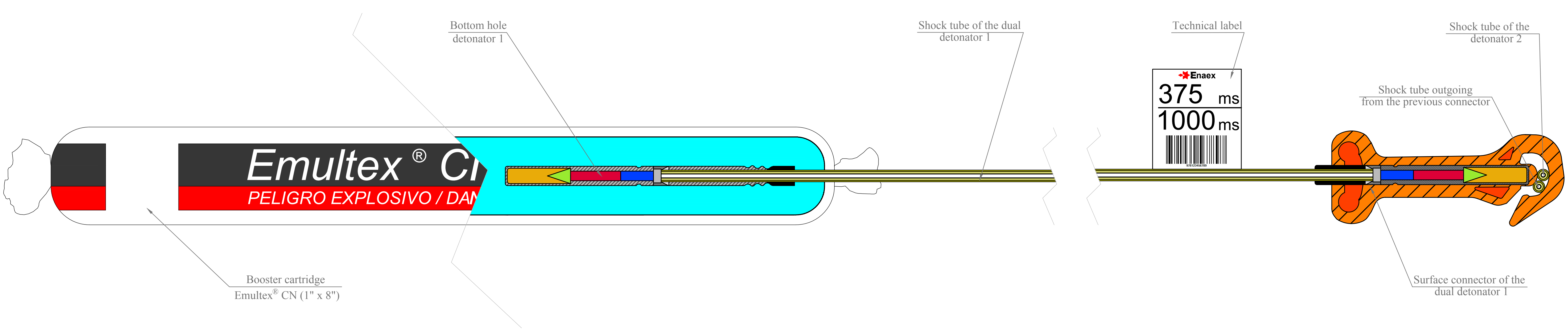
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

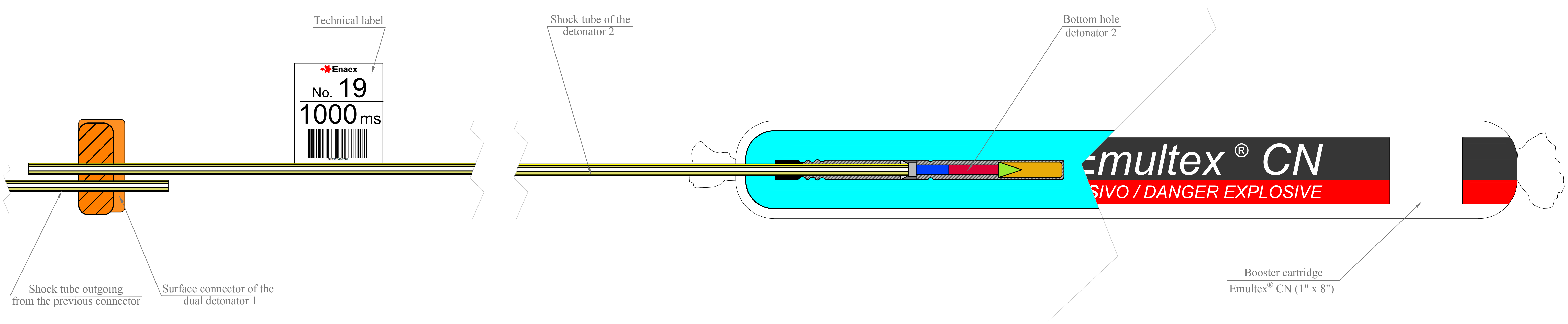
Detail of the junction booster - bottom hole detonator - surface connector (dual detonator system) - Hole 1
Scale: 1:1

Enaex 1

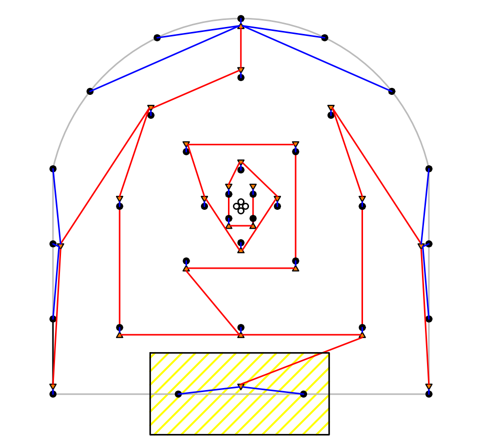


Detail of the junction booster - bottom hole detonator - surface connector (dual detonator system) - Hole 2
Scale: 1:1

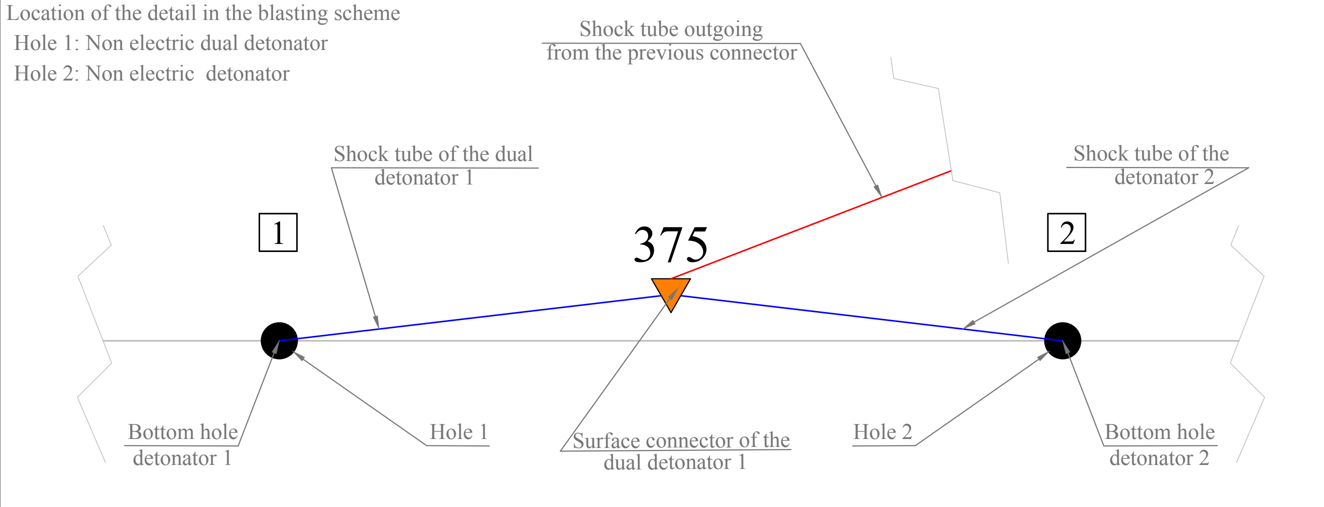
Enaex 2



Location of the detail in the blasting scheme
Scale: 1:50



Location of the detail in the blasting scheme
Hole 1: Non electric dual detonator
Hole 2: Non electric detonator



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
DETAIL OF NONEL CONNECTION BETWEEN HOLES OF THE CONTOUR (FLOOR)
Scale: 1:1

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu
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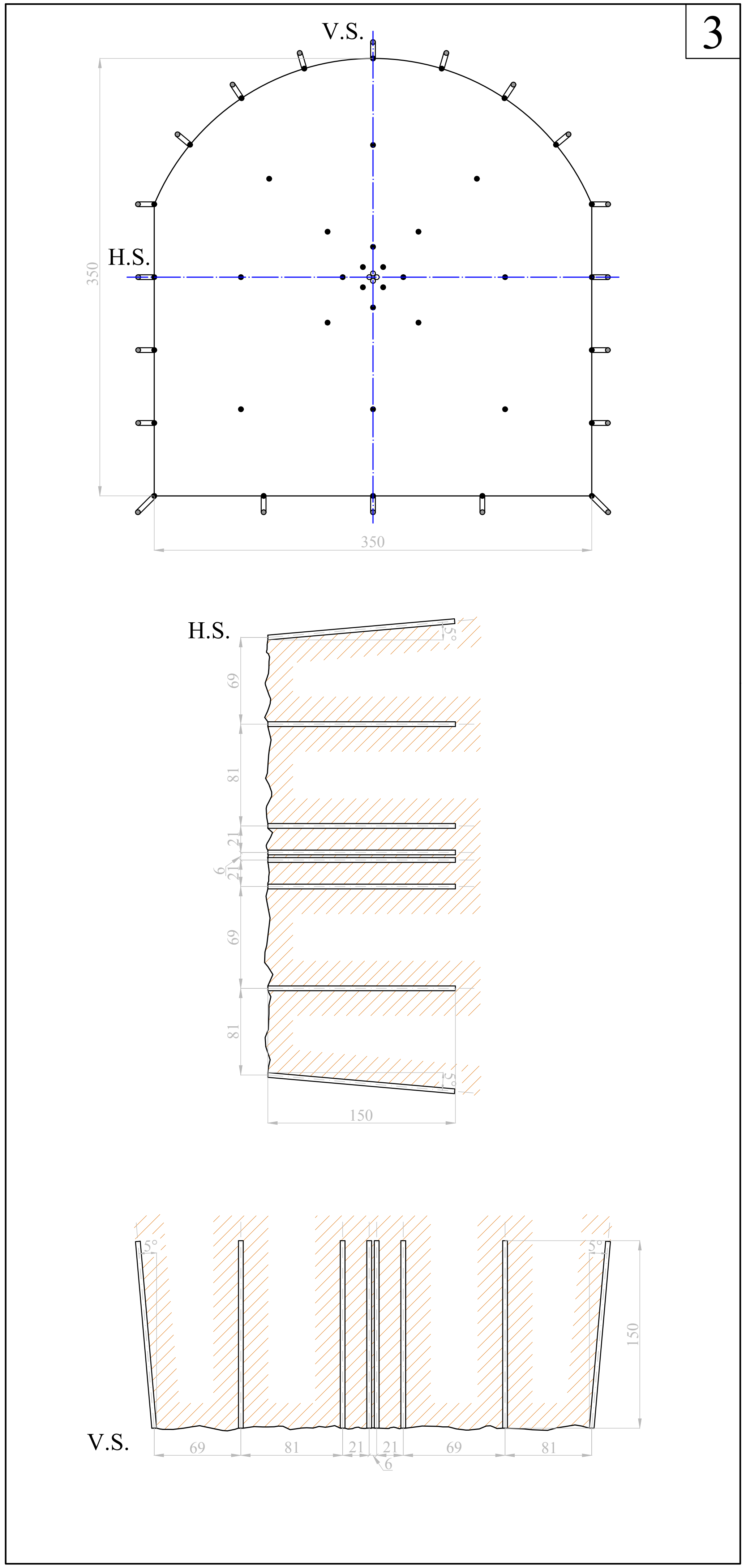

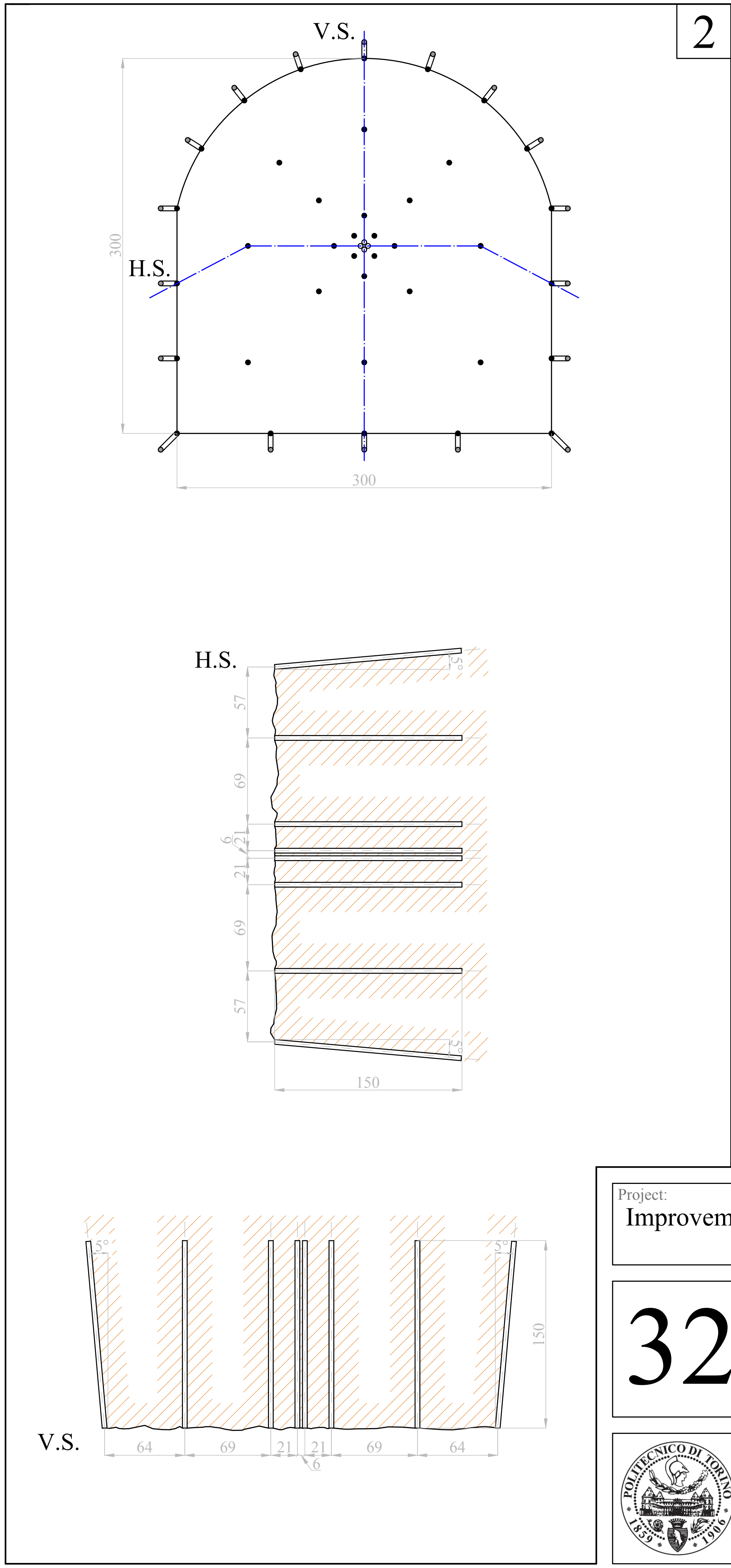

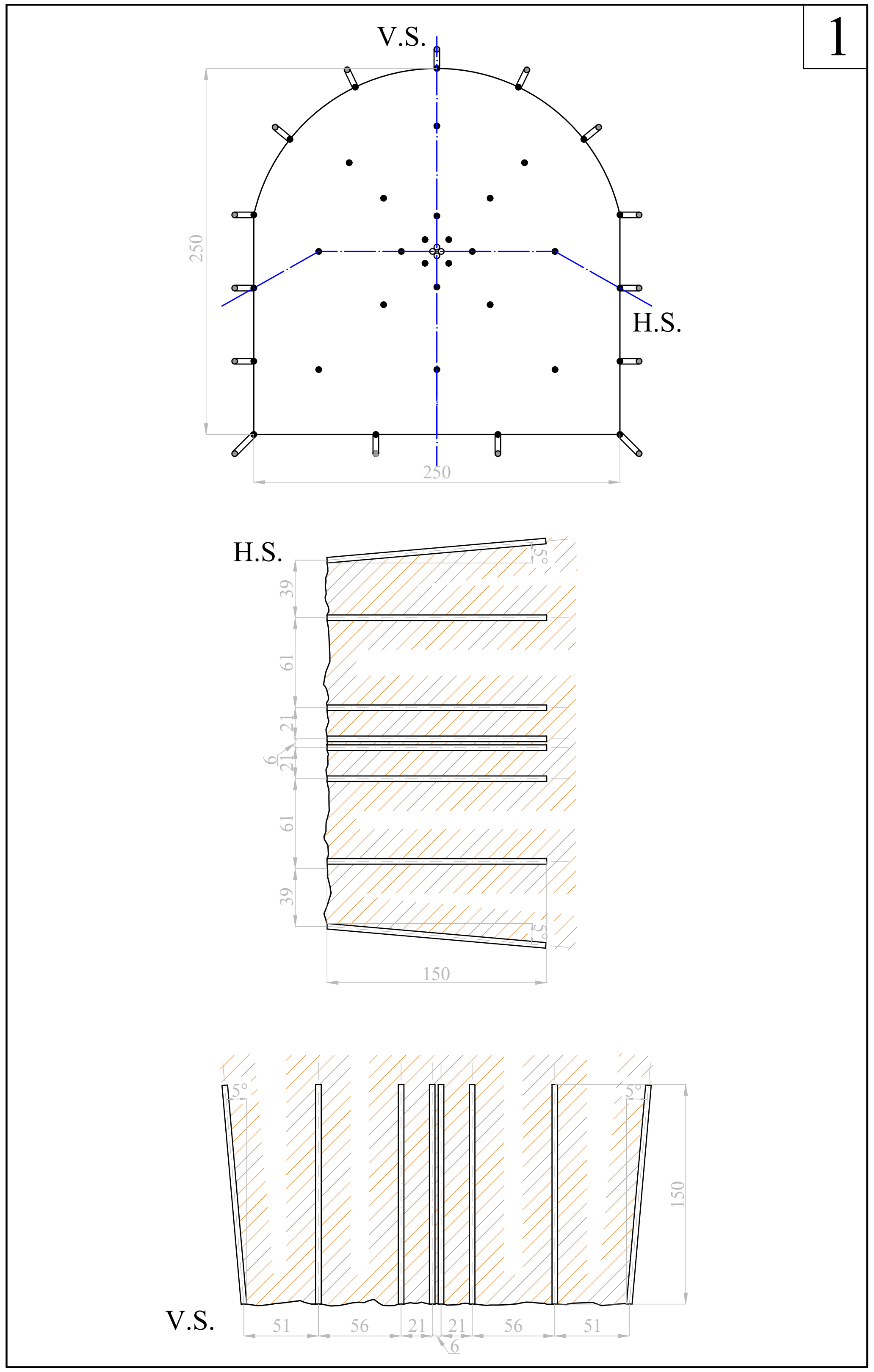

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:

PHASE 2: SECTIONS AND DEFINITION OF THE LOOK-OUT ANGLE

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

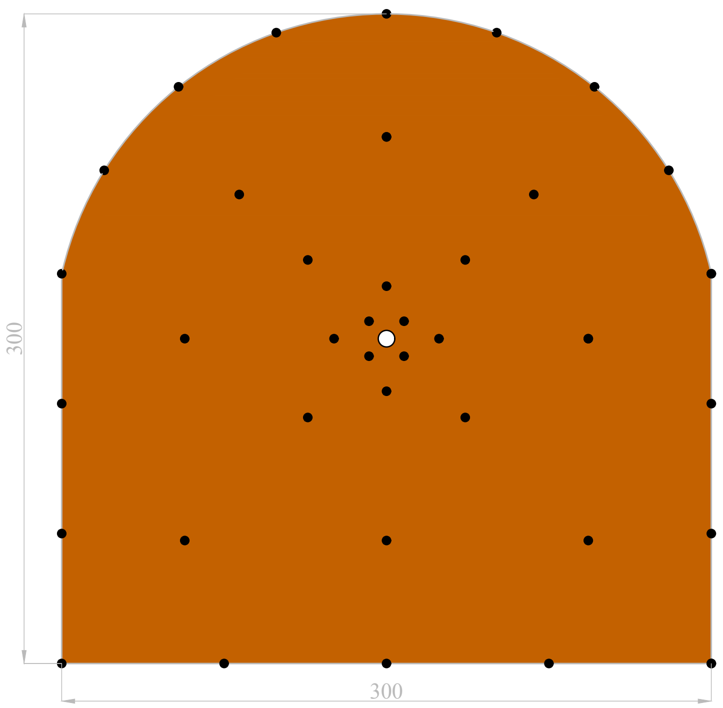
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Instant of the detonation sequence at 1000 ms

1

Empty central hole volume (considering the fictitious diameter): 0.007 m^3
Rock volume blasted at this stage: 0 m^3
Rock volume blasted until now: 0 m^3

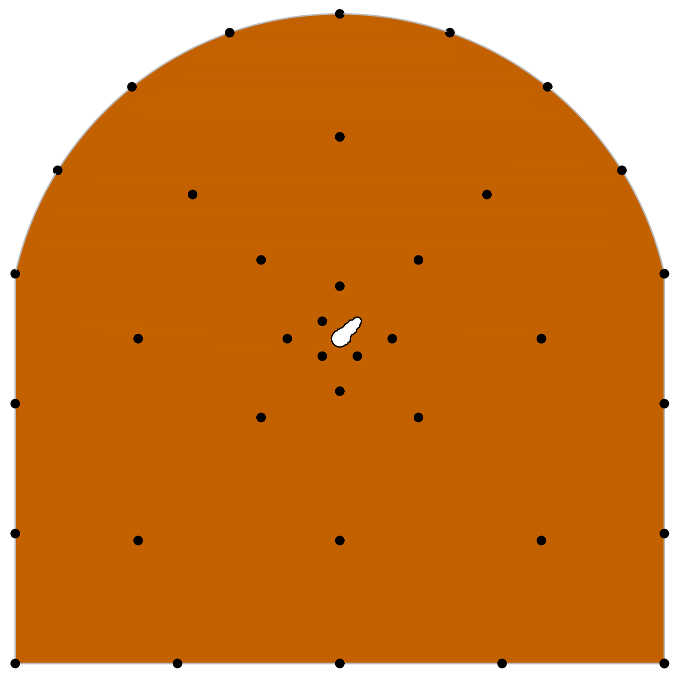


Charge per delay: 0 kg

Instant of the detonation sequence at 1025 ms

2

Rock volume blasted at this stage: 0.007 m^3
Rock volume blasted until now: 0.007 m^3

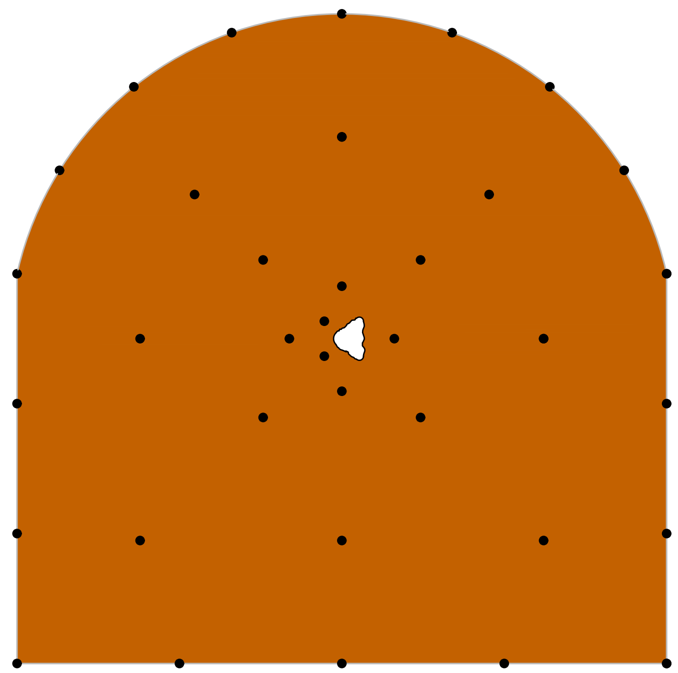


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1050 ms

3

Rock volume blasted at this stage: 0.014 m^3
Rock volume blasted until now: 0.021 m^3

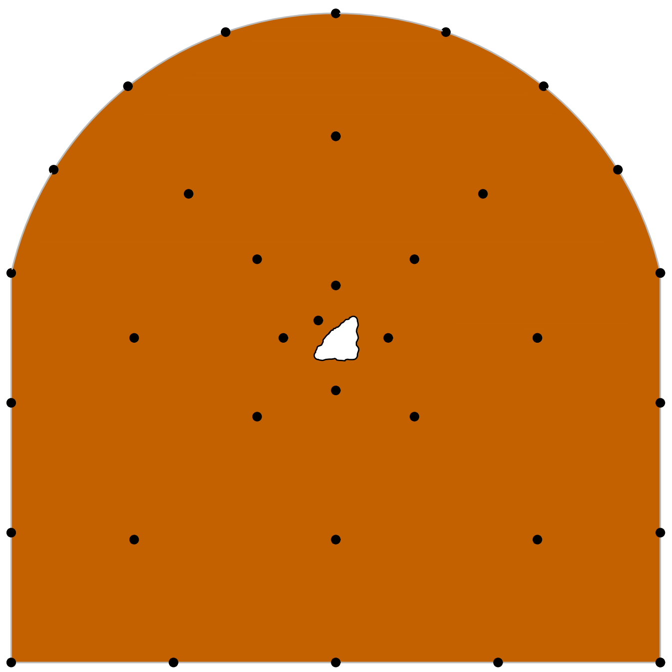


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1075 ms

4

Rock volume blasted at this stage: 0.013 m^3
Rock volume blasted until now: 0.034 m^3

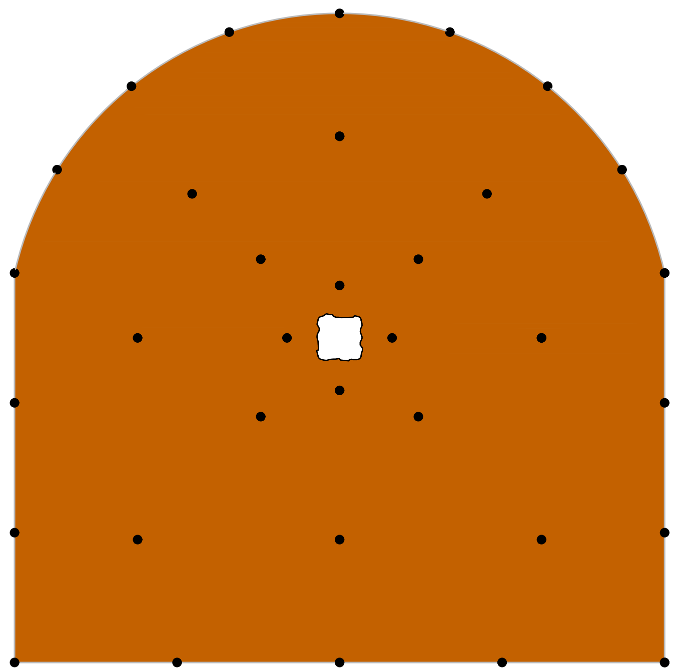


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1100 ms

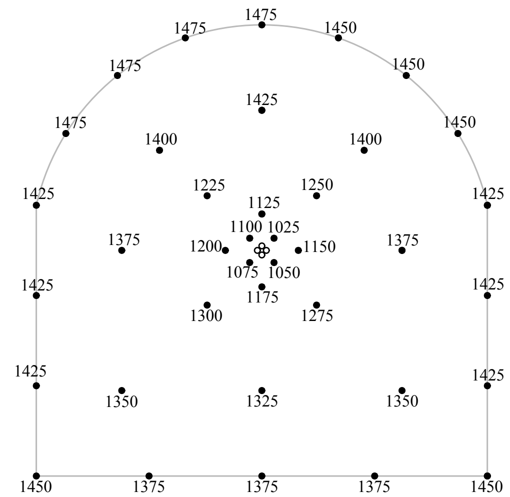
5

Rock volume blasted at this stage: 0.019 m^3
Rock volume blasted until now: 0.053 m^3



Charge per delay: 1.181 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (1)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

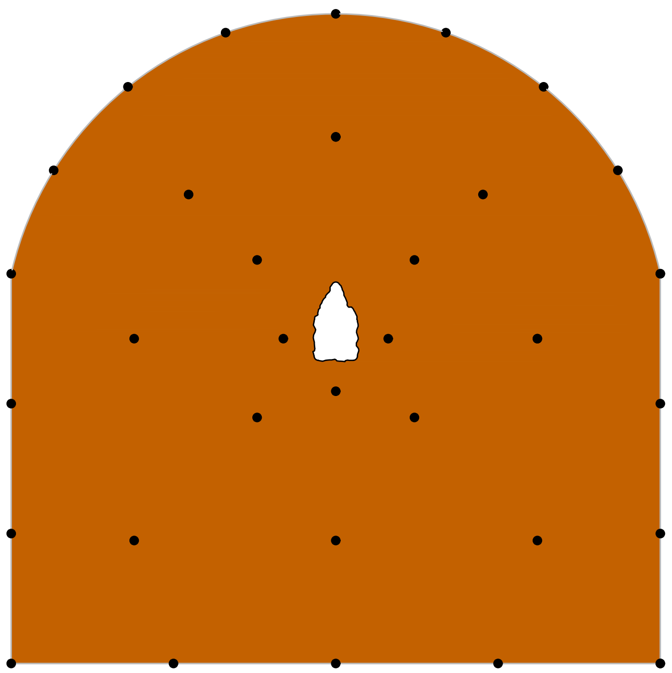
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Instant of the detonation sequence at 1125 ms

6

Rock volume blasted at this stage: 0.025 m^3
Rock volume blasted until now: 0.078 m^3

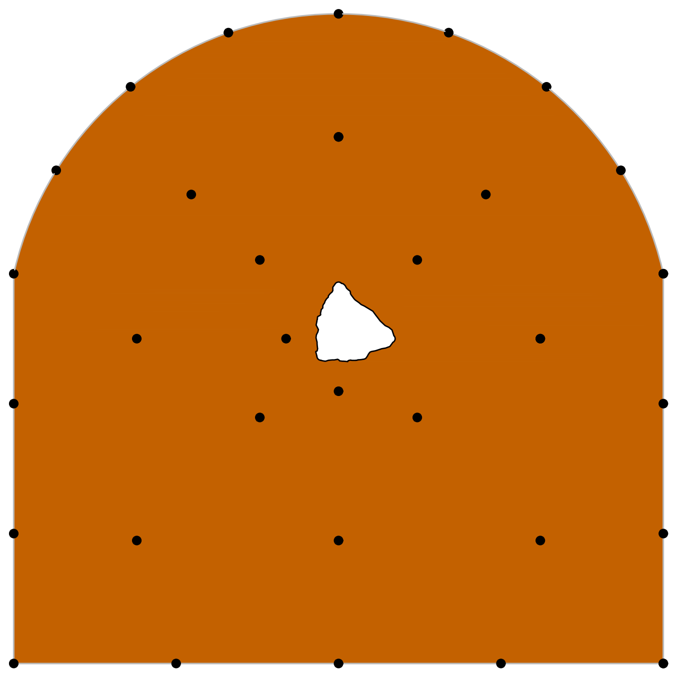


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1150 ms

7

Rock volume blasted at this stage: 0.042 m^3
Rock volume blasted until now: 0.120 m^3

