

Diagnosis of the production cycle in the small shaft sinking

Diagnose do ciclo produtivo no aprofundamento de pequenos poços

Diagnóstico del ciclo productivo en la profundización de pequeños pozos

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Abstract

Shaft sinking is a classic activity in underground mines. In shafts with small cross-section or in mines with low mechanization indices it is common to use hand pneumatic drills and blasting by explosive gelatin in cartridges, employing natural draft or flexible ducts with axial fans for gases and fumes exhaustion, muck removal by hand shoveling into hoistable dumping buckets. System of this type has been studied here, consisting of a rectangular cross-section shaft (3.7 m x 2.0 m), with final depth of 94 m, excavated in order to obtain samples for a pilot-scale mineral processing testwork, before the open pit mine's industrial startup. The shaft had a concrete collar and its walls were supported by wooden sets spaced 1.5 m and 25 mm thick wooden planks as liners. This shaft has been excavated in schist rocks belonging to the metamorphosed hydrothermal deposit of copper and gold located in Chapada (municipality of Mara Rosa, Brazil). Daily production worksheets covering one month campaign were statistically analyzed, encompassing the entire cycle of mining operations, namely drilling, charging and blasting, fumes exhaustion, mucking, wall and face trimming and scaling, and assemblage of support system. Operation downtimes were also quantified. Statistical analysis of productivity indices allowed the detection of critical points of the operation and the establishment reference for similar mining operations.

Keywords: Mine work; Underground mine; Small mines; Statistical distribution.

Resumo

O aprofundamento de poços é uma atividade clássica em minas subterrâneas. Em poços de pequena seção transversal ou em minas com baixos índices de mecanização é comum o uso de perfuratrizes pneumáticas manuais e desmonte por gelatina explosiva em cartuchos, empregando tiragem natural ou dutos flexíveis com ventiladores axiais para exaustão de gases e fumos, remoção de material desmontado por paleamento manual e transporte em caçambas basculantes içáveis. Um sistema deste tipo foi estudado, constituído por um poço de seção retangular (3,7 m x 2,0 m), com profundidade final de 94 m, escavado para se obterem amostras para ensaios de beneficiamento mineral em escala piloto, antes posta em marcha industrial da cava a céu aberto. O poço era munido de colar de concreto e suas paredes eram sustentadas por quadros de madeira espaçados de 1,5 m e pranchões de madeira de 25 mm de espessura como revestimento. Este poço foi escavado em rochas xistosas pertencentes ao depósito hidrotermal metamorfoseado de cobre e ouro localizado na Chapada (município de Mara Rosa, Brasil). Foram analisadas estatisticamente as planilhas de produção diária de um mês de campanha, abrangendo todo o ciclo das operações, nomeadamente: perfuração, carregamento e desmonte, exaustão de fumos, remoção do material desmontado, aparelhamento de paredes e faces e montagem de sistema de escoramento. As paradas de operação também foram quantificadas. A análise estatística dos índices de produtividade permitiu a detecção de pontos críticos da operação e o estabelecimento de referência para operações mineiras similares.

Palavras-chave: Trabalho mineiro; Mina subterrânea; Pequenas minas; Distribuição estatística.

Resumen

El hundimiento de pozos es una actividad clásica en las minas subterráneas. En pozos de pequeña sección o en minas con bajos índices de mecanización es común el uso de perforadoras neumáticas manuales y voladuras por gelatina explosiva en cartuchos, empleando tiro natural o ductos flexibles con ventiladores axiales para extracción de gases y humos, remoción de material fragmentado con pala manual y baldes basculantes elevables. Aquí se ha estudiado un

sistema de este tipo, consistente en un pozo de sección rectangular (3,7 m x 2,0 m), con profundidad final de 94 m, excavado con el fin de obtener muestras para ensayos de procesamiento de mineral a escala piloto, antes de la explotación a rajo abierto de la mina. El pozo tenía collar de hormigón y sus paredes estaban soportadas por juegos de madera espaciados 1,5 m y tablonces de madera de 25 mm de espesor como revestimientos. Este pozo ha sido excavado en rocas esquistosas pertenecientes al yacimiento hidrotermal metamorfoseado de cobre y oro ubicado en Chapada (municipio de Mara Rosa, Brasil). Se analizaron estadísticamente las hojas de trabajo de producción diaria que cubrían una campaña de un mes, abarcando todo el ciclo de las operaciones mineras, a saber, perforación, carga y voladura, agotamiento de humos, remoción de escombros, recorte y escalado de paredes, y ensamblaje del sistema de soporte. También se cuantificaron los tiempos muertos de operación. El análisis estadístico de los índices de productividad permitió la detección de puntos críticos de la operación y el establecimiento de referencia para operaciones mineras similares.

Palabras clave: Labores mineros; Minas subterráneas; Pequeñas minas; Distribución estadística.

1. Introduction

The execution of mine workings as shafts and tunnels differs from any other engineering task because the material in question is the rocky massif, which is constituted by rock, families of discontinuities and often water. The uncertainties about the rocky massifs' properties lead to variability of methods and precautions during construction, which involves excavation, support and equipment for ore exploitation. Excavation involves the disruption of pre-existing stresses, loads and deformations that must be counteracted by the initial or permanent support.

Shaft sinking is a classic activity in underground mines, because in many of them this kind of access is developed up to a certain depth in which the orebody is known, the exploration being responsible for revealing the possible need or not to continue to deepen this access. Underground mines advance on average 50 – 60 m/year (Neto, 2010). Several mines had to take this kind of decision in recent years in Brazil, as mine of Cuiabá mine (AngloGold, Brazil, gold, cut and fill method) with expansion in progress, Baraúnas mine (Caraíba, Brazil, copper, sublevels with retreating method), Fazenda Brasileiro mine (Yamana, Brazil, gold, sublevels with retreat mining) and São Bento mine (São Bento Mining company, Brazil, gold, cut-and-fill in sublevel method, closed in 2007).

The factors that determine the degree of difficulty and underground excavation costs are mainly related to geology, geomechanics and hydrology. Small section shafts are important for given situations. Mining the narrow veins, whose power is usually less than 2 m and variable dip, requires variable procedures and appropriate infrastructure and equipment. Attempts to increase productivity and working conditions (mechanization) in narrow shaft excavation and drifts are carried out continuously in mining operations.

Proper choice of method can reduce the shaft excavation time and ensure uninterrupted operation. As the amount of development works varies from mine to mine, overall costs can reach different sums. Excavation costs for underground mine development are about US\$ 1.850.00/m for galleries and drifts in general, according to Cotica (2009), in the order of US\$ 1.000.00 to US\$ 1.200.00/m for galleries with 4.5 m x 4.7 m of cross-section and approximately US\$ 3.000.00/m for circular shaft with 3 m in diameter, with simultaneous lining and excavation, according to Lack (2005).

As the amount of development services varies from mine to mine, overall costs can reach different sums. For instance, the Morro mine (Tabiporã company, Brazil, gold, room and pillar method) had 30 km of excavations in 2005; on the other hand, the Morro Agudo mine (Votorantim/Nexa company, Brazil, zinc, room and pillar method) had rate 900 m/month (Hashimoto et al., 2014) in developed works; while the Ipueira mine (Ferbasa company, Brazil, chromite, sublevel caving) had rate 700 m/month. On the other hand, Diavik mine (Rio Tinto company, Canada, diamond, sublevel stoping and descending cut-and-fill method), in transition to underground mine, is building 20 km of excavations for the various purposes.

