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MINING IN A WORLD OF INNOVATION

October 18-21, 2016 • Rio de Janeiro /RJ • Brazil

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Created in 2009 with the purpose of stimulating innovation, competitiveness and sustainability in Vale's operations, Vale Institute of Technology (ITV) is a non-profit organization that develops research activities, with emphasis on H2 and H3 horizons as well as training of resources human at post-graduate level. ITV has two locations: one in Belém in the State of Pará, focused on matters related to Sustainable Development, and the other in Ouro Preto, MG, dedicated to Mining and Mineral processing topics.

At the ITV unit in Belém, Vale has been offering a *Master's Degree Program in the Sustainable Use of Natural Resources in Tropical Regions* since 2012. This is a unprecedented initiative in Brazil as it is the first course offered in conjunction with a company from the mineral sector to obtain certification from the Coordination for the Improvement of Higher Education Personnel (Coordenação de Aperfeiçoamento de Pessoal de Nível Superior - Capes), of the Ministry of Education.

In Minas Gerais, ITV launched a pioneering partnership with Universidade Federal de Ouro Preto (UFOP) to structure the *Master's Degree Course in Instrumentation, Control and Mining Process Systems, and Master's Degree Course in Mineral Engineering*. This course has been approved by Capes in 2015 and classes began in 2016. Another innovative partnership that has been being developed is with the ISPT-Instituto Superior Profissional de Tete, Mozambique, consisting in the offer of courses for the Master's Degree Program in Mining at ISPT. These courses are supported financially and operationally by Vale and Capes.

ITV also provides Specialization courses in Mining for Vale's employees and other courses for professional qualification, as well as research activities comprising levels from scientific research to post-doctoral degree.

Luiz Mello

In partnership with Capes, Vale offers academic postdoctoral scholarships for Brazilians and foreigners to researchers who are selected to work at Vale Institute of Technology units, further bringing together the academic community and Vale, expanding the potential generation of innovation and wealth for Vale and Brazil.

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It is a pleasure for us to participate in the 24th edition of the World Mining Congress - WMC 2016, being held for the first time in Brazil, and we can introduce you to some of the technological, research and innovation solutions in the Mining Sector. It is our commitment to share knowledge, innovation and technology towards the sustainable development of the operations and processes in global mining.

I hope that everyone enjoys the most of the World Mining Congress!

Luiz Mello
CEO of Vale Institute of Technology
Technology and Innovation Executive Manager of Vale

José Fernando Coura

On behalf of the Brazilian Mining Association - IBRAM and its associates, I would like to offer a warm welcome to all the participants of the 24th edition of the World Mining Congress - WMC 2016. This is the first time that the WMC, recognized as one of the most important world mining events, is being held in Brazil. The central theme of this congress is "Mining in a World of Innovation", one of the most current and important issues in the management of mining-sector businesses.

The 24th WMC began to take shape in 2012 when representatives from businesses and entities of the mining sector, as well as the Brazilian government, joined forces to support the country's bid, before the International Organizing Committee, to host the congress (IOC). This was well-deserved, given Brazil is one of the international exponents of mining.

The presentation of the Brazilian bid was made by IBRAM's presidency in conjunction with our Director of Mineral Issues, Marcelo Ribeiro Tunes. It fell to him to deliver the speech underlining the qualities that make IBRAM suitable to organize such an event, of the city of Rio de Janeiro (RJ) to attract and host event participants, and the Brazilian mining industry; factors which proved decisive in convincing the IOC members to choose Brazil as the host of the event in 2016.

With this significant vote of confidence, we are certain that the 2016 WMC will be the stage of an intense diffusion of knowledge, of discussions on the best way forward, and deep analyses of the current and future landscape of the mining industry. Without a doubt, it will also serve as a way to strengthen relationships and enable dialogue between the most diverse actors of the sector's extensive production chain on an international level.

We know that the last few years have been challenging for the mining industry and "innovation" is the key word for new business and the future of the sector itself. The economic environment has altered the rhythm of supply and demand, impacting ore prices and making it more difficult for mining companies to outline their next steps both locally and globally. Nevertheless, this moment offers an opportunity for mining to lay the way for a return to greater productivity in the future.

This is the proposal of the 24th edition of the WMC, amongst others. We also intend to technically and scientifically promote and support cooperation to develop more stages in the sustainable development of operations and processes in the mining sector.

With an optimistic vision of the prospects of the mineral sector, I hope that IBRAM, via this grand event, can awaken the public interest to debate the future of mining and identify innovative actions to further strengthen the mining industry around the world.

We wish everybody an excellent World Mining Congress!

José Fernando Coura
CEO of the Brazilian Mining Institute

Murilo Ferreira

Brazil has a historic vocation for mineral extraction activities, and since the mid-18th century they have practically dominated the dynamics of its economy. Rich in world-class minerals, the country has emerged as one of the leading global players in the mining industry, and it is now the second largest iron ore producer and one of the most significant agents in international trading and exports of this commodity.

The mining industry has become one of the most important pillars of Brazil's development. Despite the decline in iron ore prices and demand in the international markets, especially due to the slowdown in Chinese consumption, and despite the end of the super-cycle, the mining sector has continued to play a key role in maintaining Brazil's balance of trade surplus.

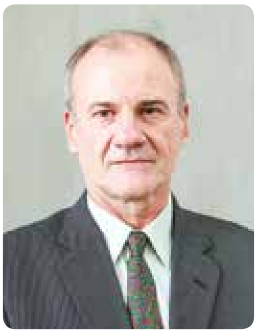
In addition to its positive impacts in the macroeconomic sphere in Brazil, mining has also become a driver of social development, particularly as it has a multiplier effect on other economic activities, contributing to the expansion of various production chains and consequently to the generation of jobs and income. It is noteworthy that in the municipalities where mining companies operate, Human Development Index ratings have been higher than the average figures for their respective states, and much higher than in non-mining municipalities.

In a country like Brazil, whose economic growth, as already mentioned, is strongly dependent upon the expansion of mining activities, the creation of the Brazilian Mining Association, which will turn 40 in December, was essential and absolutely necessary. This is a date to be celebrated, above all because IBRAM has played its role to support and strengthen mining activities with dynamism, efficiency and innovative practices. The sector's companies and organizations can count on a body that assertively and competently represents, coordinates and integrates them, defending their interests and generating conditions conducive to the sustainable development and competitiveness of their businesses.

The holding in Brazil of the 24th edition of the World Mining Congress, organized by an entity of IBRAM's quality, is a milestone and an excellent opportunity for the sector to share ideas, discuss, reflect and find stimuli and feasible ways forward at a time when we need to face the end of the mining super-cycle. The theme of the Congress could not be more appropriate, and I am sure that by its end, promising directions will have been mapped to strengthen the mining industry across the world.

Murilo Ferreira
Chief Executive Officer, Vale S.A.

Professor Jair Carlos Koppe



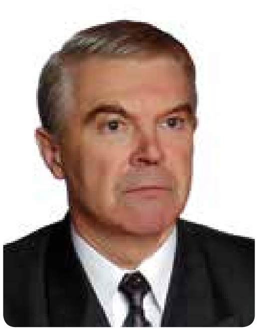
Mining has been extremely important to the World's economic growth and prosperity for centuries. The mining industry is currently facing an economic and social crises that can impact strongly the mineral production and productivity. In this scenario several challenges must be addressed, among them complex mineral deposits of low grades, water, social and environmental issues as well as declining commodity prices. Considering that the world is changing dramatically in all aspects this is the moment for innovation in mining. The WMC 2016 is under the umbrella of Mining in a World of Innovation in the proper moment. This is a nice opportunity to change our ways in mining technology considering the new evolving technologies such as automation, sensors, cloud computing, data analytics that can increase the mining production and efficiency in the entire value chain. Let's take this moment to spread our experience among academy, industries, practitioners and professionals of the mining sector focusing in the future of a world in constantly innovation.

We would like to thanks all the contributions done by the authors invited speakers and participation of delegates that will make WMC 2016 a very successful meeting. Special thanks to the members of the Scientific Committees that helped in the paper analysis ensuring the quality of the conference.

Welcome to the WMC 2016.

Professor Jair Carlos Koppe
Congress Chairperson

Józef Dubiński



Hermínio Oliveira

The 24th World Mining Congress is one of the most important mining events worldwide and is going to be held in Rio de Janeiro, Brazil, from October 18 to 21, 2016. The premiere of the World Mining Congress took place 58 years ago, in September 1958, in Warsaw, Poland. Currently, the WMC organization gathers 45 mining nations from all over the world.

Each World Mining Congress, which takes place in a different host-nation, is always a great mining occasion for the international community that represents science and industry figures involved in the exploration of mineral assets. We can assert that this congress points to the most significant directions for global mining development and determines priorities for the activities of all institutions related to mineral activity. The same approach is going to be adopted during the 24th World Mining Congress, which is going to concentrate on the theme of "Mining in a World of Innovation". Nowadays, and increasing number of countries hold great knowledge potential on mining. The challenges aforementioned demand mutual cooperation, exchange of technical knowledge and professional experience, as well as assistance to those in need. Personally, I believe that our generation of the world mining society – the heirs of our illustrious ancestors – will follow their accomplishments and guide the organization of the World Mining Congress into a new direction, to assure many more years of effective services to global mining and to the people who have taken part in this challenging activity, yet still necessary for all humankind.

Józef Dubiński
Professor and Doctor of Engineering
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Summary

BLASTING ENERGY MANAGEMENT TO MAXIMIZE PRODUCTIVITY IN A TYPICAL BRAZILIAN QUARRY

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BLASTING ENERGY MANAGEMENT TO MAXIMIZE PRODUCTIVITY IN A TYPICAL BRAZILIAN QUARRY

ABSTRACT

Pavimentadora e Construtora Falchetti is a typical Brazilian quarry operation. It is located at Tubarão, Santa Catarina state. The crushed stone produced is a hard granite, where the comminution starts using drilling and blasting and later a jaw and a cone crusher. This study shows the optimization by increasing powder ratio, changing the type of explosive used and the introduction of simple indicators to help understand where the energy (and the money) is wasted. Knowing that comminution commences in the pit with the rock fragmentation by blasting and continues at the primary crushing and cone crushing, the primary objective was to improve overall efficiency, specially by minimizing stoppages created by boulders at the jaw crusher, what was approximately 29,57%.

With the use of simple and inexpensive measurement techniques at the muckpile and primary crusher feeder, detailed size distributions per blast were available for interpretation. Later a comparison between initial particle size to feed the crusher and all sequential costs related in the decreasing rock size process were studied. It was easy to realize that there is a direct relation between explosive energy use and comminution efficiency.

KEYWORDS

Efficiency, Blasting, Comminution, Explosive energy

INTRODUCTION

Drill and Blast plays a significant role on quarry production chain, been the first step for the whole process. In order to achieve the designed production for the mine, it is essential that the rock obtained by blasting have fragments compatible with the loading/hauling equipment's and especially with the primary crusher. As loading and transporting operations contribute to a significant portion of production cost, it would be advisable to spend more energy input in the blasting operation in order to gain considerable energy savings in downstream operations. Improved fragmentation not only reduces the energy requirement in size reduction process, but also improves loading rates and reduces maintenance of machinery (Sastry and Chandar, 2008).

A quarry challenge is to achieve satisfactory fragmentation results and respect the increasing environmental restrictions at the same time. As presented by Lusk & Worsey (2005), the expanding urban environments are presenting a new challenge, where quarries located as close as possible to the cities but far enough from commercial and residential land have now suburban and shopping malls as well as neighborhood. Therefore, it is fundamental to manage blasting energy, maximizing rock fragmentation and minimizing the percentage lost as vibration or noise.

CASE STUDY

This study was conducted at the Falchetti granite quarry (Figure 1), located in southern Brazil, with a production capacity of 3000t per day. The quarry excavation process is a traditional descending bench method, with varying heights from 12m to 15m. Because of geology and environmental constrains, different patterns are required for certain portions of the mine. Blast pattern usually is 2 x 4m with blastholes 63,5mm (2 ½") diameter, emulsion explosives and shock tube as initiation system.

Figure 1: Falchetti quarry overview and neighborhood.

The granite rock mass is highly heterogeneous, where it may be found in the same blasting pile different compositions. The fractures are intense and irregular; also, there are two intrusions of igneous rocks. Blast performance is highly influenced by those geological parameters and comparisons between benches from different portions of the mine must be avoided. With that in mind, an investigation was conducted under similar benches mainly on the lower portion of the mine. Because of intense fracturing, water is present in most parts of the rock mass and there is no dewatering system available for the blasters, so packaged or bulk emulsion is often used.

Operational and environmental restrictions are related to buildings near the quarry. It can be noted at Figure 1 the urban area close to the pit. In addition, 300m away is the Morro do Formigão tunnel located at highway BR 101 (South-North), highway SC 390 (East-West) close to the community. The National Transportation Department (DNIT) established the maximum allowed charge per delay (kg of explosive/delay) for the rock blast in order to assure the tunnel structure security. In addition, the company also follows the limits set by the Brazilian standard regulation for blasting in urban areas (NBR 9653-2005). The quarry historically detonates one blasthole per time in a "serpentine" sequence (Munaretti, 2013).

METHODOLOGY

A total of four shots were analyzed from April/15 to February/16 where energy consumption, ROM production at the primary crusher and muckpile size distribution were monitored and compared to the old regular blasts.

Table 1: The four shot analysis.

Burden x Spacing	Diameter (mm)	Explosive	Initiation system	# holes	Height (m)	Powder factor (kg/m³)	Cost (R$/ton)
2 x 4	76,2	Packaged emulsion	Shocktube	100	13,45	0,49	1,66
2,3 x 4,6	76,2	Bulk emulsion	Shocktube	100	13,37	0,55	1,63
2,15 x 4,3	63,5	Bulk emulsion	Electronic	145	12,54	0,44	1,76
2 x 4	63,5	Bulk emulsion	Shocktube	156	12,5	0,48	1,53

The quarry traditionally used 2m x 4m burden to spacing, 63mm (2 ½") blastholes, 2m stemming with drill cuttings and packaged emulsion for a "serpentine" blast pattern or one by one linear sequence. Quarry management relied on the fact that the quarry is located on an urban area and the blast design would not present any risk of high ground vibration. The constraints on the design and old myths about blasting usually presented several problems for hauling, loading and crushing. Before the study started, the authors noted:

- Excess of boulders;
- Crusher production below the expected;
- Crusher often running empty;

- Low loading efficiency of the excavator and need of secondary breakage by dropball;
- Trucks on a line for long time waiting for the excavator to load;
- High costs for truck maintenance;

The granite presents intense fracturing, water is often present and the management states that it is not cost effective to dry the holes by dewatering or even using liners in this particular operation. Because of that, emulsion is the only option. The blasters were also used to attach an extra shocktube at a higher portion of the blasthole trying to guarantee the explosive column blast sequence, which represent an extra cost about 4% of the total blast. After the tests with the new procedures, this old myth was left behind.

According to the primary crusher nominal production, the goal was to obtain maximum particle size about 20,32cm (8”), which would increase hauling, loading production and transportation. The second objective was to increase primary crusher production while reducing its final product size from 12” to 8”. The authors also monitored the primary crusher energy consumption expecting a significant evidence that blasting affect crushing and grinding results directly, and that large savings can be achieved (Eloranta, 1995). Minimum changes were performed, just using the best blasting practices, such as adequate stemming height and material, bulk explosive for better coupling, adequate timing and initiation sequence, low scattering delays for detonators, and adequate burden and spacing. In order to respect environmental constrains and also provide adequate blast relief, the blastholes would be fired in a sequence one by one, however not using the historical serpentine pattern, instead a diagonal pattern was be used (Figure 2) for maximum burden relief and fragmentation (Munaretti, 2013).

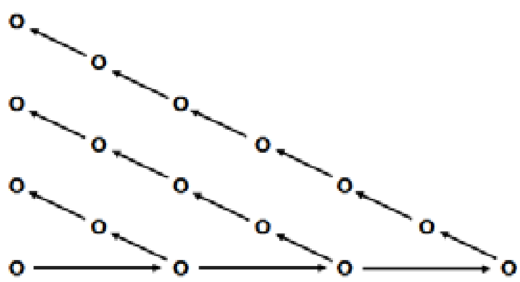

Figure 2: Diagonal blasting pattern used.

In order to measure size distribution, a digital image analysis technique was used, where a 12.5cm radius yellow ball was the scale. A rock fragmentation size analysis software (Split-Desktop-academic license) was chosen (Figure 3).

Figure 3: Muckpile image sampling.

RESULTS

The production results from the test blasts were compared to the ones obtained from March/13 to February/15. This period was chosen as the hauling, loading and crushing equipment were the same as the ones monitored for this work. Older records were not registered for this work, as they were not reliable and could lead to unreal comparisons.

As it can be seem in Table 2, production increase was about 29,57%. Although the results presented refer to medium values, high production levels could be achieved with daily production results exceeding the crusher nominal production capacity of about 3000t/day.

Table 2: Production comparison.

Period	(t/day)	(h/day)	Flow rate (t/h)
Mar/13 - Feb/15	1.815,33	6,48	281,67
Apr/15 - Feb/16	**2.352,13**	6,80	**355,84**

According to the mine management, the production increase seems to be driven mainly related to two factors: (i) stemming height reduction and change from regular drill cuttings to 9,5mm aggregate as stemming material. (ii) Change from packaged emulsion to bulk emulsion which increased the coupling and the powder factor (Munaretti 2002). Those changes significantly reduced the particle size, improving the production, as the boulders were projected to the front portion of the pile. The company was historically performing secondary blasting by dropball, practice that tends to be minimal now after the tests. It was also noted that bulk emulsion reduced significantly the explosive loading time and was more effective due to better coupling, fast loading and less ergonomically problems for the blasters compared to packed emulsion.

A significant boulder decrease was observed at the front portion of the muckpile. It was common to observe boulders larger than the crusher opening in the front portion of the pile, generated by excessive stemming. Figure 4 shows a picture comparing two blasts realized in 2012 and 2016 at the same bench and pit portion. It was possible to note the difference in the muckpile size distribution, especially in the front portion. The blast in Jul/2012 used packaged emulsion, 2 ½" blasthole, burden/spacing 2m x 4m and stemming of 2m with drill cuttings. This blast required excessive secondary breakage and rock fragments larger than 800mm were observed, resulting in high costs. Excessive secondary breakage with dropball increased loading/hauling cycle time considerably, and as consequence the primary crusher worked empty, increasing unitary costs.

The blast in Jan/2016 used bulk emulsion, 2 ½" blast hole, burden/spacing 2m x 4m and stemming of 1,2m with crushed stone. This blast allowed for loading/hauling cycle time reduction, also as the particle size distribution was more suitable for the primary crusher feeding higher production was achieved. Furthermore, in Figure 4 it may be noticed the particle size difference in the front part of the muckpile, occurrence of boulders generated from the stemming portion were reduced in the blast in Jan/16 as a consequence of better explosive energy use.

Figure 4: Blasting in Jul/2012 (left) / Blasting Jan/2016 (right).

Figure 5 shows the size distribution for the muckpile for the Jan/2016 test and also three other additional tests. The results were considered satisfactory, where it was possible to achieve 80% to 60% percent passing at 8" (200mm) sieve. Secondary breakage was minimal.

Figure 5: Particle size obtained on the tests

In addition, energy consumption on the primary crusher was monitored and the results show a trend. It was also noticed that the energy consumption was not regular as expected, because of several other factors that may affect the results. Even though, the results indicate that it is possible to increase the production in the primary crusher without raising the costs at the same proportion, in other words, producing more with less.

The energy consumed can change in three ways. First, if the feed size to the primary crusher is decreased, less energy will be required to crush the ore to the same product size. Second, a decrease in Work index Wi related to additional macro and microfracturing within individual fragments. Third, an increased percentage of undersize that bypasses stages of crushing thereby decreasing the percentage of total tons crushed (Eloranta and Workman, 2003). The first affirmative made by Eloranta was the initial observation realized in the field, where an energy consumption-measuring device was installed in the primary crusher panel and tests were done with different size feeds on the crusher. As expected, when crushing boulders, the crusher current increased significantly, which means higher crushing costs. The increase in the percentage of undersize that bypasses the primary crushing stage allows the reduction of jaw wear, which also extends its life.

Figure 6 shows the relationship between production and energy consumption at the quarry. Improve blast performance is not a simple matter of increasing powder factor but instead, the whole blast design and quality control, mainly stemming, explosive coupling, initiation sequence, delay scattering, adequate relief, muckpile format, burden and spacing.

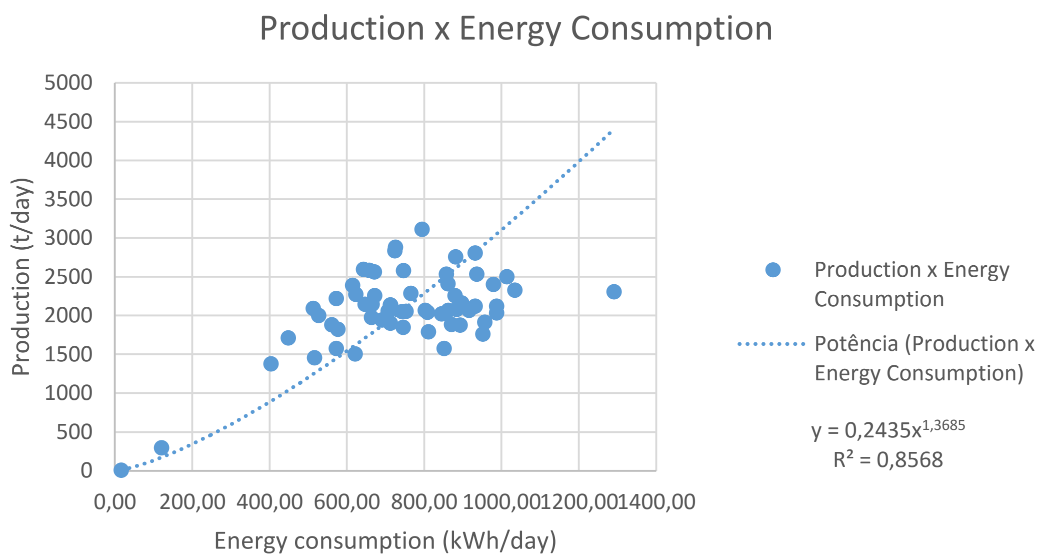

Figure 6: Production x Energy consumption at the mine (April/2015 – February/2016).

The quarry runs about 20 days per month, since the production increase was of 536,8 t per day, and considering the final product sale price of 25 R$/t, the company gross revenue increase was R$ 3.220.800,00 per year.

The production cost was estimated before and after the blasting changes. A few assumptions were made in order to compare the scenarios:

- Trucks, excavator fuel consumption and maintenance costs were kept constant, not depending on the quality of the material generated by the blast but only on the amount of hours worked;
- Primary crusher energy consumption, consumable wear and maintenance costs were kept constant, not depending on the feeding material but only on the production achieved.

The costs were considered starting from drilling until the primary crushing. The total costs can be calculated through the following equation:

$$Total\ cost = D\&B + \frac{\left(Truck_{\frac{R\$}{h}} x\ h\right) + \left(Excavator_{\frac{R\$}{h}} x\ h\right) + \left(Labor_{\frac{R\$}{dia}}\right) + \left(Crusher_{\frac{R\$}{h}}\ x\ h\right)}{Production_{t/day}}$$

The total production cost before the testing was estimated in 6.16 R$/t, while the total production cost was estimated in 5.43 R$/t for the new blast design. This represents a reduction of 11.85% on the production costs from drilling to primary crushing.

CONCLUSIONS

Blasting is a key part on the production chain and is direct related and determinant to all subsequent operations such as loading, hauling, transportation, crushing and grinding. Because of that, its continuous improvement should be always the first goal of any mining or quarrying company.

The advances in blast fragmentation provided significant financial feedback. No investments were made on this mini *drill-to-crusher* project, as the measurement techniques used were simple and inexpensive. The best blast practices are mandatory to achieve best results and require a minimum understanding of geology heterogeneity, its relation with correct explosive placing (drilling), blasting energy use, adequate stemming, burden-spacing pattern, and initiation sequence. Blast design improvement should consider not only the powder factor but instead look at it as a whole comminution process, also considering the whole process costs.

The first results show a projected financial gain of about R$ 3 million per year on gross revenues and also a production cost reduction of 11,85%, keeping the same equipment and labor without any investment. For future work, it is planned to try ANFO or heavy ANFO instead of bulk emulsion which could lead to significant increase in profitability. To achieve this a dewatering system will have to be used.

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CHALLENGES IN MODELLING GEOMECHANICAL HETEROGENEITY OF ROCK MASSES USING GEOSTATISTICAL APPROACHES

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CHALLENGES IN MODELLING GEOMECHANICAL HETEROGENEITY OF ROCK MASSES USING GEOSTATISTICAL APPROACHES

ABSTRACT

Modelling rock mass geomechanical heterogeneity plays an important role in optimization of mine planning and design. Understanding the spatial variability of rock mass engineering properties allows for a better prediction of the rock mass behaviour during excavation (i.e., rock mass stability, blast-induced rock mass fragmentation), and during comminution (crushing and grinding behaviour of rock). Geostatistical techniques are commonly used for resource modelling (i.e., modelling ore grade and tonnage). Their application for geomechanical attributes also becomes routine for many mining operations; however, the spatial modelling of the engineering geomechanical properties has its own challenges and difficulties. Interpretation of the geospatial model of geomechanical properties without a good understanding of these challenges, assumptions and limitations could lead to decision-making that does not necessarily result in appropriate consequences. This paper addresses major challenges with spatial modelling of geomechanical attributes using the information from a case study of an open pit mine in Quebec, Canada, where more than 200,000 m. of geotechnical borehole data, drilled and logged in the pit area since 1949, was used for 3D block modelling of various geomechanical attributes. The properties that are modelled include RQD, RMR, and intact rock strength. Unlike ore-grade data, some of these geomechanical variables are non-additive and direction dependent, which makes the process of 3D block modelling more challenging. Geomechanical data are often scattered or widely dispersed across the orebody and are generally sensitive to core drilling technology used for rock sampling. In addition, these properties often show complex multivariate relationships. Using simple geostatistical estimation methods, such as kriging, for geomechanical variables creates a unique model based on the available dataset. However, the krige model is not reliable in representing the extreme rock mass quality. In other words, the 3D block model overestimates the poor quality rock masses and underestimates the good quality rock masses. Using conditional simulation for spatial modelling of rock mass geomechanical properties can generate multiple equi-probable models which allow for the reduction modelling biases related to the estimation of non-additive variables.

KEYWORDS

Geomechanical properties, 3D block model, geostatistics, heterogeneity modelling

INTRODUCTION

Design of an optimal open pit mine is a challenging problem for mine engineers due to the spatial variations of geomechanical properties of rock masses that form the pit area. A thorough understanding of the geomechanical and geological behaviour of the entire rock masses across the mine is crucial for optimal mine planning and design. Understanding the heterogeneity of rock masses' geomechanical properties enables the mine design engineers to identify the high-risk zones and the mine sectors that require more exploration and data collection attempts. In open pit mine projects, modelling of the geomechanical heterogeneity can improve the reliability of pit design and prediction of the rock mass behaviours during the excavations (i.e., rock mass stability, blast-induced rock fragmentation) and prediction of rock mass behaviour during comminution process (identification of geometallurgical domains). For example, such models can be considered as inputs into the blast design process in order to determine the optimal powder factor of an individual blast required for a target fragmentation (Bye, 2011). Consequently, this would improve the mine-to-mill value chain by enhancing the loading rates, transportation and mill throughput.

The common practice in open pit design is to create a deterministic boundary model of the subsurface geomechanical properties (Hack et al., 2006). In this boundary model, which is generally used in the earlier stages of a mine design with presence of limited data, the rock mass in the mine site is divided into a number of geotechnical domains according to the collected data of subsurface condition and engineering judgements (expert knowledge). Basically, some structural features, such as changes in lithology, presence of major structures and characteristics of joint sets, are considered as the boundaries of the geotechnical domains (Read and Stacey, 2009). Each geotechnical domain is assumed to be uniform with in which the geomechanical properties are homogeneously distributed. An average value is generally assigned to rock mass properties of each domain, and it is assumed that the average value represents the property across the domain. These average values of rock mass geomechanical properties are then used in numerical models for slope stability analysis to determine the safety factor. Finally, a sensitivity analysis is generally performed to determine the effects of each geomechanical property on the pit slope stability in order to explore the variation of the geomechanical properties of rock masses. This conventional deterministic method ignores the local spatial variability of the rock mass properties within each geotechnical domain because it assumes that the geotechnical domain is homogeneous. Research indicates that predictions made based on these conventional deterministic boundary models have a lower probability of being accurate and the resulting design decisions are generally questionable (Hammah et al. 2009).

An alternative approach for modelling the geomechanical heterogeneity in open pit mining is to apply geostatistical methods. Geostatistics, or theory of regionalized variables (Matheron, 1963), is a branch of applied statistics that models a natural phenomenon by quantifying the variability of one or more of the so-called regionalized variables (Journel and Huijbregts, 1978). In the geostatistical techniques, both random and structured characteristics of the regionalized variables are modelled in a mathematical context. Geostatistical techniques were originally developed for ore reserve, where the main regionalized variable of interest is the ore grade. The early efforts of using geostatistical methods in geomechanics were done by La Pointe (1981), who applied geostatistics to investigate the spatial distribution of fracture properties in order to improve the design of rock structures. Since then, many researchers have applied the geostatistical interpolation and simulation methods for analysis of spatial distribution of geomechanical properties. Some examples are the works by Vatcher et al. (2016), Doostmohammadi et al. (2015), Ferrari et al. (2014), Egaña and Ortiz (2013), Marache et al. (2009), Stavropoulou et al. (2007), Marinoni (2003), Ayalew et al. (2002) and Giles (1993). The principle reason behind the slower adoption of the geostatistical techniques in the geomechanical mine design projects can be attributed to different challenges in the application. Modelling the spatial heterogeneity of the geomechanical properties using the geostatistical tools requires special attention to the specifications and limitations of the geomechanical variables. A better understanding of these challenges in the spatial modelling of such variables leads to careful selection of the spatial modelling approach in order to achieve the most reliable representation of the deposit. Each of the challenges and restrictions could provide some uncertainties in the process of spatial modelling of the geomechanical variable using these geostatistical techniques.

This paper focuses on the challenges in applications of the geostatistical methods for the heterogeneity modelling of geomechanical variables. Of interest is the Rock Mass Rating (RMR) geomechanical classification system developed by Bieniawski (1976). RMR is commonly used for geomechanical characterization of rock masses and estimation of their mechanical properties. RMR is calculated as the sum of the ratings for its constituent parameters including intact rock strength; joint spacing (the mean spacing between discontinuities and can be calculated as the reciprocal of the fracture frequency); Rock Quality Designation (RQD) measured as the ratio of total length of all pieces of rock core larger than 10 cm (4 inch) to the total length of core run; joint condition (a measure of joint surface roughness and alteration, and its filling material); and water condition (a measure of ground water condition in the project area). An iron ore open pit mine in Quebec, Canada, was used as a case study to model spatial variation of RMR geomechanical attribute using the geostatistical simulation method. Lessons learned from this case study were then used to discuss the major challenges with the use of geostatistical methods for modelling geomechanical heterogeneity of rock masses.

Geostatistical Modelling of Geomechanical Properties in Mont-Wright Open Pit Mine

Mont-Wright (MW) is a major iron ore deposit that is owned and operated by ArcelorMittal Mines Canada (AMMC) Corporation. The mining complex is located in northern Quebec in Canada, 16 km. west of the mining town Fermont. It is also located approximately 1,000 km northeast of Montreal. The mine site can be accessed from the closest city in the south, Baie-Comeau, via Highway 389. Mont-Wright has operated since 1975 and in total 1,285 million tonnes of iron ore have been extracted up to 2012 (Savard and Jean, 2012). The iron ore concentrates of the MW are shipped to the AMMC's pelletizing plant and shipping terminal located at Port Cartier on the north shore of the Gulf of St. Lawrence through a 416 km private railway owned by AMMC. Figure 1 demonstrates the Quebec regional location map of the MW mining complex. There are several active mining regions in Mont-Wright, including Paul's Peak, Pit A, Pit B and C-prim. The pit of study in this paper is the Paul's Peak pit, which is the largest open pit within the mining complex. Figure 2 depicts the Paul's Peak open pit looking from the east wall. Currently, the Paul's Peak pit extends ~1800 m of length, ~600 m of width and ~320 m of depth. The pit will be expanded to a depth of ~600 m by year 2045.

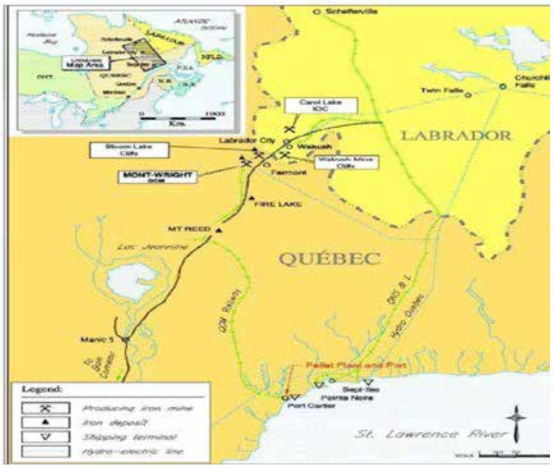

Figure 1. Map showing the location of MW mining complex in Quebec-Canada

Figure 2. Paul's Peak open pit (looking from the east)

Mont-Wright iron ore deposit is a part of the highly folded and metamorphosed south-western branch of the Labrador Trough, where the most important rock type is the specular hematite iron formation containing recrystallized quartz and hematite with minor magnetite and iron silicates (Savard and Jean 2012). The complex structural geology of the Mont Wright arises from the existence of sever folding into a series of synclines and anticlines, as well as the presence of a secondary folding that has made wide zones of specular hematite up to 300 m in width (Savard and Jean 2012). Figure 3 shows the spatial distribution of rock types in a plan view of the Paul's Peak pit. The view shows a folding system in the far east of Paul's Peak.

Since exploration of Paul's Peak started in 1949, 871 drill holes with a total length of more than 200 km have been drilled and logged for geological and geotechnical information. Figure 4 shows a crosssection of the pit with the RMR logging information of some drill holes have been superimposed. The geomechanical properties measured/assessed along the boreholes include RQD, RMR, fracture frequency, joint condition and intact rock strength. Table 1 provides a summary of the simple statistics for the geomechanical properties in the drill-hole dataset. This table indicates that the rock mass in the Paul's Peak formation has, on average, a good quality of RMR and RQD with very high intact rock strength. In terms of fracturing, the rock masses have closely spaced joints, and their surfaces are characterized on average as slightly rough with hard joint walls.

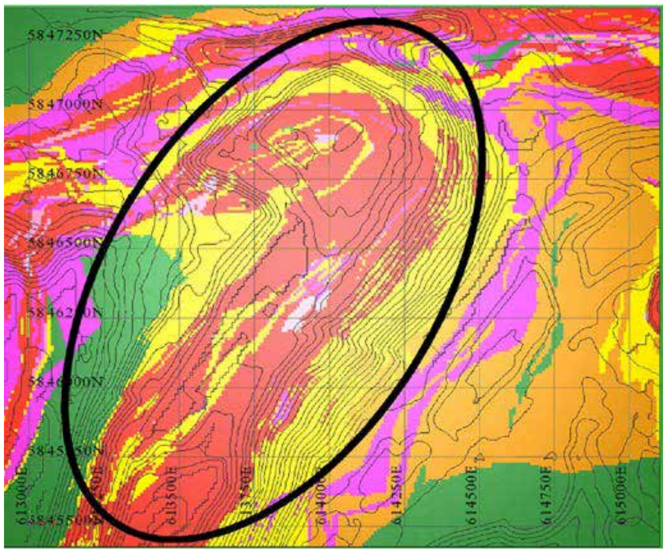

Figure 3. Lithological map of Paul's Peak open pit mine (year 2013)

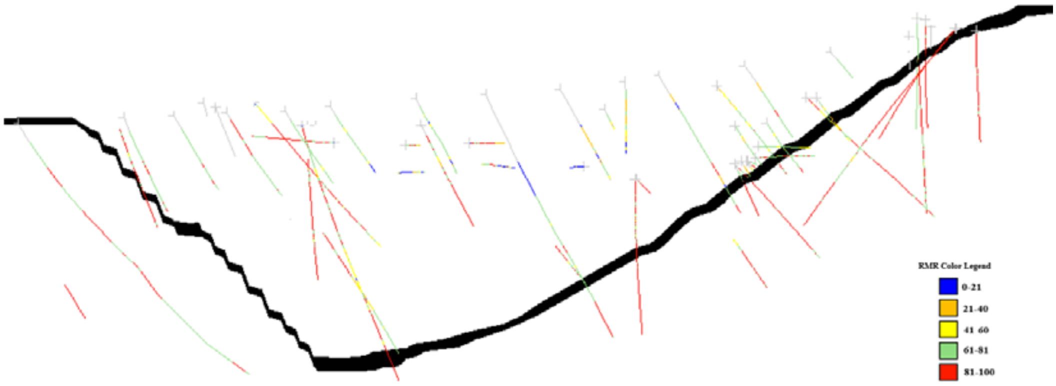

Figure 4. A cross section of the Paul’s Peak pit showing the logged values of RMR along drill holes.

Table 1: Basic statistics for geomechanical properties in Paul's Peak

Property	No. of samples	Min.	Max.	Mean/Mode*	Median	STD	CoV	Description**
RQD (%)	65796	0	100	80	90	25	0.31	Good
Fracture Frequency (m^{-1})	65794	0	65	5.3	4.0	6	1.29	Joints Closely Spaced
UCS (MPa)	65176	5	380	201	193	89	0.44	Very High
Joint Condition	65176	0	25	17.2/20.0	20.0	5	0.31	Slightly rough/hard joint wall
RMR	65176	12	100	78	81	16	0.21	Good

*Mode is recorded only for the Joint Condition
**Descriptions are based RMR 1976

Using the geomechanical data of the drill-hole dataset, five 3D block models were developed using the sequential Gaussian simulation (SGS) technique for each of the constituent geomechanical properties of RMR including: RQD, joint condition, UCS and fracture frequency. The water condition was assumed as dry, which is the common condition at the pit. The resulting 3D block models were then combined according to the RMR_{76} procedure in order to develop the corresponding five RMR block models (summing of the ratings for the RMR constituent parameters of each block). Table 2 compares the qualitative results of simulated 3D block models and the geomechanical drill-hole data. According to this table, the built 3D block models honour the quality of rock mass in terms of RQD and UCS. However, the simulated 3D block models of fracture frequency provide rock masses of moderately spaced joints in comparison with the closely spaced jointed rock mass logged in the geomechanical drill-hole data. This indicates that the simulated block models slightly underestimate the fracture frequency (from mean 5.3 $m.^{-1}$ in the geomechanical drill-hole data to mean 3.75 $m.^{-1}$ in simulated block models). Consequently, the resulting simulated block models of RMR slightly overestimate the RMR values from good quality in the geomechanical drill-hole dataset to very good quality in simulations; however, this overestimation is slight, as the average of simulated block models of RMR is 82, which is a little more than mean 78 recorded in the geomechanical drill-hole data. Figure 5 shows the Q-Q plot for the simulated RMR values versus the RMR values from the rock sample data. The simulation honours the extreme rock mass qualities (low and high rock mass qualities). Figure 6 displays the 3D block model of RMR which has been mapped on the Paul’s Peak walls. This figure shows the heterogeneity of RMR in the Paul’s Peak pit. The majority of the rocks in the Paul’s Peak pit are very good quality in terms of RMR.

Table 2: Comparison of simulation results and drill-hole data

Geomechanical Property	**Qualitative Description**	
	Drill-hole Data	**Simulation**
RQD (%)	Good	Good
Fracture Frequency (m^{-1})	Joints Closely Spaced	Joints Moderately Spaced
UCS (MPa)	Very High	Very High
RMR	Good	Very Good

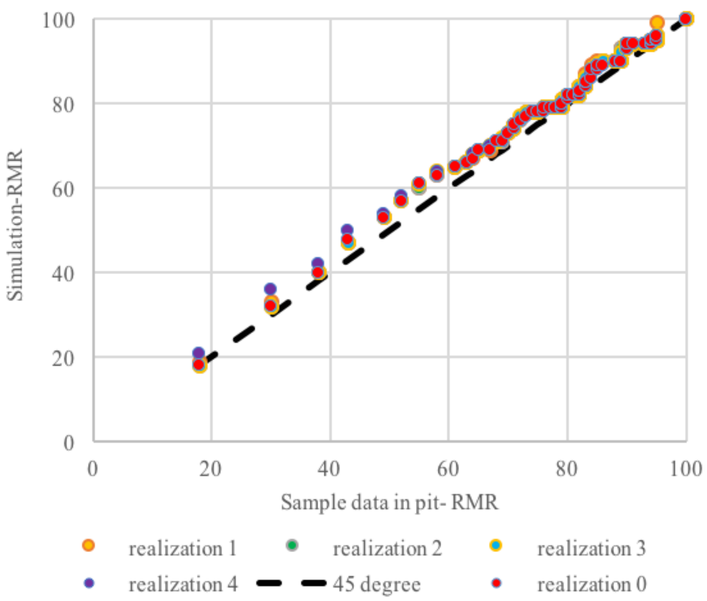

Figure 5. The Q-Q plot of the modelled RMR against the measured RMR values in the pit.

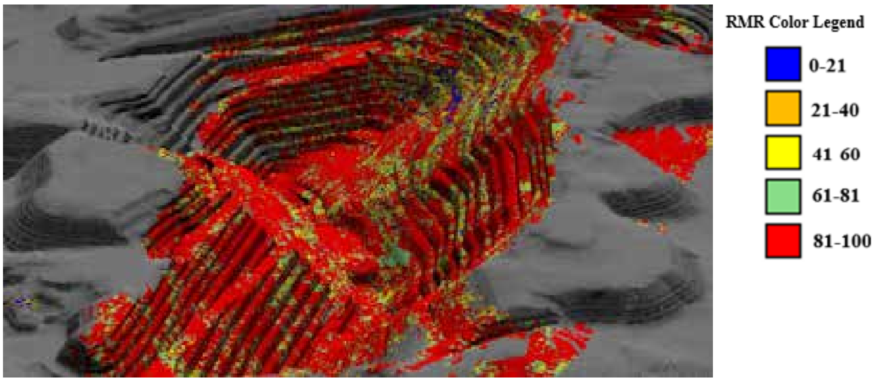

Figure 6. One realization of the RMR model mapped on the Paul's Peak pit walls (Looking towards East).

During the process of spatial modelling of geomechanical properties of the rock masses in the Paul’s Peak open pit mine, several questions were raised regarding the reliability of the developed spatial models and the nature of the input data used for the modelling. Application of geostatistical modelling for geomechanical data has some challenges, which are common in most of the geomechanical modelling projects and are discussed in the following section.

CHALLENGES IN GEOSTATISTICAL MODELLING OF GEOMECHANICAL PROPERTIES

Lack of geomechanical data

Despite the wide use of geostatistical techniques for 3D modelling in mineral and oil-and-gas exploration projects, geotechnical projects are slowly adopting such methods for spatial modelling of geomechanical properties. This is due to the fact that the geostatistical approaches require sufficient amount of data for the statistical inference. Traditional open pit mine design and optimization have been limited by the amount of detailed geomechanical data. Unlike ore grade data, which is generally closely and systematically spaced across the orebody, geomechanical information is often scattered or widely dispersed in the area of study, which can make it difficult to apply geostatistical approximation techniques properly. Although many exploration drill holes are generally drilled to delineate an orebody, a limited budget is allocated to geomechanical borehole drilling. Most of the core samples from exploration drillings that contain ore are typically split for assay as soon as possible after drilling. This renders measurement of geomechanical attributes very difficult and destroys opportunities for collecting valuable geomechanical data from mineral exploration boreholes that generally have a higher density and better coverage of the area of investigation. One of the reasons that mining and exploration companies hesitate to undertake a geotechnical core logging prior to splitting of the core samples is that the conventional geotechnical core logging process is a very time-intensive activity. Re-logging of the exploration cores after their handling and storage also causes significant deviation in the recorded data (Macciotta et al. 2014) and can be technically difficult (Blenkinsop and Doyle 2010). In the Paul’s Peak open pit mine, the majority of the exploration drill holes were logged to collect geomechanical properties prior to use the core samples for assay analysis. This additional effort provided adequate amount of input data for spatial modelling of the geomechanical attributes of the open pit mine. The simulation models demonstrate more uncertainty in the pit boundaries where the geomechanical data are more widely dispersed.

Subjectivity in geomechanical data collection

Unlike the ore grade data, in which the rock samples are carefully selected and assayed in the lab to measure the constituent elements, there is a significant degree of subjectivity in the geotechnical data collection during the site investigation (Egaña and Ortiz, 2013). This subjectivity comes partly from the subjective definitions of the geomechanical properties, and inconsistency in reports (e.g., two core loggers report two different values for a geomechanical variable of a core run). Logging the joint condition based on joint surface roughness, shape of the joint surface and joint wall strength is a very subjective process. This subjectivity in characterization also appears in logging other geomechanical properties such as RQD and fracture frequency. RQD is measured as the ratio of total length of all pieces of sound, slightly or moderately weathered rock cores longer than 10 cm (4 inch) to the total length of core run (Deere and Deere, 1988). Two different core loggers could interpret the definition of the *sound, slightly or moderately weathered piece of rock* differently.

Another source of subjectivity in geomechanical characterization is the estimation of a geomechanical property using other measured variables. This is also true for the logging of ore grades in the polymetallic deposits, where the values of some metal grades are estimated from other variables through regression, deconvolution and construction of a metal balance sheet (Journel and Huijbregts, 1978). A similar situation can be observed in the geotechnical logging, where, for example, the values of UCS for a rock is reported using an experimental regression between the point load strength index and rock hardness index (ISRM 1981) and the Uniaxial Compressive Strength. The values of RQD are sometimes estimated using the relationship between RQD and the measured values of fracture frequency.

In the Paul's Peak open pit mine case, the geomechanical database used for the geostatistical modelling was logged by more than 83 different core loggers between 1949 and 2013. The experts who logged the cores did not all have the same level of experience and geotechnical knowledge. However, it is practically impossible to remove the uncertainties associated with data subjectivity from the database.

Bye (2011) states that it is necessary to recognize the uncertainties caused by the lack of data and data subjectivity, and consequently identify the confidence level of the 3D spatial models using geostatistical techniques. Both lack of data and subjective assessment of some geomechanical parameters could significantly reduce the accuracy of the developed spatial models and consequently influence the decision-making process for mine planning and design.

Homogeneity

The data used for geostatistical estimation and simulation should come from a consistent statistical and geomechanical population (Egaña and Ortiz, 2013). One of the major sources of the uncertainty in the geostatistical approaches is the inhomogeneity of the data used for modelling. Homogeneity of the data is generally required for the statistical inference. Journel and Huijbregts (1978) suggest that homogeneity or representativeness of data used for the geostatistical modelling must be constant through time and space.

In many mature mining projects like Paul's Peak open pit operation, the homogeneity of the geomechanical data through time and space is difficult to attain. The inhomogeneity of the geomechanical data in time and space can be attributed to the advance of technologies and equipment used in different periods of time. For example, newer core drilling technologies and the use of triple tube core barrel would lead to better core recovery and generally better logged values for RQD and fracture frequency than older technologies and equipment.

The temporal/spatial homogeneity of data can be acquired only if constant methodologies are followed through the time and space for drilling, sampling and logging of the geomechanical variables. This can be achieved by using advanced quantitative logging methods, such as high definition digital photography and image analysis of rock core samples; online non-destructive hardness testing of core samples; and detailed geophysical characterization of drill cores (Dunham and Vann, 2007).

Additivity

The geostatistical estimation methods, such as ordinary kriging, estimate the values of a regionalized variable (such as ore grade) using arithmetic averaging of the values in the sampled locations. These methods assume that the regionalized variable being estimated is additive or linear. A regionalized variable is additive when the arithmetic mean of its values is its average. The additivity assumption is valid for regionalized variables, such as the ore grade where the support is constant (Journel and Huijbregts, 1978). Unlike the ore grade, the geomechanical variables such as RQD and RMR are not additive because their linear average would not necessarily be representative of the geomechanical behaviour of rock mass. For example, the stability of a pit slope is more highly influenced by areas where the rock mass is of low quality (low RMR or RQD); therefore, if a spatial model cannot accurately reproduce these poor rock mass quality zones, the model will fail to represent the ground condition for the slope stability analysis. The non-linearity of RMR also arises because RMR is calculated through a non-linear combination of its constituent variables (Egaña and Ortiz, 2013). In calculating RMR, each geomechanical variable (RQD, UCS, joint spacing, joint condition and water condition) is rated according to the class to which its value belongs.

Another factor that makes a regionalized variable non-additive is the support change effect. The support change has impacts on the additivity of the RQD, fracture frequency, intact rock strength and consequently the RMR. The support in the geomechanical logging is usually considered as the core run length. Therefore, it is essential that the geomechanical variables are logged and reported in consistent and constant core run lengths.

The estimates resulting from the linear averaging (e.g., using ordinary kriging) of the non-additive geomechanical properties can only be of use for visualizing the trend and obtaining some comprehension of the spatial features (Deutsch, 2013). The linear estimators provide the local average estimate where the histogram of the original data is not honoured (Journel and Huijbregts, 1978; Deutsch, 2002; Eivazy et al., 2015). Such local estimates are biased low or high depending on the type of geomechanical property distribution and the type of linear estimation (Deutsch, 2013). Eivazy et al. (2015) demonstrated that using geostatistical estimation methods for non-additive geomechanical properties like RMR can significantly overestimate the poor rock mass classes (low RMR value) and underestimate the very good rock mass classes (high RMR value); consequently, the estimated histogram becomes narrower than the sampling dataset. Unlike the linear estimators, the geostatistical conditional simulation techniques could be a better choice for spatial modelling of the geomechanical variables because they honour the histogram of the original data and do not over/underestimate the extrema of such variables, or eliminate the biases associated with modelling of non-additive variables. This was clearly demonstrated in the case study of the Paul's Peak open pit mine. In fact, conditional simulation methods are the only practical ways for modelling the heterogeneity of non-additive geomechanical variables because they realistically represent the short-scale variability and avoid the biases of the linear estimation methods. Such conditional simulation techniques also generate a collection of point estimates inside a block.

Direction Dependency

The geomechanical variables such as joint spacing (or fracture frequency) and RQD measured along a drill hole or a scanline survey are direction dependent. Depending on the structural heterogeneity of rock mass, the values of RQD and fracture frequency logged in the process of core logging can significantly vary when the orientation of the drill hole or the scanline traverse changes. Since RQD and joint spacing are the two components in the calculation of RMR, this variable can also show directional behaviour. To address the direction dependency in spatial modelling of geomechanical variables, it would be ideal if these geomechanical variables were logged using the drill holes/scanline traverses in various orientations. This allows for better detection of rock mass geomechanical anisotropy. In practice, this may not always be possible because the exploration drill holes are generally oriented with respect to the orientation of orebody.

Upscaling

The upscaling from the point estimates to the 3D block models is a vital step in the modelling of the regionalized variables. The upscaling of the estimated values of ore grade into 3D blocks is a routine matter in the geostatistical modelling. This is due to the fact that ore grade is an additive regionalized variable and the arithmetic average of the estimated values within a block can truly represent the upscaled value of the 3D block. However, the upscaling of the geomechanical variables is a challenging and complex task. The challenge in the upscaling of such geomechanical variables arises from the non-linearity and/or non-additivity nature of these variables. The function or blend response model (Dunham and Vann, 2007) that could effectively and realistically upscale a geomechanical variable is an important question that one should think about it before starting the modelling process. The matter of upscaling of the geomechanical variables is a key factor in selecting an appropriate geostatistical technique (Bye, 2011). In this regard, a thorough understanding of each geomechanical variable and its impacts on the downstream decision makings (e.g. pit slope design, geometallurgical modelling) is essential. This understanding can help in selection of the proper geostatistical tool and the upscaling approach. Depending on the type of the geomechanical variable, geostatistical linear, non-linear or simulation techniques might be appropriate. The upscaling problem of geomechanical properties becomes even more challenging if one decide to use the 3D block models of these properties as an input data for numerical modelling.

CONCLUSIONS

This paper has illustrated the challenges in applying geostatistical methods for modelling the heterogeneity of geomechanical properties. These challenges include the lack of geomechanical data in many mining projects; subjectivity of geomechanical variables in the data collection process; direction dependency of the geomechanical variables; and non-additivity, non-linearity and upscaling issues associated with these variables. These challenges are the major obstacles for the application of geostatistical techniques in the geomechanical projects. Each of these challenges and limitations can influence the choice of the proper geostatistical techniques that can be used for heterogeneity modelling. Without a good understanding of these challenges and limitations, spatial modelling of the geomechanical variable might not lead to a reliable interpretation of the modelling results and may adversely impact decision makings by mine design engineers. Through a case study of an open pit mine elaborated on in this paper, it was shown that the best possible choice of geostatistical methods for heterogeneity modelling of the geomechanical variables could be the sequential Gaussian simulation (SGS). In this case study of an iron ore open pit mine, the geomechanical variables were modelled in 3D block models using the sequential Gaussian simulation method. The developed models effectively identify the high-risk zones of weak rock mass quality that are susceptible to pit slope instability and require more monitoring efforts during exploitation.

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COMBINING FLEXIBILITY WITH CONVEYOR BASED MINING

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COMBINING FLEXIBILITY WITH CONVEYOR BASED MINING

ABSTRACT

During the recent mining boom, the industry largely followed the traditional path of truck & shovel mining, as it provided a simple and flexible way to expand production rates. Today, most mining companies struggle with excessive operating costs. By contrast, conveyor based haulage has consistently been shown to be more cost effective, in particular as surface mines are getting deeper and with higher stripping ratios or lower grades. In hindsight, many would have been stronger financially today had they invested in a conveyor based mining method. However, there are some challenges to overcome when replacing truck- with conveyor haulage, one being a lesser degree of flexibility. Flexibility comes back to finding a method to load the conveyor, as required by practical mine design. In conventional In-Pit Crushing & Conveying, a mobile or relocatable crushing station is utilized. The former eliminates truck haulage and provides the greater cost savings opportunity, while the latter utilizes short distance truck haulage with more flexibility but at a higher cost. Either method is constrained by the shovel capacity and reach. This paper introduces a different method to load the conveyor using dozer pushing. This must not be confused with the traditional use of dozer pushing, where material is simply pushed into mined out voids. In this method, dozers are used to push blasted material to feed mobile crushers, which in turn load conveyors. Several factors contribute to the flexibility of this method. For starters, there is no technical restriction on bench height. Within financially viable limits, you can vary the bench height to efficiently work undulating ground or dipping seams. If working multiple machines on different levels, you can adjust each bench height to balance capacity and rate of advance. Over time, the number of crushing stations can gradually be increased, rather than facing the full capital investment up front. Unlike shovels, the dozer fleet is highly scalable, whether contracted or owner-operated. Although the crushing stations should preferably be mobile and self-propelled, they are normally only relocated every 10-12 days. This allows for high availability and provides a cushion in the schedule for operations and maintenance. As mine operations change long term, the crushing station is adaptable and can even be reconfigured for alternative uses. Besides introducing the overall concept, the main thrust of this paper is to explain how dozer pushing adds flexibility to In-Pit Crushing & Conveying, for short- as well as long-term planning.

KEYWORDS

In-Pit Crushing and Conveying, IPCC, conveyor based mining, dozer push, mobile crusher, continuous mining, hybrid crusher, flexible mining, PF400

BACKGROUND

The predominant surface mining method in the world is truck & shovel mining. Mining is done on level benches where large shovels load haul trucks, which in turn transport waste to a dump or ore to a crushing plant. Extensive infrastructure is required, e.g., with haulage roads and access ramps, along with equipment like graders and water trucks to ensure access and proper working conditions. Anybody who has worked around mines know that roads and ramps continually change to keep up with the expanding mine. For all the costs associated with this, truck & shovel mining offers one significant advantage: Flexibility! Firstly, the truck fleet can relatively easily be increased or decreased, in particular if contractors are utilized or the mine owner-operator works multiple mines. Secondly, the net capacity of the system is independent of the reliability of individual trucks; if one trucks needs maintenance it can be replaced by another. Thirdly, it gives short-term mine planners a tool to modify the plans and shift capacity from one spot to another. In metals mining, there is a culture of "chasing the next best ore," and as more information becomes available of grades this encourages continuous adjustments to the mine plan. As a matter of fact, mining universities teach students that it maximizes Net Present Value (NPV) to extract the highest grades first. Whether or not this truly maximizes shareholder value and supports long term profitability is a much broader topic, but flexibility is embraced as a way to maximize NPV. In that spirit, the pit is usually designed to reach good ore quickly, after which it may be expanded. Often the consequence is that there are no final pit walls in the early years of operation. Although costly, it is technically not a problem to create new haul roads during new pushbacks. The effect of NPV on mine design for different methods has been well illustrated by Knights & Nehring (2014).

A separate mining method is to use dozer pushing. Traditionally, this is used in surface coal mines at the bottom of the pit. Once the ore (coal) has been extracted, the adjacent waste (overburden or interburden) is simply pushed to the void in order to expose new coal. The dozer push operation is usually preceded by drilling and cast blasting. Where applicable, this method is extremely practical and cost effective. Dozers have proven highly capable of pushing blasted material, and it is fundamentally much cheaper to push for a short distance than to load with a shovel and haul with trucks. This paper does not describe this mining method. However, it does leverage the proven experience and known operating costs of dozer pushing, in order to push to a crushing station rather than a void.

THE CASE FOR CONVEYOR BASED HAULAGE

In-Pit Crushing & Conveying (IPCC) is a general term for a more continuous mining process. A crushing station is located close to the mining face, for the purpose of reducing the material size so that it can be conveyed rather than hauled by trucks. The primary driver behind this is that haulage cost is one of the most significant costs in mining, and conveyors are substantially more economical than trucks. A more correct name would thus be "conveyor based haulage," rather than IPCC, since this is the true purpose of the method. The destination of the conveyor can be a crushing plant or a waste dump, in which case the material is dumped using a large spreader. Fundamentally, there are two types of IPCC systems:

- **Fully mobile**. In this configuration the crusher is mounted on crawler tracks to follow the shovel directly at the mine face. This eliminates the need for haul trucks.
- **Relocatable/fixed**. In this case, shovels load haul trucks, but the trucks only travel a short distance to the crushing station. This reduces the size of the truck fleet, and more so if the crusher is relocated periodically closer to the mine face than if left in a fixed location for life-of-mine. If the crusher is located "in-pit" the walls must accommodate conveyors; if located "ex-pit" the mine design is not affected by the IPCC system.

Which type of system is preferable depends on a host of factors, but clearly there is a trade-off between cost and flexibility. Fully mobile systems have the potential to minimize the cost, whereas the relocatable/fixed configurations are a compromise between cost and flexibility.

The general arguments for IPCC can be summarized as:

1. **Safety**. Safety is orders of magnitude higher for continuous mining equipment than for mobile equipment, as used in truck & shovel mining. Statistics from the Queensland Government (2015) confirm this. Bear in mind the massive number of conveyors around stockyards and processing plants.
2. **Dust**. IPCC generates up to 50% less dust. Turnbull (2012) addressed the environmental aspects.
3. **Water usage**. As well, water usage can be reduced up to 50%.
4. **Noise**. And again, noise can be reduced by up to 50%.
5. **Operating expenses (OPEX)**. OPEX can be reduced by 50-75% per tonne (depending on the type of system and parameters for a specific mine).

So, why is IPCC not the predominant surface mining method? There are many reasons, including the shape of the deposit, life-of-mine, availability of electric power, and concerns about capital expenses (CAPEX). Generally, the up-front CAPEX is higher for an IPCC operation than for truck & shovel. If the life-of-mine exceeds about 15 years, though, the CAPEX is usually comparable. Whether or not IPCC has a strong financial case must be evaluated for each individual mine; there is no general argument that one mining method is better than another or that one concept fits all applications. But even when the quantitative, financial numbers support IPCC, the qualitative, flexibility arguments often lead back to truck & shovel mining. Specifically, the issues pointed to are:

- **Final walls**. Continuous operation makes it hard to relocate ramp conveyors; it is best to install them on a final wall at the start of mining (obviously extending the conveyors with increasing depth). Long-term mine planning is more critical and the plans must be followed.
- **Face length and shape**. For fully mobile IPCC, efficiency improves if the mining face is long and straight. Mining blocks must be extracted in sequence and as-planned; there are few options to chase high grade or delay removal of waste.
- **Capacity**. The conveying system and spreader has a fixed capacity, while shovels and crushing stations have big capacity steps. There are limited opportunities to incrementally increase or decrease capacity over time.
- **Reliability**. As a series system, the reliability of the system is contingent upon good maintenance practices. There simply are not the redundancies built in to a parallel system like truck haulage. It requires a shift in mining culture and mindset, similar to that for a processing plant.

There are different options to address these flexibility concerns, but the bottom line is that it is a trade-off between reducing cost and increasing flexibility. As explained by Oberrauner and Ritter (2013), “IPCC is less flexible than trucks – Plan for it, live with it.” That said, the current woes of the mining industry are rarely attributed to a lack of flexibility, but rather excessive costs. It seems logical that in-pit crushing and conveyor based haulage will play a greater role when the market rebounds. In 2015, Carter summarized it as “mine operators are focusing more on ways to achieve lower cost per ton of material moved and less on production system flexibility.” With the modified dozer pushing method described herein, considerable flexibility can be added, making IPCC less of a compromise for traditional miners.

DOZING TO A MOBILE CRUSHING STATION

In this dozer pushing method, dozers replace the shovels or excavators normally used in IPCC. And rather than pushing to a mined-out void, dozers feed material to a loading point, which is followed by a complete conveying system and a waste spreader. The loading point is a mobile crusher, but modified for this application. Various components of this type of system have been in operation for many years, e.g., at Drummond in Colombia, Yallourn in Australia, and Jacinth-Ambrosia in Australia. What is new, and making this method interesting today, is innovation which combines a number of design features. These features include a hopper designed for loading with a dozer, a crawler mounted platform to provide mobility, and a powerful primary crusher capable of sizing blasted material.

PROCESS DESIGN

Overview

Figure 1 illustrates how an overall process might appear in a coal strip mine. In this example, the mine advances on a single level using one shiftable face conveyor. The principle would be the same for a lower level, each with one or two crushing stations. Drilling and blasting is illustrated on three benches, but could be done as a single step if allowed by the geology. Spreading of waste could be in-pit backfill or at an external dump. The figure shows a secondary continuous system with a Bucket Wheel Excavator (BWE) removing the coal seam. This is not the most common scenario and various types of processes could be used to supplement the dozer push operation, including truck & shovel of coal seams or a dragline for a lower interburden.