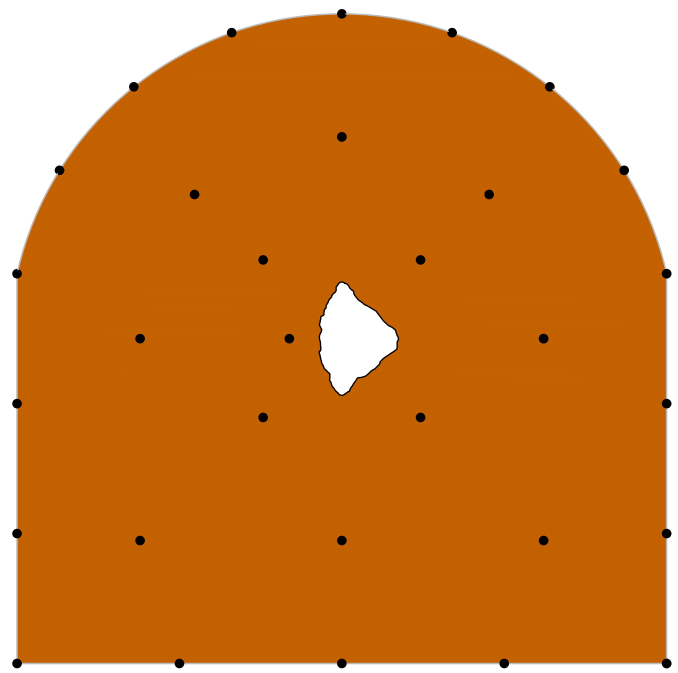


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1175 ms

8

Rock volume blasted at this stage: 0.044 m^3
Rock volume blasted until now: 0.164 m^3

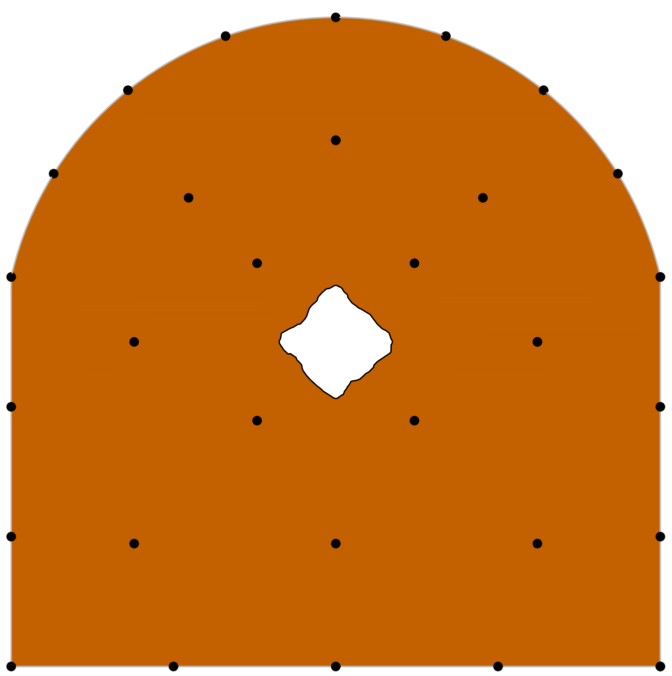


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1200 ms

9

Rock volume blasted at this stage: 0.056 m^3
Rock volume blasted until now: 0.220 m^3

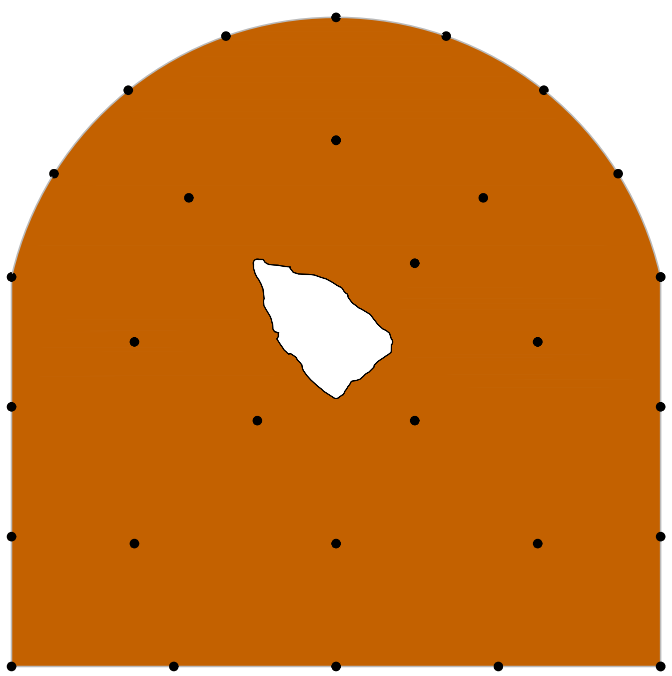


Charge per delay: 1.181 kg

Instant of the detonation sequence at 1225 ms

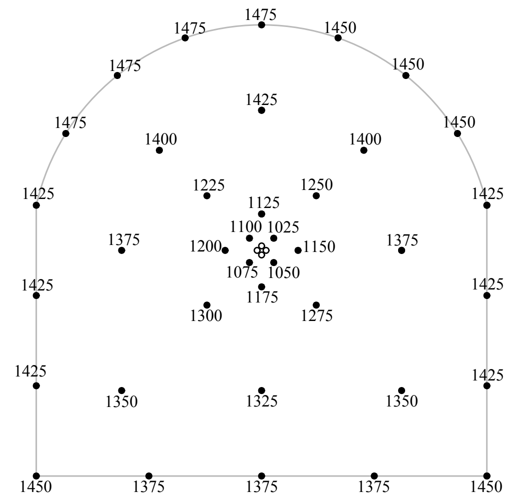
10

Rock volume blasted at this stage: 0.103 m^3
Rock volume blasted until now: 0.323 m^3



Charge per delay: 1.181 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (2)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

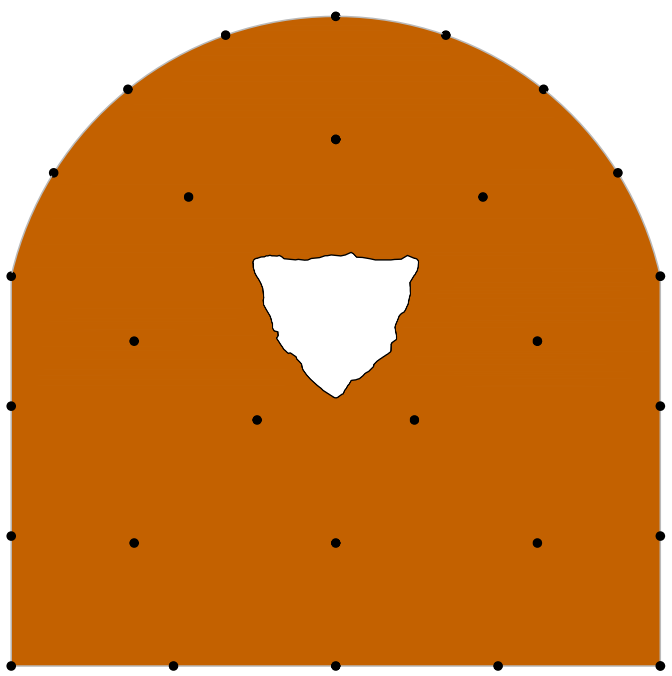
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

11

Instant of the detonation sequence at 1250 ms

Rock volume blasted at this stage: 0.167 m^3
Rock volume blasted until now: 0.490 m^3

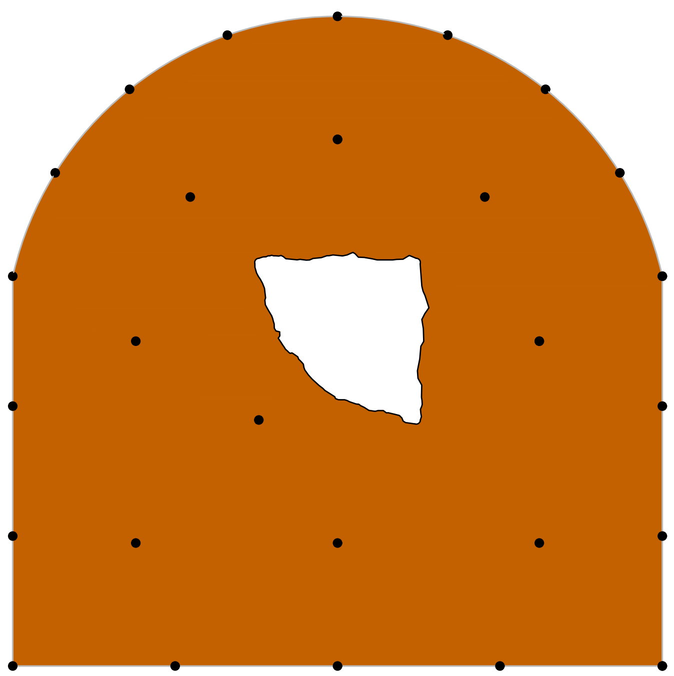


Charge per delay: 1.181 kg

12

Instant of the detonation sequence at 1275 ms

Rock volume blasted at this stage: 0.166 m^3
Rock volume blasted until now: 0.656 m^3

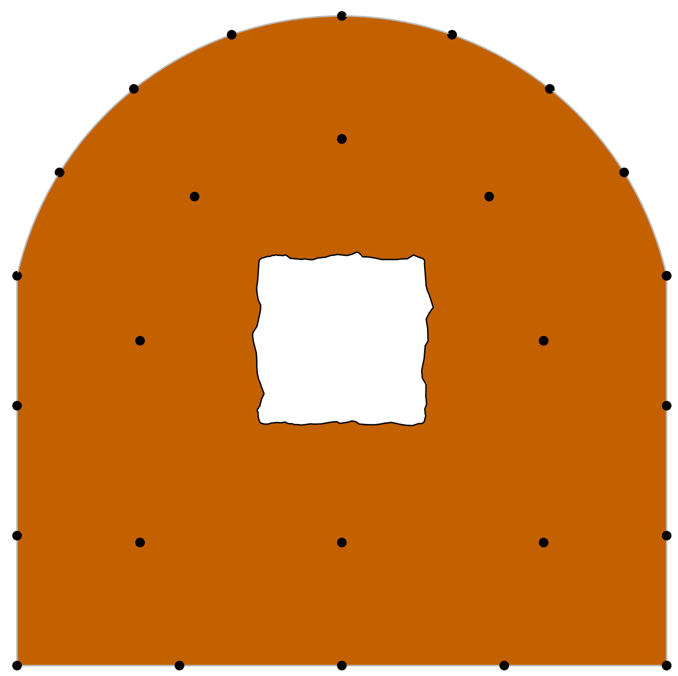


Charge per delay: 1.181 kg

13

Instant of the detonation sequence at 1300 ms

Rock volume blasted at this stage: 0.231 m^3
Rock volume blasted until now: 0.887 m^3

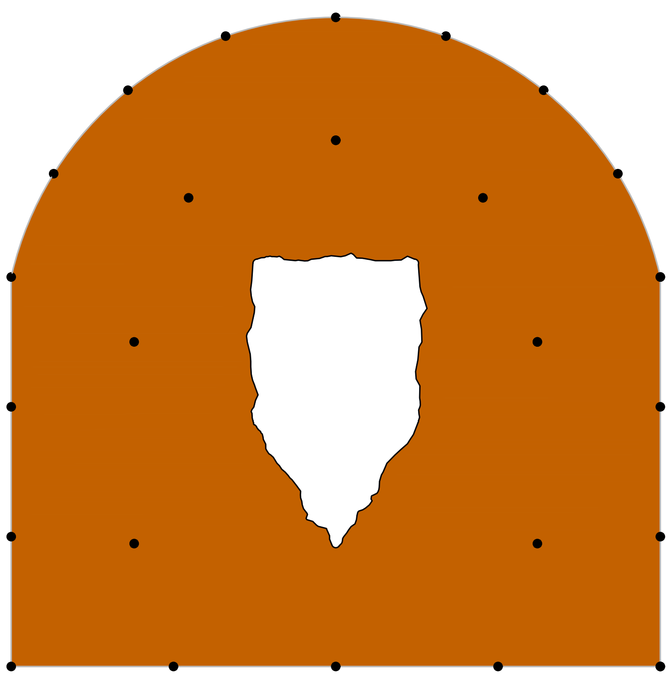


Charge per delay: 1.181 kg

14

Instant of the detonation sequence at 1325 ms

Rock volume blasted at this stage: 0.332 m^3
Rock volume blasted until now: 1.219 m^3

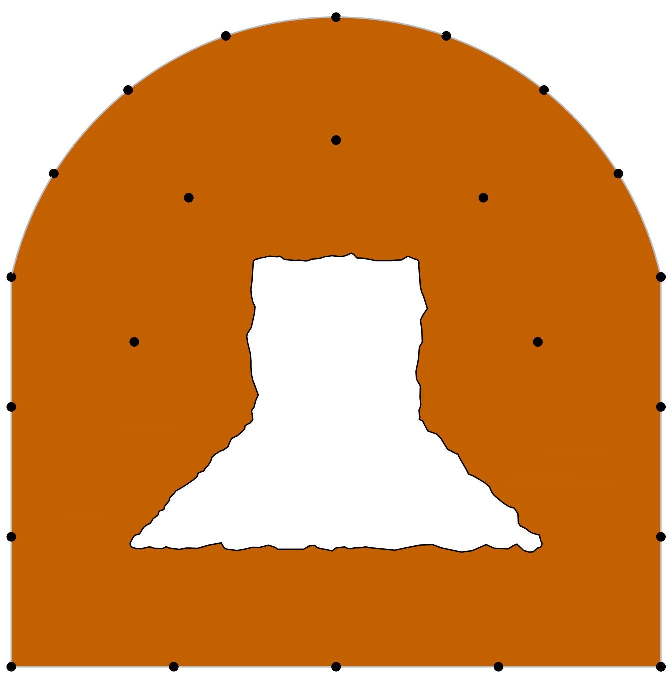


Charge per delay: 1.124 kg

15

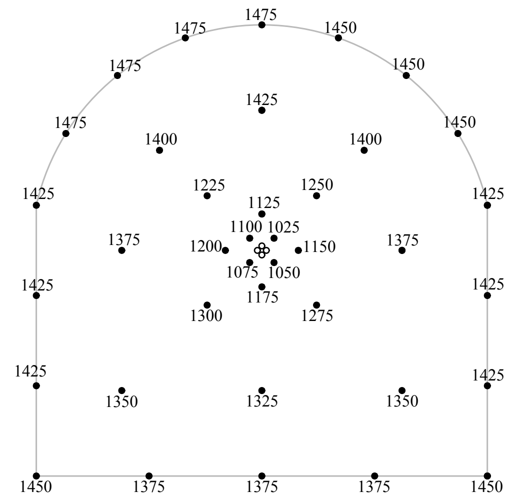
Instant of the detonation sequence at 1350 ms

Rock volume blasted at this stage: 0.857 m^3
Rock volume blasted until now: 2.076 m^3



Charge per delay: 2.248 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (3)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

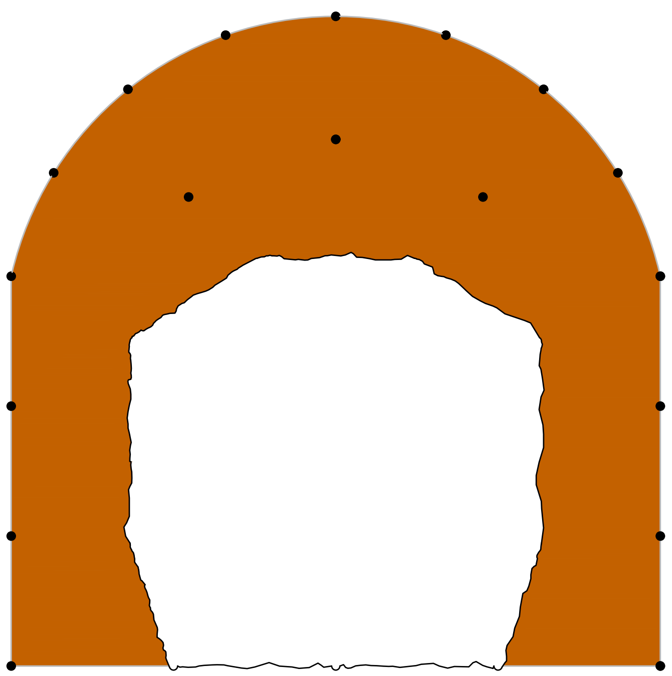
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Instant of the detonation sequence at 1375 ms

16

Rock volume blasted at this stage: 2.827 m^3
Rock volume blasted until now: 4.903 m^3

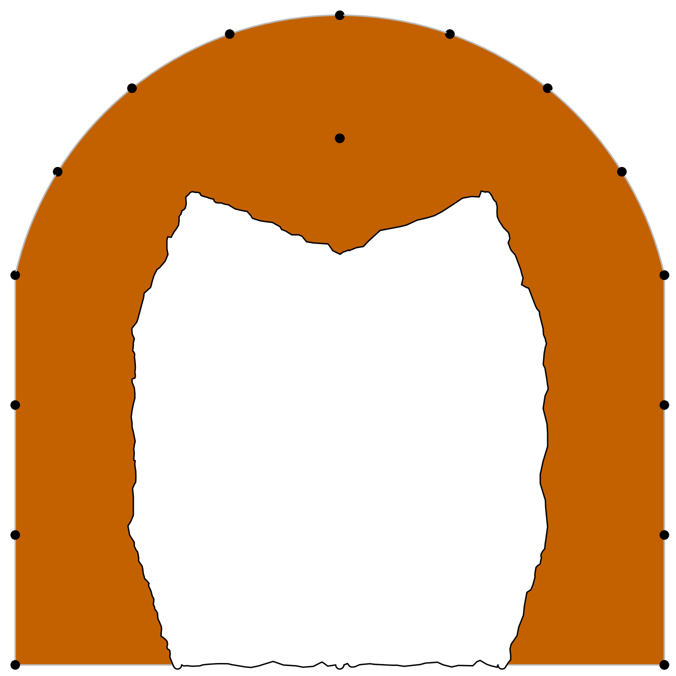


Charge per delay: 6.148 kg

Instant of the detonation sequence at 1400 ms

17

Rock volume blasted at this stage: 0.643 m^3
Rock volume blasted until now: 5.546 m^3

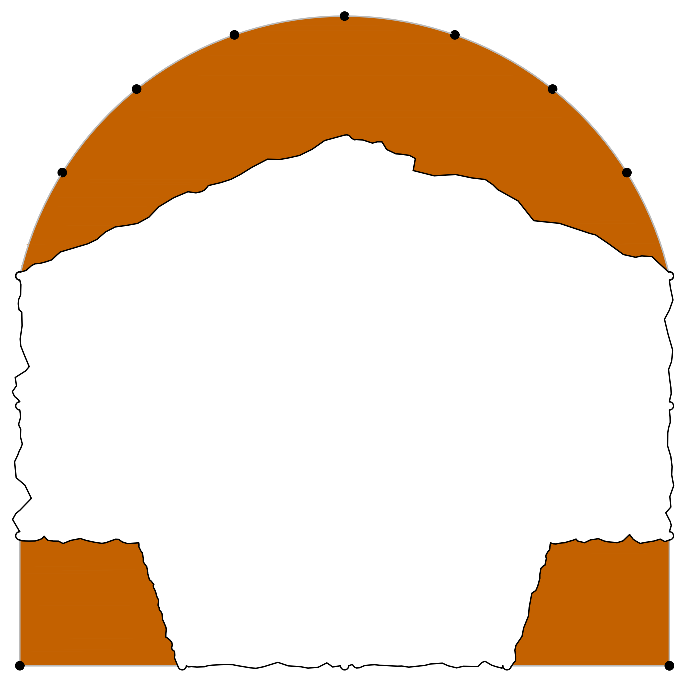


Charge per delay: 2.248 kg

Instant of the detonation sequence at 1425 ms

18

Rock volume blasted at this stage: 3.029 m^3
Rock volume blasted until now: 8.575 m^3

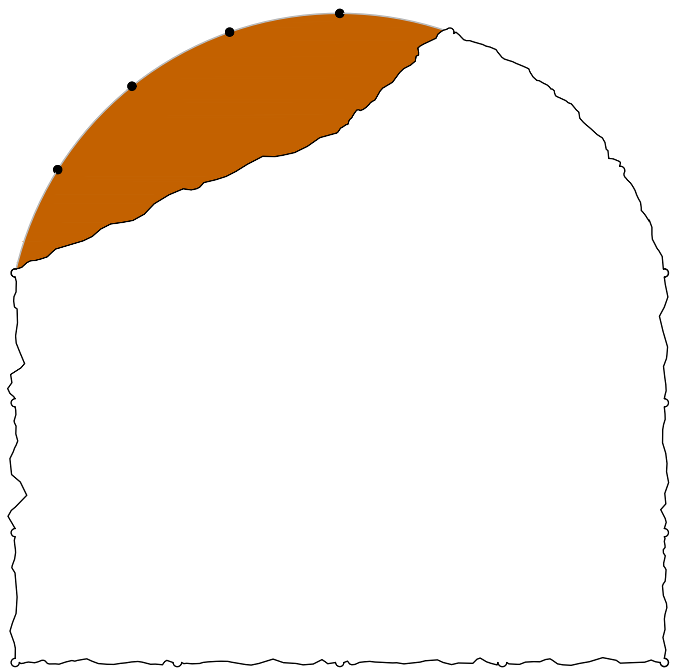


Charge per delay: 1.844 kg

Instant of the detonation sequence at 1450 ms

19

Rock volume blasted at this stage: 2.126 m^3
Rock volume blasted until now: 10.701 m^3

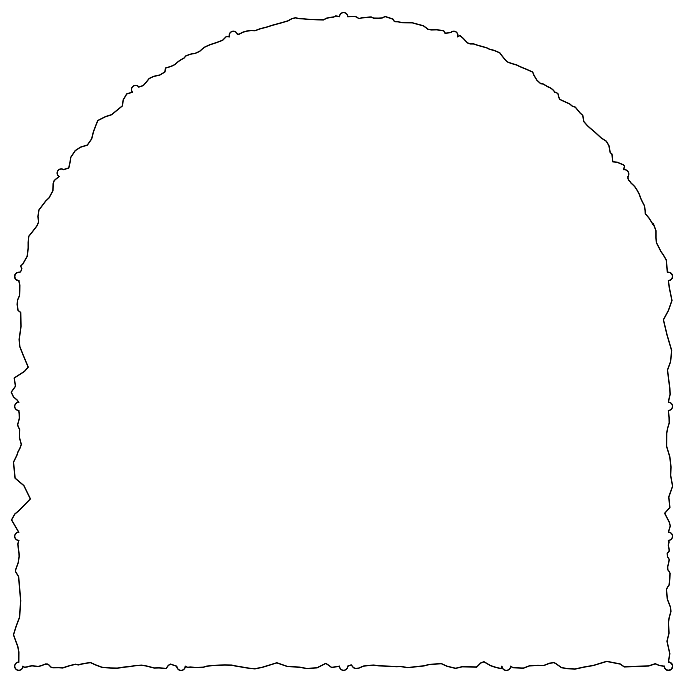


Charge per delay: 2.960 kg

Instant of the detonation sequence at 1475 ms

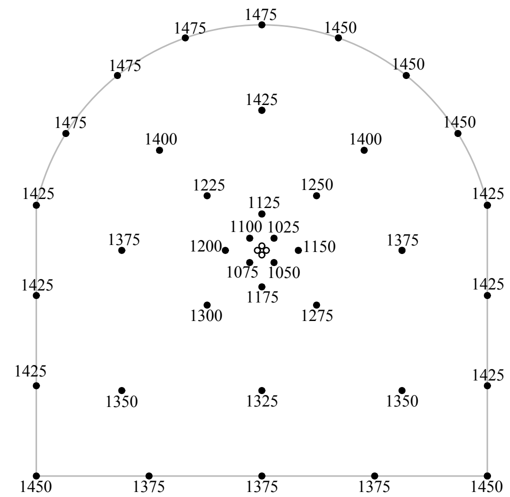
20

Rock volume blasted at this stage: 1.426 m^3
Rock volume blasted until now: 12.127 m^3



Charge per delay: 0.480 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 2: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (4)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

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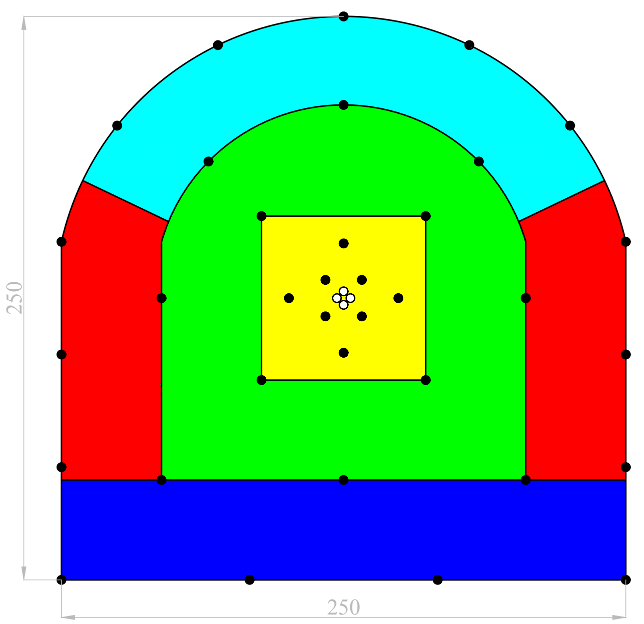
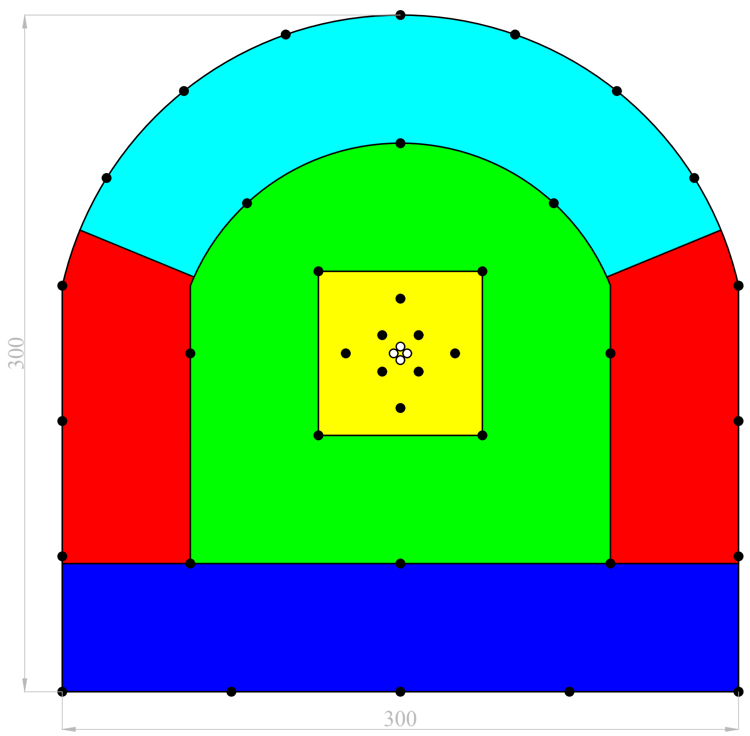


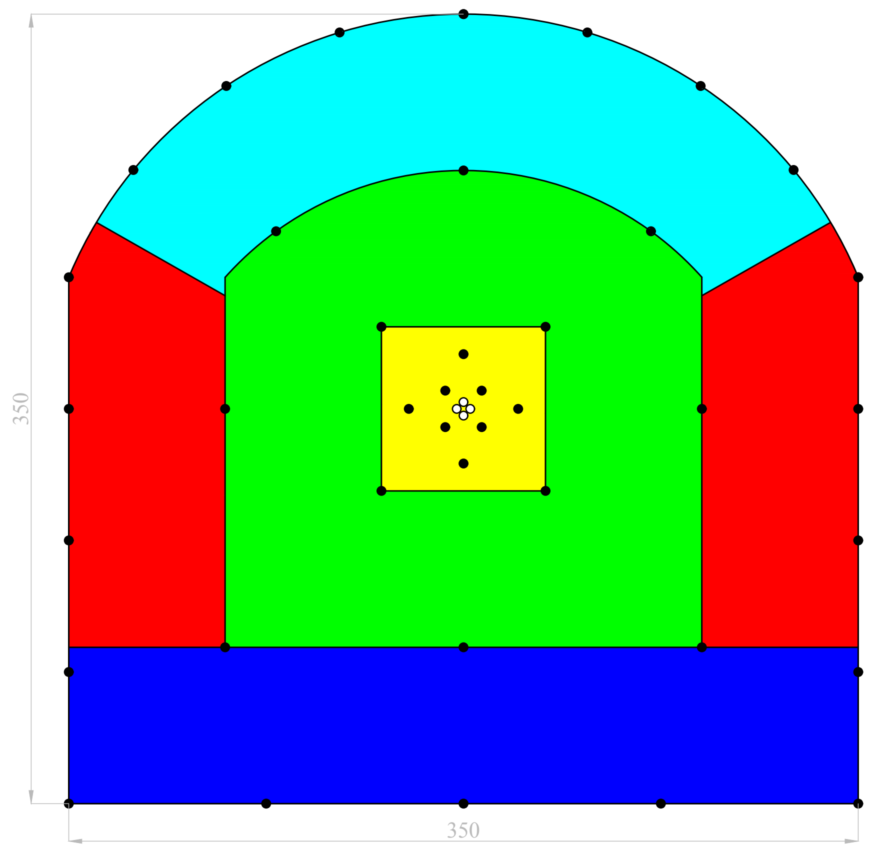
Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	2.863
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	3.170
Contour (floor) holes	Volume blasted (m^3)	1.657
	Amount of explosive (kg)	5.200
	Powder factor (P.F.) (kg/m^3)	3.134
Contour (walls) holes	Volume blasted (m^3)	1.642
	Amount of explosive (kg)	0.720
	Powder factor (P.F.) (kg/m^3)	0.438
Contour (roof) holes	Volume blasted (m^3)	1.461
	Amount of explosive (kg)	0.600
	Powder factor (P.F.) (kg/m^3)	0.411
Volume blasted by contour holes (walls + roof) (m^3)		3.103
Amount of explosive for the contour holes (walls + roof) (kg)		1.320
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.425
Volume blasted by contour holes (floor + walls + roof) (m^3)		4.760
Amount of explosive for the contour holes (walls + roof) (kg)		6.520
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.370

2



Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	3.919
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	2.295
Contour (floor) holes	Volume blasted (m^3)	2.551
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	2.548
Contour (walls) holes	Volume blasted (m^3)	2.340
	Amount of explosive (kg)	0.720
	Powder factor (P.F.) (kg/m^3)	0.308
Contour (roof) holes	Volume blasted (m^3)	2.490
	Amount of explosive (kg)	0.840
	Powder factor (P.F.) (kg/m^3)	0.337
Volume blasted by contour holes (walls + roof) (m^3)		4.830
Amount of explosive for the contour holes (walls + roof) (kg)		1.560
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.323
Volume blasted by contour holes (floor + walls + roof) (m^3)		7.380
Amount of explosive for the contour holes (walls + roof) (kg)		8.060
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.092