Recognizing the inherent variability of geological conditions, various shaft sinking methods have been developed so that excavation and support may vary according to the conditions found. The purpose and the geological-geotechnical constraints

define the method of excavation (Fujimura et al., 2001)

Underground mines usually include at least one excavation with greater depth range for temporary access or as part of the permanent structure. Shafts designed for permanent functions have typical useful diameters between 5.0 and 10.0 m. They serve for production, service, orebody assessment, mine development, manway or emergency exit (USA Army, 1997).

Depending on the mine depth, the shaft sinking can consume 60 % of development time. Because of this, the appropriate choice of a method to minimize shaft excavation time and ensure uninterrupted operation is of great importance.

In shafts with small cross-section or in mines with low mechanization indices it is common to use hand pneumatic drills and blasting by explosive gelatin in cartridges, employing natural draft or flexible ducts with axial fans for gases and fumes exhaustion, removal of blasted material (muck) by hand shoveling into hoistable dumping buckets.

System of this type has been studied here, consisting of a rectangular cross-section shaft (3.7 m x 2.0 m), with final depth of 94 m, excavated in order to obtain samples for a pilot-scale mineral processing testwork, before the open pit mine's startup.

The shaft had a concrete collar up to the first 20 m depth. As pointed out by Zhang and coworkers (Zhang et al., 2015), blasting in the vicinity of concrete mass may cause positive or negative effects on the liner resistance (at concrete's early ages, blast induced vibration have even increased the strength within the first 1 – 2 days). In present work, no monitoring of such effects in loco was done. After such a collar the shaft walls were supported by timber frames spaced 1.5 m by posts and fastened by hooked hanging bolts. The wall lining was completed using 25 mm thick wooden planks. The shaft has been excavated in schist rocks lithotypes belonging to the metamorphosed hydrothermal deposit of copper and gold located in Chapada (municipality of Mara Rosa, Goiás State — Brazil). The deposit is located about 8 km southwest of the town of Alto Horizonte.

The shaft was divided in three sections by timber dividers (buntons); the central compartment contained the electric cables and ventilation ducts and water piping; one of the side sections was for hoist system (lifting and descent of the muck bucket, equipment and personnel) showed in foreground of Figure 1–a. The third section of the shaft consisted of the stairs.

As timbering is concerned, hooked hanging bolts make the assembly of the shaft set easier and serve as structural reinforcement and fastening element for adjacent frames. There were eight hanging bolts (6-mm diameter threaded steel rods provided with hooks) for tensioning contiguous frames, according to the classical design advocated by Boky (1967) and Gardner and Johnson (1932).

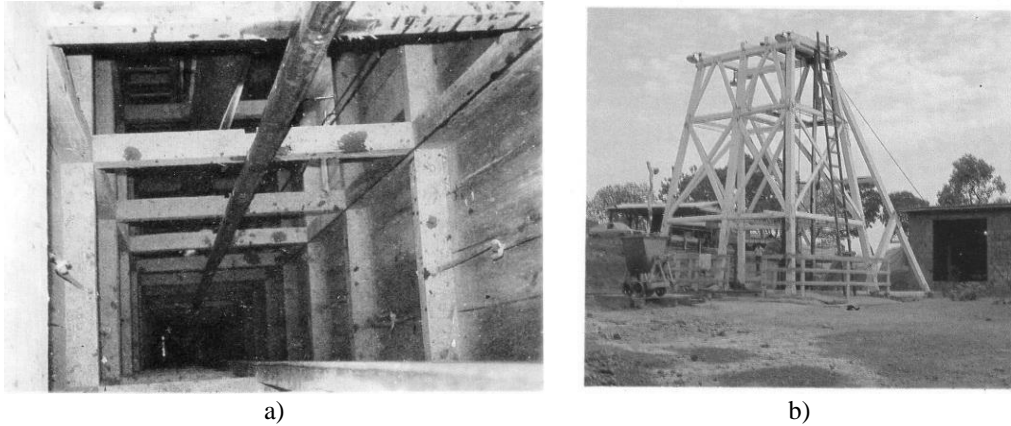
The shaft's interior space is displayed in Figure 1–a, with details of the reinforced timbering, as well as vertical guides for avoiding the swivel and pendulum movement of the bucket (impregnated with grease for lubrication). For its part Figure 1–b shows an overview of the extraction tower or shaft framework and the winch house. The tilting mucking bucket in standby (on its haulage cart's half bearings) is seen in the foreground.

Geologically speaking, Chapada deposit is closely related to metavulcanosedimentary sequences associated with hydrothermalism, resulting in massive sulphides with mineralization of copper, gold, iron and molybdenum.

This occurrence is in the so-called Mara Rosa magmatic arc, temporally and spatially related to the model of magmatic evolution of a collision belt (Oliveira et al., 2007). The average ore content is 0.44 % Cu and 0.35 g/t Au. Feldspathic biotite-schist with disseminations of chalcopyrite and pyrite is the main ore type crossed by the exploration shaft at Chapada mine (Oliveira et al., 2007).

According to Ramos Filho et al. (2003) there is locally "a sequence comprising feldspathic schist biotite, biotite-microcline gneiss (felsic metavulcanic), amphibolite and hydrothermal alteration products such as epidosite, pyrite-magnetite-quartz-sericite schist, gedrite-anthophyllite schist, kyanite schist and smaller proportions of staurolite-kyanite-ortho-amphibolite".

Figure 1 — Exploration shaft at Chapada mine: a) view of the tri-sectional shaft displaying the wooden frames fastened by hanging bolts and liners (dark spots are grease splashes from hoisting system); b) Shaft's sinking headframe (with one sheave), showing in foreground one empty dumping bucket.



Source: Authors' file.

The deposit is controlled by shear zones. Kuyumjian (1989) suggests that epidiosites and rocks rich in epidote would have resulted from the volcano-exhalative interaction of heated marine aqueous solutions and basaltic rocks. The ore zone coincides with the structural trend of the major isoclinal folding axes of north-east direction and westward flanks. This zone is 1.5 km long, 0.5 km wide and 80 m thick (Ramos Filho et al., 2003).

As geotechnical parameters are concerned, Cintra (2003), in an extensive work aiming at forecasting of copper and gold grades by neural network, has presented the Chapada deposit's histograms for the rock quality designation (**RQD**), which is the ratio between the sum of the lengths of the core pieces equal to or greater than 10 cm, and the total core run length. It is a classical indicator of rock quality in situ. **RQD** descriptive statistics for Chapada were: average: 93.98 %, median: 99.06 %; standard deviation: 12.39 %; coefficient of variation 13.2 %. Working further those data, it was possible here to get a good-to-fit theoretical distribution for the **RQD**, shown in the results section.

2. Methods

The indices belonging to the entire cycle of excavation operations were collected from production sheet records over a month (from February 2 to February 28). Namely the unit operations are: drilling, charging, and blasting, blast gas exhaustion (ventilation), wall scaling and trimming, and placement of the support system (timbering). Operation stops were also quantified. Statistical analysis of productivity indices allowed the detection of critical points of the operation and the benchmark statement for similar mining operations, with low level of mechanization.

After debugging the raw database, a descriptive statistical analysis was performed and classical nonlinear regression techniques were applied in order to test the goodness-to-fit of some theoretical statistical distributions, like Gaussian, Hill's, Harris' and Weibull's distributions, according methodological procedures preconized by Lopes et al. (2020) and Lopes et al. (2022).

