Technical and operational aspects of tunnel rounds in artisanal underground mining

Aspectos técnicos e operacionais de desmontes com explosivo na mineração artesanal subterrânea

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Abstract

In the world today, due to the current high price of gold, thousands of artisanal small-scale mines operate without the financial or technical capacity to upgrade their production methods, often using equipment and working methods that were forsaken decades ago in the conventional mining industry. This article illustrates, with a practical example, that it is possible to achieve the modern requirements of quality, safety and productivity, while operating with mining equipment that basically possesses the same characteristics as that employed in the 1950s. The operation involved the excavation of a 6 meter long tunnel and over 25 stoping sections in both rock and concrete by drilling and blasting in an underground gold mine. The article describes how the main operational problems were overcome. An effective drilling pattern was designed in order to be easily achievable with the limited local equipment. The behavior of the local nitrate explosive was effectively predicted and managed. The interaction with the drilling teams was managed in order to determine a meeting point between their traditional working methods and the scientific view of the engineer. At the end of the article some general guidelines are proposed for the management of similar situations, in order to improve the efficiency, productivity and safety of drill and blast operations in small-scale mines.

Keywords: underground mining, small-scale mining, drill and blast, pyrotechnic ignition

Resumo

Atualmente no mundo, devido ao elevado preço do ouro, milhares de pequenas minas artesanais operam sem a capacidade financeira ou técnica para atualizar seus métodos de produção, muitas vezes utilizando equipamentos e métodos de lavra que foram abandonados há décadas na indústria mineral convencional. Esse artigo ilustra, com um exemplo prático, que é possível atingir os modernos requisitos de qualidade, segurança e produtividade, mesmo operando com equipamentos de lavra que possuem basicamente as mesmas características dos utilizados na década de 1950. A operação envolveu a escavação de um túnel de 6 metros e o desmonte de mais de 25 seções em rocha e concreto, em uma mina de ouro subterrânea. O artigo descreve como os principais problemas operacionais foram superados: a concepção de um padrão de perfuração eficaz que foi facilmente alcançável com o limitado equipamento local, a previsão do comportamento e a gestão do local explosivo com base em nitrato e a interação com as equipes de perfuração, a fim de determinar um ponto de acordo entre seus métodos de trabalho tradicionais e a visão científica do engenheiro. No final do artigo, são propostas algumas diretrizes gerais para a gestão de situações semelhantes, a fim de serem melhoradas a eficiência, a produtividade e a segurança das operações de perfuração e desmonte em minas subterrâneas de pequena escala.

Palavras-chave: *lavra subterrânea, pequena mineração, perfuração e desmonte, iniciação pirotécnica.*

1. Introduction

Artisanal mining (AM) is typically dedicated to the exploitation of marginal deposits of very valuable materials (such as gold, diamonds and other gemstones). The profitability of such activities, despite their low efficiency and productivity (Peixoto and De Lima, 2000), results from the high market prices of these commodities (Veiga, 1997, Hilson 2002, Hentschel et al., 2002, Shena and Gunsonb 2006, Hruschka and Echavarría 2011, Seccatore et al., 2014). Until recently, AM was performed almost exclusively in easily accessible areas such as alluvial or low-consolidated deposits located near to the natural surface. Underground mining of deep deposits was essentially out of the reach of artisanal operations until a few years ago (Veiga 1997, Hilson 2002). Only in the last years, has the big gold price boom of the late 2000s changed this condition. Nowadays, even inefficient operations characterized by very low production can justify the initial investment in underground mining equipment, with the recent price of gold making such activities profitable. In 2012 a Canadian and Brazilian joint-venture (JV) decided to invest in a project dedicated to implement an artisanal mining operation (described in the next section) into a small-scale industrial unit. For doing so, the mine owner (MO) and the JV set up a new dedicated company aimed to overcome the lack of capital that has prevented, up to now, proper planning and technical upgrade for the extension of the operation.