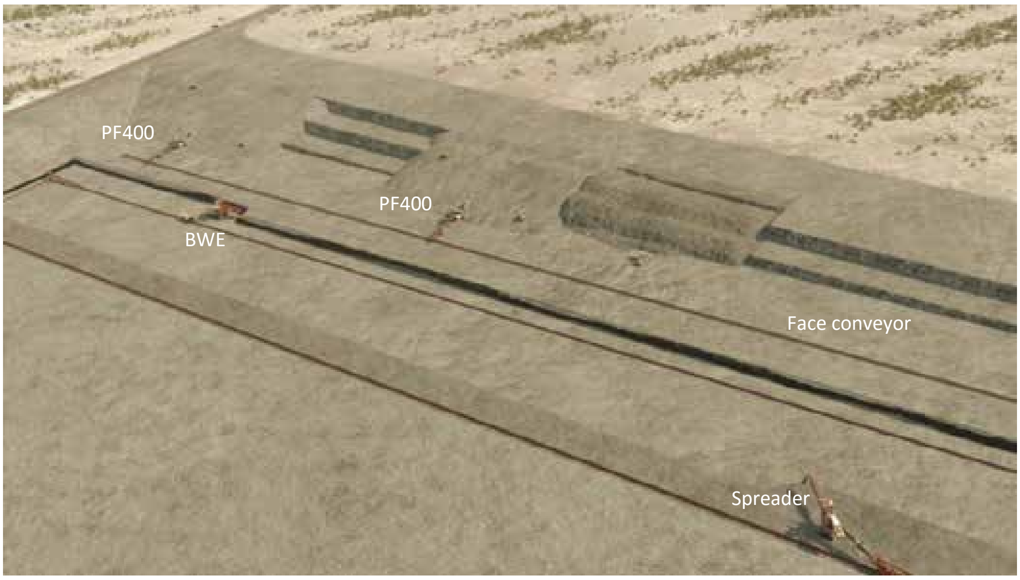

Figure 1 – Example of a mine layout for dozer pushing, using two mobile PF400 crushers, an around-the-pit conveying system, and a spreader for back filling

The dozer push mining method is most simply illustrated for a coal strip mine, but can be applied to hard rock mining as well. It is beyond the scope of this paper to address block size, grade control, etc., but the approach is similar to any fully mobile IPCC system. Functionally, the dozer push system can work with ore as well as waste, with some additional considerations:

- Adapt the block model and Selective Mining Unit for a shape and bench height that can be effectively delivered with dozer pushing.
- Use only one crushing station per face conveyor. Ore and waste cannot yet be conveyed simultaneously.
- If the ratio of waste to ore is high, or stockpiling and blending is necessary, use truck & shovel for ore movements.
- If the ratio of waste to ore is low, build designated ramp conveyors for waste and ore along with adjustable transfer stations inside the pit. Plan the mining sequence to balance loading between the conveyors.

With respect to the overall process, the dozer push method provides flexibility in both mine design and capacity. Unlike shovels, excavators, and BWE's, dozers are not limited to a set bench height. For dipping or undulating seams, the bench height is simply adapted to the geology. There are economic constraints on dozer efficiency, but no technical limit on bench height. Geological strata are rarely perfectly horizontal, with softer and harder layers within the same bench height. An intermediate bench would allow optimized drilling & blasting without the constraint of, e.g., an exact 10 m digging height.

Capacity is somewhat flexible as well. Each crusher can be expected to have a net throughput of about 15 Mt/a, which can be considered medium-size in IPCC. Hence, there is an opportunity to increase the number of stations as production ramps up for a new mine; all CAPEX is not required up-front! An interesting aspect of capacity is the rate of advance if there are multiple levels. By slightly increasing the height of the lower bench, the rates of advance will match. For example, if the face length is 10% shorter on the next lower level, the bench height could be 30 m on the upper- and 33 m on the lower level.

Dozing capacity

The capacity of the actual dozing operation is critical. The driving factors for an individual dozer's capacity are the angle of the push and the distance. The fuel tank limits the maximum short-time working grade to about 33% (1:3), with the most practical average being about 25% (1:4). Operator comfort is also factor, but autonomous dozers already exist. A bench height of 30 m leads to a maximum pushing distance of 120 m, with an actual face advance of 45 m. An overlap of the "fans" results in an effective block volume of about 270,000 m^3 (30 × 45 × 200 m).

Net capacity of the dozers varies over time for each block, but can also be managed by scaling the dozer fleet. There are various ways to manage and optimize dozer production over time, but expect an average of about 1,500 t/h and dozer. The analysis herein is based on utilizing three dozers, and accounting for interference between the units, the net average for the fleet is reduced to 3,000 t/h. For flexibility, the number of dozers can be easily increased or decreased over time to adjust the production. It can even be contracted to avoid CAPEX. It is assumed that only the largest, standard mining dozers are used, e.g., Caterpillar D11 or Komatsu D475. A larger D575 exists, but is custom built.

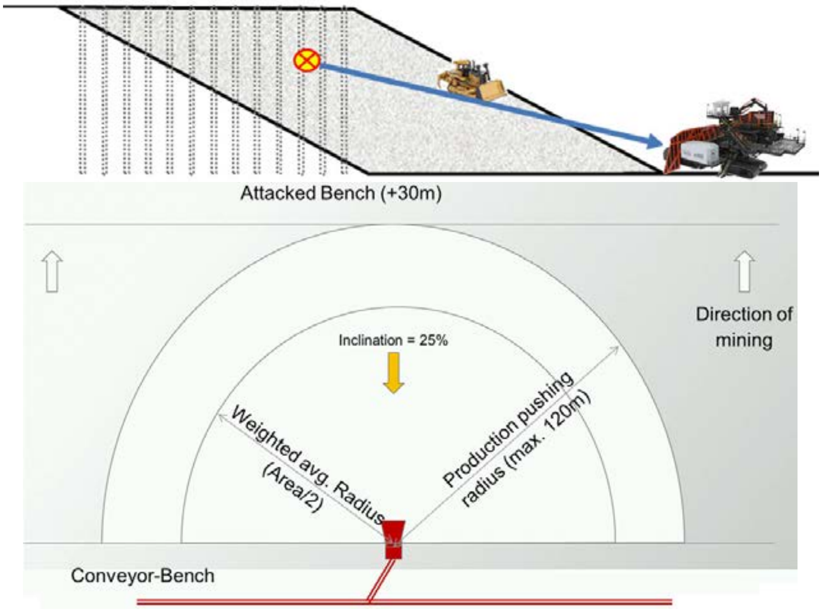

Figure 2 – Geometry of dozer pushing to a crusher at the lower bench

Figure 3 – Dozer pushing onto the crusher's apron feeder, with the material forming a "natural hopper"

Relocations

Based on the 30 m bench height, each fan will be mined out after 10-12 days, and the crusher relocated along the face conveyor. The crusher and link bridge are self-propelled in order to minimize the loss of production. A key step when preparing the new fan is to excavate a pocket into which the PF400 drives. By shortening the average dozing distance by 10 m, dozer productivity increases 10-20%.

If two PF400 operate on the same bench, the relocations can be staggered to smooth capacity of the combined system. Horizontal advance will vary, but assuming that two crushers are used on a 2 km long face, the face conveyor would have to be shifted every 2 months.

Drilling and blasting can take place at another section along the face and not interfere with the excavation process. The link bridge provides enough distance to minimize the risk of damage to the conveyor during blasting.

Figure 4 – Crusher and link bridge prior to relocating to a new fan along the face

Operating hours

This straight-forward mining sequence increases the available operating hours relative to a normal fully mobile IPCC system. The crusher is stationary during operations and the face conveyor is shifted only forward, and only once as the full face advances. Operating time of IPCC systems and the losses due to conveyor shifting have been analyzed by the University of Freiberg (Ritter, R., Herzog, A., & Drebenstedt, C., 2014). The additional operating hours of the dozer pushing method supports flexibility by providing a scheduling cushion for operations and maintenance. All calculations herein are based on 5,000 operating hours per annum, which is conservative and highly attainable.

DESIGN FLEXIBILITY

The crushing station used in dozer pushing may appear to be highly specialized, but it is actually more flexible than any other crusher for IPCC. Even with the best of planning, things change over the life-of-mine, which can be 20 or 30 years. The ideal solution in one 5-year period may not be ideal in another period. The PF400 has a number of design features that make it adaptable and reconfigurable.

Crushing range

The heart of the station is the crusher itself. Because of height and weight limitations, roll crushers are the preferred choice in higher capacity mobile crushers. Sandvik's Hybrid crusher is capable of processing anything from fine, wet, and sticky materials up to 1.2 m blocks of 200 MPa hardness. There is certainly a strong correlation between the blast fragmentation curve and the performance of the crusher, but this style of crusher has the flexibility to cover a wide range of feed material.

Truck feed with final dozer push

During times when it is not practical to install face conveyors, the crusher can be used as a relocatable station. In its simplest form, the machine is just parked on a lower bench wherever it can be connected to a conveyor. No modifications, steel-, or concrete structures are necessary. Short-haul mine trucks of any size are used to dump feed material on the upper bench. A single dozer can easily do the final push into the natural hopper. There is virtually no cost and minimal time to relocate the crusher and keep haulage distance to a minimum.

Truck feed with freestanding hopper

If the crusher is intended for relocatable use for an extended period, it is preferable to install a freestanding hopper with truck bridges. The crushing station is parked underneath this hopper. Since the station already has a shield, no additional retaining walls are necessary. The freestanding hopper is a modular structure that can be moved relatively quickly. Still, with a moderate investment an additional structure can already be setup ahead of time. The crusher itself is mobile and relocates very quickly.

Figure 5 – Truck feed when using the station as a relocatable crusher. Left: Final dozer push. Right: Freestanding hopper

Direct shovel feed

The dozer-push front end is interchangeable with a traditional hopper design. This converts the PF400 to direct feed by a shovel or excavator. This traditional mobile application has its advantages in lower bench heights or when grade control is critical. Multiple shovels and excavators are available to match and fully utilize the 4,500 t/h design capacity of the crusher. Four crawler tracks provide a highly stable platform, completely eliminating the need for support legs. As the shovel dumps into the hopper, the crawler mounted link bridge avoids the oscillations affecting cantilevered discharge conveyors seen on older mobile crushing stations.

Dragline feed

Draglines are commonplace and have a very low OPEX per tonne moved. However, they have limited reach and rehandling is an increasing problem as overburden thickness increases. The PF400 is a cost-effective solution to bring the material onto a conveyor. Spotting is important to avoid damaging the crusher or making an excessively large pile; a software solution already exists from ACARP (Australian Coal Association Research Program). When the dragline walks, a small dozer can support the clean-up and the crusher is ready to move.

Discharge to trucks

In certain situations, conveyors are not accessible from the mining face. This could be during mine development, changes in mining direction, or regular box cuts. The PF400 and the existing dozer fleet can be utilized to load trucks, eliminating the need for additional benches, ramps, and temporary shovels or excavators. If the regular bench height is 30 m, a double-bench of 60 m is entirely possible.

Figure 6 – Alternative usages of the crushing station. Left: Direct shovel feed. Middle: Dragline feed. Right: Discharge to trucks

FINANCIAL EXAMPLES

Whether IPCC with dozer pushing is a suitable and economic method must be assessed in each case. There are no universal truths. The recommended sequence is to first do a desktop study to screen if a mine is a candidate, before proceeding with a full-blown mine planning study or equipment design study. The most common mistake is to first do a truck & shovel mine plan and then try to fit IPCC into that plan.

That said, several studies have indicated substantial savings for IPCC with dozer pushing. Over its lifetime, total costs (CAPEX plus OPEX) have been calculated to be 35-43% lower than the corresponding truck & shovel operation. Specifically, these studies have included removal of overburden, mineral sands mining, and waste dump relocation. Figure 7 is representative of these studies, with numbers specifically chosen for an Indonesian coal mine removing 30 Mt/a of overburden. The relevant truck & shovel scenario uses 100 short-ton haul trucks. Note that the total costs track fairly closely up to Year 7; after the up-front CAPEX is written off the IPCC scenario with dozer pushing has about half the total annual cost. Figure 8 shows the split between different steps in the IPCC process. Even if crushing is an added cost, conveyor haulage is so much cheaper than truck haulage that the net savings is US$0.63 per tonne.

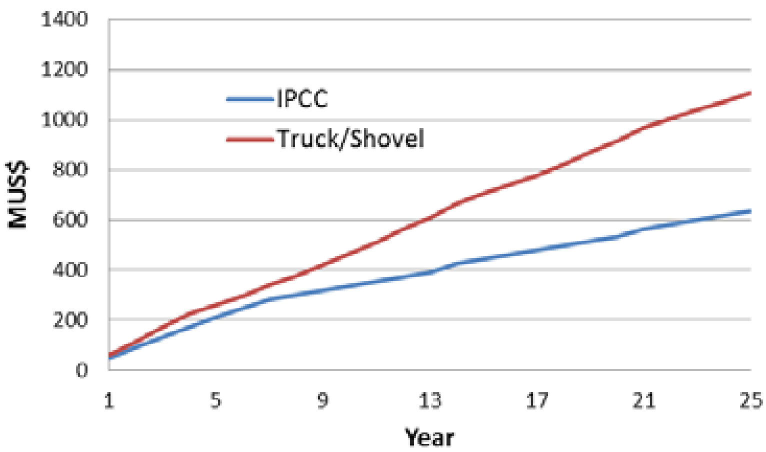

Figure 7 – Total accumulated total cost over life-of-mine, including CAPEX and OPEX, comparing IPCC with dozer pushing and truck & shovel mining

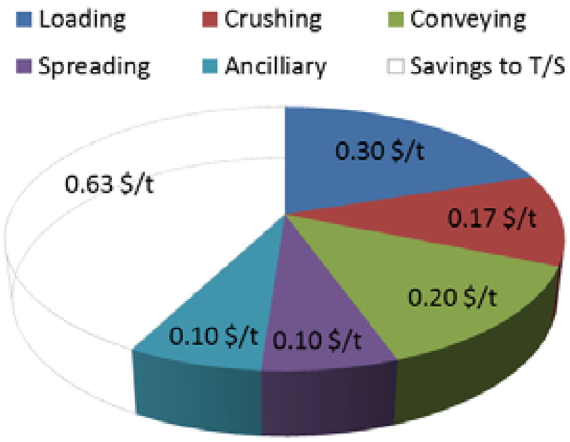

Figure 8 – Total cost for IPCC with dozer pushing, including the saving compared to truck & shovel (T/S)

CONCLUSIONS

It is falsehood to write off IPCC for a lack of flexibility. While truck & shovel mining offers the most flexibility, conveyor based mining with in-pit crushing is not as inflexible as traditionalists believe. In general, it should be approached as a different mine method, and not as an afterthought to a truck & shovel mine plan. Flexibility is commonly interpreted as the ability to continually adjust the mine plan. Trucks can be dispatched to wherever they are needed. For mine planning, there is little need to create final walls, maximize the face length, or to keep them straight. The required capacity to remove waste or overburden changes over time, as does the haulage cycle-time, but the size of the truck fleet can be changed. Reliability of individual trucks is not critical. This type of flexibility can be partially mimicked with short-haul trucks feeding a relocatable crusher, followed by a conveyor system. However, that also compromises the cost advantage of conveyor based haulage.

By shifting the focus to the loading method, this paper explains an alternative approach to combine conveyor based haulage with flexibility. The basis of this method is to use dozers instead of shovels, with profound implications. The flexibility with this method can be divided into three groups:

- **Flexibility compared to truck & shovel.** Capacity for dozers is more flexible than for shovels; the dozer fleet is highly scalable, including the use of contractors. The bench height is adaptable to the geology without the shovel's constraint in vertical reach. A height of about 30 m is optimum, but there are no technical limitations.
- **Flexibility compared to other IPCC systems.** Relocating the mobile crushing station only every 10-12 days increases the annual operating hours and allows for good preventive maintenance. With each crusher processing about 15 Mt/a, capacity can be added over time, avoiding all CAPEX up-front. It is also possible to match the rate-of-advance at different levels with different bench heights. If no face conveyor is accessible, the crusher can temporarily discharge to trucks.
- **Flexibility to change method over life-of-mine.** The physical design of the crushing station allows it to be reconfigured for different mining methods. It can be fed with trucks, either via a hopper or a final dozer push, directly by a shovel, or even a dragline. With a hybrid roll crusher, materials ranging from fine and sticky up to 200 MPa hardness can be processed.

Dozer pushing is a proven method, and the in-pit crushing technologies and references exist. Studies of overburden removal, mineral sands mining, and waste dump relocations show that pushing to mobile crushers reduce total cost by 35-43% compared to truck & shovel mining. By combining cost savings with a good flexibility, dozer pushing promises to increase the use of conveyor based mining.

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COMPARATIVE ANALYSIS OF ECONOMICS, OPERATIONAL AND ENVIRONMENTALS BENEFITS IN THE MINING METHODS TERRACE MINING AND OPEN PIT MINING: GYPSUM MINING CASE

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ABSTRACT

This study aims to demonstrate the operational, economic and environmental benefits occurred with the change from the mining method "Open Pit Mining" to the method "Terrace Mining" in gypsum mining at the enterprise Royal Gipso, in the county of Araripina, Pernambuco. In the Gypsum Pole of Araripe, most of deposits applies the method of mining Open Pit Mining by simple benches, once this method has been shown efficient from an operational point of view. However the purpose of this study is to show the improvements that this new method is providing for the entire gypsum production chain, including the significant environmental benefits that this change has generated. For the analysis of this study it was made a survey of operational data (unit operations, equipment cycle time measurement), geometric (benches and access) and operating costs of the mining methods Open Pit Mining and Mining Terrace. All data that will be presented were obtained through technical visit to Mining Ponta da Serra, located in Araripina. Based on the results of this analysis can be inferred that changing from the Open Pit Mining method to the Terrace Mining one, it indicates a considerable improvement throughout the operating process of the mine, as well as reduces the environmental impacts caused by the mining activity. Therefore we conclude that this new technique that has been applied to the Mining Ponta da Serra is the right path to a sustainable future, both from an operational point of view and the environmental.

KEYWORDS

Gypsum; Gypsum Pole of Araripe; mining methods; Open Pit Mining; Terrace Mining; environment

INTRODUCTION

Gypsum is a mineral formed in sedimentary deposits of chemicals and which is deposited by evaporation and precipitation processes. It has wide application in the construction industry to be the main raw material for the manufacture of plaster. In Brazil, the main gypsum deposits from the economic point of view are in the Sedimentary Basin of Araripe and are located in two Brazilian states: Pernambuco, with the largest representation (Gypsum Pole of Araripe) and Ceará, with less potential for exploitation. The Gypsum Polo (Figure 1) is formed by the cities of Araripina, Bodocó, Ipubi, Ouricurí and Trindade, and the mines that compose it are responsible for about 95% of gypsum produced in Brazil.

In this Polo, the gypsum occurs in Santana Formation, fossiliferous, which is characterized by the presence of lime, clays and shales of reduced thickness. (Menor, 2012). One of the companies belonging to this Polo is the Royal Gypsum Mining Ltda., engaged in the exploitation of industrial mineral gypsum and manufacture of precast plaster, and its headquarters is located in Sítio Lagoa de Dentro in Araripina/PE. The area object of this study is the Mine Ponta da Serra, belonging to the Royal Gypsum, located on the farm Ponta da Serra, district of Gergelim, county of Araripina/PE. The gypsum deposit is located in the central part of the area and covers an area of 23.1 ha. The gypsum that occurs in this region has several allotropic forms, which are popularly called Boró, Cocadinha, Rapadura, Estrelinha, Johnson, and Ore Floor. Samples were collected from several existing varieties (Figure 2).

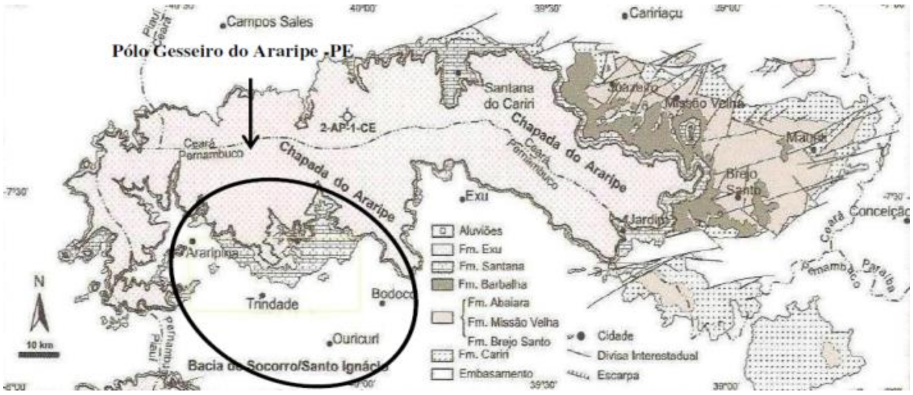

Figure 1 - Regional geology including Araripe Basin and DNPM research area and the municipalities that compose the Gypsum Pole of Araripe

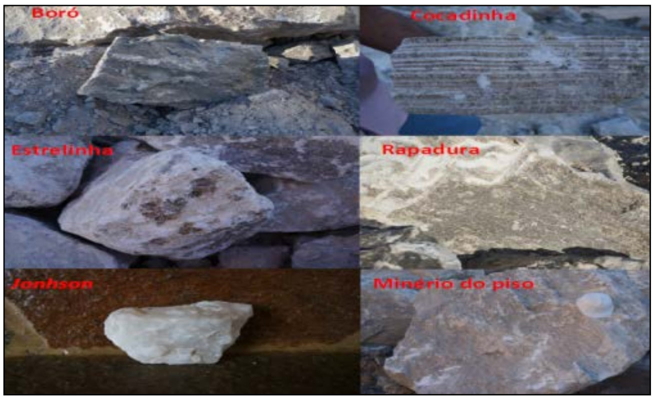

Figure 2 - Mineralogical varieties of gypsum that occur in Gypsum Pole of Araripe

The mines of Gypsum Polo present the best geological and location conditions for extraction of gypsum because the deposits are presented in layers forming tabular deposits. Because of this feature, the mining method used in the region is almost the same for all mines.

Until the year 2012, the Mine Ponta da Serra operated with mining methodology Open Pit – Crater Mining, with simple benches (Figure 3). The overburden-ore ratio was about 0.43 cubic meters of overburden for a ton of gypsum. According to Bastos (2013), the bench height ranged from 10 to 20 meters high, and the overburden layer had an average thickness of 22 meters. The unit operations of mining consisted of stripping, primary disassemble (explosive), secondary fragmentation (hydraulic breakers), loading and transportation. All equipment operating in mining were outsourced and mine production was approximately 12,000 tons per month. Products marketed in mining are classified as ore A, B and C, which have high quality and prices in the market. Most of the sales is the ore type A (Cocadinha, Rapadura, and Estrelinha) with an average price of $ 21/ton, about 80% of production.

However, this mining method caused a big problem regarding the generation of large bulks of overburden (Figure 4) due to the movement of large volumes of land and overburden coverage, which implied the need for acquisition of easement areas disposal of such bulks, becoming this an operational and economic bottleneck for operation of the method since there is limited availability of physical space for send-off. Moreover, it propitiated a major environmental impact generated by these bulks deposited in areas near the pit, and with the depletion of the ore, was forming large craters that generated an environmental liability quite costly, which implied an enormous difficulty to implement an environmental remediation project that met with satisfaction all the requirements of the National environmental Council - CONAMA.

Figure 3 - Mining front of mine Ponta da Serra by Open Pit – Crater Mining method

Figure 4 - Overburden bulk resulting of Open Pit – Crater Mining method

Before that, there was the need for replacement of the mining method Open Pit - Crater Mining for Terrace Mining (figure 5). This method is similar to strip mining to allow the recovery of the area concurrently with mining, with the difference that its operationalization form large terraces to replace the strips, and the deposition of the overburden is held at 180 ° from the mine on local where the ore has been exploited. It is employed in the mining of sedimentary deposits and other mineral deposits that have low cohesion, being applied in the presence of thick covers or when the footwall of the ore has a steep dive. The implementation of Mining Terrace may or may not require the development of multiple stands both for extraction of ore and for the removal of overburden being the number of banks depending on the depth of the excavation (Bullivant, 1987).

In this context, this paper aims to evaluate the application of mining method Terrace Mining, comparing it to the Crater Mining, to optimize the activities of extraction of gypsum in Araripe with respect to a change in the philosophy of exploitation going to adopt methods mining in the form of terraces, much more suited to the exploitation of horizontal deposits of thin layers and depth. Adopting this methodology exploitation, we had the possibility to carry out the filling operations and the disposal of overburden coverage in areas already mined with economic, operational and environmental advantages over Crater Mining.

Figure 6 - Deposition of overburden in the pit – Terrace Mining

METHODOLOGY

After the change of the mining method, a times and movements study was performed to compare the real benefits of this new methodology over the old method, the Crater Mining, which have been translated and interpreted in a plausible way, as the main objective for the continuity of business processes has been reached. This study was conducted through a data collection work in the field for a two-week period in February 2014, which basically consisted of measuring equipment cycle time in the stripping operation of overburden material, which allowed a comparative analysis of operating costs after the change of the method of mining Open Pit - Crater Mining for Mining Terrace. It was used a truck PEX 7060, totaling 30 samples. It is noteworthy that the equipments are all outsourced and the lease of such equipment is the higher cost of generating factor for mining.

Stripping Cycle Time

By the year 2012, mining Royal gypsum used the open pit mining method type Open Pit - Crater Mining with the following cycle of unit operations: stripping (loading and transport of overburden to a send-off outside the mining area); rock blasting (drilling and blasting); secondary fragmentation of the ore; loading of ore; Ore transportation to the crushing or customers.

In the unit operation of stripping were used to transport the overburden: 4 tracings trucks, plates PEX 7060, PEX 7370, PEI 1926, and PEI 1796, with a nominal capacity of 16 m^3, and a hydraulic excavator Hyundai manufacturer, R320LC-7 model, year 2011, with a nominal capacity of 1.73 m^3.

With the implementation of Terrace Mining, the cycle of unit operations ore disassemble remained the same. The differences relate primarily to the manner of execution of the stripping operation and disposal of overburden, especially the overburden transport route in Terrace Mining, which implies the optimization of overburden transport due to the change of the overburden deposition site for a nearest location in front of descobertura, thus allowing reducing the number of trucks, and the possibility of environmental recovery of the mined area simultaneously with the mining.

In the evaluation of the cycle time in the field, through chronometers, were measured: truck loading time in stripping (LT), truck round trip time to the deposition site of overburden (RTT), dumping time (DT), and truck travel time back (TTB). The cycle time stripping of the Crater Mining was obtained through reports assigned by the company, and this time was recorded during the stripping of a front work area headquarters was 710 meters away from the overburden bulk (overburden bulk out the pit). The cycle time of this operation for the Terrace Mining method was obtained through field surveys at stripping to expose the ore bench located 450 meters away from the site of deposition of overburden (waste bulk in the pit). The round trip distance (RTD) and return (RD) of the trucks to the path of overburden bulk were computed through the odometers of trucks. Measurements also possible to estimate the productivity of hydraulic excavator.

RESULTS AND DISCUSSION

Comparison of costs excavator hydraulic

Data that characterize the mine:
- Average thickness of gypsum layer: 11 m;
- Average thickness of the overburden layer: 21 m;
- Coefficient of swelling clay: 0.4;
- Correction factor for the volume of the corresponding landfill = 1 + 0.4 = 1.4
- Specific mass of gypsum in situ: 2,3 t/m^3;
- Angle of repose of natural slope: 75°.

Determination of the technical elements of mining:
- Volume gypsum to blasting monthly: 8695.65 m^3;
- Deposit area to be discovered monthly: 790.51 m^2;
- Seat width to be disassembled: 20.00 m;
- Monthly mining advance: (790.51 m^2) / (20 m) = 39.52 m;
- Volume of overburden in situ, to be dismantled monthly: (790.51 m^2) x (21 m) = 16600.7 m^3;
- Volume of overburden considering the swelling: (16600.7 m^3) x 1.4 = 23240.99 m^3;
- Work regime: 8 hours/day – 25 days/month – 300 days/year.

The volume of overburden to be excavated daily is given by:

$$V = (23240.99\ m^3/month)/(25\ days/month) = 929.64\ m^3/day \tag{1}$$

From the data collection in the field, it was found that the excavator cycle time is, on average, 20.58 s per cycle and 170 caçambadas per hour. So, it was possible to calculate the bucket volume needed according to the volume of overburden:

$$V_C = (116.20\ m^3/h)\ x\ (170\ caçambadas/h) = 0.7\ m^3 \tag{2}$$

According to the equation proposed by Souza (2001), that was determined the average hourly productivity of the hydraulic excavator, obtaining the value P_h = 106.9 m^3/hora. Considering the work regime, it has the following estimated monthly production equipment:

$$P = (106.9\ m^3/h)\ x\ (8\ h/day)\ x\ (25\ days/month) = 21373\ m^3/month \tag{3}$$

Therefore, the required number of excavators for obtaining the required production month is:

$$N_e = (23240.99)/(21373) = 1.09 = 1\ unit \tag{4}$$

The excavator used in Mine Ponta da Serra has 1.7 m^3 of bucket capacity, and in this case was carried out the same previous calculation (hourly output and excavators number) to determine the necessary amount of excavators. Considering the same values of the parameters, except for bucket capacity, we obtained the value 151.4 m^3/h for hourly production, and the value 30278.3 m^3/month for the monthly production. Therefore, the required number of excavators for obtaining the required production month is:

$$N_e = (23240.99)/(30278.3) = 0.77 = 1\ unit \tag{5}$$

The result shows that the excavator that was used in the operation of Crater Mining was overestimated, because the 1.2 m^3 bucket capacity loader is sufficient to supply the production demand. The model used previously was the R330LC-9S Hyundai manufacturer with a bucket 1.73 m^3. The ideal device would be a model of 0.7 m³ bucket capacity, but in the region, which is closest, is Hyundai's manufacturer, R220LC-9 model, 1.2 m^3.

The stripping operation of the mine is outsourced. Thus, the rental cost of the old excavator was R$ 130.00/h, resulting in a monthly cost of R$ 26000.00; diesel oil consumption for this equipment was approximately 25 L/h, and the unit cost of diesel was R$ 2.27. Considering the work regime 200 h/month, the monthly cost of diesel oil for this equipment was R$ 11350.00, totaling an annual cost of R$ 136200.00. With the implementation of Terrace Mining, these costs were significantly reduced, as shown in the following table:

Table 1: Comparison between excavators

Cost data with equipment and Diesel	**R220LC-9 (1.2 m³)**	**R330LC-9S (1.7 m³)**	**Difference between excavators**
Rental cost (R$/month)	20.000,00	26.000,00	6.000,00
Diesel consumption (L/month)	4.000,00	5.000,00	1.000,00
Diesel cost (R$/month)	9.080,00	11.350,00	2.270,00

According to the table data, the operation of 1.2 m^3 excavator provided savings of R$ 72000.00 in the rental of equipment, and R$ 27240.00 in annual cost of diesel, with a total annual savings of R$ 99240.00.

Comparison of costs overburden transport

Open Pit - Crater Mining

The specification of the trucks operating in mining is the Volkswagen manufacturer, model 31320, nominal capacity of 16 m^3. In operation of Crater Mining, were used 4 units operating in outsourced mine. Trucks cycle times were obtained considering an average distance of 710 meters from descobertura bench to the overburden bulk. The excavator used for measurement of time was the R330LC-9S (1.7 m^3). The table 2 shows the data of truck cycle time of stripping operation of Open Pit.

Table 2: Truck cycle time of overburden transport – Open Pit Mining

Samples	**LT**		**RTT**		**RTD (m)**	**TTB**		**RD (m)**	**DT**	
	h:min:sec	**min**	**h:min:sec**	**min**		**h:min:sec**	**min**		**h:min:sec**	**min**
1	00:03:53	3,884	00:04:09	4,15	700	00:02:11	2,183	698	00:01:14	1,23
2	00:03:53	3,884	00:04:09	4,15	700	00:02:11	2,183	700	00:01:15	1,25
3	00:03:55	3,917	00:04:11	4,187	714	00:02:14	2,233	697	00:01:16	1,27
4	00:03:57	3,95	00:04:12	4,2	703	00:02:14	2,233	699	00:01:18	1,3
5	00:03:57	3,95	00:04:12	4,2	710	00:02:15	2,25	701	00:01:18	1,3
6	00:03:57	3,95	00:04:13	4,22	718	00:02:15	2,25	705	00:01:18	1,3
7	00:03:58	3,967	00:04:13	4,22	720	00:02:16	2,267	709	00:01:18	1,3
8	00:03:59	3,984	00:04:13	4,22	707	00:02:17	2,283	706	00:01:18	1,3
9	00:04:01	4,017	00:04:13	4,22	717	00:02:17	2,283	696	00:01:20	1,33
10	00:04:01	4,017	00:04:13	4,22	704	00:02:17	2,283	703	00:01:20	1,33
11	00:04:02	4,034	00:04:13	4,22	700	00:02:17	2,283	702	00:01:21	1,35
12	00:04:02	4,034	00:04:14	4,23	708	00:02:17	2,283	690	00:01:21	1,35
13	00:04:02	4,034	00:04:15	4,25	716	00:02:17	2,283	704	00:01:21	1,35
14	00:04:02	4,034	00:04:15	4,25	712	00:02:18	2,3	691	00:01:21	1,35
15	00:04:02	4,034	00:04:09	4,15	705	00:02:18	2,3	700	00:01:21	1,35
16	00:04:03	4,05	00:04:16	4,27	708	00:02:19	2,317	702	00:01:22	1,37
17	00:04:03	4,05	00:04:16	4,27	720	00:02:19	2,317	709	00:01:22	1,37
18	00:04:03	4,05	00:04:17	4,28	717	00:02:19	2,317	699	00:01:22	1,37
19	00:04:04	4,067	00:04:17	4,28	721	00:02:19	2,317	701	00:01:22	1,37

20	00:04:05	4,084	00:04:17	4,28	715	00:02:19	2,317	707	00:01:23	1,38
21	00:04:05	4,084	00:04:18	4,3	717	00:02:19	2,317	695	00:01:23	1,38
22	00:04:05	4,084	00:04:18	4,3	707	00:02:20	2,333	702	00:01:23	1,38
23	00:04:06	4,1	00:04:18	4,3	704	00:02:20	2,333	696	00:01:24	1,4
24	00:04:06	4,1	00:04:19	4,32	708	00:02:20	2,333	704	00:01:25	1,42
25	00:04:07	4,117	00:04:19	4,32	720	00:02:20	2,333	690	00:01:25	1,42
26	00:04:08	4,134	00:04:20	4,33	718	00:02:21	2,35	698	00:01:25	1,42
27	00:04:08	4,134	00:04:20	4,33	705	00:02:21	2,35	703	00:01:25	1,42
28	00:04:11	4,184	00:04:21	4,35	703	00:02:22	2,367	704	00:01:27	1,45
29	00:04:11	4,184	00:04:22	4,37	700	00:02:22	2,367	704	00:01:27	1,45
30	00:04:12	4,2	00:04:23	4,38	700	00:02:22	2,367	699	00:01:28	1,47
Average	00:04:03	4,04	00:04:16	4,3	709,9	00:02:18	2,3	700,5	00:01:21	1,36

Applied basic concepts of transport scaling to get the cycle time of a dump truck, there was obtained:
LT = 4.04 min
RTT = 4.30 min
DT = 1.36 min
TTB = 2.30 min

Thus, the total time of the truck cycle is 12 minutes. As the material it is a compact clay, the bucket fill factor is 0.8, which allows to calculate the number of cycles to load a truck, obtaining the value of 12 cycles.

With the equation proposed by Pinto (1999), assuming a 80% efficiency for transporting and 70% for loading operation, it was found that the number of trucks needed was 4 trucks. This amount corresponds to the amount of trucks that operated in the mine Ponta da Serra, executing 73 cycles/day.

In Mining, trucks are also outsourced and the freight is $ 18.50 per trip. To attend the required production, the trucks must perform 73 cycles/day. So, considering the amount of $ 18.50, the monthly cost of freight with 1 (one) truck will be R$ 33762.50 and consequently the annual cost is R$ 405150.00.

Terrace Mining

For Terrace Mining method, the cycle times of the trucks have been raised considering an average distance transport 450 meters from the front of work to the place of deposition of overburden, and are shown in the table 3.

Table 3: Truck cycle time of overburden transport – Terrace Mining

Samples	**LT**		**RTT**		**RTD (m)**	**TTB**		**RD (m)**	**DT**	
	h:min:sec	min	h:min:sec	min		h:min:sec	Min		h:min:sec	min
1	00:03:53	3,88	00:01:24	1,4	500	00:00:56	0,93	400	00:01:14	1,23
2	00:03:53	3,88	00:01:17	1,28	500	00:01:01	1,02	400	00:01:15	1,25
3	00:03:55	3,92	00:01:21	1,35	500	00:01:03	1,05	400	00:01:16	1,27
4	00:03:57	3,95	00:01:28	1,47	500	00:01:05	1,08	400	00:01:18	1,3
5	00:03:57	3,95	00:02:00	2	500	00:01:00	1	400	00:01:18	1,3
6	00:03:57	3,95	00:01:00	1	500	00:01:05	1,08	400	00:01:18	1,3

7	00:03:58	3,97	00:02:00	2	500	00:01:00	1	400	00:01:18	1,3
8	00:03:59	3,98	00:01:00	1	500	00:01:00	1	400	00:01:18	1,3
9	00:04:01	4,02	00:01:00	1	500	00:01:03	1,05	400	00:01:20	1,33
10	00:04:01	4,02	00:01:00	1	500	00:01:04	1,07	400	00:01:20	1,33
11	00:04:02	4,03	00:01:00	1	500	00:01:00	1	400	00:01:21	1,35
12	00:04:02	4,03	00:02:00	2	500	00:01:00	1	400	00:01:21	1,35
13	00:04:02	4,03	00:01:00	1	500	00:01:00	1	400	00:01:21	1,35
14	00:04:02	4,03	00:02:00	2	500	00:01:10	1,17	400	00:01:21	1,35
15	00:04:02	4,03	00:01:21	1,35	500	00:01:09	1,15	400	00:01:21	1,35
16	00:04:03	4,05	00:01:00	1	500	00:01:00	1	400	00:01:22	1,37
17	00:04:03	4,05	00:01:03	1,05	500	00:00:59	0,98	400	00:01:22	1,37
18	00:04:03	4,05	00:01:20	1,33	500	00:00:36	0,6	400	00:01:22	1,37
19	00:04:04	4,07	00:01:07	1,12	500	00:01:02	1,03	400	00:01:22	1,37
20	00:04:05	4,08	00:01:09	1,15	500	00:01:01	1,02	400	00:01:23	1,38
21	00:04:05	4,08	00:01:00	1	500	00:01:00	1	400	00:01:23	1,38
22	00:04:05	4,08	00:01:00	1	500	00:01:00	1	400	00:01:23	1,38
23	00:04:06	4,10	00:01:11	1,18	500	00:00:49	0,82	400	00:01:24	1,4
24	00:04:06	4,10	00:02:00	2	500	00:01:00	1	400	00:01:25	1,42
25	00:04:07	4,12	00:01:00	1	500	00:01:06	1,1	400	00:01:25	1,42
26	00:04:08	4,13	00:01:00	1	500	00:01:00	1	400	00:01:25	1,42
27	00:04:08	4,13	00:01:00	1	500	00:01:03	1,05	400	00:01:25	1,42
28	00:04:11	4,18	00:01:00	1	500	00:01:05	1,08	400	00:01:27	1,45
29	00:04:11	4,18	00:02:00	2	500	00:01:04	1,07	400	00:01:27	1,45
30	00:04:12	4,20	00:02:00	2	500	00:00:58	0,97	400	00:01:28	1,47
Average	00:04:03	4,04	00:01:19	1,32	500	00:01:01	1,01	400	00:01:21	1,36

Applied basic concepts of transport scaling to get the cycle time of a dump truck, there was obtained:
LT = 4.04 min
RTT = 1.32 min
DT = 1.36 min
TTB = 1.01 min

Thus, the total time of the truck cycle is 7.7 minutes. As the material it is a compact clay, the bucket fill factor is 0.8, which allows to calculate the number of cycles to load a truck, obtaining the value of 12 cycles. With the equation proposed by Pinto (1999), assuming a 80% efficiency for transporting and 70% for loading operation, it was found that the number of trucks needed was 2 trucks. The number of trips has not changed, since the carrying capacity of the trucks has not changed.

As the trucks are also outsourced and the freight is $ 18.50 per trip to a distance of up to 1000 meters, initially the unit cost of leasing the trucks did not change. However, as the average transport distance was reduced, fuel consumption also decreased, which allowed a renegotiation in the rent. Based on diesel fuel consumption data informed by the an outsourced company responsible for transporting, for the period of 1 month, and considering a distance of 1000 m was made a comparison of consumption for the four trucks (table 4). The average distances were 710 m (Open Pit Mining) and 450 m (Terrace Mining).

Table 4: Diesel consumption comparison of truck

Plates of truck	(Km/L)	***Open Pit Mining* (710 m) R$/trip**	***Open Pit Mining* (710 m) R$/cycle**	***Terrace Mining* (550 m) R$/trip**	***Terrace Mining* (550 m) R$/cycle**
PEI -1796	1.33	1.21	2.42	0.93	1.86
PEI -1926	1.20	1.34	2.69	1.04	2.08
PEX -7060	0.60	2.69	5.37	2.08	4.16
PEX -7370	0.77	2.09	4.18	1.62	3.24
Média	0.98	1.83	3.66	1.41	2.83

* considering the diesel unit value of R$ 2.27

Considering the number of 73 cycles per day multiplied by the cost of travel of the truck applied to both methods of mining, the cost of diesel fuel was R $ 267.18/day for Open Pit method - Crater Mining, while for Terrace Mining this cost was reduced to R$ 206.59. This difference resulted in a 23% reduction in the cost of diesel.

According to information by company INGESEL Ltda., which is responsible for the leasing of trucks, the cost of fuel represents about 49% of the total cost spent on the rental of trucks. Thus, with this cost reduction with diesel, there was a 40% reduction in truck freight, and the rent reduced to R $ 11.10 per trip. Consequently, the monthly cost of shipping 1 (one) truck used in overburden transport decreased to R$ 20,257.50, and the annual cost to R $ 243,090.00, which generated savings of R$ 13,505.00 per month and R$ 162,060.00 per year.

CONCLUSIONS

By replacing the method of mining Open Pit - Crater Mining for Terrace Mining method, the main change occurred in the stripping operation, specifically the deposition of overburden system.

The cost for the acquisition of an area for rental of the send-off was approximately R$ 71,000.00, and stood at a distance of 3.5 kilometers, which increased the unit cost of freight by 60%; therefore, as in the operationalization of the Terrace there is no need to purchase this area, this cost has been eliminated.

In the hydraulic excavator cost comparison, the lifting of descobertura operation showed that the excavator used at the time of operation of the Crater Mining (R330LC-9S, Hyundai) with the bucket capacity of 1.7 m^3 was oversized for operation, since a volume of 0.7 m^3 bucket is enough to meet the required production. Thus, adequate excavator for this operation is the R220LC-9, with bucket of 1.2 m^3, which is the equipment available in the area. The replacement of the excavator at the end of one year enabled a reduction of cost of R$ 99,240.00 in descobertura operation.

When comparing costs of transporting the overburden, the cycle time in the operation of Crater Mining was 12 minutes to an average distance of 710 m transport. In this case, it was needed four trucks of 16 m^3 to meet the required production. With the application of Terrace Mining was reached a total cycle time of 7.7 minute, reducing the overburden transport distance to 450 m. In this case, it takes only 2 trucks of 16 m^3 to meet the required production (40% reduction in truck fleet).

Considering that the value of the freight transport is the same for both methods and the number of trips also does not vary for the two methods (73 trips a day), initially the total cost of transport remained constant (R$ 33,586.75) for both methods. However, due to reduced transport distance to the Terrace Mining method, there was a reduction in consumption with diesel oil about 23%, which resulted in reducing the amount of freight to R $ 11.10. As in the period in which this project was carried out 4 (four) trucks were still in operation, the comparative study showed savings of R$ 54,020.00/month and R$ 648,240.00/year.

The operating cost of mining with the Crater Mining descobertura operation represented 37% of total cost; to the mining method Terrace Mining rose to 36% (with outsourced trucks), with an improvement in the efficiency of equipment and lower environmental impact.

In the environmental viewpoint yielded another important benefit associated with the topographic rehabilitation of the pit, occurring in parallel with the ore exploitation operations, minimizing environmental impact and allowing the future use of the area.

The adequacy of equipment, reducing operating costs, decreased overburden transport distance and the use of the area already mined as a deposit of overburden, it is estimated a reduction of annual expenses of R$ 330,000.00 using the mining method Terrace Mining.

The movement of overburden deposition bulks provides many problems, among which can be highlight: construction and maintenance of overburden dumps (send-off), causing loss of surface area for future use, problems instability and great visual impact on mining.

With this new methodology already implemented in the Mine Ponta da Serra gave a very important step with regard to economic and environmental sustainability for the entire production chain of gypsum, which in the near future will provide a change for the better in all Gypsum Pole of Araripe.

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CURRENT SITUATION OF OPEN PIT MINING OF MONGOLIA AND TECHNICAL INNOVATION TO BE IMPLEMENTED

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CURRENT SITUATION OF OPEN PIT MINING OF MONGOLIA AND TECHNICAL INNOVATION TO BE IMPLEMENTED

ABSTRACT

In respect with the intensifying open pit mining in Mongolia, it applied today, there are enormous technical and technological development of the open pit mining of Mongolia including Erdenet and Oyutolgoi copper mines with the mining capacity of 30-35 million tons ores and 6 coal open pit mines with with annual mining capacity of 5-15 million tons of coal in Tavantolgoi and Gurvantes coal fields, as well as number of small-scale open pit mines for extracting of iron ores, fluorspar, thermal coal and construction materials commissioned in the last 10 years. The utilization of the mining modern equipment such as Liebherr, CAT and Komatsu excavators with 10-54m3 shovels and the surface mining transportation systems with using BelAZ, Komatsu and CAT heavy dump trucks which will result in challenging the mining productivity to the international standards. In the near future, Mongolian open pit mining is expanding with the introduction of the technical innovation coped with the optimal solution for the development of surface mines towards to the reduction of environmental pollution and the high extraction and reducing high level mining costs with increased productivity.

KEYWORDS

Mineral reserves, Mining projects, Mineral extraction, Miming investment, Mineral properties

INTRODUCTION

Mongolian mining industry established and started its activity 94 years ago. In recent decades, significant development in mining sector of Mongolia has occurred by constructing largest open pit mines in relation with mineral price increases at the world market. There are currently 173 open pit mines operating in Mongolia. As categorized by mineral types, there are 80 gold mines, 12 fluorspar mines, 30 construction materials mines, 27 thermal coal mines, 9 coking coal mines, 12 small and large scale iron ore mines, 4 cupper-molybdenum and copper-gold mines (some are under development phase) and one zinc-base metal mine.

In order to make technical and technological assessment for open pit mines, it is grouped by their coal production capacity. Open pit mines can be classified into following 4 groups according to their production capacity:

A. Small scale mines with annual production capacity up to 100 thousand m3 rock mass (150 mines or 86.7% of total mines)
B. Medium scale mines with annual production capacity from 500 thousand m3 to 5 million m3 rock mass (12 mines or 6,9% of total mines)
C. Large scale mines with annual production capacity from 5 million m3 to 15 million m3 rock mass (9 mines or 5,2% of total mines)
D. Extra-large scale mines with annual production capacity over 15 million m3 to 5 million m3 rock mass (2 mines or 1,2% of total mines)

As shown in the below Figure, there are over 100 mln.t of mine production in all open pit mines in Mongolia, of which 80 percent of the total production are referred a dozen of mines categorized under the C and D types of the open pit mines. The exploitation of few number of open pit mines with high production capacity is crucial for the development of the open pit mining in Mongolia.

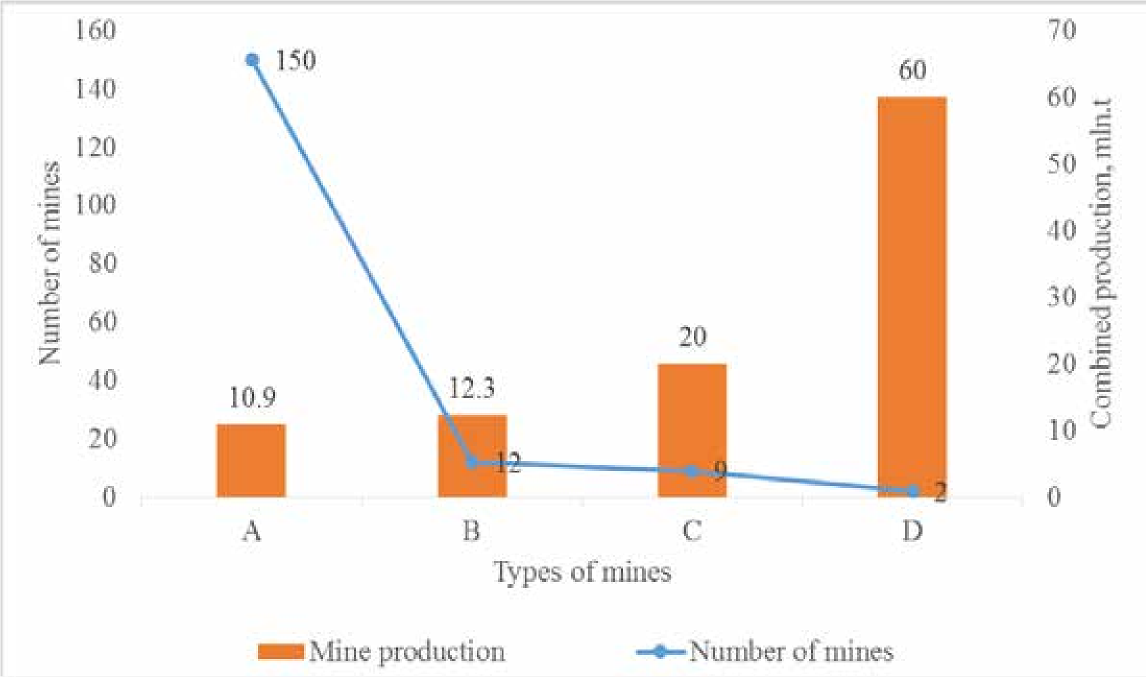

Figure 1 – Mine types and ration on mine production and numbers

Small scale or A type mines include gold mines, operating at small placer gold deposits, all fluorspar and construction materials open pit mines, small iron ore open pits and thermal coal mines, operating for local energy supply.

Medium scale or B type mines can be represented by large and medium scale coal mines, such as Baga nuur, Shivee-Ovoo, Aduunchuluun and iron ore open pit mines, operating at well explored Bayan gol, Tumurtei and Tumur tolgoi iron ore deposits.

Large scale or C type mines comprise of 8 large capacity coking coal open pit mines, all located in Tavan tolgoi and Gurvan tes coal basins in Southern Mongolia. These mines have production capacity to produce 10-15 million tons of coal annually, but production volume by rock mass are less than 10 million m3, due to low stripping ratio.

D type mines with extra large scale include Erdenet copper and molybdenum mine and Oyu Tolgoi ore open pit mine. These open pit mines have annual capacity to mine 30-35 million tons of ore and extract 15-20 million m3 of rock mass including overburden.

Current technological and technical situation and further required renovations for sample open pit mines of each above said group are explained below:

Small scale open pit mines mainly use hydraulic excavators with shovel capacity of 1-5 m3 for the excavation. These type excavators become more popular since 2000's and light hydraulic backhoes allow to make excavation from the upper level to lower benches. Some fluorspar and construction materials open pits still use limited number of 5-cubic meters shovels. These machines also are used at some fluorspar, placer gold, construction materials and iron ore mines. Transportation at the mines is being performed by 10-15 tons ordinary dump trucks. These kind of equipment are also used at small-scale local coal mines. All small mines, except placer gold mines use mining system with transportation of overburden materials to external stockpile. Hard rock materials at mines are prepared by drilling and blasting, mostly using small (100-120 mm) diameter blast holes. Drilling in soft rocks is done by rotary drill rigs and in hard rocks – by drill rigs with DTH hammers. Excavators and loaders of Hyundai, Komatsu, CAT, and Liebherr and dump trucks produced in Korea and China are being used in the small scale open pit mines. For auxiliary mining works, CAT and Komatsu bulldozers are used.

Technologies and equipment used for small scale open pit mines are similar to those in the worldwide open pit mines in respect with productivity and mining parameters. The introduction of the specific equipment and technologies which are suitable for the particular geological and mining conditions would reflect to the increase the mining productivity and decrease the mining cost. For instance, it's recommended to use draglines for overburden in placer gold mines and higher capacity dump trucks for technological transportation at mines.

Therefore, high operating cost in small open pit mines does not allow to offset the cost of efficiency and ecological costs for rehabilitation. Due to the unqualified land management at small mines, the ratio of the destructed land per unit of mining area has been increased.

Medium scale open pit mines have rather high efficiency and mining performance indices, because of using higher capacity modern mining equipment. These mines mostly use 5-20 cubic meters shovels and draglines in combination with 40-75 tons dump trucks for transportation. Larger coal and iron ore mines and limestone quarries belong to this group, for example Baga nuur and Shivee-Ovoo thermal coal mines in central region with annual production capacity of 2 – 5 million tons of coal. Stipping ratio at these open pit mines is averaged to 4,5-6m3/t and angle of coal seam dip -15-200 . Mine operation is started from coal seam outcrop, therefore in further mine-geological condition may be complicated, increasing volume of overburden removal. The mines commonly use shovel/truck mining system both for coal production and overburden removal. Run-off-mine coal, mined by shovel/truck system is crushed at CHP up to 100 mm size and supplied to power stations of Ulaanbaatar and other cities by railway. Coal, sized up 300 mm and over is supplied to other customers by auto transport.

Hydraulic backhoes also are widely used for coal production. Direct casting system with 15-25m3 walking draglines is used for overburden removal and creation of the internal stockpiles. Smaller sized draglines can be used for the management of stockpile of overburden materials. Hydraulic backhoes also are widely used for coal production at other mines.

The existing problem of these medium scale mines is referred to the overburden lag behind in mining technology due to increased volume of overburden materials, long distance transportation and low productivity and efficiency of hauling equipment. The truck transportation is high costly compared with the railway transportation, used to in the past there was used considered as more suitable for horizontal coal deposits. Another major problem for the operation of the state owned coal mines is financial constraint, associated with low coal pricing (12-14 US$/ton), regulated by the Government, finally that hinders adequate technical renovation and environmental protection at mines. The urgent renovation action for the thermal coal mines is to be aimed to introducing fewer, more efficient, high productivity equipment and systems to eliminate lags in overburden removal. In connection with this, continuous mining technologies, including bucket wheel excavators and conveyor transporting system could be investigated and introduced in the near future.

Large scale open pit mines include coking coal mines and copper open pit mines which are newly established in the recent decade. New coking coal mines with the annual production capacity of 5-15 million tons operate at Tavantolgoi and Gurvantes coking coal basins in the southern border area of Mongolia. Distance between the basins is over 400 km. The mining operation have been started in 2010 followed by the intensiveness of the domestic and foreign investment coped with a sharp rebound in prices of coking coal at the world market. Table 1 shows production capacity and capacity utilization of export oriented coking coal mines in recent years.

Table 1 – Production capacity and capacity utilization (%) of new coking coal mines

No.	Name of open pit mines and company	Production capacity, mln.t	Operating years				
			2010	2011	2012	2013	2014
a.	*Tavantolgoi basin*						
1.	East Tsanhi */Erdenes Tavan Tolgoi State-owned company/*	15,0	-	0,9 6%	2,5 17%	2,8 19%	0,9 6%
2.	West Tsanhi */Erdenes Tavan Tolgoi State owned company /*	20,0	-	-	-	-	3,6 18%
3.	Tavantolgoi */Tavantolgoi JSC/*	10,0 mln.t	5,2 50%	6,2 62%	2,3 23%	2,5 25%	0,6 6%
4.	Ukhaa Khudag */Energy Resource LLC/*	15,0 mln.t	3,9 26%	7,1 47%	9,2 61%	7,8 52%	4,6 31%
b.	*Gurvantes area*						
1.	Nariin sukhait 1 */MAK LLC/*	14,0	5,0 36%	5,3 38%	5,4 39%	2,4 17%	4,0 29%
2.	Nariin sukhait 2 */Qinhua-MAK-Nariin sukhait JVC/*	2,0	1,8 90%	1,7 85%	1,6 80%	0,5 25%	0,5 25%
3.	Ovoot tolgoi */South Gobi Sands LLC/*	8,0	2,5 31%	4,6 56%	1,3 16%	1,9 24%	1,6 20%
4.	Khuren shand */Usukh zoos LLC/*	2,0	-	-	-	-	0,8 40%

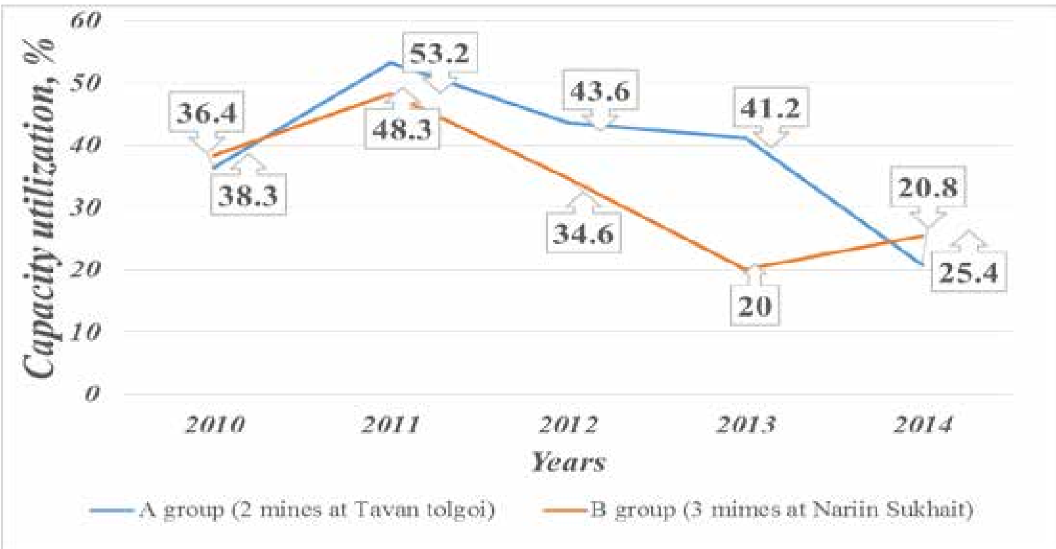

Figure 2 – Capacity utilization of open pit coal mines

As shown in the Figure 2, the capacity utilization of mines is below 50%, accordingly low investment efficiency and utilization of mining equipment productivity. On the other hand, the capacity utilization of mines is highly relied on coking coal demand and prices at the Chinese market.

These open mines use identically same mining technology and equipment for both coal production and overburden with the conventional shovel/truck mining system.

Due to the recent development of the new mines in early phase, the stripping ratios for initial years are not so high (3.7-5.6 m3/t). Overburden materials are transported to the external waste stockpiles, because lower coal seams will be mined in further. For this reason, the construction of the internal stockpiles is not usable in the short term.

As an example, the production situation and equipment fleet of Ukhaa khudag open pit coal mine are described below. Backhoe type hydraulic excavators of various capacity are commonly used at the mine, depending on structure of coal seam and properties of rock mass. Overburden stripping is carried out by Liebherr 996 excavators with 33 m3 bucket (4 units), Liebherr 9400 with 22 m3 bucket (2 units) and Hitachi 3600 with 22 m3 bucket (3 units). Three units of Liebherr 9250 hydraulic excavators with 15 m3 bucket are used for coal production. Two units of Hitachi 1200 excavators with 6.7 m3 bucket are on standby and partially used for coal production.

Transportation equipment fleet of Ukhaa Khudag mine consists of 105 heavy duty Caterpillar dump trucks of various loading capacity: CAT773 (56 tons), CAT777 (93 tons), CAT785 (134 tons) and CAT793 (227 tons). Dump trucks of higher loading capacity (CAT793 and CAT785) are used mostly for transportation of overburden materials and smaller dump trucks (CAT 773 and CAT777) - for coal transportation. Average distance for overburden transportation is 4.6 km and for coal transportation - 6.1 km.

Currently, 3 units of Sandvik D45KS (229 mm), 2 units of CAT MD6290 (229 mm), and 2 units of Sandvik DP1500 are used for drilling blast holes. External transportation of coal from mine site to border ports carried out by Chinese double trailer trucks.

As mentioned, production capacity utilization of coking coal mines is lower, accordingly equipment utilization is also low. In the meantime, only use of high cost truck transportation at open pits increases overall mining cost, thus reducing competitiveness of Mongolian coking coal at the market.

Simultaneous operation of many open pits in adjacent licensed areas of Tavan tolgoi and Gurvantes basins require certain coordination. These issues include creating of unified external transportation system, some infrastructure facilities and networks, overall management of land planning and environmental protection, including waste stockpile locations, possible lease and use of mining equipment, maintenance workshops, joint planning and introducing new technologies, for example cyclic-continuous and continuous technologies etc.

Extra large scale open pit mines include currently operating Oyu Tolgoi and Erdenet copper open mines and Tsagaan Suvarga mine, which is under construction.

Oyu Tolgoi open pit mine started operation in 2011 and reached to its full capacity in 2014. Daily production capacity of the mine is 95-100 thousand tons of ore and annual capacity is to be 35-48 million tons of ore. Life of the open pit mine, operating at copper porphyry type deposit is estimated to 30 years.

The technology and equipment, used at Oyu tolgoi open pit is similar to the Erdenet copper open pit, which uses transport mining system with transportation of overburden materials to the external waste stockpile. Both mines use high productivity, modern mining equipment. For example, overburden materials are excavated by electric shovels with 54 m3 bucket capacity, ore mining is performed by hydraulic excavators with 34 m3 bucket capacity, ore and overburden transportation – by Komatsu 930E dump trucks of 290 tons loading capacity. Atlas Copco PV 351D and 351E drill rigs are drilling blast holes with 320 mm diameter.