2.1 Studied parameters

In line with the production cycles of shafts excavation in competent rocks (such as those explained by classics such as

Gardner e Johnson, 1932; Sanada et al., 2015; and Dowis, 1972), the following variables were selected for this study, since they are part of a normal operating cycle when sinking a small cross-section shaft, employing low coefficient of automation.

Preparation for drilling

This step encompasses prior to drilling operations, such as assembly of compressed air pipelines and other utilities, marking up of the face before drilling using usually spray ink, and lowering of pneumatic rock drills and jackhammers from the surface.

Drilling

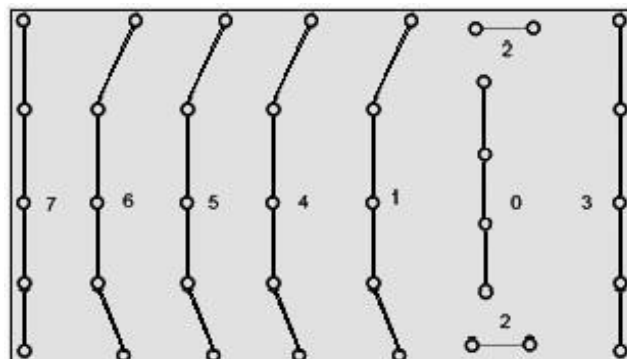
Operation performed by means of two pneumatic rock drills (Atlas Copco HR 658 model with consumption of 0.058 m³/s at 600 kPa; impact frequency of 34 Hz) operated manually totaling between 38 and 42 blast holes employing rock drills with integral hexagon shanks 1.2 m in length and 27 mm in diameter. The drill bit type was conventional cross bit type. In line lubricators type BLG 30 from Atlas Copco were used. Jackhammers (MQB series from Brobras), with air specific consumption of 0.032 m³/s at 600 kPa, and impact frequency of 20 Hz were used for face trimming and pneumatic scaling of side walls.

Charging and blasting

This operation was started with introduction of a booster (primer) consisting of a gelamonite 75 % cartridge (0.156 kg per cartridge), provided with electric cap, at the bottom of each of the holes. After that they are filled with water resistant explosive gelatin (Bragel 60 %, with 0.184 kg per cartridge and resistant to water for at least 32 h but no more than 72 h). Naturally, the filling up the blast holes and explosive charge compaction inside them were done with help of wooden rods, to avoid sparks.

The blasting was started electrically and detonating cord and delays multiples of 25 milliseconds were employed. The fire lines are shown in Figure 2. As safety is concerned, the electrical self-potential induced by redox reactions of the sulphides present in the ore was not enough to trigger the spontaneous detonation of the electric caps.

Figure 2 - Typical drilling and initiation pattern employed (rows are numbered according to delay sequence).



Source: Authors' own elaboration.

Fan cut type was adopted for the drilling pattern instead of the bench or stope type, which is more suitable for large cross-section shafts. The numbers of the hole sets refer to the numbering of electrical blasting caps (indicating time lapses of 25 ms). Therefore, the caps were instantaneously ignited at row 0 and the caps had a 175 ms delay at row 7.

The holes in row 0 had the minimum slope (in relation to the shaft axis), whereas lines 1, 4, 5 and 6 had gradually

increased slopes. The lines 2, 3 and 7, in turn, were drilled vertically (since they were finishing holes).

As staff sizing is concerned, underground team consisted of one foreman, one jackhammer (drill runner), one jackhammer helper (drill helper) and one handyman (shoveler).

Front ventilation

Ventilation of the work front was made using vinyl canvas conduits (from Sansuy or similar) coupled to axial exhauster-based system. The interconnection of the contiguous segments was made with use of metal rings (clamps).

Preparation for mucking

This kind of operations are the last activities before the mucking operation (and after a blasting cycle) and they were denoted preparation for mucking (or pre-mucking) and included the inspection following a blast, detection of any misfired holes (with unexploded explosive devices), wall trimming and scaling operation (if necessary and done manually), and investigation of timbering integrity (which can be damaged by fly-rocks or air blast).

Mucking

This operation was basically the removal of the blasted rock (by hand shoveling) and it was performed by all working underground team, with surface aid from hoist team. The mucking unities were steel buckets with U-shaped bottom. After each bucket filling, the bucket was hoisted by steel cable attached to the winch. Upon reaching the surface, the bucket was disengaged from the winch hook after fitting of its pivot axis in the half bearings of the cart, which was then hauled for short distance. The haulage cart was tilted in a dumper truck on wheels that carried the blasted material to the stockyard. The bucket was then returned by the cart to the shaft's mouth, and, after being re-engaged to the winch hook cable, it was descended to the underground work front, closing the cycle of this activity. Naturally they used up two buckets to save cycle time, since the descent of the new empty bucket was made immediately after the disengagement of the loaded bucket at surface. The average time of engagement maneuver was 60 seconds.

Inspection visits

Inspections were operations of non-systematic character, consisting of geological inspection of rock walls and measuring face advance with fiberglass tape.

Erection and fastening of timbering

The frequency of this operation was half the frequency of whole sinking cycle. Therefore, ideally, the wooden sets and side planks were assembled and fastened for lining after two cycles of shaft's face advancement, keeping lagging of 15 m. The squared wooden beams used to make the sets (frames) had a cross-section of 0.2 m by 0.15 m.

The more recent three-compartment sets or frames were tightened with hanging bolts and nuts (8 carbon steel bolts 15.9 mm diameter rods, per frame) and fastened to the already assembled lining. The timber bearers (bear sets) were regular sets with end plates extended and wedged 0.5 m in both sides of the shaft rocky walls. They were sized to carry the full weight of the next two wooden sets. In sequence and at the right time, those two more regular sets were mounted and secured to the predecessor frame via hanging bolts.

Timbering was kept at a distance of 15 m from the bottom face of shaft, in order to avoid damages due to collisions by rock fragments (fly-rocks) during the blasting stage, or by blast-induced shockwaves. For estimate this distance implied in calculation of the scaled distance (SD), which is given by the following equation.

$$SD = \frac{\text{distance from explosion point, m}}{(TNT \text{ equivalent mass per shot, kg})^q} = \frac{d}{(w_{eq})^q} \quad (1)$$

With respect to the value of the parameter q , if we are considering a spherical detonation wave surface (from a point load), we have $q = 1/3$; this is the Hopkinson–Cranz scaling law, according to Bajić et al. (2016). On the other hand, considering a cylindrical wave front, $q = 1/2$ (square root scaling law) (Schneider, 2002; Kumar et al., 2016). As pointed out by Kumar et al. (2016) the peak particle velocity (v_{peak}) is a good index of damage associated with vibration. They collected many empirical data and proposed a general equation (Equation 2) for the particle peak velocity, for several kinds of rocks, forecasting peak particle velocity from rock strength and scaled distance. The regression analysis resulted exponent $k_1 = 0.642$ and $k_2 = 1.463$.

$$v_{peak} = \frac{f_c^{k_1}}{\gamma \times SD^{k_2}} = \frac{f_c^{k_1}}{\gamma \times \left[\frac{d}{\sqrt{w_{eq}}} \right]^{k_2}} \quad (2)$$

Where: f_c — uniaxial compressive strength (UCS) of rock [Pa]; γ — unit weight [N/m³].