2. The Mine

The mine is located close to the town of Portovelo, in southern Ecuador in the province of El Oro. This area has historically been dedicated to gold extraction since the pre-Columbian age (the name “El Oro” meaning “The Gold”). The area of interest was industrially exploited by the USA-based SADCo Company from 1896 until the 1950s; later the property passed to a publicly-owned Ecuadorian company, and finally the mine closed in the 1980s. After the mine’s closure, artisanal miners entered in the tunnels left by the mining companies in the area, in order to mine the small portions of the mineral veins left in place. In 1991, with the “Ley da Mineria” (Mining Law), the Ecuadorian government legalized small-scale mining activities, allowing the miners to move away from dangerous methods, providing access to technical national and international cooperation programs (COSUDE project, PRODEMINCA project amongst others) and attracting investors (Sandoval, 2001). The mine that is the object of this study is legally operating under Ecuadorian law as a small-scale mine. After the legalization, the mining techniques improved in the region. Despite this, it is possible to observe that in the majority of mines the technical level remains very low and the working methods are semi-artisanal (Sandoval, 2001). The veins exploited in the mine are part of the Portovelo epithermal vein system. According to Van Thournout et al. (1996), the volcanic-hosted altered-mineralized system covers an area of more than 150 km2. More than 30 gold-bearing veins are identified up to now. They dip at angles of 45 to 70°, have lengths of hundreds of meters up to 2.5 km, and thickness of 0.8–1.5 m, in some places reaching 3–5 m (Vikentyev et al., 2005). In the mine object of this study, the embedding rock is hard andesitic basalt, and two veins of economic value are intercepted. The mine is developed on a single level, the access to the ore body is provided by an adit (approximatively 2 x 2.5 m and 850 m long) connected to the veins by drifts driven alongside them. The JV manifested the interest to explore those tunnels and carry out core-drilling for the determination of mineral grade. It has been necessary, therefore, to perform underground blasts to open a 6 meter long tunnel to host the drilling machine. Since the drilling machine cannot be disassembled, in order to bring it in place, it was necessary to demolish the massive concrete wall protecting the entrance and to stope blast over 25 sections of the adit in order to widen it. The MO provided the equipment and workforce to perform this preliminary excavation. This gave the operations their peculiar quality, having to achieve the modern requirements of quality, safety and productivity while operating with mining equipment that basically possesses the same characteristics as that employed in the 1950s.

3. Materials

The available equipment consisted of the following items.

Drilling equipment: A single pneumatic drill was available. It was a manually-handled model (Figure 1a) with a rotary-percussive mechanism, using water as drilling fluid and sustained by a pneumatic jack. The only rods available had a length of 1.20 m or 1.60 m. The bits had a diameter of 1”1/2. The average drilling time for one blasthole was about 4–5 minutes.

Loading and hauling equipment: The loading equipment consisted of a single pneumatic loader (locally called lampón, see Figure 1c). The hauling equipment consisted of electric locomotors and a few hauling wagons (burras, Figure 1b) which were unloaded by lateral tipping.

Explosives: In the mine two kinds of explosives are used:

i) Dynamite (NGL 80%) as booster and bottom charge, in cartridges of 1”1/8 x 8”;

ii) Ammonium Nitrate (AN) as column charge, in handmade paper cartridges (Figure 2a). Due to the unavailability of other explosives and due to the experience of the workers in the use of these products, they were employed both for tunnel driving and for blasting the concrete door. The dynamite is industrially produced and is delivered to the mine already packed in cartridges. The nitrate explosive is common fertilizer, and is packed in handcrafted paper cartridges for ease of handling. It is composed of granular A.N. (34%N) without the addition of fuel oil. The F.O. was eliminated in the preparation of the charges when the miners empirically observed the consequences of the very negative oxygen balance of poorly mixed ANFO. In order to reduce the hazard of suffocation or air poisoning, ventilation shifts of 4 to 6 hours are left after each blast to ensure the evacuation of blasting fumes and the substitution of exhausted air.