Stripping ratio of the open it is 1.5-2.4 m3/t according to mine design. Depth of open pit is increasing quickly due to steep inclination of ore body and reached to 180 meters for recent 3 years.

At present, the open pit has particular geo mechanical problem, for example some parts of pit wall side are collapsing due to cracks and fractures that requires detailed geo mechanical investigations and study of influences on wall stability from blasting operations. There is arisen need for investigating and introducing conveyor systems and special lifting equipment to transport ore from lower levels to the surface in connection with deepening mine. It will facilitate to cost reducing and keeping smooth mine production capacity.

One of extra large scale mines is Erdenet open pit mine, which producing 30 million tons of ore per annum. The mine has been in operation for cumulative 38 years since 1978. The open pit mine occupies 1.5 x 2.5 km area in the northwest part of the deposit. The expansion scheme of the open pit mine has started in 2014 in the central part of the deposit, southeast of existing open pit that will facilitate in increasing of ore reserves and extension of open pit life.

Erdenet ore open pit mine uses conventional shovel/truck mining system with transportation of overburden materials to the external waste stockpiles. Excavation equipment fleet of the open pit consists of Russian made electrical shovels EKG-10 and EKG-15 (8 units) with 10-15m3 bucket capacity and Liebbher excavators with 18m3 bucket capacity (3 units). For transportation of ore and overburden materials, there used BELAZ-75131 dump trucks (26 units) of 130 tons loading capacity. Drilling of 250mm blast holes is carried out by 6 units of Russian made drill rigs SBSh-250. Current open pit mine depth is 165 m, but researchers consider that mine depth may reach to 240 meters and more, because in further the mine is to be concentrated on stockwork sulphide type deep ore deposit.[3] Table 2 shows Erdenet ore mine depth, expansion parameters of mine take off area, waste stockpile and other operation indices.

Table 2 – Some major parameters of Erdenet copper open pit mine

No.	Indicators	unit	2000	2005	2010	2015	Remarks
1	Mine depth	m	60	105	135	165	
2	Mine area	ha	295	332	343	388	
3	Area of external stockpiles	ha	264	312	535	644	
4	Average transportation distance	km	2.1	2.6	3.3	2.9	Average of ore and overburden transportation
5	Rock mass excavation	mln.m^3	15.0	18.6	16.4	21.6	
6	Overburden removal	mln.m^3	6.0	7.9	5.6	9.2	Planning is not smooth
7	Ore production	mln. t	23.0	27.5	27.5	31.2	
8	Stripping ratio	m^3/t	0.3	0.29	0.2	0.29	

Erdenet open pit mine uses rather large number of medium sized mining equipment produced in Russia, which has of low capacity compared with the production capacity, however, the equipment fleet has lower price and maintenance cost, providing lower production cost and overall cost effectiveness.

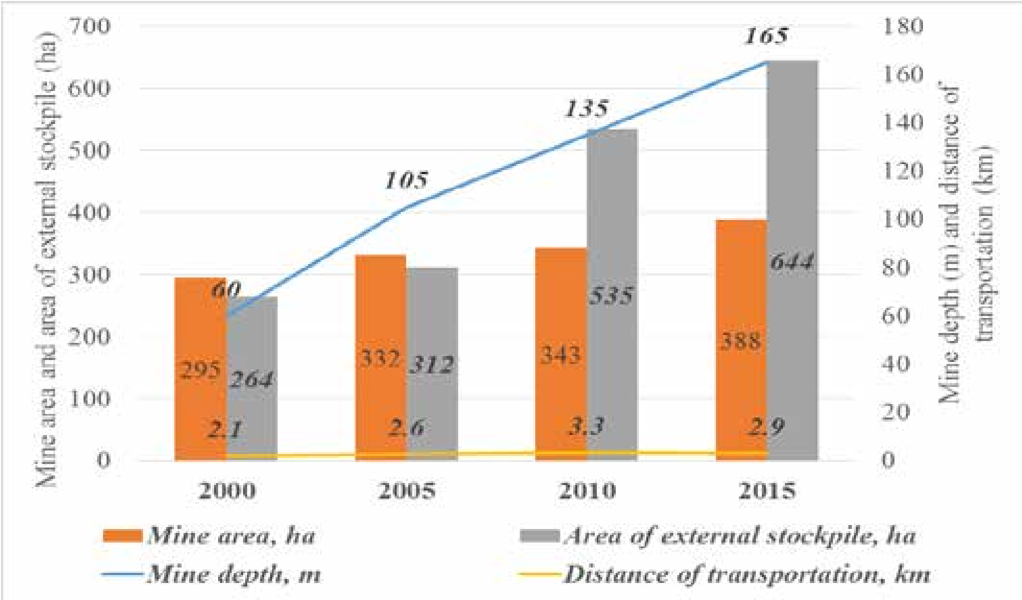

Figure 3 – Mining parameters of open pit mine of Erdenet Mining Corporation (as of example D category open pit mines)

Consequently, it is required to introducing conveyor transportation and bunker system to lift up ore materials to ground level from lower horizons, in terms of increasing transportation distance and inclination, also mine production capacity up to 35-40 million tons per year. In this regard, some researches [4] are being conducted presently.

CONCLUSIONS AND SUGGESTIONS

1. Open pit mining method is dominated in the mining sector of Mongolia, currently it accounts for 99.7% of the total mining and open pit mining is increasing steadily in coming years. This paper assessed current situation and describes possible directions of technical and technological development at open pit mines of Mongolia.

2. All open pit mines were classified into 4 groups, according to their production capacity, analyzed by their current technological level and identified possibilities towards to the further improvement.

3. Specialized mining equipment need to be widely introduced at small placer gold, primary gold ore and coal open pit mines. The environmental contamination should be reduced through the implementation of proper technical renovation and land management with good practices.

4. Medium scale open pit mines include major thermal coal mines, which have of the important contribution to energy supply of the country. But these open pit mines experience several problems, such as financial constraint, insufficient technical renovation and significant lags in respect with the overburden stripping. In order to eliminate lags of overburden removal, its required to conduct mining operation as of perpendicular direction to the deposit strike,introduce conveyor and railway systems for transportation of overburden materials, where possible, increase equipment utilization and efficiency.

5. Newly established large scale coking coal open pit mines are concentrated in two main basins of Southern Mongolia, that requires effective coordination for infrastructure development, including external transport system, land use and management, environmental protection, equipment utilization etc.

High attention needed to pay for increasing equipment utilization and efficiency, reducing production cost, introducing new kind of cyclic-continuous and continuous mining technologies.

6. Technical and technological situation of Oyu Tolgoi and Erdenet extra-large scale open pit mines was analyzed and reviewed. Priority direction of technical renovation at these open pit mines will be focused on introducing conveyor and bunker system to transport ore materials to the surface from lower horizons, in connection with deepening pit operations

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DILUTION CAUSED BY MINE EXECUTION INEFFICIENCY IN OPEN PIT MINING

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DILUTION CAUSED BY MINE EXECUTION INEFFICIENCY IN OPEN PIT MINING

ABSTRACT

Developing a good mine plan is not an easy work; to do it correctly, all particularities, deposit characteristics and operational conditions must be considered. Among these characteristics, dilution and loss can be cited as factors that sometimes are neglected in mining operations or it is simply assumed a correction factor due to the difficulty to be implemented. Geological intricacies, geometrical aspects and operational characteristics, like experience of the operator and use of technologies will directly influence the occurrence of dilution. Dilution and loss are factors that should be seriously considered for tonnage and grade estimates during the reserves assessment procedures. In this study it was determined a methodology to quantify dilution during short term planning in open pit mining. The proposed technique consists in identifying the blocks belonging to periodical plans and its neighborhood considering the nature of the contacts and the differences in grades between adjacent blocks. With this method, it is possible to estimate the internal dilution block by block caused by operational inefficiency and also the dilution caused in the contact slopes at the end of the excavated dig line. The technique takes into account the equipment's inability to perfectly remove each block (respecting excavation limits), and the influence of bench slope angle when mining the boundary blocks. The obtained results demonstrate the need of systematically quantifying dilution, since the quantities of material incorporated or left behind might be significant and could have a great impact on tonnage and grades reconciliation between planning and production.

KEYWORDS

Mine planning, dilution, grade control.

INTRODUCTION

Dilution is defined as waste material that is taken during the ore extraction process, but there are multiple assumptions involved within this definition. It assumes that the operation knows with fair precision the location, shape and grade of a given ore block, and that the equipment operators are capable of mining at a similar resolution (Crawford, 2004). However, there are several sources of uncertainty about the accuracy of the information above such as: the geological model may contain misinterpretations; block grades are based on estimates of spatially scattered data, and the operator may not have the ability to visually select the materials in the field.

According to Sinclair and Blackwell (2002), dilution can be subdivided in two categories: internal and external to the ore. Internal dilution can be considered from the perspective of volumes of waste rock within an ore zone or the inherent diluting effect resulting from either increasing the size of SMUs or the effect of blocks misclassification from sampling and analytical errors during grade control. External dilution can be related to minimum mining width, contact dilution and over break of slopes relative to planned mining boundaries. In addition, Villaescusa (1998) defines loss as the economical material that is not mined due to geological aspects and operating conditions.

This work focuses in quantifying mining dilution in open pit mining. Usually, dilution is better controlled in underground operations than open pit mining due to the selectivity of the underground methods, where this parameter has a higher influence in the operation. Dilution is defined as the incorporation of waste material to ore due to operational inefficiency to separate materials during mining process, considering physical processes, operating and geometric configurations of the mining with the available equipment (Câmara, 2013).

Ore losses and dilution occur during all stages of mining and, while several models are capable to investigate the influence of dilution, its quantification is the greatest challenge (Pakalnis et al., 1995).

However, according to Butcher (2000), dilution can be controlled at acceptable levels, through the implementation of correct mining engineering principles.

METHODOLOGY

The main idea of this work is to characterize each block and its neighborhood contained in a certain set of polygons representing the mine plan (Figure 1). Figure 1 is a typical plan view of a deposit where it can be observed the model represented by the colored blocks in the background representing different cut off grades that occur in the ore deposit, according to the legend provided. In yellow, there are polygons representing the mining areas planned for the period and in green, the pit limit.

Through a routine implemented using JavaScript and an HTML interface running inside DATAMINE Studio 3, it is possible to estimate the grade dilution based on differences between planned blocks and their adjacencies and define the mined percentage of those contiguous blocks. The algorithm calculates the internal dilution of the planned blocks (only when there are waste blocks, or low grade ore, adjacent to an ore block) as well as the influence of slope angle in adjacent blocks to the mine plan in open pit mining. The excavation imperfection (in this case it is incorporated as dilution) can be chosen according to other factors already estimated, especially the selectivity provided by the mining equipment and operator's skill.

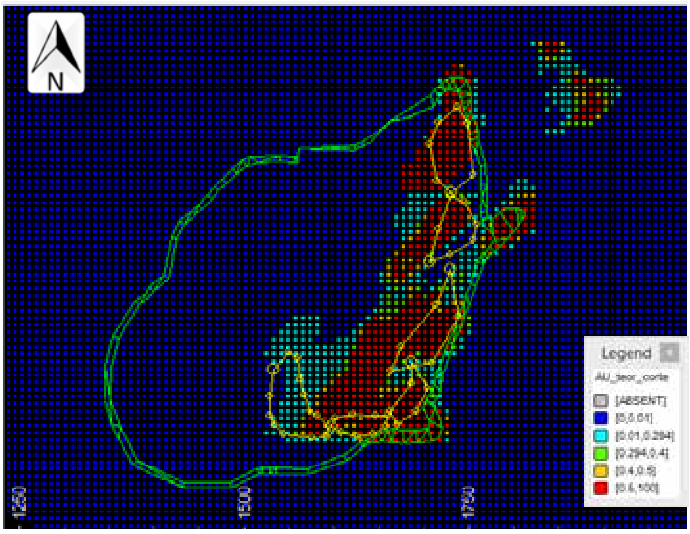

Figure 1- Planning polygons superposed to the block model

To apply the methodology, the first step is identifying the blocks located inside the planning polygons, with their respective grades. These polygons are located in a given reference level, normally associated with the base of the blocks (or bench floor). The polygons are then projected to a distance equivalent to the bench height, depending on the operation approach. Linking the bottom and top polygons now defines a solid representing the volume to be removed from the bench(or portion) and weight averaged considering its masses, allowing one to calculate the value of the interest variables that are expected to be accomplished according to the targets defined by the mine planning.

After creating the solids from the 2D planning polygons, it is possible to identify the blocks that are contained within these solids and the blocks that are in contact with the boundary blocks. All these steps are executed in the algorithm, summarized in the script interface shown in Figure 2.

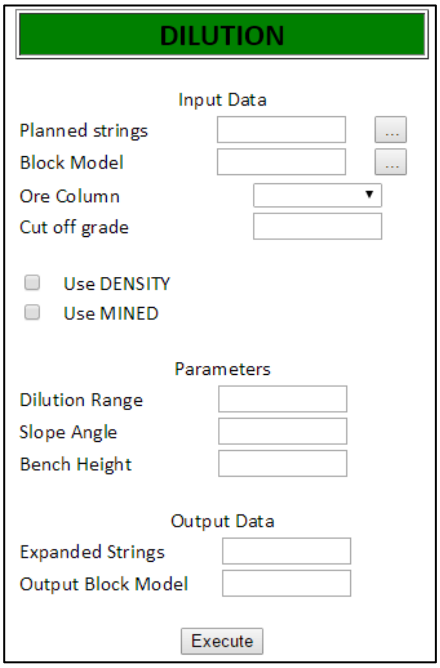

Figure 2- HTML interface of the dilution script

Through commands that transform the polygons in solids according to the bench height used in the operation, and after selecting the blocks contained within these polygons, it is possible to calculate the internal dilution as well the contact dilution and losses of blocks located in the boundaries exposed to the influence of the slope angle.

The major advantage of this script is the speed execution of the processes involved in the dilution calculation and its automaticity in use. The user needs to insert as input data the strings corresponding to planned polygons, the block model and set operating conditions: bench height, slope angle and dilution range (which is the imprecision, in meters, of the equipment when mining each block). With these data, the script identifies the blocks that lie within the planned polygons, the nature of the contact blocks and calculates the dilution for the period.

The script also calculates the percentage of the block within the planned polygon, in order to be more precise about the results of total tonnage found. Using this percentage is a user's choice, checking the "Mined" checkbox. The "Density" is also a user´s choice, however it is recommended that whenever the block model has this column it should be used, thus it will be calculated the actual tonnage and not the volume.

Quantifying Dilution

Two types of dilution and loss are calculated and reported:

- Internal dilution, (planned ore block – planned non-ore block), which will take into account the dilution range chosen by user (Figure 3);
- External dilution, planned ore blocks that have waste external contact block in its neighborhood. From the slope angle and the block size chosen is calculated the contact block mass´ that are incorporated to the planned ore block (Figure 4).
- Loss, planned block portion that are not mined due to the influence of slope angle. It is always calculated together with the external dilution.

The dilution per block is calculated in two different ways:

1. For ore blocks that have internal contact with waste blocks (equation 1):

$$\text{Diluted Grade} = \frac{((\text{M1 x Mined1 x G1}) + (\text{M2 x Mined2 x G2}))}{((\text{M1 x Mined1}) + (\text{M2 x Mined2}))} \qquad (1)$$

Where,

- M1 is the ore block mass, calculated by:
  Block mass = XINC x YINC x ZINC x Density

  Where,

- XINC, YINC and ZINC are the block dimensions in X, Y and Z directions respectively;
- Mined1 is the percentage of block within the polygon;
- G1 is the block grade;
- M2 is the mass of waste block adjacent to be added, calculated by:
  Adjacent block mass = Dilution range x YINC x ZINC x Adjacent Density
- Mined2 is the percentage of adjacent block within the polygon;
- G2 is the adjacent block grade (waste or low grade).

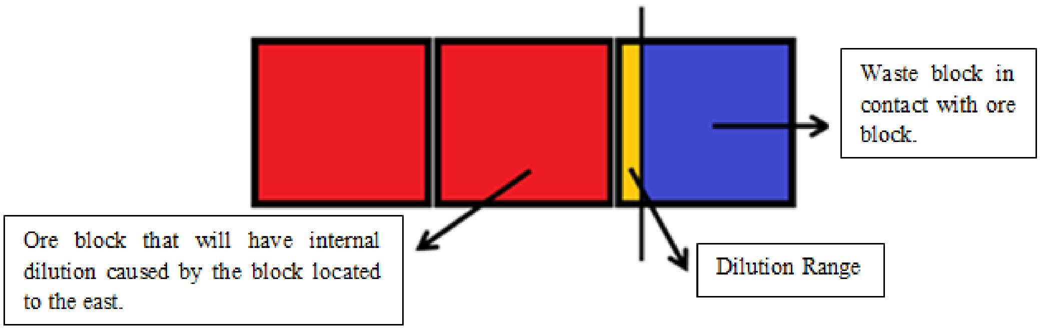

Figure 3 - Internal dilution

2. For ore blocks that have external contact with waste blocks (slope angle) (Equations 2 to 6):

$$\text{Mass of Loss} = \frac{(\text{Influence of slope angle x YINC x } \left(\frac{\text{ZINC}}{2}\right) \text{ x Density})}{2} \qquad (2)$$

$$\text{Mass of Dilution} = \frac{\left(\text{Influence of slope angle x YINC x } \left(\frac{\text{ZINC}}{2}\right) \text{ x Adjacent Density}\right)}{2} \qquad (3)$$

$$\text{Loss} = (\text{Mass of Loss x Grade}) \qquad (4)$$

$$\text{Dilution} = (\text{Mass of Dilution x Adjacent Grade}) \qquad (5)$$

$$\text{Diluted grade} = \frac{(\text{M1 x G1})+\text{Dilution-Loss}}{(\text{M1})+(\text{Mass of Dilution})-(\text{Mass of Loss})} \qquad (6)$$

Where,

- M1 is the ore block mass, calculated by:
  Block mass = XINC x YINC x ZINC x Density
- G1 is the block grade;
- Influence of slope angle is the triangular prism volume formed by the slope angle of the block that is mined.

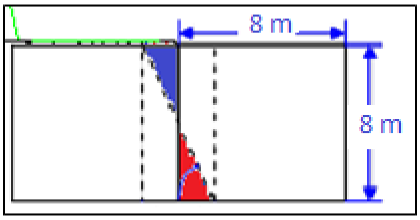

Figure 4 - External dilution, caused by slope angle influence

When a block model is estimated, usually the result is a set of prismatic blocks with X, Y and Z coordinates and orthogonal faces, without any kind of inclination. However, when the pit is designed with operational conditions and rock stability the slope angle should be considered.

If blocks contact is ore/waste, the dilution will correspond to the triangular prism volume formed by the slope angle configuration. In this study, it was considered a 60° slope angle and block dimensions of 8m in all directions. It is also considered that the bench face crosses the middle of the bench height. Using trigonometry (Equation 7) it is possible to calculate the amount of the block that is not mined (loss) and also the tonnage incorporated from the contiguous block (dilution). Both, loss and dilution affect the average grade of the block, which, at the end of the process, will affect the reconciliation in tonnage and grades.

$$\tan 60° = \frac{8}{x} \qquad x = \frac{8}{\tan 60°} \qquad x=4.61 \qquad \frac{x}{2} = \frac{4.61}{2} \qquad \frac{x}{2}=2.3 \qquad (7)$$

To illustrate how much 2.3 meters influence in mining, the volume of the triangular prism formed by the bench´s slope angle is calculated below (Equation 8):

$$\text{Influence of slope angle}= \left(\frac{\text{ZINC}}{2}\right) \times 2.3 \times \text{YINC} \times \text{Density}$$

$$\text{Influence of slope angle}= \left(\frac{8}{2}\right) \times 2.3 \times 8 \times 2.7$$

$$\text{Influence of slope angle} \approx 200\ \text{m}^3 \qquad (8)$$

This volume corresponds to 14% of the total mass of a block with dimensions of 8m, 8m by 8m in the x, y, and z, respectively, which represents a great influence. The product of the mass by the ore grade allows to quantify loss and multiplying by the waste grade (contact block) the dilution can be calculated.

To calculate the dilution of ore blocks that have external contact with waste blocks (slope), the block portion inside the planned polygon ("Mined") was not used, because, in this example, it is assumed that it is in full contact with the waste block.

The methodology proposes to calculate dilution for two types of contact: internal, when ore blocks have contact with non-ore blocks; and external, when blocks have external contact with non-ore blocks. In this study, the calculation of the diluted grade is given by the sum of the two types of dilution shown by the equation 9:

$$\text{Diluted grade} = \frac{\sum(\text{Masses x Grades})}{\sum(\text{Masses})} \tag{9}$$

RESULTS AND DISCUSSION

To validate the methodology, the script was tested using monthly dig lines related to planning polygons and calculate the dilution and losses considering the block model. The tested files are from an open pit gold deposit, and the planning polygons consist in dig lines design in the benches, containing ore and waste blocks. This dig lines represent the portion planned to be mined, according to mine plan. All scripts procedures were performed by an average of 1 to 2 minutes.

The output is a new model with all the original fields plus a new field containing the diluted grades after the calculations. The results shown in Figure 5 are a comparison between the planned average grades against diluted average grades in six months, considering external and internal dilution and loss for a typical set of polygons representing the short term mine planning.

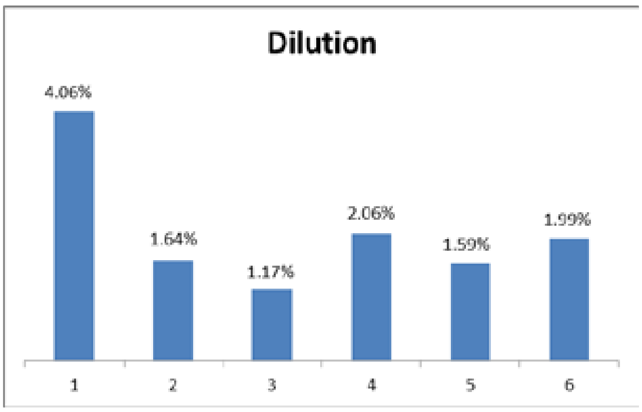

Figure 5 - Dilution per period

In order to estimate the dilution for the analyzed period it was calculated an average, weighted by the mass mined in each month. Thus, it is possible to obtain the average value of dilution of the period, which in this case was 2.1%. Assuming an average value for all months, could mask differences that happen in each month, like a difficulty to access determined area, or mining more waste to liberate the ore, etc. Using this methodology, the dilution is calculated, not a fixed value.

CONCLUSION

Crawford (2004) comment that contact dilution is determined by resolution; specifically, the ability to accurately define and then mine along the limits of an ore zone, such that the available mining equipment can successfully extract the material at those limits. This work allowed, in addition to calculate the contact dilution, (called here internal dilution), the calculation of external dilution and loss. These calculations aim to estimate an average dilution for the period.

The use of a dilution factor in the estimated block model, before mining, is a way to predict grades and masses closer to the mined. When a block model is estimated and the actual dilution is not considered, an arbitrary number is normally set to be the dilution used in the model. This arbitrary value often does not take into account all the factors that actually cause dilution.

The consideration about the deposit geometry, more specifically benches, slope angle and execution efficiency were the items that determine the calculation of dilution. It is known that each feature will contribute in the calculation, but it is the combination of those aspects that will determine the total dilution calculated.

This study demonstrates an attempt to make a simple and direct approach of a parameter that is known to be complex and difficult to control, and that the majority of companies in the mining sector or simply disregard this effect or when considered it was used inadvertently and obscurely by applying a fixed number or factor for the entire deposit without proper understanding the source of the problem and the consequences that this problem can have to when mining different deposits. In this case, the deposit tested is an open pit gold deposit, which is mined using mechanical extraction. The methodology could be adapted for different deposits and different operational characteristics, for example, to calculate dilution in a deposit mined using drill and blast, the dilution range would change to a percentage of the block that is diluted with his adjacent, instead of using the equipment imprecision (in meters) when mining the block.

The methodology has been tested in different types of deposits and the idea is to keep doing tests to check the impact of dilution considering the variability inherent to each deposit and operational conditions. Normally dilution is a number chosen by heart and not very well explained and calculated. This methodology details a way to incorporate dilution in the short term mine planning and to be systematically considered along the mine life.

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EVALUATION OF SHIELD WITH CHAINS ON TIRES OF OFF-ROAD TRUCKS 777 F/G OF BAUXITE MINE IN PARAGOMINAS MINING S/A

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EVALUATION OF SHIELD WITH CHAINS ON TIRES OF OFF-ROAD TRUCKS 777 F/G OF BAUXITE MINE IN PARAGOMINAS MINING S/A

ABSTRACT

The mining of bauxite is performed by the method known as strip mining. This method is developed in strips due to the morphology of the deposits, which generally present tabular geometry and are close to the surface. To perform the mining it is necessary to remove the overburden in the mining area in order to expose the ore, and this operation can be held by different equipments according to the height of the layer of waste rock. In the Mineração Paragominas S/A (MPSA), located in the city of Paragominas - PA, the layer of waste rises over 13 meters high. The overburden removal is then performed in three ways depending on that height: using bulldozers CATERPILLAR D11 with trident-shape ripper attached (H>8m), bulldozer and hydraulic excavator to remove the remaining material (13m< H >8 m), and off-road trucks CATERPILLAR 777F/G and excavator system for waste layer recess, operation with bulldozer D11 and hydraulic shovel. The off-road trucks, which are characterized by having greater adhesion and resistance to operate in unfavorable conditions, are responsible for 15% of all movements of waste in MPSA, ensuring best rates of overall equipment effectiveness, in addition to continuity and synchronism of the overburden removal process. However, in the MPSA area, the historical levels of precipitation in the period from December to May are quite high, which directly impacts the mining system. Faced with this problem, shields were installed on the external rear tires of the off-road trucks of Mineração Paragominas S/A, pulling them in order to facilitate the operation of such equipments. The results were satisfactory, for the chains increased the trucks rate of Physical Use, ensuring the continuity of methodologies of overburden removal productively, and ensure a safe operation, since the chains prevent slippage and/or lateral slippage during cornering and maneuvers.

KEYWORDS

Bauxite; overburden removal; off-road trucks; chains

INTRODUCTION

Bauxite is the world's leading source of alumina, but there are significant differences between the various deposits, characterizing this rock for specific uses, depending on its mineralogical composition and associated impurities. Some bauxites have the approximate composition of gibbsite, however mostly form a mixture, containing impurities such as silica, iron oxide, titanium and other elements. As a result, bauxite is not considered a mineral species, and a rigid classification, “bauxite” name should be used in reference to the rock (Bauxite). The main constituents of the rock are: gibbsite, a boehmite and diaspore. About 95% of all bauxite extracted in the world is used for the production of alumina by the Bayer process, of which 90% is extracted aluminum. The remaining (10%) is intended for other purposes such as refractories, abrasives, clay cements, chemical industry etc. (Sampaio & Neves, 2012).

The company Mineração Paragominas SA (MPSA), belonging to the Norwegian Norsk Hydro, is located in the municipality of Paragominas, which is distant 360 km from the capital Belém, in Pará State. The mining and processing of bauxite operations are concentrated in the Plateau Miltonia 3, and the bauxite concentrate is transported by a ore pipeline 244 km long to the refinery Alunorte Hydro, which is located in Barcarena/PA (Pereira, 2012).

Bauxite mining of Mineração Paragominas S.A. involves complex operations, since it is a differentiated deposit to be formed by thick layers of sterile material, which range from 2 to 20 meters. The average thickness of the overburden is 14 m and the mineable ore layer has average thickness of 1.65 m (figure 1).

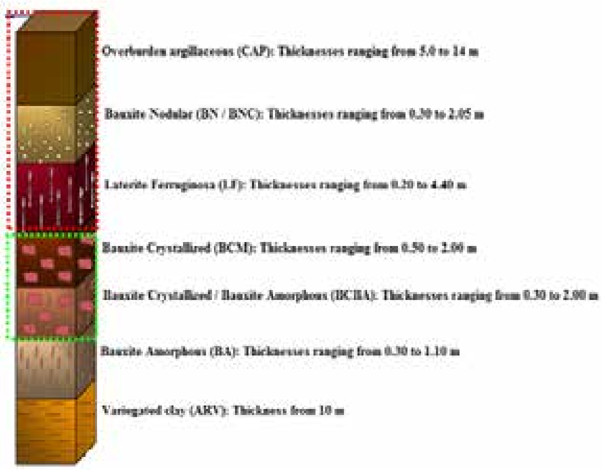

Figure 1 - Lithological profile Plateau Miltonia 3

The four layers overlying the ore layer are: CAP layer (overburden argillaceous) is sandy-clay surface layer of yellowish (clay Belterra) with an average of 11 meters thick; BN layer (Bauxite Nodular Pisolítica and / or Concrecionária) where bauxite is presented in the form of yellow-reddish nodules with small gibbsite crystals micro to criptocristalizados; the BNC layer (Bauxite Nodular Crystallized) also has bauxite in nodular form, reddish, but with visible gibbsite crystals to the naked eye; and the LF layer (Laterite Ferruginosa), which has high resistance to be rich in iron oxide, and then used as lining of access and roads of mine (Pimentel, 2012).

The ratio sterile/ore in MPSA bauxite mine is around 5,5m³ sterile per ton of ore. This characteristic influences directly and significantly in the stripping operation, which consists of removal of waste material overlying the ore layer, which makes necessary to create strategies in the execution of this operation. Thus, depending on the thickness of the overburden layer, stripping can be performed using only bulldozers D11 (direct stripping); using bulldozers more hydraulic excavator (pre-cut and tipping, respectively); and using the sistem off-road trucks more hydraulic excavator to demean (figure 2), with tractor operation (pre-cut) and hydraulic excavator (tipping) (Ferreira, 2012).

Figure 2 – Stripping using hydraulic excavator + off-road truck

The off-road trucks are responsible for about 15% of all sterile handling of MPSA, ensuring better operational performance index, besides continuity and synchronism stripping process (Pimentel, 2013). Dump trucks are dimensioned for heavy construction services. They are of great tonnage and dimensions that are beyond the normal, which prevents their use in normal traffic roads, and are characterized by being more robust, with more grip and resistance to operate in adverse conditions and terrain difficult to access (Ricardo & Catalani, 2007).

However, in the region of MPSA, the historical levels of precipitation rainfall in the period from December to May are quite high, which impacts directly and significantly the mining system and other activities of the mine, including the operation of off-road trucks. The graph in Figure 3 shows the average rainfall of the years 2008-2013 (Pimentel & Magalhães, 2013).

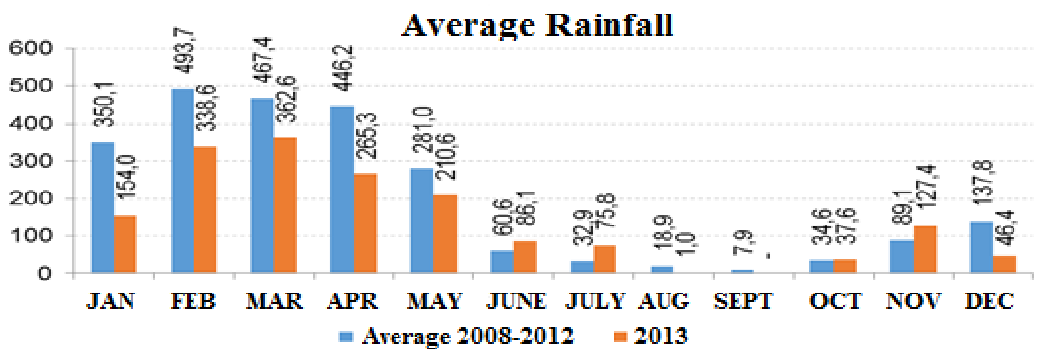

Figure 3 – Average rainfall 2008 to 2012 MPSA

In addition to the high rainfall, there is another factor that affects the operability of trucks: the clayey characteristics of formation material of the tracks and accesses where travels these trucks, because due to saturation of the material, there is low adherence between the tread of tires and the ground and, therefore, there is the appearance of gnats and barriers, which combined with the high sinking, increase the rolling resistance coefficient, making it reach the operating point of impossibility of trucks (Pimentel and Magalhães, 2013).

The operation of off-road trucks is significantly affected, and the resulting impact is a reduction of Use Global Physics (UT), which is a related index to losses by stops where the equipment is available but is not being used (unproductive hours internal and external). In this case, the UT trucks is greatly reduced by the fact they are unable to operate or expend a lot of time to make their cycle time. In the figure 4, the chart shows the UT of CAT 777 for the years 2011 and 2012 of MPSA.

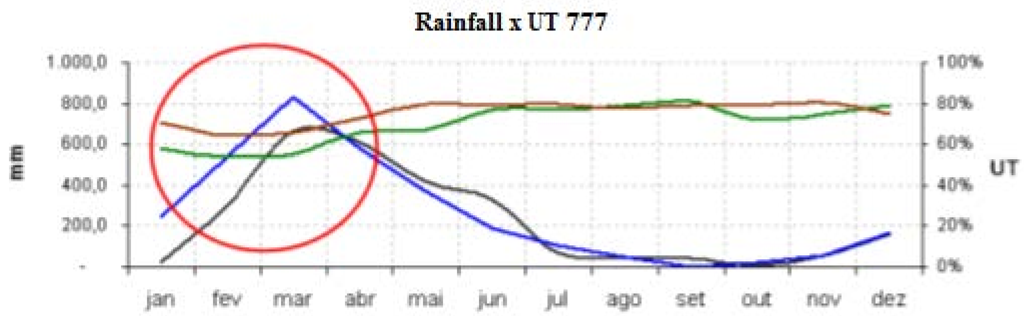

Figure 4 – Rainfall x UT CAT 777

As can be observed in the period between January and April, the rainfall incidence is more intense, impacting directly and significantly the Physical Use of trucks. Faced with this problem, an evaluation of the implementation of a technique known as shielding was performed, consisting of a protective coating with the use of a dense and flexible steel mesh consisting of forged elements and rings union soldiers (current), tires outboard rear of off-road trucks CAT 777 Mining Paragominas SA, in order to increase the grip conditions and, consequently, the trafficability of the equipment, allowing its operation more continuously during periods of high rainfall. Combined with the application of screens, actions related to infrastructure ramps and loading squares and tilting to reduce the hours of stopping these trucks were held, increasing the time traffic during rainy times and returning more rapidly its operations after function stops these moments.

METHODOLOGY

The methodology consisted in the installation of shielding on external rear tires of off-road trucks CAT 777F. The shields were installed in three trucks in 2014, enabling the collection and comparison of UT operating rates in the rainy season of all CAT 777 trucks fleet and cycle time evaluation and trafficability conditions on slippery slopes of trucks with shields and unshielded. One of the trucks still remained unshielded, and had its operation evaluated in relation to the others.

The shielding installation process is relatively simple and can be done either by a technical supplier of shields (RUD), and by MPSA employees.

The evaluation of the operation allows to evaluate the traffic of off-road trucks with and without chains, assessing the amount of downtime, up loaded, down empty, and maneuvering in the areas of loading and unloading. The chains were installed in 03 trucks from the month of January: CAT 777F 6501, CAT 777F 6502 and CAT 777F 6503. We evaluated the percentage of downtime by bad weather and consequently the use Physics (UT), in the period from January to April 13, 2014, and the shields remained in trucks.

RESULTS AND DISCUSSION

During the evaluation, there was no performance difference in travel by trucks with and without chains on dry roads: they achieved the same cycle time and UT index. However, the wet roads, the truck unshielded kit presented in 03 events, enormous difficulty climbing the ramp, and the other 07 could not even rise due to the lack of adherence between the propellant train and the ground. This can be seen in figure 5, which shows that the clay and moist soil fills the grooves of the tread band, reducing adherence and resulting in the slipping.

Figure 6 – Absence adherence between the tire and the ground

The clay soil causes not only the slipping, but also a high degree of sinking and generation of gnats, creating more forces opposed to truck movement to the point of impossible operation because for each sunk centimeter, it is required strength 0.6% of the machine weight to overcome the resistance, not to mention the natural resistance to the ramp. So, whereas the displacement force exceeds the others resistances, it can be said that there is movement. However, with regard to the friction or adherence (Fa), one must consider the existence of a minimum value in the direction opposite to the shifting and tangential to the periphery of the tire and is always higher than the tractive effort, i.e., from the moment: $F_{a} = f.P_{m}$, is set at the slipping condition. That way whenever the tractive effort does not exceed the adhesion force, no slipping and the tire rolls over the surface (Ricardo & Catalani, 2007).

Trucks with shields were able to make their moves, as there was an increase in adhesive strength and conservation of tractive effort (figure 6):

Figure 6 – Tire shielded - greater adherence to contact area

Although they have greater friction conditions, it was observed that in situations of heavy rain and high degree of saturation of the terrain, even with current equipment skidded and hence the operation had stopped. However, the time of resumption of the same activities was less than what was without power.

From the data collected in the period from January to April 13, 2014, it obtained the percentage of downtime hours of the trucks with shield as well as unshielded, as can be seen in the graph of figure 7:

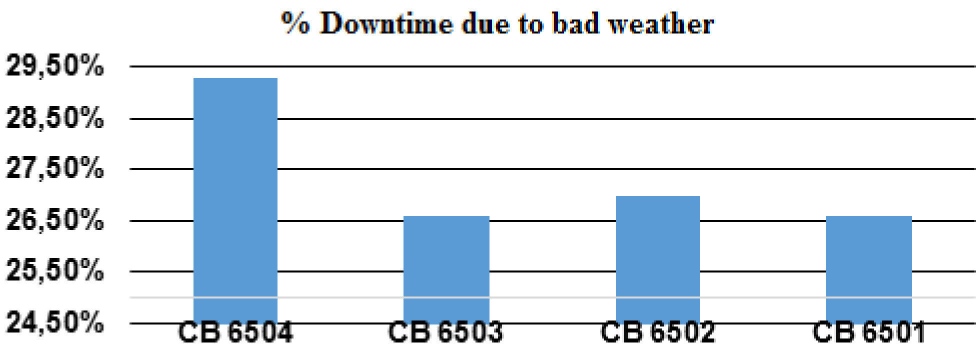

Figure 7 – Percentage of downtime due to bad weather the fleet CAT 777F

The truck 6504 was the one that was without the chains. With this, their time of recovery after the rain and stopped for rain are higher than other trucks, impacting thus their UT. It can be noticed that the trucks with shields had a performance in terms of UT 3.5% higher than the unshielded.

Time chainless truck cycle is zero, because it does not run. Already with chains manage traffic normally, as if the roads were dry. However, at some point, when it is too wet, it is necessary to stop the operation of the fleet, for the operation of the truck is compromised.

It was also observed that there was no marked wear on the tires, since the material of roads (clay) is non-abrasive and chains are used only in rainy periods. The useful life of the tires is MPSA or more least 15,000 hours and there was no significant reduction in its useful life.

CONCLUSIONS

In Mining Paragominas SA, the off-road trucks play a fundamental role in the stripping operation, but has its UT compromised by rainfall and conditions of local access roads.

So, the application of shields with chains in external rear tires these trucks is essential, because they ensure the operation thereof, in that the chains increase adherence, so that there is greater traction conditions and transmitting the force necessary for the displacement.

The evaluation helped to confirm that the use of current increased the UT CAT 777F fleet by an average of 3.5%, reducing downtime hours by bad weather in relation to the truck unshielded, which were unable to operate. That way you are ensuring or increasing the operating frequency of such equipment in rainy periods, enabling the assurance of continuity of methodologies stripping and consequently achieving better sterile material handling rates.

Moreover, the trucks equipped with standard showed no slippage or lateral slipping during turns and maneuvers, thus favoring the security and integrity of the operation. It wasn't also necessary the realization of any relevant maintenance on the shields, only tensioning adjustments made by the company.

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EXPERIENCES FROM EXPERIMENTAL MINING IN BRAZIL

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ABSTRACT

The Experimental Mine (EM) of the Research Center of Responsible Mining of the University of São Paulo became the subject of investigation in a few years after its development; it is an open-pit quarry currently exploiting marble and gneiss, used to produce industrial limestone and construction aggregates. It is a developing enterprise, dealing with the challenges of a technological upgrade from a small-scale operation to the characteristics of a medium-sized company. The Experimental Mine Project (EMP) was born to attend a double demand: to provide research and development (R&D) support to a growing company and to provide experimental opportunity for a field of knowledge such as mining engineering that requires a large scale for its experiments.

The main challenges of the EMP are related with the remaining small-scale mining features, such as large variety of equipment available, high level of operational flexibility, scarcity or absence of mine planning, being focused on daily operations. In such an environment, the first role of the EMP was to evaluate in a quantitative way the effects of unit operations over the whole mining process.

The current excavation technique is by drilling and blasting. Many experimental campaigns have been conducted on site, with different purposes. One of the main research lines was to increase the productivity of the quarry by lowering production costs and improving the quality of the product, then optimizing the entire production cycle; the relationship between the unit costs of drilling and explosives were evaluated, as well as the link between the blast design and some factors affecting the downstream processing of the product.

The paper describes the methods employed to conduct the research and the improvements to be pursued, with the due consideration to the influence and interference of the many parameters involved, from the rock-mass characteristics to the final products.

KEYWORDS

Blast design, rock fragmentation, downstream processing, damage control, comminution energy

INTRODUCTION

In designing a blast, the geometry is a very important factor, but also the amount and type of explosive and the timing sequence play an important role. Everything should be established in view of the desired effect, and containing, as much as possible, the side effects (Mancini & Cardu, 2001). Drill and blast is an important step in this process and the results such as fragmentation, muck-pile shape and looseness, dilution, damage and rock softening affect the efficiency of downstream processes (Richard et al., 1982; Roy & Singh, 1998; Konya & Walter, 1990; Oriard, 2005). The importance of blasting to downstream processes has been studied and discussed by many researchers. Nielsen and Kristiansen (1996) investigated the effect of blasting on crushing and grinding operations and discussed how to evaluate the application of the comminution system. They pointed out that the gap between mining and mineral processing should be harmonized, and suggested that blasting could be considered as the first step of the integrated comminution process for the optimization of the mine operations. Size reduction represents one of the most energy-intensive and costly processes in the excavation of rocks. Drill and Blast, being the first operation in the size reduction chain, may have a significant downstream effect (Kim, 2010). One of the goal of the research conducted at EM under this aspect was to examine the effect of different timing sequences on fragmentation, although the subject has been extensively treated in many aspects (Katsabanis et al., 2006; Kim, 2010; Stagg, 1987). In the same way, it is quite common for quarry operators to be concerned with fragmentation when difficulties in drilling and loading are encountered, or when a large amount of oversize is produced, resulting in a general loss of productivity in secondary blasting: this was the problem encountered at the quarry site under study and, on this basis, a number of experimental blasts were performed and the blasting size reduction effect was recognized, as shown below.

The influence of the blasting activity on the grindability of the basted material has been thoroughly researched in recent years. Nowadays it is widely accepted that blasting produces two effects on the broken rock: it induces visible fracturing, that is measurable by means of image analysis or sieving and, at the same time, induces invisible fracturing, which means a system of micro-fractures, invisible at the naked eye, that are detectable only by microscope analysis but show their direct effect by decreasing the grinding energy to reduce the material to a desired particle size distribution; it can be said that this effect "softens" the material.

The downstream effects of the first aspect are widely researched and discussed (Mackenzie 1967; Clerici et al., 1974; Scott, 1996; Božic, 1998; Sastry & Chandar, 2004; Morin and Ficarazzo, 2006; Mansfield & Schoeman, 2010; Seccatore et al., 2011; Cardu et al., 2012; Dompieri et al., 2012). Rather precise models have been developed over the years to achieve a good prediction of the output of the blast in terms of particle size distribution, such as the widely-used KUZ-RAM (Cunningham, 1983, 2005) and the SWEBREC (Ouchterlony et al, 2006) among other studies. Nielsen and Kristiansen (1996) pioneered the research on the latter subject, showing that increases in P.F. lead to higher presence of micro-fractures in the material, therefore achieving significant reductions in Bond Work Index (WI, Bond 1961). Workman and Eloranta (2003) quote Nielsen and Kristiansen's data (sedimentary iron ore - taconite - as an experimental material), and show the comparison between WI of the blasted material (WIB) with the one of the intact rock (WIR), leading up to -74% in the WIB with a P.F. = 0,42 kg/m^3. Katsabanis et al (2008) obtained similar but less incisive results in different granite formations: P.F. = 1,15 kg/m^3 led to the maximum reduction of WIB of -11% compared to WIR. Workman and Eloranta (2003) point out that, since mill feed is relatively small (usually smaller than 19mm (3/4"), the fractures that survive at this size must have at a microscopical level an effect on the WIB. On the other end, Katsabanis et al. (2004) discuss the limits of the influence of blasting on grinding efficiency. From what previously reviewed, it is evident that the effects of micro-cracking highly depend on the lithotype. Workman and Eloranta (2009) discussed the different ways to achieve higher P.F. for the benefits of grinding: this can be done by increasing the amount of explosive per hole or by changing burden and spacing, that is to say varying the distribution of charges. Nonetheless, Eloranta's considerations on this subject remained theoretical and based on an economical point of view. In the present work, the influence of different geometrical distribution of the charges on the grindability of the blasted material was then investigated empirically.

Moreover, It is a well-known fact that the performance of blasting has a significant effect on loading, secondary breakage and crushing, as established by numerous studies (Cheatham, 1968; Chi et al., 1996; Michaux and Djordjevic, 2005; Kanchibotla et al., 1999; Nielsen and Lownds, 1997; Kojovic, 2005; Nielsen and Kristiansen, 1996; Workman and Eloranta, 2003 and 2009; Roy & Singh, 1998; Konya & Walter, 1990; Oriard, 2005). Nonetheless, these researches have been carried out in major mining operations, in both metallic and non-metallic ores, in situations where there was a proper control of all the factors involved, from drilling to crushing. All the proper controls led to refined fragment size distribution prognosis models, such as the most recent Swebrecf unction (Ouchterlony, 2005). The present research takes a look at a completely different situation, such as a small mining operation in Brazil. Mining operations such as the small-scale situation analysed here should not be confused with what is defined in literature as "Artisanal and small-scale mining" (Seccatore et al., 2015): the definition here adopted merely refers to the scale of production, that encompasses ventures that produce between 10 and 100 mtpa, but still are considered industrial operations, in the Brazilian context. Such small mines are characterized by a large variety of equipment availability and a high level of operational flexibility. Therefore in this context, mine planning is usually scarce or absent, and mine management generally focuses on daily operations. Therefore, the analysis of the effects of unit operations, such as blasting, over the whole mining process is often neglected. To overcome these aspects, the research was focused to propose answers to the following questions: the dependence of the performance of drill and blast on the loading, secondary breaking and crushing time in a small-scale limestone quarry; the way to properly analyze the operations in a small-scale quarry, considering the restrictions due to the scale and the constrains of costs.

THE EXPERIMENTAL MINE

The Experimental Mine of the Research Center for Responsible Mining of the University of São Paulo, Brazil, is a quarry, located at the city of Taubaté, Brazil, exploiting dolomitic limestone for acting as acidity buffer for agricultural soil or as an aggregate for civil construction. It is a small operation, that faced constraints due to old methods that were considered the regular operation procedure before the beginning of the Experimental Mine Project. At present, since the start of the project, there have been great changes in the regular practices as well as the continuity of the modernization of the equipment used.

The goal of the research is to depict a way to improve the rational use of the explosive energy for the benefits of the quarrying process. An extensive literature review shows how a good fragmentation by blasting favourably influences the profitability of the whole mining process. Many methods allowing prediction and estimation of the fragmentation are available: if carefully and reasonably used, they can be very helpful to obtain an optimal fragment distribution which lowers the total cost of the whole production process and not only that of drill and blast. The quarry operates with two different crushing systems, one for the aggregate and another for the acidity buffer. The reason for this is due to the lower specification standards that the aggregate has on the market. While the aggregate goes through a primary- and a secondary crusher only, the limestone for use as an acidity buffer must go through a cycle of grinding and milling in order to achieve the particle size distribution for industrial standards.

GEOLOGY OF THE BASIN

The area under study is inserted into the *Terreno Embu*, which includes the Central Segment of the *Mantiqueira Province* (Heilbron et al, 2004), called *Ribeira Belt*, which origin is Neoproterozoic/Cambrian. The region is divided into five tectono-stratigraphic areas (Howell, 1989) separated by thrust faults, sometimes by oblique shear zones (Heilbron et al, 2004). The complex corresponds to the *Embu* meta-sedimentary structures (Hasui, 1975), and constitutes the *Açungui Group*. Meira (2014) identified maximum ages of sedimentation through the dating of zircon grains, very old: Paleo-proterozoic and Meso-proterozoic. The lithological associations were divided into three stratigraphic units (Fernandes, 1991): the *Rio Una* Unit, superior and dominated by mica-schists and quartzites; *Rio Paraibuna Unit*, dominated by quartzite and silicates, with intercalated biotite gneisses and amphibolites; and *Redemption Serra Unit*, composed of gneiss, amphibolites, and marbles (Heilbron et al, 2004). The main metamorphic *Embu* complex is in the region of sillimanite-muscovite, locally reaching the sillimanite-K-feldspar zone (Vlach, 2001), and relates to the first two of the five phases of deformation affecting the *Embu Complex*. The first phase of deformation was observed only in relict structures associated with high temperature parageneses. The second deformation phase generated the main foliation folds and small recumbent folds, and is associated to a quite intense mineral lineation (Heilbron et al, 2004). The third deformation phase is related to reverse and tight bends. The fifth phase is delayed and transversal to the NE orientation. The shear zones bordering the ground are milonitic and vertical, and control the placement of granites (Janasi et al, 2003). Meira (2014) identified bimodality of metamorphic ages of *Embu Complex*, between 650-600 My and 600-560 My, and the metamorphism occurred in the first indicative range of metasedimentary successions at depths of up to 25 km, reaching amphibolite facies. The newest metamorphism is characterized by an almost isobaric decompression up to about 10 km deep, at temperatures between 550-600° C. The large shear zones mark the third phase of deformation (mylonitic foliation).

The rock at the quarry is exploited along an elongated strip according to NE-SW direction, and was mapped by Orcioli (2010), who identified four main rock types: dolomitic marble; biotitic gneiss, commonly milonitic; amphibolite, which occurs between marble and biotitic gneiss and granite, intrusive in metasedimentary units (Figure 1). According to the author, the sequence shows a structure that is cut by a thrust fault in direction WNW-ESE, in the central portion of the polygon of the quarry.

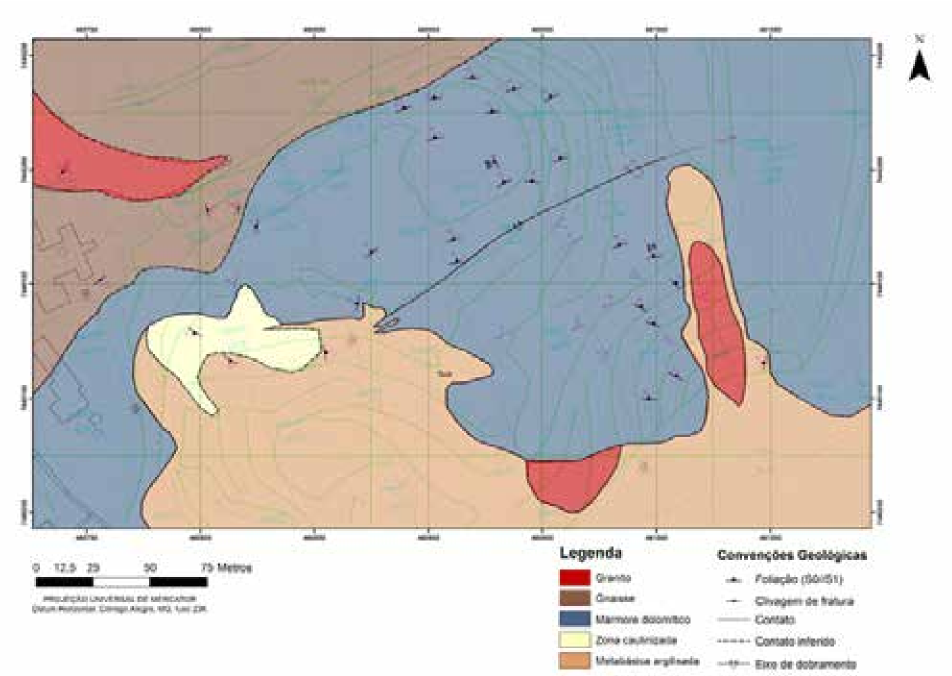

Figure 1. Geologic map of the quarry following Orcioli, 2010 (unpublished-internal report).

In the area under study several experimental campaigns have been carried out for the geotechnical characterization of the material; four structural domains were detected and, through the *Dips* software, fracture families for each domain were investigated. The frequency of fractures for each lithotype was determined on field, and used for the calculation of the J_v index. The geomechanical characterization of the rock mass was based on data acquired from experimental investigation. Particular emphasis was given to the differences between the various rock types, by association of the geotechnical parameters to the structural features, textural and mineralogical characteristics of the rock types. The results showed that the rock mass has, in general, good features, with values of RQD preferentially high: from 50 m depth under the ground level, there are no significant intervals with values less than 50 (average values being from 69 and 100).

DRILL AND BLAST OPERATIONS

Drilling was performed until the third quarter of 2014 using light pneumatic crawl mounted rigs, quite out-dated. This equipment was later changed for more modern models of crawl mounted rigs by a local manufacturer. In both equipment's the drilling operation is controlled manually by the operator. The borders of the drilling area are determined by means of optical topography; then, the locations of the holes collars are determined with pre-dimensioned wooden rods and marked on the ground using small boulders found on site. The blasting process involves charging the holes with 50.8 mm cartridges of explosive emulsion, primed by a strand of detonating cord along the hole, leaving an upper stemming of about 2 m. The stemming is done using a mixture of two gravels coming from two of the quarry's product piles, without the particulate generated by the drilling operation. The blast is fired by safety fuse and a fire cap that initiate the main line of detonating cord; delays are provided by means of relays (17 ms series). The standard deviation presented on the delay time around the nominal delay is quite large, due to the quality of the manufacturing process of blasting products, as well as the issues that arise from pyrotechnical delays, provided by local suppliers. The typical blast parameters are given in Table 1.

The situation presented at the beginning of the research showed no proper control of drilling, no proper attention payed to the firing sequence and distribution of the relays amongst the holes, and issues with the geometry presented on the blast. This situation proved to be ideal to analyse a typical low efficiency operational condition.

During the implementation of the project, the researchers have been able to change the habitual practices, introducing the following improvements:

1. Drilling is supervised in order to grant correct collar positioning, inclination and use of torque and thrust feed;
2. Charging is adjusted to the bench conditions, leaving intermediate stemming to reduce linear charging according to the drill speed rate encountered and to the presence of depressions or indents on the bench face;

Table 1. Typical Blast parameters.

Parameter	Symbol	Unit	Value
Hole diameter	ϕ	mm	63,5
Spacing	S	m	2
Burden	B	m	2
Blasthole length	Hdl_{bh}	m	6 to 9 m
Sub-drilling length	l_{sub}	m	0,5
Hole inclination	α	°	75
Stemming length	l_s	m	2
Specific Charge	q	kg/m^3	0,40 to 0,47 kg/m^3

3. The point of initiation of the firing sequence must take maximum advantage of the free surfaces to favour the movement of the blasted material;
4. Simultaneous holes in the firing sequence (due to linear combination of 17 ms relays) must be located as far away as possible, to avoid undesired cooperation of charges that may induce the explosive energy to work with shear effect instead of producing fragmentation.

THE INFLUENCE OF CHARGE DISTRIBUTION ON TH ECOMMINUTION ENERGY

Small-scale blasts have been performed on 14 marble blocks with different Powder Factors (PF) and charge distributions. For every PF, charges have been designed to simulate concentrated and distributed geometries. In particular:

- Holes were drilled with Ø = 6 mm (diameter), L = 60 mm (hole depth);
- Each hole was charged with a strand of detonating cord with linear charge of 10 g/m, achieving a 6g charge per hole;
- To each block has been assigned a desired theoretical PF (PF_{th}). Adjusting the number of holes per block allowed to achieve a PF (PF_{real}) as close as possible to PF_{th}; adjusting the geometry of the holes allowed to simulate different charge distributions.

Two charge distributions were simulated as follows:

a. Concentrated charges simulate open-cast blasts with large-diameter holes, with large burden and spacing.

b. Distributed charges simulate bench blasts with small-diameter holes and reduced burden and spacing. Examples are shown in Figure 2. Three control blocks have been fragmented by mechanical means for comparison of the results. All blast tests took place on field due to safety reasons.

The blasted material was then analysed considering the particle size distribution and the Work Index WI. Results obtained suggest that:

1. Concentrated charges lead to a particle size distribution closer to the "dust-and-boulders behaviour". Distributed charges lead to a more uniform particle size distribution;

2. When increasing the specific charge, distributed charges lead to a greater reduction in particle size than concentrated charges.

3. Material blasted with distributed charges presents a steeper decrease of WI at the increasing of the specific charge.

Small charges with a distributed geometry transmit more uniformly the explosive energy to the rock, leading to better fragmentation and higher induction of micro-fractures. This reduces the total comminution energy necessary to grind the blasted material to the desired particle size. Looking at the application on field, these results suggest that for any given powder factor, choosing to reduce the drilling diameters and increasing the specific drilling will benefit the comminution circuit.

TESTS WITH NON-COAXIAL CHARGES FOR CONTOUR BLASTING

Contour blasting is commonly performed by employing linear charges, decoupled from the boreholes. To achieve the best results in terms of rock breakage and respect of the excavation profile, blasting theory suggests that charges should be inserted coaxial to the holes to grant uniform distribution of the explosive energy and therefore obtaining a uniform Radius of Damage. Nonetheless, non-coaxial charges are often employed in blasting practice. Non-coaxial charging methods include the employ of high-power detonating cord (40 to 100 g/m), low-power detonating cord connecting small-diameter cartridges (commonly 10 g/m detonating cord priming 1" cartridges) or string loading (a thin layer of bulk emulsion pumped with controlled flow and controlled extraction of the injecting rod). This research was focused on evaluating the effects of the first two charging methods on the quality of final walls. Different drilling geometries and charging configurations were applied (Figure 3).

Figure 2. Left: Example of a block with drilling set out to simulate concentrated charges for PF_{th} = 0,3 kg/m^3; right: Example of a block with drilling set out to simulate distributed charges for $P_{\cdot th}$ = 0,3 kg/m^3.

On the other hand, when the rock is poor, any quality of the final wall is hardly achieved at all, in spite of any care in the details of execution of smooth blasting (Figure 4). It is concluded that any design criterion and theoretical approach modelling the effects of contour blasting cannot ignore the features of the rock mass.

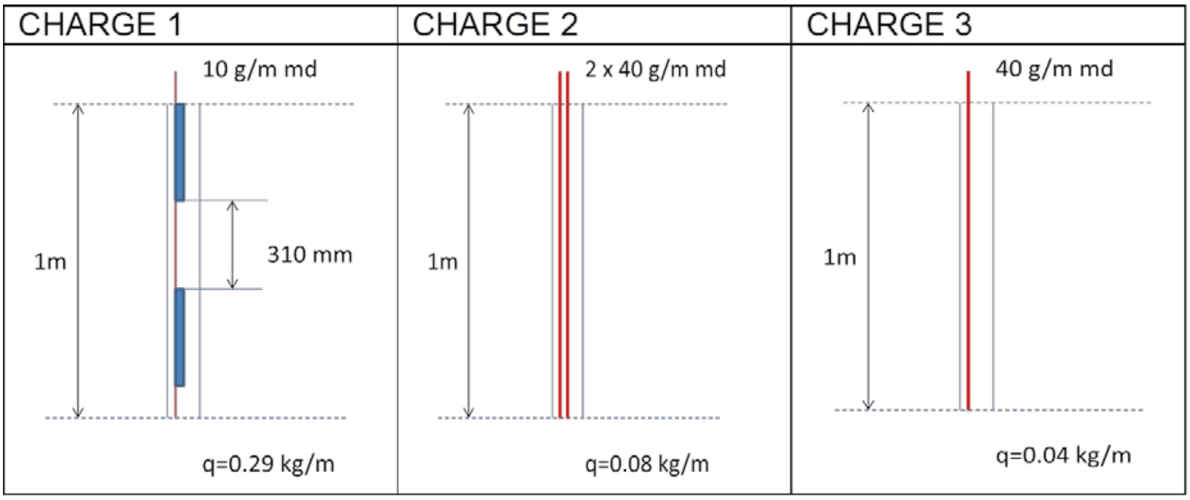

Figure 3. Details on loading adopted for experimental blasts1, 2 and 3.

The Half-Cast Factor (HCF), the Over-break (OB) and the Under-Break (UB) were evaluated as control indicators. Rock Quality Designation (RQD) and Rock Mass Rating (RMR) were used to classify the rock mass. The research was aimed to push contour blasts to their limits, observing for which geometry and charge configuration the blast lost its design threshold with respect to the final wall for every given rock mass. Results show the operational limits of non-coaxial charges: in good-quality rock, smooth blasting with decoupled linear charge of 40 g/m can be extended to a spacing S = 22Øf with little or no detectable drawbacks in terms of final wall quality, in contrast with theoretical formulae for the determination of the radius of damage.

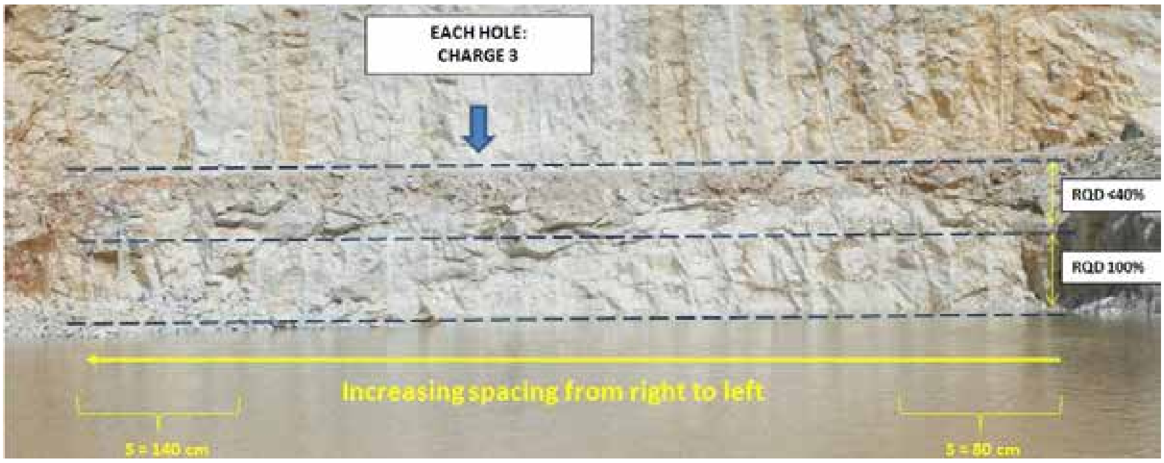

Figure 4. Results of a blast, which was performed by progressively increasing the spacing between the holes. It can be observed that HCF=100% for every spacing in RQD=100% and HCF=0 in RQD<40%.