Working in the same direction, in the present study the data referring only to schists were pinched from Kumar et al. (2016) in order to produce a particular equation for this type of rock. Kumar et al. (2016) report the unit weight value of 26.000 N/m³ for the schists studied. In turn, the uniaxial strength average for such schists was adopted, resulting: $f_c = 58.33$ MPa. As a result, taking the average attributes of the experiments reported with schists, the following values were obtained for schists: $k_1 = 0.613$ and $k_2 = 1.443$.

As a confirmatory remark with respect to the safe distance of facilities from blast face, Sanada et al. (2015) have reported a ventilation shaft sinking at Mizunami Underground Research Laboratory (Japan) and have pointed out that blasting was only carried out after the scaffold has been raised from 20 to 30 m above the shaft face in order to avoid damage by fly-rocks.

Lunch break

Whenever possible, this part of the operational cycle was to be superimposed over the period of inactivity due to blast gas exhaustion from the work front. This procedure had an impact on the nominal measurement of lunch stop times and the front ventilation times, since there was a partial overlap between these two activities.

Coffee break

This step had the purpose of energy recovery. Sometimes, interval between these stops was swelled or contracted, when it could take advantage of overlapping on downtime required (as for detonation front ventilation or corrective maintenance of the winch or pumps).

Drainage ditch excavation

This step had a sporadic character, aiming to improve the pumping of water accumulated in the work front, which was exhausted by sequence of pneumatic pumps in series.

Ground water pumping

The water upwelling was high and the drainage was done by submersible centrifugal pumps, pneumatic type Brobras/Zaba, with a manometric head of 30 m, discharge diameter of 63 mm, compressed air supply diameter of 25 mm and maximum air consumption of 30 liters per second (0.03 m³/s). Reservoirs in the shaft side walls were excavated every 20 m,

constituting intermediate pumping stations. The front-end pumps had an 18-hour day regime, while the intermediate-station pumps were operated 8 hours a day. To meet this regime, the side-wall reservoirs, for the temporary storage of water, had volume of 3 m x 2 m x 2 m, connected to the shaft by a small drift.

Maintenance in the hoisting winch

In order to improve productivity, whenever possible, the hoist system maintenance was superimposed over scheduled shutdowns (such as stops for blast gas exhaustion or lunch).

2.2 Optimized production criteria

Adopting the use of several simultaneous mucking buckets, the conditions for uninterrupted production for the deepening of a vertical shaft are systematized as follows. Equation 3 gives the effective volume required for each bucket:

$$V_b = \left(\frac{2 \times h}{v} + \tau_{man} \right) \times \frac{Q_v}{n} \quad (3)$$

Where: V_b — bucket volume [m³]; h — shaft depth [m]; v — lifting speed [m/s]; τ_{man} — maneuver operation time (bucket engagement and disengagement) [s]; Q_v — required volume flow rate (during the production cycle) [m³/s]; n — number of buckets [-].

The disengaging/release maneuver includes the transfer of the bucket to the tilting cart or its tipping over the conveying equipment that should take the blasted material to the unloading point. Of course, the total number of buckets extracted per blasting cycle is the ratio of the total muck mass and the useful mass capacity of the hoisting system, that is to say:

$$N_b = \frac{k_{swell} \times \mu \times \eta \times \rho_s \times S \times z}{(C_{win} - q \times h) \times \phi - m_b} \quad (4)$$

Where: N_b — number of buckets [-]; k_{swell} — swelling coefficient of blasted material [-]; μ — coefficient of overbreak [-]; η — coefficient of utilization of blast holes [-]; ρ_s — in situ average density of rocks (solid) [kg/m³]; S — project area of the shaft cross section [m²]; z — depth of the blast holes [m]; C_{win} — nominal mass capacity of the winch [kg]; q — linear density of the hoist rope [kg/m]; h — shaft depth [m]; ϕ — Hoisting winch safety coefficient ($\phi < 1.0$) [-]; m_b — mass of the empty bucket [kg].

Note that Equation 3 multiplied by the bucket filling factor and the apparent density of the bulk material can replace the denominator of Equation 4, thereby allowing the interconnecting of the production parameters. Buckets of nominal volume equal to 0.2 m³ were used in the excavation of the shaft object of this study. The winch employed was a Hercules (Model W-20) with nominal power of 14.9 kW (20 hp). The steel winch cable with 16 mm diameter (5/8 ") was type 6 x 19 (6 strands with 19 wires per strand). The linear density of the cable used was approximately 1.05 kg/m. The cable cruising hoist speed was about 1.0 m/s.

In turn, the distance from the shaft to the discharge point (in meters), as a function of surface haulage speed, must meet the following inequality:

$$d_{haul} \leq \left(\frac{\tau_{load} + \tau_{unload}}{2} \right) \times v_{sur} \quad (5)$$

Where: d_{haul} — distance from the shaft axis to the dumping point [m]; τ_{load} — loading time of the bulk material [s]; τ_{unload} — unloading time of the bulk material [s]; v_{sur} — haulage velocity on the surface [m/s].

That is, the speed of the conveyance equipment and the haulage distance must be compatible with the extraction time of bucket cycle.

3. Results and Discussion

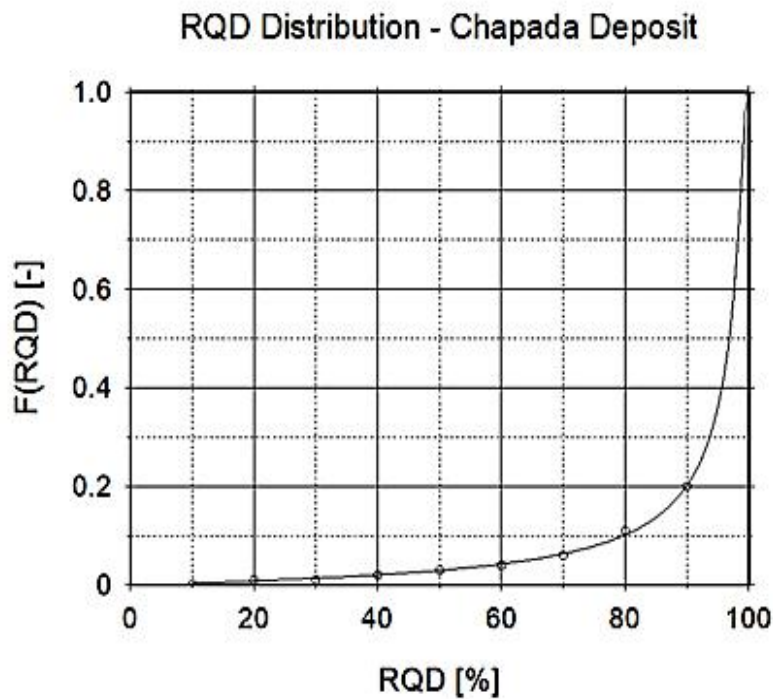
3.1 Rock quality designation distribution (RQD)

The handling of the RQD probability density histogram, previously empirically determined by Cintra (2003), has allowed the present authors to establish the cumulative distribution of this parameter for the Chapada deposit, which can be described by a truncated Weibull–Rosin–Rammler equation given by the equation:

$$F(RQD) = 1 - \exp \left[- \left(\frac{\frac{RQD}{100 - RQD}}{\frac{k^*}{100 - k^*}} \right)^p \right] \quad (6)$$

The regression parameters of Equation 6 were: $k^* = 97.85 \%$; $p = 0.918$, with determination coefficient of $R^2 = 0.9989$. Figure 3 displays the adherence between theoretical curve and data.