3



Cut holes	Volume blasted (m^3)	0.711
	Amount of explosive (kg)	14.172
	Powder factor (P.F.) (kg/m^3)	18.371
Production holes	Volume blasted (m^3)	5.435
	Amount of explosive (kg)	8.992
	Powder factor (P.F.) (kg/m^3)	1.654
Contour (floor) holes	Volume blasted (m^3)	3.631
	Amount of explosive (kg)	6.500
	Powder factor (P.F.) (kg/m^3)	1.790
Contour (walls) holes	Volume blasted (m^3)	3.598
	Amount of explosive (kg)	0.720
	Powder factor (P.F.) (kg/m^3)	0.200
Contour (roof) holes	Volume blasted (m^3)	3.192
	Amount of explosive (kg)	0.840
	Powder factor (P.F.) (kg/m^3)	0.263
Volume blasted by contour holes (walls + roof) (m^3)		6.789
Amount of explosive for the contour holes (walls + roof) (kg)		1.560
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.230
Volume blasted by contour holes (floor + walls + roof) (m^3)		10.421
Amount of explosive for the contour holes (walls + roof) (kg)		8.060
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		0.773

	1	2	3
Cross section (m^2)	5.58	8.05	11.08
Design pull (m)	1.5	1.5	1.5
Blasted volume (m^3)	8.368	12.070	16.628
Volume to be removed (m^3)	11.715	16.898	23.279
Total amount of explosive (kg)	29.684	31.224	31.224
Overall Powder Factor (P.F.) (kg/m^3)	3.547	2.587	1.878
Holes diameter (mm)	38.1	38.1	38.1

	1	2	3
Holes number (-)	41	42	44
Holes density (holes/m^2)	7.35	5.22	3.97
Total drilled length (m)	61.5	63	66
Look - out angle (°)	5	5	5
Specific drilling (S.D.) (m/m^3)	7.349	5.219	3.969
Total number of detonators (-)	61	65	67
Detonators specific consumption (D.C.) (detonators/m^3)	7.289	5.385	4.029

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
MAIN DATA ON THE PHASE 2 SCHEMES

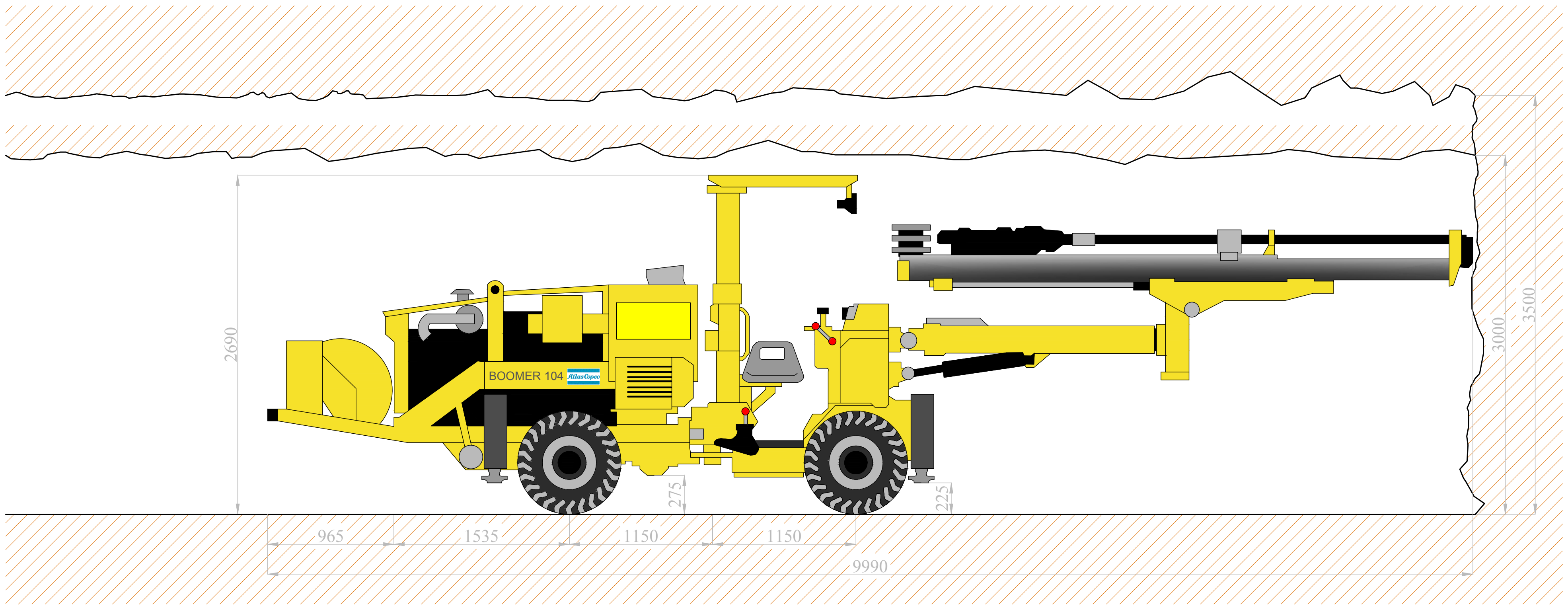
POLITECNICO DI TORINO

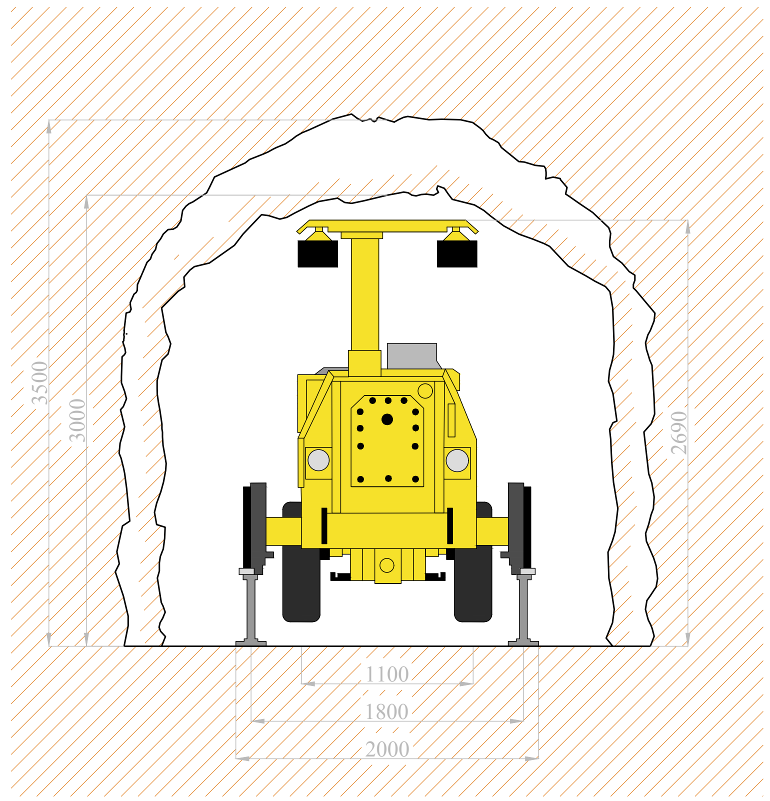
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

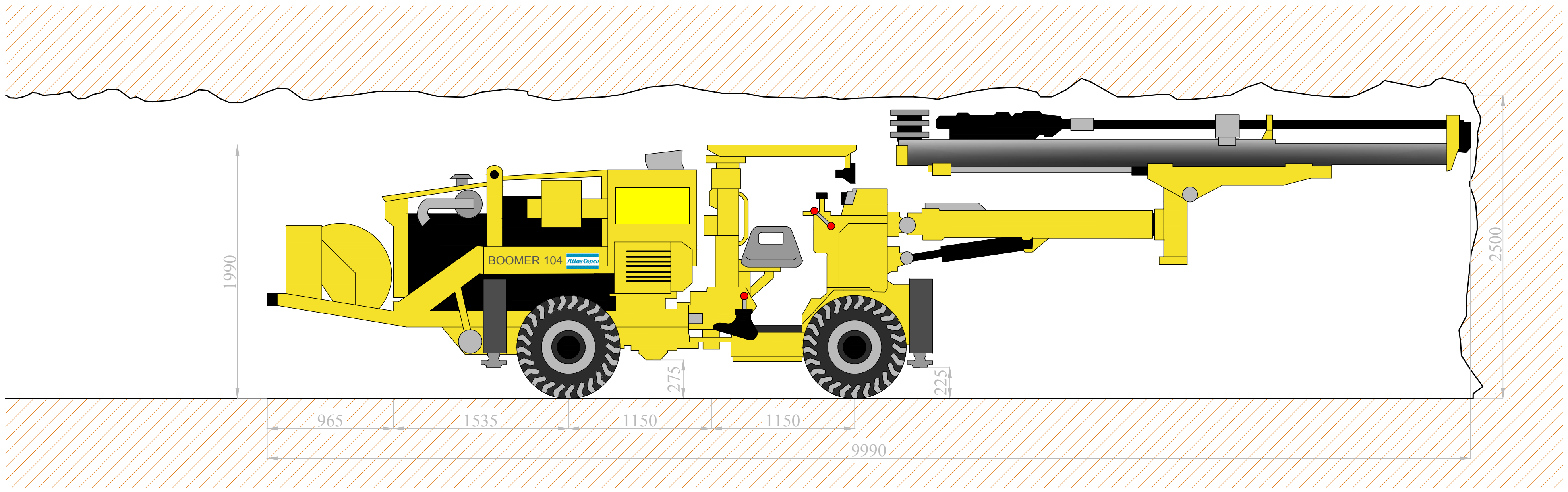
Supervisor:
Prof. Marilena Cardu

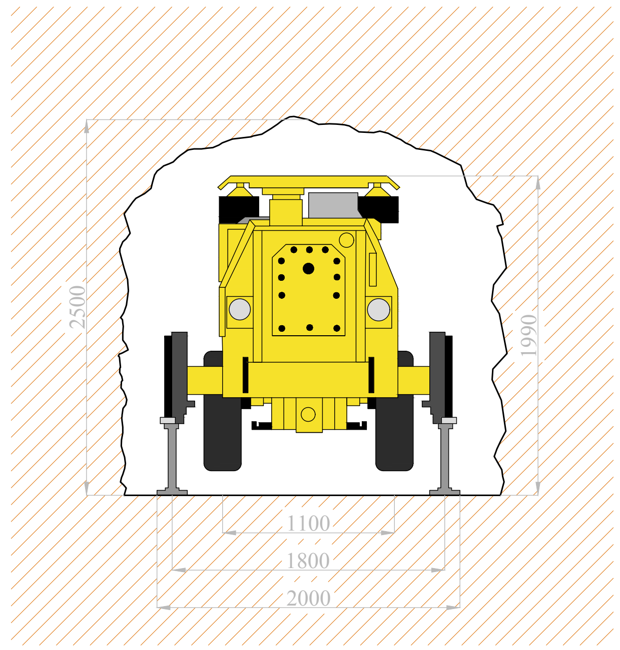
A.Y. 2018 - 2019

Size of the boomer and its positioning on the section
Scale: 1:50









Photos of the Boomer 104 Atlas Copco, available for drilling

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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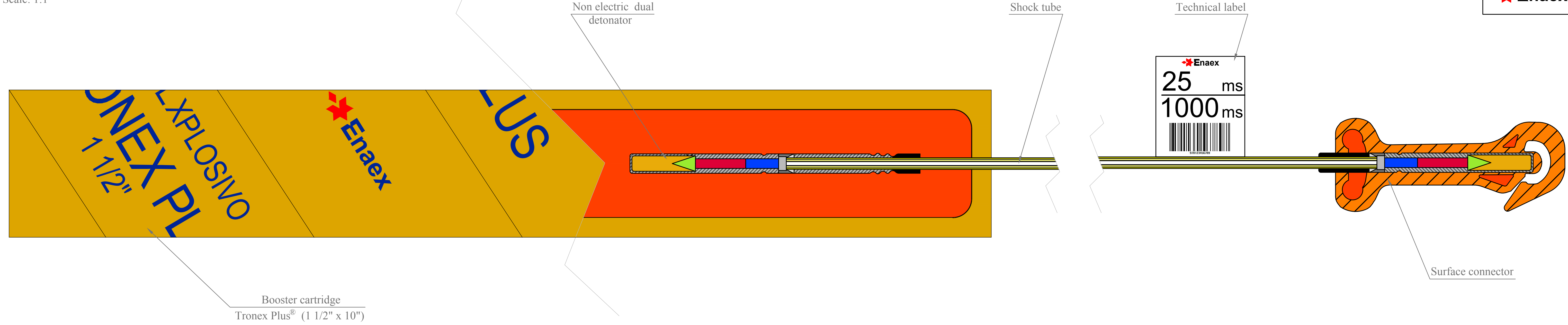
Technical drawing:
SIZE OF BOOMER AND ITS POSITIONING ON THE SECTION
Scale: 1:50

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu
A.Y. 2018 - 2019

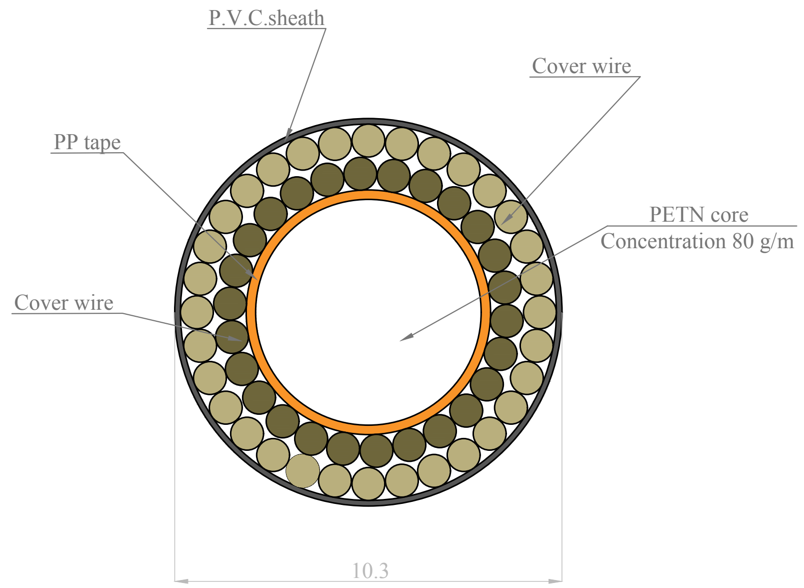
Detail of the junction booster - bottom hole detonator - surface connector (dual detonator system)
Scale: 1:1



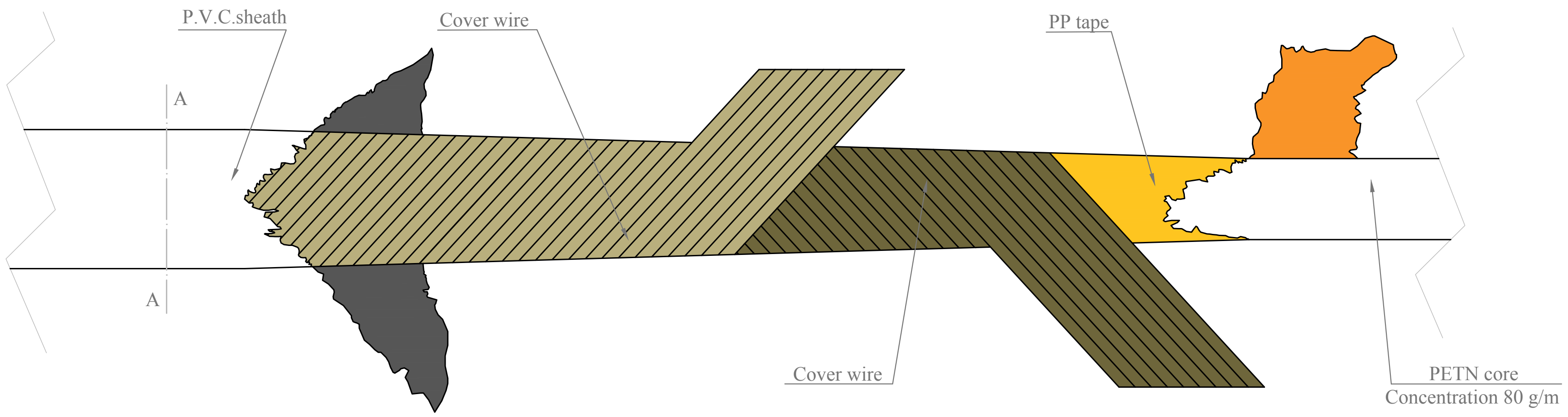
Detail of a booster cartridge
Scale: 1:1

Detail of a column charge cartridge
Scale: 1:1

Section A - A of the detonating cord used in contour (walls and roof) holes
Scale: 5:1



Detail of the detonating cord used in contour (walls and roof) holes
Scale: 2:1



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: DETAIL OF EXPLOSIVES USED

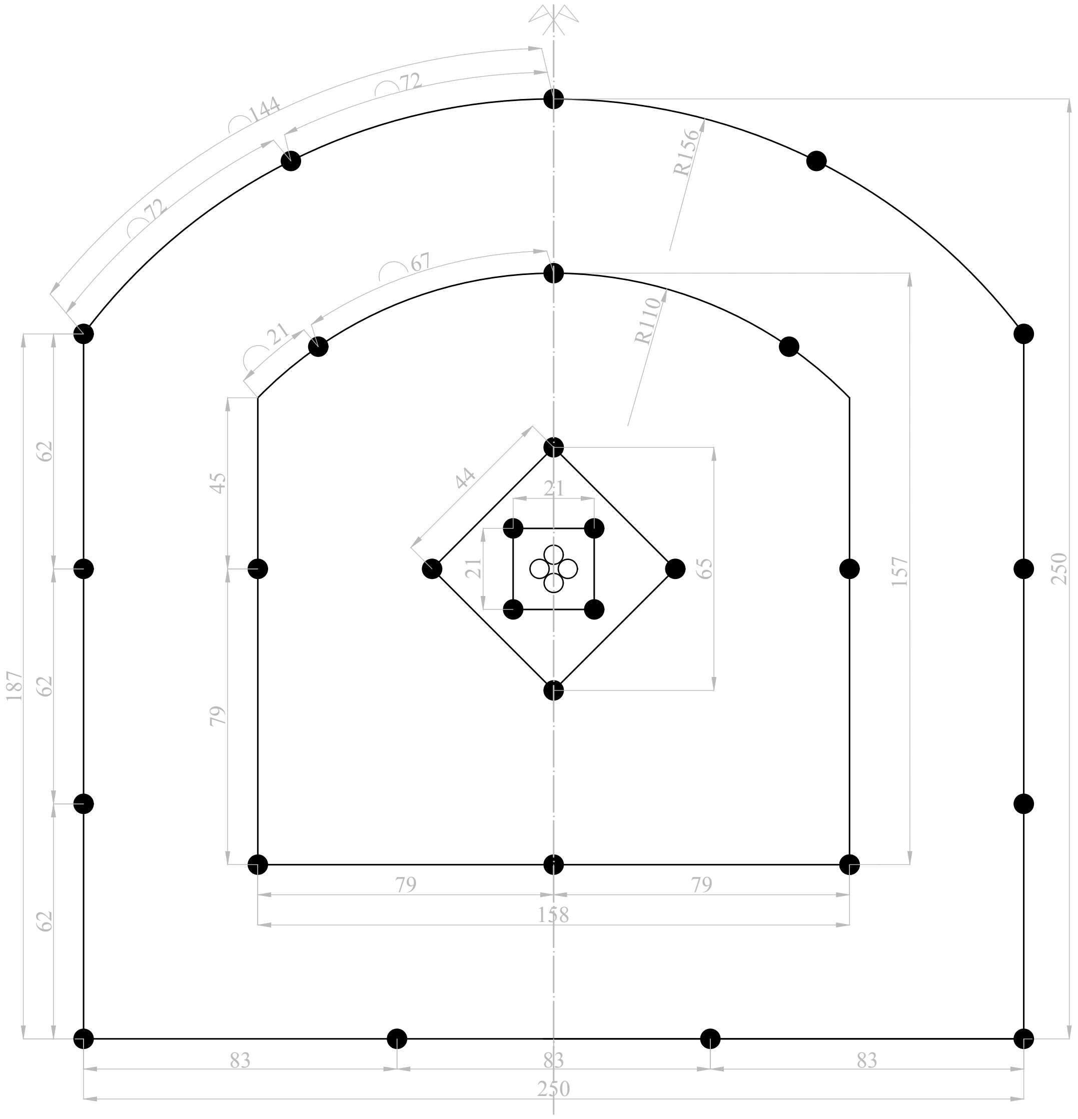
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Drilling scheme
Scale: 1:20



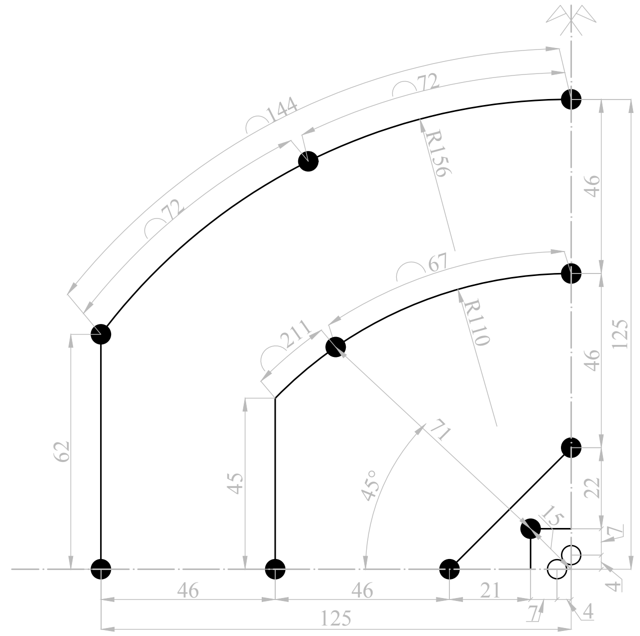
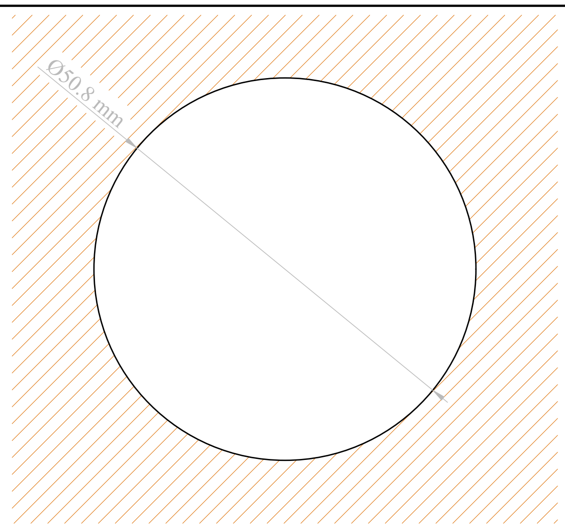
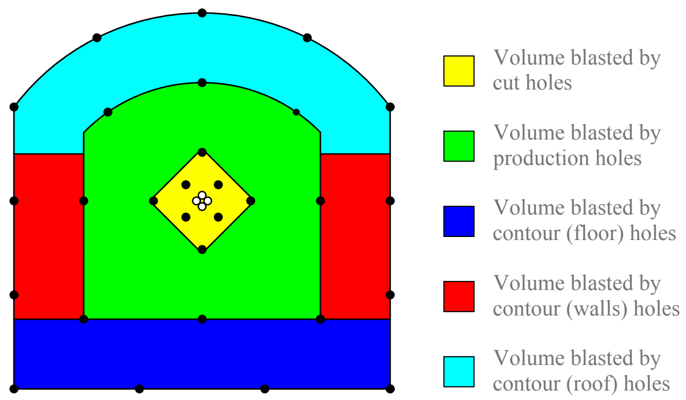


Image of the Boomer 104 Atlas Copco, available for drilling

Cross section of a hole
Scale: 1:1



Classification of the blasted volumes
Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: DRILLING SCHEME FOR SECTION 2.5 x 2.5 m
Scale: 1:20

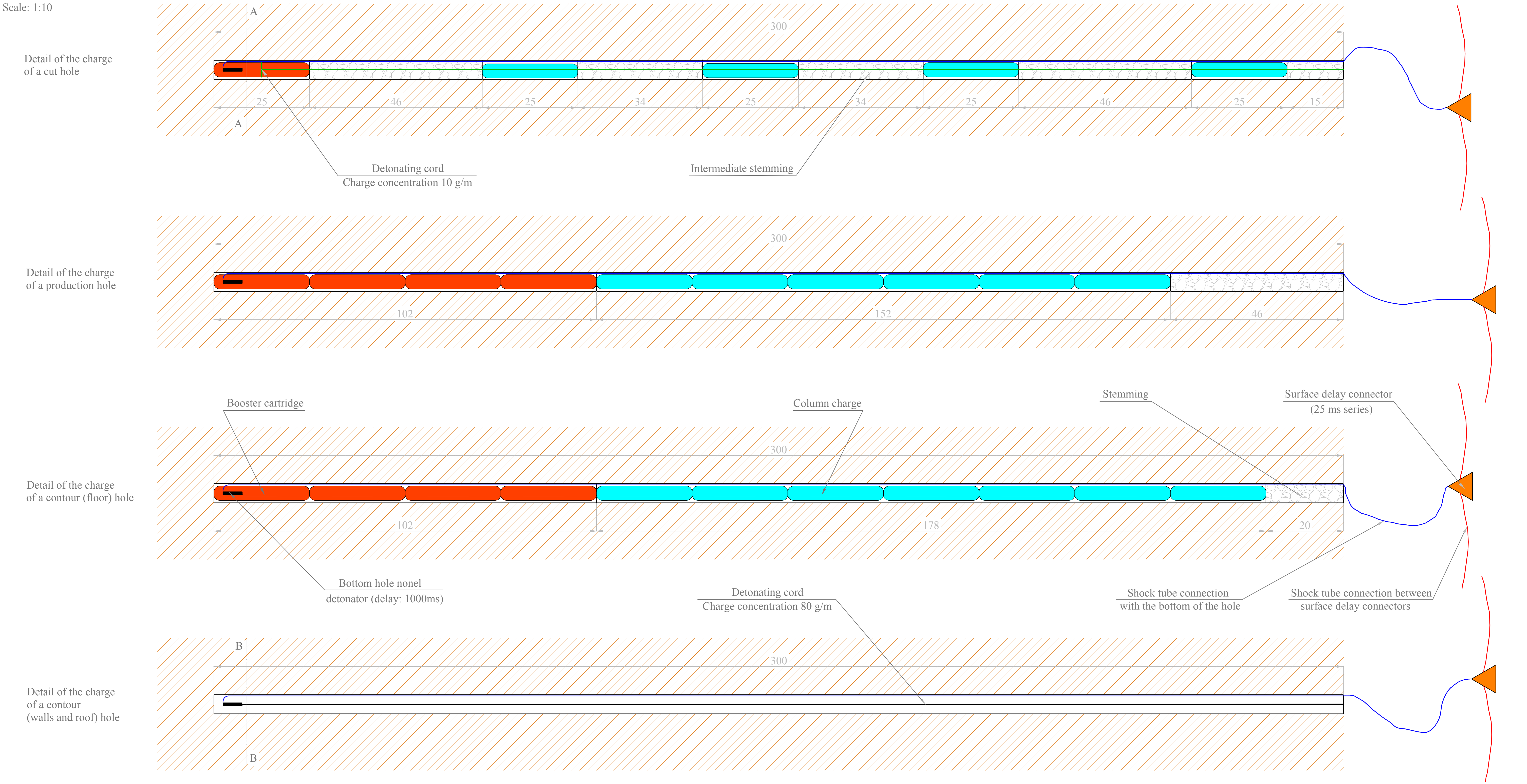
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

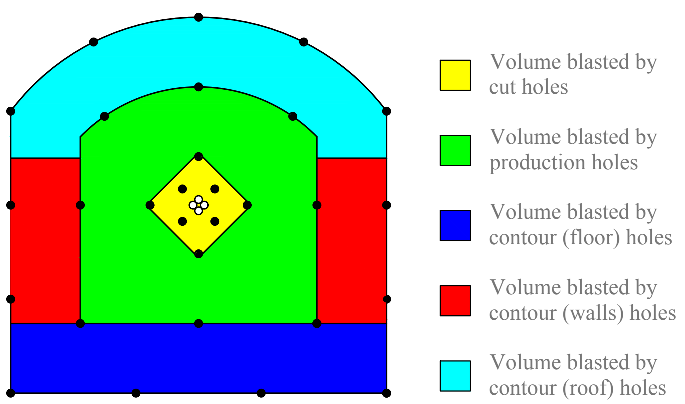
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

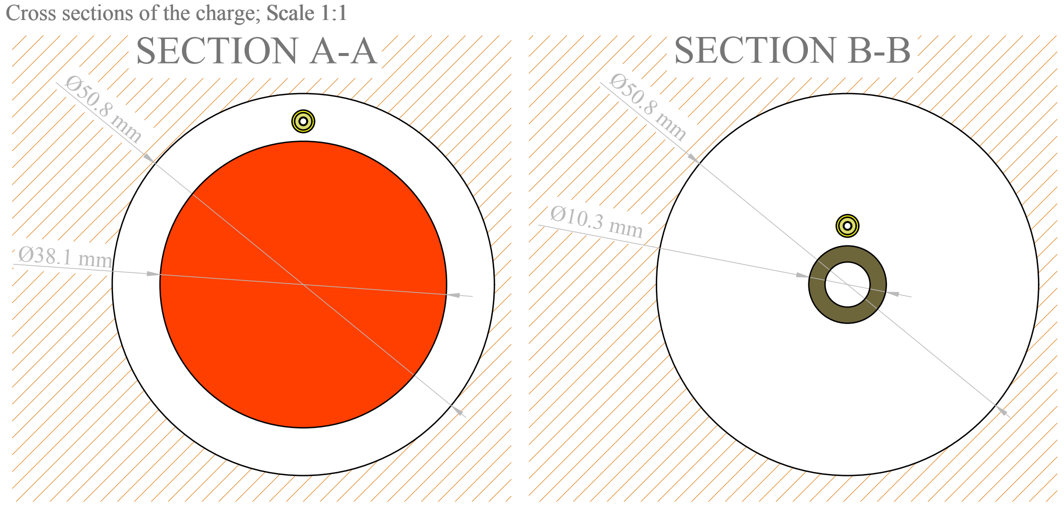
Detail of the charge
Scale: 1:10



Classification of the blasted volumes
Scale: 1:50



Cross sections of the charge; Scale 1:1



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: SCHEME OF THE CHARGE FOR SECTION 2.5 x 2.5 m
Scale: 1:10