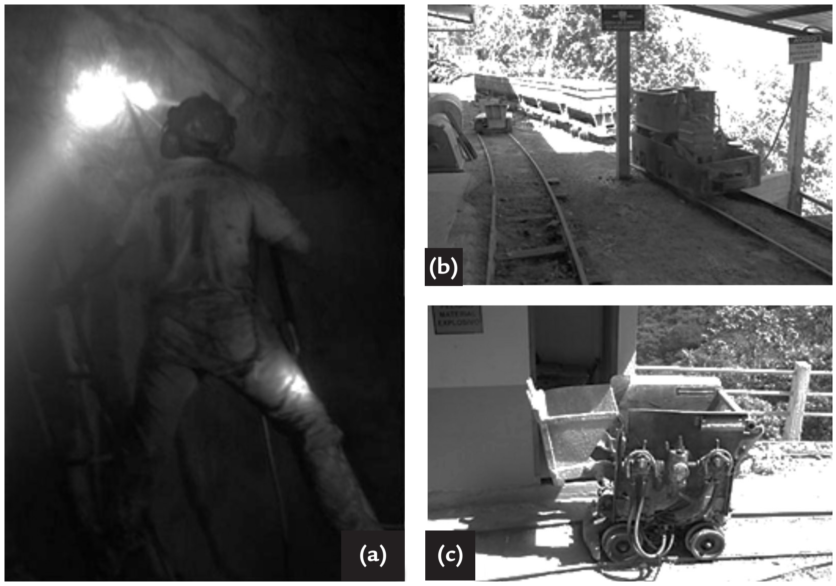

Figure 1
(a) Operator handling the manual pneumatic drill;
(b) Hauling wagons (back) and electric locomotors (front);
(c) Pneumatic loader. In b/w for a better resolution of the pictures taken underground

Figure 2
(a) Ammonium Nitrate bag and empty newspaper cartridges;
(b) charging of a blast and
(c) operators ready to ignite the fuses.
In b/w for a better resolution of the pictures taken underground.

It is still doubtful whether the ammonium nitrate has either the behavior of a detonating explosive or that of a low explosive (deflagrating explosive) under such conditions. According to dedicated studies (Oommen & Jain, 1999) AN decomposes at around 230°C at a pressure of 760 mmHg, and above 325°C it deflagrates. The same authors note that, if confined, ammonium nitrate may detonate at between 260 and 300°C with a detonation velocity ranging between 1250 and 4650 m/s. Such VODs are close to the VOD values indicated by others authors for commercial ANFO (94% of A.N. and 6% F.O.), ranging between 2000 and 4400 m/s (Oloffson, 1988) and between 2400 to 4750 m/s (ISEE, 2011). According to the effective results in terms of fragmentation, it is possible to hypothesize the explosive behavior of the whole blasthole as follows: a) the primed dynamite works effectively as a high detonating charge, producing toe breakage and inducing shockwaves towards the collar that create primary cracking; b) the AN column works as a very loose detonating charge, producing secondary cracking and mainly opening the fractures and moving the pile by means of gas expansion. In order to assess the true behavior of the explosive, it would at least be necessary to analyze the real composition of the product because many accessory materials like chloride ions and heavy metals can catalyze the decomposition, modifying the performance of the explosion. Some field tests should also be carried out in order to analyze the behavior of the product (VOD measurement) and the composition of the exhausted gases under the true operating conditions.

Ignition system: The dynamite cartridges are primed with a blasting cap, and ignited with a safety fuse. The blasting caps come already secured to 2 m of safety fuse and are inserted into the dynamite cartridge outside of the tunnel (Figure 2b). This working method poses two main problems. The first is a safety problem, since the cartridges are transported into the mine already primed (ISEE, 2001). The second problem is both a technical and a safety problem. Since the length of the fuse is fixed and is the same for all the mines, the precise timing of the firing pattern is practically impossible. The fuses are lit with a lantern (Figure 2c), and the firing sequence depends of the order of ignition of the fuses. Really poor control of the timing has an impact on the result of the blast which has to be taken seriously into account. Moreover, the blasters only rely on their experience to know whether the time to leave the blast area is sufficient compared to the length of the fuse. Even if this mine this method has not yet created problems, it cannot be considered a safe practice.

4. Blast design

The design of the blasts was performed after preliminary surveys of the site, taking into consideration the limitations of the available equipment. The main limitations imposed on the design were:

i) the section of the existing tunnels: roughly 2 m wide x 2.5 m high, but with a very irregular profile;

ii) the diameter of the drilling bits: only 1"1/2 was available; the theoretical pull of 1.60 m is imposed by the rod's length;

iii) the size of fragments: due to the size of the loading and hauling equipment (see Figures 1b and 1c) fragmentation had to be taken into account, since large boulders could not be loaded or hauled;

iv) the impossibility of the accurate control of the blast timing.

Cut holes: Due to the reduced section of the tunnel, the opening cuts were designed with parallel holes. The only drilling diameter available made it impossible to realize large-diameter uncharged holes. Hence an opening cut with four central holes drilled close together (see Figure 3b) has been designed to obtain an equivalent diameter of 88 mm. The remainder of the opening cut was calculated with the method suggested by Oloffson (1988). Only three rounds of the opening cut were designed, since the width of the tunnel offered no space for a fourth.