Figure 3 — Distribution for rock quality designation (RQD) of Chapada deposit after Cintra’s experimental data (Sanada et al., 2015).



Source: Authors’ own elaboration.

3.2 Productivity of excavation method

The excavation productivity was 0.25 t per man per hour, comparable to the open stopping mining (manual or semi-mechanized underground mining method); and lower than the mean productivity of shrinkage stopping mining (with manual drilling, extraction system already falling into disuse). This is consistent with a mine development stage and with the number of people involved. Some of the performance indices quantified for the work in the period studied can be seen in Table 1, below. The swell (bulking) factor is 1.82.

Table 1 — Performance indices for the shaft sinking.

Parameters	Values Obtained
Average yield	67.21 buckets/ m of advancement
Bulk volume rate:	13.44 m ³ /m of advancement
Volume in situ by theoretical advance:	7.4 m ³ / m of advancement
Unit mucking time cycle:	9.3 minutes per bucket
Average drilling velocity:	0.247 m/min

Source: Authors' own elaboration.

The average unit consumption of gelamonite and Bragel, and other productivity rates can be appreciated in Table 2, below.

Table 2 — Unit consumption and explosive charging ratio.

Obtained Parameters	Gelamonite 75 %	Bragel 60 %
Cartridges/m of advancement	34.6	120.6
Cartridges/m ³ (in situ)	4.68	16.29
kg/m ³ (of bulk material)	0.402	1.650
kg/m ³ (in situ)	0.730	2.998

Source: Authors' own elaboration.

Using the theoretical curve in the figure of loading or powder factor (*Pf*) against heading area (*S*) (Schneider, 2002), for several rock types and drill of diameter of 31.7 mm, a non-linear regression was performed. The resulting equation (with coefficient of determination: $R^2 = 92.20\%$, and for heading area greater than 3 m²) is given (in SI unities) by:

$$Pf = 12 \times \exp[-0.8028 \times (S - 3)^{0.5486}] + 1.57 \quad (7)$$

As can be seen, the lower the face area the greater the loading factor. It is due to more constricted rock movement in more confined spaces. From the equation, the expected value for the present case ($S = 7.4$ m²) would be 3.53 kg/m³ (for a hypothetical drill diameter of 32 mm). The actual value was 3.73 kg/m³, fully consistent value, since drills with a diameter of 27 mm in fact were used.

From these blasting value and adopting 15 m as minimum distance between timbering and the shaft bottom, the minimum scaled distance can be calculated as presented in Table 3.

Table 3 — Scaled distance calculation to set timbering lagging.

Explosive	Nominal TNT Content (“grade”)	Cartridge mass	Cartridge number	Mass per shot [kg]	Equivalent TNT mass per shot
✓ <i>Primer (Gelamonite):</i>	75 %	0.156 kg	34.6	5.3976	4.05 kg
✓ <i>Charge (Bragel):</i>	60 %	0.184 kg	120.6	22.1904	13.31 kg
Total TNT equivalent mass:					17.36 kg
Minimum distance from blast face:					15.0 m
Minimums scaled distance for the timbering at full blasting charge without delays:					
✓ <i>with parameter $q = 1/3$ (cubic root):</i>					5.79 m/kg ^{1/3}
✓ <i>with parameter $q = 1/2$ (square root):</i>					3.30 m/kg ^{1/2}
Partial charge per delay, with parameter $q = 1/3$ (cubic root):					11.08 m/kg ^{1/3}
Partial charge per delay, with parameter $q = 1/2$ (square root):					9.52 m/kg ^{1/2}

Source: Authors’ own elaboration.

Although the equation based on the schist data collected (Kumar et al., 2016) employs the scaled distance with $q = 1/2$ (cylindrical shock wave), if we apply it to the case under study, but taking the Hopkinson–Cranz scaling law (spherical shock wave), a peak particle velocity of 0.0693 m/s would be obtained instead.

The following table (Table 4) gives a summary of the distribution of time effectively worked in sinking a shaft with rectangular section of 3.7 m by 2.0 m, with walls supported by four bolted wood frames and lined with 25 mm thick wood planks. The total sinking of the shaft was 20.8 m during the study period (28 days, with four off Sundays). That is, on the first of February, the shaft had depth of 57.2 m to end on the 28th of the same month, 78.0 m.

Table 4 — Average time distribution for shaft sinking activities.

Activity	Total Time [min]	Percentage [%]
Mucking	13.064	43.29
Preparation for mucking (pre-mucking)	2.177	7.21
Inspection visits	160	0.53
Timbering assembly	4.178	13.84
Preparation for drilling (pre-drilling)	665	2.20
Drilling	3.524	11.68
Charging and blasting	2.018	7.43
Stop for lunch	820	2.72
Ventilation (gas exhausting)	986	3.27
Coffee/snack break	1.080	3.58
Excavation of drainage ditch	300	0.99
Water exhausting	716	2.37
Maintenance on the winch	270	0.89
Total working	30.180	100.00

Source: Authors’ own elaboration.

Adding to the preparation stage and the productive operations themselves, we have 85.65 % of the time spent. The

maintenance, inspection and auxiliary operations (ventilation and drainage) added result 8.05 %, while the stops (lunch and coffee) add up to 6.30 %.

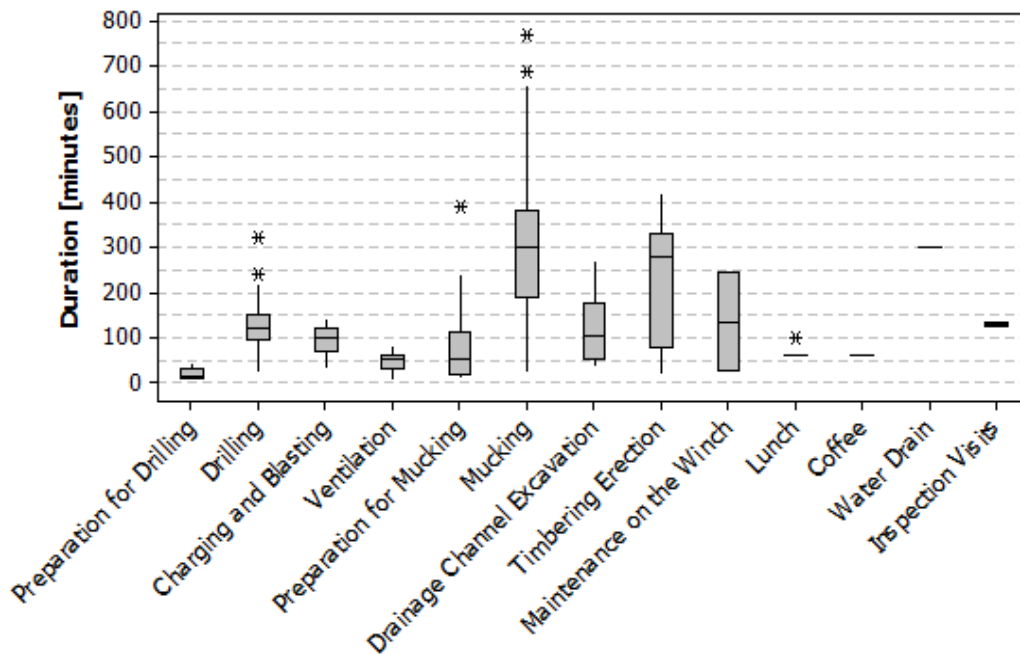
As for the chemical insalubrity aspect of the work environment, three cases are noteworthy. First, the handling of nitroglycerin-based explosives, which has a strong vasodilator character, tended to induce headache in new workers less accustomed to handling explosives. Another unhealthy aspect was the one caused by incomplete exhaustion of the gases generated in the front detonation. Part of this last unhealthy condition is also due to the slow release of the gases produced by the explosion that percolated to the rocky massif through the schistosity of the rock. This problem was analyzed in detail and with an emphasis on carbon monoxide (CO) by Harris and Mainiero (2008).