POLITECNICO DI TORINO

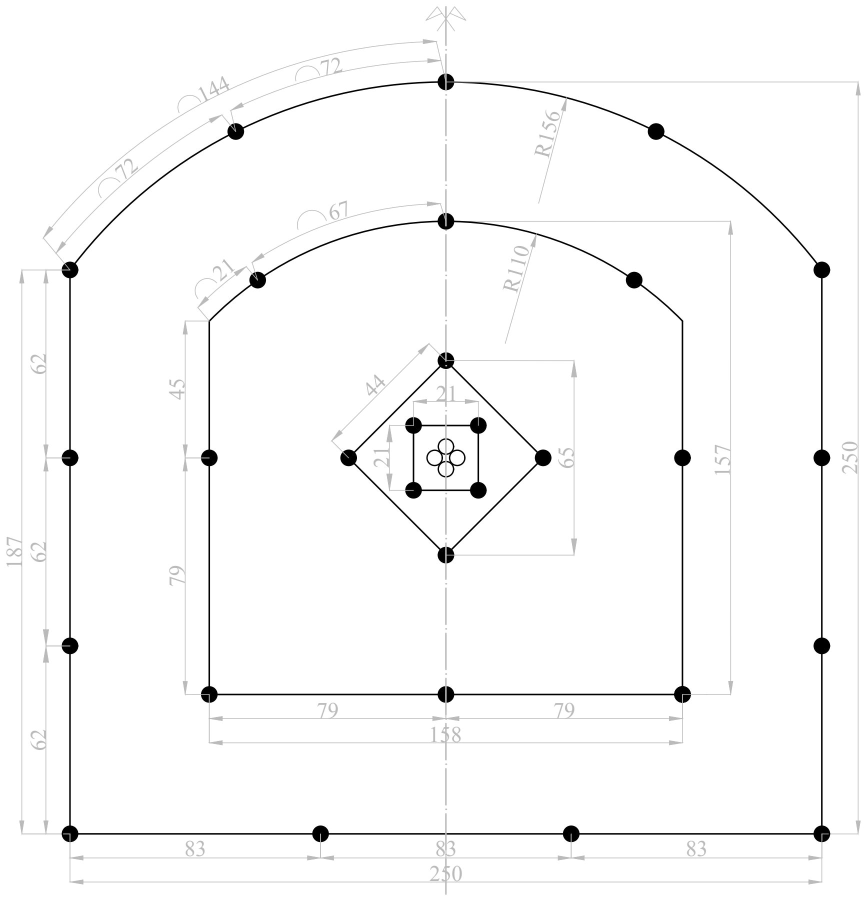
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

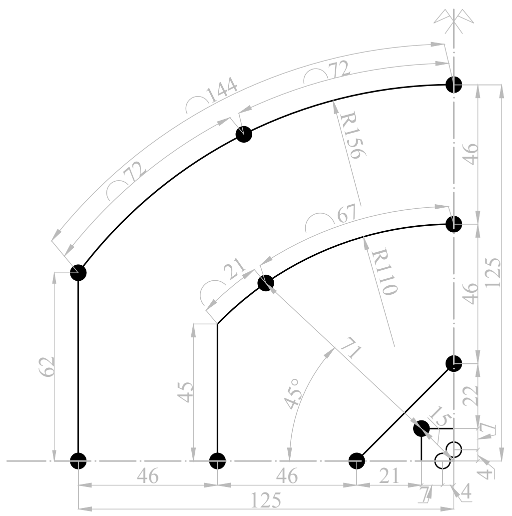
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Drilling scheme

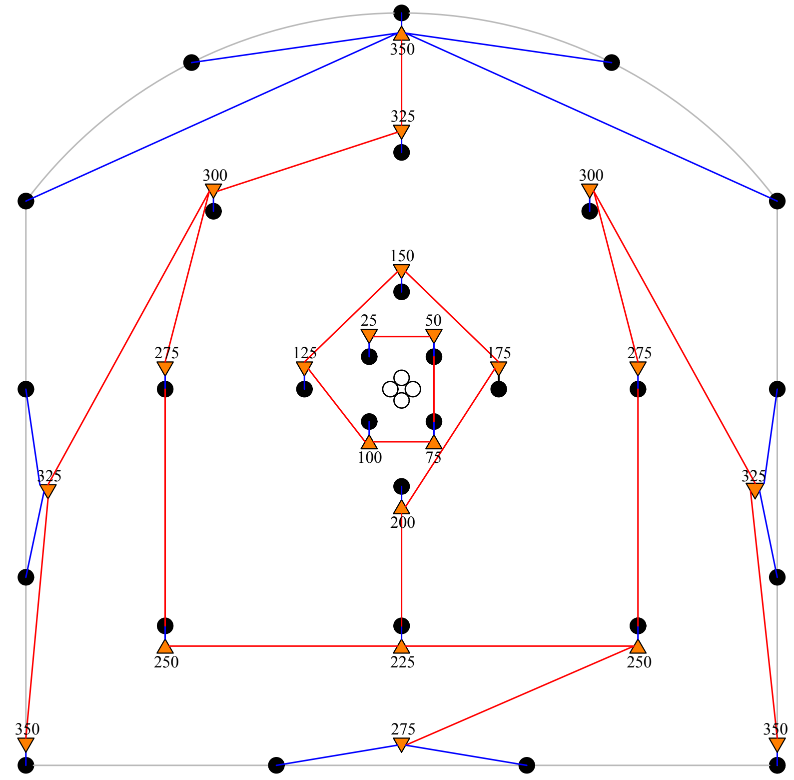
Scale: 1:25





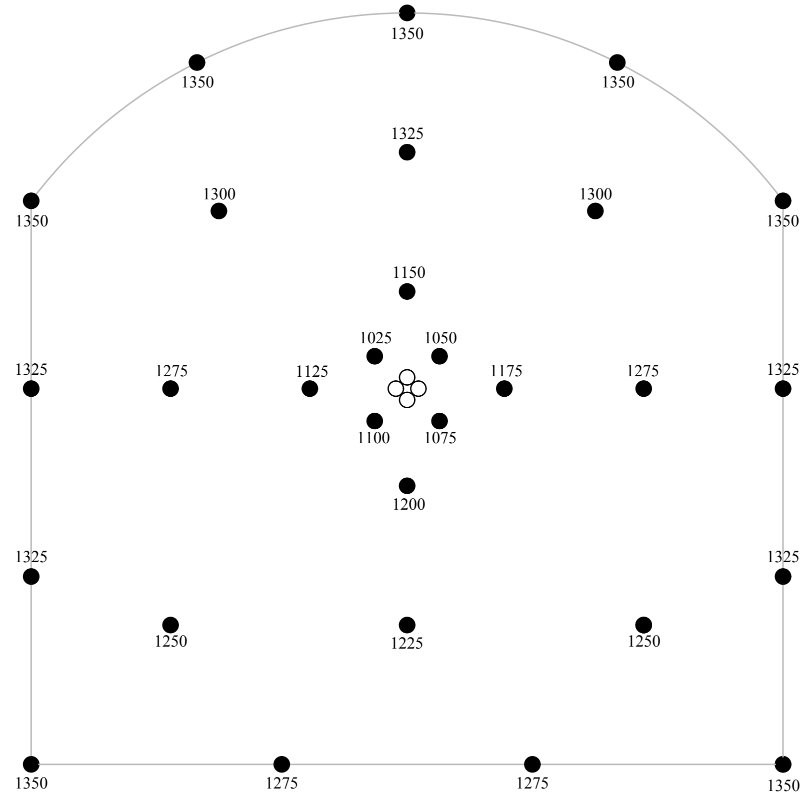
Trigger times of the surface delay connectors

Scale: 1:25



Trigger times of the in-hole detonators

Scale: 1:25

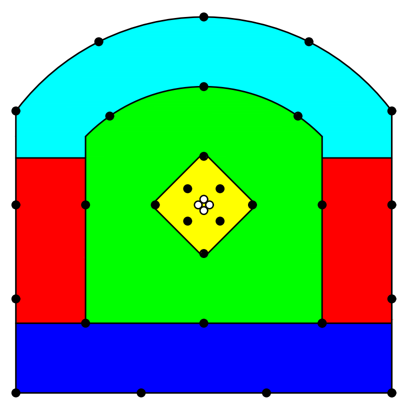


Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
8 (1200 ms)	25 ms	1000 ms	1	1
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	2	2
11 (1275 ms)	25 ms	1000 ms	3	4
12 (1300 ms)	25 ms	1000 ms	2	2
13 (1325 ms)	25 ms	1000 ms	3	5
14 (1350 ms)	25 ms	1000 ms	3	7

Classification of the blasted volumes

Scale: 1:50



Volume blasted by cut holes

Volume blasted by production holes

Volume blasted by contour (floor) holes

Volume blasted by contour (walls) holes

Volume blasted by contour (roof) holes

Photo of the non electric dual detonator

Enaex

Surface delay connector (25 ms series)

Shock tube connection with the bottom of the hole

Bottom hole nonel detonator (delay: 1000ms)

Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:

Simone BEZ

Badge Number: 236349

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Technical drawing:

PHASE 3: TRIGGER TIMES FOR SECTION 2.5 x 2.5 m

Scale: 1:25

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

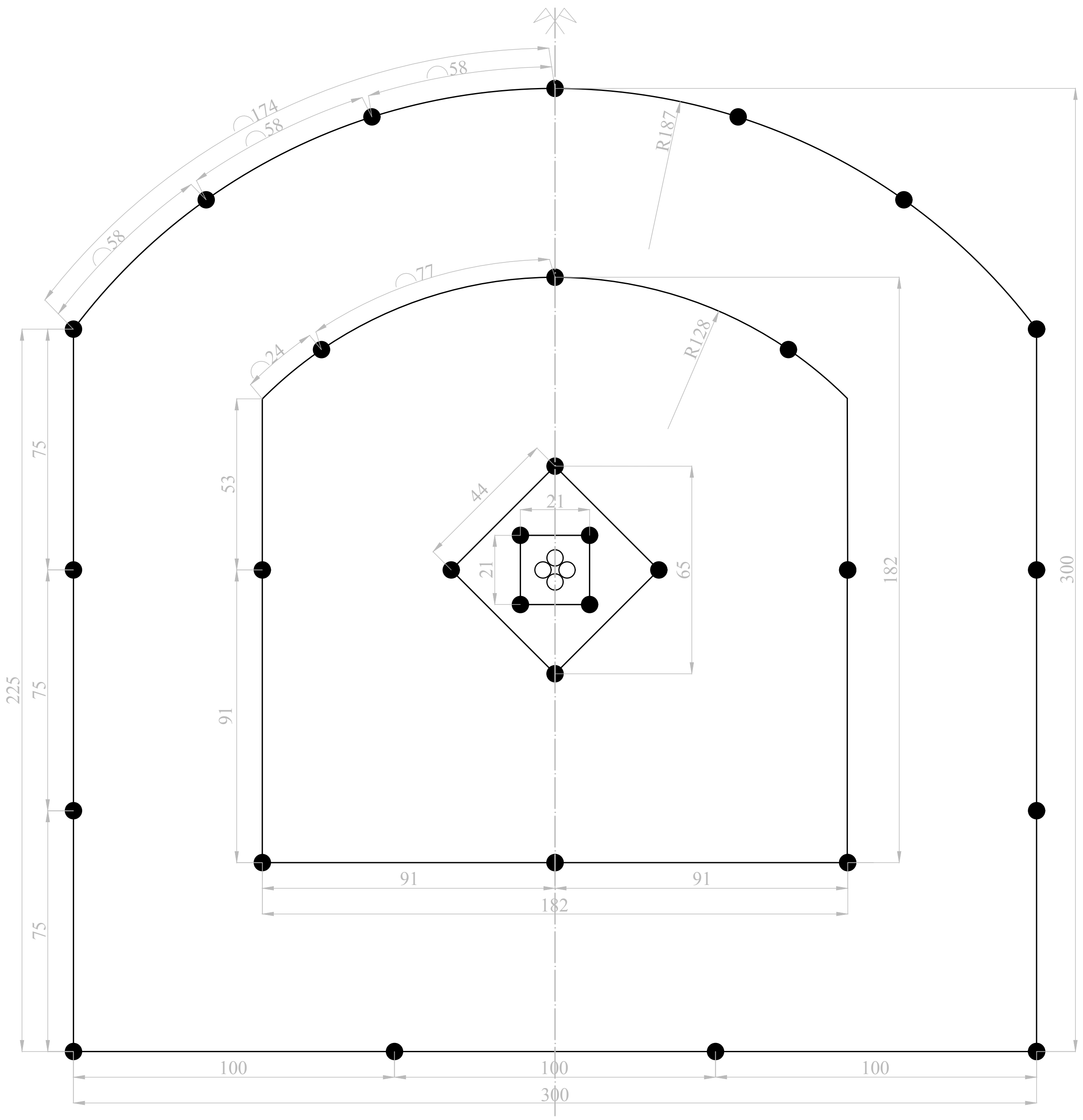
Supervisor:

Prof. Marilena Cardu

A.Y. 2018 - 2019

Drilling scheme

Scale: 1:20



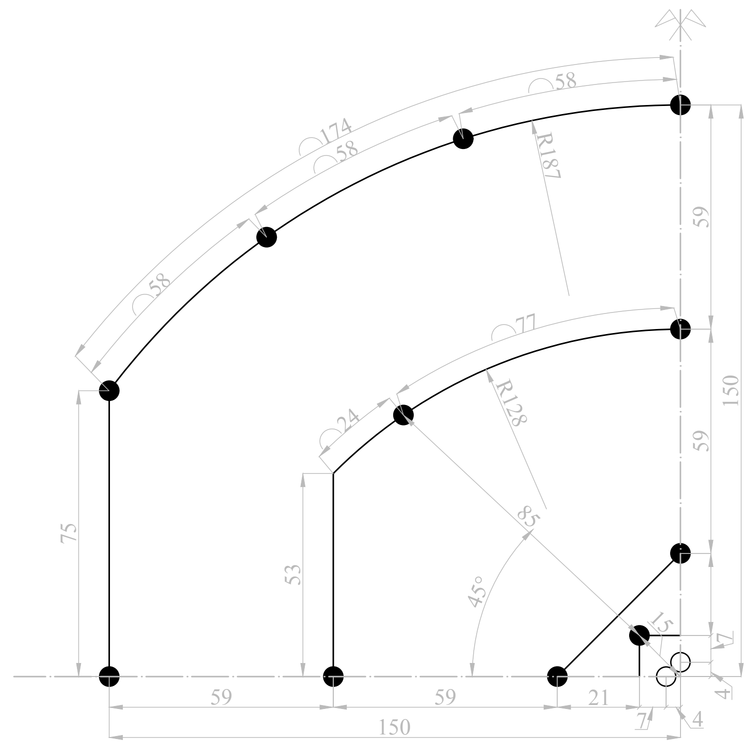
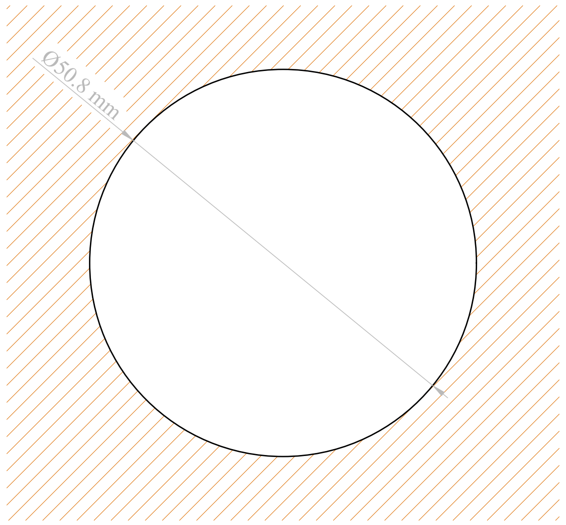


Image of the Boomer 104 Atlas Copco, available for drilling

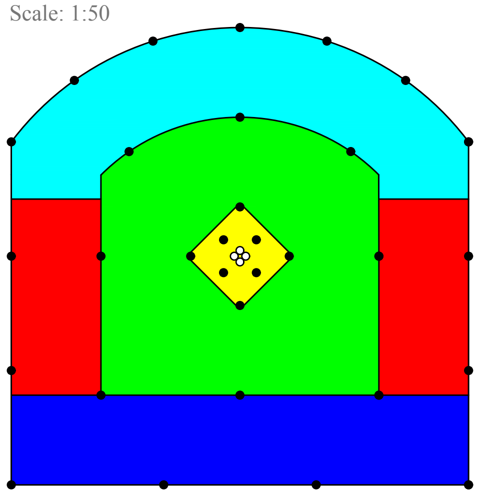
Cross section of a hole

Scale: 1:1



Classification of the blasted volumes

Scale: 1:50



Volume blasted by cut holes

Volume blasted by production holes

Volume blasted by contour (floor) holes

Volume blasted by contour (walls) holes

Volume blasted by contour (roof) holes

Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:

Simone BEZ

Badge Number: 236349

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Technical drawing:

PHASE 3: DRILLING SCHEME FOR SECTION 3.0 x 3.0 m

Scale: 1:20

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:

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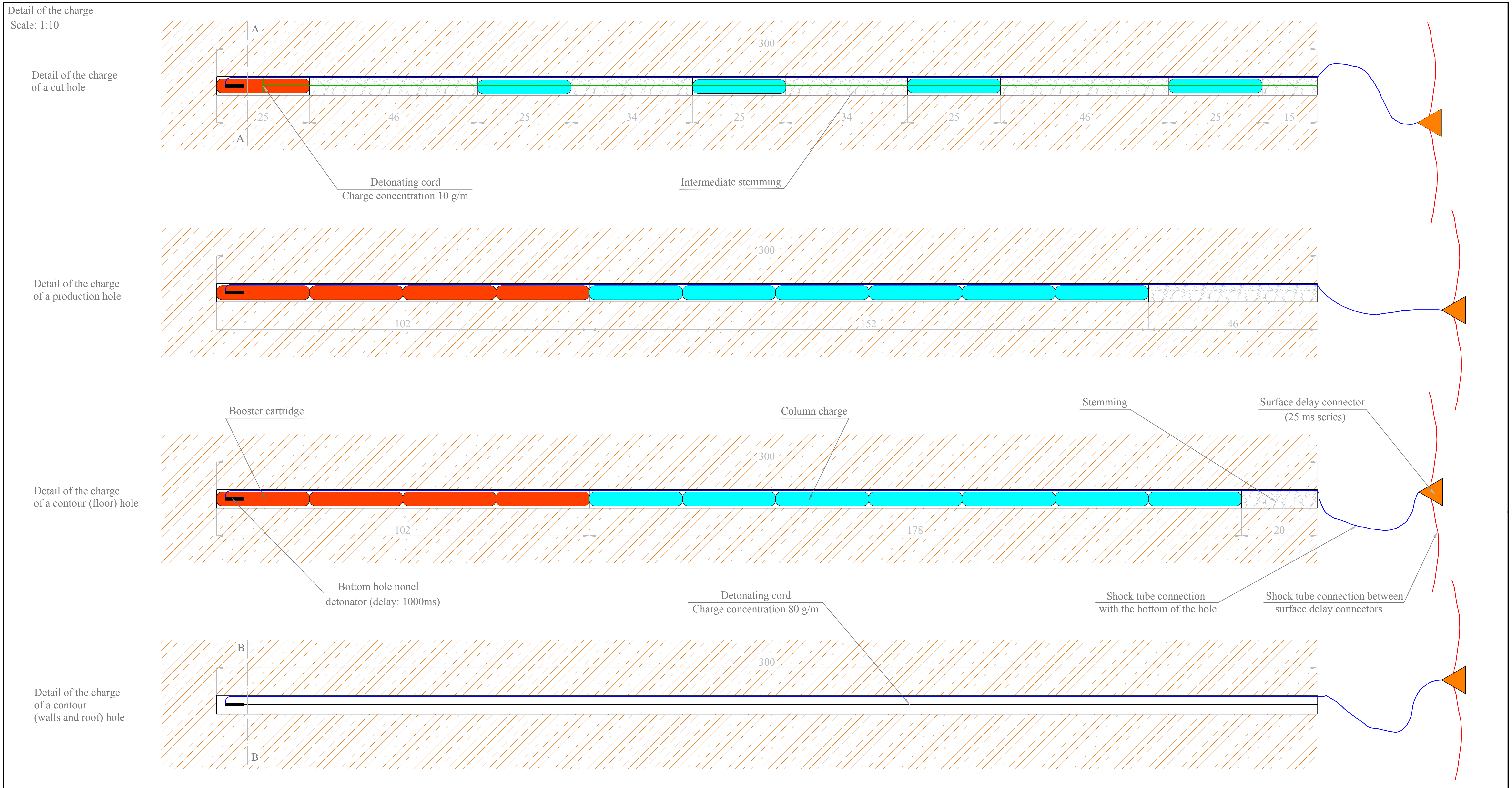
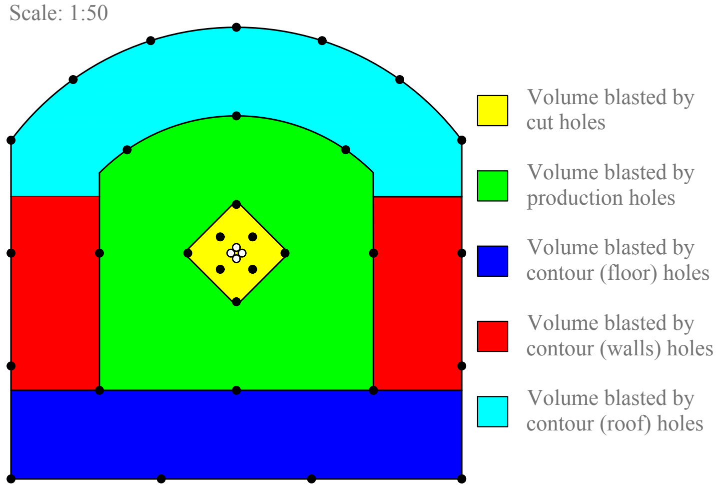
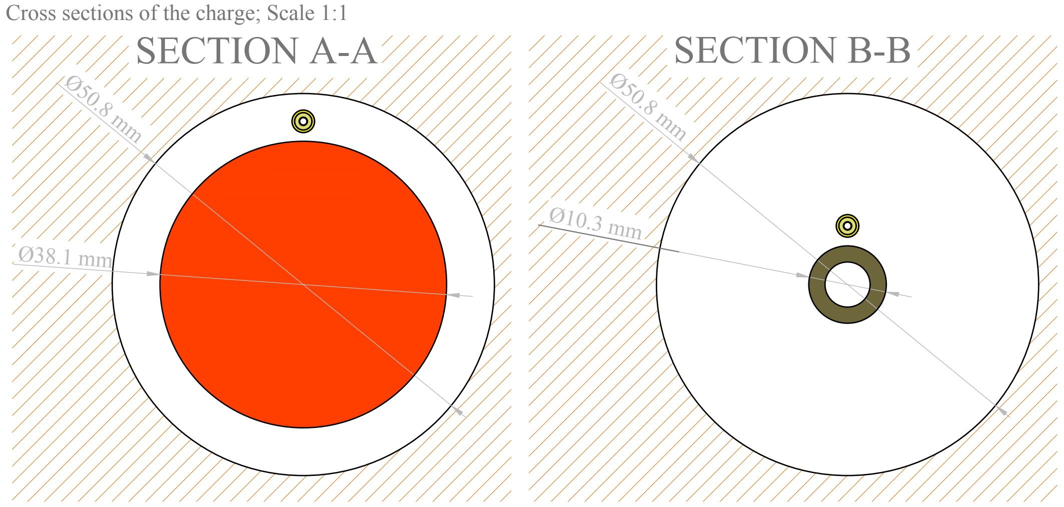
A.Y. 2018 - 2019



Classification of the blasted volumes
Scale: 1:50



Cross sections of the charge; Scale 1:1



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: SCHEME OF THE CHARGE FOR SECTION 3.0 x 3.0 m

Scale: 1:10

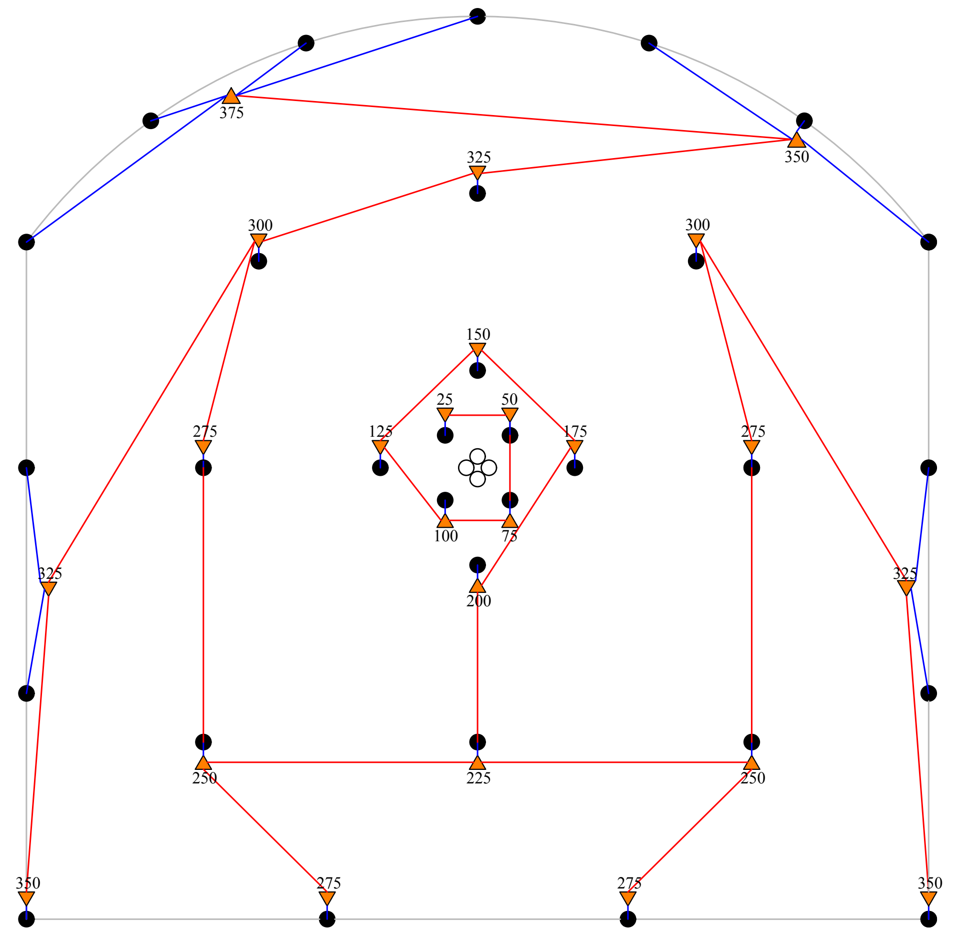
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

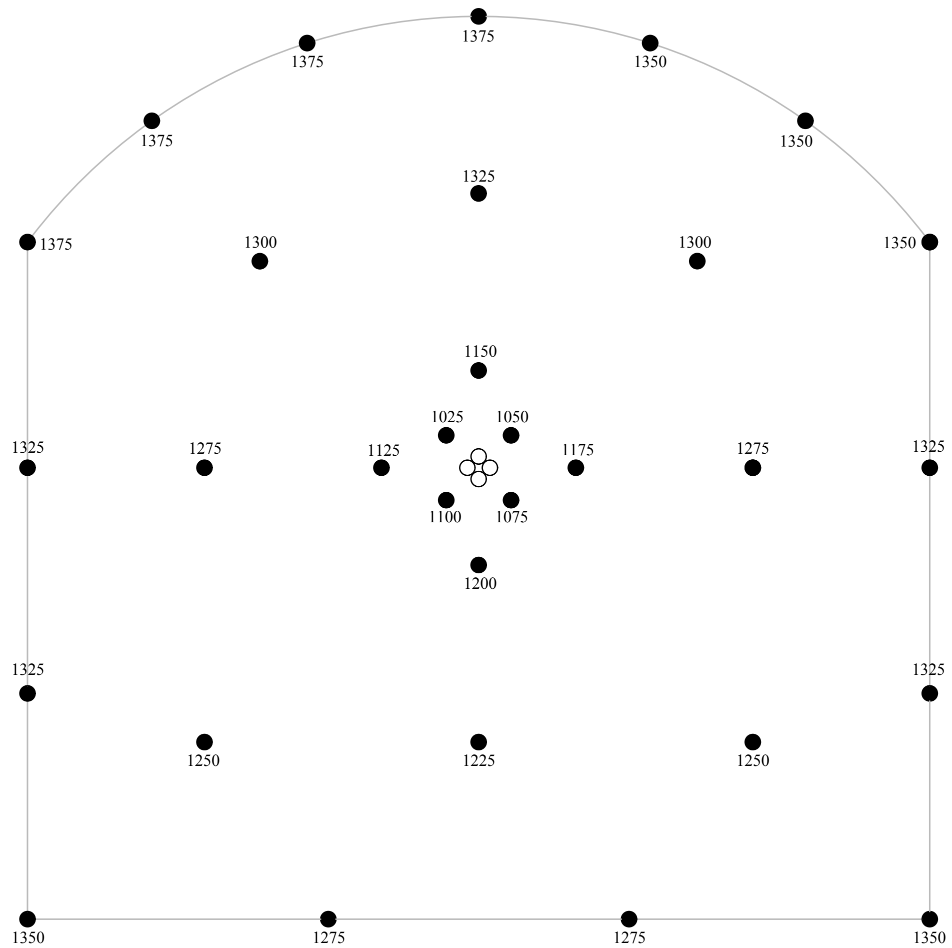
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	2	2
11 (1275 ms)	25 ms	1000 ms	4	4
12 (1300 ms)	25 ms	1000 ms	2	2
13 (1325 ms)	25 ms	1000 ms	3	5
14 (1350 ms)	25 ms	1000 ms	3	5
15 (1375 ms)	25 ms	1000 ms	1	4