KEY PERFORMANCE INDICATORS (KPIs) TO EVALUATE THE INFLUENCE OF THE DETONATION SEQUENCE IN ROCK FRAGMENTATION

To monitor any changes that a modification of PF, blasting pattern or timing can cause to the downstream process, some Key Performance Indicators (KPIs) were chosen. They possess the common characteristics of being representative and simple to obtain on field: the objective is to create a database that can provide some valid correlations between different factors, and this is possible only if the operators can get in autonomy and simplicity these values. The KPIs selected are listed, defined and described in Table 2. Changes had to be gradually introduced for practical and economic reasons. Only few blast tests were completely designed and performed according to the new blasting method. The project involved a delay between each blast-hole; moreover, relays delayed by 42ms have been used for the first time, so as to increase the time difference between the ignition of the blast-holes of a row and those of the next.

The measured parameters showed good results: homogeneous muck-pile, with a low percentage of fines and a reduced volume of blocks whose size requires secondary breaking; good quality of the final wall, with acceptable values of HCF and B_b; absence of fly-rock; the high cost caused by a high number of delay units was compensated by a reduction in the cost of drilling, thanks to the increase of spacing that has allowed realizing fewer holes to cover the entire surface to drill. As an example, some details of a blast (the last of the experimental campaign) are shown in Figure 5.

Considering the results obtained, it can be noticed that:

- The shape of the muck-pile S_m improved significantly over time: this has the strong advantage of increasing the productivity of the loader, reducing the idle time that consisted in gather the material spread away from the pile;
- The back-break B_b is strongly reduced. Thanks to a lower loading/hole, and to a different timing, it was possible to obtain a better precision of the residual wall;

- The evaluation of the parameter S_b is obtained by the work of the hydraulic hammer, that is greatly reduced. Despite some blast were made in an area where the rock was particularly strong, a better dimensioning of both the charging and timing of the blast has led to a better quality of the product. By comparing the performance of S_b with the PF employed in the examined blasts, it can be noticed that to a greater amount of explosive/m^3 does not necessarily correspond a greater fragmentation;
- The accuracy of the result obtained as the regularity of the residual wall is concerned is greatly increased, giving rise to a very high HCF;
- As it can be seen on the performance of F_y, fly-rock is completely absent in the last two blasts. This results in increased safety and reduced projections of material away from the pile.

Table 2. KPIs selected for the research

KPI	Symbol	Unit	Definition	Reason for Choice
Shape of the Muckpile	S_M	-	It is the dimensionless ratio of the spread length of the muckpile, normalized with the length of the blasthole before blasting.	This value gives a realistic idea of the expected productivity in phase of mucking and loading due to the easiness of operations of the loaders on the restricted quarry floors.
Backbreak	B_b	*M*	It is the average value of retreat of the bench edge with respect to the alignment of the contour holes. In the case analyzed, it was calculated every 2m, which were measured along the contour of the quarry boundary.	Back-break can generate various problems, whether due to stratification of the rock, or to design errors. To assess this factor, some changes were attempted such as modifying the charge distribution along the blast-holes and varying the blasting sequence, relying on the constant arrangement of the layers.
Specific incidence of secondary breaking	S_b	h/m^3	It is the time of work of the hydraulic hammer employed for secondary breaking, normalized to the volume of the bench before blasting. The operator of the hummer, after a short training, can take this value autonomously.	The good or bad outcome of a blast in terms of particle size can be evaluated according to how many hours the hydraulic hammer has worked on a muck-pile to reduce oversize blocks below the threshold size value.
Half-Cast Factor	*HCF*	%	Dimensionless ratio or the percentage between the total length of half-casts observable on the wall remaining after the blast and the total length of contour holes drilled and blasted. It can be taken as an indicator of severe (HCF null), moderate (HCF medium) or low (high HCF) mechanical damage of the residual wall.	HCF is an indirect indicator of the damage induced by contour blast-holes. Good guidance of the fracture occurs only if the pressure in the holes does not greatly exceed the minimum necessary to obtain the detachment;
Flyrock	F_y	*M*	Flyrock was defined as the distance of loose fragments encountered outside of the shaped muckpile.	It is an important indicator of safety, as well as another indicator of the effective energy use, since a thrown fragment is a sign of energy wasted in ballistic rather than in fragmentation.
Cost of the Blast	*C*	*Cost/ m^3*	The cost analysis includes: · explosives: cost/kg of emulsion; · drilling: cost/drilled meter; · secondary breaking: cost/liter of fuel (hydraulic hammer); · fine to be separated before going to the primary crusher: cost/liter of fuel consumed along the deviation path to the sieving plant.	The last two factors were considered as representative of the first consequences of the outcome of the blast. The costs were referred to the bench volume (cost/m^3), and reported as unitary costs. For confidentiality, costs have been normalized to the cost of the first blast analyzed, taken as C=1; the costs of the other blasts are reported proportionally.

Figure 5. General view of the bench before (left) and after (right) the blast. The free surface, being left by the previous blast, dimensioned according to the proposed method, is smooth and slightly damaged. The muck-pile generated by the blast has a suitable geometry.

THE EFFECT OF DRILL AND BLAST PERFORMANCE ON TIME OF LOADING, SECONDARY BREAKAGE AND CRUSHING

The quarry under study uses three 25 t trucks for the transport of the material, and one excavator for mucking, hauling and dumping into the truck. In order to properly measure all the events, cycle times were recorded by using a stopwatch from a safe point of observation of the whole pit and the primary crusher (Figure 6). Data were recorded on specific sheets for each truck, allowing a single staff member to properly monitor all the operations relevant to this research.

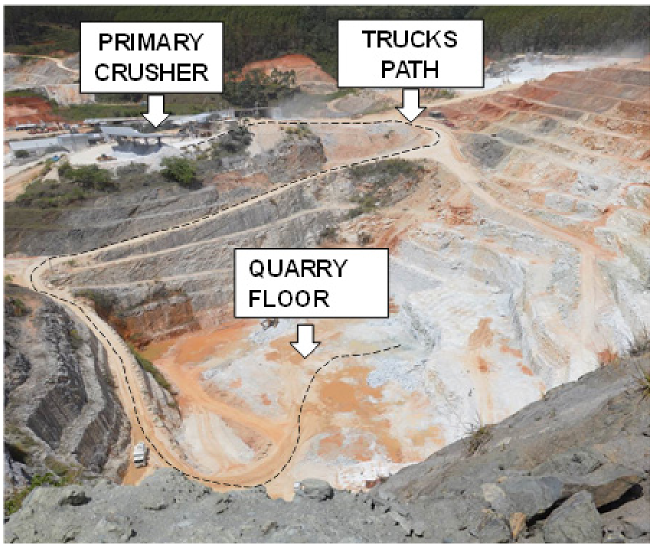

Figure 6. Overview of the quarry from the measuring point.

An analysis of cycle times was performed, considering different scenarios that contemplate different qualities of blasting performance. The quality of the performance is associated to the adherence to good practices of drilling and blasting execution, as supervised by the project team on site. It was evaluated the effect of poor blasting performance on the downstream process, quantified by the loss in terms of operational time when compared to regular or outstanding blasting performance.

The results show how the events that slow the cycle down happen on an irregular basis. From a managerial point of view, this creates a bias for decision-making: were the negative events are a regular issue, this would necessarily lead to a correction and a review of the adopted practices. Since the occurrence of deviating events is scattered along time and does not follow a regular sequence, delaying events are not perceived and therefore neglected. The better blasting practices introduced allowed for smaller and more evenly distributed cycle times, and reduced by an order of magnitude the usage of secondary breaking equipment.

The results led to state that the effects on the downstream process of small-scale operations are visible in the statistical analysis of cycle times. So far, it was achieved a proper indicator that allows to infer the issues that arises from improper blasting practices: the standard deviation of cycle times distribution. The analysis of the standard deviation measured in the distribution of cycle times show that, when exposed to material coming from poor blasting, the primary crusher is randomly fed with oversize material, leading to delays that have unpredictable consequences on the work cycle as a whole.

The main consequence observed are truck lines. Also, that cycle times associated to poor blasting practices are more irregular and unpredictable; cycle times associated to standard blasting practices lead to more uniform flows of operation in the cycle as a whole.

This work is important because it discusses the impact of such operational losses on an industrial environment characterized by high operational flexibility and lack of planning. Solutions are suggested and evaluated for the improvement of the small-scale mining process by Research Center for Responsible Mining team.

CONCLUSIONS

The Experimental Mine provides wide scenery about future tests and research, as characterized by a relatively simple organization of blasts and control of downstream conditions. Regarding the blast initiation, the organization of the circuit with detonating cord leads to various drawbacks. In particular, the risk of severing the circuit is common and is essential to avoid cross connection lines. The goal for the medium term is the shift towards shock-tube or electronic systems. This would allow the chance to experience various combinations of delay times. Another goal is to continue the recording of the drilling speed, in order to create a three-dimensional map of variation of geomechanical strength along the whole quarry. The influence of timing, within the possibilities offered by detonating cord, was examined in this study. As far as fragmentation and the importance of timing are concerned, results indicates that timing, leading to stress wave interaction, is important. Therefore, selection of delay timing can be of significant benefit to downstream processes as well as enhanced fragmentation itself. Blasting is an application of energy and energy distribution. Powder factor, distribution of charges and timing affect the outcome as a complex system. The work needs to be further improved by additional experiments, also with the aim of assessing the mutual influence between some important parameters, such as examining how blasting results are affected by the rock mechanics properties and the effects of charge distribution and initiation timing on the breakage.

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FRAGMENTATION IMPROVEMENT IN BLASTING AND ITS BENEFITS IN MINE AND PLANT OPERATIONS THROUGH TECHNOLOGY AND INNOVATION

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FRAGMENTATION IMPROVEMENT IN BLASTING AND ITS BENEFITS IN MINE AND PLANT OPERATIONS THROUGH TECHNOLOGY AND INNOVATION

ABSTRACT

This paper presents the benefits obtained in mine and plant operations through drilling and blasting optimization at Salobo Mine, Vale, in the last 20 months. When facing challenging environments and market conditions, mining companies search for cost reductions and productivity increase through the application of innovative technology in products and services, and new blasting techniques to add value in its operations. This paper focuses on improving fragmentation in ore zones, generating benefits in the downstream operations such as loading, transporting, crushing and milling. Implementation of high-energy explosives and electronic detonators, as well as the application of optimum blast designs and timing are key to improve productivity. The fragmentation improvement in ore zones has enabled the improvement of loading dig rates and trucks fill factors, increasing primary crusher availability by reducing oversize, and increasing overall throughput in the mill plant by increasing fines from blasts.

KEYWORDS

BLASTING, MINE TO PLANT, THROUGHPUT, ENERGY, EXPLOSIVES, FRAGMENTATION, PRODUCTIVITY, FINES.

INTRODUCTION

Salobo is an open pit copper mine with 15m benches. Mine operation counts on 12¼ inches drill holes and application of pumped explosives. Front shovels excavate the blasted ore and waste, and trucks transport them to the process plant and waste pile respectively. Ore goes to primary gyratory crushers, HPGR mills, ball mills and flotation.

The objective of this project is to optimize mine and plant processes by using Mine to Mill concepts. Fragmentation distribution changed due to drill and blast designs optimization. The effects of fragmentation in the downstream operations were measured and are reported in this paper.

In nonferrous metal mining operations it is generally necessary to crush and grind the ore to fine particle sizes to achieve effective mineral liberation. Decreasing ore grades are driving the pursuit of higher efficiencies in mineral processing, requiring ever-finer comminution (G.F. Brent, et al.2013).

In Mine-to-Mill optimization, the target blast fragmentation for each rock type can be determined by considering all downstream processes as a whole: Loading, hauling, crushing, and mineral liberation (K.Kum, 2012).

Various mine-to-mill studies have measured increases in mill productivity that are as described to improved fragmentation from the blasting operations (Scott et al. 2002, Rantapaa et al. 2005).

The focus in the mine is to reduce oversize and increase fines in the blasts to improve crusher and mill productivity. Introduction of high-energy explosives in the mine, and new blast designs deployment were critical to deliver the results.

Rock types and properties

Table 1 describes the existent rock types and its characteristics. Ore zones are basically represented by lithologies BDX, XMT, in most of the cases.

Table 1

Lithology	Avg. UCS (MPa)	SD (MPa)	Compressive Strength (MPa)
Biotita Granada Xisto – BDX	127	55	10 - 320
Xisto Magnético – XMT	144	64	4 - 290
Hematita Dura – HD	125	63	10 - 290
Granada Xisto – DGRX	173	78	10 - 300
Milonito – ML	133	57	20 - 280
Quartzo Milonito – QML	140	61	10 - 260
Diabásio – DB	179	50	50 - 250
Metavulcânica Básica – MTB	159	62	60 - 290

This mining company classifies the rock types depending on the ore grades and some of the characteristics, in order to define blast patterns. MSAT is a high-grade ore, MSMT a medium-grade ore and MSBT a low-grade ore. MSAT have ore grades higher than 0.8% of copper, and the ore feeds the process plant. MSMT and MSAT have from 0.3% to 0.8% copper. They feed the plant only when MSAT presents very high copper grades. Normally these materials go to a temporary pile, and will be processed at the end of the mine life. Table 2 presents drill and blast parameters according to this classification.

METHODOLOGY

The methodology to implemente the Project and analyse the results is the Mine to Plant. The objective of this methodology is to be constantly looking for opportunities to maximize plant treatment in an integrated way and to uncover new solutions that allow to successfully approach future challenges.

Concerning increases in the hardness of the rock at the blasting and plant processing stages due to the enlargement and deepening of the mine, which means that we are always performing a complete revision of all the process's steps (J. Alarcon, et al.2015).

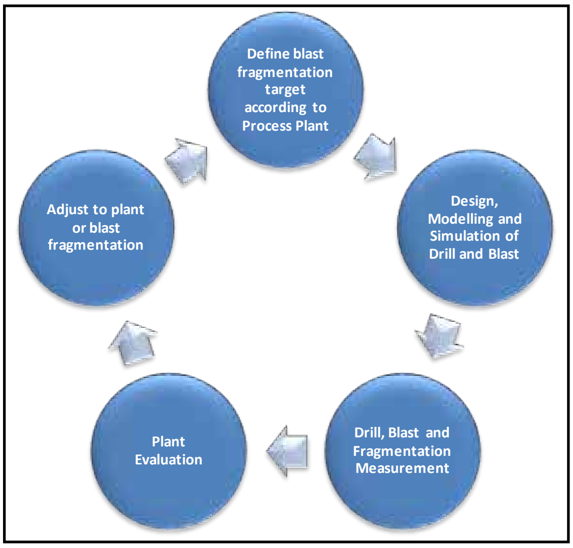

Figure 1: Mine to Plant Methodology

It was measured a baseline containing some key performance indicators (KPI's). The KPI and its values represent the scenario before the project implementation. It was measured the same KPI's during the implementation of the solution, in order to compare the results, which are shown further on.

Table 2: Baseline blast designs before project starts.

D = 12. ¼"	**Baseline Blast Designs 2014**					
Rock Classif.	Burden (m)	Spacing (m)	Sub-drill (m)	Stem. (m)	Area (m^2)	PF (Kg/m^3)
MSAT	5,70	6,50	2,00	4,50	37,05	2,02
MSMT	6,00	7,00	2,00	5,00	42,00	1,71
MSBT	6,00	7,00	2,00	5,00	42,00	1,71
MIS	6,00	7,00	2,00	5,00	42,00	1,71
OXI	6,00	7,00	2,00	5,00	42,00	1,71
Waste Rock	6,50	7,50	2,00	6,00	48,75	1,35
Waste Quart.	7,00	8,00	2,00	6,00	56,00	1,18
Waste Sapro.	8,00	9,00	2,00	7,00	72,00	0,83

Table 3 describes the explosives properties used in the project. Orica started the project with standard emulsions (straight and blended 70/30). Then it has migrated to more energetic explosives (straight and blended 70/30). All pumped emulsions based on ammonium nitrate. It considers standard emulsions for scenarios 1 and 2, and energetic explosives in scenario 3, as described in Figure 2. The explosives used during the Baseline stage are pumped emulsions based on ammonium nitrate. Its properties were obtained from the catalogue in the supplier's website.

Tabela 3: Explosives properties.

Explosives Properties	Blended Emulsion Baseline * 70/30	Straight Emulsion Baseline *	Standard Blended Emulsion Orica 70/30	Standard Straight Emulsion Orica	Energetic Blended Emulsion Orica 70/30	Energetic Straight Emulsion Orica
Cup density (g/cm3)	1.10-1.25	1.10-1.25	1.15	1.15	1.15	1.15
Average in hole density (g/cm3)	-	-	1.25	1.25	1.25	1.25
VOD (m/s)	5,000	5,500	6,562	6,478	6,632	6,577
Effective Energy (MJ/kg)	-	-	2.59	2.40	2,71	2.57
REE (%)	99-106	65-101	112	104	118	112
RBS (%)	128-165	95-103	174	162	184	174
Gas Volume (l/kg)	1,013	1,039	992	1,003	903	1,000

* Other explosives supplier

Figure 2 describes in-hole explosives and accessories distribution in three different scenarios.

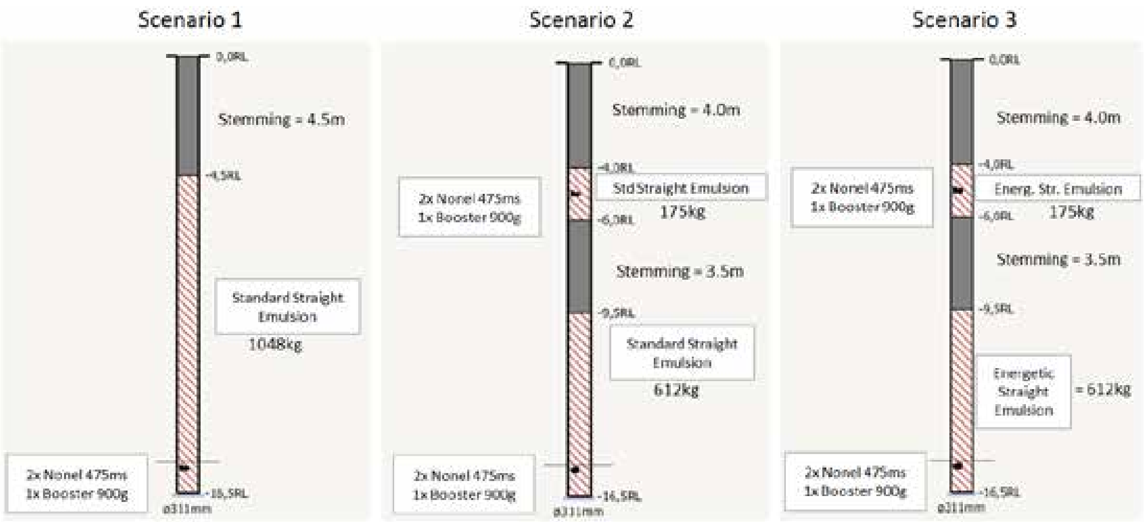

Figure 2 – In-hole loading configuration.

Scenario 1 represents the standard explosives charging design. Same design as Baseline, simply changing the explosive type and reducing sub drill by 0.5m. Stemming height of 4.5m, explosive charge length of 12m.

Scenario 2 represents the application of two decks of explosives, in order to increase fragmentation in the stemming region.

Scenario 3 represents the application of a higher energy explosive, using the same design as scenario 2, in order to increase fines to the process plant.

RESULTS AND DISCUSSIONS

Table 4 describes the new blast designs applied during the project. Relevant differences in blast patterns (area in m²) and powder factor (%) in comparison with Baseline parameters. It generated drill and blast savings and delivered an adequate fragmentation to the mill plant due to the increased explosives energy and better in-hole energy distribution.

Table 4: Blast designs during Project implementation.

D = 12. ¼"	Current Designs 2016						Difference	
Rock Classif.	Burden (m)	Spac. (m)	Subdrill (m)	Stem. (m)	Area (m2)	PF (Kg/m3)	Area (m2)	PF
MSAT	5,00	7,50	1,50	4,50	37,50	2,02	1,21	-5,15%
MSMT	6,00	9,00	1,50	5,00	54,00	1,28	28.57	-25,46%
MSBT	6,00	9,00	1,50	6,00	54,00	1,17	28.57	-31,94%
MIS	6,00	9,00	1,50	7,00	54,00	1,06	28.57	-38,43%
OXI	6,00	9,00	1,50	7,00	54,00	1,06	28.57	-38,43%
Waste Rock	6,00	10,00	1,50	7,00	60,00	0,95	23,08	-29,83%
Waste Quartz.	6,50	11,00	1,50	8,00	71,50	0,71	27,68	-39,48%
Waste Saprol.	7,00	13,00	1,50	9,00	91,00	0,49	26,39	-40,66%

Effective productivity of shovels increased by 4.0% (Shovel type 1) and 8.5% (Shovel type 2) as shown in Figure 3. There is an increase of rock mass hauled per trip increased by 0.4% to 2.6%, according to the type and capacity of the trucks, as shown in Figure 4. It represents the positive effect of double decks of explosives over the fragmentation in the stemming zones and the generation of fines due to explosives energy.

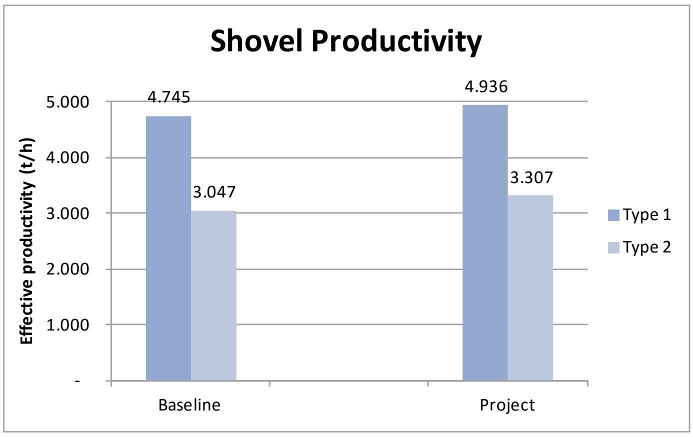

Figure 3 – Effective shovel productivity.

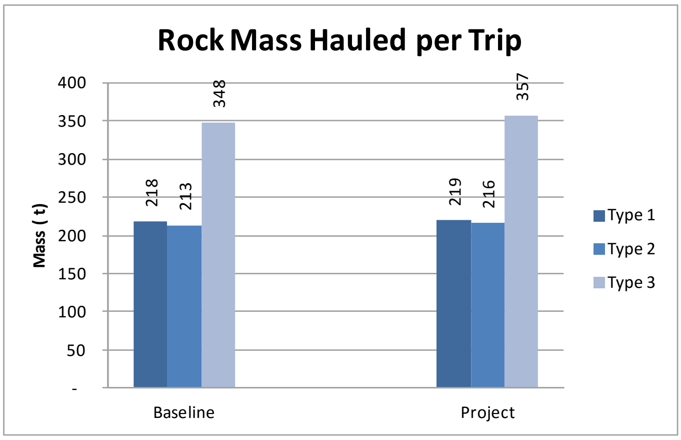

Figure 4 – Rock mass hauled per trip.

Figure 5 represents the process plant flowsheet at Salobo. Consists in primary gyratory crushers, secondary cone crushers, roller press (HPGR), ball mills, flotation, regrinding, dewatering and filtering.

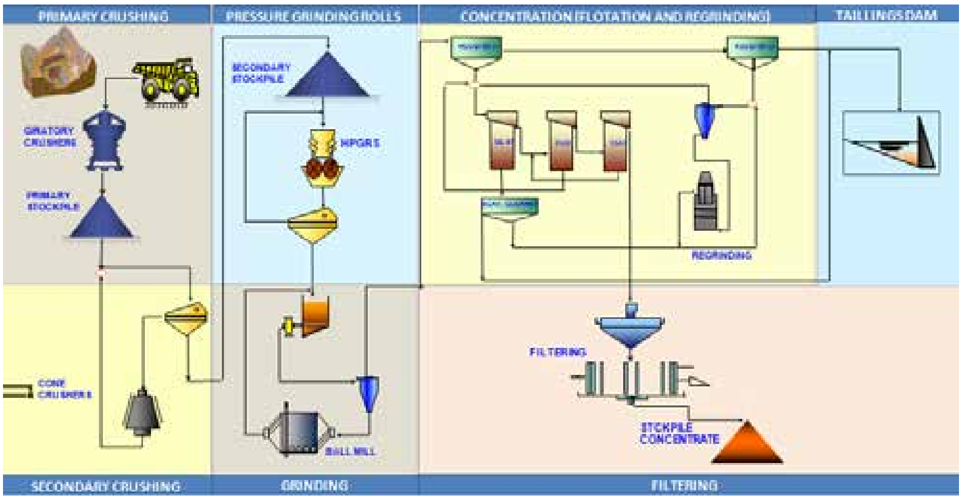

Figure 5: Process plant flowsheet at Salobo.

Different blast designs, blast patterns, and energetic products applied in the blasts are evaluated in the plant using the Mine to Plant methodology. KPI's monitored in primary crusher are stoppage time due to oversize (events per 100Ktons ROM fed), efficiency of the rock breaker in the crusher, and maintenance improvements.

Fragmentation improvements in blasting have contributed to reduce P80 and top size, which enabled an improvement in primary crushers' performance.

Two primary gyratory crushers operate with 6 inches open size setting. Due to fragmentation improvements, reduction of top size and P80 it could be reset to 5½ inches, in order to reduce P80 of the ore to feed secondary crushers and improve subsequent processes.

The quantity of events of primary crusher stoppages is monitored weekly. The reduction of events has contributed to increase primary crusher availability. In this case, the additional available hours were separated considering three main subjects: maintenance improvements, rock breaker improvements, and fragmentation improvements in blasting.

Figure 6 represents the reduction of stoppages in primary crushers due to oversize (events per 100Ktons). Figure 7 indicates the additional production considering each improvement.

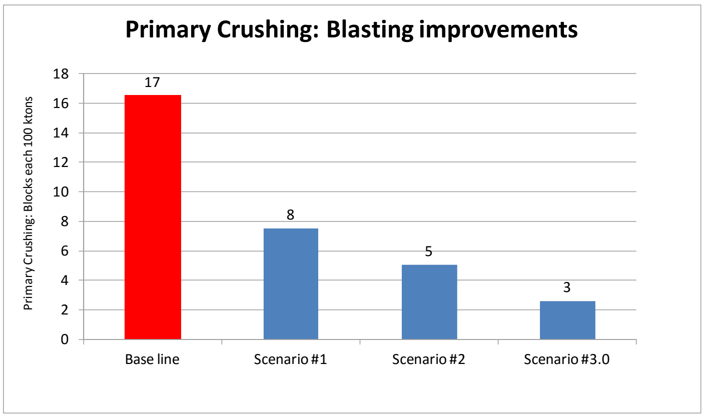

Figure 6: Primary crusher – Reduction of stoppages due to oversize

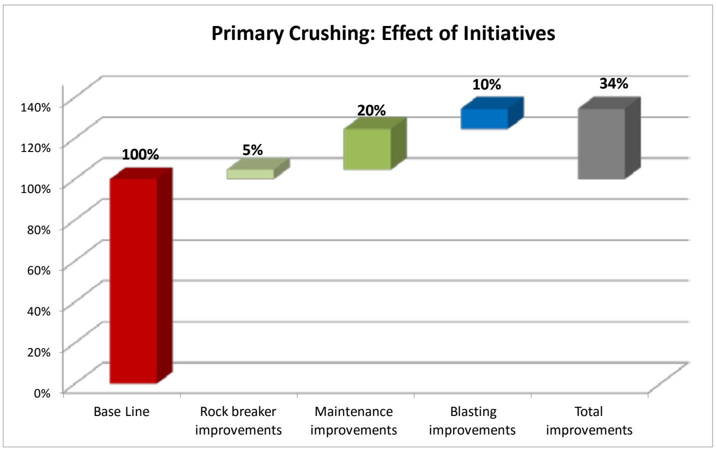

Figure 7: Primary crusher – Production Increase

The three aspects have contributed with 34% additional production in primary crushers, which represent 43Ktons per day (dry basis). The effect of fragmentation itself represents 10% of the additional production, or 4.3Ktons per day.

In the plant, three KPI's related to fragmentation are measured: throughput (tph) of fresh feed ore, percentage of circulating mass in HPGR mills (roller press), and percentage of fines below 8 mm in fresh feed ore. Through the application of high-energy explosives, there is an increase in fines generation, reduction in the circulating mass in HPGR mills, and increase in mill throughput.

Figure 8 represents a statistical empiric model relating plant throughput with the percentage of circulating mass in HPGR.

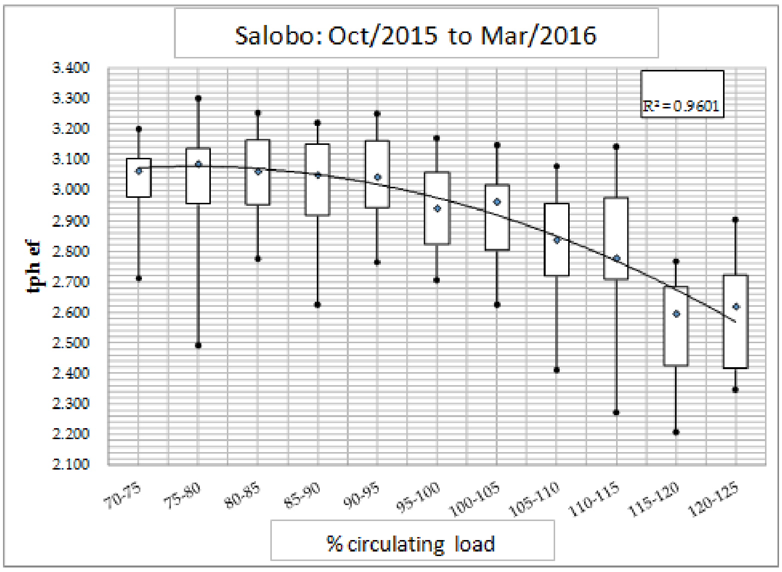

Figure 8: Correlation between plant throughput and circulating mass. Empirical model.

Figure 9 presents the fragmentation results after blasting in the three different scenarios, as described. Blasting with double decks of explosives indicates improvements in fragmentation in the stemming zone, by reducing top size and P80. It is noticed by the reduction of stoppages in primary crusher (scenarios 2 and 3).

Blasting with a single deck of explosive charge (scenario 1) indicates an increase in the percentage of fines. This effect can be demonstrated by the reduction of the circulating mass in the roller press (HPGR) and the percentage of fines below 8 mm in fresh ore feed.

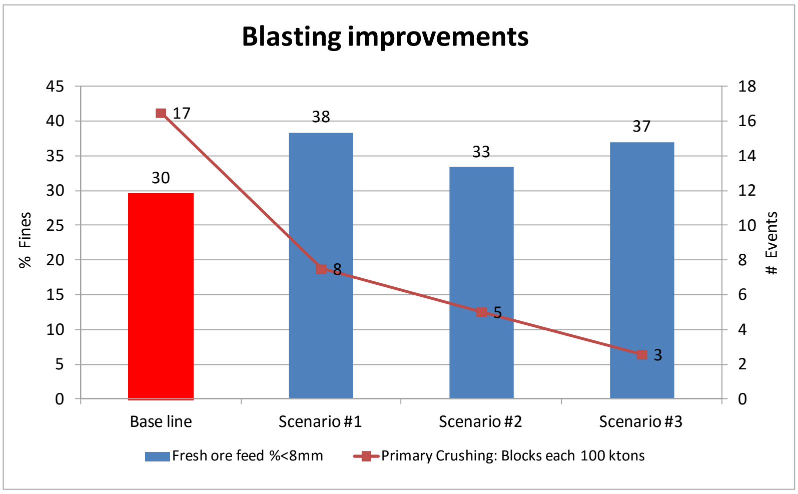

Figure 9: Blasting improvements – Oversize and Fines

CONCLUSIONS

The use of high-energy explosives and innovative blast designs and techniques allow improving rock fragmentation by blasting.

The use of double decks of explosives charge, reducing the stemming length on the top of the hole represented an improvement in fragmentation, particularly the coarser zone of the curve (top size and P80). This meant an increase in the average shovel productivity of 5.7% and rock mass transported per truck per trip of 2.6%. In the plant there is a 10% increase production in the primary crushers due to the reduction of oversize material, representing additional 4.3Ktons of ore per day.

An increase of 8% in the explosive energy generates more fines in blasts. It also compensates the effect of the intermediate stemming deck. Results indicate a 12% increase in fines, which affects positively the reduction in the circulating mass in the HPGR circuit and the increase of throughput.

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GEOREFERENCED MONITORING SYSTEM FOR OPEN PIT MESH NETWORKS

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GEOREFERENCED MONITORING SYSTEM FOR OPEN PIT MESH NETWORKS

ABSTRACT

Nowadays, Open Pit Mines need to manage several applications in order to increase safety and optimize their production (such as the Anti-Fatigue Systems and the Mine Fleet Management Systems) and all of them depend of an underlying network infrastructure. Wireless Mesh Networks are widely used as the platform where all this systems can operate, thus it is essential to monitor the Mesh Network signal coverage. This paper tackles the troubleshooting of the coverage holes that are constantly formed by the continuous changes in geography. The goal is to know where the vehicles are losing connectivity in order to prevent network issues. There are several Network Monitoring Systems such as the Solarwinds Network Performance Monitor that stores different connection parameters from devices, and shows them through time, unfortunately these systems do not provide us the geographic locations of the vehicles. This paper presents a Network Monitoring System that shows the most important signal parameters from the Mine Fleet Vehicles on a Map (using different colors to distinguish between the different signal quality levels) and through time (using Signal Strength Vs. Time Graphs). This approach let us know immediately where the signal coverage is weak, null or corrupted (due to channel interference), making the troubleshooting straightforward. We have developed the entire system in Cerro Verde Mine. The user interface was designed as a Web Application and the server side was written as a Service that basically uses the SNMP protocol to collect the signal parameters from each vehicle, and uses the PI AF SDK to extract all the vehicle's geographic coordinates from our PI server (from OSIsoft). This Georeferenced Monitoring System is already running at Cerro Verde and gives us a constant source of information that we can use as a feedback to continuously improve the signal coverage on the Open Pit.

KEYWORDS

Open Pit Mesh Networks, Georeferenced Network Monitoring System, Simple Network Management Protocol (SNMP), PI Asset Framework Software Development Kit (PI AF SDK)

INTRODUCTION

At Cerro Verde the most important mine systems hang on the Wireless Mesh Network, this is why it is crucial to guarantee full wireless coverage on the most important places of the pit such as the banks where the shovels work, the main roads of the haul trucks, the dumps, etc. As we can see when we think about wireless coverage we immediately associate geographic places, but the fact is that when we think about the best known Network Monitoring Systems we associate Network Oriented Line Graphs. This was the motivation for us to develop a map oriented Network Monitoring System. At the present time, we no longer need to carry out periodic drive tests, because this system gives us a quick view of the wireless coverage in the Pit and it tells us on time when a maintenance is needed.

The approach presented in this paper takes full advantage of the existent MEM (Mobile Equipment Monitoring) methodology that Freeport-McMoran has adopted since 2008 which basically consist of a set of data-loggers (one for each haul truck and shovel), that send all the vehicle's performance related data to a centralized server which in turn is connected to the PI server of Cerro Verde (Larson, 2013). In short the system that was developed gathers the geographic coordinates and the operating status of the mine fleet vehicles from the PI server of Cerro Verde and gathers all the wireless signal related data directly from the access points of the vehicles through SNMP protocol.

METHODS

On this section we first briefly explain the operating principle of the system, and then describe how the client and the server side of the system were developed.

Georeferenced Monitoring System Operating Principle

The following paragraphs roughly describe the operating principle of the system. Please refer to Figure 1 for better understanding.

The PI server receives each second all the data related to each vehicle (such their geographic coordinates, their load status, etc) through the Mesh Network.

The Georeferenced Monitoring System Server gathers all the signal coverage data from each Access Point, also through the Mesh Network. It then connects to the AF PI Server and request the geographic coordinates and status from each mine vehicle. Then the AF PI Server immediately starts to pull this data directly from the PI server, and at the same time it starts to send back this information to the Georeferenced Monitoring System Server. Finally all the data is stored in a local SQL Database.

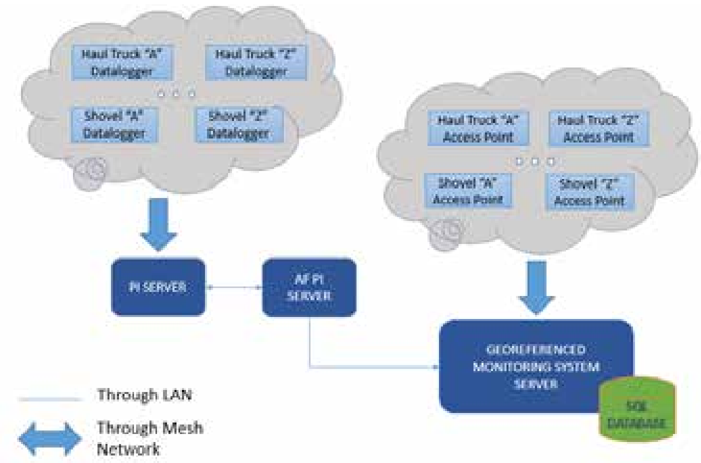

Figure 1 – Operating Principle Diagram

Client Side

The client side of the system was designed to be free of add-ons and plugins, for this purpose it was written using only web standard languages (HTML5, CSS3 and Javascript). The map engine uses the Javascript Google Maps API that provides great control over the markers functionality and even allows us to overlay a custom topographic map of the Pit. For the Time versus Signal Strength Graphics we first thought of using a well know Javascript based Graphic Library as D3.js or Chart.js, but because of our very specific requirements, we ended up writing our own Chart engine using only HTML SVG (Scalable Vector Graphics) elements.

Server Side

The server side was written entirely in C# using NET Framework 4.0 and is divided in two parts, the first one and the most important is a Windows Service that gathers data from each vehicle (Access Point signal parameters, GPS coordinates and the vehicle's operating status) and stores this data on a SQL

Data Base, and the second one is the Web Application that handles all the users requests and connects to the Active Directory Database from Cerro Verde for authentication purposes, restricting access to only authorized users.

Signal Coverage Data Gathering

All the signal coverage parameters related to Mesh Networks as signal strength, SNR (Signal to Noise Ratio), connected up time, parent Access Point, etc. are collected through the SNMP protocol (version 2 community based) using only Get-Next request messages. In order to reduce the wireless bandwidth usage, it was important to make sure that each SNMP request embrace all the requested OIDs (SNMP Object Identifiers) corresponding to the signal coverage parameters (Harrington, Presuhn & Wijnen, 2002). For a detailed description of the SNMP version 2 protocol please refer to the publication of Case, Fedor, Schoffstall and Davin (1990) and the publication of Presuhn (2002b).

It was first used the Open Source SnmpSharpNet Library for C# in order to facilitate the SNMP operations, and then we decided to write our own SNMP v2c library, in both cases, the best approach to carry out all the requests was to use a dynamic thread pool.

Vehicle's GPS Coordinates and Operating Status Gathering

The main source of the geographic coordinates and the Dispatch Status of the mine fleet vehicles is our PI server that is constantly being feed by all the Matrikon Data Loggers that each Haul Truck and each Shovel has.

It was also decided to gather the Operating Status from mine fleet vehicles to distinguish the haul trucks and shovels that are actually working from the vehicles that are stopped or under maintenance. The source of this parameter is actually each vehicle's mine fleet management device.

At the beginning we used the PI SDK for C#, then we started using the AF PI SDK for C# because it provide us more speed and performance and because the PI SDK is being deprecated (Osisoft, 2011; Verhoeff, 2015). Therefore the Windows Service extracts all this information through a multi-threaded based approach that uses the AF PI SDK for C#.

It is important to note that the PI Server stores all the data from vehicles in Tags Elements as we can see in Figure 2 whereas the AF Server pulls this data from PI Server and shapes this information into objects as we can appreciate in Figure 3 (Isolutions Inc., 2012; Osisoft, 2011).

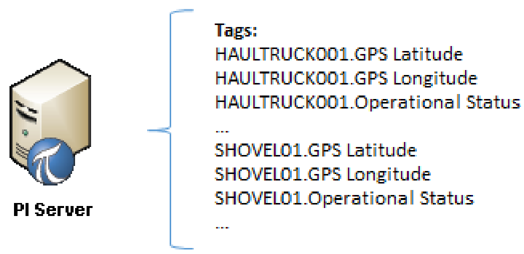

Figure 2 – PI Server Tags Elements

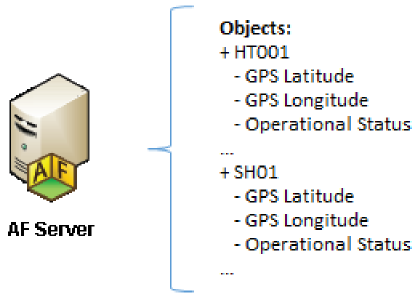

Figure 3 – AF Server Objects Elements

Client and server Communication

The only mean of communication between the client and the server side was AJAX (Asynchronous Javascript And XML) technology, which is also standardized and allow us to re-write if necessary the whole server side in another language without involving any changes on the client side.

RESULTS

On this section we will present a set of tables indicating which parameters we extract from PI server and which values we extract from each access point, then we show other tables comparing the performance of the different methods that were tested. Finally we show a set of images that demonstrate the basic usage from the georeferenced monitoring system.

As we can see in Table 1, three parameters are extracted from the AF/PI Server, the latitude and longitude from the mine fleet vehicles and their Operating Status which basically tell us if the corresponding vehicle is actually working or not.

Table 1 – Parameters Extracted From AF/PI Server

Name	Value	Unit
GPS Latitude	-90<=x<=90	Decimal grades (°)
GPS Longitude	-180<=x<180	Decimal grades (°)
Operating Status	Operative, Stopped	----

On Table 2 we can appreciate the OIDs that are gathered directly from the access points of the mine fleet vehicles. For more information about SNMP OIDs, their function and their value types, please refer to the publication of McCloghrie, Perkins and Schoenwaelder (1999) and the publication of Presuhn (2002a).

Table 2 - OID Values Extracted From Access Points

OID Description	OID Name	OID Value Type	OID Value Range
Hostname	1.3.6.1.4.1.9.2.1.3	Octect String	----
MAC Address	1.3.6.1.2.1.2.2.1.6	Octect String	----
Model	1.3.6.1.2.1.47.1.1.1.1.13	Octect String	----
Signal Strength	1.3.6.1.4.1.9.9.273.1.3.1.1.3	Signed 32 bit Integer	-100<=x<=0
SNR	1.3.6.1.4.1.9.9.273.1.3.1.1.4	Unsigned 32 bit Integer	0<=x<=100
Parent Name	1.3.6.1.4.1.9.9.273.1.2.1.1.13	Octect String	----
Channel	1.3.6.1.4.1.9.9.272.1.1.2.5.1.3	Signed 32 bit Integer	1<=x<=14

Bellow on Table 3 we compare the mean time required to pull the corresponding data from the AF/PI Server for 100 vehicles, using the PI SDK and the AF PI SDK.

Table 3 - Mean AF/PI Server Data Pulling Speed for 100 vehicles

Software Development Kit	Mean Data Pulling Speed (seg.)
Using PI SDK	2.3
Using AF PI SDK	0.8

Finally on Table 4 we compare the mean time required to pull the corresponding data from the access points of 100 vehicles using the well known Open Source Library SnmpSharpNet and our SNMP Implementation.

Table 4 - Mean SNMP Data Pulling Speed for 100 vehicles

Snmp Implementation	Mean Data Pulling Speed (seg.)
Using SnmpSharpNet Library	2.1
Using our SNMP v2c Implementation	1.9

Now we will show the final result of the Georeferenced Network Monitoring System through images. The system has basically two main uses, the first one consist of seeing the last reported georeferenced signal coverage data from all the vehicles from the mine fleet, as we can appreciate on Figure 4.

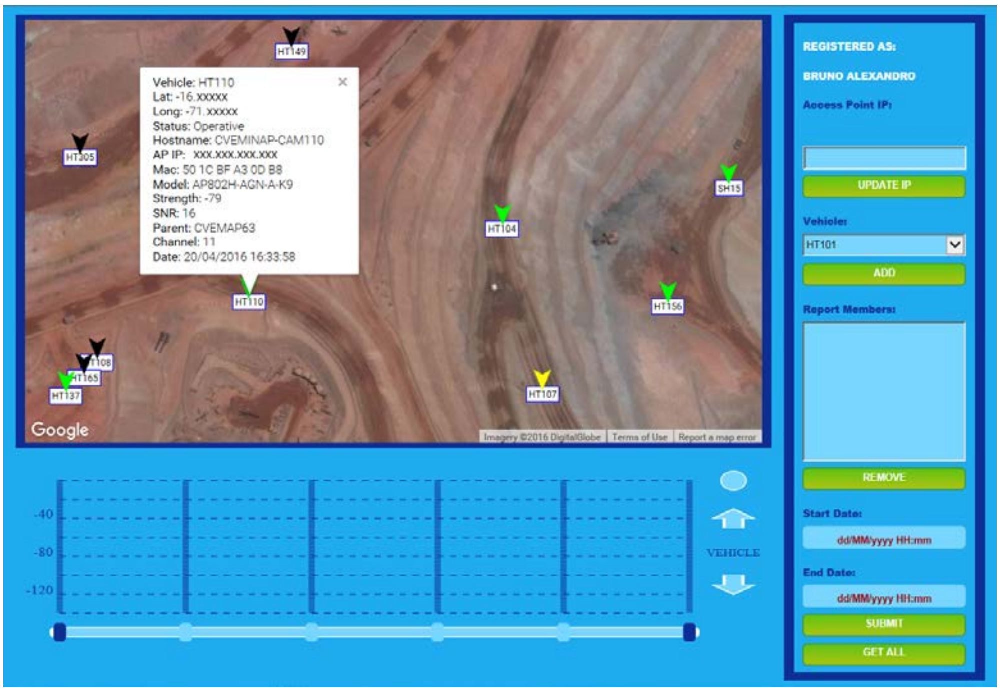

Figure 4 – System main functionality let users check the last reported signal coverage data

Four different colors were used in order to distinguish between the different levels of signal quality. These different levels of signal quality were based on the signal strength reported by each vehicle, and the range of values was defined by experience. Please refer to Table 5 in order to see the range of values corresponding to each signal quality level. Note that the black color correspond to the case when the SNMP request times out.

Table 5 – Signal Strength Range Values

Quality Level	Value (dBm)	Color
Good	0>=x>-70	Green
Acceptable	-70>=x>-80	Yellow
Bad	-80>=x>=-100	Red
Null	----	Black

The second main use of the Georeferenced Network Monitoring System is to show the behavior of the signal coverage of the different vehicles through time, as we can appreciate in Figure 5.

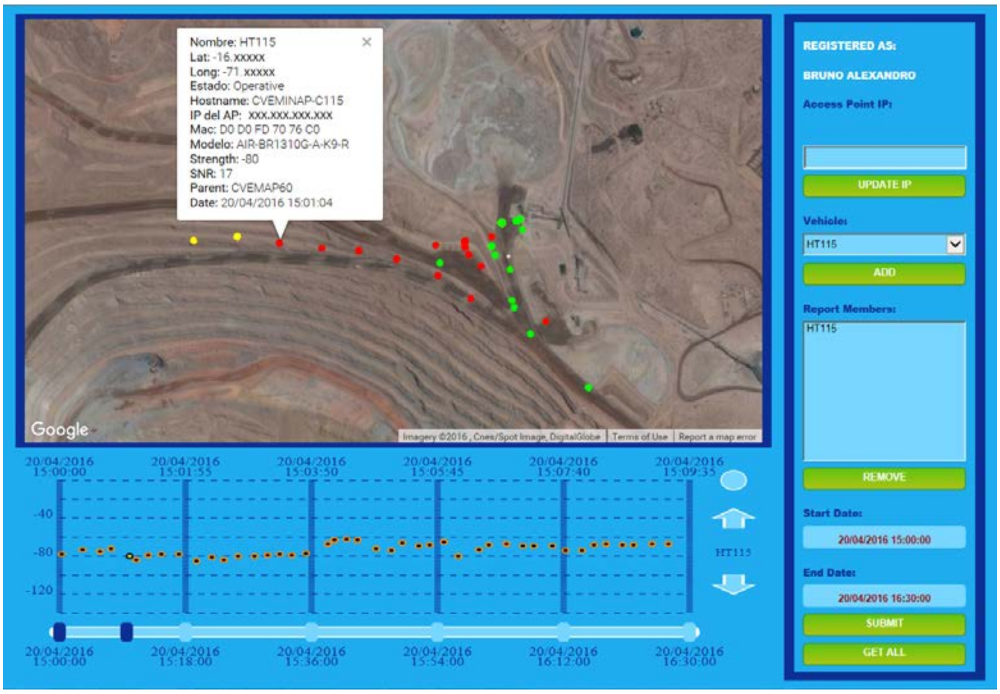

Figure 5 – System second functionality let users see the signal coverage behavior of vehicles through time

DISCUSSIONS

On the Results section we have first tested different approaches in order to optimize the Georeferenced Network Monitoring System performance. This also helped us to learn more about the different technologies that allowed us to design and develop this system.

Then we could see that our system is very useful to quickly see the current state of the wireless network coverage all over the pit. This main functionality at the present time allow us to know exactly where the coverage holes are, and therefore it also allow us to fix those problems faster. The second functionality of our system basically help us to distinguish if any problem was caused by any wireless network issue or not. This is also important because not all the problems related to the functionality of the main Mine Applications are caused by network issues, but many times they are generated by application's software bugs, hardware failures and power supply issues.

Another approach to accomplish a Georeferenced Network Monitoring System would be to program the Data Loggers in order to collect the signal coverage data directly from them. The problem would be that any necessary change of the system would involve the reconfiguration of all the data loggers, which would be at the end very impractical. Also we have to consider that a data logger wouldn't collect any additional information about the signal coverage status in comparison with a dedicated server, this is because when an Access Point doesn't have wireless coverage it simply lacks of useful information.

CONCLUSIONS

This Georeferenced Network Monitoring System is a new approach with many advantages over the current common Network Monitoring Systems, because it shows the signal coverage data on a map.

The time that normally takes to find out where exactly any wireless related problem is, decreases significantly with the usage of this Georeferenced Monitoring System.

The extraction of the Operational Status of the mine fleet vehicles is also very convenient, because let us distinguish when a vehicle is really having wireless network issues, or when it is under maintenance or stopped and the access point is turned off.

It is important to note that this system was developed without spending any money on hardware other than the system's server, because we had already all the other technologies deployed. This could be the case of much more other Mines that can test a very similar approach.

ACKNOWLEDGEMENTS

This work would not have been possible without the support of Cerro Verde Mine and Freeport Corporation that provided us all the necessary tools in order to develop this Georeferenced Network Monitoring System. A special thanks to all the members of the IT Department of Cerro Verde, without their ideas and criticism, the goal would not been achieved.

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GEOSTATISTICS APPLIED TO GEOMETALLURGICAL MODELING

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GEOSTATISTICS APPLIED TO GEOMETALLURGICAL MODELING

ABSTRACT

Many factors influence on ore processing efficiency and a better understanding of these aspects and their impact on the processing plant can help to improve the ore recovery. The construction of a geometallurgical model is fundamental for achieving this objective, since the knowledge of ore properties allows a more accurate forecast of mass recovery by the process, improving mine planning. Most geometallurgical variables are non-additive, i.e., the output value from a combination of samples does not depend only of the values and masses of the initial samples, but also from a complex relationship with other variables. Due to these complex relationships, it is not recommended the use of conventional estimation methods like Inverse Distance to a Power or Ordinary Kriging (OK), once this estimates use a linear weighted average. Non-linear geostatistical methods were developed to estimate a local probability distribution of possible values for a variable. Among these methods, Indicator Kriging (IK) can be used to estimate the probability of a block to be above (or below) a determined cut-off or the likelihood to belong to a certain class or category. This study uses IK combined with the information from the geometallurgical tests to build a short term block model of a phosphate mine at Vale Fertilizantes S/A. It is expected the use of a geometallurgical model for the mine planning will improve the recovery prognostic of each selective mining unit and may be used as a tool to help in the decision making process beyond the simple cut-off grade on the P_2O_5 grade. At the end, the model generated by the linear methods (OK) is compared against the model proposed using IK. The prevision by the IK geometallurgical proved to be accurate and the results were compared against production figures.

KEYWORDS

GEOMETALLURGY, GEOSTATISTICS, INDICATOR KRIGING, NON-ADDITIVE VARIABLES

INTRODUCTION

Ore characterization is mandatory in all mining projects. The results normally obtained from the pilot plant can integrate the models used for predicting the processing plant recovery. This process is referred as geometallurgy (Braga, 2015).

In this scenario, geometallurgy provides the interaction among physical/chemical rock properties such as mineral assemblage, hardness and chemical composition, with process variables including mass recovery and energy consumption. Therefore, it is possible to estimate with accuracy and reasonable precision the process performance, helping to improve mine planning and project risk evaluation (Mendonça, 2015).

The incorporation of geometallurgical variables in short term block model helps in predicting processing plant efficiency. However, it is necessary to be cautious when estimating geometallurgical models. Mass recovery and concentrate grades are non-additive variables, i.e., their values do not average linearly.

Inverse Square Distance and Ordinary Kriging (Matheron, 1963) are estimation techniques that use a weighting average to combine sample values to estimate a block depending on its distance and covariance, respectively. Since results are based on the weighted average of the data without considering their relationship with other variables or their non-additivity, the use of such estimation technique is not recommended for building a geometallurgical model. Conversely, indicator kriging (Journel, 1982) estimates probabilities by a categorical transformation of a dataset based on cutoff grades or thresholds. As a result, it is possible to derive at each block the probability to be above or below a determined cut-off. This paper evaluates the applicability of indicator kriging as estimation method for building a geometallurgical model of a phosphate mine located at a carbonatite complex in central Brazil.

METHODOLOGY

The processing plant efficiency is probably one of the most influential aspects to improve profitability in a mining project. Due to this fact, it is vital to have a deep understanding of the geometallurgical aspects of the ore, as they directly affect recovery. Through the application of a methodology to estimate the geometallurgical model, it is expected to be possible to predict the processing plant performance and introduce data to assist mine planning.

The carbonatite complex used in this case study belongs to an ultramafic-carbonatitic alkaline intrusion related to ultrapotassic intense magmatism of the Upper Cretaceous (Gibson et al., 1995). The complex is composed by silicate (predominantly ultramafic), carbonatite and foscoritic rocks containing significant phosphate and titanium deposits, which are currently being mined for apatite (Brod et al. 2000). The apatite (P_2O_5) and anatase (TiO_2) deposits are located at the weathering mantle on these alkaline rocks. The supergene concentration of these minerals is given by solubilization and leaching of the most mobile components contained in the original rocks. Moreover, in shallower horizons, apatite was partially transformed into secondary phosphate. However, in deeper portions, it remained in the weathering mantle as resistive mineral. The anatase is a product of weathering originated from the decalcification of perovskite ($CaTiO_3$) in the original rock.

In this case study, the prior data separation into domains precedes the estimation. The classes were separated according to the weathering zones, since for this kind of deposit geometallurgical behavior is directly related to the weathering intensity, the domains description is shown in Figure 1, presented in a typical cross section.

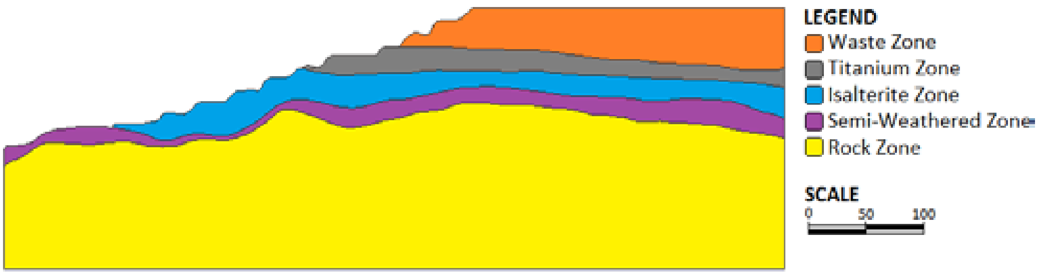

Figure 1 – Weathering Zones Separation.

This study starts by geometallurgical tests to obtain data. A pilot plant mimics the processing plant on a smaller scale, in order to provide indicators of process recovery and concentrate grades. The variables considered for this case study was the concentrate grade, also the metallurgical and mass recovery, as shown in Table 1. The dataset is heterotopic comprising 5369 samples. Univariate statistics for each variable is depicted in Table 2.

Table 1 – Variables and their description.

Variable Name	Description
P_2O_5CON	Apatite Concentrate
RECTOT	Metallurgical Recovery
RMTOT	Mass Recovery

Table 2 – Samples basic statistics.

Variable Name	Number of Data	Minimum	Maximum	Mean	Standard Deviation
P_2O_5CON (%)	2434	11.50	38.00	34.59	2.70
RECTOT (%)	5369	1.77	99.32	56.49	14.10
RMTOT (%)	5369	0.78	55.10	14.98	6.32

Since the samples of the concentrate grade have different mass recoveries, it was necessary to transform it into accumulated variables, by weighting each grade sample in function of its mass.

Knowing that these geometallurgical are non-additive was proceeded the estimative by indicator kriging, which transforms the original dataset (u_a) into categories based on threshold limits (Z_k) according to, as shown in Equation (2). In this case, the cut-off grades were chosen considering the deciles of the distribution.

$$i(u_a, Z_k) = \begin{cases} 1, & Z(u_a) \leq Z_k \\ 0, & Z(u_a) > Z_k \end{cases} \tag{1}$$

The spatial continuity analysis started with the indicators defined for the median (Q50). As all the variograms have the same form and spatial continuity major axes, with large continuity for low grades and short for the high grades, the median model was applied with a range reduction factor for the higher quantiles, and an increase factor for the lower quantiles. It was not possible to use domains separation for the spatial continuity analysis due to the small number of samples within some domains in the dataset. The multiple indicator probabilities were kriged and the E-type model derived from each estimated block conditional probability distribution function.

The E-type mass recovery estimates were validated using swath plots, comparing sample averages against block averages within regions and histograms reproduction. These validations proved to be ok and the models were accepted. The next stage was to compare the block models derived from IK and OK and evaluate the geometallurgical model constructed against production results.

RESULTS AND DISCUSSION

Average Difference Analysis

The average difference measures the discrepancy of two values. In this case study, the difference was calculated for each block value (n blocks in the model) from the IK and OK estimates, with the objective to identify if there is a significant divergence between the methods. An average difference near to zero indicates the methods lead to similar results on average, while very negative or positive results shows substantial discrepancy between the methods. The average difference was calculated using Equation (4):

$$Average\ Difference = \frac{1}{n}\sum IK_{estimate} - OK_{estimate} \tag{2}$$

The differences should have average error close to zero and minimum spread, i.e. the average close to zero (unbiasedness) is not sufficient, since high and low magnitude errors can compensate each other (need also to be precise). The average difference histograms for the three variables considered in this study are displayed in Figure 2.

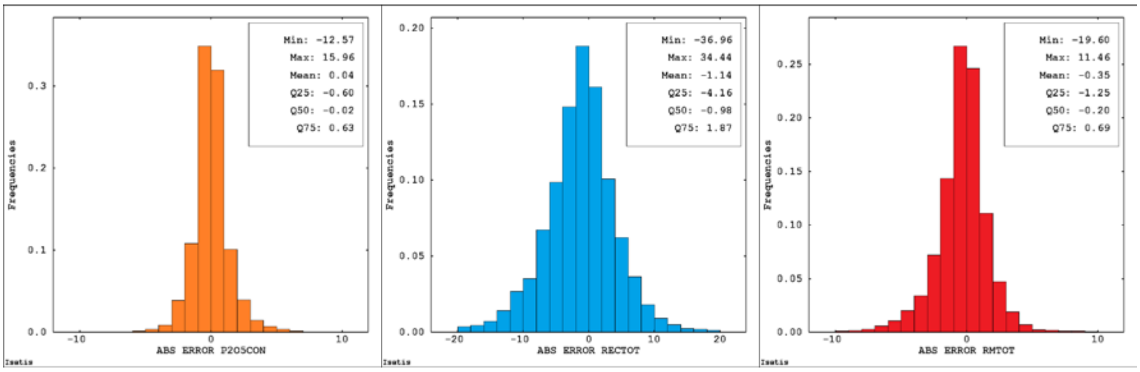

Figure 2 - Absolute differences for the variables P_2O_5CON, RECTOT and RMTOT, respectively.

Significant differences in the two estimation methods were detected. Mass recovery (RECTOT) presented an average difference of -1.14%, indicating that the blocks were overestimated by OK, when compared to the IK estimates. From the histogram of differences, it is noticed that 50% of the blocks presented difference exceeding 1.87%. Considering the importance of mass recovery at the processing plant, these errors were considered significant.

The metallurgical recovery (RMTOT) presented an average difference near zero, not showing global difference between the two models. Even tough global differences were not noticed, 25% of the blocks error exceeded 1.25%. The apatite concentrate (P_2O_5CON) also showed differences, but less significant. Although this comparison is not capable to identify which estimation method is more accurate, it shows that ignoring non linearity and non-additivity can significantly affect geometallurgical variable estimates.

Estimated Grades vs. Real Grades

This analysis is introduced as a reconciliation scheme between the grades obtained from the processing plant (herein referred as real grades) and the grades estimated by the kriged models. It is expected that the IK and OK estimated block models approximate on average the real grades.

Ten planned blending piles were selected along 2015 production, and it was compared the average grade of all blocks forming each pile against the real grade. The closer to zero the difference is, the more similar the real and estimated grades are.

It is important to observe that in this study case there is no sampling at the blending piles. Therefore, it became impossible to detect if the differences founded between the estimated and real grades are due to mine planning deviations, processing plant inefficiency or the estimated method itself. This fact limits some of the analysis but do not invalidates the comparison made.