Thirdly, one should mention that due to its abundance of water in the sulfide-bearing mineralized layers, the water acidity (resulting from sulfide oxidation) has provoked limb ulceration (sometimes severe lesions), especially in legs and feet of underground staff, despite the use of waterproof raincoats and long-legged rubber boots. It was virtually impossible to remain dry in the underground environment, under torrential action of acidic water percolating from the shaft walls.

3.3 Statistical analysis Obtained Parameters

The analyzed data indicate that there is a wide variation for some categories of data, while others have very low variation (Figure 4). The breakfast and lunch classes are almost deterministic in nature and presented always at the same time and so were not statistically analyzed. Winch maintenance operation, and basic depletion had no sufficient amount of data to be analyzed statistically. In sequence, Table 5 summarizes the descriptive statistics for duration of unit operations in shaft sinking. Timbering operation had a significant amount of data, but it is clearly bimodal, as can be seen below (Figure 5).

Figure 4 — Boxplot of the parameters studied.



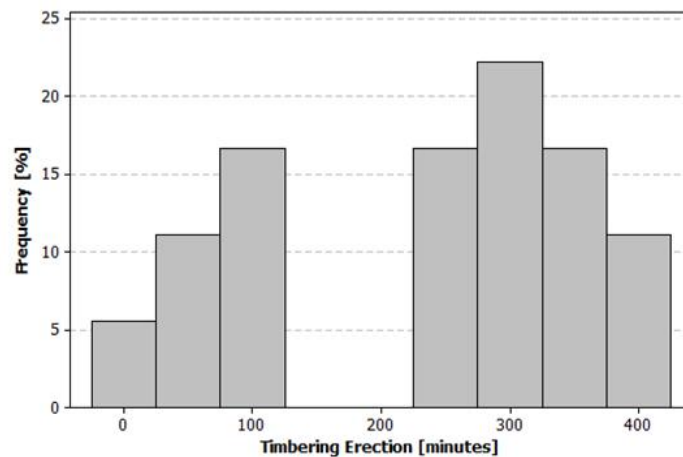
Source: Authors' own elaboration.

Table 5 — Experimental descriptive statistics for shaft sinking unit operations.

	Preparation for Drilling	Drilling	Charging and Blasting	Ventilation	Preparation for Mucking	Mucking	Water Drain	Timbering Assembly
Minimum [min]	6.0	20.0	30.0	4.0	10.0	20.0	35.0	19.0
1° Quartile [min]	10.0	95.0	69.5	31.3	20.0	190.0	63.8	87.5
Median [min]	15.0	122.0	98.0	50.0	50.0	300.0	105.0	277.5
3° Quartile [min]	30.0	147.5	119.3	60.0	110.0	380.0	139.5	317.5
Maximum [min]	45.0	320.0	140.0	81.0	390.0	770.0	270.0	420.0
Average [min]	19.3	130.5	93.3	44.8	75.1	318.6	119.3	232.1
Standard deviation [min]	11.4	62.8	28.5	19.6	82.9	177.0	84.3	131.6
Coefficient of variation	59.1 %	48.1 %	30.5 %	43.7 %	110.4 %	55.6 %	70.6 %	56.7 %
Number of occurrences [-]	21	27	24	22	29	41	6	18

Source: Authors' own elaboration.

Figure 5 - Histogram for timbering operation.



Source: Authors' own elaboration.

Regarding Figure 5, the most time-consuming operation consisted essentially in the introduction of bearing sets (bearers) in order to furnish in-rock anchorage to the structure, while the shorter class of operation consisted of the fastening of timber frames with hanging bolts. It was expected that operations of shorter duration would be more frequent, while the more time-consuming operations would occur with longer time intervals.

For the sake of clarity, the data were normalized by dividing for the maximum value of the variable. Then, the normality of the data collected was investigated by the Anderson-Darling normality test (Table 6). When the p-value is greater than 0.05 (the adopted level of significance), the data distribution could be considered normal (the null hypothesis cannot be discharged). Nevertheless, the statistical analysis of the data was also tested for other theoretical distributions usually found in the literature. Nonlinear regression analysis of the experimental data showed that even if the null hypothesis cannot be rejected, statistical adherence was higher for some of these distributions, as will be seen in Table 6.

Table 6 - Anderson–Darling test for normality.

Variable	P-value
Pre-drilling	0.828
Drilling	0.580
Charging and Blasting	0.681
Ventilation	0.799
Pre-Mucking	0.717
Mucking	0.112

Source: Authors' own elaboration.

The kurtosis of a distribution defines the flattening of the curve in relation to the normal distribution. Negative kurtosis indicates that the distribution curve is flatter and the data are more dispersed and varied. For its part, positive kurtosis indicates more concentrated distributions and therefore with higher peaks. On the other hand, the distortion is a measure the degree of asymmetry of a distribution. The distortion can be positive or negative if the change is to the right or left respectively. A zero-distortion distribution is perfectly symmetrical with respect to the median, like, as instance, the Gauss or normal distribution. Table 7 shows these parameters concerning the here studied unit operations.

Table 7 — Shape parameters of the data distributions.

Description	Pre-Drilling	Drilling	Charging and Blasting	Ventilation	Pre-Mucking	Mucking
Kurtosis	- 0.60	2.31	- 0.73	- 0.37	6.90	0.24
Distortion	0.68	1.03	- 0.32	- 0.39	2.36	0.58

Source: Authors' own elaboration.

The cumulative probabilistic distributions studied were those from Hill, Gaudin–Meloy, Gates–Gaudin–Schumann, Harris, and Weibull–Rosin–Rammler. The Hill distribution, Harris distribution and Weibull distribution had high goodness-to-fit for the data and for that reason the equations are presented in Table 8. Pre-drilling proved to be better described by the Weibull distribution, while drilling and pre-mucking operations fit more closely to the Hill distribution (Table 9). The Harris distribution results in better adherence to loading and detonation operations, ventilation, and mucking operations.

Table 8 — Used theoretical distributions.

Distribution	Equation
Right truncated Hill	$Y = \frac{\left(\frac{x}{x_{max} - x}\right)^a}{\left(\frac{x}{x_{max} - x}\right)^a - \left(\frac{x_{50}}{x_{max} - x}\right)^a}$
Hill	$Y = \frac{1}{1 + \left(\frac{x}{x_{50}}\right)^{-\lambda}}$
Harris	$Y = 1 - \left[1 - \left(\frac{X}{X_{max}}\right)^a\right]^b$
Weibull	$Y = 1 - \exp\left[-\left(\frac{X - \delta}{X_*}\right)^n\right]$

Source: Authors' own elaboration.

Fitted distributions for pre-drilling, drilling and pre-mucking are shown in Figure 6, whereas Figure 7 corresponds to the analogous cumulative distributions of mucking, charging followed by blasting, and ventilation.