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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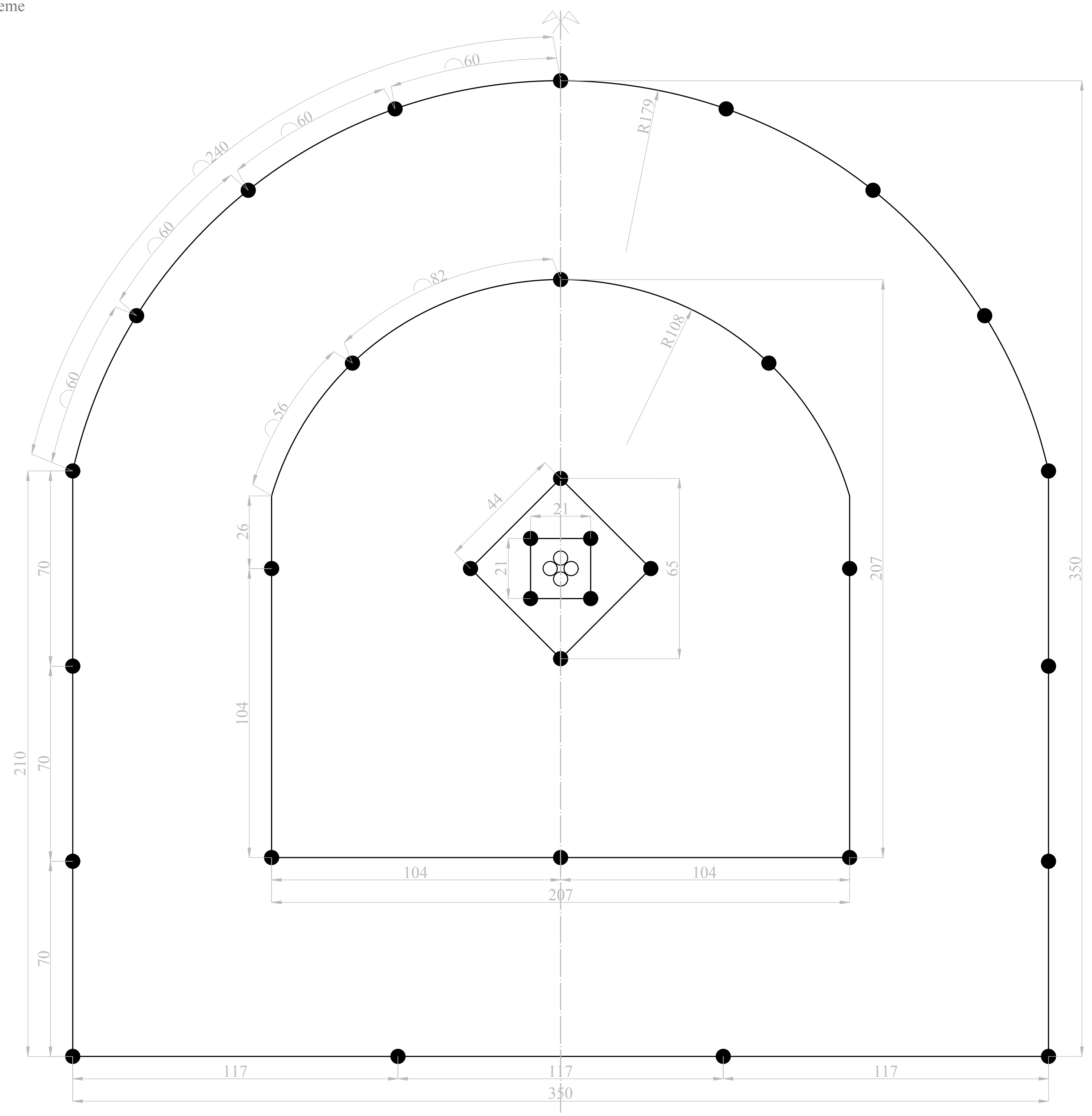
Technical drawing:
PHASE 3: TRIGGER TIMES FOR SECTION 3.0 x 3.0 m
Scale: 1:25

POLITECNICO DI TORINO
Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

Drilling scheme

Scale: 1:20



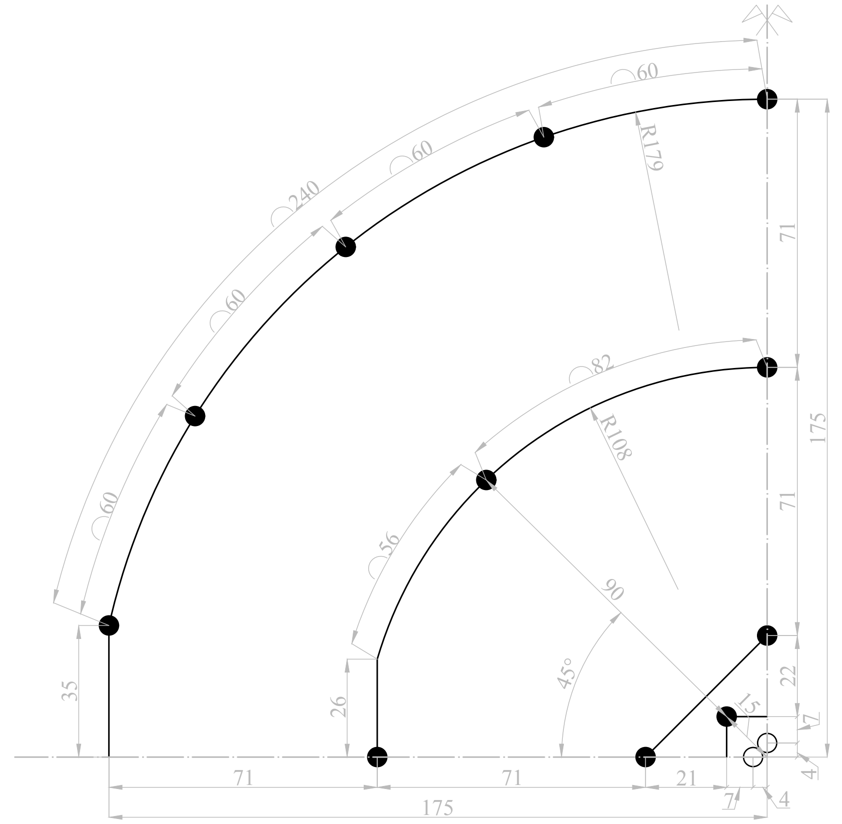
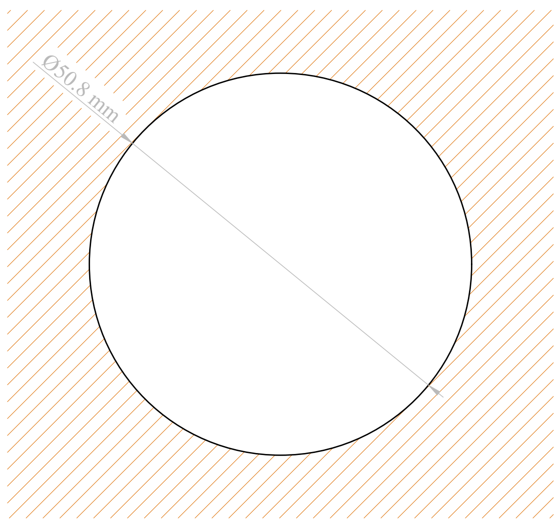


Image of the Boomer 104 Atlas Copco, available for drilling

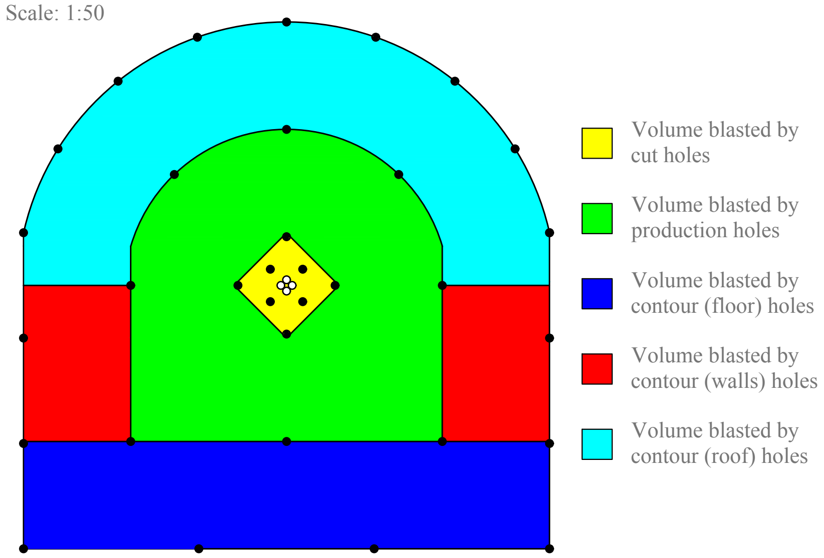
Cross section of a hole

Scale: 1:1



Classification of the blasted volumes

Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:

PHASE 3: DRILLING SCHEME FOR SECTION 3.5 x 3.5 m

Scale: 1:20

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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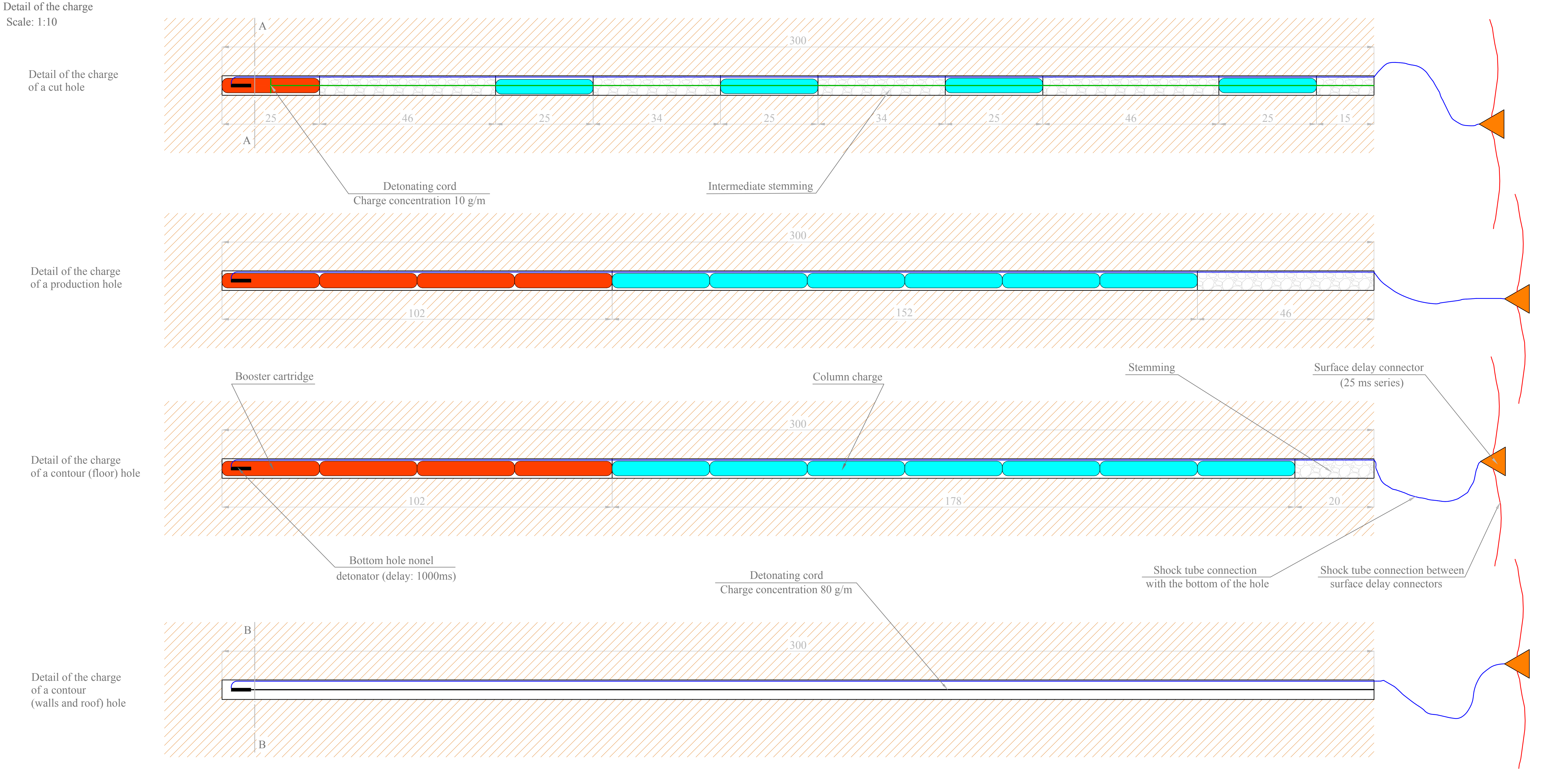

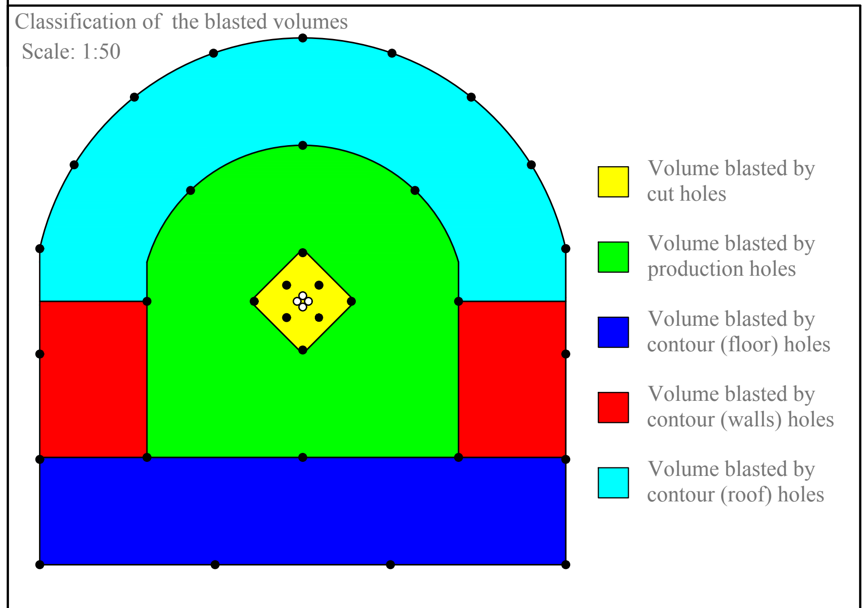

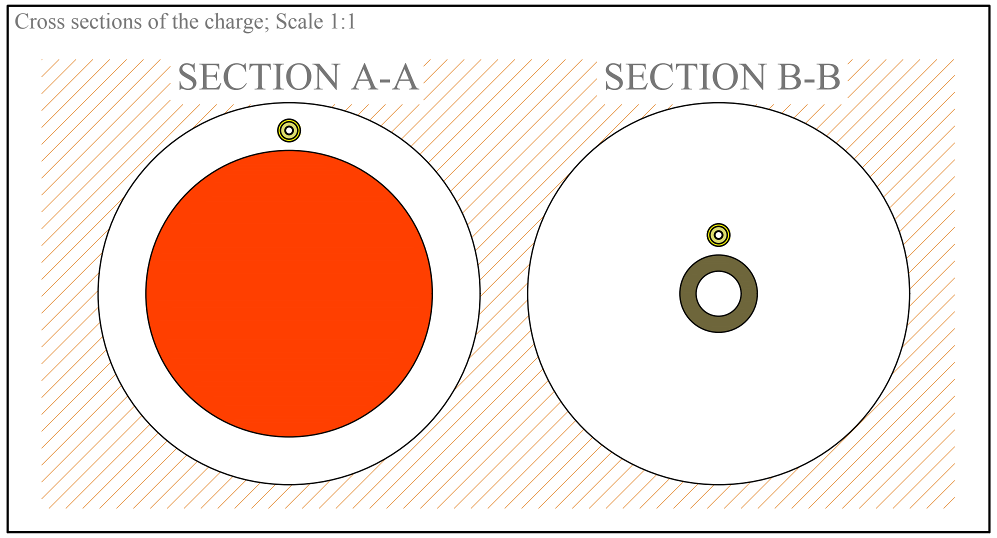

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: SCHEME OF THE CHARGE FOR SECTION 3.5 x 3.5 m

Scale: 1:10

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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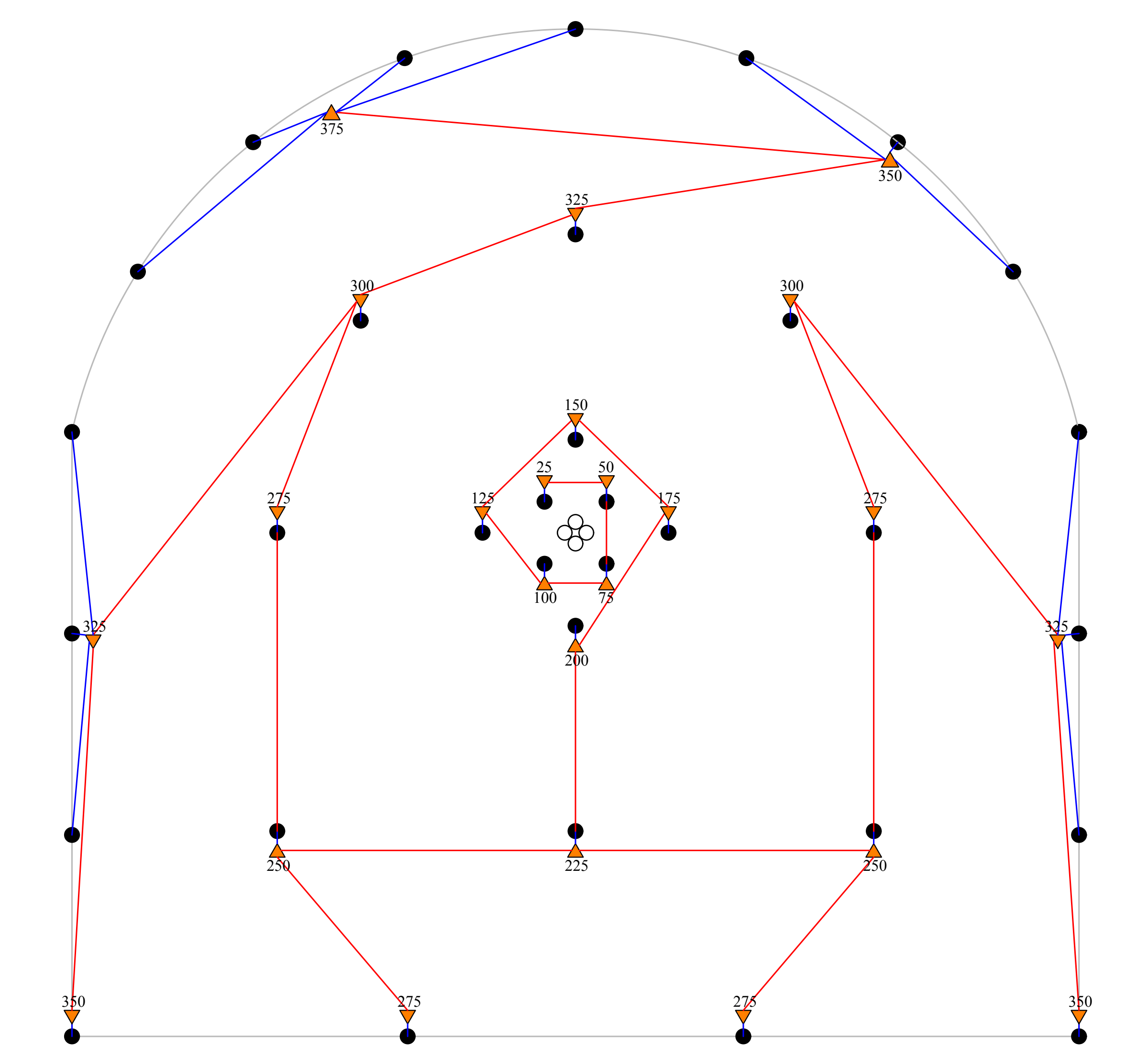
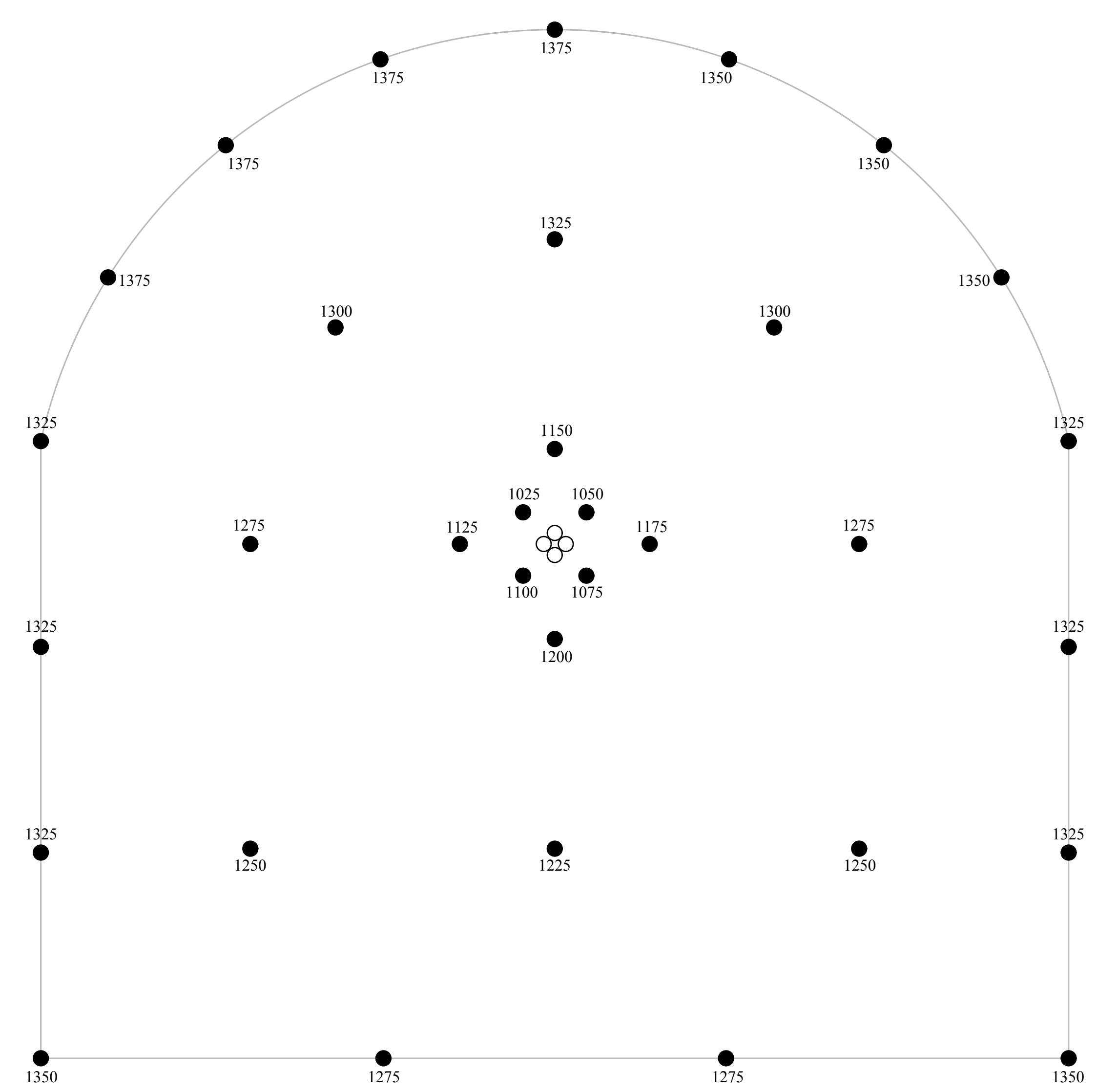
Trigger times of the surface delay connectors
Scale: 1:25



Trigger times of the in-hole detonators
Scale: 1:25



Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
1 (1025 ms)	25 ms	1000 ms	1	1
2 (1050 ms)	25 ms	1000 ms	1	1
3 (1075 ms)	25 ms	1000 ms	1	1
4 (1100 ms)	25 ms	1000 ms	1	1
5 (1125 ms)	25 ms	1000 ms	1	1
6 (1150 ms)	25 ms	1000 ms	1	1
7 (1175 ms)	25 ms	1000 ms	1	1
8 (1200 ms)	25 ms	1000 ms	1	1

Detonation order (detonation time)	Delay of the surface connector	Delay of the in - hole detonator	No. of the surface connectors	No. of the bottom hole detonators
9 (1225 ms)	25 ms	1000 ms	1	1
10 (1250 ms)	25 ms	1000 ms	2	2
11 (1275 ms)	25 ms	1000 ms	4	4
12 (1300 ms)	25 ms	1000 ms	2	2
13 (1325 ms)	25 ms	1000 ms	3	7
14 (1350 ms)	25 ms	1000 ms	3	5
15 (1375 ms)	25 ms	1000 ms	1	4

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
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Badge Number: 236349

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Technical drawing:
PHASE 3: TRIGGER TIMES FOR SECTION 3.5 x 3.5 m
Scale: 1:25

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
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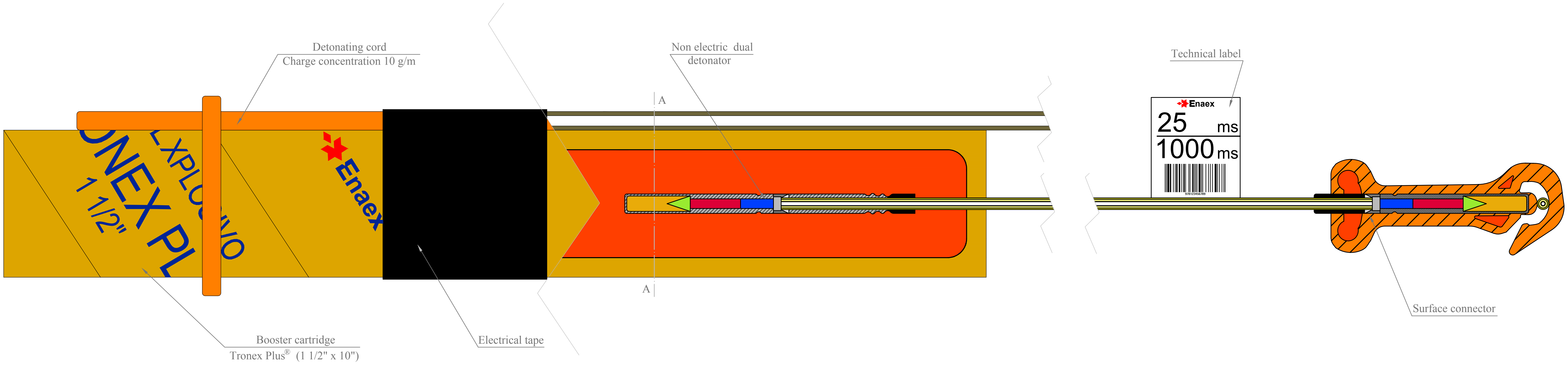
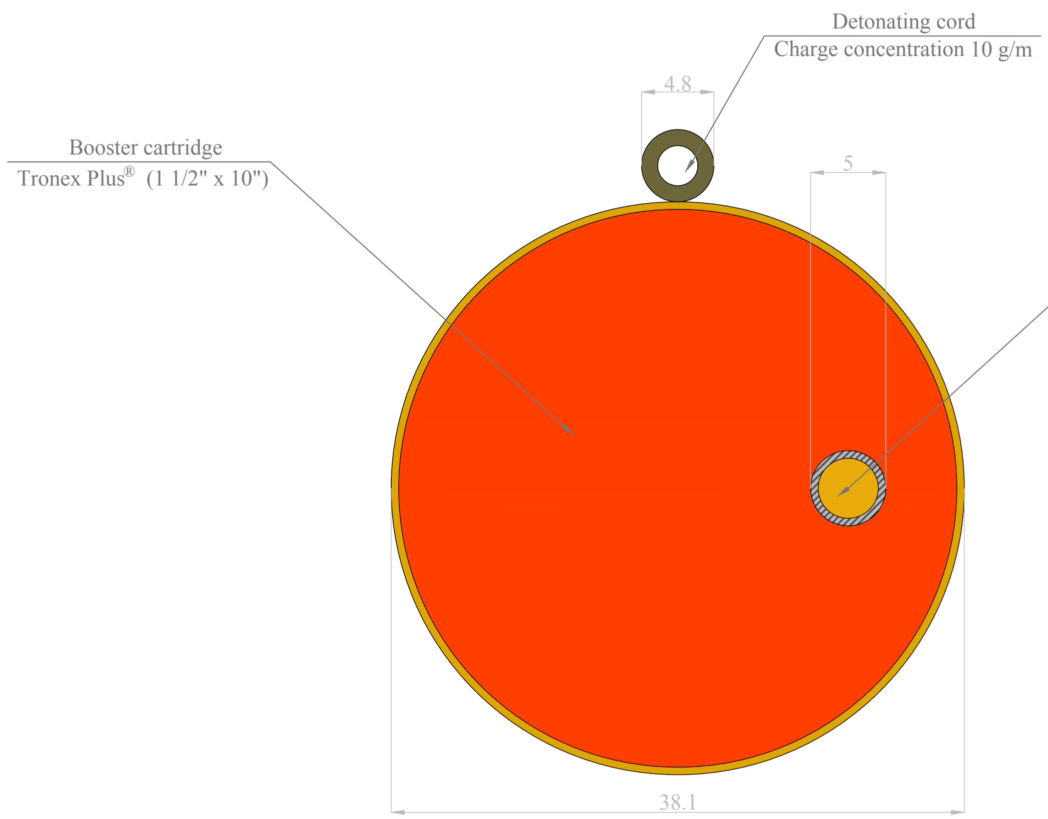
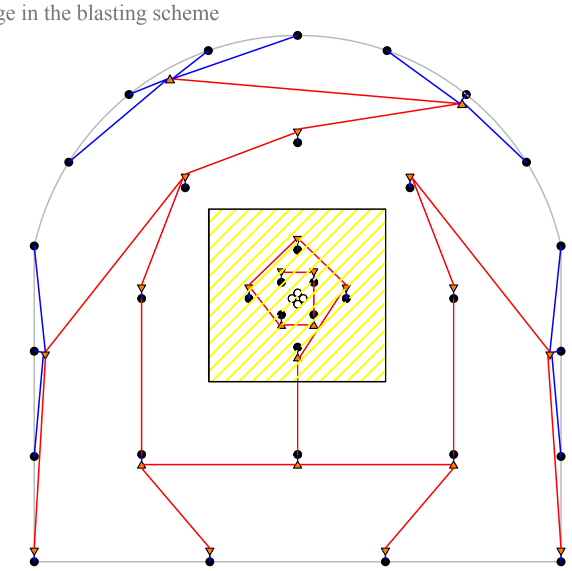
Detail of the junction booster - bottom hole detonator - propagation detonating cord - surface connector (dual detonator system)
Scale: 1:1



Section A - A of the junction between booster cartridge, bottom hole detonator and propagation detonating cord
Scale: 2:1



Location of the detail of the charge in the blasting scheme
Scale: 1:50



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
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Technical drawing:
PHASE 3: DETAIL OF THE BOTTOM HOLE CHARGE USED IN THE CUT HOLES