IK models approximates better the metallurgical recovery (RMTOT) for 70% of the analyzed piles and was more accurate than OK. The average relative error reaches -10% for IK prediction, while OK to -20%. The reconciliation results for each planned pile tested can be seen in Figure 3.

For mass recovery (RECTOT) both kriged models led to similar results. Their reconciliation results are depicted in Figure 4. For the apatite concentrate (P_2O_5CON), 60% of pile grades were better approximated by IK derived block grade estimations (Figure 5). The average relative error was -1% with IK, while the OK presented a relative error of -2%.

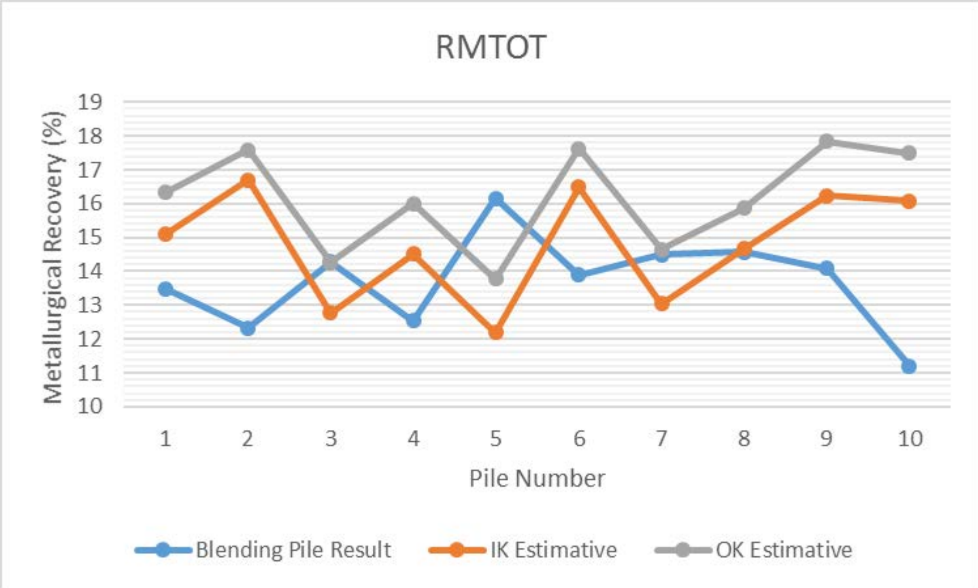

Figure 3 - Metallurgical recovery reconciliation results.

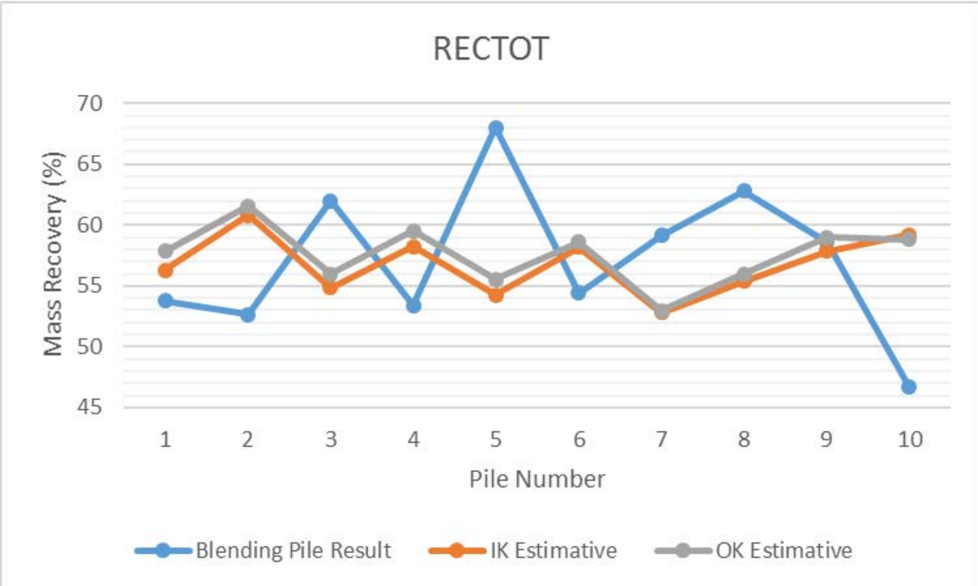

Figure 4 - Mass recovery reconciliation results.

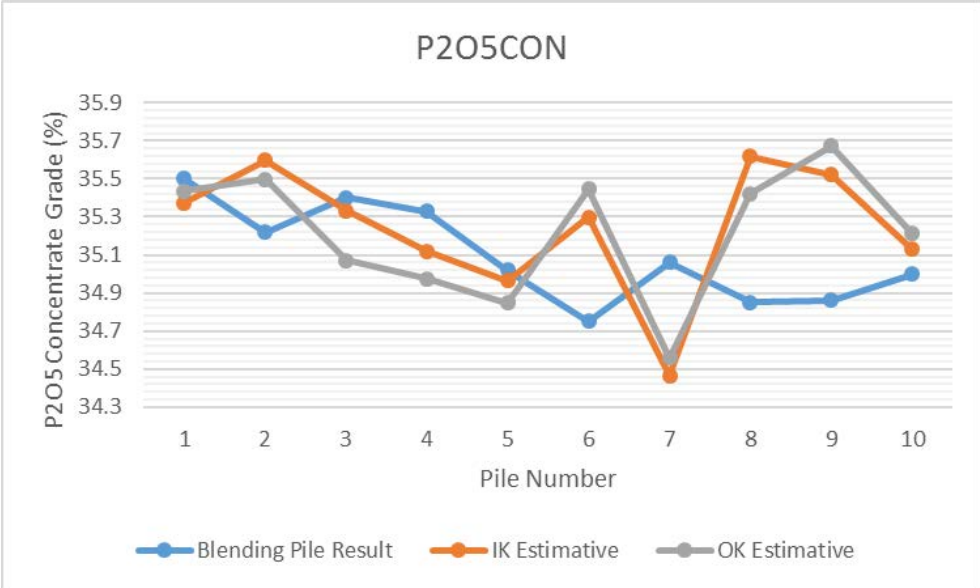

Figure 5 - Apatite concentrate grades reconciliation results.

It is possible to observe a considerable difference between the measured at the plant results and the estimated ones. The probable causes for this discrepancy are: the upscaling effect (error) from the pilot plant tested to the processing plant, estimation error at each block value associated with the interpolation process (kriging minimizes the error but do not eliminate it), discrepancies caused by operation lack of geometric adherence between the planned blocks to be mined and the ones really extracted, and last but not least important the sampling error at the processing and pilot plants. Even though these probable causes are known, it is not possible to accurately nominate which one is more or less responsible for this discrepancy.

CONCLUSION

Combining mineralogical characterization into mineral processing studies is of paramount importance for defining ore types and to understand their behavior at the processing plant. Incorporating geometallurgical response into mine planning can lead to a more effective and profitable operation.

The use of non-linear methods to deal with non-additive geometallurgical variables provide more precise and accurate models. In this case study, results from reconciled models against production data showed IK is more accurate than OK for the purpose of estimate geometallurgical variables.

The error analysis showed a significant difference between the estimation methods. This is important to show the impact on the chosen method to estimate a block model. The reconciliation analysis presented a slight improvement in estimating using IK, even considering the discrepancies between the real and estimated grades. These discrepancies are not only caused by the estimation method, but for a combination of factors.

In future studies, other alternatives will be investigated to build models with geometallurgical variables modelling including stochastic simulations.

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IMPROVEMENT OF DRAINAGE WELLS EFFICIENCY

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IMPROVEMENT OF DRAINAGE WELLS EFFICIENCY

ABSTRACT

Drainage wells are commonly used to dewatering mining areas. Due to their nature and relatively short lifespan, they are intensively operated, and thus deteriorate much faster than drinking water wells. The pumping system operating contributes to significant costs, this due to the consumption of electricity, drilling, rehabilitation, and the purchase and repair of pumping equipment. The costs associated with water pumping increases simultaneously to well-loss increase. The well-loss is a phenomenon occurring naturally in water wells. However, it can be additionally activated by pump unit over-sizing.

The literature describes many cases of cooperation pump units and pipe-line, however these are related to stationary pumping systems, where the pump unit operates at a constant pumping head. The potential cost and energy savings from pumping systems is great. Recent studies have found that pumping systems account about 20 percent of world energy usage. Efforts that minimize wasted energy in these systems would not only have substantial economic savings and also important environmental benefits.

Drainage wells are intended to decrease of water level in the aquifers, so the pump unit co-operates with a variable pumping head and variable well-discharge. Moreover, pump unit co-operation depends on all elements of hydraulic system, including aquifer and well zone.

Dewatering wells are the basic drainage method used in lignite production, with a few thousand wells in operation across the Europe. Due to intensive exploitation, their life-cycle is from a few to several years. Their rapid deterioration comes as a result of mechanical and chemical clogging. The phenomena of high-speed water flow and significant drawdown occurs much faster than drinking water-wells.
The research conducted in the lignite mine pumping plant indicates that improvement of wells efficiency and maintenance cut-cost could be achieved by proper geo-hydraulic parameters recognition and pump unit selection.

KEYWORDS

groundwater, drainge well, critical velocity of seepage, step drawdown test, well discharge, pump unit, pumping rate, well loss, power savings

INRTODUCTION

Drainage wells are commonly used in civil engineering for dewatering urban and industrial areas. Due to their nature and relatively short lifespan, they are intensively operated, and thus they deteriorate much faster than water-supply wells. Operating of pumping systems contributes to significant costs, this being due to the consumption of electricity, drilling, development, rehabilitation, and the purchase and repair of the pumping equipment. The costs associated with raising water from the wells increase with well ageing, which is a phenomenon occurring naturally in wells. However, it can be additionally activated by the pump unit over-sizing. The paper views at the excessive well loss issue.

Sub-surface drainage is one of the most important auxiliary processes in construction and mining, ensuring safe working conditions, but it also has a great impact on production or construction costs as well as on the environment. Therefore, there is a huge potential to save cost and energy from optimizing pumping systems. Recent studies found that pumping systems account for about 20 percent of world energy usage (Yates & Weybourne, 2001). The improvement of the operation and performance of a pumping plant can result in reductions in energy costs even to 50%. Minimizing of energy waste in these systems would not only have substantial economic advantages, but important environmental benefits as well (Hodgson & Walters 2002). Taking into consideration subsurface drainage, waste energy is concerned with two basic elements: groundwater wells and pumping equipment. Energy savings could be achieved by the proper pump unit selection and minimizing of hydraulic losses. Cooperation of the pumping equipment with the hydraulic system of dewatering wells is the main issue discussed in these paper.

The scientific literature describes many cases of cooperation between pump units and pipe-lines. However, it is related to stationary pumping systems, where the pump unit operates at a constant pumping head. The dewatering wells are intended to decrease of water level in the rock mass, so the pump unit co-operates with a variable pumping head and variable aquifer discharge. Moreover, the pump unit co-operation depends on all elements of the hydraulic system including aquifer and wellbore.

Well ageing causes well loss which is time dependent. However, well loss is a naturally occurring phenomenon that depends on the hydraulic features of the aquifer, the well construction materials and the borehole drilling, development and operation (de la Loma González 2013, Houben & Treskatis 2007). Increase of flow velocity around a wellbore mobilizes fine particles, which are then transported through the porous structure. This may leads to clogging of slots of the screen, thus reducing both porosity and permeability (Blackwell at al 1995). Other reasons for the short lifespan of wells is chemical clogging, occurring as a result of the quick drawdown and aeration of the screen pipe (de Zwart et al. 2006, de Zwart 2007, Hitchon 2000, van Beek 1989). Clogging of the gravel pack and well screen leads to the increased well loss, energy consumption and operating costs. The phenomena of the high-speed water flow and huge drawdown with screen pipe aeration occur much faster in drainage wells than water-supply wells.

The well loss can be evaluated by the well pumping test. In the Jacob equation (1947), the well loss is described as the non-linear (non-laminar) flow regime component CQ^2 (e.g. Helweg 1994, Kawecki 1995, Singh 2002, Singh 2008, Treskatis et al. 1998). In some cases changes of the drawdown cannot be explained by the Jacob formula. Rorabaugh (1953) adopted formula, where power exponent in the expression CQ is different from 2. Some researchers (e.g. Atkinson at al. 2010, Motyka & Wilk 1986) noticed that in fissure and fissure-karst aquifers, the presence of turbulent flow conditions is possible. In these cases, high pumping rate moves turbulent flow regime into aquifer, which is the reason of power exponent changes.

However, in contrast to above, this paper presents the results of step-drawdown pumping tests analyzed jointly with pump characteristics. It is assumed that deviation from Jacob equation is limited by pumping head of pump unit. Moreover, it is assumed that the pump operating point influences on well pumping test performance. These are main objectives of the work presented in this paper.

INITIAL ASSUMPTION

The flow regime in the porous media can be described in two different ways: the laminar flow – expressed by linear relationship, and the turbulent flow - expressed by the quadratic dependence on the pumping rate. According to Darcy's law, the flow rate depends on the hydraulic gradient. In the natural conditions, the water flow takes place in a laminar regime. The growth of hydraulic gradient takes place only in the near well-zone. It is observed in the well as a well-loss. It has to be explicitly emphasized that well-loss is limited by pumping head curve. It is obvious that the operating point is an intersection point of pump and well-system curves.

So Far, most of publications are focused on stationary pumping plant selection, pump unit life cycle cost performance and energy saving capabilities (e.g. Frenning 2001, Kaya et al. 2008, Papa et al. 2014, Yates & Weybourne 2001). In the most frequent cases of stationary pump units, total pumping head (*H*) could be predefined (Sulzer 2010, Jones at al. 2011, Shiels 2001, Larralde & Ocampo 2010). In such cases, pump selection is relatively easy. However, the well drawdown curve is unique for each well and could be obtained during the step drawdown test, when total pumping head *H* is changed. Many recent interpretations of well pumping tests focused on well development assessment (e.g. Shekhar 2006, Jha et al. 2006, Sethi 2011) and the evaluation of aquifer parameters (e.g. Avci 1992, Avci at al. 2010, Dufresne 2011, Mathias & Todman 2010, Louwyck et al. 2010, Karami & Younger 2002). However, little attention has been paid to the impact of pump unit selection on interpretation of pumping test data and well loss growth.

In the case of dewatering wells, the pumping head *H* is changeable in time not only because water well deterioration. Very soon after well development, the cone of depression is enlarged. The pump unit becomes undersized on pumping head. At the same time, the well discharge is also decreasing. Consequently, pump units must be replaced simultaneously according to the groundwater level decrease. Changes of the drawdown in the mine and their influence on the pump unit type is presented in Figure 1.

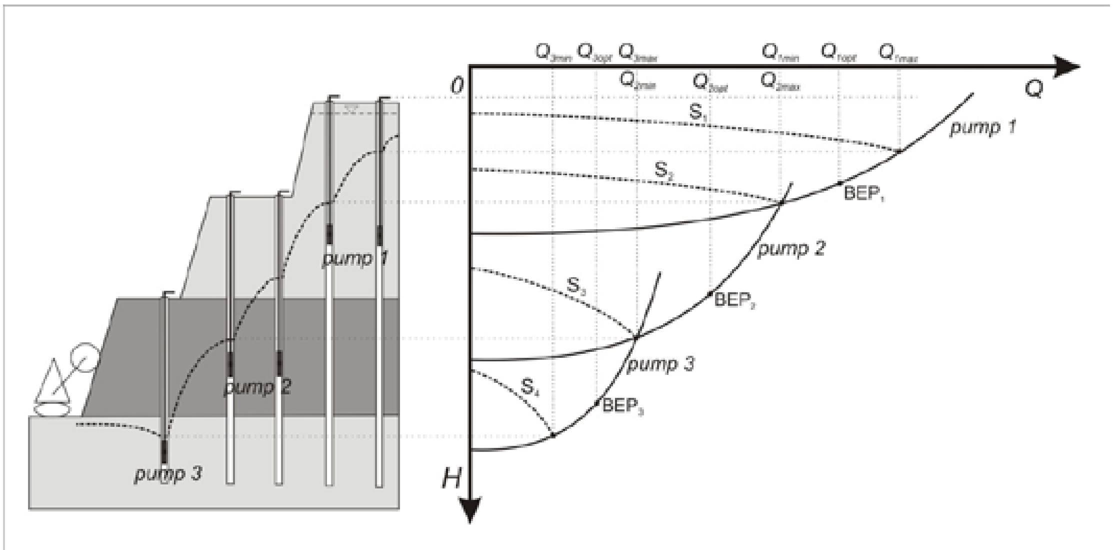

Figure. 1. Influence of drawdown growth on the pump cooperation point; where: Q_{min} is a minimum acceptable pumping rate-, Q_{opt} is an optimal pumping rate-, Q_{max} is a maximum acceptable pumping rate - in relation to the BEP of the pump unit, H – pumping head, S is the actual drawdown.

Selection of pumps in these conditions is much more complicated in comparison to stationary pumping plants. Application of large safety factors when determining head requirements will lead to pump unit oversizing. The flow rate then has to be regulated with a valve to avoid an additional pressure loss on the well screen. This would not only cause increases in investment costs, but also unnecessarily high energy consumption and premature wear of the pump unit (Sulzer, 2010) as the pump then operates under load. Either way of dealing with an oversized pump unit will lead to an increase of energetic and financial expenditures.

According to dewatering wells, where the drawdown is growing in time, the pump unit requires exchange to follow drawdown changes. Head losses changes in the well and their influence on cooperation with the pump curve is presented in Figure 1.

In the practical solution exceeding of critical well discharge depends on pump settings. In general, pump oversizing could be obtained in three different ways:
1. oversizing of pumping rate, where $Q>Q_{crit}$,
2. oversizing of the pumping head, where $Q \le Q_{crit}$, and $H_P>H_L$
3. both: pumping rate and head oversizing, where $Q>Q_{crit}$ and $H_P>H_L$

When the pump selection is proper:

1. well discharge is equal to (or less than) well critical discharge,
2. pumping head is equal to hydraulic losses in well system,
3. well loss can be described by the Jacob formula on each step of drawdown test,
4. pump unit and water well cooperation point take place in best efficiency point (BEP).

RESULTS

In this study, 45 pumping tests have been carried out for 41 wells. Research has been conducted in a lignite mine, where hundreds of wells are working in different hydrogeological conditions. In general, manufacturer characteristics of pump have been used to evaluate of well-pump cooperation.

The well screen diameter is 406 mm in most of the wells. In general, drainage is conducted in the porous (shallow) and fractured (deeper) aquifer. All wells are more than 100 m deep and consist of several to over a dozen screens. Although, depending on the location of the well, the water table is reduced in relation to the natural groundwater table.

The first approach for step drawdown test interpretation for the evaluation of hydraulic parameters of geological formations have been made in the previous publication (Polak at al. 2015). For the presentation of the pump unit influence on the step drawdown test, the results of 4 drawdown tests have been chosen in relation to the pump head curve. The cases of wells W389, W20 and W58 are typical examples of pump oversizing in the actual state of the well quality.

The first example presented in Figure 2 is a step drawdown test in the well W389 drilled in the porous, confined aquifer consisting of medium- and small gravel sands. The static water level is 83 m below the actual ground surface. However, the water level in this point of the cone of depression is 290 m below the pre-excavated ground surface. The well is 143 m deep with 4 active well screens. The total length of the screens is 17 m. Hydraulic conductivity of the aquifer is $1.2 \cdot 10^{-4}$ m/s. Critical well discharge is about 5.2 10^{-2} m^3/s. However, BEP of the pump is much less. The drawdown curve is continuous in the whole range of the pump curve. A difference between a pump and well characteristics in the last pumping stage is a hydraulic pipeline-loss and also a difference between a nominal pump curve and a real curve, which could be reduced in relation to those presented in Figure 4. The results indicate that drawdown curve could be described in full agreement with the work of Jacob (1947).

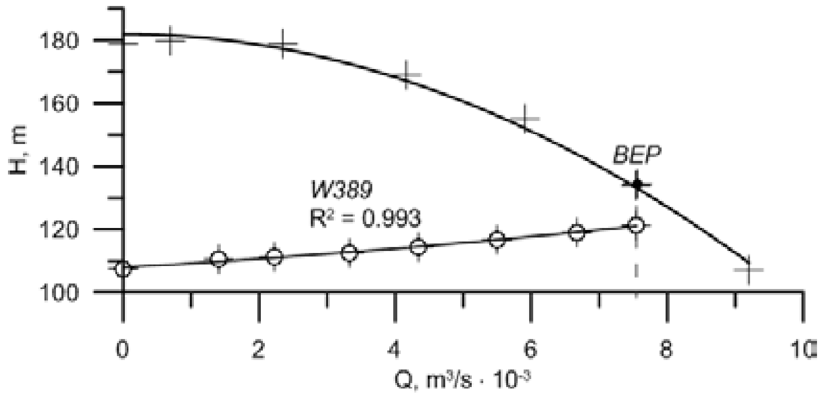

Figure 2. Step drawdown test in the well W389

The second example presented in Figure 3 is a step drawdown test in the well W20. The well was drilled in the confined aquifer consisting of Jurassic limestones. The static water level is 205 m below the ground surface. The well is 250 m deep with 2 active well screens. The total length of well screens is 12.8 m. Hydraulic conductivity of aquifer is $3.5 \cdot 10^{-5}$ m/s. The critical well discharge is about $2.13 \cdot 10^{-2}$ m^3/s and is equal to pump BEP. However, current pumping rate is about 50% less. The drawdown curve is continuous and could be described by the Jacob formula. The difference between a pump and well characteristics in the last stage of the pumping test is a hydraulic pipeline loss. The result of the step drawdown test shows that the pump is oversized on pumping rate. The pump unit and well cooperation take place on the left side of pump characteristics and that is energy consuming. It also causes premature wear of the pump unit. In this case, rehabilitation of the well is required for acceleration of well discharge.

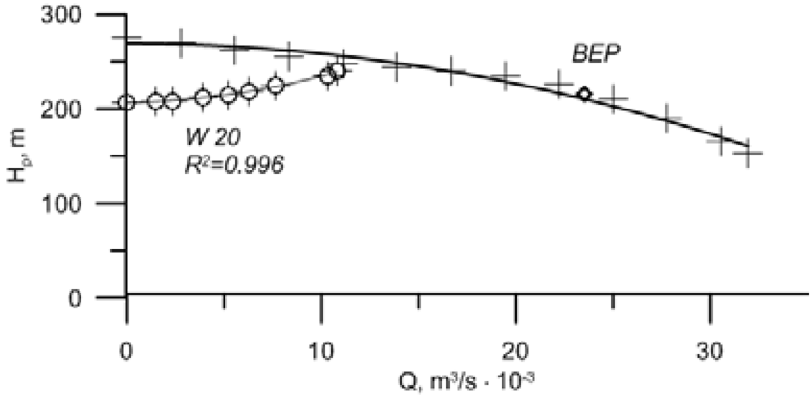

Figure 3. Step drawdown test in the well W20

The third example presented in Figure 4. is a step drawdown test in the well W59 drilled in the fractured, confined aquifer consisting of Jurassic limestones. The static water level is 87 m below the actual ground surface. However, the water level in the cone of depression is 125 m below the pre-excavated ground surface. The well is 250 m deep with 9 active well screens. The total length of the screens is 64.3 m. Hydraulic conductivity of the aquifer is $3.5 \cdot 10^{-5}$ m/s. The critical well discharge is about 0.1 m^3/s. However, BEP for the pump is much less. The drawdown curve is continuous only when the valve is partially open. For the last two steps of the drawdown test a well loss is additional, and it is called an excessive well loss (E_{WL}) in this paper. The difference between a pump and a continuous well Jacob curve results in oversizing of the pump unit head. The well system and pump unit cooperation take place on the left side of pump characteristics. However, it could take place for the pump where a curve would be lower than 50 m. This case shows that usage of the current pump unit causes energy consumption and it could be reduced by pump unit replacement. According to actual pumping rate, power savings in this particular case would be 16 kW. Moreover, working with an excessive well loss leads to the aeration of screen pipe and chemical clogging, which reduces the life-span of the well. In this particular case, rehabilitation of the well is required for acceleration of dewatering.

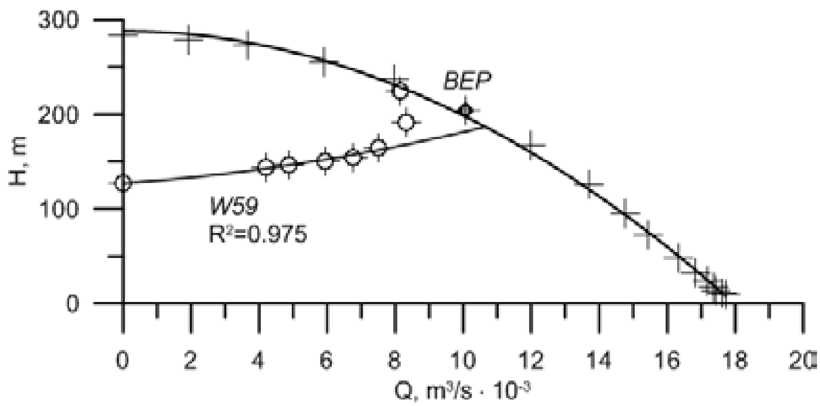

Figure 4. Step drawdown test in the well W59

The fourth example presented in Figure 5 is a step drawdown test in the well W48 drilled in the porous, confined aquifer consisting of fine and medium gravel sands. The static water level is 22 m below the actual ground surface. However, the water level in the cone of depression is 57 m below the pre-excavated ground surface. The well is 100 m deep with 4 active well screens. Total length of well screens is 17 m. The weighted average hydraulic conductivity of aquifers is $5.0 \cdot 10^{-5}$ m/s. The critical well discharge is about $3.3 \cdot 10^{-2}$ m^3/s and BEP for the pump is less. The drawdown curve is continuous only when the valve is not fully open. For the last step of the drawdown test there is an additional well loss, called an excessive well loss (E_{WL}) in this paper. The difference between this point and the continuous well drawdown curve is the result of oversizing of the pump unit. Moreover, the difference between the pump curve and the drawdown in the last point of drawdown test is a hydraulic pipeline loss. However, pump curve lowering is the main reason of difference between curves. The pump unit and well cooperation take place on the left side of pump characteristic. However, it could take place for the pump unit where the curve is lower than 30 m and BEP has to be moved to the left. This case shows that usage of the current pump unit is due to energy consumption and could be reduced by pump unit replacement. Taking into consideration actual pumping rate, power savings in this particular case would be 4 kW. Similarly to the well W59, working with an excessive well loss leads to quick drawdown in screen pipe and chemical clogging. Moreover, the pumping rate in this state of well quality is also too high. It probably leads to mechanical clogging acceleration. Oversizing of the pump unit reduces life-span of the well. In this particular case, rehabilitation of the well is required for acceleration of dewatering

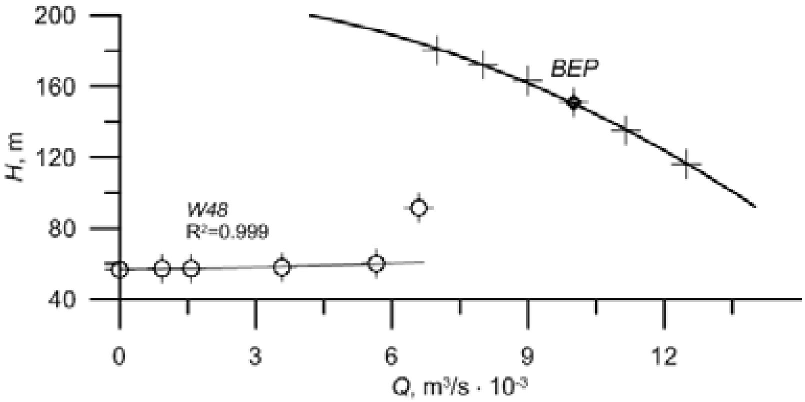

Figure 5. Step drawdown test in the well W48

DISCUSSION

In presented studies, 45 pumping tests have been carried out for 41 wells. A critical discharge has been evaluated while using the Sichardt formula. In all cases the drawdown curve has been compared to the pump head curve. In 23 dewatering wells the cooperation well-pump point takes place in BEP of the pump. Chosen example (Well W389) has been presented in Figure 4.

An excessive well-loss has been found in 18 wells in the last steps of drawdown tests:
In 5 wells the pump unit has been oversized by pumping rate – an example has been described above and presented in Figure 3 (Well W20),
In 5 wells the pump unit has been oversized by pumping head – an example has been described above and presented in Figure 4 (Well W59),
In 8 wells the pump unit has been oversized by pumping rate and head - an example has been described above and presented on Figure 5 (Well W48).

Results of the analysis lead to following observations:

1. Pumping rate could be oversized in relation to the critical well discharge. This case could be noticed particularly in new developed wells. Exceeding of the critical apparent velocity of a seepage in the beginning of life-span of the well leads to mechanical clogging (permeability decrease) of the well-screens by fines. It limits discharge of the well.
2. When pumping rate is oversized and a discharge of the well is limited by aquifer properties, the pump unit is working on the left side of the pump curve and the drawdown curve is corresponding to the Jacob equation. Then this phenomenon leads to energy consumption.
3. When the pump unit is replaced and the new pump is oversized by pumping head, the well–pump unit cooperation point could be close to BEP of the pump. However, excessive well loss at the end of the step drawdown course is observed. Excessive well loss, described by Rorabaugh by using the power exponent *n,* can be detected in this case. When the additional drawdown includes active well-screens of the well the excessive well loss leads to quick

drawdown in the screen pipe and chemical clogging.

4. Oversizing for both the pumping head and rate is also observed. In these particular case mechanical and chemical clogging also occur. However, pumping is energy consuming because pump unit-well cooperation takes place on the left side of BEP. The drawdown curve can be described by the Rorabaugh formula also in this case.

In the conducted research 3 types of pump unit oversizing have been detected. The reduction of well-loss can be achieved by partially closing of the outlet valve. However, it is unfavorable from energy the consumption point of view. Energy savings could be achieved by pumping equipment replacement or well rehabilitation. When the dewatering process is in progress, renovation or drilling of a new well is an option to be taken into account.

In the mine dewatering plant where research has been conducted, the total accumulated value of excessive well loss was about 520 m. Taking into account changes of pumps efficiency, potential power savings have been evaluated. In the calculations it was assumed that pumping rate in all particular wells have to be maintained. The total oversized capacity is approximately about 190 kW. In Figure 6. the values of excessive well loss and the surplus of power consumed as a result of the excessive well loss in the individual wells is presented.

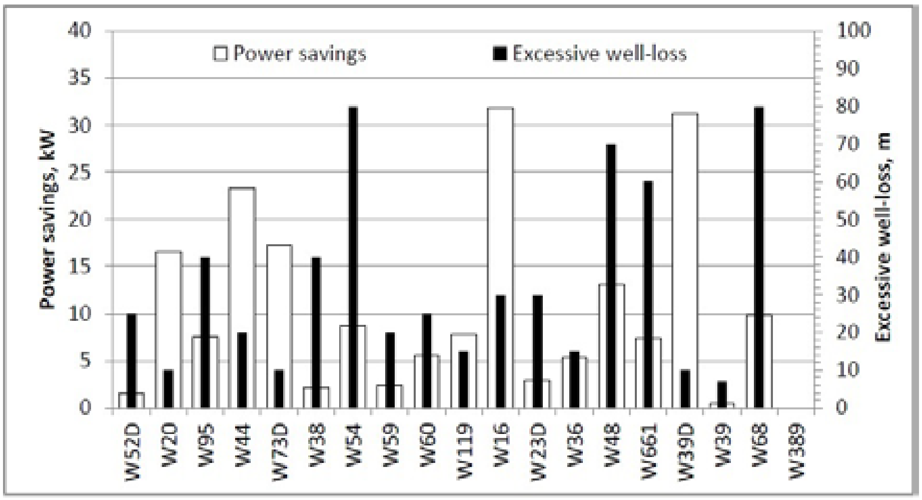

Fig. 6. Excessive well loss and potential power savings in tested dewatering wells

CONCLUSION

In this study, multi-step pumping tests have been carried out in order to determine well quality and energy losses. The considerations presented in the paper indicate that the step drawdown test should be extended to pump unit cooperation. It leads to the conclusion that pump unit test performance could be carried out in the well. When the steady-state flow is achieved, drawdown and pumping rate are measured. The pumping test could be carried out for the manufacturer pump unit curve. However, optionally the pumping head on the pipeline could be measured to test the technical state of the pump unit and pipeline as well.

The step-drawdown curve is the continuous Jacob function as long as the pressure balance in the well is controlled by hydraulic losses in the pipeline. When the pump is oversized, the pressure balance is achieved by excessive head losses inside the well. The process occurs in the last stage (stages) of the pumping test. This phenomenon has been described by Rorabaugh for 3 step drawdown test as a conversion of the Jacob formula with *n*-index. Taking into consideration the results of pumping tests, this conversion is limited by the pump unit curve. So, the excessive well loss can be observed for fractured aquifers. However, it can be also observed for clogged dewatering wells in porous media, where head of the pump is oversized and pipeline hydraulic loss is out of control.

The research conducted in the mine pumping plant indicates that the excessive well loss is affected by oversizing of pump units. In the most common solution the pumping rate is controlled by regulating the pressure loss in the pipeline with a valve. This may have positive impacts on the life-span of the well. However, it also causes a waste of energy and reduces the life-span of the pump unit particularly when the cooperation point is outside of BEP. So, the well loss and energy waste should be reduced by pump unit replacement. However, when the dewatering process needs to be accelerated, rehabilitation of the well is required.

Oversizing of the pump unit is achieved in 3 different ways. Oversizing of pumping rate accelerates mechanical clogging. Oversizing of the pumping head is due to acceleration of chemical clogging. Oversizing of the pumping head and rate together is equivalent to energy consumption, mechanical and chemical clogging acceleration. Extended step drawdown tests including pump test performance for a newly developed dewatering well allow us to the elimination of processes leading to premature well deterioration and to significant energy savings.

ACKNOWLEDGMENTS

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IMPROVING TRANSSHIPPING EQUIPMENT DURING IMPLEMENTATION OF COMBINED TRUCK-RAILWAY TRANSPORT

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IMPROVING TRANSSHIPPING EQUIPMENT DURING IMPLEMENTATION OF COMBINED TRUCK-RAILWAY TRANSPORT

ABSTRACT

Analysis of rock mass transshipment from dump trucks to railway transport on deep open pit mines is conducted. Newly designed dumping station of a deep open pit mine without an excavators is offered, it consists of a bunker, located on non-operational pit bank above railroads, access roads and an upper unloading platform. Unlike known methods, here, pikes are constructed above railroads, connected to corresponding access roads for entrance of dump trucks to the upper transshipping area, where accumulated bulk from rock mass, which is transshipped, is located near the rock mass bunker. Safety of the dump trucks' operation on dumping stations could be achieved through implementation of developed protection device.

KEYWORDS

Deep open pit mines, combined truck-railway transport, dumping stations, dumping station without an excavator with protection device.

INTRODUCTION

Analysis of dumping stations operation with excavators

Steeply dipping mines are characterized by significant decrease of mining operations, decrease of working zone area, large amount (30 and more) of pit banks being mined simultaneously, difficulty of conducting stripping and mining deep part of an open pit mine during mining . Due to the problems mentioned above, two types of transport are used for transportation of rock mass on deep open pit mines: trucks and railway. Practice shows, that highest technical-economic indexes are inherent to electrified railway transport using locomotive unit. However, low gradeability, significant curve radii, difficulty of train blocks interchange do not allow independent usage of railway transport at a depth of more than 300-350 m (Rakishev & Moldabayev, 2012; Drizhenko et al., 2008).

Dump trucks are highly mobile, do not require flat roads on open pit mine, could overcome significant uplifts and operate in constrained conditions. Their carrying capacity is relatively small, and cost of rock mass transportation is higher 8-10 times compared to railway transport. Consequently, its usage is limited to 120-150 m. Thus, combined type of transportation is usually used for transportation of rock mass during mining of deep horizons. Different combination of its constituent units is explained by tendency of using benefits of railway transport and truck efficiently, avoiding their deficiencies if possible. Efficiency of using varying schemes of open pit mine transportation depends on the level of technical-economic indexes during transportation of rock mass, and also on combined loading-unloading operations. Here, organization of transshipping operations plays an important role during combined use of dump trucks and railway transport.

Currently, dumping stations are mainly used on deep open pit mines, equipped with single-bucket excavator and loading machines. Rock mass on such points is kept in accumulating storage facilities. Capacity of a storage facility varies from 20 000 to 300 000 m^3. They have a length of 100-300 m, width of 25-80 m and a height up to 12 m. During transshipment of rock mass from dump trucks to railway wagons railroad is planked along the lower area of the storage facility. Dump trucks are unloaded on the upper area. There are excavator operated dumping stations with rock mass storage facility located on the slope of the pit bank, on horizontal area and recess of a pit bank. In the first case, temporary non-operational side of an open pit mine is used (Figure 1), constructed from stable rock mass, along which bulk storage facility from transshipped rock mass is formed. Wherein, upper area of the pit bank serves as a maneuver- dumping station for trucks (Drizhenko, 2011; Rakishev & Moldabayev, 2015).

DS (Dumping Station) is divided into three zones by width: unloading of dump trucks, borderline (section area) and operation of an excavator. Section area has a length of 10 m, length of the rest of zones is 40 – 150 m. Such dumping stations have only one-sided transshipping front. Main advantages of them are short construction terms and narrow platforms, used as a storage facility on an open pit mine. That is why; they are usually used for short period on one point (moveable storage facilities). Volume of stored rock mass is determined by length of the storage facility and width of single excavator stope.

Horizontal area should have enough space for maneuver-transshipment operations of dump trucks on accumulated pile of storage facility during construction of rock mass storage facility on that same area. Maneuver-transshipment area has a safety shaft of rock mass with a height up to 2 m around the perimeter. One (one sided transshipment front) or more (two sided front) excavators could be placed on dumping station of such type. Width of the storage facility on the upper area should be no less than 35-50 m, on the lower part it could be up to 65-80 m (Figure 2).

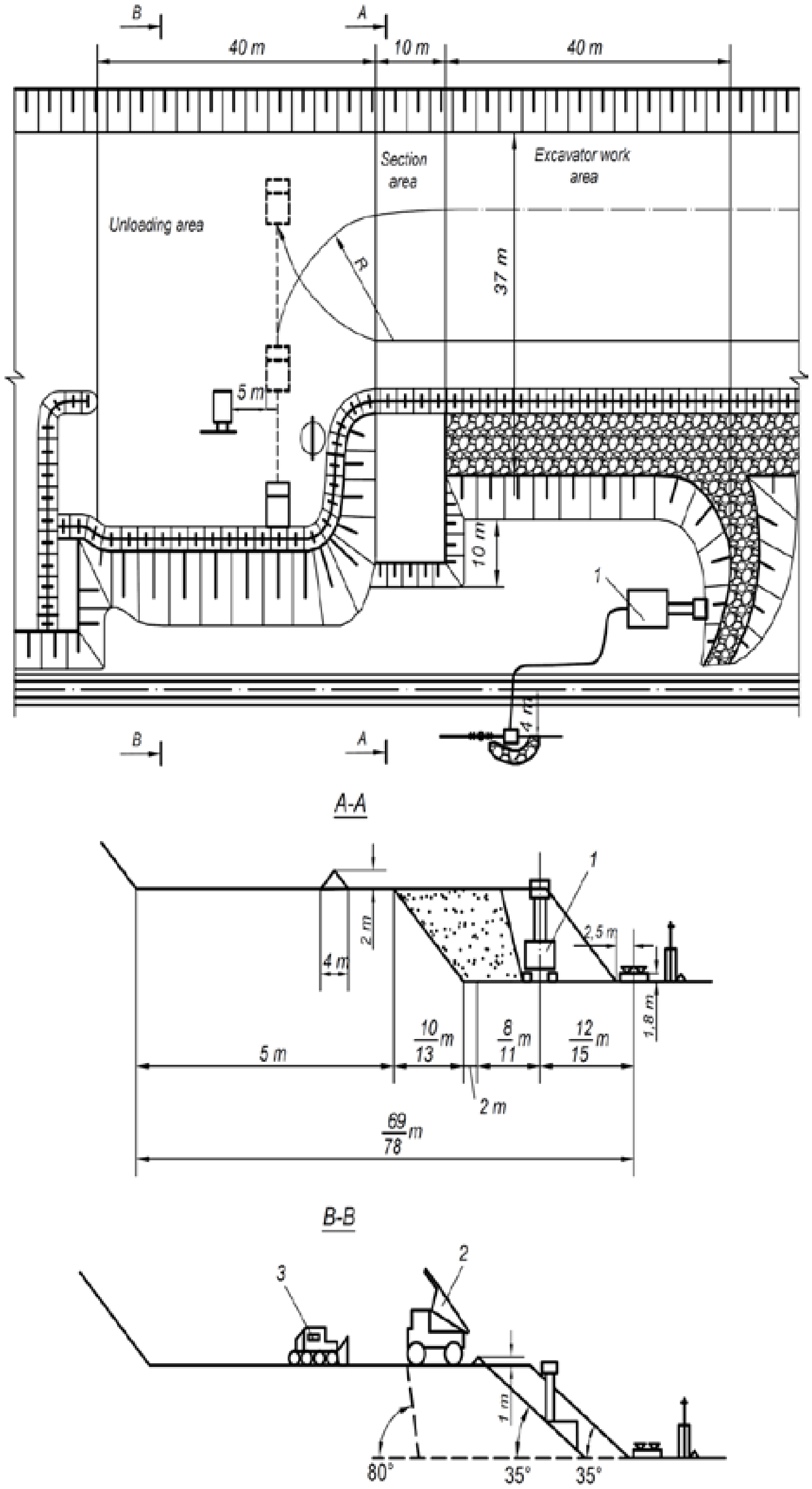

Figure 1 - Scheme of excavator operated dumping station on pit bank slope of an open pit mine: 1 – transshipping excavator; 2 – dump truck; 3 – bulldozer

Figure 2 - Excavator operated dumping station on an open pit mine

When rock mass storage facility is constructed on the recess of the pit bank, the dumping station setup would be the same as above. Excavator is also located on the recess and moved along the storage facility. Recess length is equal to the length of the storage facility, its width is 10-15 m, and depth is 2.5-4 m. Railway is constructed on the area of the pit bank along the recess. Dump trucks turn around on maneuver-unloading area, which is located on the upper platform of a pile of storage facility, and transferred to the recess. Only part of a storage facility which is limited by the width of the recess is operated. Storage facilities, with transferred rock mass on the horizontal area and recess, are usually used on upper horizons and surface of an open pit mine, also on platforms near entrances of exit trenches. By safety rules, height of the storage facility should not exceed maximum height of an elevator scoop.

Main disadvantages of an excavator operated dumping stations are high capital expenses, due to the use of expensive transshipment equipment and bulldozers used for storing rock mass; high cost price of transshipment; long period of loading railway transport (45-50 min.), determined by the productivity of excavators and loading devices being used; significant areas, occupied on an open pit mine; dumping station downtime due to the relocation of excavators and removal of railways.

Dumping stations during implementation of combined truck-railway transport

Introduction of high-power excavators with scoop capacity of 20 m^3 and more excludes the possibility of direct loading of modern construction dump trucks due to the low heat input capacity. Device of special directing platforms and dumping stations with accumulating bunkers is needed for their combined operation.

Dumping stations are divided into stationary, semi stationary and mobile by its classification features (Moldabayev et al., 2015; Sultanbekova et al., 2015). Stationary dumping stations are usually located on surface or mined upper horizons, and are used throughout the whole operation period of a mine. They are constructed from monolith or fabricated reinforced concrete, rarely from metals. Stationary dumping stations are usually used during loading of railway transport by transport systems, containing conveyor or skip hoists due to the significant increase of haulage length by dump trucks.

Semi stationary dumping stations are used to receive rock mass, transferred by a truck. They are usually located on lower horizons, maintained by railway transport, within non-operational or temporary non-operational side of an open pit mine. After 4-6 years they are usually relocated with expansion of railway communications to the deeper horizons. Wherein, dumping stations are usually meant to be located on depth of an open pit mine with step of 45-60 m and scattered around the perimeter, so that distance the of truck transportation does not exceed 1.2-1.5 km. Dumping stations are designed for combined operation of high-power excavators and railway transport directly on operating horizons.

Along with rock mass transfer from one type of transport to another with single-scoop excavators, there is also a practice of using draglines, direct loading of dump trucks with carrying capacity of 5, 10, 27 and 40 tons through guiding device or loading bin using belt conveyor, feed apron and vibratory feeder, also self-flowing outlet. Loading of wagons by belt conveyors and feeders are conducted on slow pace of railway transport, by excavators – with railway transport. Only one wagon is loaded at a time. Dumping stations with simultaneous loading of five or more wagons by vibratory feeders are used abroad (Drizhenko, 2011). In general case, output per shift of a dumping station is calculated by using following formula:

$$E = (T_c - t_n) \times p_d \times t_n \div \left(\frac{t_n \times p_{d.o}}{p_p - E_c} + t_{v.o}\right) \times \left[\left(\frac{p_d}{p_{d.o}} - 1\right) + t_o\right] \quad (1)$$

where, T_c, t_n – duration of a shift and planned breaks in operation respectively, hours; t_n – carrying capacity of a wagon, tons; $p_{d.o}$ – number of wagons (dump trucks) being simultaneously loaded; p_d – number of wagons in a railway transport; p_p – number of simultaneously operating loading machines; E_c – technical productivity of a loading machine, tons/hour; $t_{v.o}$, t_o – period of time allocated for exchange of wagons and railway trains during loading, hours.

Best usage conditions, using a group of feeders, current supply of railway trains for loading allow to significantly increasing productivity of dumping stations compared to equipping them with open pit mine excavators. Duration of railway trains loading has rectilinear characteristic and significantly decreases with increase of productivity of transshipping equipment and number of wagons in a railway transport. Productivity of dumping station rises with the increase of railway transport's carrying capacity (Figure 3).

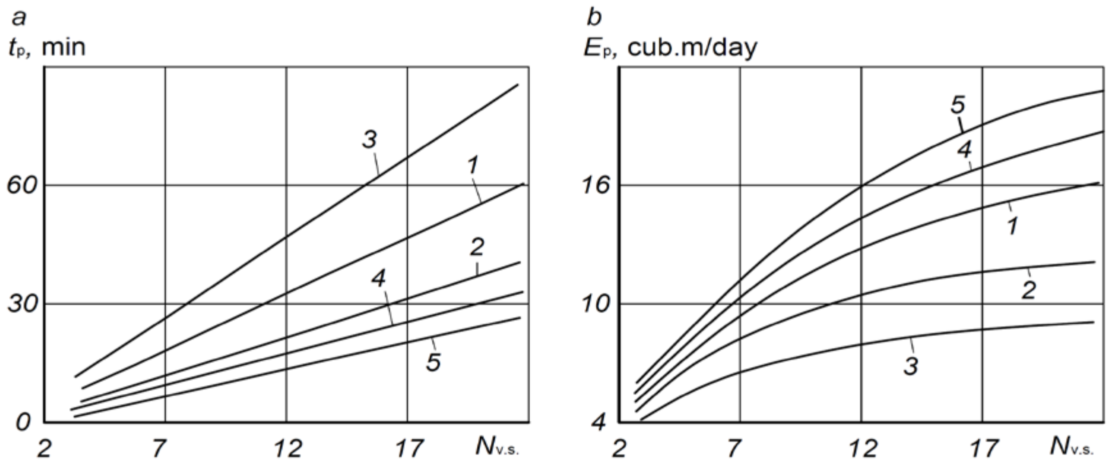

Figure 3 - Dependence of change of railway trains' loading time t_p (a) and productivity of a dumping station E_p (b) on number of trailer dump trucks 2BC-105 ($N_{v.s}$): 1,2,3 – loading one dump truck simultaneously with one, two and three vibratory feeders VPR-3K (VPR-4); 4 – loading four dump trucks simultaneously with three vibratory feeders each; 5 – loading of EKG-8I excavator

Productivity of dumping stations mainly depend on a number of loading machines, their productivity, number of dump trucks used with railway transport, schemes of railway trains supply and change as can be seen from Figure 3,. Open pit mine excavators have higher productivity compared to draglines; thus, it is widely used. Productivity of conveyors and vibratory and apron feeders depends on width and speed of a belt. Thus, technical productivity of conveyors with belt width of 1200-2000 millimeters changes from 1000 up to 6000 tons/hour. For vibratory feeders VPR-4 and VPR-3K, it is 1500-2000 tons/hour, apron feeders with belt width of 1500-1800 millimeters – 2000 tons/hour.

Railway transport is loaded with two excavators to decrease its downtime. Arranging dumping stations with simultaneous loading of 1-5 wagons and more with group of feeders allows decreasing downtime of railway trains being loaded up to 8-15 minutes. Rate of rock mass supply to dumping station should correspond to the productivity of transshipping equipment to ensure normal functioning of dumping station.

Dumping stations without excavators

Unloading storage facilities have an area of no less than 50×250 m, which significantly increases amount of work on spacing of open pit mine side. Storage height of 10-12 m does not provide enough safety for loading of dump trucks at the upper area. Unloading expenses of one ton of a rock mass are comparable to loading it on the sides. At the same time, points equipped with high performance transshipping devices allow significantly increasing loading speed of trains, releasing excavators for their intended use on open pit mines and dumps, increasing performance of operating personnel, decreasing electricity and materials expenses (Drizhenko et al., 2001; Dudchenko et al.,2001; Drizhenko et al.,2002; Drizhenko & Simonenko,1985; Dudchenko et al.,1991; Dudchenko et al., 1990).

Trestle and bunker dumping stations are used for truck-railway transport on small open pit mines. Capacity of a bunker of a dumping station is 1000-4000 m^3. All these dumping stations are mounted on an open pit mine on bulk trestle or pit bank. Trestle points, construction-wise, consist of trestle, maneuver-transshipment area with emphasis on wheels of dump trucks and railway for installment of wagons being loaded. If bulk trestle is used, their height exceeds the height of a wagon for 0.5-1 m. Direct transfer of rock mass from dump trucks to dump trucks is possible on such dumping stations.

Bunker dumping stations could be constructed on an open pit mine as stationary, semi-stationary and reinforced concrete or mobile metallic constructions. They are completed by high-power vibratory feeders and located on pit banks, consisting of solid rock mass. Dumping stations should correspond to specific requirements during operation on an open pit mine: to be demounted comparably easily and in a short period of time, transported and mounted on new working place; be operationally reliable and non-complex in maintenance and use; withstand high seismic and hitting loads from massive explosions on an open pit and falling parts of rock mass being loaded; occupy less territory of an open pit side and area on the pit bank; fit into current open pit mine side construction. Semi-stationary mobile and mobile dumping stations fully correspond to the above requirements, such dumping stations are usually located on non-operating and operating sides (Drizhenko et al.,2001; Dudchenko et al.,2001; Drizhenko et al.,2002).

Semi-stationary dumping stations are located on the recess of a pit bank massif using two schemes of equipment layout: on the space under a pit bank of non-crushed rock mass massif and on artificially constructed metallic or reinforced concrete frame, piled with fine-grained rock up to the level of vibratory feeders installment (Drizhenko & Simonenko,1985; Drizhenko et al.,2015). Wherein, frontal (receiving) bunker wall in rock mass, prone to destruction, are additionally reinforced with removable blocks. Mobile dumping stations could also be assembled on the base of mobile devices (Dudchenko et al.,1991), located along prepared pit bank slope (Dudchenko et al., 1990). It is practical to arrange protective cushion fill from rock mass in less stable cracked rock during possible destruction of pit bank slope; sectional frame is placed between loading device and slope to maintain such cushion fill.

Dumping stations without excavators are located on small areas, which decreases mining-constructional work amount on spacing of open pit mine walls. Dumping station without excavator of a deep open pit mine is proposed, which includes bunker, located on non-operational pit bank above railways, access roads and upper loading area. In contrast to well-known methods, pike is constructed above railways, connected with corresponding access roads for entrance of dump trucks to the upper loading area, where near the bunker accumulating pile of rock mass, being loaded, is set up (Drizhenko et al.,2015).

In this regard, developed technical solutions allows significantly decreasing rock mass transportation distance by dump trucks, implementing complete filling of a bunker and creating certain load reserve on a loading platform by constructing accumulated pile of rock mass, and thereby decrease downtime of railway trains being loaded. Excluding excavators from use allows significantly decreasing operating costs for loading operations. Wherein, block structure has an influence on dumping stations' operation effectiveness, which allows using its structure many times during advancement of mining operation front and depth of mining. Doing so allows using dump trucks at an average distance of rock mass transportation up to 1.5-2 km with decrease of their total number.

Transportation distance decrease, for example, during operation of Kachar open pit mine, with transportation of rock mass by a truck to dumping station having deadlock location of railway by only reducing dump trucks route allows recouping the costs of trestles and gaining economic effect of 56 000 USD annually. Protection device is offered to increase maneuvering safety of dump trucks on the loading area (Anisimov et al.,2015). Unlike well-known methods, metallic grid in the upper part of the stand is equipped with frame connected to the counterweight on the base and fastened movably on the stand with possibility of displacement in vertical plane and fixing in the given position. Wherein, basis of the stand is constructed as slides with possibility of displacement in the given direction. Economic effect from changing metallic grid with anchor mount for one dumping station will be around 120 000 USD.

On deep open pit mines transshipment is usually conducted on dumping stations, which are equipped on rock ledges with height of 12-15 m. Reinforced concrete fence is constructed or safety shaft, by safety rules it should be placed of the limits of natural slope angle of rock mass in the massif and have height of no less than 1 m (Table 1), is piled from loaded rock mass on upper area of pit banks to prevent downhill fall of dump trucks during loading.

Table 1 – Parameters of thrust shaft at a time of unloading dump trucks

Title	Index			
Carrying capacity of dump trucks, tons	120	130	180	200
Total mass of loaded dump truck, tons	210	235	343	355
Base of dump truck, meters	5,3	5,3	6,65	6,1
Tire type,	33.00R-51		40.00R-57	
External diameter of tire, millimeter	3022	3022	3575	3575
Width of tire section, millimeter	900	900	1140	1140
Width of dump truck by rear wheels, meters	6,14	6,47	7,78	7,78
Height of rear edge of body at a period of unloading above horizontal area, meters	1,08	1,2	1,6	1,5
Distance from rear wheel axle to rear edge of body at a period of unloading, meters	2,5	2,85	3,4	3,49
Minimum proportions of safety shaft in unloading zone of a dump truck, in meters:				
– height	1,0	1,0	1,0	1,0
– base width	2,5	2,5	2,5	2,5
– area of base, m^2	16,2	25,5	27,8	27,8

Recommendations on dump trucks' movement intensity at a time of unloading are not specified. Supplying enterprises with high-power dump trucks requires more accurate regulation of their safe movement conditions. Reinforced concrete fences have long construction period, capital-intensive and cannot be displaced if dumping station contours are changed. Cleaning of spill under thrust is done using special equipment. Safety shafts are constructed from rock mass being transferred, cleaned, displaced and built up during the process of usage by bulldozers.

During unloading dump truck moves in reverse and abuts to high-wall of safety shaft by rear wheels (Figure 4). Vertical G (H) and resultant τ (H) forces from dump truck's mass act downwards and does not cause any destruction of pit bank massif. Inertia force of moving dump truck F_u (H) is significantly low and balanced by shift reaction on cross section of safety shaft under rear wheels. Wherein, $F_u = a\ G$, where a – deceleration of a dump truck movement while hitting safety shaft, m/sec^2.

From equation of slow motion velocity:

$$V_к = V - at \tag{2}$$

$$a = \frac{V}{t} \tag{3}$$

where, $V_к$ and V – final and initial velocities of dump truck movement to dumping station m/sec; t – deceleration time, sec.

Shift reaction of safety shaft R (H) is as follows:

$$R = C_c\, S \tag{4}$$

where, C_c – safety shaft's rock mass clutch, pascal. By (Drizhenko, 2011) for loosened rock mass of iron mines, piled on solid base, $C_c = 0{,}5 \cdot 10^4$ Pa; S – base area of safety shaft in the contact zone with rear wheels of a dump truck, m^2, which is equal to

$$S = (B+C)\, h_b\, ctg\,\varphi - 0{,}25\, b^2 tg\,\varphi \tag{5}$$

here, *B, b* – external and internal width of a dump truck gauge, m; *C* – external width of prism cross section of safety shaft due to rear wheels impact, m; h_b – height of safety shaft, m; φ – internal friction angle of piled rock mass in safety shaft, degrees.

Ratio $n_з = \dfrac{R}{F_u}$ shows value of stability margin of safety shaft from cross section. It is considered, that operation safety of dump trucks during unloading could be achieved if $n_з \geq 5$. From Figure 5 it can be shown, that safety of dump trucks with carrying capacity of 120-200 tons with safety shaft height of $h_в$ = 1 m is achieved by initial movement speed of V = 5-6 km/h and braking in 20 seconds. Increasing braking duration to 25 sec allows dump trucks to move to dumping station with the speed of 7-10 km/h. These numbers are marginal for dump trucks with carrying capacity of 120 tons.

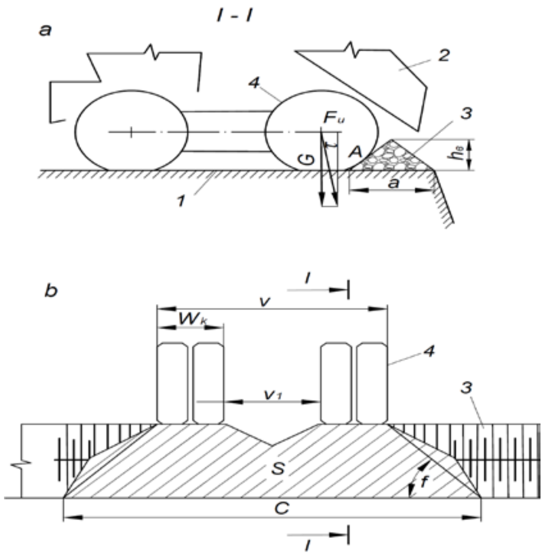

Figure 4 - Calculation of safety of dump truck while reaching safety shaft; a – cross-section; b – plan view: 1 – pit bank; 2 – dump truck; 3 – safety shaft; 4 – rear wheels

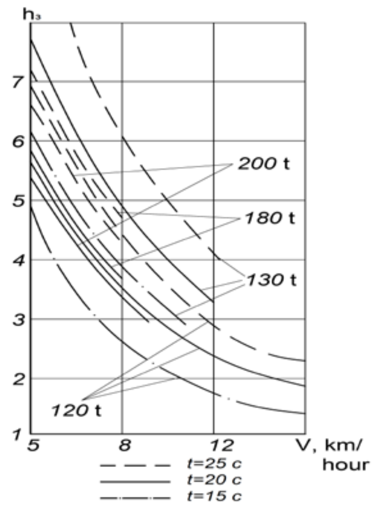

Figure 5 - Graph of change of safety shaft's stability margin n_3 due to dump truck's initial movement speed V and its deceleration time until stopping t

There is a possibility of piling safety shafts with height of 1.5-1.8 m for more powerful machines. Rock mass on them are compressed by rear wheels over time, whereupon, value of C_c increases. In this regard, dump trucks road speed to the unloading point could be increased up to 12 km/hour, braking time could be decreased to 15 sec.

Main technical-economic indexes of semi-stationary and mobile dumping stations of combined truck-railway transport are shown in Table 2. Displacement of dumping station to new location is done in no less than 1-2 years. Displacement duration of semi-stationary and mobile dumping stations is determined by taking into account combination of assembly and dismantling operations in time.

Table 2 – Main technical-economic indexes of semi-stationary and mobile dumping stations

Index	Dumping station productivity, million tons/year		
	4-6	8-10	10-12
Dimensions of dumping station, m:			
Length by a front of a pit bank	20-25/15-20	25-30/30-45	40-50/45-60
Length of maneuver area for dump trucks	50	80	80
Width of maneuver-unloading area	45-50	45-50	45-50
Width of lower transport area	15/20	15/20	15/20
Bunker capacity, tons	650/520	800/700	1200/1160
Number of vibratory equipment on point	1/1	3/3	3/6
Productivity of vibratory equipment, thousand tons/hour	1,5	1,5	2
Number of operating personnel	6	7	8
Power consumption, thousand kW/h	300	900	1006

Note: nominator – semi-stationary dumping station, constructed on a bunker recess of a massif; denominator – mobile dumping station, constructed on a prepared pit bank slope.

Depending on required productivity, dumping stations are equipped with 2-3 vibratory feeders with productivity of 1500-2000 tons/hour. Duration of loading railway trains decreases from 20-24 to 8-12 minutes. This increases their turnover. In this regard, labor productivity on combined transport of basic open pit mines increases 1.2-1.4 times, power consumption decreases 2.3-2.5 times, 1-2 railway trains are freed. All excavators and bulldozers, used for transshipment, are directed to an open pit mine to perform their primary functions of excavating and loading rock mass.

CONCLUSION

Rock mass transshipment during its displacement to the surface is ubiquitous on all deep open pit mines for truck-railway transport. It is done by open pit mine excavators with scoop capacity of 5-12.5 m^3, which leads to long downtimes of trains during loading. Open pit mine side part with placement of dumping stations on separate pit banks with length up to 200-300 m and width of 40-60 m has slope angle of no more than 20°, which constrains required intensity of deepening of an open pit mine. Large amounts of rock mass, stockpiled in an unloading storage facility, leads to freezing of funds and high costs of train loading. Developed designs of dumping stations with bulldozers and vibratory feeders, but without excavators do not have such deficiencies. However, accumulating capacity of bunker is not high (up to 5 000-10 000 tons), which constrains its commercial development. At the same time, rock mass reserves on dumping station could be increased through consideration of blasted rock amount on sides and, if necessary, replenish with them current needs of transshipment operations by given plan. Currently, the work being conducted to replace vibratory feeders with more reliable feeders of special construction for surface mining operations, which will allow to completely abandon the need of using dumping stations with excavators on deep open pit mines.