Stoping holes: The stoping holes were distributed with an angle of breakage of around 90°, and the charge was calculated following the method of Olofsson (1988). Considering that the column explosive is AN with a low disruptive power, a lower burden was left for the wall holes and one as low as possible for the floor holes. The calculated charge is shown in Figure 2b.

Firing sequence: The firing sequence was designed considering that two operators ignite the fuses at the same time, each one operating on half of the face (making the coordinated contemporary ignition of two holes on opposite sides of the face possible, see Figure 2c), and that no more than three fuses can be jointed and ignited at the same time.

5. Drilling and blasting operations

Drilling team: The MO company provided two drilling teams, composed of a drill operator and his assistant. The operator handles the machine and drills the blastholes (Figure 1a), while the assistant helps in the handling of the rod to position the bit, operates the admission valves of compressed air and water, and helps during the re-positioning of the grip of the pneumatic jack. The drill operator, who is more experienced, is also the blaster in charge. Team no.1 was composed of two rather young miners, who were available to perform schemes different from those ones they were used to, and who were curious to learn new techniques. Their work was accurate and precise. Team no.2 was composed of a slightly older drill operator who was very attached to the techniques he had always adopted. This resulted in a resistance to perform new techniques, and in order to follow the design, constant supervision of the drilling operations was needed. The accuracy of drilling and therefore the quality of the result depended heavily on the team, which was higher when team no.1 operated.

Accuracy and result of the drilling: Some differences between the design and the realization are unavoidable in each working site. Nevertheless, in artisanal mining both human and technical factors can lead to great discrepancies that must be considered and evaluated. The main problems encountered were the following:

i) depending on the team and on the willingness of the drill operator to follow the instructions of the engineer, the results could vary widely from the design;

ii) it has been necessary to regularize the blasting pattern because the drillers find it operationally easier to visualize and hence realize a grid based on a squared and regular pattern, even in the opening cut (Figure 3b);

iii) the manual drill could not reach the level of the roof holes, and therefore such holes were realized at steeper angles with respect to the horizontal plan than the designed look-out angle (Figure 2b).

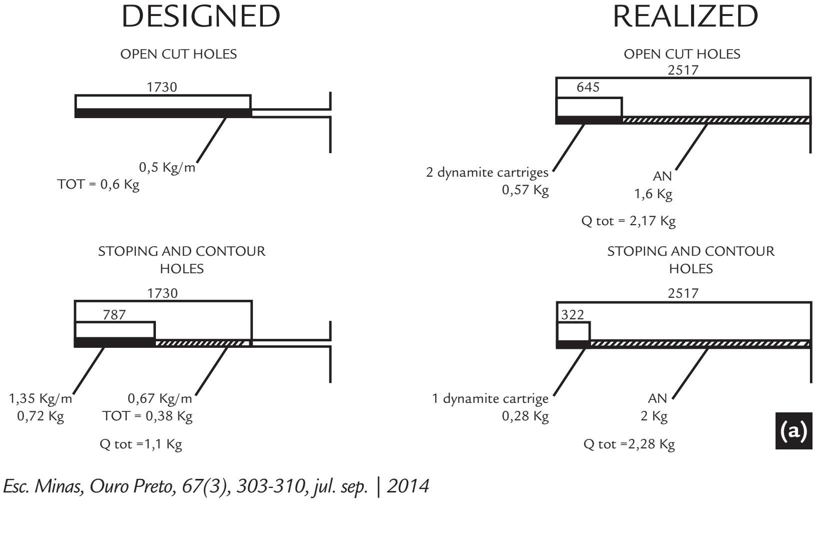

Figure 3- a
Differences between the planned and realized charging of the holes;