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio
Supervisor:
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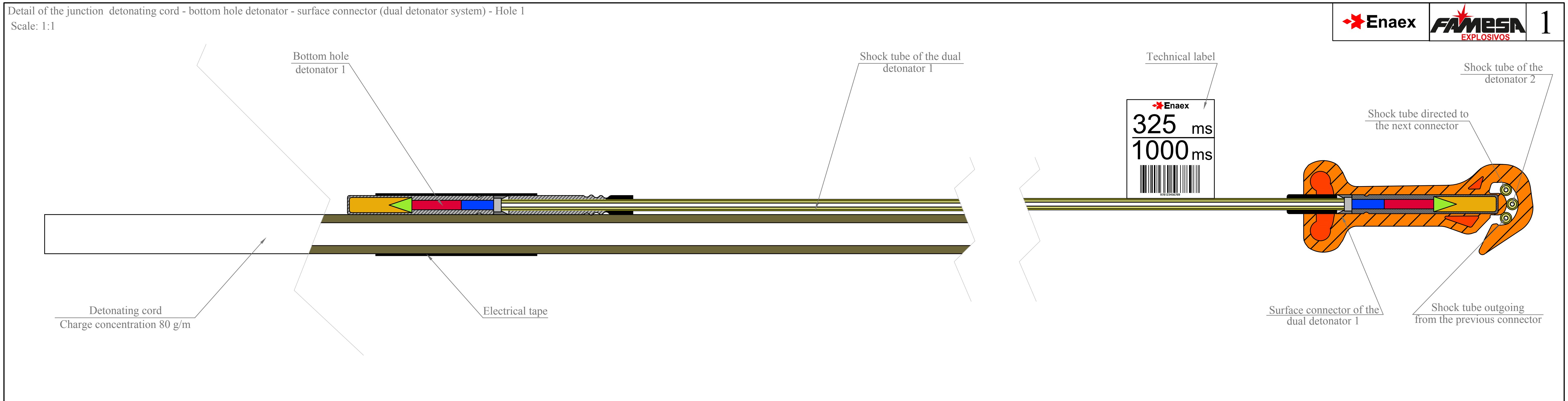
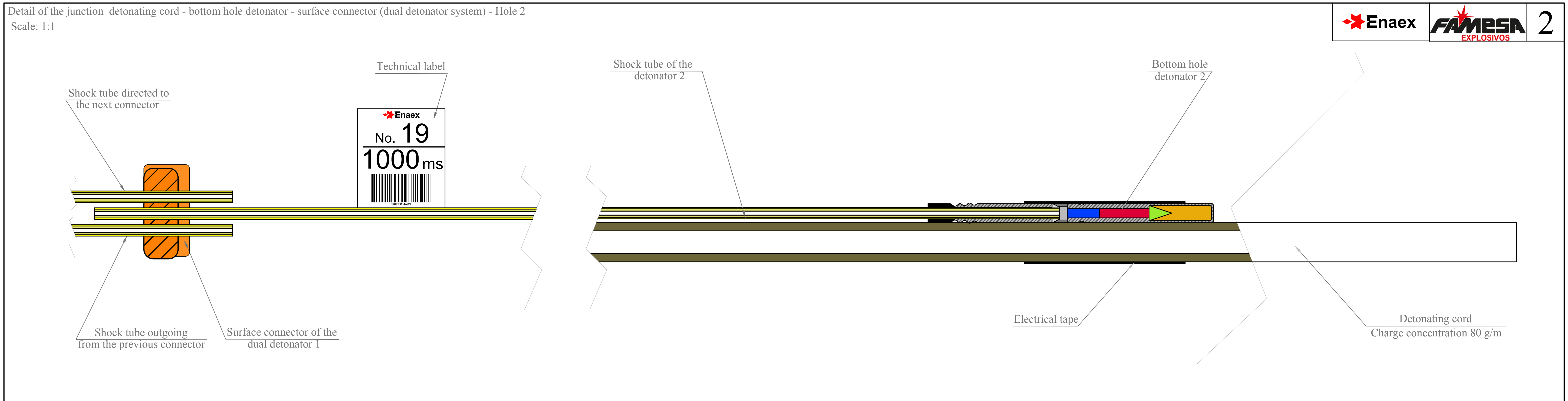
Detail of the junction detonating cord - bottom hole detonator - surface connector (dual detonator system) - Hole 1

Scale: 1:1



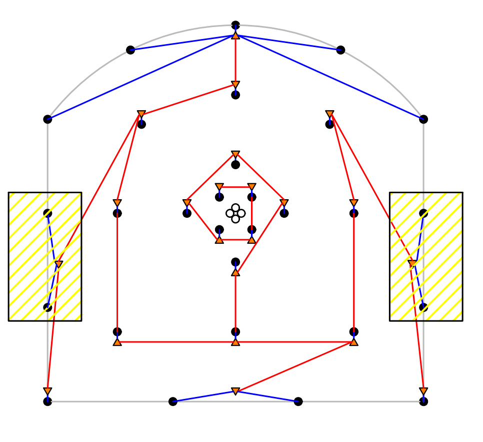
Detail of the junction detonating cord - bottom hole detonator - surface connector (dual detonator system) - Hole 2

Scale: 1:1



Location of the detail in the blasting scheme

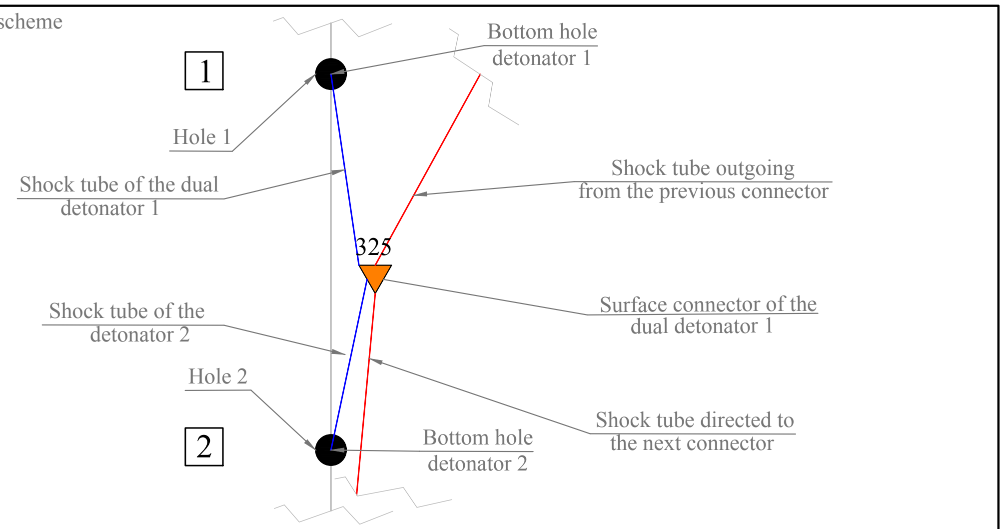
Scale: 1:50



Location of the detail in the blasting scheme

Hole 1: Non electric dual detonator

Hole 2: Non electric detonator



Project:

Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:

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Technical drawing:

DETAIL OF NONEL CONNECTION BETWEEN HOLES OF THE CONTOUR (WALLS)

Scale: 1:1

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:

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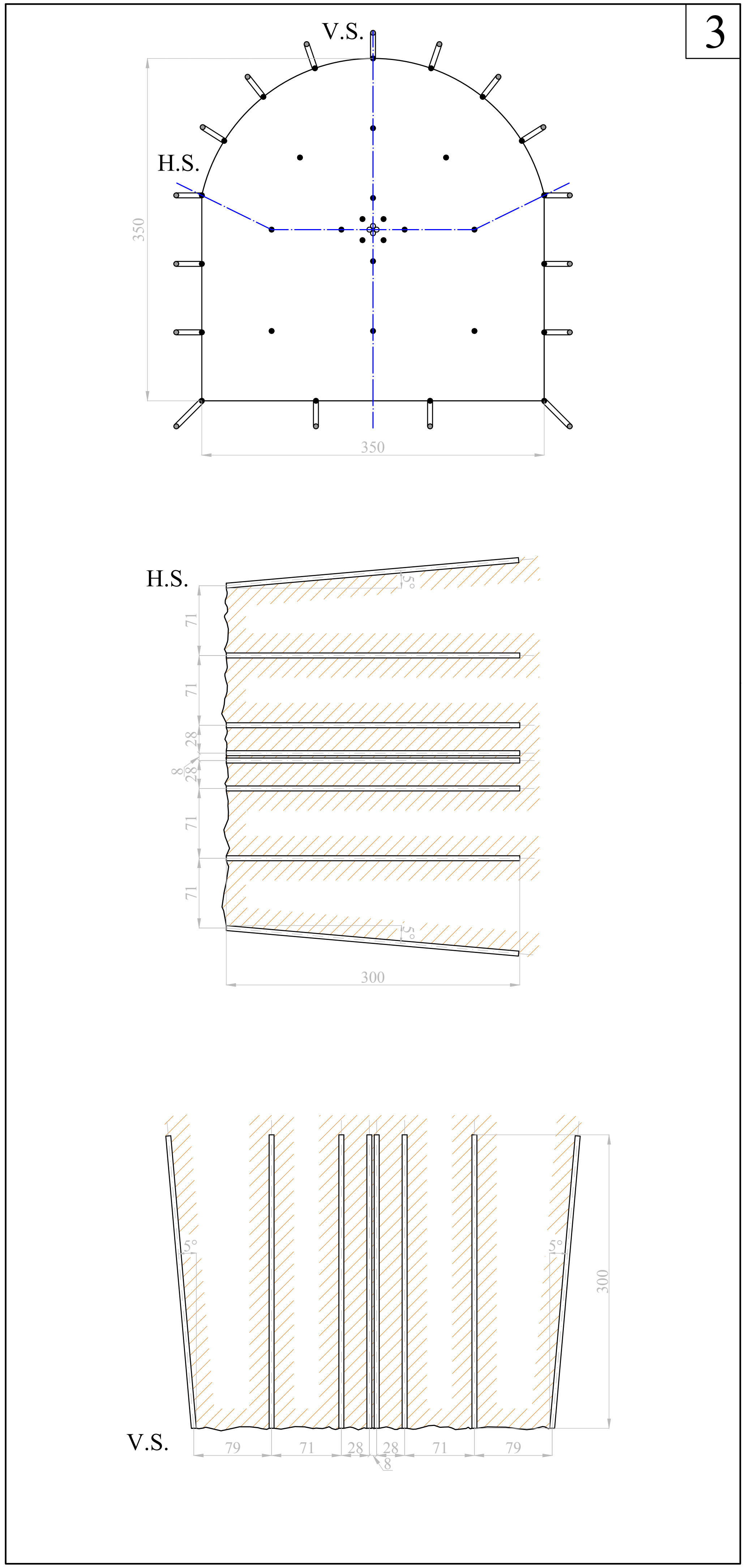

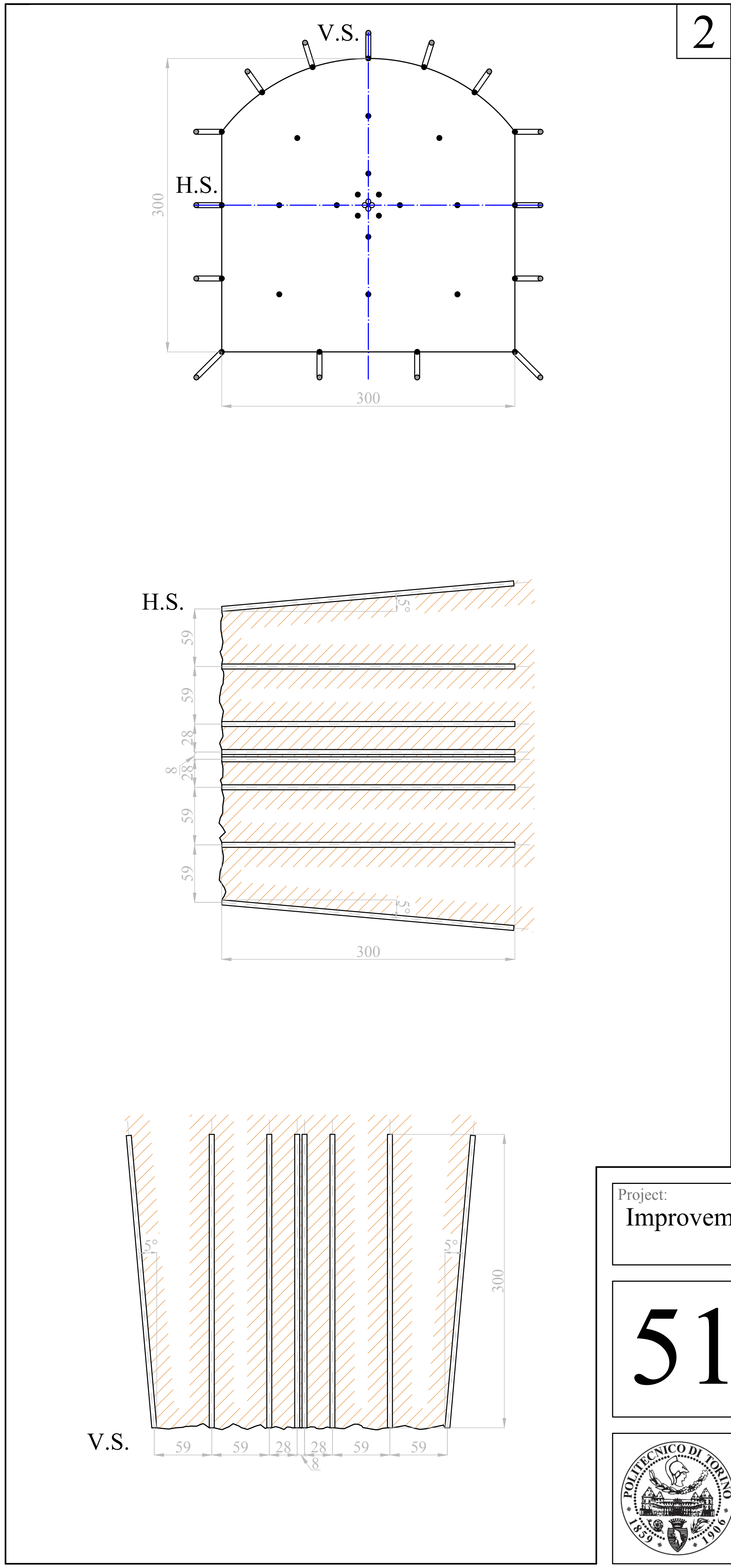

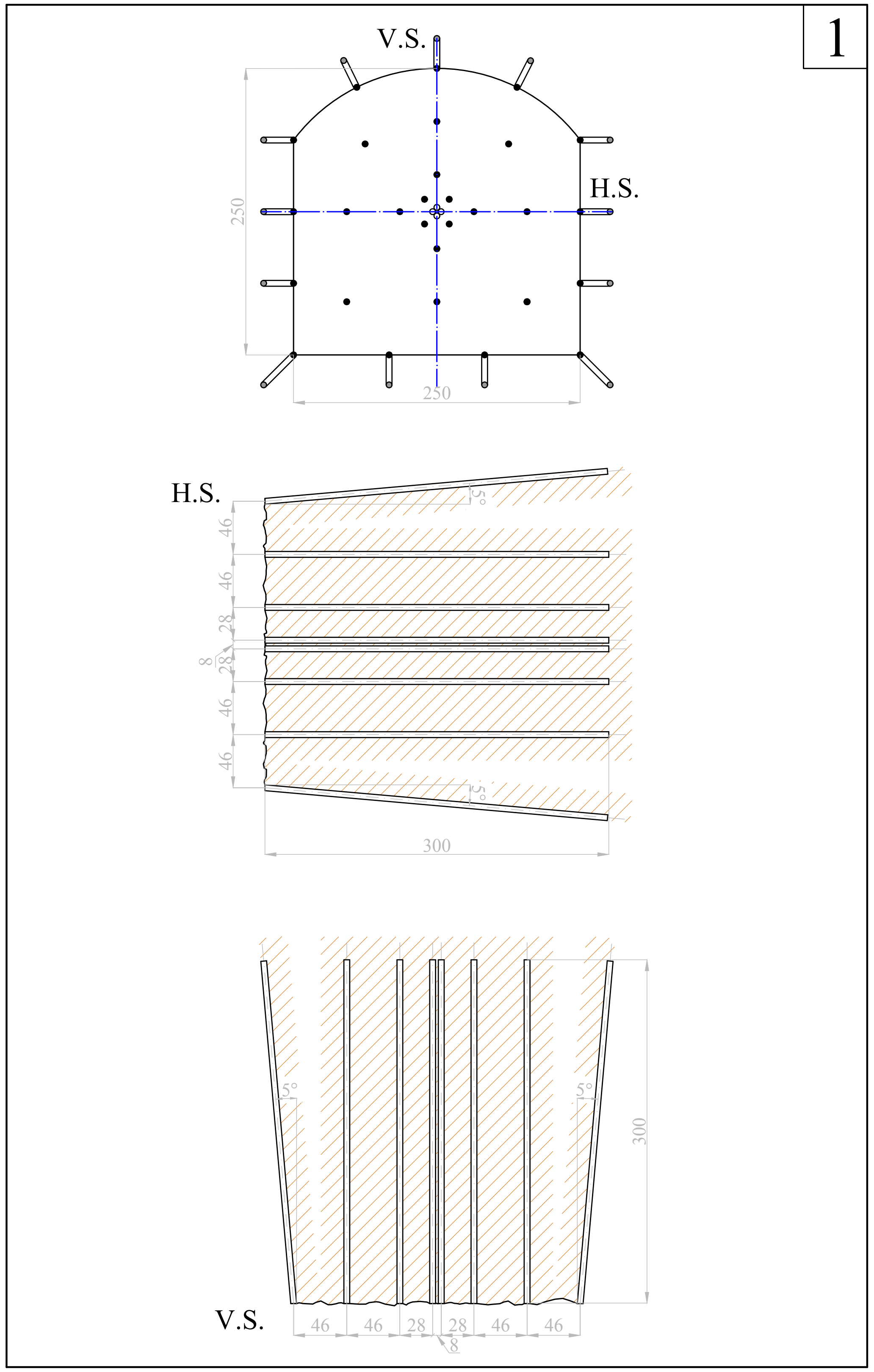

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
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Technical drawing:
PHASE 3: SECTIONS AND DEFINITION OF THE LOOK-OUT ANGLE

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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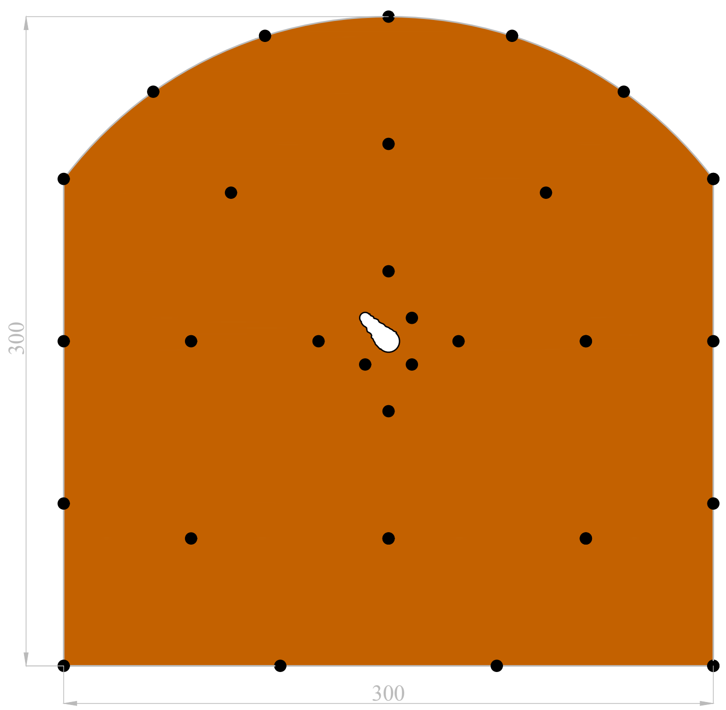
Instant of the detonation sequence at 1025 ms

1

Empty central hole volume (considering the fictitious diameter): 0.024 m^3
Rock volume blasted at this stage: 0.024 m^3
Rock volume blasted until now: 0.024 m^3

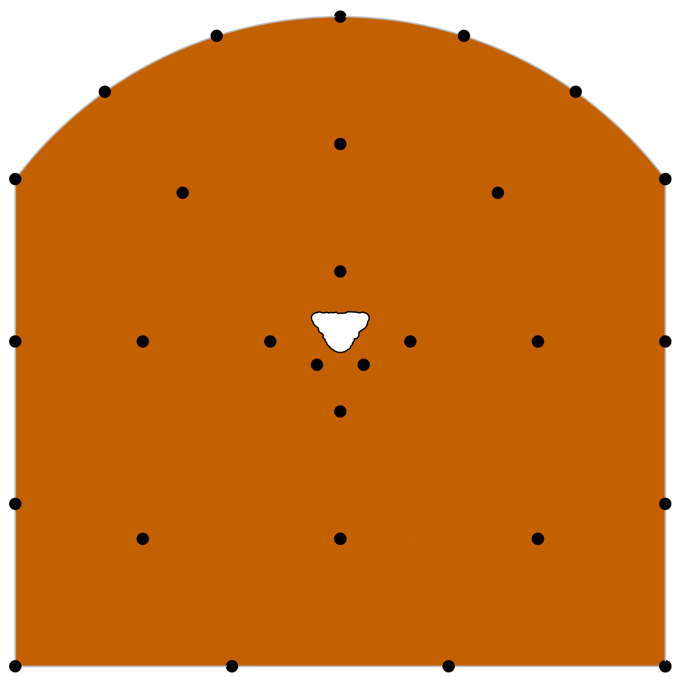


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1050 ms

2

Rock volume blasted at this stage: 0.051 m^3
Rock volume blasted until now: 0.075 m^3

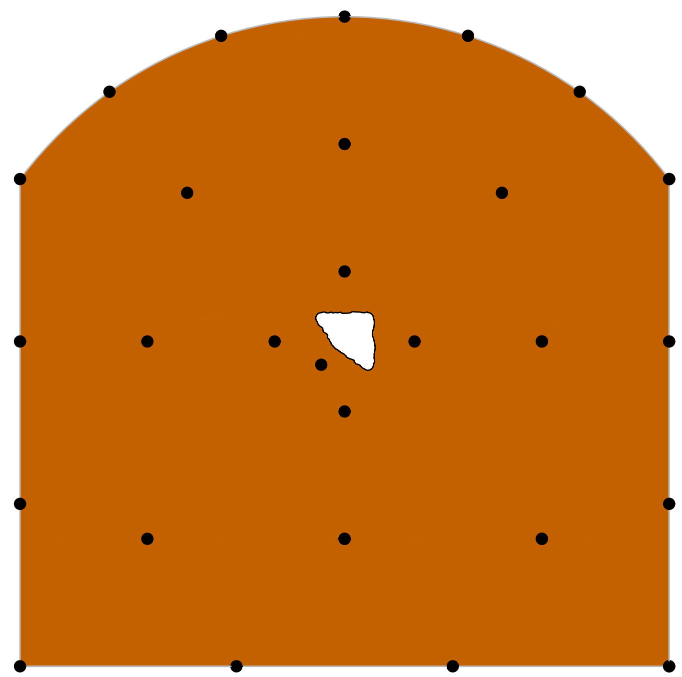


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1075 ms

3

Rock volume blasted at this stage: 0.047 m^3
Rock volume blasted until now: 0.122 m^3

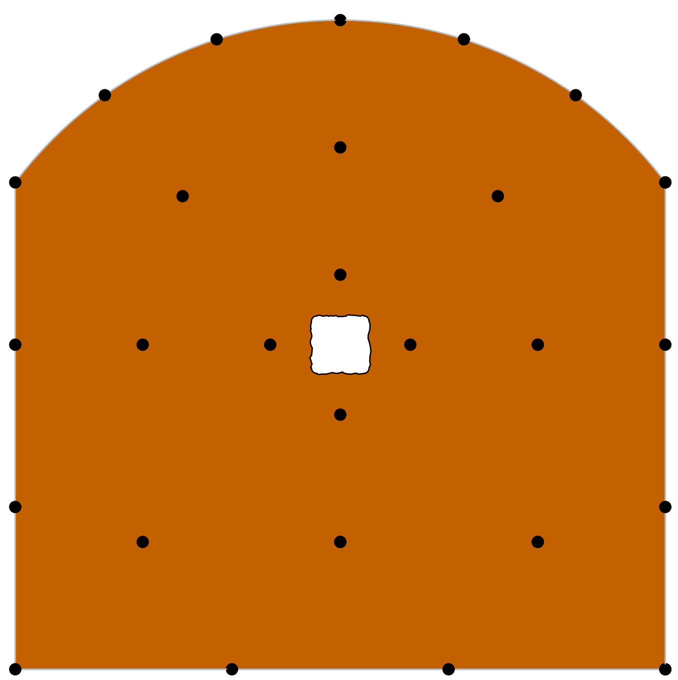


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1100 ms

4

Rock volume blasted at this stage: 0.068 m^3
Rock volume blasted until now: 0.190 m^3

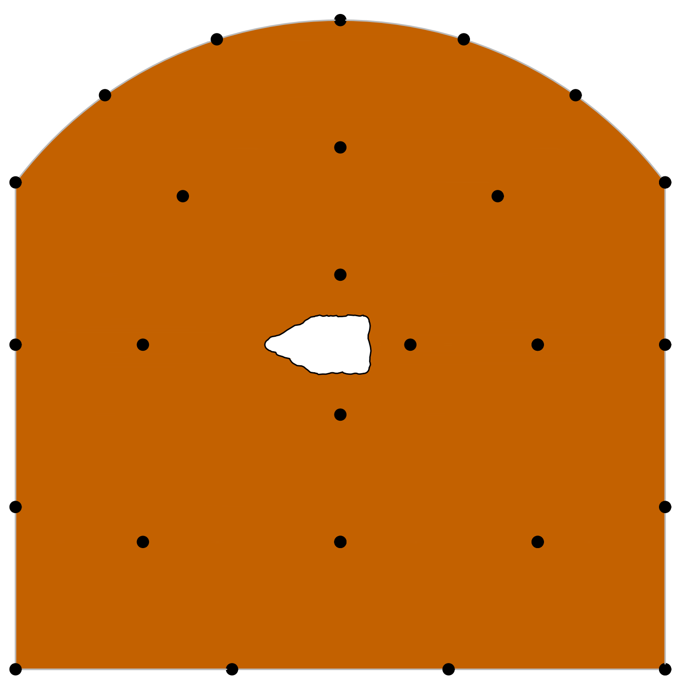


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1125 ms

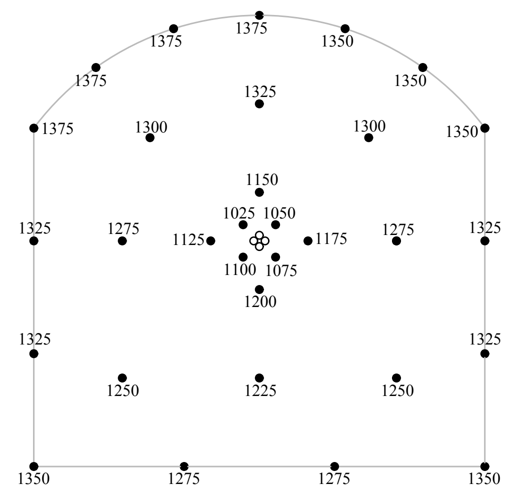
5

Rock volume blasted at this stage: 0.090 m^3
Rock volume blasted until now: 0.280 m^3



Charge per delay: 1.680 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (1)

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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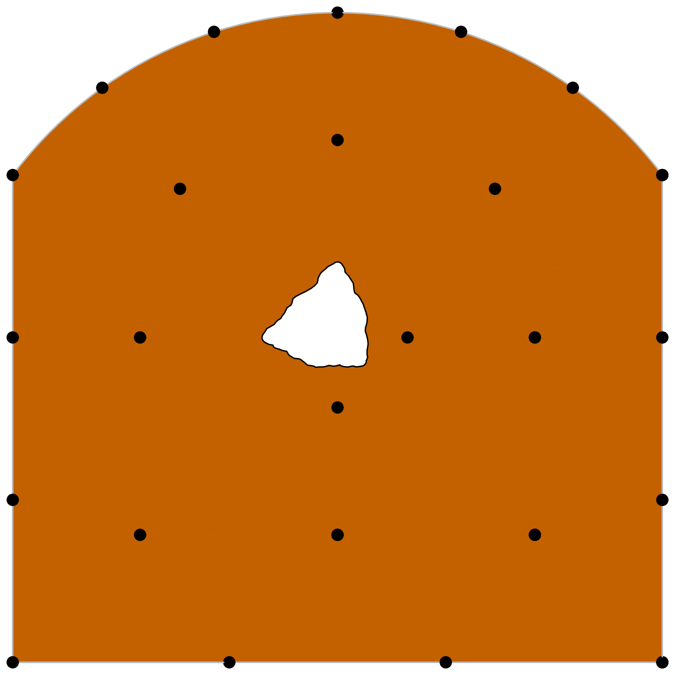
Instant of the detonation sequence at 1150 ms

6

Rock volume blasted at this stage: 0.144 m^3
Rock volume blasted until now: 0.424 m^3

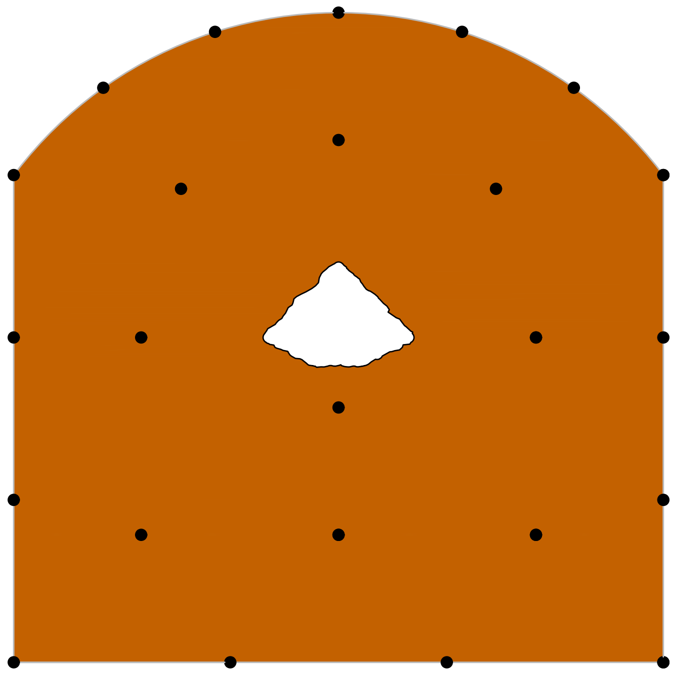


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1175 ms

7

Rock volume blasted at this stage: 0.139 m^3
Rock volume blasted until now: 0.563 m^3

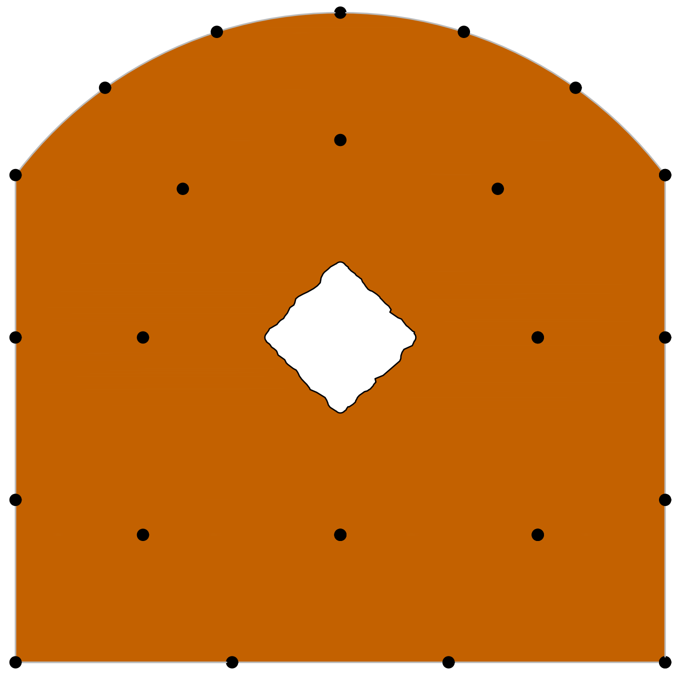


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1200 ms

8

Rock volume blasted at this stage: 0.202 m^3
Rock volume blasted until now: 0.765 m^3