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ISOVIBRATION AND ISOSUSTAINABILITY FOR BLAST VIBRATION CONTROL IN OPEN PIT IRON MINE

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ISO-VIBRATION AND ISO-SUSTAINABILITY FOR BLAST VIBRATION CONTROL IN OPEN PIT IRON MINE

ABSTRACT

In rock blasting with explosives, only 5 to 15% of the energy released by blasts is used effectively to fragment rock. It means that the largest part is transferred to the surrounding environment in form of side effects, which are likely to cause significant structural damage and human discomfort. This is the most common cause of concern, and there have been protests by people living near areas where these operations are conducted. Explosive energy is effectively used in rock fragmentation process. Consequently, most of the energy transferred to the surroundings brings on collateral effects and may cause negative effects. Shortly after an explosive detonation in the drill hole, two characteristics zones are formed: transition zone (hydrodynamic, plastic, fragmented and broken) and elastic or seismic zone. Vibration near to explosive charge detonation may reach 500 to 5000 mm/s in the later zone may reach 4000-5000 mm/s depending on explosive energy quantity and gradually attenuation with the distance when blasting wave propagation it is known the attenuation varies with the distance. Methodology proposed for blasting vibration control in open pit iron mine consist in following steps: ground vibration characterization by extensive seismic in situ blasting tests, ground vibration and air blast law calculated based on in situ testing data and using these specific equations and human discomfort and physical structure damage blasting vibration standards determined the maximum explosive charge per delay. This methodology applied in important iron mine of Brazil. An innovative methodology is proposed or sustainable control of blasting vibration based on iso-vibration and iso-sustainability index. This methodology validated in one Vale's iron mine.

KEYWORDS

Blasting, vibration, control, iso-vibration, iso-sustainability.

INTRODUCTION

The exploitation of ore resources in open pit mines frequently involves rock-blasting operations. Nevertheless, according to Dinis da Gama (1998), only 5 to 15% of the energy released by blasts is used effectively to rock mass fragmentation. It means that the largest part is transferred to the surrounding environment in form of side effects, which are likely to cause possible structural damage and mainly human discomfort. This is the most common cause of concern for people living near areas where these operations are conducted.

For the process of blasting cylindrical explosive charges in rock hole surrounding perpendicular section to an explosive charge axis, three zones are defined: explosive cavity, transition zone, and seismic or elastic zone (Figure 1).

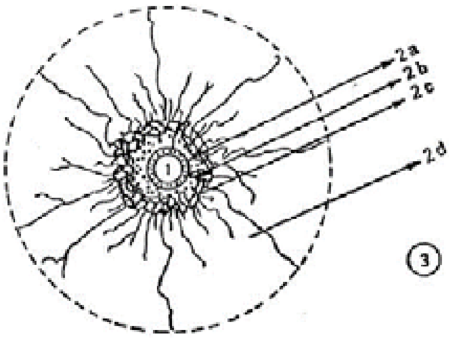

Figure 1 Zone defined in a rock mass around an explosive charge after a blast (according to Atchison, 1968, cited by Dinis da Gama, 2001)

Where: 1 is the explosive cavity; 2 the transition zone; 2a the hydrodynamic zone; 2b is the plastic zone; 2c is the fragmented zone; 2d is the fractured zone; and 3 is the seismic or elastic zone.

The first zone, where the explosion occurs, is originally occupied by an explosive charge (Q) and it is associated with hydrodynamic mechanisms of detonation. The second zone has a greater extent and may be divided into four distinct zones: hydrodynamic, plastic, fragmented and fractured. Finally, there is a third region called elastic or seismic zone comprising non-fractured intact material, indicating that the tension occurring here would be below the elastic limit of the rock. This zone is significant for ground vibration problems resulting from rock excavation with explosives.

In relation to the dynamic behaviour of the rock mass near to explosive charge detonation, Figure 2 shows an example of the study carried out by Holmberg (1982).

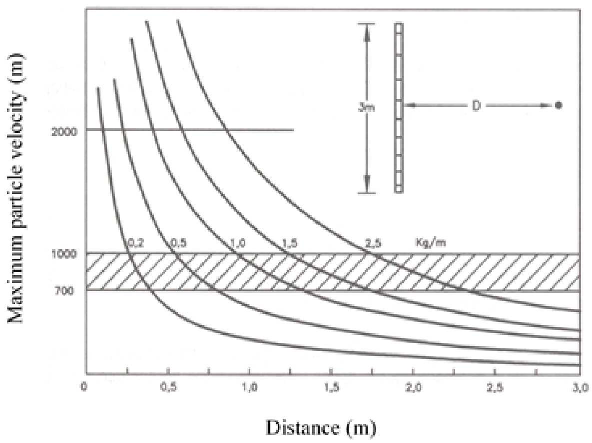

Figure 2 Particle velocity in function of distance and linear explosives concentration to 3 m length of explosive charge and maximum hole at the distance of 3 m. (Holmberg, 1982)

The particle velocity vibration (V) and the propagation velocity of P waves (V_p) show relation with dynamic properties of blasting charges and of the rock mass, and could be expressed by the following equations:

$$V = \sigma_d \left(\frac{V_d}{2\sigma_i} - \frac{1}{\rho_e V_d} \right) \tag{1}$$

$$V_p = \frac{2\sigma_i \rho_e V_d}{\rho_r (\rho_e V_d{}^2 - 2\sigma_i)} \tag{2}$$

Where: σ_d is the dynamic tension of the rock mass, V_d is the velocity of blasting, σ_i is the initial tension of the rock due to blasting the explosive charge, ρ_e is the explosive density and ρ_r is the rock density. The initial tension in the rock σ_i due to the blasting of the explosive charge, is expressed by equation 3.

$$\sigma_i = \frac{\rho_e V_d{}^2}{2(1+r)} \tag{3}$$

Where: r is the transmission ratio results from the division between explosive impedance and rock mass impedance expressed by $r = \rho_e V_d / \rho_r V_P$.

This paper aims to provide a methodology, which could predict propagation of blasting induced vibrations as well as its respective attenuation laws regarding rock mass dynamic characteristics. Furthermore, the human discomfort of the surrounding community were quantified using a sustainability model based on standards and a calculus of the maximum explosive charge per delay was carried to exemplify the methodology.

BLASTING INDUCED VIBRATIONS AND AIR BLAST LAWS, VIBRATION SUSTAINABILITY

The most accepted mathematical model to determine particle vibration velocity was proposed by Dinis da Gama (1993) and Holmberg (2000), which considers the energy (Q) of the vibration source and the distance (D) from the blast as key variables (Equation 4). The coefficients a, b and c are determined by multiple regression analysis from the data obtained from in situ measurements.

$$v = aQ^b D^{-c} \tag{4}$$

Vibration sustainability concept is based on environmental sustainability of open pit mines, and blasting induced vibrations. It aims to prevent damage to sensitive structures and human discomfort in the surrounding communities.

According to the mathematical model to quantify sustainability developed by Navarro Torres et al. (2016), blasting induced vibration is expressed by blasting environmental sustainability index SI_v applied for $x_i \leq X$ condition, expressed by equation (5), as follows:

$$SI_v = 1 - \left| \frac{x_i}{X} \right| \tag{5}$$

Where: x_i is the vibration caused by rock blasting and X is the vibration limit allowed by standards.

The environmental sustainability assessment regarding blasting induced vibration ranges from 0 to 1 and it respects the following conditions:

a) If $x_i = X$ or $x_i > X$, so $SI_v = 0$;

b) If $x_i = 0$, so $SI_v = 1$

Furthermore, based on this quantitative criterion, Navarro Torres et al. (2015) proposed to characterize the environmental sustainability level regarding blasting induced vibrations, as presented in Table 1.

Table 1 Proposed blasting vibration environmental sustainability level

Sustainability level	Colour	SI_v
High		$0.60 < SI_v \leq 1.00$
Medium		$0.20 < SI_v < 0.60$
Low		$0.00 < SI_v \leq 0.20$

Particle velocity vibration and frequency are the most used parameters to assess the side effects of rock blasting. Frequency plays a key role in vibration analysis due to the buildings response depends on the local ground vibration's frequencies (Kahndelwal & Shing, 2006). In addition, ground vibrations, which may cause damage to buildings, are characterized by several international standards, for instance, the Portuguese NP 2074-2015 standard (Table 2) as well as ABNT-NBR 9653:2005, the Brazilian standard (Table 3).

Table 2 Maximum peak particle velocity recommended by Portuguese standard NP 2074:2015

Type of structure	Dominant frequency		
	Frequency <40 Hz	10 Hz < Frequency <40 Hz	Frequency >40 Hz
Sensitive	1.5	3.0	6.0
Current	3.0	6.0	12.0
Reinforced	6.0	12.0	40.0

Table 3 Maximum peak particle velocity recommended by Brazilian standard ABNT-NBR 9653:2005

Frequency range (Hz)	Peak particle vibration velocity limit
4 a 15	Starting at 15 mm/s and increase linearly up to 20 mm/s
15 a 40	Above 20 mm/s increases linearly to 50 mm/s
Above 40	50 mm/s

Brazilian standard ABNT-NBR 9653:2005 for air blast control measured beyond the mining operations area shall not exceed 100 Pa or 134 dBL.

Regarding the comfort of adjacent living areas of mining operations, the ABNT-NBR 9653:2005 Brazilian standard recommends eight procedures, nevertheless does not define allowable vibration values. Consequently, the present study adopts the Queensland Government of Australia Standard.

Transport Noise Management Code of Practice Volume 2: Construction Noise and Vibration of Queensland Government of Australia (2014) – TNMCP, states limits, aiming minimise human discomfort for blasting induced vibrations, as described in Table 4. It was elaborated based on several blasting vibration criteria for human comfort, such as: Australian Standard AS 2187.2:2006, ANZEC 'Technical

Basis for Guidelines to Minimise Annoyance due to Blasting Overpressure and Ground Vibration', 1990, Environmental Protection Act Section 440ZB and EHP EcoAccess 'Noise and Vibration from Blasting', 2006.

Table 4 Human comfort vibration limits to minimise annoyance from blasting of Queensland Government of Australia

Location	Blasting Limit (Resultant PPV)
Buildings of special value or significance (may include historical buidings, monuments)	2 mm/s
Sensitive land uses	Not more than 5 mm/s for 9 out of any 10 consecutive blasts and not more than 10 mm/s for any blast
Occupied non-sensitive sites, such as factories and commercial premises	See AS 2187.2:2006, Table J4.5 (A)

Moreover, CETESB D7.013 São Paulo Brazilian standard recommends the resulting particle vibration shall not exceed 4.2 mm/s measured outside the mining enterprise limits.

METHODOLOGY

The research methodology adopted for blasting induced vibration characterization is based on the four following steps:

1st Step: Intensive in-situ vibration monitoring including open pit blasting area, region between the pit and the surrounding community and the later.

Using seismographs, it was measured the following parameters for each event:

- Particle peak vibration (v)
- Explosive charge per delay for each blast (Q)
- Distance between detonation source and vibration measurement equipment (D)
- Vibration frequency (F);
- Geophone location (UTM coordinates).

2nd Step: Computing vibration laws of local testing, using statistical tools.

In this step, it was carried out a multiple linear regression analysis of the database using equation (6).

$$v = aQ^{b} D^{-c} \qquad (6)$$

Where: a, b, c, are coefficients obtained from multiple linear regression analysis

3rd Step: Calculation and representation of iso values for vibration and proposed sustainability index, using kriging interpolation, for the studied area.

Using the obtained equation in the previous step, values of blasting induced vibrations could be estimated for different locations and explosive charges. This new database allows the interpolation by kriging to infer the spread of values in the studied area and plot contour maps of vibration. The equation (7) and the vibration values obtained were used to generate a database to plot contour maps of sustainability index.

$$SI_v = 1 - \frac{x_i}{X} \tag{7}$$

Where: SI_v is vibration sustainability index, x_i is possible vibration caused by blasting and X is permissible vibration considered by blasting induced vibration standards.

4th Step: Determination of maximum explosive charge per delay for blasting induced vibration prediction.

In order to mitigate possible structural damage and human discomfort, equation (8) was used to calculate maximum charge per delay.

$$Q_{\max} = \frac{\sqrt[b]{\left(v_L . D^c\right)}}{a} \tag{8}$$

Where: Q_{max} is the maximum explosive charge per delay and v_L is permissible vibration for structure damage and human discomfort recommended by blasting induced vibration standards.

RESULTS AND DISCUSSION

In order to validate the methodology of iso-vibration and iso-sustainability for blasting induced vibration prediction in open pit iron mine, the study was conducted in a Vale's mine located in Minas Gerais, Brazil.

The first step of the methodology was carried out using simultaneously eighteen GEOSONICS seismographs which recorded 4170 events in 57 different points, also using explosive charges per delay ranging from 800 kg to 2100 kg of emulsion (Figure 3).

Figure 3 Monitoring points of vibration

The vibration propagation law (Equation 9) resulted from the field data with a good correlation ($R^2 = 0.86$). The Figure 4 depicts the process carried out using the software LABFIT and it represents the scatter plot of database.

$$v = 900Q^{0,27}D^{-1,27} \tag{9}$$

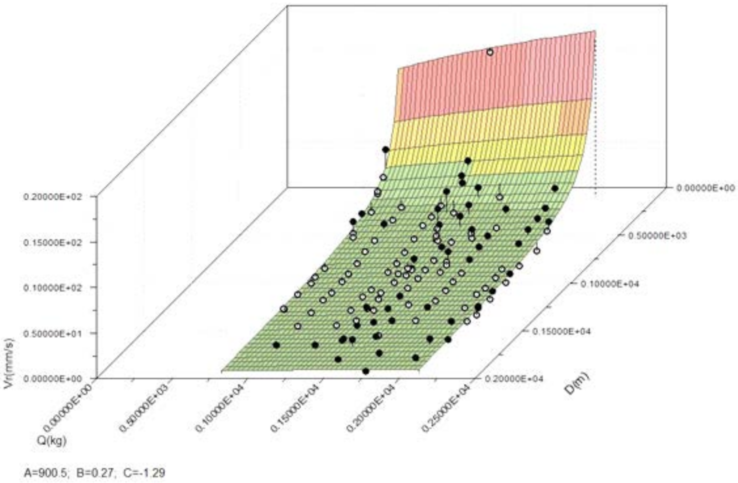

Figure 4 Vibration (Vr) versus explosive charge per delay (Q) versus distance (D).

The contour maps (Figures 5 and 6) were obtained using the software SURFER, as described by the third step. They illustrated the spread of the vibration and sustainability index as proposed by the methodology.

Figure 5 Iso-values of vibrations contour map

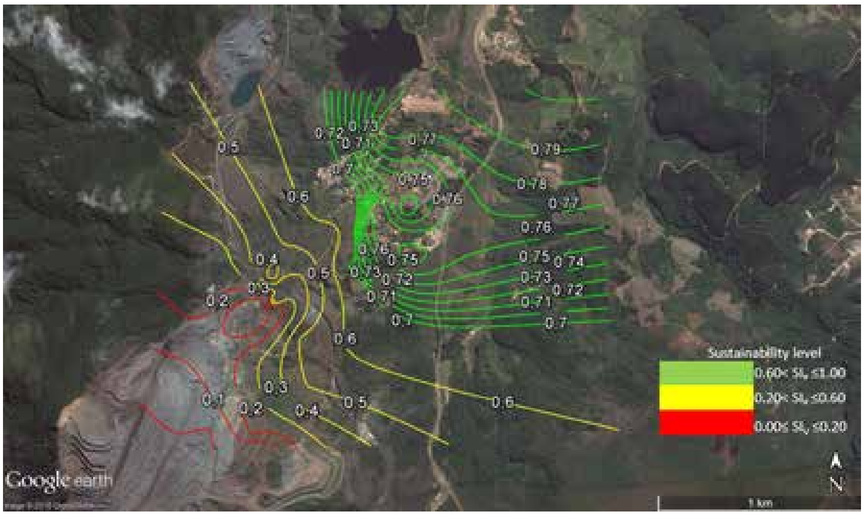

Figure 6 Iso-values of sustainability index contour map

Regarding the Brazilian ABNT-NBR 9653:2005 and TNMCP Australian standard, it is shown in Table 4 that for the studied case, the acquired measurements did not exceed the threshold. The registered frequencies were all below 10 Hz.

Table 4 Summary of measurements regarding ABNT-NBR 9653:2005 and TNMCP Australian standard

Assessment indicator	Standard	Threshold limit(mm/s)	Local assessment	Observation
Iso-vibration (structural damage)	ABNT-NBR 9653:2005	15	0.3 - 0.8	No damage
Iso-vibration (human discomfort)	TNMCP	2	0.3 – 0.8	No human discomfort
Iso-sustainability (human discomfort)	TNMCP	2	0.7 – 0.79	Good level of sustainability

Finally, it was calculated the maximum admissible explosive charge per delay according to equation (9). Brazilian ABNT- NBR 9653: 2005 does not mention the types of structures, which could be affected by blasting induced vibrations, t states only the maximum admissible peak particle velocity regarding the frequency. Therefore, to calculate the maximum explosive charge per delay it was used the Portuguese standard NP 2074, for instance. It defines the maximum peak particle velocity considering the frequency and the type of structure (1.5 mm/s for sensitive buildings, 3 mm/s for ordinary buildings and 6 mm/s for resistant building assuming frequencies below 10 Hz). The maximum explosive charge results are depicted in Figure 7.

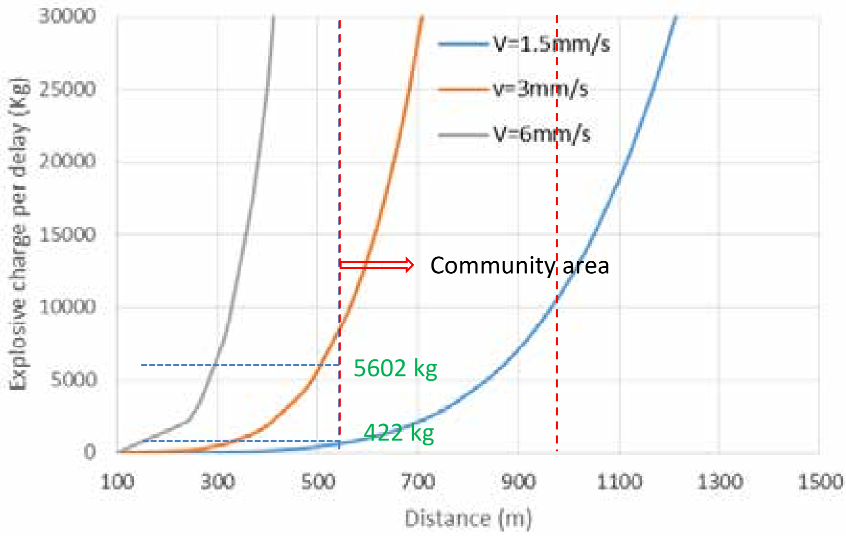

Figure 7 Maximum explosive charges per delay regarding NP 2074

As shown in Figure 7, the maximum explosive charges per delay were calculated considering NP 2074. For sensitive structures, 422 kg is recommended for a building 500m away from the blasting source and 5602 kg for ordinary buildings. In case of resistant buildings, there is no concern, the maximum explosive charge per delay once the results recommend an extremely high charge. The mine operations in Vale’s iron ore mine, commonly use 250 kg of emulsion per hole, consequently it would correspond to approximately 2 holes detonated instantaneously, for sensitive buildings and 23 holes for ordinary buildings in the community.

CONCLUSION

The methodology proved consistent and feasible, fieldwork was successfully carried out by the team. Dependent parameters of the rock mass characteristics (a, b and c) show congruence with those found in the literature. It can be said that the characterization of the dynamic behavior of the land was carried out correctly.

The propagation law of vibration found have a high correlation coefficient, which indicates the representativeness and offers reliability to the database as well as their results. Thus, it is possible to predict the local propagation of blasting induced vibrations.

The monitoring campaign did no present structural damage, according to ABNT-NBR 9653:2005, due to blasting induced vibrations in the this Vale's iron mine and surrounding community.

Human discomfort is a subjective concept. Although, several standards, as well as TNMCP Australian standard, point out maximum limits. The database obtained do not exceed thresholds limits proposed by this standard.

The blasting operations has resulted in a good level of sustainability, considering the human discomfort at the community area. The iso-sustainability map works as an important marker to open pit mining operations, since it evaluates the human discomfort in a quantitative manner.

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LINEAR PROGRAMMING IN THE ANALYSIS OF ECONOMIC FEASIBILITY OF MULTIMINE PROJECT

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ABSTRACT

Production costs of iron ore varies according to the specificity of each mine. In the mining phase, some of the largest operational costs are associated to the haulage distance and the stripping ratio. In this context, the implementation of the multiple mines project and the conveyor belts system at the Itabira Complex Mines, Brazil, aims to optimize these two indicators. The multiple mines methodology focuses on reducing the overall stripping ratio of the complex, and the implementation of conveyor belts mitigates the increase of the haulage distance. Two mines and a stockpile are available to feed three different concentration plants. The goal is to achieve the best way to feed the plants considering the economic aspects. A linear programming model is proposed to determine the best iron ore mass feeding the plants from different mines and the stockpile, taking into account the least haulage distance and the smallest stripping ratio. This optimization model was developed using LINGO Software to interface with EXCEL and evaluate several scenarios. The results showed that the method of multiple mines is a technically and economically viable to be implemented at the Itabira Complex.

KEYWORDS

Linear programming, multiple mines, conveyor belts

INTRODUCTION

In the current situation iron ore prices decrease, saving costs and adding value to the products are the pillars for the survival of companies and mines in the mining market. With this focus, Vale S/A built a new processing plant and is adapting its old plants in Itabira complex, Brazil, to receive low grade iron ore to produce a pellet feed with high iron content (62% Fe) with higher sale value compared to standard products.

Production costs of iron ore varies according to the specificity of each mine. In the mining phase, some of the highest operational costs are associated to the haulage distance and the stripping ratio. With the possibility of feeding all the processing plants with a common iron ore grade, a new feeding plant method can be applied: the multiple mine method. This method combines multiples mines feeding various processing plants ensuring an overall stripping ratio (REM) to the mining complex, and in the case of Itabira Mining Complex represents a significant reduction in this index. Due to the large expansion of the Itabira mines, the deploying multiple mine method planning created a problem, the significant increase in the haulage distance (DMT), so to mitigate this situation it was proposed an in pit crushing/conveyor belt (TCLD) system.

The Itabira Mining Complex needs a new feed processing plant schema for the years 2017 to 2021, considering that after 2017 the plants will be fed with low grade iron ore (42% Fe). Following this new processing plants configuration, a new proposition of run of mine distribution was carried out using the multiple mines method as well as an implementation of mobile crushers and conveyor belts. The goal was to achieve the best result to feed the plants considering the economic aspects. A linear programming model was proposed to determine the best iron ore mass feeding the plants from different mines, taking into account the minimum haulage distance, the smallest stripping ratio and guaranteeing the grade of the run of mine (ROM).

The mathematical model took in consideration the ore grades, the stripping ratio and the haulage distances between the mines and the processing plants. This optimization model was developed using LINGO Software to interface with EXCEL. An objective function was developed to evaluate the different scenarios helping to define the best ore processing plant feeding schema. The results showed that the method of multiple mines is technically and economically viable method for mine planning at the Itabira Complex.

LINEAR PROGRAMMING IN MINE PLANNING

The multiple mine method can be seen from the operational research perspective. Operational Research (OR) is the study applied to problems that include the management and coordination of the activities and is considered one of the most appropriate scientific method to investigate similar problems. Following this idea, the OR becomes an important tool for optimization and rationalization of resources, aiding the decision-making process.

There are several models and mathematical programming tools for solving the proposed problem. Among them, the linear programming (LP) has been the most studied method. This method is one of the most versatile and simple to be used and some examples of applications in mining are described.

Carvalho Junior (2005) used a linear programming model to represent the all production process associated with coal production, from mining to marketing. This model seeks to quantify the risk associated with service product specifications of each customer of Copelmi´s mining company.

Moraes (2005) proposed a linear goal-programming model to optimize the composition of lots of iron ore. The model aims to determine the recovery of stored product , so that the mixture meets the limits of quality specifications and predetermined quantity by the client and respects the operational site restrictions in the Cauê Plant, located in Itabira Complex.

Carvalho, Koppe and Costa (2012) analyzed the impact of the uncertainty associated with the input parameters in a mine planning optimization model. A mathematical model representing the coal production process was taken in consideration. This model was optimized using the linear programming concept whereby the best solution was disturbed by the stochastic behavior of one of the main parameters involved in the production process.

Martins (2013) created a linear programming model responsible for allocating the loading equipment in the mining faces and inform travel distances that each truck must perform, respecting restrictions like ore grade, ore particle distribution and masses to be delivered in each discharge point. This model was applied to the Brucutu mine, Minas Gerais, Brazil.

MATHEMATICAL MODELLING

Prado (1999) describes LP as a technique that allows the merging of several variables based on a linear function of effectiveness (objective function), while simultaneously satisfying a group of linear restriction for these variables. According Ravindran, Philips and Solberg (1987), the construction of a LP model follows three basic steps:

(i) Identifying the unknown variables to be determined and representing them using algebraic symbols;

(ii) Listing all the restrictions of the problem and expressing them as linear equations in terms of the decision variables defined in the previous step;

(iii) Identifying the objective or criterion for optimizing the problem and representing it as a linear function of the decision variables. The objective can be either a maximizing or a minimizing function.

Following these steps a LP model was created to represent the mines, the stockpile and the concentration plants from Itabira Complex. The ROM comes from the mines that compound the complex: Minas do Meio (MM) and Conceição (CE), and from the Itabirite Stockpile (DEP _IC). The aim of this model is to seek for the smallest stripping ratio, the shorter haulage distance to deliver the best grade and the required mass of ROM.

Inputs

The first step to create the model is determining the inputs:

Minas(i) = Mines;

Us(j) = Concentration Plants;

Parâmetros(k) = Parameters;

Rom_i = Run of mine, iron ore mass, in tons, available at the mine i;

Cap_Usinas_j = Iron ore quantity to be offered for each plant j;

Rem_i = Stripping ratio of each mine i;

$disp_j$ = Availability of concentration plant j;

$dest_{ij}$ = If the mine i can feed the the concentration plant j;

$meta_{jk}$ = Grade target of each plant j to each parameter k;

wm_k = goal deviation weight of the k parameter;

$teor_{ik}$ = Grade available in each mine i for each parameter k; dpmik = positive deviation from target;

dnmik = negative deviation from target;limite = mass limit in tons which can be sent to Plant Cauê from Conceição mine and Itabirite stockpile simultaneously.Decision variables

From the input data, the decision variable was defined as:
xij = ore quantity in tonnes to be transported from the mine or stockpile i to the plant j.

Objective function

Set the decision variable, the next step of mathematical modeling is the definition of the objective function represented by Equation 1, and aims to minimize the stripping ratio and the haulage distance and ensure the minimum iron content that feeds the plants, as expressed in Equation (1):

$$\min \sum_{i \in Minas} \sum_{j \in USinas} dist_{ij} x_{ij} + \mathrm{Re}\, m_{ij} x_{ij} + \sum_{j \in USinas} \sum_{k \in Parametros} wm_k dpm_{jk} + wm_k dnm_{jk}$$

Restriction

For this study, the set of constraints corresponding to the boundary conditions is formed by:

Use of ROM of each mine or iron ore stockpile

It should be used in most of the available ROM at each mine or stockpile.

$$\sum_{j \in USinas} x_{ij} \le Rom_i \quad \forall i \in \mathrm{Minas} \tag{2}$$

Plant capacity

The capacity of each plant should be respected.

$$\sum_{i \in Minas} x_{ij} = cap\_Us_j \quad \forall j \in Us \middle| disp_j = 1 \tag{3}$$

Availability of plants for particular mine or stockpile

Unavailable plants can not be used.

$$\sum_{i \in Minas} x_{ij} = 0 \quad \forall j \in Us \middle| disp_j = 0 \tag{4}$$

Mine or stockpile ore transport for a plant

There is ore transport from the mine or stockpile i to the plant j if dest(i,j)=1.

$$\sum_{j \in USinas} x_{ij} = 0 \quad \forall i \in Minas \middle| dest_{ij} = 0 \tag{5}$$

The feed limit of Cauê plant with ore from the Itabirite stockpile and Conceição mine

Mass from Conceição (CE) + Itabirite stockpile (DEP_IC) sent to Cauê plant must be less than or equal to limit.

$$\sum_{i \in Minas} x(M\_CE, U\_CA) + x(M\_DEP\_IC, U\_CA) \ge \lim ite \tag{6}$$

Level of assurance of grade in the plants feed

The target for the control parameter must be satisfied wherever possible.

$$\sum_{i \in Minas} (teor_{ik} - meta_{jk}) x_{ij} - dpm_{lj} + dnm_{lj} = 0 \quad \forall k \in Parametros, \forall j \in Us \tag{7}$$

Ensuring non-negativity and completeness

$$x_{ij}, dnm_{ij}, dpm_{ij} \geq 0 \quad \forall i \in Minas, \ \forall j \in \mathrm{Us} \tag{8}$$

Computational implementation

The model based on mathematical programming presented in this paper was implemented using the software and Optimizer *LINGO*, version 10.0, of *Lindo Systems Inc.* interfaced with *Microsoft Excel*, using spreadsheets.

THE CASE STUDY

To quantify the improvement by using the multiple mines method, a comparison was made between the haulage distance and stripping ratio generated by the optimizer with the indicators of the conventional mining method appointed by the Medium Term Planning. In a second step, all improvements were converted in money terms and those results were compared with the current mining method, showing the best operational viability of iron ore mass transport systems.Inputs

After 2016 all the Itabira plants will be able to receive the run of mine with grades of 42% Fe. The complex capacity is 74Mt of ROM, this capacity is shared by 26Mt for the Cauê Plant (CA) and 48 Mt for both Conceição Plants (CEI and CEII – 24Mt each).

The mines and the stockpile will have a variable availability of ROM in the period 2017-2021(Table 1). The grades of the mines and stockpile are represented in the Table 2.

Table 1 – Availability of Run of Mine: CE – Conceição Mine, MM – Minas do Meio Mine and DEP_IC – Itabirite stockpile

	Mass Mt				
Origin/ Year	2017	2018	2019	2020	2021
CE	53.90	54.00	54.00	54.00	54.20
MM	35.70	36.00	36.00	36.00	36.00
DEP_IC	14.00	14.00	14.00	14.00	14.00
Total	103.60	104.00	104.00	104.00	104.20

Table 2 – Grade of Run of Mine: CE – Conceição Mine, MM – Minas do Meio Mine and DEP_IC – Itabirite stockpile

		Grade Run of Mine				
Year	Mine	Fegl %	Sigl %	P %	Algl %	Mngl %
2017	CE	42.32	37.07	0.020	0.92	0.111
	DEP_IC	41.00	38.55	0.015	0.50	0.054
	MM	42.92	36.66	0.018	1.08	0.146
2018	CE	42.02	37.50	0.020	1.02	0.890
	DEP_IC	41.00	38.55	0.015	0.50	0.054
	MM	42.71	36.76	0.018	1.08	0.146
2019	CE	42.32	37.07	0.020	0.92	0.111
	DEP_IC	41.00	38.55	0.015	0.50	0.054
	MM	42.93	36.54	0.016	1.06	0.151
2020	CE	42.48	38.33	0.024	0.92	0.140
	DEP_IC	41.00	38.55	0.015	0.50	0.054
	MM	43.12	35.63	0.015	1.07	0.182
2021	CE	42.49	38.28	0.024	0.92	0.140
	DEP_IC	41.00	38.55	0.015	0.50	0.054
	MM	43.58	34.93	0.015	1.08	0.185

Fegl: total Fe Sigl: total SiO_2 P:phosphorus Algl: total Al Mngl: total Mn

The stripping ratio of each mine was obtained from the long-term planning data, which is represented in Table 3. For the stockpile the stripping ratio was considered null.

Table 1 – Mine stripping ratio: CE – Conceição Mine; MM – Minas do Meio Mine

Mine/ Year	2017	2018	2019	2020	2021
CE	*0.41*	*0.39*	*0.39*	*0.45*	*0.45*
MM	2.17	2.05	1.89	1.60	1.65

The haulage distance was obtained in long-term mining plans, considering all the possible destinations for the Itabirites during the five years studied. An alternative transportation scenario was taken into consideration; this is the ore truck transportation to the semi mobile crusher BSM Camarinha (semi mobile crushing system).

This alternative scenario in the multiple mine method considers that the ore from Conceição mine and the stockpile feeds the Conceição I and Conceição II originals crushers and the BSM Camarinha, the BSM receives only ore from Minas do Meio mine. The haulage distance of each origin to the destiny is showed in table 4.

Table 4 – Haulage distance

	Haulage Distance Km				
Origin/Destiny	2017	2018	2019	2020	2021
CE/ CE1	*3.76*	*3.66*	*3.47*	*3.82*	*4.16*
CE/ CE2	2.26	2.32	2.52	2.86	3.22
CE/ BSM CAM	*10.80*	*10.70*	*10.51*	*10.86*	*11.20*
DEP_IC/ BSM CAM	7.70	7.70	7.70	7.70	7.70
DEP_IC/ CE2	*0.80*	*0.80*	*0.80*	*0.80*	*0.80*
DEP_IC/ CE	2.95	2.95	2.95	2.95	2.95
MM/ BSM CAM	*3.46*	*2.84*	*2.48*	*2.17*	*1.90*

For the case study of multiple mine method it was considered only a combination of constraint between mines and concentration plants, therefore it is not allowed to feed the CEI and CEII plants with ROM from the Minas do Meio. This matrix is represented in the table 5.

Table 5 – The plants feed matrix

Origin/Destiny	CEI	CEII	CA
CE	1	1	1
DEP_IC	1	1	1
CA	0	0	1

ANALYSIS OF RESULTS

The results of the programming model proposed in this case study will be analyzed following certain procedures:

• Initially operational indicators were analyzed;
• Subsequently, capital gains related to scenario were analyzed.

Results of operational indicators

To quantify all gains, the scenario was compared to the original scenario presented by the long-term planning. The haulage distance (DMT), stripping ratio (REM) and compliance with the quality of at least 42% of iron in the feeding plant were the operational indicators considered in this analysis.

The mathematical model returned the following results for DMT and REM compared with the values of the original plan (Table 6). This scenario shows a decrease of stripping ratio and a small increase in haulage distance, but guarantees delivery of the grades needed to feed the plants.

Table 6 – The DMT and REM results

	Year	2017	2018	2019	2020	2021
Original Plan	REM	1.11	1.05	0.99	0.91	0.93
	DMT Km	4.30	4.40	3.95	4.10	4.21
New Scenario	REM	0.50	0.67	0.44	0.46	0.46
	DMT Km	4.57	3.63	4.52	4.72	4.96

For this scenario, it was obtained the following annual grades for the plants, as presented in table 7.

Table 7 – Annual grades for the plants feed

		Grade Run of Mine				
Year	Plant	Fegl %	Sigl %	P %	Algl %	Mngl %
2017	CEI	42.27	37.13	0.020	0.91	0.109
	CEII	42.12	37.30	0.020	0.86	0.102
	CA	42.00	37.50	0.018	0.81	0.100
2018	CEI	42.00	37.52	0.020	1.01	0.874
	CEII	42.00	37.52	0.020	1.01	0.874
	CA	42.00	37.50	0.017	0.84	0.108
2019	CEI	42.32	37.07	0.020	0.92	0.111
	CEII	42.05	37.37	0.019	0.84	0.099
	CA	42.00	37.47	0.018	0.81	0.100
2020	CEI	42.48	38.33	0.024	0.92	0.140
	CEII	42.30	38.36	0.023	0.87	0.129
	CA	42.00	37.80	0.018	0.78	0.113
2021	CEI	42.49	38.28	0.024	0.92	0.140
	CEII	42.42	38.29	0.024	0.90	0.136
	CA	42.00	37.67	0.018	0.75	0.108

Fegl: total Fe Sigl: total SiO_2 P:phosphorus Algl: total Al Mngl: total Mn

Results of capital gains

For a better analysis of operational indicators, all indexes will be transported to the same basis of comparison, and the costs associated with the base that best correlates all these indexes. The costs associated with the haulage distance and the stripping ratio ore accumulated in 2015 until May were obtained from internal controls of VALE S/A. These values were associated with the gain or demerit of 1% of each indicator studied, therefore, all parameters were transformed into percentage values of each indicator reported by long-term planning.

The scenario generated savings of approximately US$ 100 million. Figure 1 shows the annual evolution of the savings of operating indexes. Positive values represent the economy compared to conventional scenario and negative values represents an increase in the conventional scenario costs.

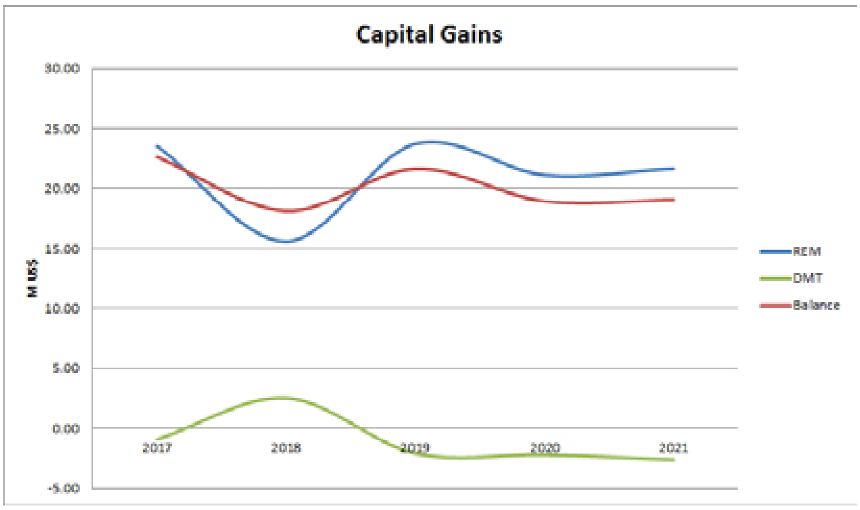

Figure 1 - Annual evolution of the saving of operating indexes

CONCLUSIONS

This paper aimed to use the OR to analyze the improvements of implementation of the multiple mines method and conveyor belts at Itabira Complex. A mathematical model that optimizes the operating variables to help the feasibility analysis on current mining methods and feeding the plants was successfully developed.

The results showed a decrease of global stripping ratio and a small increase in haulage distance, but guarantees delivery of the grades needed to feed the plants as well as showed a significant savings of approximately US$ 100 million. So this study ensures the feasibility of the multiple mines method and the conveyor belts at Itabira Complex.

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MINE PROJECT EVALUATIONS IN THE RISING OF UNCERTAINTY: REAL OPTIONS ANALYSIS

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ABSTRACT

This paper presents an economic evaluation model for open pit mining projects based on real option valuation (ROV) approach. Price movements are simulated using binomial tree method, whereas costs uncertainties are modeled using ROV technique. We implement the proposed methods for a hypothetical gold mine case study. Results indicate that management might be inclined to commit the investment if relying on the NPV rules as if the NPV is positive in value. Nevertheless, results obtained from ROV analysis suggested that project might have higher values if deferring the investment into a specified timing and committing it once uncertain condition has become resolved.

KEYWORDS

DCF Methods, ROV Approach, System Dynamics Modeling Methods, Uncertainty

1. INTRODUCTION

Evaluating economic values of mining projects is a complex task. Normally, mine analysts use the standard DCF method to evaluate the project value in prefeasibility and feasibility studies stages. The drawbacks of using the DCF have been its utmost assumption that the discount rate is all-in-one parameter that covers up project risks. The DCF method forecasts the future cash flows, and discounts them by the discount rate, then subtracts it by the investment costs to estimate the NPV of a project. Projects with positive NPV are accepted while projects with negative NPV are rejected. The investment decisions based on the DCF-NPV rules are an irreversible decision feature in which decision is irreversible. In other words, it overlooks the value of managerial flexibility for readjusting the decisions by reacting to changes in uncertain conditions. Many attempts consist of Samis et al. (2007), Moyen et al. (1996), Laughton et al.(2000), Frimpong and Whitting (1997), Tufano (1998), Steffen (1997), Koushavanh et al.(2014), and Azimi et al. (2013), have incorporated the uncertainty variables in the project valuation models. Yet, most of the mine project evaluations are performed in the spreadsheet. The seminal works of Black and Scholes and Merton (1973) have a significant contribution to the origin of the ROV method. In fact, a real option approach derives from financial option pricing. Brennan and Schwartz (1975) published the first groundbreaking paper adopted option pricing theory to valuing the copper mines. Numerous works include Cortaza et al. (2008), Dimitrakopoulos and Sabour (2007), Samis et al. (2001), Sabour (2001), Haque et al. (2014), Mayer and Kazakidis (2007), Dixit and Pindyck (1995), Longstaff et al. (2001), Schwartz and Trolle (2010), Trigeorgis (2002), Slade (2001), Mardones (1993), Martinez et al (2009), have performed ROV to evaluate the values of energy and mineral resources investments projects. This paper applied the real option decision framework (RODF) presented by Inthavongsa et al. (2016). The RODF offers the management the right, but not obligation to maintain, defer, expand, and shutdown, its operation upon the predefined conditions. Such an investment opportunity comes with costs and it payoffs in options premium. It is implied that to acquire an option, one must pay a premium upfront, including future expenses if the option is exercised. Previous works consist of Cooke (2004), Tan et al. (2010), Kasiri and Sharda (2013), Johnson et al.(2006), Ford and Sobek (2005), Barghav and Ford (2006), have incorporated ROV and SD methods to evaluate economic value of various projects. Binomial option pricing is an alternative approach to solve real options problems (Cox et al., 1979). Braodao et al. (2005b) developed a decision tree based on binomial method, Ajak and Topal (2015) applied binomial method to value switching options for the mines. Culik (2015) presented binomial option pricing models with changing volatility. This article proposes an integration of the real options valuation (ROV), binomial decision tree (BDT), and system dynamics (SD), to allow for the dynamic evaluation process of sequential decisions for mine projects. In this study, the complementary strengths of the SD method and the binomial tree are employed to extend the capability of ROV in project evaluations. In fact, combination of evaluation methods extends capabilities of each technique. ROV provides valuation technique as a continuing process to remedy the uncertainties, in which managerial flexibility is considered from the start and during the implementation of projects. SD comes with not only its predictive capability but also the applicability to support the design of framework. SD modeling methods can capture the dynamic nature of stochastic parameters of model variables. It has complementary advantages in representing mathematical complex in forms of graphical representations. The binomial decision tree offers an intuitive and transparent technique for analyzing sequential decisions based on the event tree analyses. Mun (2002) emphasized that no matter which models are used, the binomial lattice is a more powerful representations of decisions than other mathematical models. Binomial lattice can be converted into a series of decision nodes. These nodes correspond to optimal decisions that should be made under specified timing and underlying variable conditions. Hence, it is expected that the successful combination of these methods would address some limitations in mine project evaluation problems. The remainder of this paper is organized as follows: section 2 describes the integration of evaluation methods. Section 3 presents the cost uncertainty model. Section 4 provides economic decision policies. Section 5 shows the implementation of proposed method. Section 6 analyses the results and discusses the key findings. Section 7 presents concluding remarks.

2. INTEGRATION OF EVALUATION METHOD

2.1 Real Options Valuation Approach

Application of ROV approach to a mine planning task has been realized for its flexibility not covered by a traditional DCF method. By definition, the mine investors (option holders) own the right, but not obligation to defer, suspend, expand, and terminate its operations, prior to and during the time of implementation of the projects. There are two types of options namely, calls (buy), and puts (sell). Reader are referred to Black and Scholes (1973) for original source, and Trigeorgis (1996), and Tomkins (1994), for more information. Real options applied option-pricing theory to evaluate the physical or real assets as opposed to financial options. If X is the strike price (the mining costs) and S_T is the final price of the underlying asset (the payoff value), the value of the call and put option can be written as (Hull, 1993).

$$Call\ payoff = max[S_T - X, 0] \tag{1}$$

$$Put\ payoff = max[X - S_T, 0] \tag{2}$$

The conditional syntax (Eq. (1) & (2)), reasons the logic strategies for the option holders to exercise their rights without an associated obligation to do so. This reflects the fact that the call option will be exercised if $S_T > X$, and will not be exercised if $S_T < X$. Similarly, the put option will only be executed if $X > S_T$ and will not be executed if $X < S_T$ (Hull, 1993). In real practices, the values generated must be greater than costs of executing an option. It is implied that management will commit the option if the value outweigh the upfront expenses. In this paper, ROV is performed to analyze the values of option-embedded in mining investment projects.

2.2. System Dynamics Modeling Approach

System dynamics (SD) provides flexible tool-suites to formulating a behavioral model capable of reproducing, by itself, the dynamic problem of interest. Fig. 1 shows the basic element of SD structures, which comprises of stock (level) and flows (rates). Stocks are fundamental to generating behavior in a system; flows cause stocks to change. Readers are referred to Sterman (2000) for more information. Tan et al. (2010) pointed out that the benefits of using SD for evaluating a risky project involved managerial flexibility is the increased realm of the project model itself. The net rate of change of any stock, its derivative, is the inflow less the outflow, defining the differential equation as follows:

$$\frac{d(Stock)}{dt} = Inflow(t) - Outflow(t) \tag{3}$$

$$d(Stock) = Stock(t_0) + \int_{t_0}^{t} (Inflow(t) - Outflow(t))dt \tag{4}$$

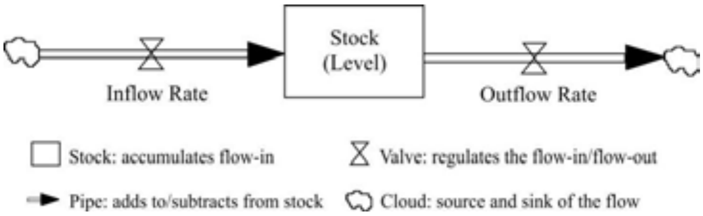

Fig. 1 SD structures

In this article, SD is used to simulate the cash flow streams of the projects. Monte Carlo sensitivity analysis is performed to generate the confident bands of the project values distribution. We used SD methods and system dynamics software, Vensim®DSS6b software, to model the ROV into graphical representations. The SD contains the built-in conditional statements that are suitable for solving the call and put syntax given in Eq. (1) & (2) (see section 2.1). It is important to note that, in this paper, the SD stands as the tools for modeling, and for solving the ROV problems.

$$IF: \{X\} : THEN: \{Y\} : ELSE: \{Z\} \tag{5}$$

2.3 Binomial Decision Tree Method

Cox et al. (1979) developed the binomial methods to valuing financial options using discrete-time option pricing techniques. Metal price uncertainty can be modeled using binomial lattice method. Projecting metal price evolution assists determining the optimal time to exercise an option. Let S_t be the spot price at time t, S_0 is the price of underlying asset at time t_0. The initial price (S_0), is assumed to following the up (u) and down (d) movements with probability (p) for up moves and $(1 - p)$ for down moves, respectively. The size of binomial tree is defined by the upstate (i) and downstate (j), and the time period (t_n), which known as the factorial of binomial. Underlying asset price will simply change at the discrete time: $t_1 = dt$, $t_2 = 2dt$, until $t_n = ndt = T$, where T is the maturity date of an option, and hence, the time step can be determined by $dt = T/n$. Fig 2(A) shows the three steps binomial tree for recombining binomial tree on the left hand side and for non-recombining tree on the right hand side (Figure 2(B)). In this paper, we provide a brief information about binomial method and refer the readers to Cox el al. (1979) for the original source, and Brandao et al. (2005), Copeland and Antikarov (2001), Mun (2006), and Rubinstein (1994), for more information. In this paper, we use binomial tree to analyze the optimal timing for decision-making.

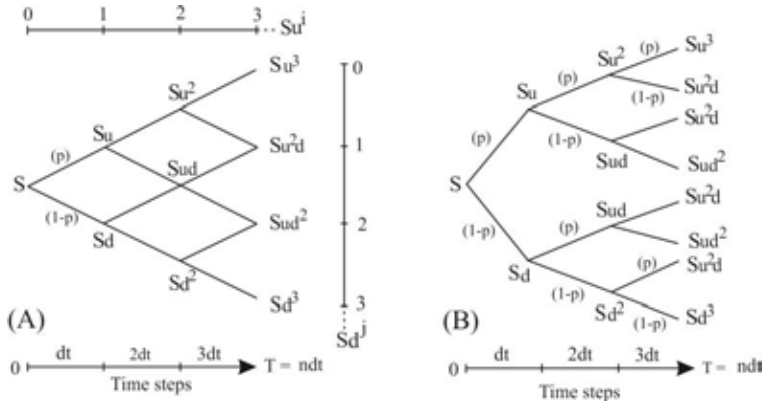

Figure 2 (A) Recombining binomial tree and (B) non-recombining binomial tree

$$S_t = S_0 * u^{i-j} * d^j \tag{6}$$

where

$$u = e^{\sigma\sqrt{dt}}, \qquad d = e^{-\sigma\sqrt{dt}} = 1/u \tag{7}$$

3. COST UNCERTAINTY MODEL

In real practices, cost uncertainty likely occurs within operating sections. It particularly occurs in the mining entity and the processing body. The mines may produce under targeted production rate, or the other way around, it may produce over the targeted production rate. In addition, cost uncertainty can be triggered by the exogenous factor, for instance, the instantaneous disease outbreaks in the regions that highly prone to this kind of risk, such as the Ebola viruses spread in South Africa and the vicinity. In this case, costs of shortage of skilled-labor and the cost of transportation might be increased and escalated rapidly. In this unusual situation, ROV technique would suggest the mines to temporary suspend its operations until the disease outbreak has completely stopped. However, such viruses-triggered cost uncertainty is beyond the scope of this paper. Thus, it is important to take into account the cost uncertainties, which might influence value of a project. This study incorporates the cost of under production and over production into models. If such costs incurred then these costs must be added into the total costs of production (C_{pro}) per ton, else, it is zero. Let $Q(q,c)$ be the total metal produced, $M(q,c)$ be the total material mined, M_c be the mining costs , $H(q,c)$ be total material processed by the mill, P_c be processing costs, and R_c be the refining costs. The proposed cost models included cost can be written as follows:

$$C_{pro} = M(q,c) * \left[M_C + \left(C_{up}^{M}(t) \middle| C_{op}^{M}(t), 0\right)\right] + H(q,c) * \left[P_c + \left(C_{up}^{P}(t) \middle| C_{op}^{P}(t), 0\right)\right] + Q(q,c) * \left[R_c + \left(C_{up}^{R}(t) \middle| C_{op}^{R}(t), 0\right)\right] + C_{fixed} \tag{8}$$

where C_{up}^{M} = cost of underproduction incurred in mining section; C_{op}^{M} = cost of overproduction incurred in mining section; C_{up}^{P} = cost of underproduction incurred in processing body; C_{op}^{P} = cost of overproduction incurred in processing body; C_{up}^{R} = cost of underproduction incurred in refinery entity; and C_{op}^{R} = cost of overproduction incurred in refinery entity.

The cost of underproduction at time t can be calculated using Eq.(6):

$$C_{up}(t) = M(q,c) \times \left(C_{UC}(t) \times UB_f(t)\right) \tag{9}$$

$$UB_f(t) = \frac{Under\ budget\ cost - Actual\ cost}{Allocated\ budget - Actual\ cost}$$

The cost of overproduction at time t can be calculated using Eq.(7):

$$C_{op}(t) = M(q,c) \times \left(C_{OC}(t) \times OB_f(t)\right) \tag{10}$$

$$OB_f(t) = \frac{Over\ budget\ cost - Actual\ cost}{Allocated\ budget - Actual\ cost}$$

Where $C_{UC}(t)$ is the penalty charge per under-produced tons of ore, and $C_{OC}(t)$ is the penalty charge per over-produced tons of ore. UB_f is under budget factor at time t, and OB_f is the over budget factor at time t.

3.3 Cash Flow Estimation

Cash flow calculation is different from one mine to another mine. We know that CF is cash streams of cash inflows subtracted by cash outflows in different time intervals, so that we write

$$CF_t = \left(S_t * Q(q,c) - C_{pro} - D_C\right)dt \tag{11}$$

where $Q(q,c)$ is total metal produced, S_t is metal price movements under binomial scheme.

In case after subtracting variable costs and associated fixed costs, project cash flow (before tax) turns negative, in this case, we do not deduct for income tax. If cash flow turns positive, the income tax must be deducted. Let D_C be the depreciation costs; hence, we can calculate free cash flow after tax using the conditional arguments below (Dehgahani et al. 2014).

$$CF_t = \begin{cases} CF_{t+\Delta t}(1-T_t) + D_C\,, & if\ CF_t > 0 \\ CF_{t+\Delta t} + D_C\,, & if\ CF_t \leq 0 \end{cases} \tag{12}$$

4. ECONOMIC DECISION POLICY

In this paper, we employed the methods presented by Inthavongsa et al. (2016) to evaluate an economic value of the hypothetical case study. Let $E[V_{project}]$ be the expected value of project, which follows the operating policy in which $E[V_{project}(V_M|V_D|V_E|V_S)]$. Where $V_M = V_0$ represents the maintain option value or base-scale mine operation value; V_D denotes deferral option value; V_E signifies expand option value; and V_S embodies the shutting down option value. We assumed five strategic operating policies depending upon the conditions at the time of evaluating the economic value of a given mine. The transition from $\{V_M|V_D\}$ implies that the deferral option is activated, and $\{V_D|V_M\}$ implies that management maintains the operating option with a decision to invest. $\{V_M|V_E\}$ represents an operating mine proceeds to expansion. $\{V_M|V_S\}$ represents the base-scale operation proceeds to the shutdown option. $\{V_M, V_E|V_S\}$ signifies that the base-case operation after expansion is turned into the shutdown option, implying that all operations are prepared for closure. Readers are referred to Inthavongsa et al. (2016) for more information.

$$E[V_{project}(V_M|V_D|V_E|V_S)] = \{V_M|V_D\}, \{V_D|V_M\}, \{V_M|V_E\}, \{V_M|V_S\}, \{V_M, V_E|V_S\}$$

We assumed that the option is considered at time t_0 and that the option holders may exercise it at any time (American-style option) prior to the expiry date (T). Another assumption is that the value of projects conforms the discrete multiplicative process. If we let $V_0 = V_M = V_D = V_E = V_S$ at time t_0,

$$E[V_t] = max\left[\left(\frac{pV_{t+1,n}^u + (1-p)V_{t+1,n+1}^d}{(1+rf)}\right) - C_{pro}, 0\right] \tag{13}$$

where

$$p = \frac{1+rf-d}{u-d} \tag{14}$$

Let $E[V_t] = E[V_{project}]$ be the value of project discounted for risk-free rate (rf). If we let K_M be the costs for maintain option, K_D be the cost for deferral option, K_E be the costs for expand option, and K_S be the costs for shutdown option. Hence, the payoff at the-end-of-period (expiry date) can be calculated as follows:

$$Call - Payoff = max\left[\left(V_{T,N} - K_{M|D|E|S}\right), 0\right] \tag{15}$$

$$Put - Payoff = max\left[\left(K_{M|D|E|S} - V_{T,N}\right), 0\right] \tag{16}$$

Let I_C be initial capital required for investments. The ENPV of project is calculated by backward induction technique

$$ENPV = -I_C + \sum_{t=1}^{T} \frac{(p * V_{T,N}^u + (1-p) * V_{T,N}^d)}{(1+rf)} \tag{17}$$

where T is the maturity ($t = 1, 2, \ldots, t \in T$), and N is the end node ($n = 1, 2, \ldots, n \in N$).
We formulated conditional argument to support a decision-making process in term of optimal decision that maximizes the value of projects.

$$NPV = max\left[NPV_{Basecase}, ENPV_{options}\right]$$

5. IMPLEMENTATION OF PROPOSED METHOD

The hypothetical gold mines is considered to demonstrate the proposed methods. It is assumed the mines evaluate the economic potentials at the feasibility study stage, where management have to make decisions between ‘commit’ or ‘not to commit’ the investment. If we assumed, as before, management also considers the flexibility to revise their decisions based upon ROV analysis. In this case, the mineral production development area covers 1100 Km^2. The total minable reserve is 45Mt with the cutoff grade of 0.85g/t. It is a rare case, but existed in some mines where wars were taking place, this is a unique assumption that there is unexploded ordnance (UXO) underneath the mining areas. Therefore, the costs for the UXO clearance are incurred, and will be included in the operating cost estimations. The unit cost for the UXO clearance is assumed to be about 0.05$/m^2$. The materials mined were classified by the cutoff grade that differentiates between ore and waste. Materials with grade above cutoff are sent to the mills. Materials with grade lower cutoff are sent to waste dumps. The royalty rate is assumed 3.5%. Tax income is assumed 35% and the risk-adjusted rate is assumed 10%. The gold price of 1080$/Oz is assumed for the base case economic evaluations. In this case study, the mining firms can delay the start-up investment for up to three years.

6. RESULT AND DISCUSSION

6.1 Analysis of Economic Value without Options

The illustrative example is kept simple for transparency of exposition; however, it contains sufficient information for tractability. The base case mine design was set up and the conventional DCF method was calculated for economic values of the projects without options. Table 1 depicts the forecasted cash flows over the life-of-project. The operating cash flows are calculated with positive in values. Discounting it by 10% and then subtracted the capital expenditures (CAPEX) of 595M$, the expected NPV yields 53.16M$ (see Table 1).

Table 1 Economic evaluations of the base case mine design using standard DCF methods

Description	Year	2016	2017	2018	2019	2020	2021	2022	2023	2024
Tonnes of material mined (Mtpa)		5	5.5	6	7.2	7.8	5.4	2.9	2.4	1.1
Gold price ($/Oz)		1080	1080	1080	1080	1080	1080	1080	1080	1081
Operating costs for mining and processing (M$)		123.08	132.60	142.05	164.51	175.64	130.70	82.12	72.05	27.85
Gold procuced (Mt)		0.361	0.397	0.412	0.419	0.433	0.433	0.209	0.173	0.125
Revenue (M$)		389.88	428.76	444.96	452.52	467.64	467.64	225.72	186.84	135.13
EBITD (M$)		376.23	413.75	429.39	436.68	451.27	451.27	217.82	180.30	130.40
Taxable income @35% (M$)		131.68	144.81	150.29	152.84	157.95	157.95	76.24	63.11	45.64
Depreciation (M$)		66.11	66.11	66.11	66.11	66.11	66.11	66.11	66.11	66.11
Net cash flow after tax and depreciation (M$)		55.36	70.23	70.94	53.22	51.58	96.52	-6.65	-20.96	-9.21
Operating free cash flow (M$)		121.48	136.34	137.05	119.33	117.69	162.63	59.46	45.15	56.91
Discounted cash flow @10% (M$)		110.43	112.68	102.97	81.50	73.07	91.80	30.51	21.06	24.13
CAPEX (M$)	595									
ENPV (M$)	**53.16**									

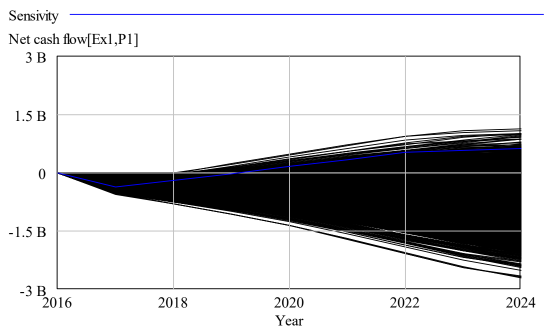

Figure 3 Monte Carlo sensitivity for net cash flow

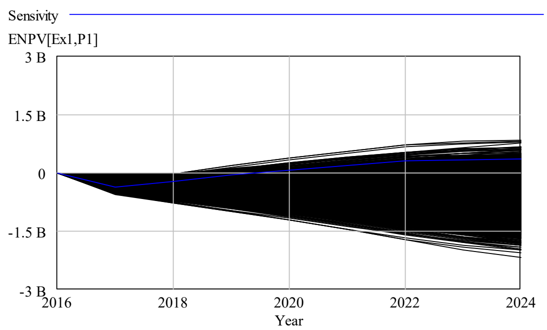

Figure 4 Monte Carlo sensitivity of ENPV

If management made a decision based on the NPV rules (NPV > 0), they would be inclined to 'commit' the investment. In this case, if the project is accepted, it is brought into the development of mineral productions. If management solely relied on the NPV rules without considering alternative options, it may impose high level of risks to the projects since the cash flows was calculated under the deterministic settings. There is evident that the projects calculated with positive NPV based on DCF leads to economic failure (e.g,Ferguson et al. 2007, Moel and Tufano, 2002). Although, the factors associated with mine failure are technically dependent from case to case for instance, inaccurate ore reserves estimation, operating cost overruns, price significantly falls, and so forth, and it takes several years to become evident.

We performed a Monte Carlo sensitivity analysis to investigate the probability of values distribution for the base-case mine design. This process involves changing the value of constant parameters (influential variables) such as gold price, discount rate, and so on. In this analysis, gold price was increased from 985$/Oz to 1500$/Oz, discount rate was set at 5% to 15%, and the simulation runtime was set at 10,000 iterations. Figure 3 shows the sensitivity results of net cash flow, where each line represents the traces of possible values over time. Figure 4 shows a similar graph of Monte Carlo sensitivity analysis for ENPV. Based on sensitivity results, it can be observed that the terminal values of the projects may be increased to about 367M$, and it may be decreased to -1.5B$. As matter of fact, the value of projects can be positive and/or negative; this implies that project may have higher and/or lower values across different time intervals. Hence, in this case, we assumed that mine analysts performed RODF to evaluate the value of projects with options. In this example, it is assumed that the management executed the deferral option upon the deferment costs of 107.66M$ over 3 years of the halting periods. The following sections provide the analysis of the project with options.

6.2 Analysis of Economic Value with Options

In this section, the ROV is performed to evaluate the economic value of the deferral option. Gold price volatility is needed as one of the input parameters to model the price movement using binomial tree technique. The annual volatility was estimated from the historical gold price data from year 1978 to year 2015. Fig. 5 exhibits that the highest volatility of 40.33% was measured for year 1980; whereas the lowest volatility of 12.58% was measured for year 2010. As can be seen in Fig. 5 that between year 2007 and mid-year 2013, gold prices were highly fluctuated. The prices fluctuating were significantly aligned with viability of annual volatility during those periods. Although the volatility was gradually decreased between year 2000 to year 2005, however, it managed to peak up to the highest value of about 33.30% in year 2008. It is perceived that the extreme viable of prices volatility occurred as a direct consequence of the global financial crisis and commensurate global recession during this period. In this paper, the annual volatility of 20.62% is used to simulate price evolution using binomial tree methods. We calculated the volatility of metal price and of the operating cost using logarithmic return method. The calculation of price volatility was carried out in spreadsheet environments. We also used spreadsheet to calculate input parameters that are necessary for building a binomial tree. Table 2 tabulates the calculated input data for the construction of binomial trees of metal price and operating cost. In ROV analysis, we first constructed the binomial tree for gold price and for operating costs. Next, we built the binomial tree for revenue. After that, we generated the binomial tree for project cash flows. Finally, we calculated the payoff for deferral option.