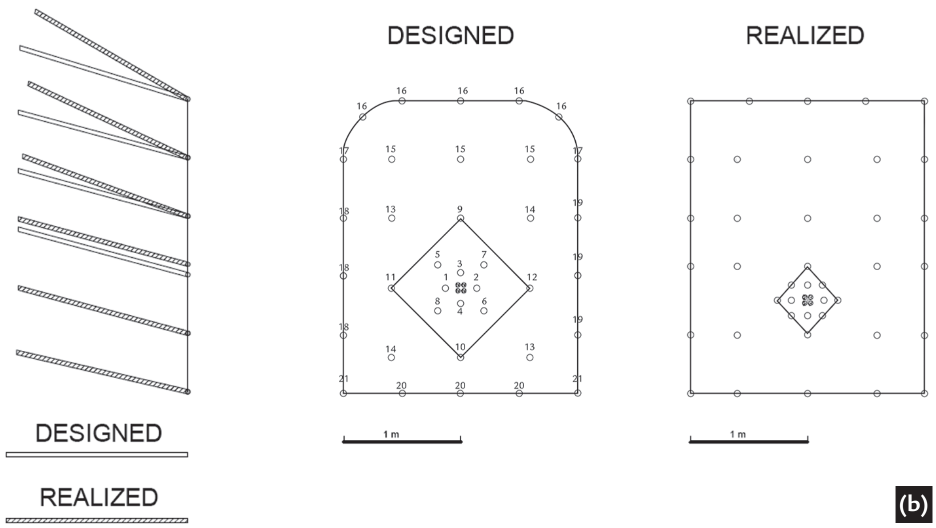

Figure 3- b
Differences between the planned and realized drilling pattern

Charging: In the mine, the holes are traditionally charged as follows: a) opening holes: 1 dynamite primer + 1 unprimed dynamite cartridge + AN up to the collar of the blasthole; b) stoping and contour holes: 1 dynamite primer + AN till the collar of the blasthole (See figure 3a).

This method overcharges the holes. Nevertheless, since psychological resistance was encountered when suggesting a different charging scheme, and considering the unknown behavior of the AN, the blasters were left free to charge the holes in the manner in which they were accustomed. Results proved that this did not affect the results in a significant way in terms of overbreak, suggesting that the behavior of AN is in the range of a low detonating explosive or a deflagrant explosive.

Symbol	Parameter	Unit	Value	
			Designed	Realized
Sh	Design section height	[m]	2.5	
Sw	Design section width	[m]	2	
Ad	Design section	[m^2]	5	
	Open cut section		0.71	0.15
L_d	Pull	[m]	1.6	1.5
Mp	Pull efficiency		0.94	
V	Blasted volume	[m^3]	8	7.5
ρ *rock*	Rock density	[kg/m^3]	2.65	
T	Tonnage of one blast	[t]	21.2	19.9
ϕ	Hole diameter	["]	1.50	1.50
ϕ	Hole diameter	[mm]	38.1	38.1
Q	Charge per hole	[kg]	1.1	2.1
L_Q	Charge length	[m]	1.1	1.5
N	No. of holes		44	44
Q_{TOT}	Charge per round	[kg]	48.4	93.2
P.F.	Powder factor	[kg/m^3]	6.05	12.43

Table 1
Differences between the designed and realized blasting pattern

Results of the blasts: A comparison between designed and achieved blast parameters is shown graphically in Figure 3 and listed in detail in Table 1. The pull efficiency is good (over 90%) despite the large dissimilarity between the designed open cut section and that realized. The dissimilarity, however, caused an increase in the burden on the production holes, and it was observed that the pull efficiency in the cut was systematically higher than in the others parts of the tunnel. In order to guarantee the regularity of the finished tunnel, the cut section in the last round was shifted from the center of the section to the lower left-hand side (Figure 4). A broad measurement of the height and width of the tunnel after each blast showed that the section had approximately the same area as the designed section, and that the overbreak was reasonably low. Some half casts were visible despite the lack of timing. Nonetheless, they were too few to allow the calculation of a half cast factor (Mancini and Cardu, 2001). Figure 4 shows the position of some visible blastholes following the last profiling round. The difference from the planned position is due to the low precision in the positioning of the rod and in setting its inclination. Regarding fragmentation, the only constraint was the size of the pneumatic loader which could not charge fragments larger than 50 cm.

The result was very positive, as no secondary breaking was required and hauling was rapid and efficient.