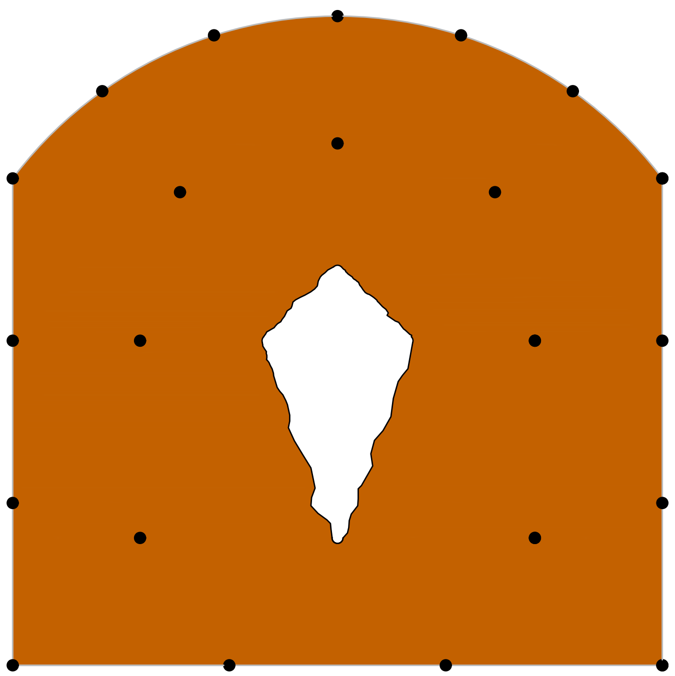


Charge per delay: 1.680 kg

Instant of the detonation sequence at 1225 ms

9

Rock volume blasted at this stage: 0.653 m^3
Rock volume blasted until now: 1.418 m^3

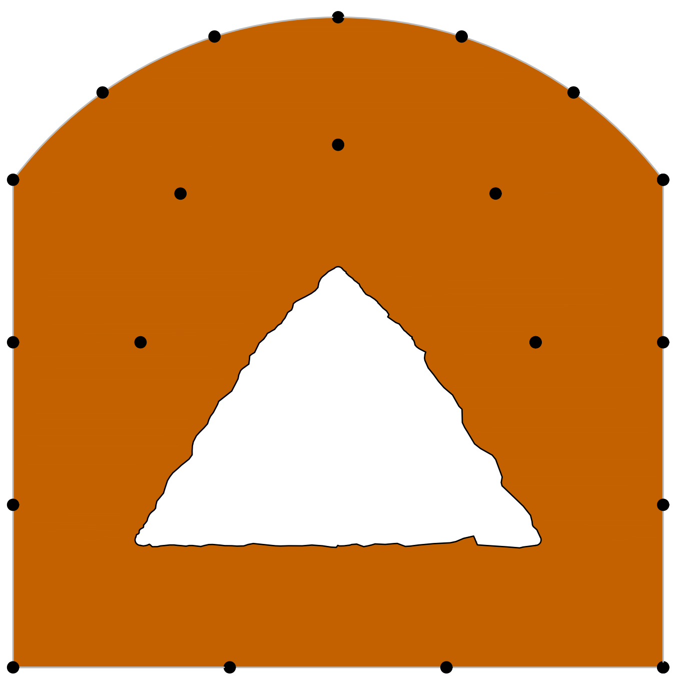


Charge per delay: 3.390 kg

Instant of the detonation sequence at 1250 ms

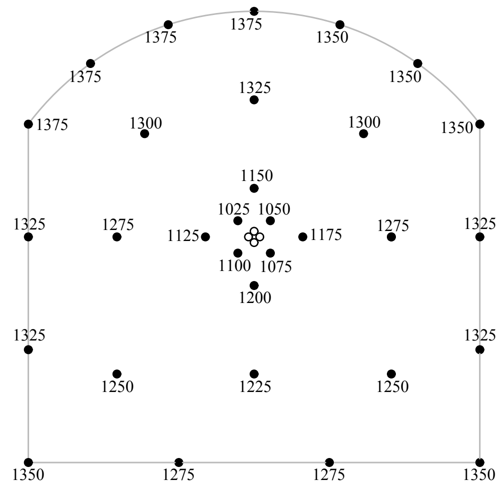
10

Rock volume blasted at this stage: 2.642 m^3
Rock volume blasted until now: 4.060 m^3



Charge per delay: 6.780 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (2)

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Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
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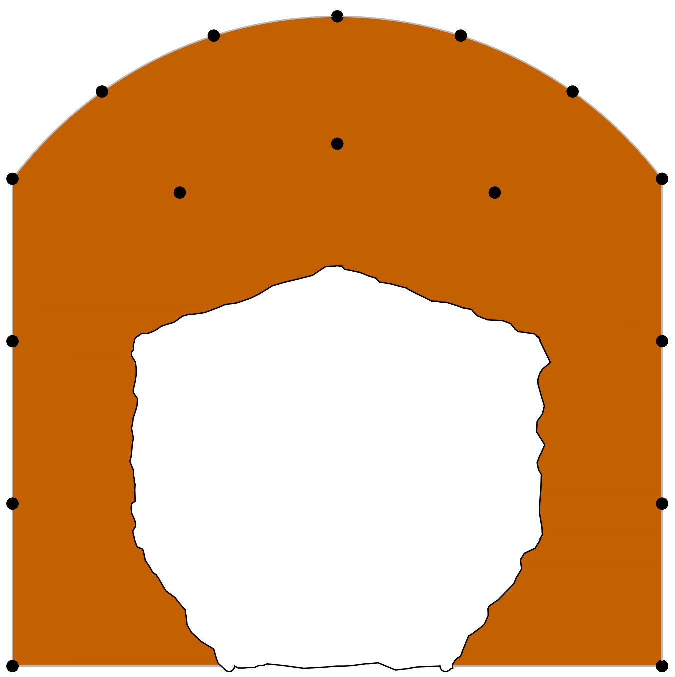
Instant of the detonation sequence at 1275 ms

11

Rock volume blasted at this stage: 4.733 m^3
Rock volume blasted until now: 8.793 m^3

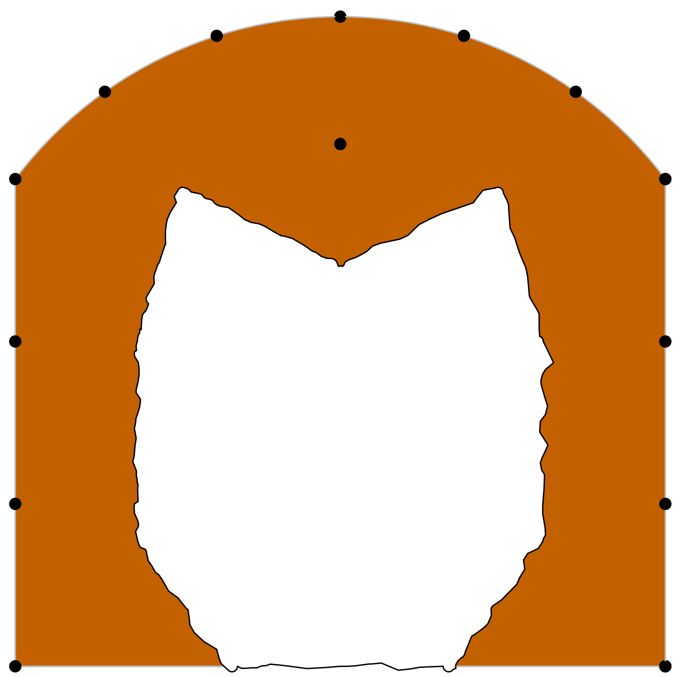


Charge per delay: 14.226 kg

Instant of the detonation sequence at 1300 ms

12

Rock volume blasted at this stage: 1.776 m^3
Rock volume blasted until now: 10.569 m^3

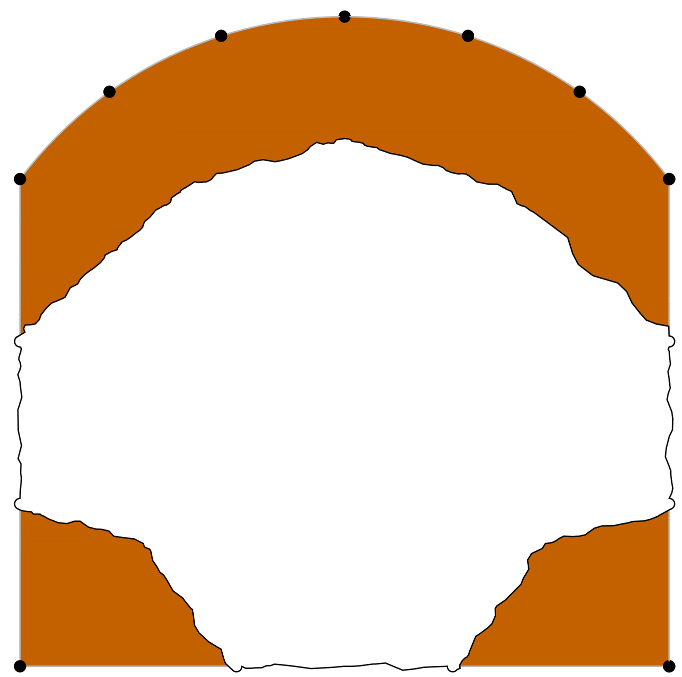


Charge per delay: 6.780 kg

Instant of the detonation sequence at 1325 ms

13

Rock volume blasted at this stage: 5.416 m^3
Rock volume blasted until now: 15.985 m^3

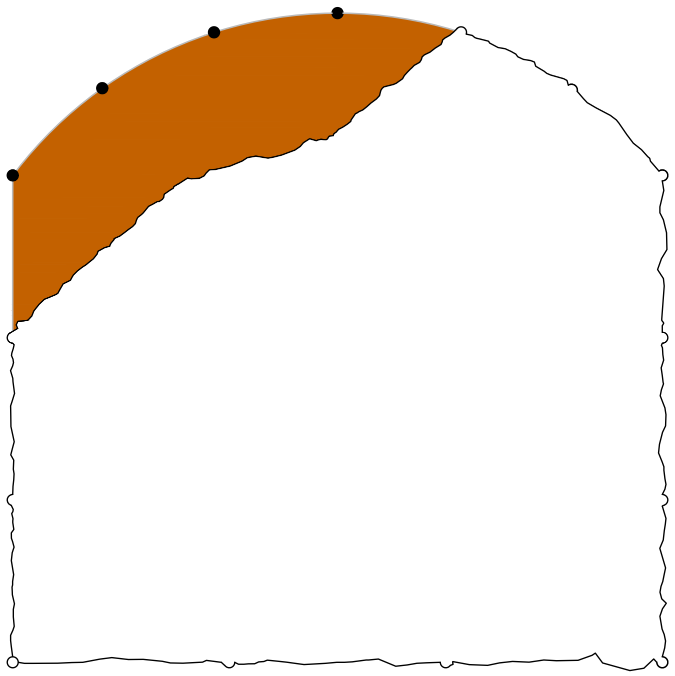


Charge per delay: 4.350 kg

Instant of the detonation sequence at 1350 ms

14

Rock volume blasted at this stage: 5.500 m^3
Rock volume blasted until now: 21.485 m^3

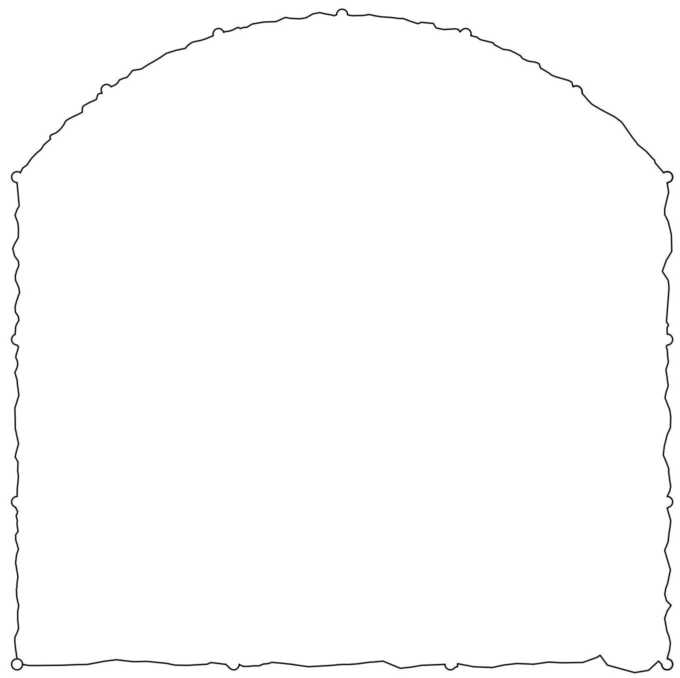


Charge per delay: 8.166 kg

Instant of the detonation sequence at 1375 ms

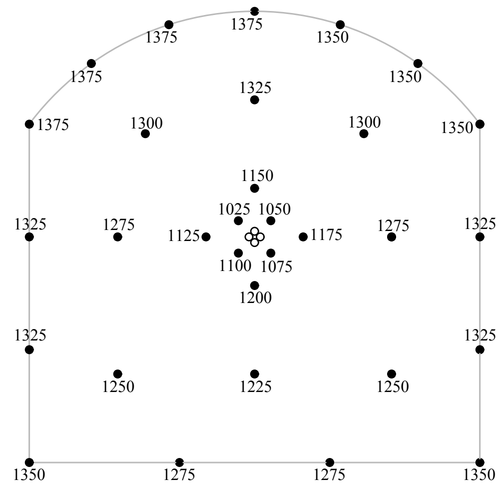
15

Rock volume blasted at this stage: 3.524 m^3
Rock volume blasted until now: 25.009 m^3



Charge per delay: 0.960 kg

Detonation sequence



Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

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Technical drawing:
PHASE 3: DETONATION SEQUENCE SCHEME FOR SECTION 3.0 x 3.0 m (3)

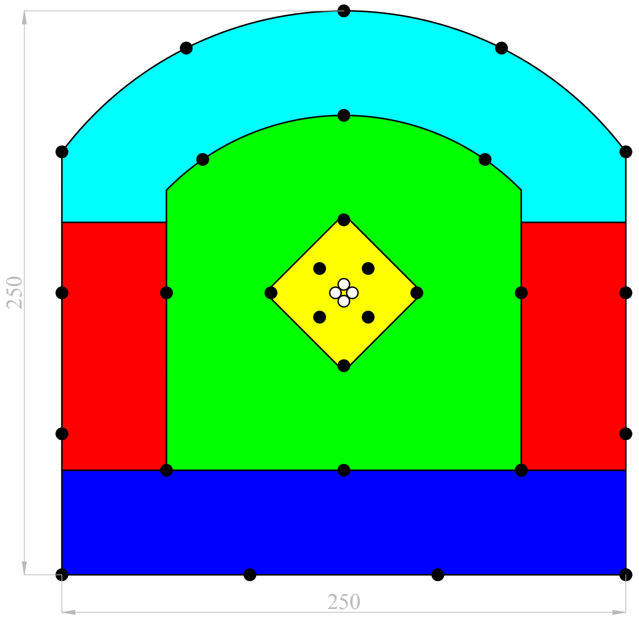
POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

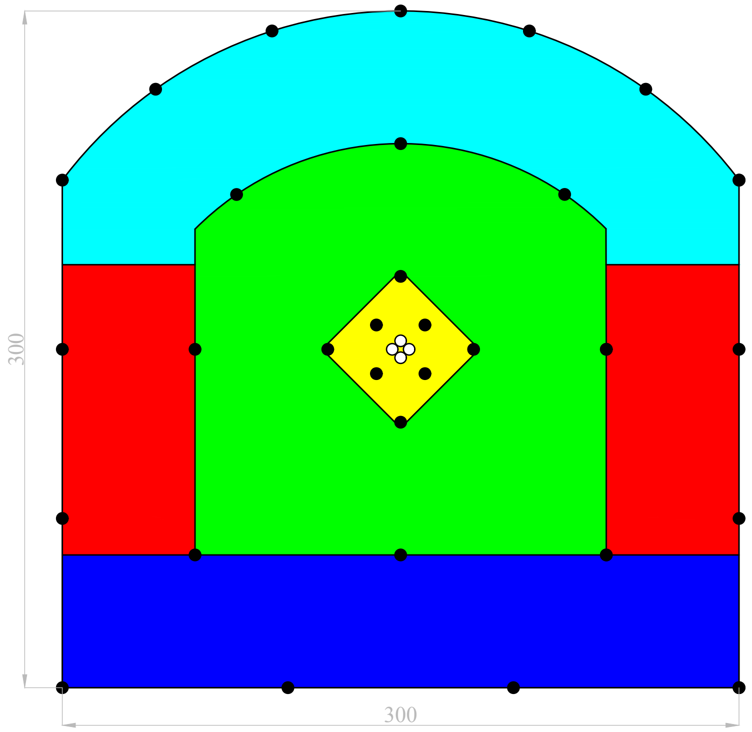
A.Y. 2018 - 2019

1



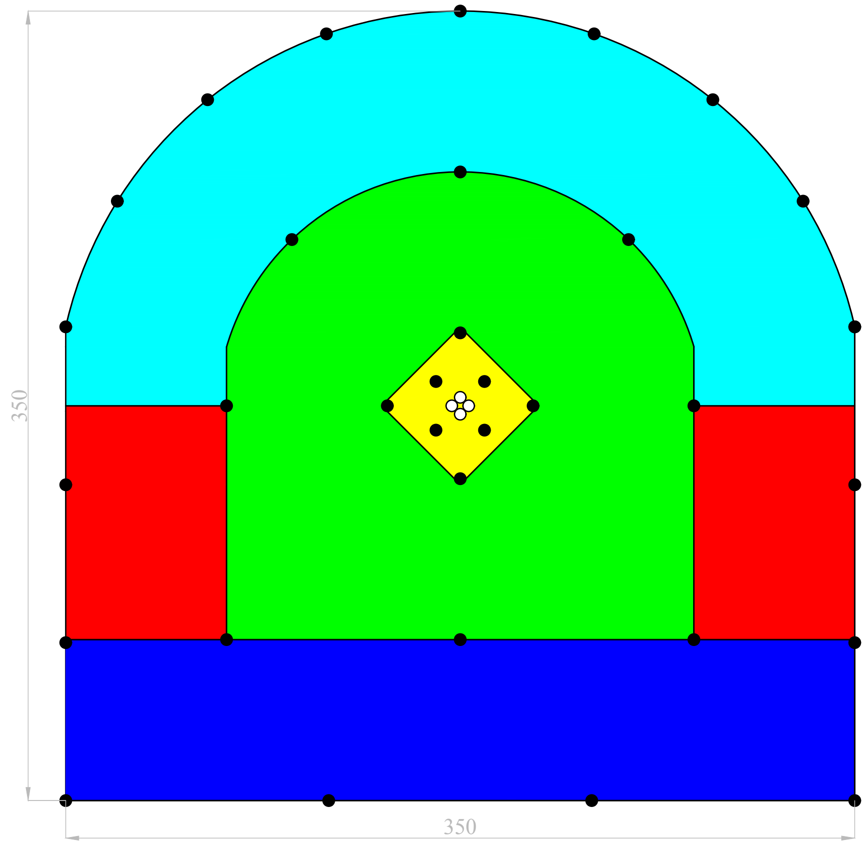
Cut holes	Volume blasted (m^3)	0.661
	Amount of explosive (kg)	13.440
	Powder factor (P.F.) (kg/m^3)	20.334
Production holes	Volume blasted (m^3)	6.185
	Amount of explosive (kg)	27.120
	Powder factor (P.F.) (kg/m^3)	4.385
Contour (floor) holes	Volume blasted (m^3)	3.457
	Amount of explosive (kg)	14.892
	Powder factor (P.F.) (kg/m^3)	4.308
Contour (walls) holes	Volume blasted (m^3)	3.034
	Amount of explosive (kg)	0.960
	Powder factor (P.F.) (kg/m^3)	0.316
Contour (roof) holes	Volume blasted (m^3)	3.844
	Amount of explosive (kg)	1.200
	Powder factor (P.F.) (kg/m^3)	0.312
Volume blasted by contour holes (walls + roof) (m^3)		6.879
Amount of explosive for the contour holes (walls + roof) (kg)		2.160
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.314
Volume blasted by contour holes (floor + walls + roof) (m^3)		10.336
Amount of explosive for the contour holes (walls + roof) (kg)		17.052
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.650

2



Cut holes	Volume blasted (m^3)	0.661
	Amount of explosive (kg)	13.440
	Powder factor (P.F.) (kg/m^3)	20.334
Production holes	Volume blasted (m^3)	8.573
	Amount of explosive (kg)	27.120
	Powder factor (P.F.) (kg/m^3)	3.163
Contour (floor) holes	Volume blasted (m^3)	5.277
	Amount of explosive (kg)	14.892
	Powder factor (P.F.) (kg/m^3)	2.822
Contour (walls) holes	Volume blasted (m^3)	4.521
	Amount of explosive (kg)	0.960
	Powder factor (P.F.) (kg/m^3)	0.212
Contour (roof) holes	Volume blasted (m^3)	5.772
	Amount of explosive (kg)	1.680
	Powder factor (P.F.) (kg/m^3)	0.291
Volume blasted by contour holes (walls + roof) (m^3)		10.294
Amount of explosive for the contour holes (walls + roof) (kg)		2.640
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.256
Volume blasted by contour holes (floor + walls + roof) (m^3)		15.571
Amount of explosive for the contour holes (walls + roof) (kg)		17.523
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		1.126

3



Cut holes	Volume blasted (m^3)	0.661
	Amount of explosive (kg)	13.440
	Powder factor (P.F.) (kg/m^3)	20.334
Production holes	Volume blasted (m^3)	10.868
	Amount of explosive (kg)	27.120
	Powder factor (P.F.) (kg/m^3)	2.495
Contour (floor) holes	Volume blasted (m^3)	7.467
	Amount of explosive (kg)	14.892
	Powder factor (P.F.) (kg/m^3)	1.994
Contour (walls) holes	Volume blasted (m^3)	4.424
	Amount of explosive (kg)	0.960
	Powder factor (P.F.) (kg/m^3)	0.217
Contour (roof) holes	Volume blasted (m^3)	9.420
	Amount of explosive (kg)	2.160
	Powder factor (P.F.) (kg/m^3)	0.229
Volume blasted by contour holes (walls + roof) (m^3)		13.844
Amount of explosive for the contour holes (walls + roof) (kg)		3.120
Powder factor for the contour holes (walls + roof) (P.F.) (kg/m^3)		0.225
Volume blasted by contour holes (floor + walls + roof) (m^3)		21.311
Amount of explosive for the contour holes (walls + roof) (kg)		18.012
Powder factor for the contour holes (floor + walls + roof) (P.F.) (kg/m^3)		0.845

	1	2	3
Cross section (m^2)	5.73	8.27	10.95
Design pull (m)	3	3	3
Actual pull (m)	2.77	2.77	2.77
Efficiency (%)	92.2	92.2	92.2
Blasted volume (m^3)	17.182	24.805	32.839
Volume to be removed (m^3)	24.055	34.727	45.975
Total amount of explosive (kg)	57.612	58.092	58.572
Overall Powder Factor (P.F.) (kg/m^3)	3.353	2.342	1.784

	1	2	3
Holes diameter (mm)	50.8	50.8	50.8
Holes number (-)	33	35	37
Holes density (holes/m^2)	5.76	4.23	3.38
Total drilled length (m)	99	105	111
Look - out angle (°)	5	5	5
Specific drilling (S.D.) (m/m^3)	5.762	4.233	3.380
Total number of detonators (-)	51	55	57
Detonators specific consumption (D.C.) (detonators/m^3)	2.968	2.217	1.736

Project:
Improvement of the drill and blast excavation technique in a Chilean artesanal gold mine

Author:
Simone BEZ
Badge Number: 236349

55

Technical drawing:
MAIN DATA ON THE PHASE 3 SCHEMES

POLITECNICO DI TORINO

Corso di Laurea Magistrale in Ingegneria per l'Ambienete e il Territorio

Supervisor:
Prof. Marilena Cardu

A.Y. 2018 - 2019

POLITECNICO
DI TORINO

TECHNICAL
CATALOGS

RIOCAP
CÁPSULA DETONANTE

Es un accesorio de voladura constituido por una vaina de aluminio que contiene un explosivo base de alto poder y otro iniciador muy sensible a la chispa de la mecha de seguridad, protegido por un opérculo de aluminio proporcionando mayor seguridad en su manejo.

Aplicaciones
- Se utiliza en todo tipo de voladuras para iniciar la detonación de una carga de explosivo primario como: dinamita, booster, hidrogel, emulsión o cordón detonante.

Ventajas
- De fácil uso y aplicación.
- Ideal para voladuras de taladros simples.
- Producto económico

MAXAM

RIOCAP

CÁPSULA DETONANTE

CARACTERÍSTICAS TÉCNICAS

	UNIDAD	VALOR
Potencia	No.	8
Diámetro Interno	mm	5,75
Longitud de Capsula	mm	45
Resistencia al impacto 2 Kg. Desde 0.9 m.	-	NO DETONA
Altura de Carga Explosiva	mm	21

EMBALAJE*

	Cantidad pzas. /caja	Peso Neto (kg)	Peso Bruto (kg)	Dimensiones Caja (cm) Largo	Ancho	Alto
CAJA INTERNA	100	0,153	0,181	6,4	7,5	5,0
CAJA EXTERNA	10000	18,800	19,600	33,0	31,0	26,0

* Medidas y pesos aproximados.

PRECAUCIONES

- Para garantizar su iniciación y buen funcionamiento, debe conectarse correctamente con la mecha de seguridad, utilizando la herramienta de engargolado adecuada para tal efecto.
- Este accesorio debe ser almacenado con productos compatibles y en polvorines aprobados por autoridad competente, alejado del calor, fuego o líquidos inflamables.
- Deber ser manipulado, transportado y utilizado con mucho cuidado, evitando golpes o impactos.

GESTION DE LA CALIDAD
RI-9000-1095

Sistema de Gestión de la Calidad
Certificados según Norma
IRAM ISO 9001 : 2008

MAXAM Fanexa, F.A.M.
Avda. 23 de Marzo, 13
4145 Cochabamba
Tel. +591 44 25 66 93
Fax. +591 44 23 28 15
contact.bo.fanexa@maxam.net

Clasificación

UN N°: 0029
DIVISIÓN: 1.1B

RIOFUSE

MECHA DE SEGURIDAD

Es un accesorio de voladura constituido por un núcleo de pólvora negra y recubierto por varias envolturas de hilo y papel e impermeabilizado por baños de asfalto y recubierto con plástico, permitiendo en esas condiciones una combustión continua, sin interrupciones de la pólvora y a una velocidad constante.

Aplicaciones

- Se utiliza para iniciar cápsulas detonantes, las cuales a su vez podrán detonar todo tipo de explosivo primario como pueden ser: dinamita, booster, cordón detonante, hidrogel y otros.

Ventajas

- De fácil uso y aplicación.
- Resistente a la tracción y abrasión.
- Producto impermeable.

MAXAM

RIOFUSE
MECHA DE SEGURIDAD

CARACTERÍSTICAS TÉCNICAS

	UNIDAD	VALOR
Peso Pólvora	g/m	6,0 ± 0,2
Tiempo de combustión	s/m	160 ± 10
Diámetro Externo	mm	5,0 ± 0,1
Resistencia a la tracción	kg	50
Resistencia al agua	-	Buena

EMBALAJE*

Rollos Por Caja	Cantidad (m/caja)	Peso Neto (kg)	Peso Bruto (kg)	Dimensiones Caja (cm) Largo	Ancho	Alto
2 de 500 m	1000	22,4	23,4	51	26	31
10 de 100 m	1000	22,3	23,3	54	28	31

*Medidas y pesos aproximados

PRECAUCIONES

- Para garantizar su aplicación debe conectarse correctamente con la Cápsula Detonante, utilizando la herramienta de engargolado adecuada para tal efecto.
- Este accesorio debe ser almacenado con productos compatibles y en polvorines aprobados por autoridad competente, alejado del calor, fuego o líquidos inflamables.
- Deber ser manipulado, transportado y utilizado con mucho cuidado, evitando golpes o impactos.

Sistema de Gestión de la Calidad
Certificados según Norma
IRAM ISO 9001 : 2008

MAXAM Fanexa, F.A.M.
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Clasificación

UN N°: 0105

DIVISIÓN: 1.4S

Enaex

Anfo Premium®

Agente de voladura de alta calidad

DESCRIPCIÓN

Agente de voladura de alta calidad, fabricado con nitrato de amonio Enaex grado explosivo de baja densidad y alta absorción de petróleo.

INFORMACIÓN DE TRANSPORTE

Agente de Voladura
Clase 1. División 1.5 D
N° NU: 0331
HDS-NCN-01

CARACTERÍSTICAS TÉCNICAS

Densidad vaciado ("pour")	0,77±3% g/cc
Velocidad de detonación	3.600 - 4.100 m/s*
	2.000 - 2.500 m/s**
Presión de detonación	28 Kbar*
Energía	3.818 KJ/Kg
Volumen de gases	1.050 L/Kg
Diámetro mínimo***	1"
Resistencia al agua	Nula

* Confinado en 3" de diámetro.
** No confinado en 3" de diámetro.
*** Confinado.

FABRICACIÓN

- Planta Río Loa, Enaex Servicios S.A
- Planta Teatinos, Enaex Servicios S.A

PRESENTACIÓN

Anfo Premium se entrega en sacos de 25 Kg, de tejido de polipropileno con bolsa interior de polietileno.

USOS

Anfo Premium es especialmente recomendable para uso en pequeño diámetro en minería subterránea y para voladuras de superficie, especialmente cuando se presenta una roca competente, en perforaciones sin agua. Se recomienda utilizarlo en zonas con buena ventilación en minería subterránea.

Enaex

Emultex® CN

Emulsión envasada de diámetro pequeño

DESCRIPCIÓN

Emulsión explosiva envasada de diámetro pequeño, sensible al fulminante Nº 8 diseñada para un amplio rango de aplicaciones en voladuras (excepto minas de carbón).