Table 2 Input data of metal price and operating cost for building the binomial tree

Input data	Volatility (%)	Up moves (u)	Down moves (d)	Risk-free rate (%)	Probability (%)
Metal price	20.62	1.13	0.88	8.00	78.44
Operating cost	22.20	1.14	0.87		75.94
Cash flow	39.38	1.27	0.79	8.00	60.00

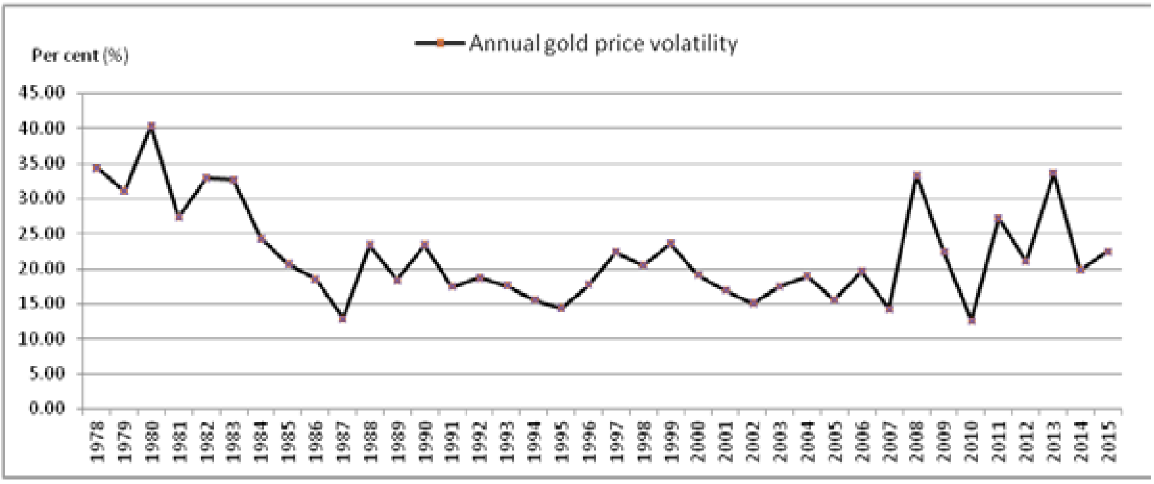

Figure 5 Estimated annual volatility of gold prices

Figure 7(i) shows the up and down movements of gold price in a binomial tree (recombining binomial tree). The evolution of price uncertainty starting with the base case price of 1080 \$/Oz and changes over time. We can observe that neutral states $(S_0 * u^1 d^1, S_0 * u^2 d^2, S_0 * u^3 d^3, and\ S_0 * u^4 d^4)$ separates between the up states and the down states. It is noticed that price will return to its initial value on the even columns. Where the up states are exponentially increased $(S_0 * u^i)$as i increased, while the down states is inversely decreased $S_0 * u^j)$ as j increased (see Figure 2). Similarly, in Figure 7(ii) cost uncertainty was simulated under binomial scheme. Project cash flow can be positive and negative depending on cash inflows and cash outflows of a particular period. Binomial tree of cash flow (see Figure 7(iii)) is built upon the calculation of revenues subtracted variable costs and associated fixed cost , for example, $CF_{2017,n1} = \big((1225 * 0.361) * (1 - 0.30) - 141 + 66.11\big) = 188.57\text{M\$}$, and $CF_{2017,n2} = \big((952 * 0.361) * (1 - 0.30) - 107 + 66.11\big) = 153.01M\$$. We used the same procedure to calculate the cash flows for the following years. After that, The payoff of options is calculated based on $max(V_{T,N} - K_D, 0)$, as shown in Figure 7(v), for example, for year 2017 at node 1, $max(188.57 - 107.66, 0) = \max(80.90, 0) = 80.90$. If values of option are kept opened, then we implement a backward induction to constructed binomial tree for the present values of projects. Present value is calculated by discounting a pair of cash flow in that year by the risk-free rate. Figure 7 (iv) shows binomial tree for present value, for example, $CF_{2018,n1} = (137.09 * 0.6 + (1 - 0.60) * 89.30)/(1 + 0.08)) = 115.56M\$$, and $CF_{2018,n2} = (89.30 * 0.6 + (1 - 0.60) * 51.61)/(1 + 0.08)) = 72.87M\$$. Figure 7(vi) displays the binomial tree for deferral option. We assumed that the payoff values were kept opened, and then we implemented a backward induction from year 2022 back to year 2016. Hence, the terminal value of calculated backward is the expected DNPV.

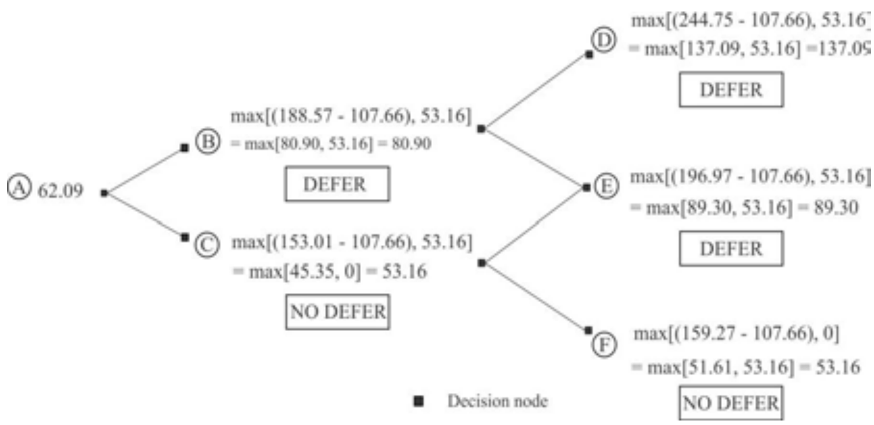

Figure 6. Binomial decision tree for deferral option

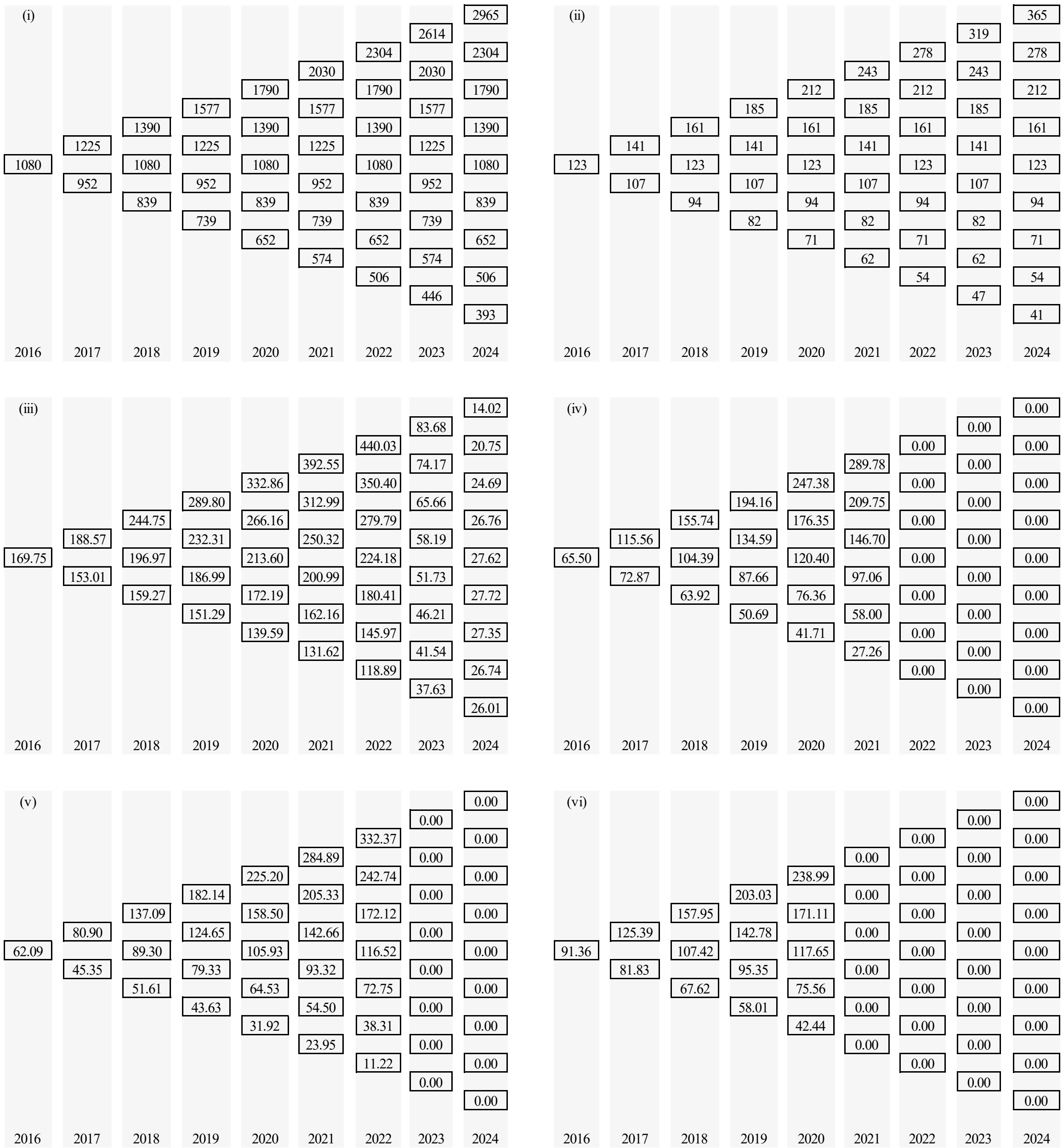

Figure 7 (i) Binomial tree for gold price, (ii) Binomial tree for costs, (iii) Binomial tree for cash flows, (iv) Binomial tree for present value, (v) Binomial tree for option payoff, and (vi) Binomial tree for ENPV

Figure 6 exhibits the binomial decision tree for deferral option before expiry date. Decision is made upon the comparison between payoffs and the base-case NPV, which is $\max[Payoff, NPV]$. If the payoff is greater than NPV, the decision is to defer, and it is not to defer, otherwise. Figure 8 depicts the SD model of deferral option. We used system dynamics software, namely Vensim®DSS6b, to develop models and to simulate the value of deferral options. It is important to note that the binomial option pricing method was implemented in the SD environments where the multiplicative process of binomial was not completely replicated. However, as can be seen in Figure 8, we applied the up-value, the middle-value, and the down-value of cash flows, to perform a binomial option pricing using SD approach. We utilized the SD model to back the decision if the managements were to choose between 'invest-now' or 'invest-late'. Monte Carlo simulation was simulated for 10,000 iterations, using random uniform in which each value has an equal probability of occurrences. Figure 8 illustrates Monte Carlo simulation analyses with high probabilities that payoff value may yield about 100M$ if deferral option is exercised. However, this value can only be obtained if management executed it. Figure 9 shows results of sensitivity analysis for deferral NPV (hereby called DNPV), which encapsulates the ranges of possible outcomes for DNPV. It shows that value may yield of approximately up to 1.73B$.

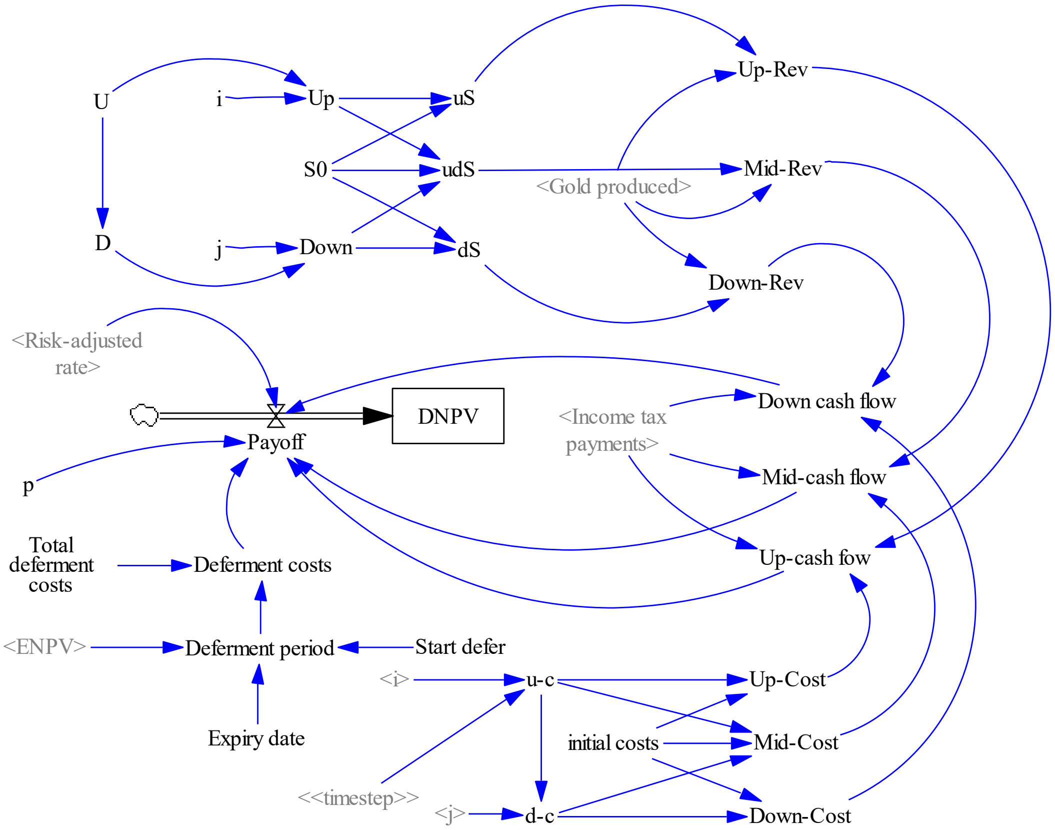

Figure 8 SD model of defer option

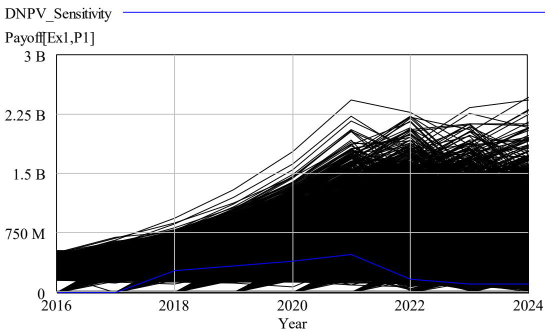

Figure 9 Monte Carlo sensitivity for payoff

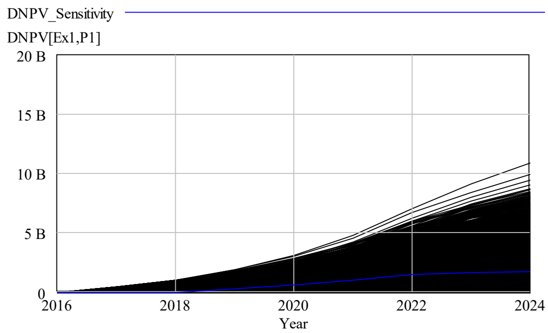

Figure 10 Monte Carlo sensitivity for DNPV

We can interpret results depicted in Figure 8 and Figure 9 that, there is no negative value appeared on the graph because the payoff was calculated by using conditional arguments given in Eq.(16). Based on real options rules the value of option is always great than zero $[(V_t > 0), 0]$, it is zero otherwise. By definition of option pricing theory, if the strategic option to be executed does not add value to the project, it better not to execute it. In the case study, we evaluate the economic values of projects by accounting for uncertainty in price movements and variation in operating costs. We assumed that it takes three years for halting. Management will have to make a decision before option is expired worthlessly. Decision is made upon the selection between base-case NPV or DNPV, $max[\$53.16M, \$91.36M] = \$91.36$M. Clearly, if management exercised a deferral option, the value might yield 91.36M$. The value of a deferral option is the different between the base-case NPV and the DNPV that is $91.36M\$ - 53.16M\$$ = 44.20M$.

7. CONCLUDING REMARK

This paper has outlined the integration of real option approach, system dynamics modeling method, and binomial option pricing, for the economic evaluation of a mining project. We demonstrated the evaluation procedure through a hypothetical gold mine. The important insight is that management might be inclined to commit the investment if the base-case NPV of is greater than zero. Yet, the cash flow calculations were carried out under deterministic settings such as static price was applied over the life-of-project. In this case, the decision made solely upon the NPV criteria may lead to undervalued of project because it does not reflect the effects of changes in the input parameters. ROV allows the better decision making under stochastic settings. It allows managements to hold the right of making or not making used of it, for example, to exercise or not to exercise a deferral option for a case study. However, such an option incurred costs, if management executed it and it payoffs in option premium. The hypothetical case study can be seen as an illustrative example for evaluating mining project when option is available. It reflected that such an option is connected with values and its probability of occurrences. The binomial tree method was used to simulate the uncertainty in the model inputs. The SD method was carried out to model and to solve options in a dynamic environment. The both methods enhanced the ROV in the ways that it visualized the calculation processes. In conclusion, we argue against the use of the stand-alone method such as DCF method for evaluating values of mining projects. Instead, the comparative approaches such as ROV and the auxiliary methods should be employed to value mining projects.

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OPEN PIT MINE WITH THE USE OF CONTINUOS SURFACE MINERS

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OPEN PIT MINE WITH THE USE OF CONTINUOS SURFACE MINERS

ABSTRACT

Because of corporative and economic need to do maintenance and to reduce mining costs over the useful life of mining enterprise, especially an open pit mining, arising from difficulties inherent in the nature of the business itself, the mining industry, sensitive to these issues, has fueled the development of new technologies and mineral production processes with great emphasis on open pit mining. As an example of these news unconventionals technologies and techniques, surface mining are presented as promising specializations of traditional models and mine mining systems, in particular to the already established Strip Mining Method. This article presents briefly this new technology and its application, as well as its major technical, economic, environmental and safety advantages applied to open pit mining.

KEYWORDS

Open pit mine, Surface Miners.

INTRODUCTION

Because of the difficulties inherent in the nature of the mining business, such as more difficult to access and mining best quality ores, increase ore transport distances to the ore treatment plant, increase between barren/ore relations, which in turn has a direct impact on drilling, blasting rocks, loading and transport of materials and their operating costs, among others, the mining industry, sensitive to these issues, to mitigate and/or eliminate such impacting factors to competitiveness and sustainability of its business, has fueled the development of improvements to the implementation of new technologies and mineral production processes with great emphasis on open pit mining. Thus, a new perspective in the light of mining engineering science also deserves attention and it is necessary. Adjustments and innovative character of adjustments in traditional methods already established for the open pit mining, in order to incorporate these welcome technological innovations in the industry, amount to more prominent position.

Technologies that prioritize cleaner, continuous, automated, economical and environmentally responsible systems are incoming to suggest a re-examination of traditional and established models and mining systems. Given these needs, surface miners are presented as promising specializations of traditional models and mining open pit mining systems, in particular to the already established Strip Mining Method.

MATERIALS AND METHODS

The development of surface mining technology began from the year 1970. At that time, the mining industry has already demanded for technological innovations in order to make them more economical and sustainable. Such development, undertook great efforts these, then visionary manufacturers, to meet the current technological stage of maturity.

In table 1, it can be seen various surface mining manufacturers by type miners present in the world market. Given that about 80% of the world surface mining market are concentrated in two major manufacturers, the German Group Wirtgen and the American Group Vermeer, and that 100% of surface miners in Brazil belong to these, will be presented during the this article, its main technological differences, as well as its main technical and economic advantages compared to application to the traditional operating methods.

Table 1: Comparison of surface miners by types

Parameters	Type of Suface Miner		
	Central drum	Front drum	Bucket Wheel
Width Cut (mm)	0-4200	0-5250	0-7100
Depth of Cut (mm)	0-800	0-800	0-800
Capacity (t)	100-3000	100-4000	100-5000
Operating Weight (t)	40-190	40-190	200-540
Motor Power (Kw)	450-1200	450-1200	450-1500
Manufacturers	Wirtgen, Bitelli, L&T e Huron	Vermeer, Tesmec, Voest Alpine	Krupp Födertechnik, Tenova TAKRAF

Source: Adapted from GHOSH (2010)

Considering only the two main manufacturers, each with their similarities and differences concerning the technological application of concepts, currently about 500 units are in operation in several mines around the world, having Asia accounted for most of these applications. Then, in the Figure 1, 2,3 and 4 is presented two models of surface miner and the distribution of surface mining units in the world for both manufacturers and equipments examples..

Distribution of Surface Mining in the World - Wirtgen

Figure 1 - 401 Surface Mining in the World. Source: Wirtgen 2012

Figure 2 – Surface Miner SM2500 – Wirtgen. Mine of Bauxite - Pará - Brazil. Source: Wirtgen 2014

Distribution of Surface Mining in the World - Vermeer

Figure 3 - 61 Surface Mining in the World Source:Vermeer 2014

Figure 4 - Surface Miner T1255 - Vermeer Mine of Limestone - China. Source: Vermeer 2014

Unlike traditional mining equipment, geological and mineralogical aspects of the mine, the mine production scales to meet specifications of mineral processing plant, mine development with handling of sterile, entailing thereby a new "lay out" of mining plan and mining sequency, become re-examined as a function of the specific features and capabilities of the equipment. These aspects became, and still are major challenges encountered in the development and consolidation of this new technology in the world market.

In the case of Miners Wirtgen, asphalt milling of wide application in the construction industry were the basis for its development. The operational concept application in mining which is carried out cutting and fragmentation of ore and/or sterile "in situ" with direct loading of these fragmented materials in trucks took from the example of application in the construction industry. In the case of Vermeer Surface Miners, the design of the technology has primarily focused on the ore fragmentation and/or sterile "in situ", for loading operation with other equipment like bulldozers and loaders into trucks.

Application in mine open pit mining

The surface miners are used primarily for bedridden deposits in horizontal plane or sub horizontal plane, in table format and with great longitudinal extension, such as lime deposits, coal, iron ore, gypsum, salt, bauxite, among others. For the distribution of mining mineral substance commercialized by Wirtgen, given the similarity of applying the technology to both manufacturers, it can be inferred by the similarity in the application of mineral substance case of miners provided by Vermeer. Figure 5 shows the distribution of the miners made by Wirtgen to December 2014 by mineral substance.

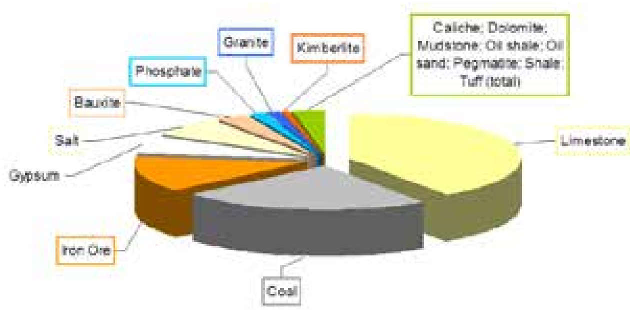


Figure 5 – Distribution mineral substance of the miners sold by Wirtgen. Source: Wirtgen (2014)

Operating concept of Surface Mining

Cutting drum in the center

Continuous cutting equipment that load trucks concurrently with the court. In good operating condition, its output can reach up to 2,500 t/h. The rock is excavated by the cutting drum in the center of the equipment. The strength equipment weight is added to the regular forces and drag forces on the cutting drum facilitating the fragmentation process. Wirtgen surface miners are representatives of this concept illustrated in the Figures 6 and 7 .

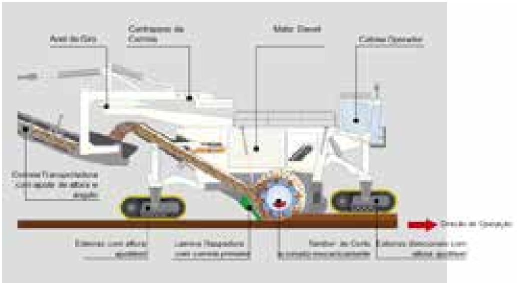


Figure 6 – Surface miner Wirtgen. Source: Wirtgen 2014

Figure 7 – Cut and load system of surface miner Wirtgen Source: Wirtgen 2014

Drum front and/or back cut

Continuous cutting equipment that do not load trucks concurrently with the court. It is worth noting that the load option is available for miners type bucket wheel from other manufacturers, which as stated, are not object of study of this article. In good operating condition, production of Vermeer miners can reach up to 4,000 t/h. The rock is excavated by the cutting drum located on the front or back of the machine using combination of forces with direction and sense of machine's displacement, facilitating in this way, the process of fragmentation. (See Figures 8 and 9)

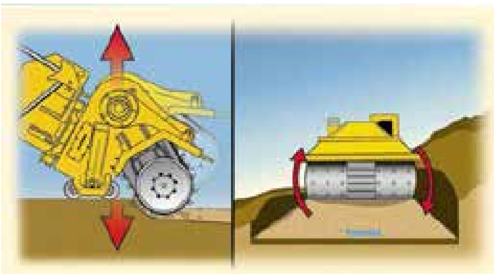


Figure 8 - Surface miner Vermeer Source: Vermeer 2014

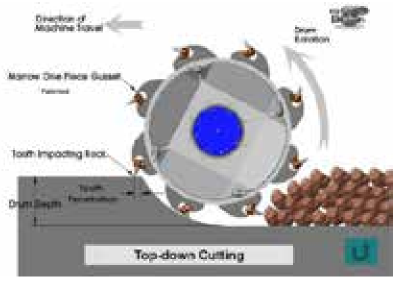


Figure 9 - Cut system of surface miner - Vermeer Source: Vermeer 2014

Advantages of surface miners in the open pit mine

Surface miner Wirtgen **Surface Miner Vermeer**

Figure 10: Operations eliminated – Wirtgen Miners Source: Adapted from Wirtgen (2010)

Figure 11: Operations eliminated – Vermeer Miners. Source: Adapted from Wirtgen (2010)

As represented in Figures 10 and 11, the surface miners of both manufacturers perform the cutting operation and fragmentation of rock "in situ" in a single operation, thus eliminating the unitary mine operations like drilling, rock blasting and primary crushing of treatment plant.

Advantages of surface miners Wirtgen

Eliminates drilling, blasting rocks, loading trucks with other equipment and primary crushing operations. As main advantage of Wirtgen miners, among others mentioned, have the direct loading in mining trucks of the cut and fragmented material. This interface requires a proper sizing of trucks units necessary for good mining production performance. The good operating synchronism between trucks used, especially with the number of units used, and surface mining will result in good performance production for the set. With direct load operation in trucks, their economic gain is with the elimination of the operation itself and their operating costs and investments with bulldozers and/or shovel loaders. It is worth noting, when used without the direct upload option, the productivity of the equipment increases considerably.

Advantages of surface miners Vermeer

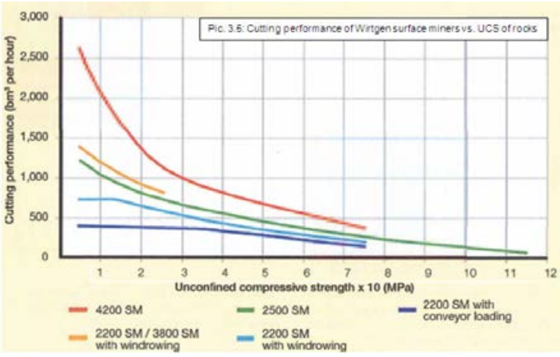
Eliminates drilling, blasting rocks, and primary crushing operations. The operational concept of not performing direct loading on trucks, exempts the interface of cutting production and fragmentation of the material with the conveyor belts. In this way, they realize high production scale, reflecting positively on its economic aspects. Even occasioning operating costs and investments in other loading equipment into trucks, their gain is mainly applying the concept of economies of scale. The operational flexibility of the machine with the cutting operations and fragmentation mining separated from loading trucks is a point to note.

In Table 2, following comparison of Wirtgen and Vermeer Surface Miners Models. It can be seen the difference between sizes of equipment given their different application concepts.

	SURFACE MINE MODELS - WIRTGEN			SURFACE MINE MODELS - VERMEER		
	2200SM	2500SM	4200SM	T1255TL - CD	T1255TL - DD	4200SM
Width Cut	2200 mm	2500 mm	4200 mm	3658 mm	3685 mm	4200 mm
Depth of Cut	0 - 300mm	0 - 600 mm	0 - 830 mm	0 - 686 mm	0 - 533 mm	0 - 711 mm
Motor Power	949 HP - CAT	1050 HP - CUMMINS	1601 HP - CUMMINS	600 HP - CAT	600 HP - CAT	1200 HP - 2 MOTORES CAT
Operating Weight	51 t	103 t	184 t	111 t	99 t	179 t

Table 2: Comparison of Wirtgen and Vermeer Surface Miners**.** Source: Wirtgen e Vermeer (2010)

It can be seen in the Figures 13 and 14 production performance curves for different types of Wirtgen and Vermeer surface miners. The curves show the maximum cutting performance of the miners due to the mechanical strength of the rocks.



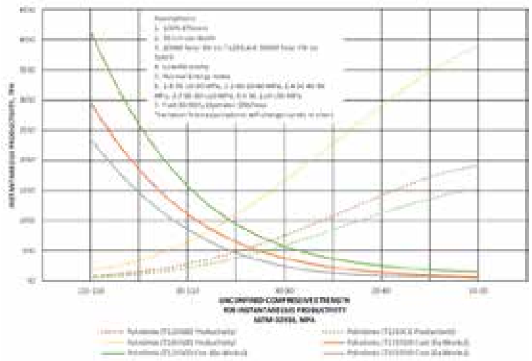


Figure 13: Wirtgen surface mine production curves   Figure 14: Vermeer surface mine production curves
Source: Wirtgen (2010   Source: Vermeer (2010)

It is noted in the graphs above, the rapid drop in productivity of miners from the small increases in mechanical strength of the rocks, with the main analysis parameter their Resistances of Uniaxial Compression Strength - UCS's. Wirtgen manufacturer states that its surface miners are able to cut ores with uniaxial compressive strength economically to a range from 0 to 100 MPa. Vermeer manufacturer claims that its surface miners are able to cut ores with uniaxial compressive strength economically to range a from 0 to 150 MPa. Note that the values are average, and can vary more or less depending on variables of other limiting application of the miners, such as their own mineralogical characteristics of rocks and mining operating conditions.

Boundary conditions for application of Surface Mining

Despite all the proven benefits of operations with surface mining in terms of quality mining and sustainability, its implementation must respect certain parameters that influence the its cutting capacity and mining performance as represented by the Table 3:

Table 3 : parameters that influence the surface miner cutting capacity

Characteristics of the rock mass	Machine Set up	Application type
Moisture content, density, Uniaxial Compressive Strength - UCS, Point load index, Young's modulus, Fracture energy, Toughness index, Brazilian tensile strength, Profile sonic speed, abrasiveness (Schimazek-F, Cerchar), Number of volumetric together viscosity, Specific power cut	Setting cutting tools (inclination angle, angle of attack, clearance angle, drill angle, arrangement of drills, type of drill, number of drills, drill material, width drum, power equipment, liquid nature cooling of the drills).	Operating mode (windrowing/ direct loading on trucks), width and length of the operational area, operator skill, specific conditions (dry/wet material, desired particle size and production)

Source: Adapted from DEY e GHOSE (2008)

The following conditions must also be evaluated regarding the aplication of surface mining:

Ground topography

The prior leveling preparation of the ground local is required. Unevenness above 0,4 meters need to be flattened with other resources. The surface miner is able to complete its operation leaving a smooth surface since the ground has been leveled by other equipment, or by itself.

Ground stability

The operational safety of surface miners is directly related to its application in stable land, because they are heavy machinery. In unstable ground, unconsolidated on underground openings, or karst terrains, there is a risk of land subsidence as a result of its operation.

Characteristics of the rock mass

It is imperative to understand the rock to be excavated before the selection of the appropriate machine. Determination of the excavation facility is one of the critical stages in the open mining projects. The evaluation of the application and selection of the mining area for an operation should be based on a careful assessment of the specific properties of the rock, since the stops in a mechanical blasting process can reduce the efficiency of the equipment, being under the technical and economic limits of viability, frustrating the whole purpose of its implementation.

Uniaxial Compression Strength Resistence

The hard rock is more difficult to dig. The uniaxial compressive strength is the most widely accepted parameter to determine the hardness of a rock sample and is generally considered the most useful guide for determining the performance of mechanical excavation.

The manufacturers of surface mining are based on this parameter and simulate performance curves based mainly on uniaxial compressive strength or the ratio between normal stress and shear the rock. Despite the various capabilities of each equipment, in general, the productivity of miners drastically drops with increasing resistance of the rock to be cut. The graphs 01 and 02 illustrates this situation.

RESULTS

With the implementation by mining companies of surface mining technology, start to incorporate the main advantages in the technical, operational, economic, environmental and safety areas. Here are the main advantages understood by the use of surface mining technology in open-pit mining.

Technical and operational advantages through the use of Surface Mining

- ✓ Reduction of the number of unit operations in the mine;
- ✓ Elimination of truck loading operations in ore and sterile mining fronts (for the case of Miners Wirtgen);
- ✓ Elimination of rock drilling operation;
- ✓ Elimination of rock blasting operation;
- ✓ Elimination of primary crushing, and sometimes to the secondary crushing;
- ✓ Realization of selective mining;
- ✓ Making regular and stable slopes;
- ✓ Making stable and regular access, eliminating cost tires and maintenance of trucks, and manufacturing and maintenance of drainages.
- ✓ Open possibility to mine blocked areas because of surrounded communities proximity.

Economic advantages through the use of Surface Mining

- ✓ Elimination of investment and operational costs to truck loading operations (for the case of Miners Wirtgen);
- ✓ Elimination of investments and operating costs related to the rock drilling operation;
- ✓ Elimination of investments in operating costs related to rock blasting operation;
- ✓ Elimination of investment and operating costs for primary crushing, including to the case of increasing of Run of Mine - ROM production.
- ✓ Selective mining realization;
- ✓ Reduction of operational costs of maintaining access and drainages;
- ✓ Reduced operating costs related to the preparation and maintenance of slopes.

Environmental advantages and security through the use of Surface Mining

- ✓ Easier and safer mining by reducing the number of unit operations in the mine;
- ✓ Eliminates impacts resulting from the use of explosives such as vibration, over - pressure, ultra - fragments releases, dust and noise pollution;
- ✓ Possibility to work near the surrounding communities of mine due to eliminate/mitigate impacts or nuisances related to drilling, blasting, loading and crushing near them. Realizing this way mining with minimum generation of noise and dust for communities.
- ✓ Making more regular and stable slopes that the traditional method;
- ✓ Making more stable and regular access to the traditional method;
- ✓ Its operating concept is more harmonized with the restrictive environmental current regulations when compared with traditional mining method using drilling and blasting rocks.

CONCLUSION

Considering the main technical, economic, environmental and safety advantages from surface mining application in the open pit mining, it is concluded that the technology is a great innovative and technological span specialization of traditional models and mining systems.

Its operating concept is more harmonized with the current restrictive environmental regulations, exonerating both companies and directly involved communities of discomforts due to drilling and blasting operations rocks and primary crushing, which enables a better relationship between them.

Given the breadth of diversity mines that perform mine mining with surface mining in the world, as well as the large amount of equipment in operation, it can also conclude about the great receptivity of technology by the mining market, consolidating its position as a system of mining cleaner, continuous, automated, secure and environmentally responsible.

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PAYLOADS ANALYSIS USING TELEMETRY

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PAYLOADS ANALYSIS USING TELEMETRY

ABSTRACT

The actual iron ore price forces the companies to reduce costs in all production steps of the mining operation. Loading and hauling materials using shovel-truck system represents the highest mining costs, particularly tires, diesel, maintenance and workforce. Annual goals of materials movement are split by months were weather conditions can affect productivity and performance of the fleet. The data file of the monthly material movement is based on the scale of each truck. Every fleet has a payload and it represents the capacity of hauling, being the value of loading a unique input extracted in each duty cycle, without taking into account the position of the material over the truck's dump. Shear forces and torsion are applied all time at the frames trucks and, when the truckloads are unbalanced, increases the forces over the frame and tires. Transport unbalanced truckloads into long distances attack the tires and decrease its end of life, causing damages like displacement of tread and flank separation. The objective of this study is to propose a methodology to decrease unbalanced truck loads reducing the mining costs. In this mission of incorporating a material position on the truck´s dump, the telemetry was taking into consideration. The analysis of the suspension pressures data values shows the truckloads positions and with this data was created a databank. This databank indicates that the operators need truck load training to reduce unbalanced loads and truck costs in the mining operation, since unbalanced loads leads to early tires and suspensions scrap. The case study was carried out at Itabira Iron Ore Mine Complex, Brazil. This methodology was able to reduce the number of unbalanced truck loads as well as the tires life was increased.

KEYWORDS

Load and haul; off road trucks; unbalanced truckloads; mining; costs, off road tires.

INTRODUCTION

The loading and hauling costs control is important particularly in a scenario of reduced iron ore price market. A common practice in large operation is hauling materials using shovel/trucks system, therefore, costs with tires, components, suspensions and frames maintenance cracks due repetitive efforts are very significant (Chamanara, 2013). Knowing and describing the operating parameters is extremely important to understand the main causes of scrapping or early withdrawal of tires in operation, among which are (Tire Maintenance Manual – Good Year/Off the road, 2008):

- Inflation pressure of tires and operation value;
- Temperature;
- Conditions of access - presence blocks, ripples, and sub- and over- elevation of roads;
- Transport speed;
- Average distance covered per cycle;
- Payload centering on the bucket trucks.

Tire consumption in an open pit mine is directly linked to the operating conditions in which they are submitted. Unbalanced truck loads and tires in operation with low pressure lead to displacement or separation of the inner layers of the tires by heating its internal structures. Shear forces and torsion are applied all time at the frames trucks and, when the truckloads are unbalanced, increases the forces over the frame and tires. Transport unbalanced truckloads into long distances attack the tires and decrease its end of life, causing damages like displacement of tread and flank separation (Michelin Guideline – Utilization and Maintenance – 2012).

An important factor that should be taken into consideration for trucks tires is the TKPH (*"tons kilometers per hour").* To calculate this factor is taken into account three variables: distance traveled, average speed and payload carried. For specification, TKPH admit that the materials transported during the cycles are evenly distributed over the truck structure. The correct load centralization on the bucket truck has its importance at this point in ensuring that each tire is carrying an approximate load of their maximum capacity, without damage the truck's productivity (payload). If large amounts of off-center loads (Figure 1) are brought into a set of tires in constant cycles, they are subject to more severe conditions, shortening the life thereof (Otraco, 2002).

Figure 1: Iron ore decentralized load image at Itabira mines, Brazil.

Analysis of the suspension pressures data values with telemetry can show the truck loads positions and with this data was created a databank. This databank was used for operators training with the goal to reduce unbalanced truck loads and reduce the truck costs in the mining operation, once unbalanced loads can bring discomfort to the truck drivers as well as scrap early tires and suspensions.

METHODOLOGY

Key Performance Indicators for Load Position

The key performance indicators for truck load positioning are rack, roll and pitch (Figure 2). Rack is the twisting force and it is common to find the term torsional motion for its description. Torsional forces applied constantly on trucks structures lead to cracking and potential repetitions may decrease equipment life and is a major cause of premature breakage of components and chassis (Chamanara, 2013).

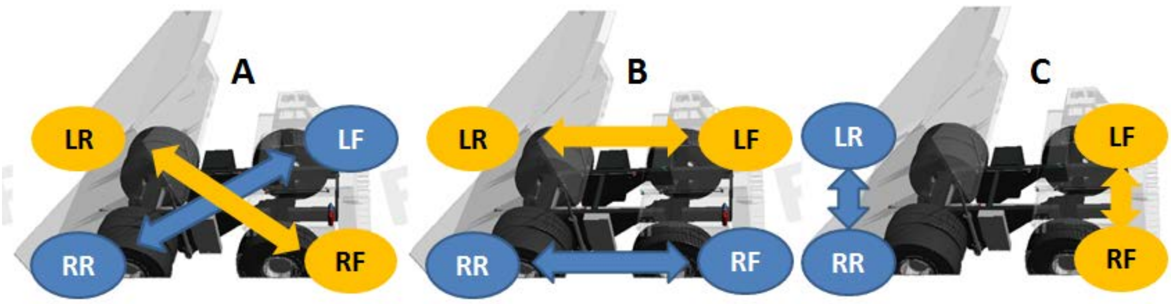


Figure 2: The interactions between the suspension pressure values. a) Rack, b) Roll and c) Pitch. LF – left front pressure suspension value, RF – right front, LR – left rear and RR – right rear.

The calculation of the rack value is given by:

RACK = (LF + RR) – (RF + LR) Figure 2 – A

Roll (bias) is understood as the difference between the sums of the suspensions on the right side by the sum of the suspensions on the left. This indicator reflects directly the positioning of loads due to the comparison between the sides. As it must seek centralized loads, the values of the key performance indicators (KPI's) in each transmission cycle is expected to be null or close to zero, reflecting little twisting force applied to the equipment during the hours of operation.

Roll calculation:

ROLL = (LF + LR) – (RF + RR) Figure 2 - B

Therefore, like the rack KPI it is also expected a value close to zero for roll, reflecting little lateral difference between the pressures of the suspensions. The third and final KPI, the pitch, is expected to find value equal to the difference between axes by subtracting the sum of pressures found on the front axle by the values of both rear suspensions, as shown in Figure 2. Generally, for each tire carrying or supporting the same load, the division of the total gross weight of operation should be 1/6 per tire. Thus, as the tire assembly has a ratio of one to two, comparing the front and rear axles, the rear axles supports 2/3 (67%) of the total load and the front axles the remaining 1/3 (33%). The calculation for pitch is:

PITCH = (LF + RF) – (LR + RR) Figure 2 - C

This difference between axes, when the payload is evenly distributed over the truck structure is 1/3 negative of the sum of the values found in all four suspensions, reflecting that the load is balanced across wheelbase (Chamanara, 2013). Chamanara explains how to compare the suspensions pressure data by collecting data via telemetry and graphically displays the values over time of a transport cycle (Figure 3).

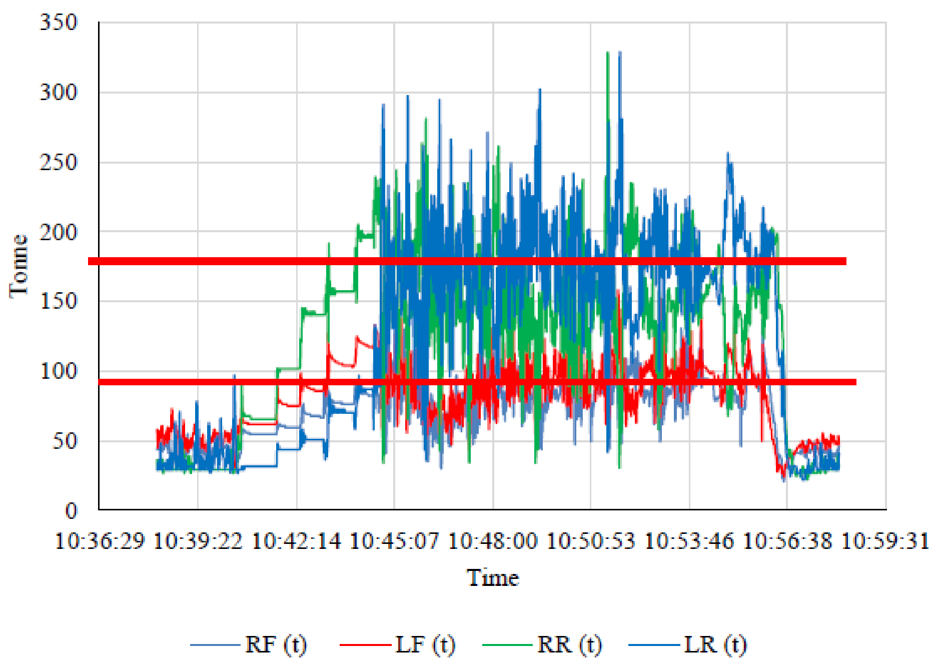


Figure 3: suspensions pressures (tons) registration of a CAT truck in operation (Chamanara, 2013).

The current practice of mining companies is exclusively to extract the truck payload to build the total movement at the end of each month, year or other period of interest. This work paper applies a methodology of registering the distribution of loads on the bucket trucks thus creating a database with these records. The database includes some information and describes the profile of mining, such as:

- Excavator / loader that did the load;
- Operating Bank - wide and maneuvering square of the conditions can be found;
- Excavated and transported material - sterile or ore;
- What was the machine operator;
- What distance transported;
- Cycle time;
- What time of day (for constant day and night operations) loads happen more often?
- Payloads transportation by truck and it´s recurrence.

The information above is used to create monthly reports to show the evolution of operators, common location of occurrences, common material, cycle time and frequency that each tire has depreciated with badly centered loads. May thus infer how harmful the transport of these loads on tire life is and create more precise management training for equipment instruction team.

Telemetry model for load suspensions pressures

Choose snippets are necessary to creation of the system Minecare trend using beacon's own Modular Mining. The so-called beacons (Figure 4) are points of reference in the graphic area of the order containing the electronic map of the mine.

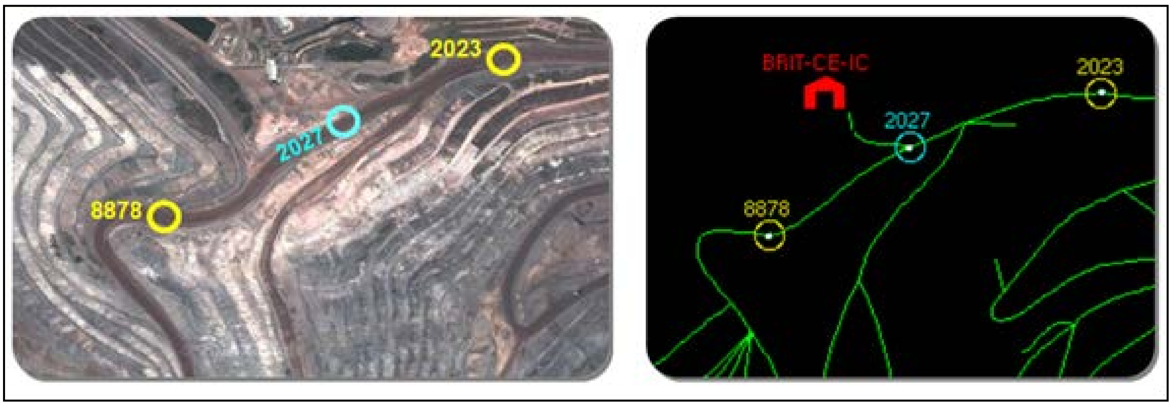
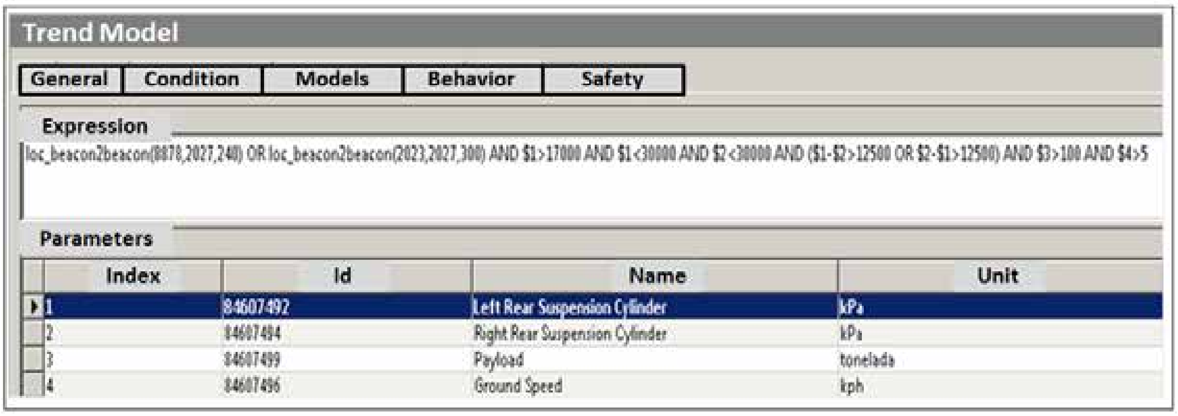


Figure 4: Mine road chosen for reading suspensions pressures and representation of the dispatch system - beacons recognition and creation of telemetry rule.

These points are used by the system to locate the equipment and optimize it, providing the next load of the truck. After defined the passages with their beacons in the dispatch system, move to field evaluation and follow-generated graphics with the respective pressures suspensions. For this, create the rules in Minecare system. See Figure 5 for an example of the rule created for the excerpts:



Index	Id	Name	Unit
1	84607492	Left Rear Suspension Cylinder	kPa
2	84607484	Right Rear Suspension Cylinder	kPa
3	84607499	Payload	tonelada
4	84607496	Ground Speed	kph

Figure 5: Trend rule model created in Minecare system - collecting pressure values of the suspensions.

Comparisons were made between the positions of the masses on the truck's bucket to calibrate the telemetry and refined classification rules of ROM loads to supply the crushing plants. The pressures of the suspensions are collected and analyzed by the trend, or telemetry rule when travelling between the reading beacons.

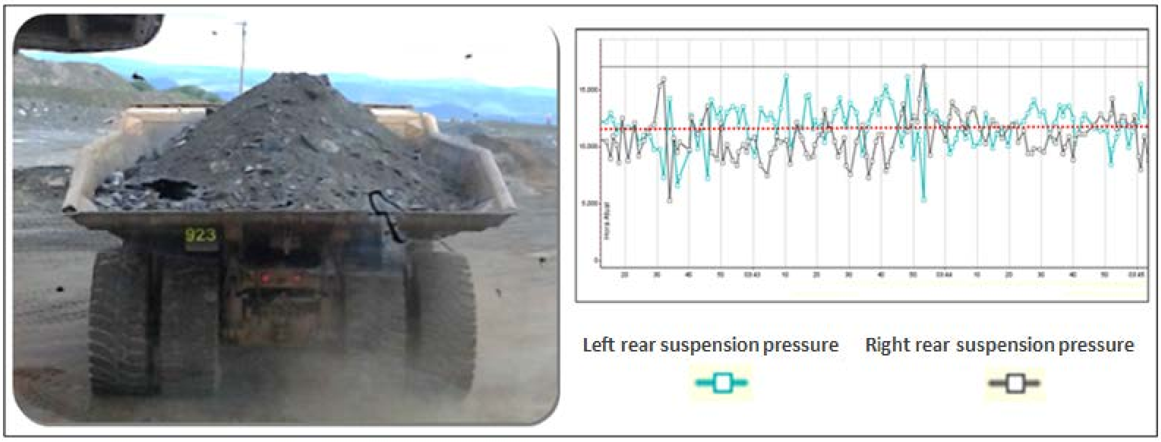


Figure 6: Centralized ROM payload and pressures of the rear suspensions.

Figure 6 shows how the pressures in the rear suspension vary during operation around the shaft. The variations are caused due to depressions in the mine roads. Figure 7 shows how the decentralized payload interferes with the right pressures experienced by the suspension during the truck route. This difference can be understood as a transfer not only to the suspensions, but also to tire components and frame (chassis) of the equipment.

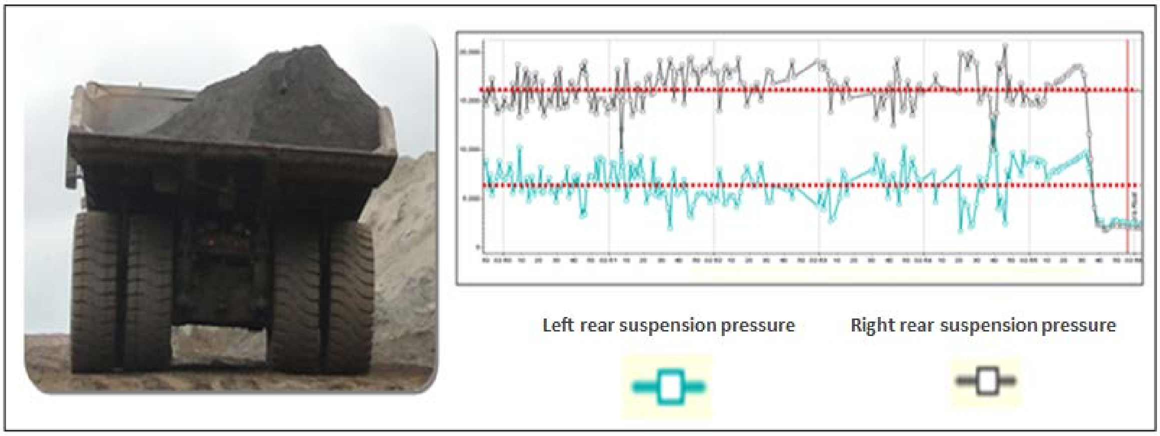


Figure 7: Decentralized load to the right.

Likewise for load decentralized to the right, the answers of the suspensions are the same of it shifted to the left, Figure 8. In both cases the effects on the equipment and the operator are the same, varying only the path on sharp curves.

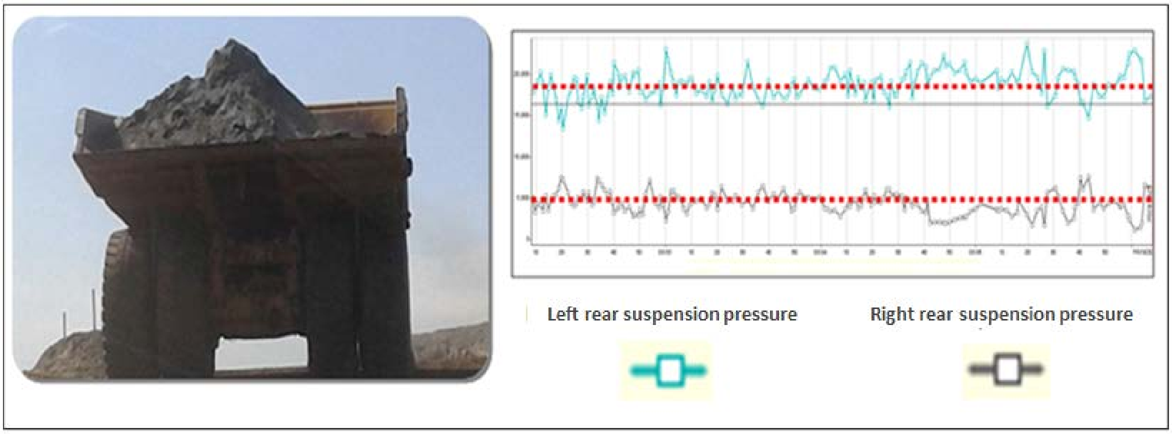


Figure 8: Decentralized load the left.

Table 1 is an example of the database created by collecting values of the suspensions. In the field "event" (5th column) the classification of load positioning is validated through the interaction of KPIs previously seen. Other information can be used for treatment of deviations, such as training needs of loader operators as well as local excavators with more decentralized loads.

Table 1: Sample database created through the records

Start /	End /	Shift /	Team /	Event /	Machine /	Machine Operator /	Truck /	Truck Operator	/Payload
3/2/2015 6:42	3/2/2015 7:26	03-FEV-15-ITA-Manhã	Turma C	OP - Carga Descentralizada a ESQ T3	PE3788	Nome do Operador de Carga	CA65922	Nome do Operador de Transporte	244
12/2/2015 22:28	12/2/2015 23:11	12-FEV-15-ITA-Noite	Turma B	OP - Carga Descentralizada a DIR T2	EC3258	Nome do Operador de Carga	CA65787	Nome do Operador de Transporte	245
12/2/2015 23:48	13/2/2015 0:41	13-FEV-15-ITA-Madrugada	Turma E	OP - Carga Descentralizada a ESQ T3	EM2466	Nome do Operador de Carga	CA65930	Nome do Operador de Transporte	238
13/2/2015 0:31	13/2/2015 1:01	13-FEV-15-ITA-Madrugada	Turma E	OP - Carga Descentralizada a DIR T2	EC3257	Nome do Operador de Carga	CA65787	Nome do Operador de Transporte	259
13/2/2015 10:57	13/2/2015 11:37	13-FEV-15-ITA-Manhã	Turma C	OP - Carga Descentralizada a ESQ T1	PE3781	Nome do Operador de Carga	CA65771	Nome do Operador de Transporte	241
13/2/2015 11:05	13/2/2015 11:56	13-FEV-15-ITA-Manhã	Turma C	OP - Carga Descentralizada a ESQ T1	PE3781	Nome do Operador de Carga	CA65919	Nome do Operador de Transporte	242

RESULTS

After the necessary adjustments in Minecare system for collection and storage of data, it began the analysis of the evolution of alarm records generated on the operation. The total material data in the three plants of Itabira complex between June to October 2015 is shown on table 2.

Table 2: Total mass delivered to the crushing plants Itabira Complex and percentage of decentralized payloads. Source: Dispatch System and Minecare

Month	ROM (t)	Cycles	Decentralized Loads	Percentage of Total ROM Loads
Jun	5.809.063	24.692	385	1,6%
Jul	6.007.377	25.614	320	1,2%
Aug	5.818.168	23.842	225	0,9%
Sep	5.217.746	20.738	190	0,9%
Oct	5.780.306	22.998	277	1,2%

The access to the processing plants in the Itabira complex were monitored by the rules created in the system for collecting and storing data from the pressures of suspensions of trucks from June to

October. All transportation equipment to move these sections were monitored and their loads classified.

In June was counted 385 decentralized loads of ROM transportation in Itabira. This represents 1.6% of total ore cargo for that month (24,692 loads delivered to the crushing plants). With the progress of the action plan and the daily monitoring team-by-team, good results were achieved, keeping the months of August and September at the percentage of 0.9 % of the total charges. Figure 10 show the evolution of the decentralized loads from June to October, 2015.

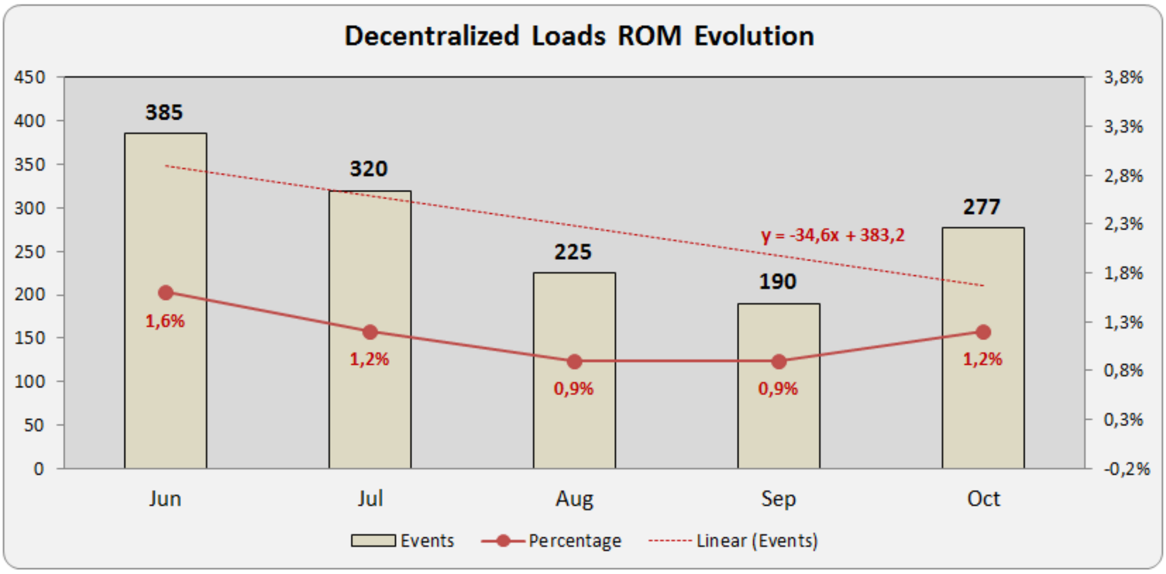


Figure 9: Evolution of occurrences of off-center loads.

In October there was interference on the mining with a hydraulic excavator, shovel type, PC5500 Komatsu, see figure 10. Narrow areas difficult maneuvering of trucks and damage mining productivity making it more complicated to center the loads. Confined spaces sometimes force shovel operators to load it from the back, which raises operational risks. This situation must be avoided, but in this particular case, it was necessary to carry out the mining to supply the hematite to the crushing plants, one of the few offering fronts of such material available that month.

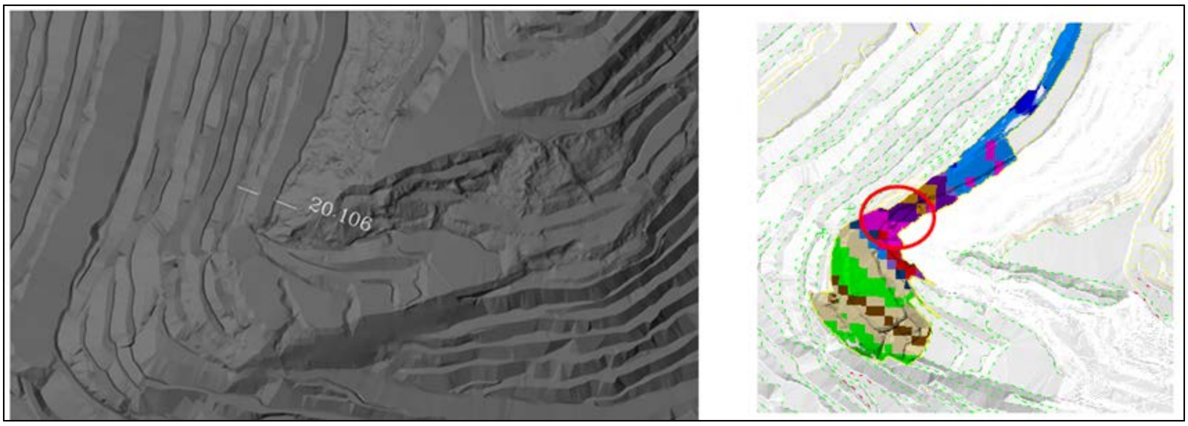


Figure 10: EM2970 the mining plan showing point of narrowing. Source: Mining Plan October 2015, short-term planning Itabira, Vale

The result was reflected in a higher generation of decentralized loads, being represented on figure 11.