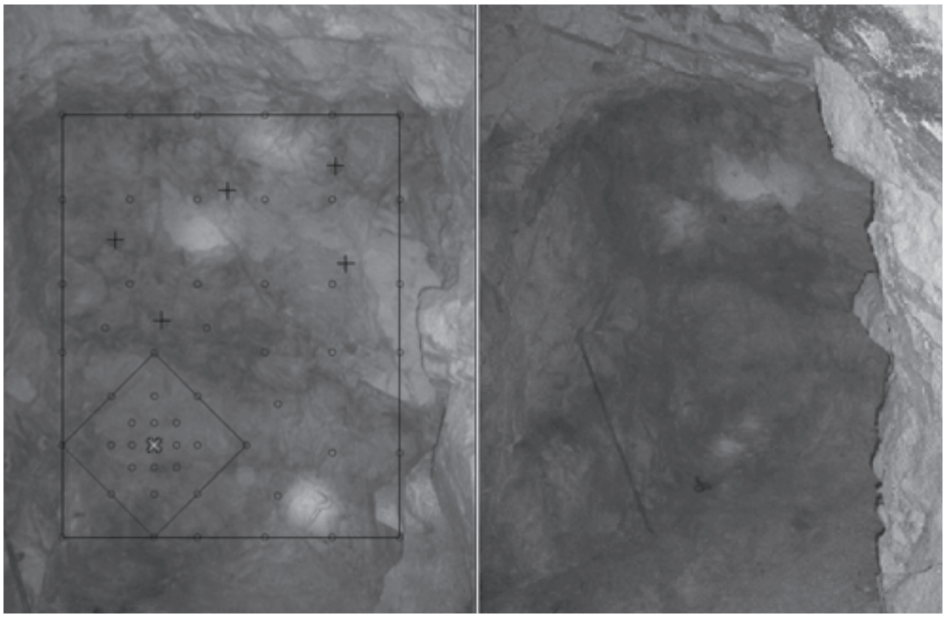

Figure 4
Final result of the blasted tunnel.
In the left picture the blast design is overlaid on the picture of the final tunnel. The impression of over-break is due to perspective, and the profile is better seen in the image on the right. In b/w for a better resolution of the pictures taken underground

6. Other blasting operations

Scaling: The presence of a set of discontinuities creating an unstable wedge made a scaling operation necessary. One hole was drilled in the roof and the wedge was blasted with only one cartridge of dynamite. The blasting of the wedge was the subject of debate with the drillers who, due to their lack of geotechnical knowledge, did not perceive the danger. The result of the blast was the detachment of not only the visible wedge but of about 1/3 m^3 of unstable rock which fell in the tunnel. This operation had a very strong psychological effect on the workers, and broke their initial resistance to the leading role of the outsourcing engineer.

Blasting the concrete frame: The massive concrete frame, built by SADCO, was located at the entrance of the adit to support the metallic door that originally sealed the tunnel. The frame was composed of 1.50 m thick concrete. The presence and position of the steel reinforcement bars was unknown. According to Olofsson (1988) the specific charge for blasting concrete varies greatly with the quality of the concrete, but overall with the amount of steel reinforcement. This author suggests distributing the explosive in as many holes as possible because commercial explosives do not cut steel bars but instead fragment the concrete around them. After a preliminary survey of the door, it appeared that in a previous similar blasting operation, a firing pattern with spacing of about 0.2 m had given a good result; as a result a similar pattern was adopted. The result of the blast was clean and accurate, the frame was totally eliminated and the fragmentation of the concrete was homogeneous. The steel reinforcement mainly consisted of drilling rods and recycled bars that were later cut mechanically.

Stoping along the adit: The widening of over 25 adit sections was performed in order to allow passage of the drilling machine. The undersized sections were individualzeded by a simple effective method: a wooden frame with the same overall shape of the drilling machine was pulled through the adit; the sections in which it got stuck were identified with spray paint. For the widening of the section, stoping blastholes diverging from the axis of the tunnel were drilled in such sections. The holes were loosely charged according to the varying burden. The results of these blasts were very poor in terms of contour, but achieved the goal of widening the section of the tunnel and producing good fragmentation to allow rapid and effective hauling.

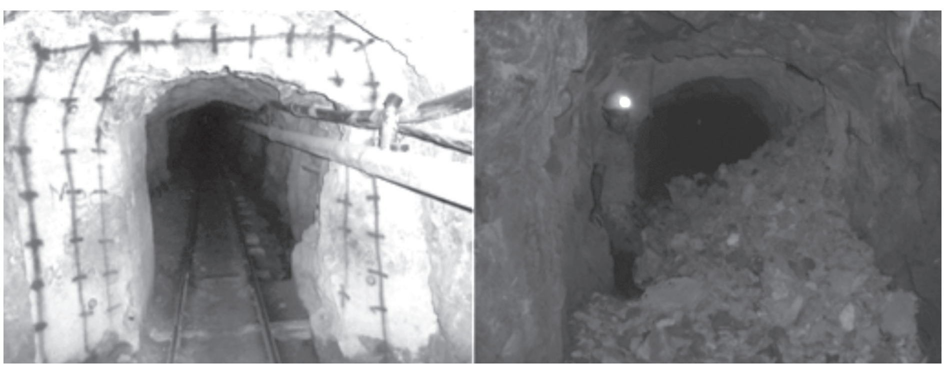

Figure 5
Preparation and result of the demolition blast of the concrete wall. In b/w for a better resolution of the pictures taken underground. In b/w for a better resolution of the pictures taken underground

7. Discussion

The execution of this work provided precious lessons about the main problems that can be encountered in analogous situations of artisanal underground operations.