INFORMACIÓN DE TRANSPORTE

Alto Explosivo
Clase 1. División 1.1D
Nº NU: 0241
HDS-EME-04

CARACTERÍSTICAS TÉCNICAS

Densidad (g /cc)	1,15 ± 4%
Velocidad de detonación* (m/s)	4.600 ± 600
Presión de detonación (Kbar)	61
Energía (KJ/Kg)	3.940
Resistencia al agua	Excelente
Volumen de gases (L/Kg)	930
Potencia relativa al ANFO	
En peso	1,01
En volumen	1,49

* Cartucho de 1 ¼" x 8" sin confinar.

DIMENSIONES Y PESO DE LA CAJA

Largo x Ancho x Alto	48 x 42 x 21 cm
	70 x 31 x 18 cm
	Bruto caja máximo 25 Kg

USOS

- Utilizar como iniciador mínimo un Detonador Nº 8.
- No abrir o "amasar" los cartuchos.
- La duración garantizada de este producto es de 9 meses. Para utilizar con mayor antigüedad solicitar asesoría a Enaex Servicios S.A.

FABRICACIÓN

- Río Loa Plant, Enaex Servicios S.A

DIMENSIONES

Tamaño** (pulgadas)	Unidades por caja	Peso unidad (g)
1 x 8	195	119
1 x 10	153	152
1 x 16	96	240,4
1 1/8 x 8	154	150,6
1 1/8 x 16	77	301,2
1 1/4 x 8	124	186,56
1 1/4x 10	100	231
1 1/4x 16	61	378
1 1/4 x 24	41	555,5
1 1/2 x 8	85	271,7
1 1/2x 10	69	333
1 1/2x 16	43	531
1 1/2 x 24	28	806,5
1 3/4 x 10	51	454,5
1 3/4 x 16	31	735,3
2 x 8	48	480,8
2 x 16	24	961,5
2 x 20	19	1200
2 1/4 x 16	18	1250
2 1/4 x 24	13	1785,7
2 1/2 x 8	30	757,6
2 1/2 x 16	14	1562,5
2 1/2 x 24	10	2272,7
3 x 8	21	1066
3 x 16	10	2272,7
3 1/2x 16	8	2901

* Tamaño y peso de cartuchos son aproximados.** Para otros tamaños consultar a Enaex Servicios S.A.

Detonador No Eléctrico

DESCRIPCIÓN

Consiste en un tubo de choque de largo determinado por el diseño de la voladura, ensamblado a un detonador de alta potencia y período de retardo para iniciar la carga explosiva. El otro extremo del tubo de choque se encuentra sellado y posee un conector plástico tipo Cobra y etiqueta adhesiva que indica el número correspondiente al retardo.

INFORMACIÓN DE TRANSPORTE

Detonador Ensamblado, No Eléctrico
Clase 1.1 B
NU: 0360

CARACTERÍSTICAS

- Un detonador de alta potencia.
- Conector plástico tipo “Cobra”, color azul.
- Etiqueta adhesiva color blanco y numerada en el tubo de choque.
- Tubo de choque color AMARILLO, con un mínimo de 25 kg. de carga de ruptura, resistente a altas y bajas temperaturas, resistente al aceite por 4 días en ANFO a 40ºC, resistente al agua.
- Iniciación confiable de Anfo, Booster APD, explosivos encartuchados y altos explosivos sensibles a cápsulas detonantes.
- La señal de iniciación en el tubo se propaga a través de pliegues, curvas, nudos y ligaduras.
- El tubo de choque queda intacto después de su uso.

(Nota: Como guía aproximada, por cada 2 m de tubo de choque usado en el ensamblaje, añadir 1 ms al período nominal de retardo del detonador)

RANGO DEL PRODUCTO

LONGITUD DE TUBOS DE CHOQUE

La longitud mínima es de 2,4 m y la máxima de 30,4 m.

BENEFICIOS

- Adecuada precisión del retardo pirotécnico.
- Tubo de choque robusto, resistente a los abrasivos y no se enreda.
- Su uso es fácil y simple.
- Componentes altamente visibles.
- Económico, inventario reducido.
- Confiable.
- Bajo nivel de ruido.

INICIACIÓN

Los detonadores no eléctricos Enaex, pueden ser iniciados con cualquiera de los productos señalados a continuación:

- Detonadores Eléctricos.
- Detonadores Electrónicos.
- Cordones Detonantes (mínimo 3g PETN/m)
- Retardo de línea troncal,

EMBALAJE

- Dimensiones por caja: 560 x 290 x 260 (mm x mm x mm)
- Peso bruto de la caja: 18,9 Kg
- Peso neto de la caja: 18,0 Kg

* Para otros largos, consultar a Enaex Servicios

Longitud (m)	Unidades / Caja
2,4	300
3,0 - 3,6 - 4,2	250
4,8 - 5,5	200
6,0 - 7,0	150
9,0 - 10,0 - 12,2	100
15,2	90
16,0 - 18,0 - 20,0	70
24,4	60
30,4	50

Detonador No Eléctrico

RETARDOS

SERIE MS

Nº	ms	SERIE
0	0	MS
1	25	MS
2	50	MS
3	75	MS
4	100	MS
5	125	MS
6	150	MS
7	175	MS
8	200	MS
9	250	MS
10	300	MS
11	350	MS
12	400	MS
13	450	MS
14	500	MS
15	600	MS
16	700	MS
17	800	MS
18	900	MS
19	1000	MS LP
20	1100	MS
21	1200	MS
22	1300	MS
23	1400	MS LP
24	1800	LP
25	2400	LP
26	3000	LP
27	3800	LP

RETARDOS

SERIE MS

Nº	ms	SERIE
28	4600	LP
29	5500	LP
30	6400	LP
31	7400	LP
32	8500	LP
33	9600	LP

SEGURIDAD

El tubo de choque no puede iniciarse por:

• Transimisiones de radiofrecuacia.

• Radiaciones.

• Corrientes parásitas o vagabundas.

• Descargas electroestáticas.

• Quemado/calor/fricción.

ALMACENAJE

• Almacenar en un lugar apropiado, fresco y seco.

• Almacenar en un lugar apropiado, de baja humedad y buena ventilación.

• Cumplir con los requisitos legales concernientes al almacenaje de su localidad.

• La vida útil del producto es de 36 meses a partir de la fecha de su fabricación.

• Las temperaturas sobre 90ºC pueden producir explosiones espontáneas.

• Hacer rotar siempre las existencias (primero en entrar, primero en salir).

Detonador No Eléctrico

APLICACIONES

Los detonadores no eléctricos Enaex son utilizados en minería de superficie como subterránea y obras civiles.

PRECAUCIONES ESPECIALES

• Manipule con cuidado. Los daños al conector o al tubo pueden producir fallas de iniciación.

• Mantenga siempre los detonadores apuntando en dirección opuesta al suyo.

• Impactos severos sobre el detonador pueden resultar en una detonación prematura.

• Nunca tirar, estirar o retorcer el tubo de choque.

• Nunca conectar la voladura o disparo antes de haber finalizado con las operaciones de carguío y personal retirado de la zona.

LÍMITES DE RESPONSABILIDAD

Esta información resume los mejores conocimientos a la fecha de su emisión sobre las capacidades del producto. Cada usuario deberá revisar previamente esta Ficha Técnica y dentro del contexto de la forma en que tiene intenciones de usar el producto. Para información adicional sobre este producto y su almacenaje, les agradecemos ponerse en contacto con Enaex a través de su administrador de servicios y/o suministro al fono: +56 2 28377777, ó al email: enaex@enaex.com.

Detonador No Eléctrico Dual

DESCRIPCIÓN

Consiste en un tubo de choque, de largo determinado por el diseño de la voladura. Contiene un detonador en un extremo con el tiempo de retardo requerido por la aplicación para iniciar la carga explosiva en el fondo de la perforación, y un detonador de Mili retardo en el otro extremo para inicio de Tubo de Choque. Este detonador No Eléctrico se encuentra alojado en un conector plástico tipo block, el cual está codificado por colores, de acuerdo al retardo del detonador de superficie que contenga.

CARACTERÍSTICAS

- Un detonador de fondo de alta potencia.
- El tiempo de retardo y longitud del tubo de choque se encuentran impresos en las etiquetas del tubo de choque.
- Conector con codificación de colores, capaz de contener 6 tubos.
- Tubo de choque color AMARILLO, con un mínimo de 25 kg de carga de ruptura, resistente a altas y bajas temperaturas, resistente al aceite por 4 días en ANFO a 40ºC, resistente al agua.
- Iniciación confiable de tubos de choque en ambas direcciones, Booster APD, explosivos encartuchados y altos explosivos sensibles a cápsulas detonantes.
- La señal de iniciación en el tubo se propaga a través de pliegues, curvas, nudos y ligaduras.
- El tubo queda intacto después de su uso.

RANGO DEL PRODUCTO

LONGITUD DE TUBOS DE CHOQUE

La longitud mínima es de 3,6 m y la máxima de 30,4 m.

INICIACIÓN

Los detonadores ensamblados Dual, pueden ser iniciados con cualquiera de los productos señalados a continuación:

- Detonadores Eléctricos.
- Detonadores Electrónicos.
- Detonadores No Eléctricos Dual ENAEX.
- Retardo de línea troncal,

INFORMACIÓN DE TRANSPORTE

Detonador Ensamblado, No Eléctrico
Clase 1.1 B
N° NU: 0360
HDS-DET-01

EMBALAJE

- Dimensiones por caja: 560 x 290 x 260 (mm x mm x mm)
- Peso bruto de la caja: 18,9 kg
- Peso neto de la caja: 18,0 kg

*Para otros largos, consultar a Enaex Servicios

Longitud (m)	Unidades / Caja
3,6	200
4,2 - 4,8	190
10,0	110
12,2	90
15,2	80
16,0 - 17,3 - 18,0	75
20,0 - 24,4 - 30,4	50

RETARDOS

TIEMPO DE RETARDOS (MS)	TIEMPO DE RETARDO (MS)	TIEMPO DE RETARDO (MS)	COLOR DEL BLOCK PLÁSTICO
	9 / 600	9 / 1000	Blanco
17 / 500			Azul
	17 / 600	17 / 1000	Azul
	25 / 600	25 / 1000	Naranja
		35 / 1000	Verde
		42 / 1000	Amarillo

Nota:

- Como guía aproximada, por cada 2 m de tubo de choque usado en el ensamblaje, añadir 1 ms al período nominal de retardo del detonador.

Enaex

Detonador No Eléctrico Dual

ALMACENAJE

• Almacenar en un lugar apropiado, de baja humedad y buena ventilación.

• Cumplir con los requisitos legales concernientes al almacenaje de su localidad.

• La vida útil del producto es de 36 meses a partir de la fecha de su fabricación.

• Las temperaturas sobre 90ºC pueden producir explosiones espontáneas.

• Hacer rotar siempre las existencias (primero en entrar, primero en salir).

BENEFICIOS

• Adecuada precisión del retardo pirotécnico.

• Tubo de choque robusto, resistente a los abrasivos y no se enreda.

• Su uso es fácil y simple.

• Componentes altamente visibles.

• Económico, inventario reducido.

• Fiable.

• Bajo nivel de ruido.

APLICACIONES

Los detonadores no eléctricos Enaex son utilizados en canteras, minas a cielo abireto, en proyectos de ingeniería civil y en excavaciones de zanjas.

SEGURIDAD

• Transimisiones de radiofrecuacia.

• Radiaciones.

• Corrientes parásitas o vagabundas.

• Descargas electroestáticas.

• Quemado/calor/fricción.

PRECAUCIONES ESPECIALES

• Manipule con cuidado. Los daños al conector o al tubo pueden producir fallas de iniciación..

• Mantenga siempre los detonadores apuntando en dirección opuesta al suyo.

• Evitar golpear el detonador.

• Nunca tirar, estirar o retorcer el tubo de choque.

• Nunca conectar la voladura o disparo antes de haber finalizado con las operaciones de carguío y personal retirado de la zona. Para una aplicación confiable, asegúrense que el detonador se encuentre en la columna explosiva.

LÍMITES DE RESPONSABILIDAD

Esta información resume los mejores conocimientos a la fecha de su emisión sobre las capacidades del producto. Cada usuario deberá revisar previamente esta Ficha Técnica y dentro del contexto de la forma en que tiene intenciones de usar el producto. Para información adicional sobre este producto y su almacenaje, les agradecemos ponerse en contacto con Enaex a través de su administrador de servicios y/o al email: enaex@enaex.com.

ALTOS EXPLOSIVOS

CORDÓN DETONANTE - PENTACORD®
ESPECIAL 80P

Descripción y composición

FAMESA EXPLOSIVOS S.A.C. fabrica el CORDÓN DETONANTE 80 P con una tecnología altamente desarrollada, que le permite satisfacer las necesidades del mercado.

El CORDÓN DETONANTE 80 P está constituido por un núcleo de pentrita (PETN), recubierto con fibras sintéticas y forrado con un material plástico.

Usos

Permite realizar trabajos en voladuras especiales tales como:

• En minería superficial: Donde resulta necesario la iniciación axial instantánea de la columna explosiva.

• En minería subterránea: En voladuras de “recorte”, “precorte” y “voladura amortiguada”, obteniéndose superficies uniformes en los contornos finales de los túneles y cámaras subterráneas.

Características técnicas

	Cordón detonante-Pentacord® Especial 80 P
Color del cordón detonante 80 P	Blanco
Peso carga (g/m)	80
Sensibilidad al fulminante	N° 6
Velocidad de detonación (m/s)	Min. 6 800

Presentación

Embalaje tipo 1.1B	Material de caja	Capacidad de caja	Peso neto (kg)	Peso bruto (kg)	Dimensiones exteriores (cm)
Cordón detonante-Pentacord® Especial 80P	Cartón	2 Rollos x 100 m	21,4	23,4	65,8 x 33,0 x 22,3

Transporte

Clase: 1
División: 1.1 D
N° ONU: 0065

FAMESA EXPLOSIVOS

MANIPULEO Y ALMACENAMIENTO

Los explosivos y accesorios de voladura de FAMESA EXPLOSIVOS S.A.C. son productos seguros, pero en manos inexpertas se constituyen en elementos peligrosos. El adquirente o usuario debe cumplir con lo establecido por las normas correspondientes, al momento de su transporte, almacenaje y uso, así como entrenar debidamente a todo el personal encargado de su manipuleo.

FAMESA EXPLOSIVOS S.A.C., no asume responsabilidad alguna por el transporte, almacenaje y/o uso inadecuado que pudiera darse a sus productos. El transporte, almacenamiento, manipuleo y uso debe hacerse en concordancia con las regulaciones y aprobaciones por la autoridad competente. Debe ser almacenado en polvorines ubicados en zonas seguras, protegidos de la lluvia y el calor; y en concordancia con la tabla de compatibilidad vigente de la autoridad competente.

ATENCIÓN

La información y recomendación aquí descrita no cubren necesariamente todas las aplicaciones del producto ni las distintas condiciones bajo las cuales éste sea utilizado. Estas se basan en la experiencia, investigación y pruebas realizadas por Famesa Explosivos S.A.C., quien no garantiza resultados favorables ni asume responsabilidad alguna, expresa o implícita en conexión con el uso de estas sugerencias. Este producto puede ser modificado sin previo aviso.

Ficha Técnica
Britacord
Cordón detonante de bajo gramaje

Britacord

Cordón detonante de bajo gramaje

APLICACIÓN

En amarres de desmontes primarios y secundarios, en Minería a Cielo Abierto, subterráneas, canteras y obras civiles.

Iniciación recomendada:
Fulminante N°8/Brinel® /Detonador Electrónico

INFORMACIÓN DE TRANSPORTE

Cordón detonante
Clase 1. División 1.1 D
N° NU: 0065

CARACTERÍSTICAS TÉCNICAS

Britacord®	NP 03	NP 05	NP 05R	NP 10R
Color del revestimiento	Verde	Azul	Azul	Naranja
Carga lineal (g/m)	3,0	5,0	5,0	10,0
Diámetro externo (mm)	3,2	3,8	3,8	4,8
Resistencia a la tracción (kgf)	50	50	80	110
Longitud del carrete (m)	500/1.000	375/750	375/750	250/500
Cantidad de Carrete por cajas (unid)	2/1	2/1	2/1	2/1
Velocidad de detonación (m/s)	6.800	6.800	6.800	6.800

EMBALAJE

Dimensiones (L x A x A cm)	27 x 27 x 27
Altura máxima de apilamiento (m)	2,0
Tipo de caja	Caja de cartón

RESISTENCIA AL AGUA

NP 03	NP 05	NP 05R	NP 10R
3,7 gr/m	5 gr/m	5 gr/m	10 gr/m
48 horas	48 horas	24 horas	24 horas

EMBALAJE

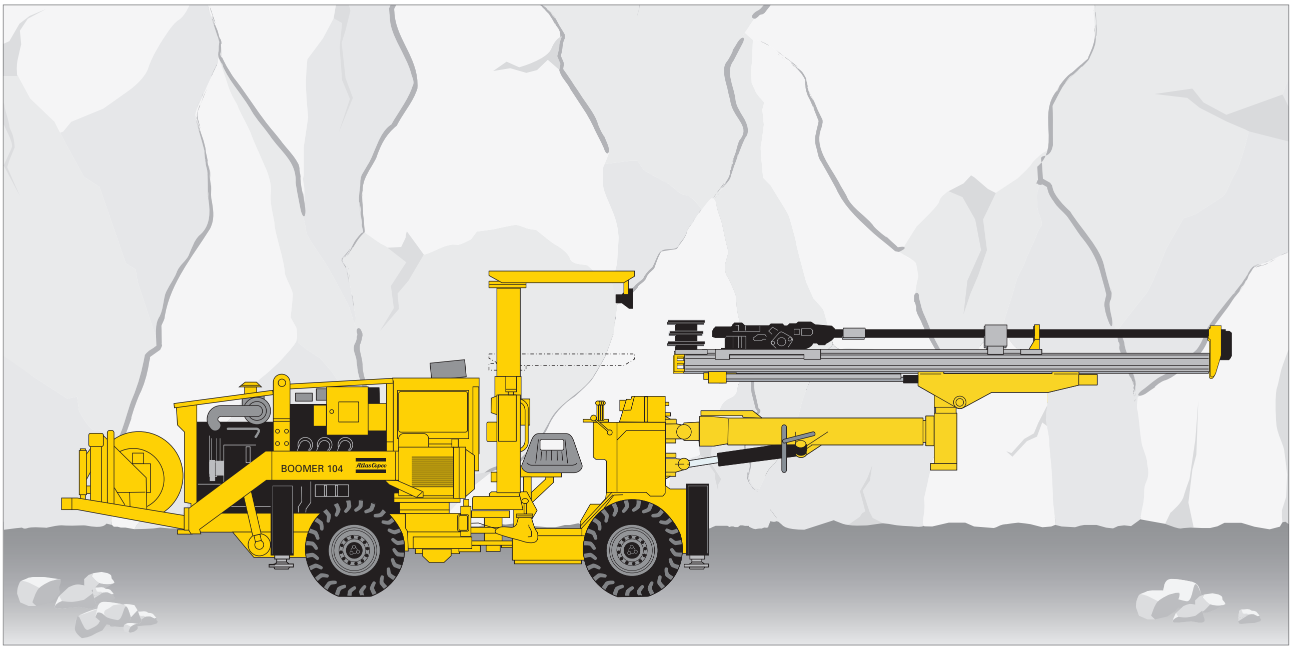
- Conservado en su embalaje original es almacenado en condiciones normales de temperatura y humedad, conforme reglamentación de la R-105*. El producto es garantizado por 24 meses, después de su fecha de fabricación.
- Reglamentación para fiscalización de productos controlados.

CARACTERÍSTICAS

- A diferencia del cordón reforzado tradicional con hilos externos y cera, este cordón de 10 gramos/m reforzado, tiene hilos adicionales que aumentan la resistencia a la tracción de 80 Kgf. con relación al no reforzado del mismo gramaje.
- El metraje va indicado en cada metro de cordón facilitando el control de stock y auditorías internas y externas.
- El PETN está recubierto por aditivos que impermeabilizan los cordones detonantes Britanite, lo que da un agregado en el empleo en pozos con presencia de agua.

Boomer 104

Equipped with a COP 1638 or COP 1838ME rock drill



Direct controlled hydraulic tunnelling and mining rig with one BUT 4B boom and either a COP 1638 or a COP 1838ME rock drill for drilling narrow tunnel cross sections.

Standard features

» Rock drill

- COP 1638 or COP 1838ME rock drill with dual-damping system for optimal drill consumable life
- Pressurized housing and mating surfaces to reduce internal contamination of the rock drill

» Hydraulic boom

- BUT 4B heavy-duty boom for fast and flexible positioning between holes

» Hydraulic feed

- BMH 2800 heavy-duty aluminium feed with high bending and torsional resistance for maximum durability
- Snap-on stainless steel sleeves and polymer contacts

» Control system

- Hydraulically controlled drilling system which incorporates the anti-jamming function RPCF (Rotation Pressure Controlled Feed force)

» Carrier

- Sturdy, articulated carrier with four-wheel drive
- Powered by a 4-cylinder diesel engine
- Four jacks for stable set-up

» General

- FOPS-approved telescopic protective roof
- Cable reel
- Electrically driven compressor
- Hydraulically driven water booster pump
- Working lights

Specifications

ROCK DRILL

	COP 1638	COP 1838ME
Shank adapter	R32/R38/T38	R32/R38/T38
Height over drill centre	88 mm	88 mm
Length without shank	1 008 mm	1 008 mm
Impact power	16 kW	18 kW
Impact rate	60 Hz	60 Hz
Hydraulic pressure	200 bar	230 bar
Rotation system	Separate rotation	Separate rotation
Rotation speed	0–340 rpm	0–340 rpm
Rotation torque, max	640 Nm	640 Nm
Lub. air consump. at 2 bar	6 l/s	6 l/s
Water consumption	1.1 l/s	1.1 l/s
Weight	170 kg	170 kg
Sound level	<106 dB(A)	<106 dB(A)

FEED

BMH 2000-series	BMH 2825	BMH 2831	BMH 2837
Total length	4 087mm	4 677 mm	5 287 mm
Drill rod length	2 500 mm	3 090 mm	3 700 mm
Hole depth	2 225 mm	2 814 mm	3 405 mm
Weight incl. rock drill	455 kg	470 kg	490 kg
Feed force	15 kN	15 kN	15 kN

» Boom

- Boom .... BUT 4B
- Feed extension .... 1 500 mm
- Boom extension .... 900 mm
- Feed roll-over .... 360°
- Feed rotation .... ±114°
- Feed dump cylinder .... +18°/-79°
- Max. swinging angle .... ±30°
- Weight, boom only .... 1 042 kg

» Air System

- Compressor .... Atlas Copco LE3
- Capacity, max at 7 bar .... 4.4 l/s
- Air pressure gauge

» Water System

- Hydraulic driven water booster pump CR5-13
- Water booster pump max capacity at 7 bar .... 80 l/min
- Min. water inlet pressure, bar .... 2 bar
- Water flow guard

» Control system

- Direct hydraulic Control System, DCS
- Basic functions such as anti-jamming included

» Electrical system

- Total installed power .... 59 kW
- Main motors .... 1 x 55 kW
- Voltage .... 380–1 000 V
- Frequency .... 50–60 Hz
- Starting method .... star/delta (380–690 V) direct (1 000 V)
- Thermal overload protection for electric motors
- Percussion hour meter on operator display
- Digital voltmeter/amperage meter in electric cabinet
- Phase sequence indicator
- Earth fault indicator
- Battery charger
- Transformer .... 1.8 kVA
- Cable reel, inner/outer diameter, mm 500/750

» Hydraulic system

- Hydraulic pumps .... 1 unit
- Pumps unloaded at start
- System pressure, max .... 150–230 bar
- Hydraulic oil tank, volume max/min .... 124 l
- Low oil level indicator
- Electric oil filling pump
- Oil filter indicator
- Water cooled oil
- Filtration .... 16 µm
- Mineral hydraulic oil
- Rock drill lubrication warning kit

» Carrier

- Deutz 4-cylinder, F4L 912W, 4 stroke prechamber diesel engine
- Power rating at 2 300 rpm .... 42 kW (57 hp)
- Torque at 1 550 rpm .... 198 Nm
- Articulated steering .... ±40° steering angle
- Catalyser
- Silencer
- Fixed seat for tramming with safety belt
- Four-wheel drive
- Hydrostatic power steering system
- Hydrostatic transmission
- Front axle .... Dana 171
- Rear axle .... Dana 171, ±7° oscillation
- Tyres .... 8.25 x R15
- Clearance outside axles .... 15°
- Hydraulic jacks, front .... 2 extendable
- Hydraulic jacks, rear 2
- Service brakes .... 2 separate circuits
- Emergency and parking brakes .... SAHR
- Fuel tank, volume .... 60 l
- Electric system .... 24 V
- Batteries .... 2 x 70 Ah
- Tramming lights .... 4 x 70 W
- Working lights .... 2 x 200 W, 24 V
- Central lubrication system
- Spirit level
- Fire extinguisher
- Gradeability at max. load on drive wheels 1:4
- Horn, beacon and reverse alarm

DRIFTER RODS

	Dimension	Min. hole diameter
mm	R32-H35-R38	45
	R32-H35-T38	45
	R32-H35-T38 Speedrod	45
	SR35-H35-R38 Speedrod	45
	SR35-H35-T38 Speedrod	45
	SR35-H35-T38	45
	SR35-R39-T38	45
	R32-R39-T38	45
	SR35-R39-T38	45
	SR35-R39-T38 Speedrod	45

SHANK ADAPTERS

	Thread	Diameter	Length
mm	R38	38	435
	T38	38	435
	R32	38	525
	T38*	38	525

**Intended for RAS and extension drilling with BSH 110*

COUPLINGS

	Thread	Diameter	Length
mm	R38	55	170
	T38	55	190

EXTENSION RODS FOR INJECTION DRILLING/RAS

	Dimension	Min. hole diameter
mm	Rnd 32 Speedrod	51
	Rnd 38 Speedrod	64

Optional equipment

» Rock drill/Drilling system
- Big hole drilling system
- Hole blowing kit with external air supply
- Water mist flushing with external water and air supply

» Feed
- BMHT 2800-series, with wider 1 510 mm front axle, max 3 700 mm drill rod

» Boom
- Feed Angle Measurement system (FAM 1)

» Protective roof
- Working lights halogen, 2 x 500 W, 230 V
- Spot light left and/or right side, 70 W
- Outlet for communication radio, 12 V

» Hydraulic system
- Hydraulic oil thermostat
- Heater kit for oil tank

» Carrier
- Fire suppression system ANSUL manual or checkfire
- Solid tyres

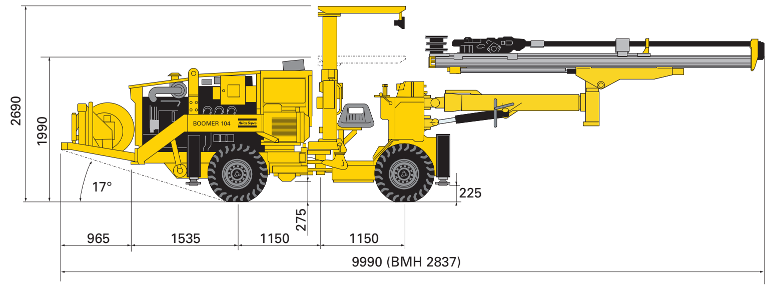
» Cabin
- FOPS-approved cabin, noise level <85 dB(A), including:
  - Air conditioning unit
  - Fixed seat

» Electrical system
- Electric cable type H07RN-F or Buflex
- Plug for cable
- Switch gear

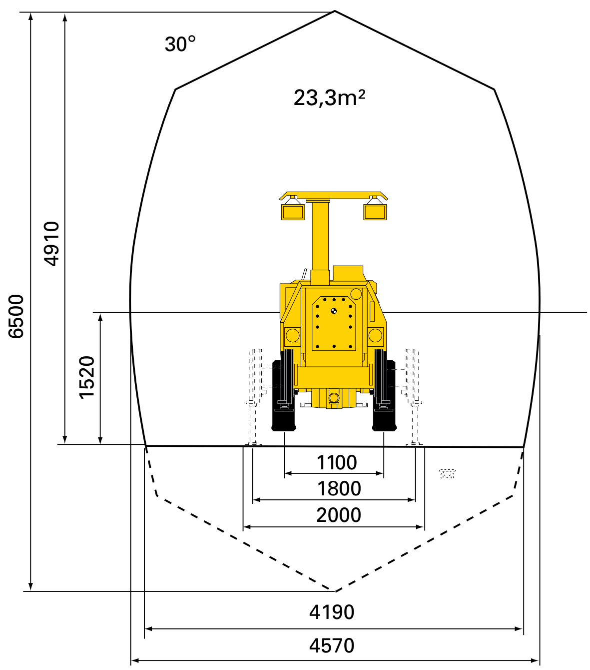
» Miscellaneous
- Rod holder for extra drill rods
- Ni-Cr plated piston rods

Measurements

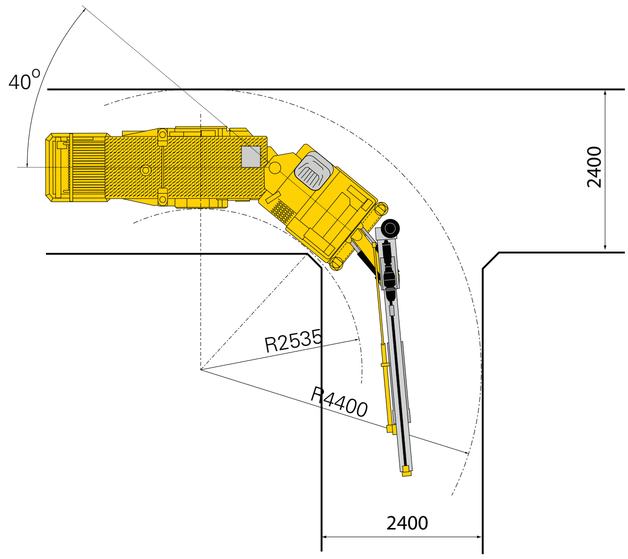
SIDE VIEW



COVERAGE AREA



TURNING RADIUS



DIMENSIONS

mm		
	Width	1 220
	Height, roof down	1 990
	Height, roof up	2 690
	Height, optional cabin	2 572
	Length with BMH 2837 feeds	9 990
	Ground clearance	275

RECOMMENDED CABLE SIZE AND LENGTH

Voltage	Type	Dimension, mm²	Diameter, mm	Length, m
380–400 V	Buflex	3x50+3x10	33	85
440–700 V	RDOT	4x35	38	85
1 000 V	Buflex	3x25 + 3x6	26	140

Recommendations are given for surrounding temperature of 40 °C and up to a height of 2 000 m.

TRAMMING SPEED

km/h	On flat ground (rolling resistance 0.05)	>8

WEIGHT

gross weight, depending on configuration		
kg	Total	9 000

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Atlas Copco