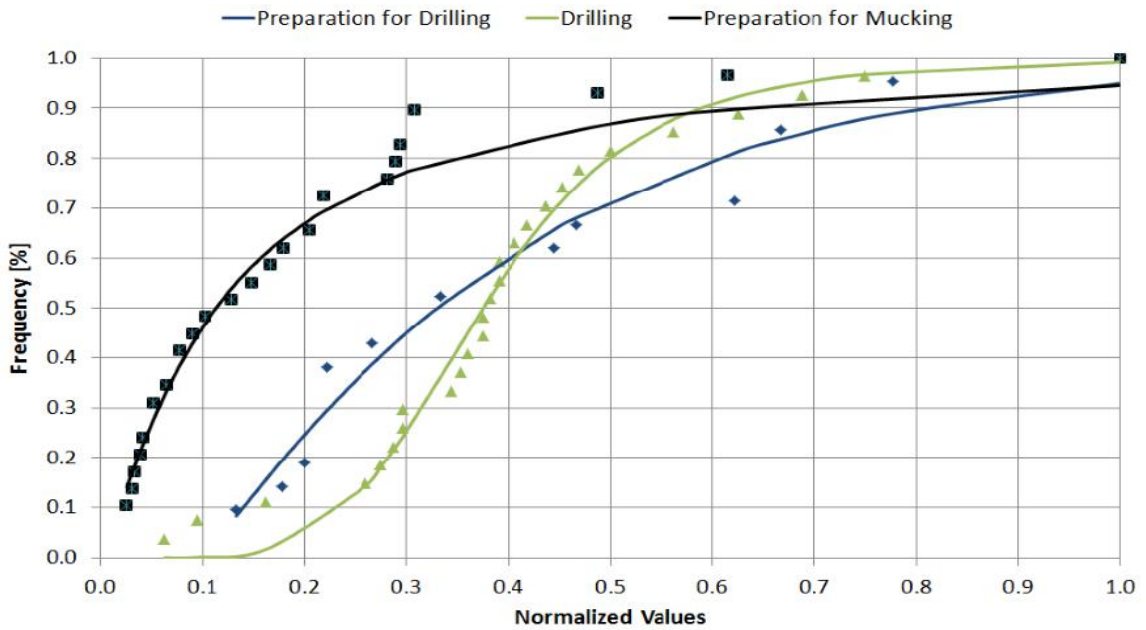
In order to allow comparison on the same scale, the data in Figure 6 and Figure 7 were normalized by dividing the absolute values by the maximum value measured for each of the stochastic variables under study.

Table 9 — Regression parameters for used distributions.

Variable	Distribution	parameters	Correlation (R)
Pre-drilling	Weibull (with $\delta = 0$)	$x^* = 0.32$ $\sigma = 0.10$ $n = 1.07$	98.4 %
Drilling	Hill	$x_{50} = 0.38$ $\lambda = 4.87$	99.3 %
Charging and Blasting	Harris	$x_{max} = 1.00$ $a = 2.42$ $b = 1.43$	99.1 %
Ventilation	Harris	$x_{max} = 1.00$ $a = 1.69$ $b = 1.53$	98.3 %
Pre-mucking	Hill (truncated)	$x_{max} = 1.00$ $a = 1.09$ $x_{50} = 0.12$	99.3 %
Mucking	Harris	$x_{max} = 1.00$ $a = 1.77$ $b = 3.52$	99.3 %

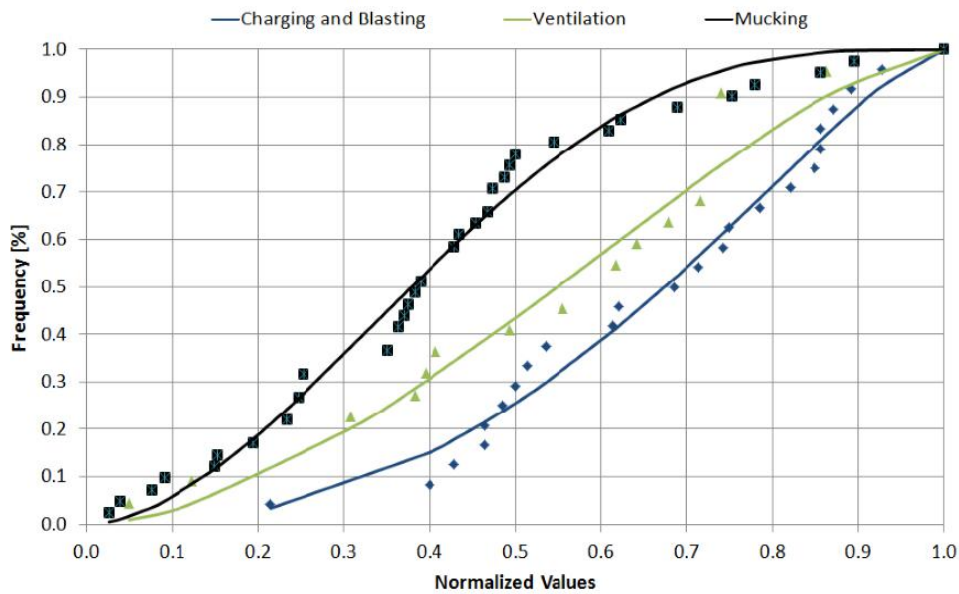
Source: Authors' own elaboration.

Figure 6 — Fitted distribution for operations: pre-drilling, drilling and pre-mucking.



Source: Authors' own elaboration.

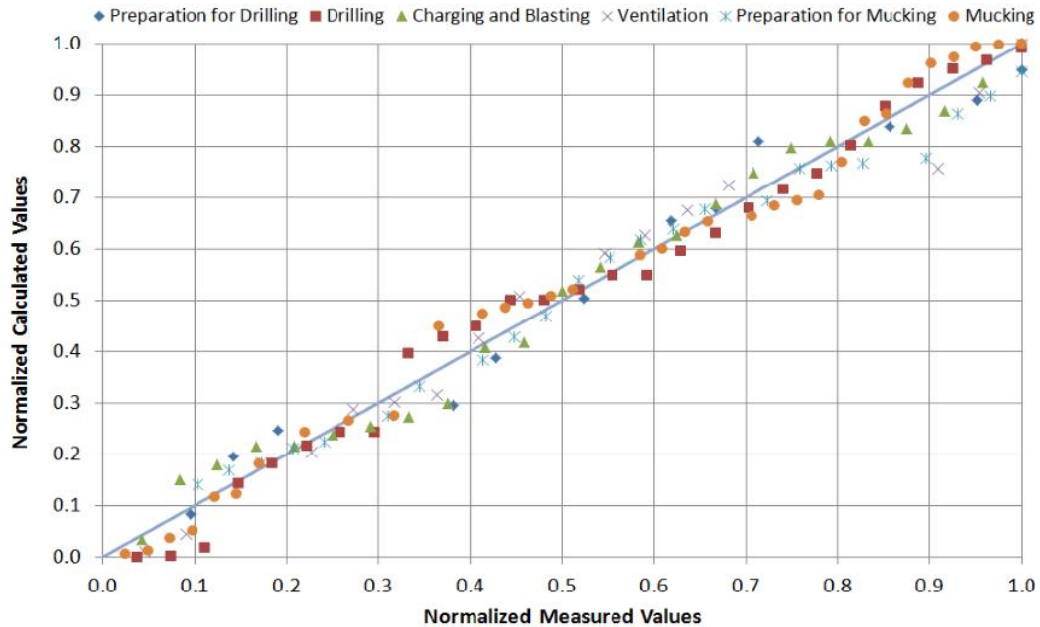
Figure 7 - Fitted distribution for operations: mucking, charging and blasting and ventilation.