In Table 2 the main criticalities identified during the work are detailed, along with their possible consequences and some preliminary guidelines for their solution.

Criticality	Consequences	Improvement
Use of outdated equipment	• High vibration level • High noise level • Mechanical hazard • Presence of dust • Low productivity	• Replacement of the equipment with a modern one (out of the reach for many small-scale mining companies) • Generalize the use of personal protective equipment, • Formation and training of the operators
Use of handcrafted explosive	• Explosive hazard • Possible formation of poisonous gas.	Both the workers and the mine management must have knowledge about explosive functioning and characteristics. This will allow them to make responsible choices.
Pyrotechnic initiation of the blast	• Explosive hazard • Impossible timing of the blast.	Shock tube initiation (Timing control can improve the blast performance, decreasing explosive consumption and specific drilling). Shocktubes are already in use in nearby artisanal mines.
Unsafe explosive handling and transport	• Poor blast efficiency • Explosive hazard	Formation and training of the operators on the principles of blast design and safe explosive handling
Absence of ventilation	• Risk of intoxication • Production slowed by the long ventilation shifts	• Short term solution: compressed air as a pressing ventilation system (it offers little reliability because it is impossible to calculate the amount of exhausted air still present at the end of the shift, and the rate of air renewal in the mine). • Long term solution: up-to-date ventilation system (the investment can be out of reach for many small-scale mining companies)
Distrust toward the engineer	• Reticence to follows instructions • Low productivity • Health and safety hazard	In future works, it must be shown as soon as possible a case of clear success of engineering knowledge (the solutions and improvements proposed by the engineer should be effective since the first attempt, because any "failure" will be eternally remembered by the miners, supporting their argument that "engineers don't know": Hentschel *et al.*, 2002).
Lack of geotechnical knowledge	• Hazard of collapses and rock falls	Formation and training of the operators on the hazards of rock fall in underground environment (rock falling is one of the main causes of accidents in small-scale mining: Hentschel *et al.*, 2002).

Table 2
Criticalities and suggested improvement

8. Concluding Remarks

All the criticalities identified during this work can be divided into three main groups:

i) use of unsuitable and unsafe supplies and equipment;

ii) Lack of technical knowledge;

iii) "Human factor". The consequences of these criticalities can be classified in the following two categories: a) productivity limitation and b) health and safety problems. It can be observed that these three categories of criticality match the characteristics commonly attributed in literature to artisanal mining: low productivity, intense labor activity, low technological knowledge, low degree of mechanization and low levels of health and safety awareness (Hentschel *et al.*, 2002, Seccatore et al., 2012). The only charactheristic of artisanal small-scale mining not present in this case is informality, thanks to the presence of the juridical framework dedicated to small-scale mining in Ecuador. During the work, some short-term solutions have been implemented, but in order to obtain significant improvements, it is evident that long-lasting solutions are needed. This is true with regard to health and safety issues because effective solutions should be based on a risk management plan that is implemented from the design phase of the mining project (NIOSH, n.d.; OHS, 2011), but it is also true with regards to efficiency and productivity issues. In order to be effective, an eventual modification of the working method must be carried out with an integrated approach: both modernizing the equipment and training the workers rather than with simple punctual corrections. A long-term solution requires large initial investment and a long time for implementation. For these reasons long-term solutions are commonly out of the reach of artisanal mining companies. Seccatore *et al*. (2013) address this specific aspect and propose a methodology for management of resources and reserves in small-scale mining, reporting an example on how to upgrade the mine object of this study to a responsible small-scale mine.

9.Acknowledgments

A special acknowledgment goes to FAPESP (Foundation for Support to the Research of the State of São Paulo, Brazil) and CNPq (National Counsel of Technological and Scientific Development, Brazil) for their funding of the research effort.

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Received: 23 May 2013 - Accepted: 24 July 2014.