Source: Authors' own elaboration.

It is possible to observe a good fit between measured data and calculated ones in accordance with each distribution. The Figure 8 shows the relationship between the values and also displays the ideal correlation represented by the solid line $x = y$ (with slope of 45°).

Figure 8 - Correlation between measured and calculated values.



Source: Authors' own elaboration.

Mancala (2017) shows a kind of "raise and fill" excavation by using mechanized vertical miner — MVM, that drills equidistant raises along the stope, with the muck material from the holes collected in a bucket and removed with aid of an adapted loader. After execution of the first sequence of holes, these are filled with cementitious material. After drying, filling provides resistance to rocky massif, enabling additional holes to be drilled in the space between the initial set accordingly. Shaterpour-Mamaghani and Bilgin (2016) have described a similar raise boring system in a ventilation shaft excavation at Kure Copper Mine, Turkey. They concluded that specific energy required is similar to that for tunnel boring machines (TBM), what allows penetration rates forecasting for raise boring machines from huge data of TBMs nowadays available in the literature.

The continuous mining of narrow veins is under study to make possible new projects, such as the Conrad Silver Project (southeast of Australia), according to Souza (2017). This company promises to execute mining of veins with minimum thickness of 0.8 m, with little or no dilution and with advancement rate of up to 4.0 m/h. If implementation is successful, this method can revolutionize the way of narrow bodies are exploited, shafts included.

As future improvements are concerned, some opportunities can be raised: testing of new forms of shaft sinking and supporting with increased mechanization and automation level; adoption of some kind of walking machine or even use of exoskeletons to increase human weight-bearing capacity such as those already in the market for handling heavy loads (Liu et al., 2015) (Liu et al., 2015; Huysamena et al., 2018).

One observes the increasing use of exoskeletons to improve the productivity of manual laborers, whether active (actively generating operative torque by actuators) or passive (spring-based) exoskeletons (Kumar et al., 2017; Linnera et al., 2018; de Vries & de Looze, 2019). Shoveling and filling buckets with blasted material in these confined spaces is a strong component for the adoption of this kind of equipment. Indeed, there has been a growing use of wearable robotic techniques in industry operations in recent years (Hiserman, 2020).

Also promising is the use of electronic sensors and electrical equipment, autonomous or remotely controlled systems (like unmanned aerial vehicles) for cavities inspection (especially in the presence of blast fumes) for detection of misfired holes and timbering damage. Shahmoradi et al. (2020) highlight the various aspects that the use of a drone can significantly improve

the safety of mine operation. Not only the intrinsic productivity gain of drone use should be considered. The natural possibility of inspecting the service front long before the complete dissipation of the explosion gases from rock blasting naturally entails a valuable shortening in of the time span between blasting step and the resumption of the operational cycle. Also, the use of expansive mortar (a non-explosive cracking agent, like the one employed by da Luz, Montenegro-Balarezo and Pereira (2003), hydraulic or thermal jet drilling, among other, are other promising possibilities for excavation and monitoring work in restricted/confined spaces, as in shaft sinking, in order to increase efficiency.

4. Conclusion and Final Considerations

Shaft sinking in competent rock is an operation that actually appears deceptively simple. Most tunnels and shafts in rock are provided with final lining, which can be: shotcrete, reinforced concrete with steel fibers, concrete rings, or mortar and concrete and also corrugated steel culvert pipes. Shafts are usually cylindrical. Rectangular cross-section shafts lined with timber are generally suitable for temporary extraction operations, under normal conditions rock. Rectangular shafts are usually longitudinally divided into compartments. Common dimensions ranging from 1.2 m x 1.8 m to 2.1 m x 3.7 m, with one, two or three compartments.

It should be pointed out that the high explosive loading ratio in the present case, compared to the conventional underground mining operation, actually results from the excavation of a mining work under confinement conditions, unlike the blasting of an entire exploitation panel, for example.

Demonstrating the importance of small section shafts, one can cite Visser (2009), which supports the replacement of traditional excavation practice of single ventilation shaft of large diameter for the excavation of three shafts of smaller diameter, distant one another of 28 m. Silva and Alcântara (2008) showed new way of sinking mine ventilation shafts at Ipueira, a chromite mine employing sublevel caving exploitation method located at Andorinhas, in Bahia, Brazil.

Various attempts to increase productivity and working conditions in shaft excavations are reported. Faurie (2010) proposes new methodology of work, with critical examination of the worker's location and operation method in workplace with replacement of less intelligent forms of execution. The referred author emphasizes that the reduction of risk exposure will naturally lead the employee to improve work and greater engagement in their tasks.

Looking prospectively into the near future, it is convenient to remember that the rapid advent of new technological resources — coupled with the cheapening of instrumentation, automation, and robotics solutions — currently allows us significant gains in productivity. Working in confined and unhealthy spaces, such as a shaft sinking operation, with a small section area, leads to fatigue due to the difficulty of locomotion, of movement development, and renewal of the oxygen content in the local atmosphere. In this line, one solution arises is the increasing use of exoskeletons to improve the productivity of manual laborers. Another tool of great potential regarding the reduction of risks in the underground work environment is the use of small drones for inspection and topographical survey of the work front, linked to each work cycle.

At least, depending on the flexibility of the labor laws at the excavation site, just in case of remote and demographically isolated areas, in need of task forces to carry out the shaft sinking work, an interesting aspect is to consider the increase in hours per shift, with the corresponding increase of overtime pay. However, the natural fall in the team productivity, under continuous work of long duration, is counter to this measure — and careful dosing must be taken into account.

Considering the temporary aspect of this type of work, this increase of hours per shift has some advantageous components, namely: greater attractiveness for the labor force (due to higher monthly earnings), lower indirect costs of workers' accommodation and feeding and — perhaps most importantly — the avoidance of potential conflicts among people living in confined environment in isolated sites, which tends to lead to the occurrence of disagreements, in particular because it is difficult to control drug abuse, such as alcohol, even under strict prohibition against its use in the camp.

As a matter of fact, the natural fatigue and less idle time resulting from work shifts with more hours also acts to avoid interpersonal conflicts during rest periods within the camp. However, care should be taken to ensure that there is no increase in the rate of accidents during longer work shifts. In any case, the consent of the team and their commitment to the agreed rules is the fundamental condition for the success of this strategy. Obviously, workers' rights, rules and safety regulations must also be strictly obeyed.

As a final remark, one must keep in mind that, no matter how classic an industrial operation is (even if it is part of those carried out under a low mechanization and automation coefficient), there is always an open field for intelligent innovation that does not demand such a magnitude of material and economic resources that it becomes impractical. And creativity is a powerful tool in that direction. The advent of low-cost computing and electronic monitoring and control systems, such as control platforms like the very affordable Arduino or Raspberry Pi, drones and ergonomic improvement devices (such as exoskeletons) — just to name a few — brought industrial productivity improvement options to an unimaginable level just a few decades ago. Even smartphones applications, such as vibrometers, decibel meters, and image and color analyzers are now freely available, which could become tools for increasing knowledge and productivity of small miners. However, the productivity gain resulting from the adoption of these new techniques was not studied in this article, due to its limited scope, and may probably be the subject of a future publication.

Undoubtedly, these are very promising topics for future research in mining, a sector generally associated (in the collective imaginary) with unsophisticated operational techniques.

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