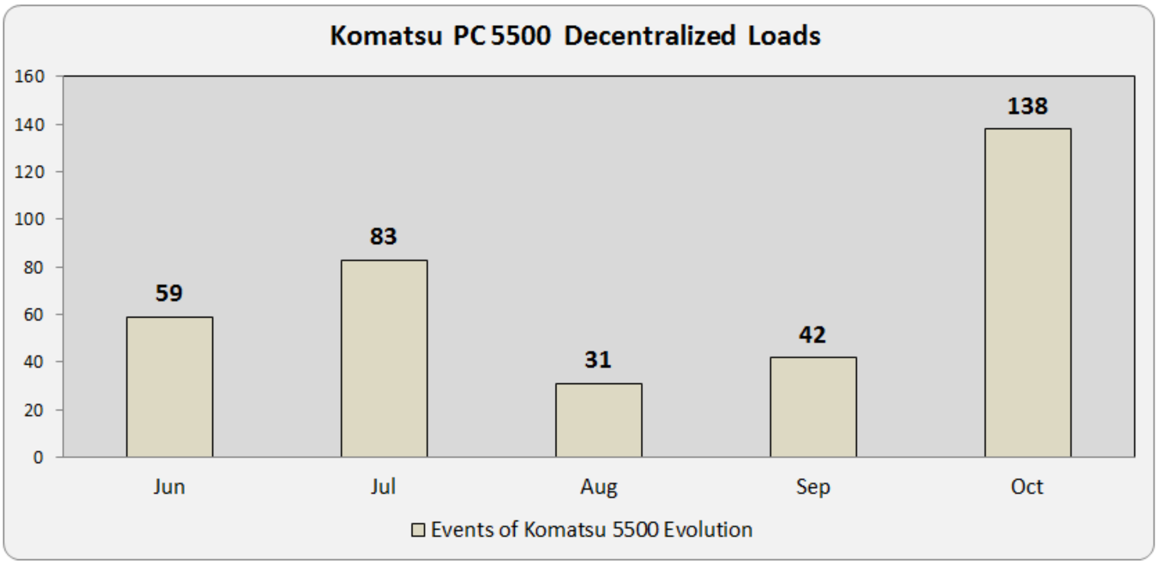


Figure 11: Evolution of decentralized load operation with the EM2970. Source: Dispatch System and Minecare - Mine Operation Itabira.

As it is shown on Figure 11, the EM2970 shovel went through points of nips during mining in Itabira Conceição mine in October and had a negative influence on the increase in instances of bad centralized loads. A total of 138 charges representing almost 50 % of the total (277 loads).

CONCLUSION

Decentralized payloads act negatively on longevity of OTR tires and must be avoided in mining operations. The directly affected operating parameters are pressure and temperature of these components. Therefore, tires in operation with these values outside specifications suffer irreversible damage, causing decrease in lifetime and increasing operating costs.

This study shows that it is possible create a database by collecting telemetry parameters, therefore increase asset management and understand the possible causes of the occurrences. The records of the occurrences contain valuable information for the operation as local of load, operators who require recycling, time of the events, equipment involved and material transported. All these data can be used for better management of training and control costs in the load and haul phases.

As June was the reference month, the initial of the effective control, a percentage of 1.6% of ROM loads were classified as decentralized. Obtained the lowest levels in August and September, 0.9 % (a decrease of 43% compared to the month of June). In October with the interference (narrowing the maneuvering area) in the mining of a hydraulic shovel excavator and consequently a rise in registrations to 1.2% the total loads.

The continuity of the actions and commitment of everyone involved in the process are of fundamental importance to maintain the levels below 1 % of the total ROM loads in Itabira Complex. That will ensure the gain calculation in hours worked of tires and the savings generated by the reduction of non-center loads.

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PRACTICAL HAUL ROAD DESIGN METHODOLOGY APPLICATION BASED ON SITE CHARACTERIZATION

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PRACTICAL HAUL ROAD DESIGN METHODOLOGY APPLICATION BASED ON SITE CHARACTERIZATION

ABSTRACT

Most of the surface mines use a truck system for hauling material, which can represent about 40% of their operational costs (OPEX). Poor road design can lead to many problems and additional costs, such as an increase in fuel and tyre consumption, ergonomic problems for operators (back problems), increased work accident risk and productivity losses. Haul roads should enable trucks to drive at their maximum safe speed, and it is estimated that a 1% increase in the rolling resistance would reduce the truck speed by 22% on high speed surface roads and 10% on pit ramps. Most of the mine roads are designed by empirical methods or just constructed by dumping and spreading material. It has been shown that such methods have the potential for increased costs as a result of the poor design. Mechanistic design, on the other hand, allows mine planners to determine the layer thickness based on limiting strain values according to local in-situ materials.

The objective of the paper is to present a low-cost approach, to design a haul road based on limiting strain values for layer thickness determination using the Dynamic Cone Penetrometer (DCP) test assessment. As a case study, a formal haul road design procedure is applied in a typical Brazilian quarry. The design process started with the construction of a special test device, the DCP. It is used for on-site material strength assessment. DCP results are correlated with California Bearing Ratio (CBR) for on-site material characterization, where poor soils should be removed and replaced or just covered with a sufficient layer thickness of selected blasted rock. Wearing course material selection is based on laboratory tests such as Atterberg Limits and particle size distribution. This process provides a vehicle friendly road that reduces operational costs such as truck maintenance costs, fuel and tyre consumption, cycle times and especially for production performance improvement through the allowance of driving safely at higher speeds. It was found that the formal design process had major economic and operational benefits.

KEYWORDS

Haul road design, Dynamic Cone Penetrometer, in-situ characterization, Wearing course selection, Structural design

INTRODUCTION

The increase in mine production led to the utilization of larger off-highway trucks, with capacities of 400t. Although, truck haulage costs can account to up to 40% of the total mining costs, mine haul roads are mostly designed based on empirical methodology which will result in inappropriate design. Good haul roads are a key to successful surface mining operations. Poorly designed, constructed and maintained roads are major contributors to high haulage costs and pose safety hazards (Hustrulid and Kuchta, 2006). A road well designed, built and in good traffic conditions will allow vehicles to ride safely and efficiently. Moreover, a good road will also allow trucks to operate at their maximum safe speed. According to Kaufman and Ault (1977) a stable road base is one of the most important fundamentals of road design. Placement of a road surface over any material that cannot adequately support the weight of traversing traffic will severely hamper vehicular mobility and drivability. Moreover, lack of a sufficiently stable bearing material beneath the road surface will permit excessive rutting, sinking, and faster overall deterioration of the travelled way. Thus, an intense frequency of maintenance will be necessary to keep the road passable.

The aim of this paper is to present a low cost haul road design approach based on site characterization through the utilization of Dynamic Cone Penetrometer (DCP) for subgrade strength assessment and simple laboratory tests for wearing course material characterization and selection. DCP results were correlated with CBR values, for an initial structural design. Subsequently a CBR based design was assessed, compared and optimized with a mechanistic approach based on limiting strain values using the software EVERSTRESS for stress and strain calculation in the layers. Special attention was given to the functional project, as the CBR design approach does take into account the wearing course material selection. For the particle size distribution, the guideline limits established by the National Department of Transport Infrastructure (DNIT) for graded crushed stone, combined with Atterberg limits and CBR testing, were used. This approach presented good potential for initial wearing course composition determination. Final composition was based on the guidelines established by Thompson and Visser (1999) for wearing course material for ideal functional performance.

CASE STUDY

The project was developed at the Falchetti granite aggregate production unit, located in southern Brazil. The quarry is developed as traditional side hill mining moving towards an open pit with daily production capacity of 3000t. Historically, haul roads were designed on the quarry by simply dumping and spreading material over the existing subgrade. Lack of design for the haul roads, resulted in high maintenance costs for the truck fleet, at the same time reducing its availability. Also, the company did not use proper equipment like roller, grader or dozer equipment for road building, usually the company used a wheel loader, which is not adequate for the service, as it takes too long to do the work and the result is not adequate.

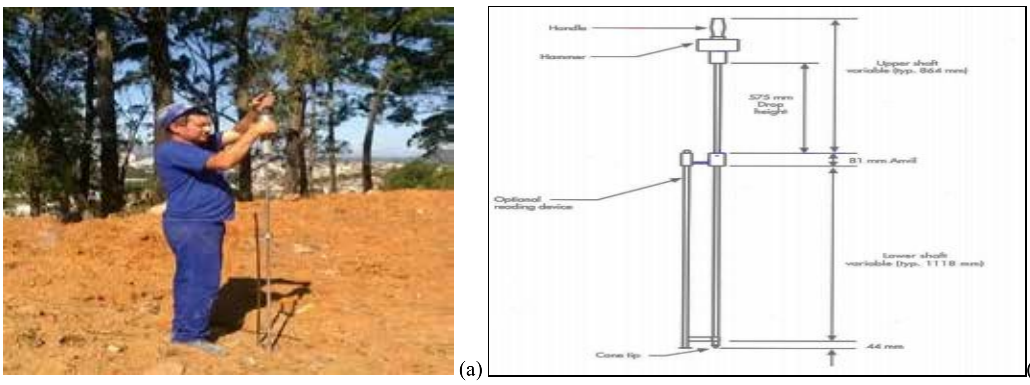
Actual access to the benches is taken through a ramp connecting the different levels of the quarry, as may be seen in Figure 1. The main ramp was built long time ago and is located west in the pit (to the right of the pit in Figure 1). The ramp presents several geometrical problems, firstly it was not built with a constant grade, presenting 16% on the initial part, 20% on the central part and 25% at final part. This design leads to excessive trucks maintenance, inefficient production and hauling costs increase. Secondly, the road is a two way path and it is not wide enough to allow two trucks to cross simultaneously, often resulting in the empty truck waiting at the bottom of the ramp, delaying the hauling/loading cycle. On the functional point of view, the most common defects noticed were dustiness, loose material and potholes. In the loading area it is common to have tire cutting due to loose material on the platform, resulting in truck maintenance cost increase and production losses as the company uses only a few haulage trucks.

Figure 1: Quarry overview

The pit will reach its eastern limit at the end of this year and will begin its expansion to the west once environmental permits are obtained. The expansion project requires the main access to be relocated allowing the company to extend the quarry´s life. The main goals of the new road project are: to postpone new haul truck purchases, as the transport distance is increasing, increase productivity and system efficiency and also maintenance costs reduction of the actual transportation fleet.

METHODOLOGY

When designing a new road, the first step is to investigate the material where it will be built over, once identified poor soils it should be removed and replaced by superior material or, when there is no adequate material available, structural design must consider sufficient cover thickness on top of the weak soil. It is generally cheaper, in terms of construction costs, to remove weak material and substitute it, instead of reinforcing the pavement. For the in situ material strength assessment, a DCP was manufactured at the quarry, following the design from the Minnesota Department of Transportation – Mn/DOT (Minnesota, 1993). As shown in Figure 2, the DCP consists of a 16mm diameter steel rod, to which a 20 mm 60° conical end is attached. It penetrates into the soil by dropping an 8kg hammer that slides on a steel rod, at a constant height of 575mm. The DCP can be found in the market for approximately US$ 600, considering the actual dollar exchange rate, the equipment would cost around R$ 2.000,00. The "home made" DCP cost was estimated in R$ 300,00, which is 15% of the commercial price. Its construction is simple, cheap and can easily be done at any mine site.



(a) (b)

Figure 2: Field testing with DCP (a) / Mn DOT DCP Design (b)

The soil was investigated with DCP for in situ strength assessment and samples were also sent to a soils laboratory for Atterberg limits test and CBR value determination. Several correlations between the DCP results and CBR values have been developed by a number of authors (Livneh & Ishia, 1987; Webster *et al.*, 1992; Trichês & Cardoso, 1998). Considering the study of Trichês and Cardoso (1998) developed in the same region in the south of Santa Catarina State over soils formed from similar geology, equation (1) was used. For mechanistic design the subgrade resilient modulus (E) was estimated through De Beer (1991), according equation (2).

$$CBR = 512{,}64 \times PR^{-1{,}25} \quad (1)$$

$$\log E = 3{,}05 - 1{,}07 \log PR \quad (2)$$

Where:

PR = DCP Penetration rate (mm/blow)

Initial structural design was based on the CBR method through its design curves, as shown in Figure 3. This methodology requires that CBR values and wheel loadings are first determined. It is important to mention that, although the method is simple, well understood and can give fairly good design guidelines for most haul roads, it should only be used as a guideline for the critical strain based method.

For the critical strain limit evaluation, the elastic modulus of the base and wearing course materials was determined from past work in the region. It is hard to evaluate mine haul road materials due to its dimensions (too coarse). Although it is fundamental to obtain materials modulus for mechanistic design, most resilient modulus testing equipment available does not support the large haul road particle size distribution. One alternative is to use equipment's such as light weight deflectometer and Benkelman beam for back calculation of layers' modulus. Another alternative is to estimate the modulus through references, such as the suggested values for granular materials presented by Thompson and Visser (1999).

Several studies have been developed in the region during the duplication works of the BR-101 federal highway. Oliveira (2000), in his work evaluates the mechanical behavior of crushed stone through back calculation, obtaining an elastic modulus of 208,5 MPa. Fernandes (2000), evaluated the mechanical behavior of graded crushed stone obtaining a modulus of 166 MPa by back calculation. The materials evaluated in both studies of Oliveira (2000) and Fernandes (2000) were obtained from a granite rock with similar characteristics of the Falchetti´s granite quarry. Although, these values are lower than suggested by Thompson and Visser (1999) they were used in this study considering they are representative of the available materials.

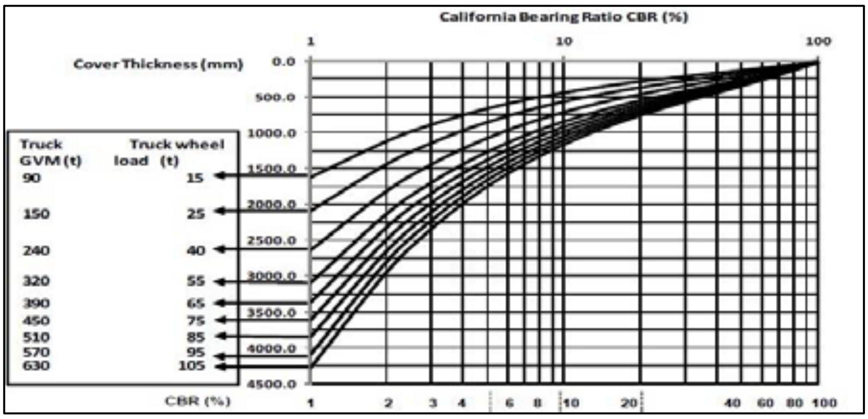


Figure 3: CBR design curves

Special attention was given to the wearing course material selection, as it does not take into account the properties of the surface material. Its composition was based on the grain size limits established on the DNIT standard for graded gravel. Thompson and Visser (1999) recommended parameter specifications for optimum functional performance, the authors suggested that based on shrinkage product and grading coefficient it is possible to predict wearing course defects. Figure 4 shows the graph to select wearing course material for ideal performance. Shrinkage product (Sp) and grading coefficient (Gc) can be calculated by equations (3) and (4), respectively.

$$Sp = LS \times P425 \tag{3}$$

$$Gc = \frac{(P265 - P2) x\ P475}{100} \tag{4}$$

Where:

LS = Bar linear shrinkage (can be approximated with the Plasticity index divided by two)

P425 = Percent passing 0,425mm sieve

P2 = Percent passing 2mm sieve

P265 = Percent passing 26,5mm sieve

P475 = Percent passing 4,75mm sieve

In figure 4, two areas are recommended for adequate materials ranges, where areas 1 and 2 are considered Ideal (1) and Operable (2), respectively. The designed wearing course material presented a Gc = 31,04 and Sp = 166,85 and it is represented in the graph by the yellow dot, falling in the operable area. However, it is expected that this material will present some wet skid resistance and loose material if not correctly maintained.

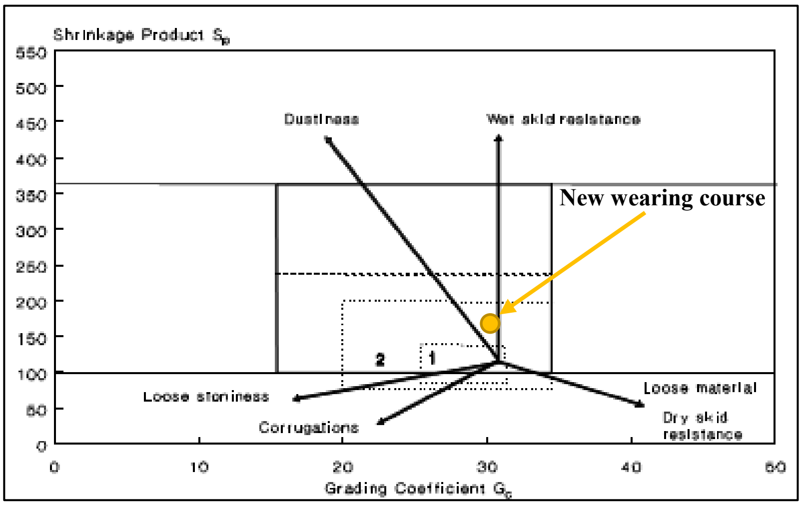


Figure 4: Material selection ranges, source: (Thompson & Visser, 1999).

RESULTS

There were made 10 DCP tests along the axis of the new road to characterize the in situ material. Soil strength was determined defining that weak material (CBR lower than 10%) that should be removed to a depth of 700mm, as shown in Figure 5(b). In the other hand, terrain profile requires cutting and filling for haul road building, where cutting volumes are considerably higher than filling. As the quarry has little space for waste storage, deeper cutting would result in an excess of waste material to be disposed. Also, a 10% CBR value is considered satisfactory for the subgrade layer requiring an acceptable cover thickness. Considering those aspects, the decision was to use material with CBR value of 10% or higher as base material, compacting the material during construction.

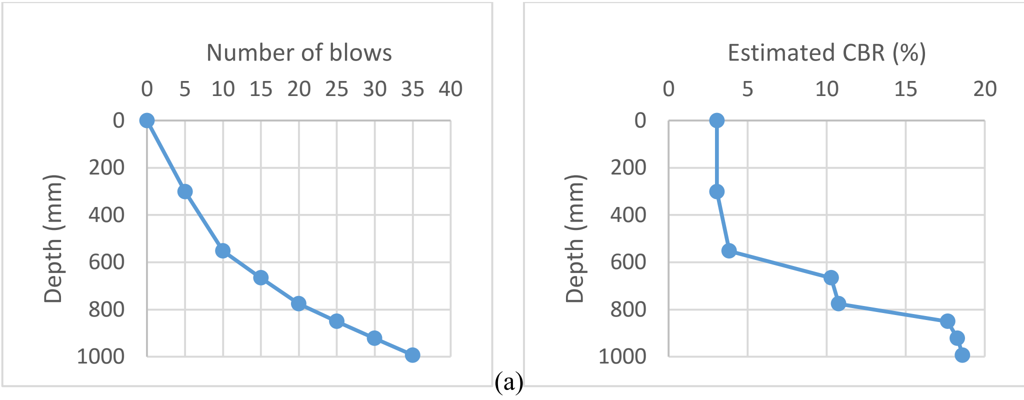


Figura 5: DCP results.

Surface layer design is slightly different from the other layers considering it must meet the general requirements as for the other layers, the design should also take care of operational aspects such as dust control, smoothness of surface, traction and rolling resistance. Material selection is usually based on local experience or guidelines related to unpaved public road construction. However, the unique service condition experienced by mine haul roads requires development of specifications tailored to those particular needs (Thompson and Visser, 2000). Wearing course material selection was based on the DNIT standards for graded crushed stone and the S_p x G_c graphic limits established by Thompson and Visser (1999). The resulting material was produced by mixing three materials in the crushing plant to ensure the correct grain size distribution and the final homogenization was made using the grader during construction.

Compacted natural gravel and crushed rock and gravel mixtures are widely used in surface mines for road construction, especially for the base and wearing layers. These materials can yield low rolling resistance and high traction, and can be constructed and maintained at a relatively low cost (Tannant and Regensburg, 2001). The mixture is composed of ¾" aggregate, 3/8" aggregate and undersized material from the secondary crusher as shown in Figure 6. Undersized material is a low cost material as it is currently discarded by the company, when studying wearing course material selection, mine planners should always look for waste material as a way of lowering the cost. Furthermore, it should also take into consideration the benefit of using waste material in the construction, reducing environmental impact with disposal.

GRADED CRUSHED STONE COMPOSITION							
ASTM	mm	DNIT - Env A		Mixture	Working Env		Tolerance
1"	25,4	100	100	100	100	100	+/- 7
3/8"	9,5	30	65	61	54	68	+/- 7
n°4	4,8	25	55	39	34	44	+/- 5
n°10	2	15	40	23	18	28	+/- 5
n°40	0,42	8	20	12	10	14	+/- 2
n°200	0,075	2	8	5	3	7	+/- 2

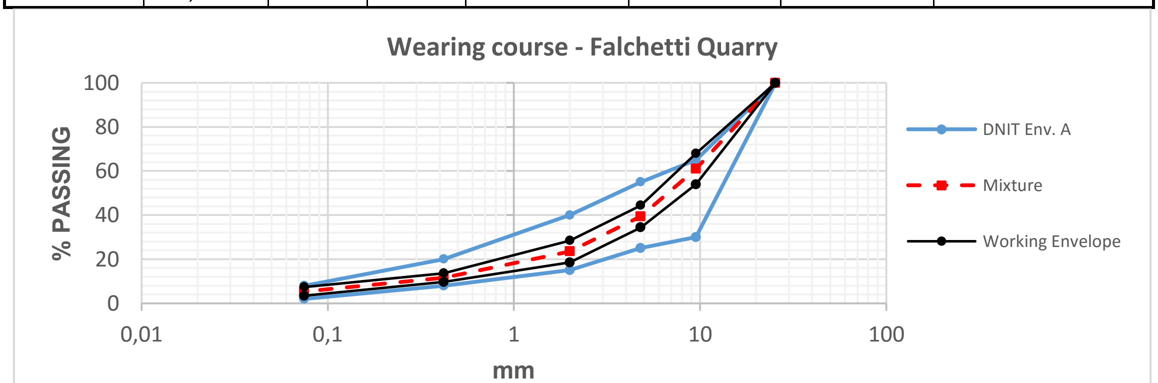


Figure 6: Wearing course composition.

The structural capacity was determined for the largest truck, a Volvo FMX 460 with 22m³ capacity. The truck has an empty operating mass of 18.930 kg and gross operating mass of 60.860 kg, resulting in a maximum wheel load of 4.944,88 kg on the rear wheels. This information, with the CBR value of the soil, wearing course and base material, was used to determine the total required cover thickness through the CBR design method. Wearing course material CBR value was determined from laboratory testing as 103%. Crushed stone base material CBR value was estimated from literature, and the value of 300% was adopted. The total cover thickness obtained was of 280 mm, comprised of 170 mm of base material and 110 mm of wearing course material.

According Tannant and Regensburg (2001), failure in the CBR method is assumed to occur when the tire penetrates a haul road layer or an upper layer´s material penetrates into the lower one, thus causing failure of the structure. Failure can occur even before such condition happens. The haul road cross-section acts as a layered beam structure. Under excessive strain, this structure can no longer act as a beam, thus losing strength, and failure becomes imminent. Consequently, it can be expected that a design using the CBR method would result in under-design in most cases, but sometimes in cases where haul roads are designed for a very short life, the CBR method can be over-conservative.

Haul road design by the CBR method was evaluated using EVERSTRESS to check if maximum vertical strain values exceeds the limiting design value of 2000 με. It was found that the design provided by CBR method, is undersized as shown in Figure 7 (blue line) and will not give enough support. Optimization of the CBR method, with the limiting strain value of 2000 microstrain, was carried out to obtain better structural response to the applied loads. As can be seen in Figure 7, using the critical strain limit design does not exceed the limiting design value for vertical strain (orange line). The optimized structural design suggests a 200mm of wearing and a 250mm thickness of base on top of a subgrade with a minimum 10% CBR. The haul road thickness had to be increased by 60% in order to fulfill the proposed design criteria, especially the limiting strain value. Although, haul road construction cost is higher according the mechanistic desing, as more quality material is used, requiring more compaction and grading activities, the long term benefits are incomparable with this additional cost. It is expected that trucks will be allowed to drive faster, from 20km/h to 30km/h, an increase of 50%. The production cost per ton will be reduced significantly as it will be possible to crush more material with the same structure considering the company has installed capacity with the excavator and loading equipment and also in the crusher, we must say that the trucks are the bottle neck so far.

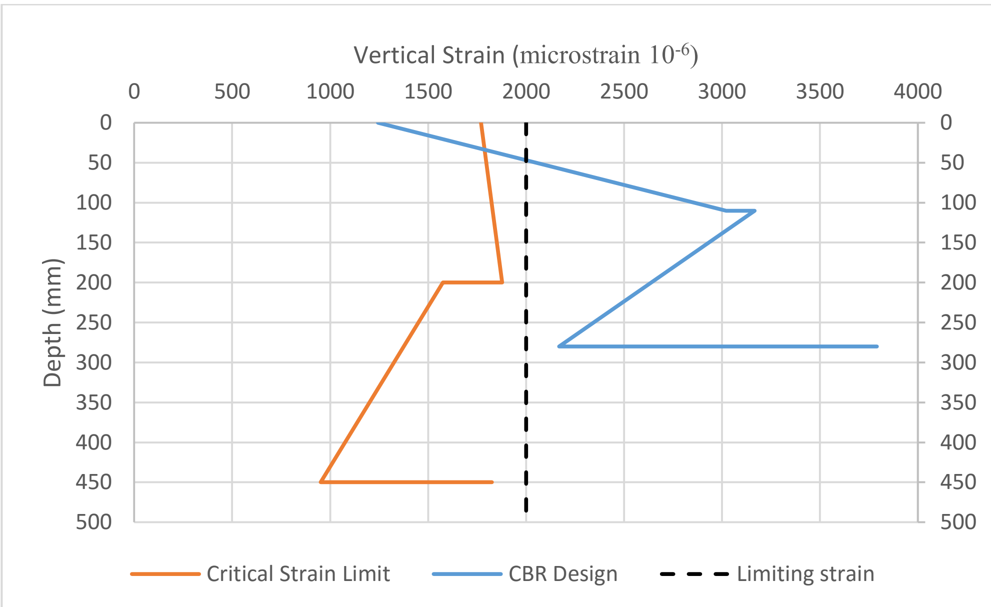


Figure 7: Design method comparison

Figure 8 shows the new wearing course and old haul road showing what was considered as sub grade material for the new road.


(a)

(b)

Figure 8: New haul road (a) and previous access (b)

CONCLUSIONS

Historical haul road design methods present deficiencies that result in operational underperformance. Lack of information about the in-situ soil over which the road is to be built is a major problem. Mining activity is highly dynamic and sometimes mine planners do not have enough time to evaluate materials, as it would take a few days for a laboratory to characterize. The DCP has been widely used during road construction or maintenance around the world due to its simplicity, low-cost, flexibility and prompt results. Moreover, it has been shown that CBR values can be obtained with high level of confidence from the DCP relationship. It is fundamental that mine planners know the resistance of the on-site material they are going to build a haul road, as sufficient cover thickness must be provided to protect the weak in-situ material from the wheel loads and volume of traffic imposed by trucks.

The CBR method should only be used for haul road design as an initial guideline for the mechanistic method. The CBR method considers failure to occur when a tire penetrates a haul road layer, but failure may occur even before it happens. The mechanistic method, on the other hand, considers failure to occur when the strain at any point exceeds the critical strain limit. The value determined as critical strain limit may vary, depending on the design life of the road and traffic intensity, but usual values are in the range of 1500-2000 microstrain. The material selected and used to build this new road presented a capacity of support that respects this limit and was optimized by the mechanistic method, after using a CBR approach as a start, providing a design with 200 mm for wearing course and 250 for base.

The CBR method does not consider wearing course material design making material selection for the surface layer arbitrary. It has been observed that DNIT standard for graded crushed stone combined with Thompson and Visser (1999) parameter specification may be used for wearing course material selection, indicating a good performance layer with low dust index.

This case study has shown a practical and simplified method of haul road design which provides an adequate structural thickness and a vehicle friendly wearing course. The method can be replicated in any mine. Although the new road is not yet in use by the time of writing the paper, initial cost benefits will be presented at the conference during the presentation.

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REDUCING COSTS IN QUARRING WITH OPTMIZED DRILLING AND BLASTING DESIGN

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REDUCING COSTS IN QUARRING WITH OPTMIZED DRILLING AND BLASTING DESIGN

ABSTRACT

This paper addresses the use of a cost saving methodology in a granite quarry. The study took place in a quarry in southeast Brazil. The main aspects considered during the study were drilling and Blasting, Hydraulic rock breaker, and crushing.

The use of Orica Shot Plus 5™ software and the Quarryman Laser profiler made possible to assess existing drilling accuracy. An extensive data collection took place to establish a robust baseline database.

With the aid of the design software, it was possible to design blasts with expanded patterns and optimized electronics timing, reducing drilling and blasting costs, while also improving fragmentation and productivity on oversize rock breaking and crushing. Overall, comparing unit costs on drilling and blasting, secondary rock breaking, and crushing of the baseline period against the demonstration period, the project team estimated the cost savings.

The results of the study showed it is possible, with optimized blasting designs and electronic initiation timing, to reduce powder factor, while increasing productivity on secondary rock breaker and crushing operations. The estimated benefits of all the cost savings were R$1.4 M/year.

KEYWORDS

Electronic detonators, open pit mining

INTRODUCTION

Traditionally, the quarrying industry control its costs by reducing costs in each unitary operation (drilling, blasting, excavation and transport, crushing) individually. The Quarry reaches its goals when all unitary operation costs are bellow established levels, but that is not necessary the lowest overall cost. Some works show the benefits of good performance in initial operations throughout the mining production chain (Thomson, 2012), (Grundstrom et. al., 2001).

For its initial position in the quarrying chain, drilling and blasting have effects in all sequential operations in the industry, and therefore can be considered most important. A good performance in drilling and blasting can generate smaller particles to improve productivity in crushing circuits (Thomson, 2012). Having increased productivity without changing significantly the equipment and total costs involved in quarrying, the unitary costs ($\$/m^3$) decrease in more productive operations.

This article addresses the applied study in drilling and blasting in a quarry in 2015, measuring cost reductions in drilling, blasting, secondary breakage, and crushing.

METHODOLOGY

An executive committee formed by the costumer's staff and Orica decided target mines and quarries for technical works on drilling and blasting. The committee chose its largest quarry as a potential target for its large-scale production (1.4 Mta). The local committee formed by unity management decided to focus on secondary rock breakage and crusher jamming issues, for considering those the operational bottlenecks at the time of the study. To have good process indicators, the local committee decided to measure the following KPIs: crushing jamming time, crushing productivity, rock-breaker work time and costs drilling and blasting

To stablish a solid baseline to measure potential benefits, Orica and Quarry crews measured the established KPIs for a month. The study took place in January/2015, and in that period, we analyzed three blasts. All the data collected was then transformed in unitary costs ($/m³) that were shared and agreed with all members of both committees

During baseline establishment, it was possible to define potential causes for performance losses in mark out, drilling and blasting. During mark out procedures, some blast holes were marked too close to the bench face, leading to relief charging using packaged explosives, leading to increased cost in drilling and blasting/m³. Some blast holes were marked too far away from the bench face, which led to poor fragmentation, and toe, which decreased crushing performance and increased drilling and blasting costs, due to re-drilling and charging the remaining toes and due to secondary breakage needed.

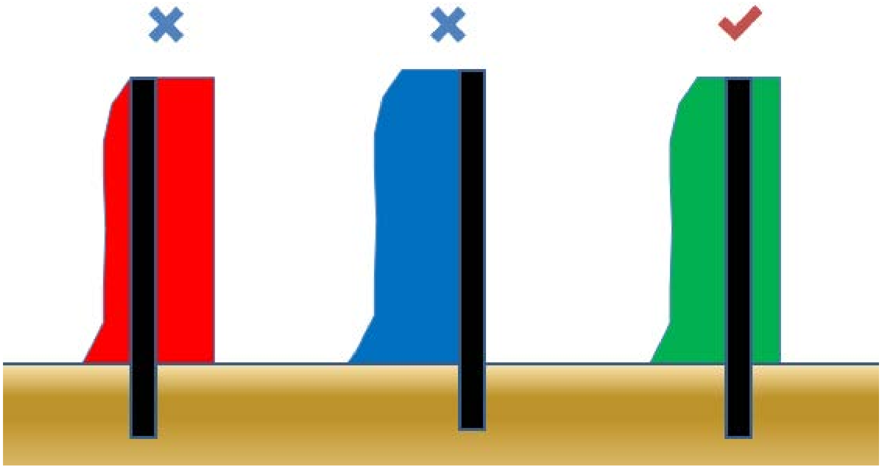

Figure 1 – Picture showing possible errors in front-row making, with a blast hole with low burden, a blast hole with heavy burden, and a blast hole with correct burden

Other source of productivity loss in drilling happened due to lack of leveling while marking out the blast holes. With the topography information acquired with the Laser Profiler TM and blast design tolls present in ShotPlus 5 Professional TM software, it was possible also to design shots with leveled holes, decreasing unnecessary blasthole filling or re-drilling

The committee also noticed that the quarry already used adequate blasting products and accessories: the blasts used ammonium nitrate emulsion Centra Gold 100TM, and electronic detonators Unitronic 600TM, but it was possible to make changes in charging and initiating sequencing to allow optimum product relative bulk strength (RBS) and blast shockwave interaction.

Orica crew marked the four blats performed in March/2015 using the Laser Profiler TM to allow optimum burden, the explosives loading procedures were changed to increase product RBS. The team also changed Blasthole initiation sequence to allow optimum burden relief rates for the granite rock, while also considering rock joint interaction in fragmentation. Using those techniques, the committee was able to design blasts using Orica Software ShotPlus 5TM using expanded patterns (10% increase)

Figure 2 – Blast performed on the Quarry

OPERATIONAL RESULTS

Comparing the results from baseline and demonstration blasts, crushing productivity increased by better fragmentation and filling capability. Drilling and blasting improved in productivity due to expanded pattern and better blast hole leveling. Fragmentation possibly improved as well, by visual comparison, and by decrease in secondary breakage time spent in the demonstration month (March/2015).

Having all the costs measured taken into account, the committee divided them by the month production during the baseline and the demonstration periods. The cost reduction calculated by all those factors was around R$104.000,00/month. Those costs were then annualized taking into account the quarry estimated annual production. The final result are shown the graph

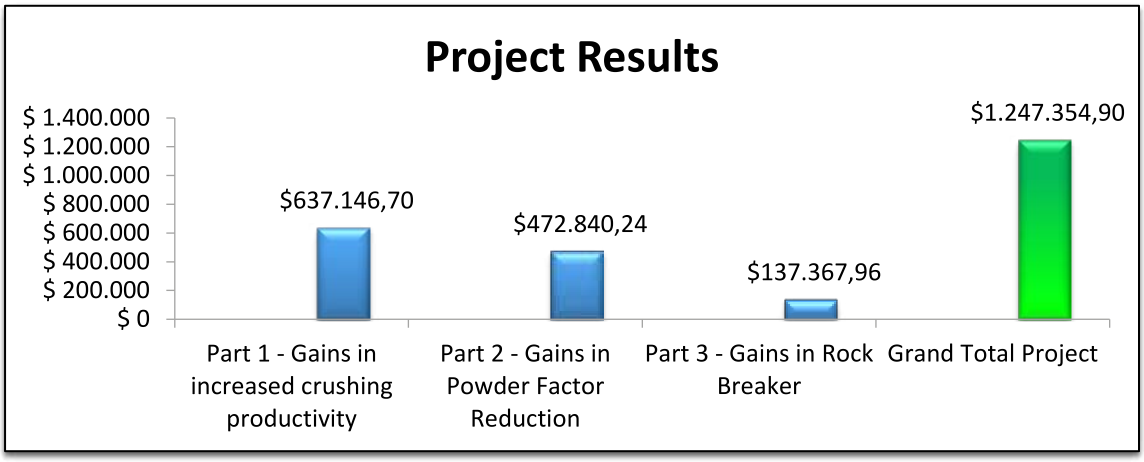

Figure 3 – Overall project results

CONCLUSION

In the recent years, more and more the industry is required to run effectively with lower costs, maintaining and improving levels of safety. The solution for these ongoing demands passes thought better understanding of the processes and an overall cost effectiveness approach.

This article showed some of the influences of effective drilling and blasting techniques in secondary breakage and crushing. It is very reasonable to deduce that benefits also extended to loading and transport, with improved diggabilty in excavators and better fill ratio of dump trucks.

Thus, the contribution of the application of good drill and blast practices not only the reduction of one unitary operation, but also the whole quarry production chain.

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REDUCING ORE DILLUITION WITH ELETRONIC DETONATORS

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REDUCING ORE DILLUITION WITH ELETRONIC DETONATORS

ABSTRACT

This paper addresses the use of advanced blasting design for the optimization of dilution and productivity in an open pit mine. The study took place in a limestone mine in Brazil in 2015. The work focused on the following aspects for optimization: minimum number of blast events to increase mining productivity, minimum dolomite/calcite ore dilution, optimum fragmentation, safety of the operation.

Shot Plus 5™ software made possible to create advanced blasting designs to induce muck pile segregation in a single blast event. The use of the Unitronic 600™ electronic detonators was fundamental to increase flexibility in muck pile movement, and to allow different ore zones to use optimum fragmentation timing. As an additional benefit, the use of remote blasting hardware increased safety of the blast procedure, compared to the conventional safety fuse initiation system.

With the aid of the advanced design software and the use of electronic detonators, it was possible to design larger blasts with increased volumes, increasing productivity, while allowing an increase in safety of the blast procedure.

KEYWORDS

Electronic detonators, open pit mining

INTRODUCTION

Traditionally, the mining industry has to deal with ore dilution to maximize ore volumes and to attend product specifications. After geological mapping, the industry blasts and mines different ore blocks separately, and volumes with different grades are blended.

This additional number of blasts created for this reason generates some issues on mining operation. Fragmentation on blast contours tend to be less effective, thus dividing the rock mass several times tends to create a larger volume of oversized material. The number of blast events also affects Drilling and Blasting, for the higher number of drilling and loading trips. Finally, the higher number of equipment evacuations performed affects the overall mining, leading to an increase in productive losses.

This article describes an advanced blasting design technique to promote ore segregation on blasting, to minimize ore dilution on a single blast event, decreasing production losses on drilling, blasting, excavation and transport fleets.

PROJECT DESIGN

On October 2015, the customer's staff contacted Orica for implementation of electronic detonators Unitronic 600™ on a limestone mine in south Brazil. The customer was interested in increase in fragmentation for ore and waste blasting, and a decrease of blasting accessories used in blast, for simplification of documentation of magazine management.

During the tests, performed October 12 and October 21, Orica used the electronic detonators Unitronic 600™ with remote blasting capability to increase fragmentation. Initial waste blast results, illustrated in figure 1, showed considerable improvement compared to conventional blasts initiated by non-electric detonators.

Figure 1- Waste material blast results

On the following days, Orica technical crew asked if any zones of mixed ore grades on the short-term mining plan. The customer's crew informed that bench 335 had calcite and dolomite zones, that were about to be blasted on the following weeks. Orica proposed blasting both zones with simultaneous initiation on calcite and dolomite zones, to induce differential rock movement and ore segregation. Figure 2 shows the bench chosen for the study with calcite zone showed in red, and dolomite zone dyed in blue.

Figure 2 – Bench chosen for study with calcite zone (red) and dolomite zone (blue)

Orica used the ShotPlus 5™ to design the intended blast with simultaneous initiation on both calcite and dolomite zones. The advanced blasting technique design is easier with the possibility of using any time with 1ms increment provided by the electronic detonators. Figure 3 shows the blast design with expected rock movement.

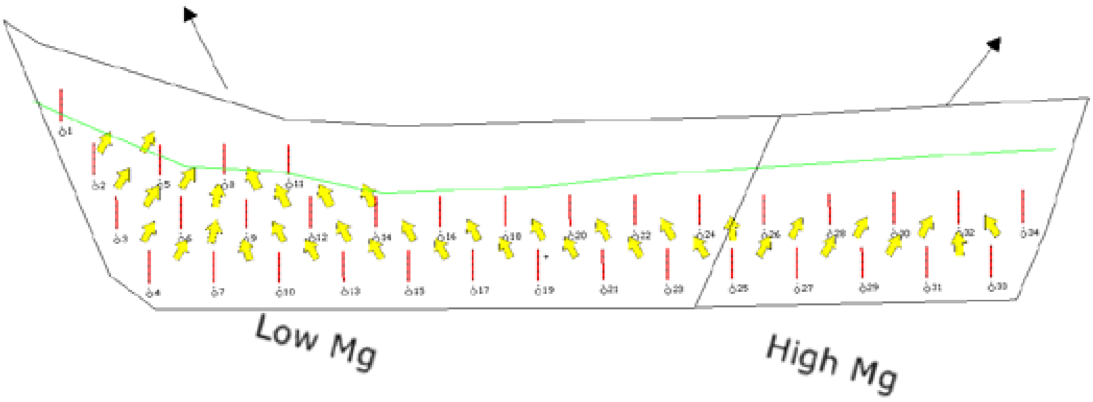

Figure 3 – Blast designed show expected rock movement

OPERATIONAL RESULTS

The blast performed on October 21 with no issues. Mining crew noticed slight segregation between calcite and dolomite zones. The initiation timing chosen for the specific test, which promoted fragmentation over rock movement, can explain this, leading to a less pronounced muck pile segregation as figure 4 shows. Nevertheless, both Orica and customer's crews considered the results positive, and further tests may induce more pronounced muck pile segregation.

Figure 4 – Blast results with slight muck pile segregation

CONCLUSION

In the recent years, more and more the industry is required to run effectively with lower costs, maintaining and improving levels of safety. The solution for these ongoing demands passes thought better understanding of the processes and an overall cost effectiveness approach.

This article showed some of the influences of the use of advanced blasting techniques to induce muck pile segregation and ore/waste separation on a single blast event. This technique allows a decrease in overall blast events, which increase not only drilling, blasting, but also excavation and transport productivities.

Thus, the contribution of the application of good drill and blast practices not only the produces benefits on one unitary operation, but also on the complete mining production chain.

SHORT-TERM SAMPLING SPACE OPTIMIZATION

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SHORT-TERM SAMPLING SPACE OPTIMIZATION

ABSTRACT

The sampling spatial distribution and procedures have a pronounced influence on the geological modeling, grade estimates, resources classification and mine planning. The collection, preparation and analysis of samples are time consuming and also expensive. Therefore, the sampling strategy must be carefully planned. The short-term sampling is a solid contribution to improve knowledge about the geological contacts and to promote the grade control from the mining operation perspective. The geostatistical analysis provides useful tools to define optimal sampling location in the short-term, avoiding lack or excess of sampling to be collected and analyzed. Insufficient samples affect grade estimations, generating values with low confidence to be assigned into the blocks. On the other hand, excess of samples generates unnecessary costs and time consumption for their preparation and analysis, which may contribute for delays during the grades model updating and consequently affect the decision making process. This work aims to demonstrate an application of a methodology to determine the optimal drill spacing for short-term purposes, using geostatistical simulation to assess the uncertainty and measure the benefit of cutting down and/or adding samples to certain areas. The techniques are tested in a phosphate deposit to find a representative grid that allows one to obtain suitable grade estimates for each block to improve the short-term mine planning.

KEYWORDS

MINE PLANNING, SAMPLING OPTIMIZATION, SHORT-TERM, GEOSTATISTICS, SIMULATION.

INTRODUCTION

The application of standardized sampling techniques helps all the subsequent stages of mining, such as, but not limited, to block model estimation, mine planning and processing. Therefore, the use of an accurate sampling protocol is essential for a better performance and accuracy of all the subsequent steps. Moreover, oversampling should also be avoided due to the high costs attached to the sampling tasks. For this reason, obtaining a suitable sample mesh becomes an essential, but not always obvious duty. In short-term operational scenario, it is important the sample collection and analysis to be efficient, so there is time to take necessary decisions in short periods.

Some authors use uncertainty measures to optimize sample patterns and/or drill spacing in a long-term scenario for resources and reserves classification. Koppe (2011) used uncertainty measures to locate new drillings on a regular grid of samples in order to reduce risk in a given transfer function. The author also mentions that the uncertainty level of a function that reflects the uncertainty of one or more geological attributes can be used in a mineral resource classification. The scenarios generated by stochastic simulation can also be used to analyze the uncertainty associated with the values of a global transfer function. When the transfer function is the ore tonnage above a given cut-off grade, each simulated scenario results in a value for ore tonnage and the uncertainty associated with this tonnage can indicate the reserves classification as probable or proven.

Emery et. al. (2009) used an optimization algorithm to find the sampling pattern that minimizes the average error from geostatistical simulation. The error depends on the ore variability, in a way that more samples provide better accuracy in the grade estimation. The minimum number of additional samples depends on the sampling objective (Koppe, 2011). In a short-term scenario, it is desired to collect a minimum number of samples to reduce costs and time.

Reducing the grade variability in processing plant feeding is fundamental, especially when the grades are low and there are contaminants. Geostatistical techniques, such as Ordinary Kriging and Indicator Kriging allow the construction of estimated models. However, these techniques do not give access to uncertainty and smooth the natural variance of the original data. Geostatistical simulation, on the other hand, allows to access the average error of the estimation for a better decision making process. This study aims to minimize the error by testing the effect of short-term samples spacing, comparing different sampling grids. The grid that shows higher spacing and minimize the error to an acceptable level will be considered optimal. The simulation algorithm used in this case study was Turning Bands (Matheron, 1973).

The first step is the simulation performance for the available data. Then, a simulated scenario is chosen as reference to obtain data with different spacing. The simulation algorithm is performed again for several spacing, and the uncertainty about the estimated concentrations is measured by the error calculation. Finally, the impact on the uncertainty by different data spacings is evaluated. This case study used data from a phosphate mine in Minas Gerais - Brazil.

Case Study

The methodology was tested using a short-term database, comprised by 299 rotary diamond drill holes, drilled in a regular grid of 25 m x 25 m, with different orientations according the geological zones. The holes have in average 35 m depth and the length of samples is standardized at 5 m.

The procedure was used to sample the main variable of interest, the apatite content (P_2O_5), in a specific region of the mine. In this region there are 650 samples, with average grade of 6% and variance of 4.75%². Figure 1 shows the plan view of the selected holes and the color legend for the P_2O_5 grades (a) and the histogram for the original data (b).

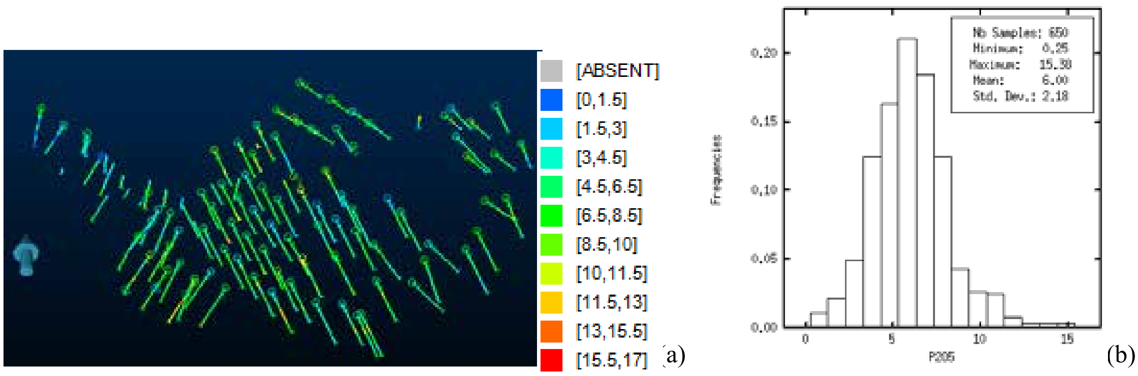

Figure 1. Sample location and original dataset histogram, respectively.

In Turning Bands the simulated values are derived from a distribution based on the associated theory with multi Gaussian random functions. The normalization of a distribution is generated from a function that can be graphically generated as shown in Figure 2, where the values corresponding to the p-quantile of the original cumulative data distribution are correlated with the corresponding values in the normal space. The histogram of the normalized data (Figure 3) follows a Gaussian distribution, with mean of zero and standard deviation of 1.

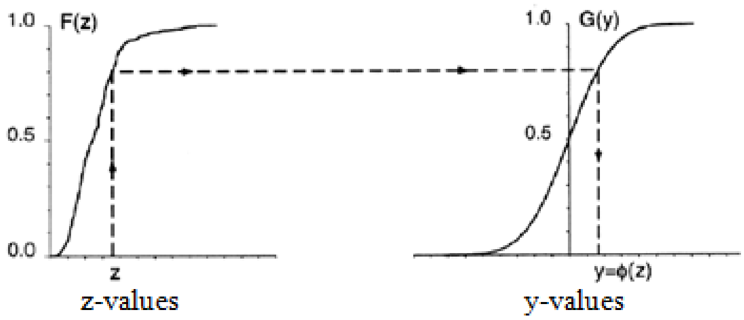

Figure 2. Process data standardization. Source: Goovaerts, 1997 – pp. 268.

After the data normalization (the histogram of original data normalized is presented on Figure 3), the analysis of the spatial continuity was performed using variograms. The variogram model of normalized data was inferred from the original data variogram model, as it has the same spatial continuity characteristics as the original data.

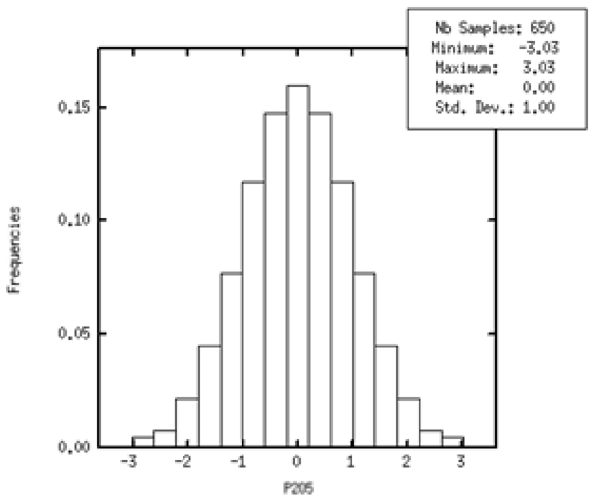

Figure 3. P_2O_5 normalized data histogram.

The variogram models may be seen in Figure 4 and the variogram equation can be seen in Equation (1). The variograms of P_2O_5 presented higher continuity in azimuth N67 with a 200 m approximately range.

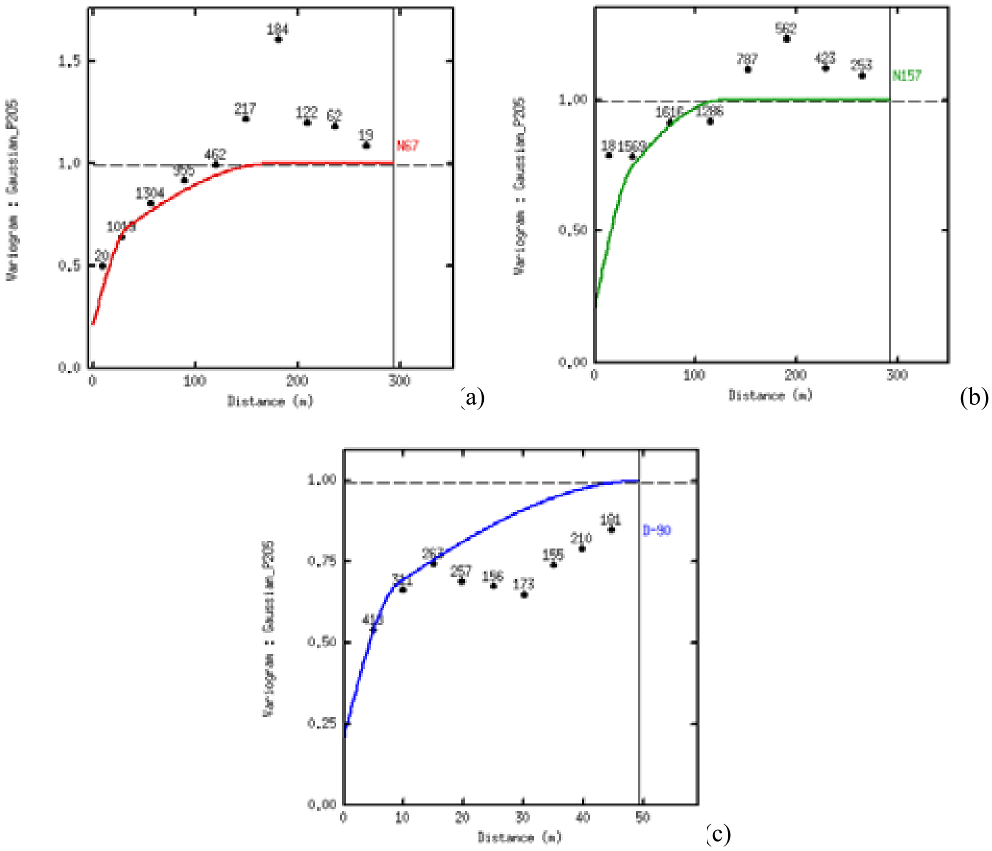

Figure 4. P_2O_5 standardized variograms in 3 directions. (a) major continuity direction (N67), (b) median continuity direction (N157), (c) minor continuity direction (D90). Continuous line represent the variogram model and dots represent the experimental variograms.

$$\gamma(h) = 0.2105 + 0.3579 \times Sph\left(\frac{N_{67}}{35} + \frac{N_{157}}{30} + \frac{vert}{9}\right) + 0.4316 \times Sph\left(\frac{N_{67}}{175} + \frac{N_{157}}{100} + \frac{vert}{50}\right) \quad (1)$$

The simulation was performed on small support scale (quasi-punctual) with grid spacing of 2m x 2m x 5m in the X, Y and Z, respectively, using the normalized data. The search parameters used in the simulation were the same as the scope of the continuity model in each direction. In total, 80 realizations for P_2O_5 grade were generated using Turning Bands simulation in the Gaussian space, and then back transformed to the original distribution.

The scenarios obtained by simulation must reproduce the histogram and variogram models of the original data, to validate the simulation. Figure 5 shows the variograms in three directions (colored lines), presenting ergodic fluctuations around the original data model (black lines).

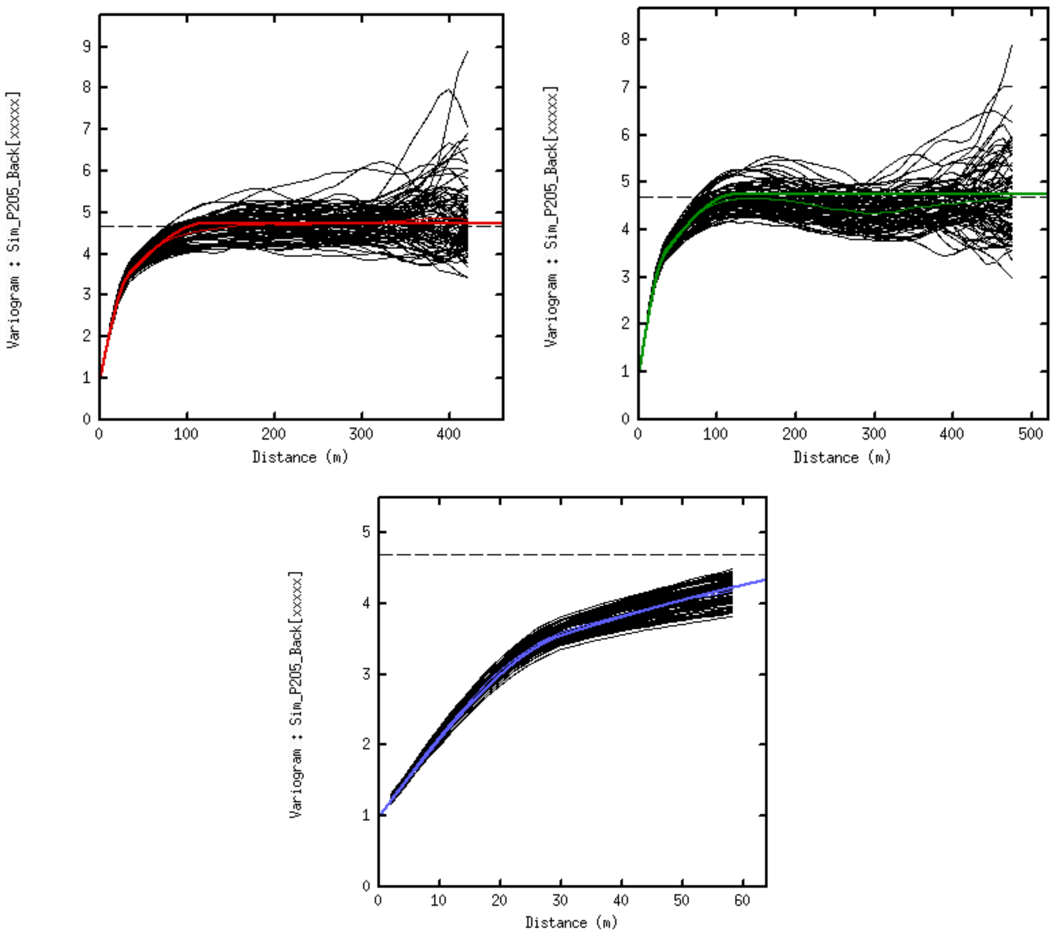

Figure 5. Experimental variograms for the 80 realizations (black lines) and variogram model's input to the simulation algorithm (colored lines).

The maps and histograms for the realizations with lowest, highest variance and the variance closest to the median are shown in Figure 6, Figure 7 and Figure 8, respectively.

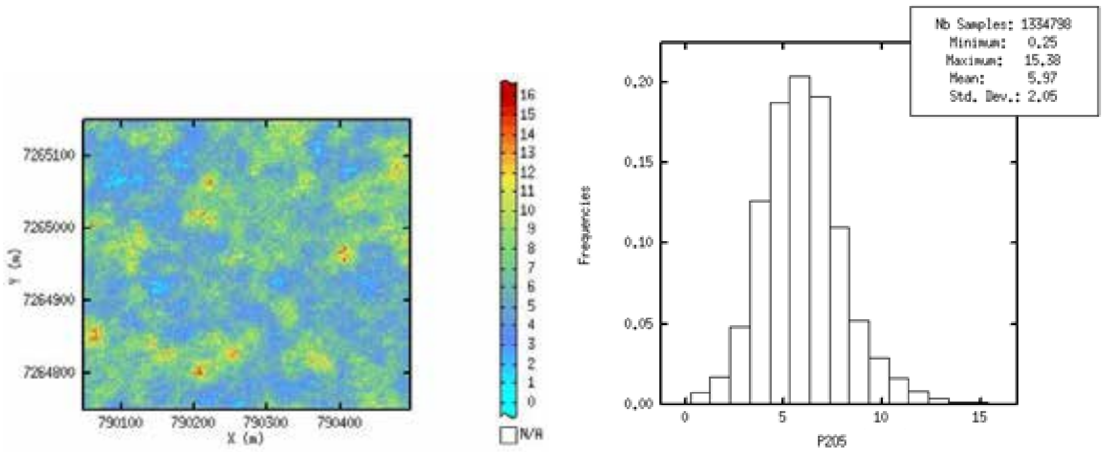

Figure 6. Map and histogram for the scenario with lowest variance.

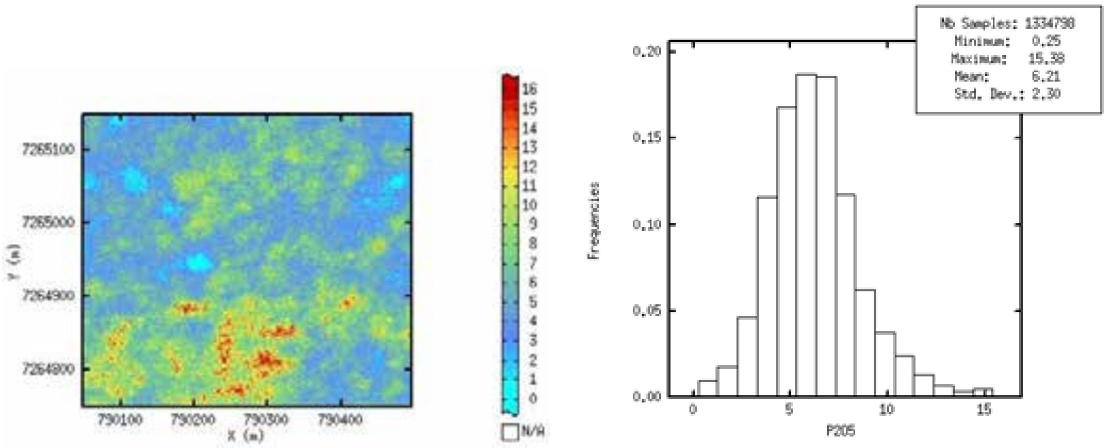

Figure 7. Map and histogram for the scenario with highest variance.

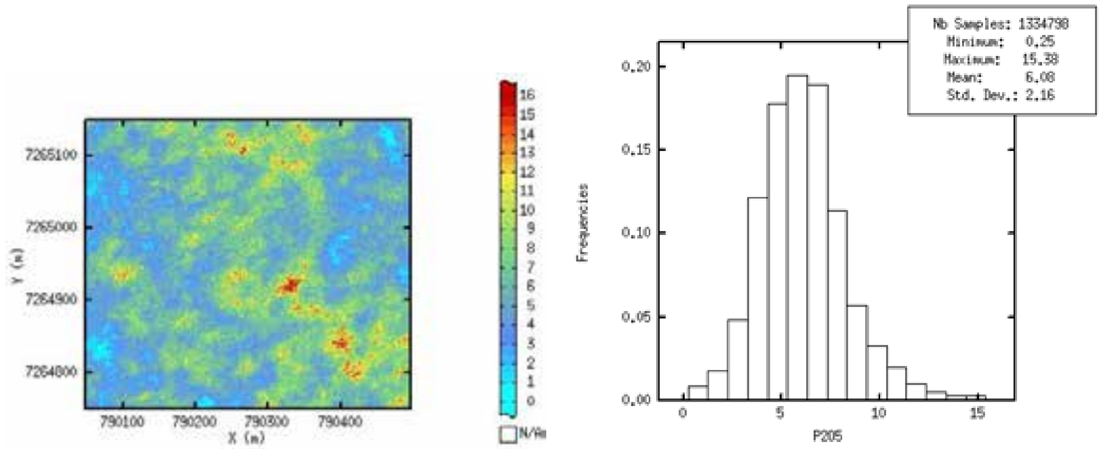

Figure 8. Map and histogram for the scenario with the variance closest to the median.

Table 1. Statistics for realizations in points support.

	Minimum (% P_2O_5)	Maximum (% P_2O_5)	Mean (% P_2O_5)	Variance (%² P_2O_5)
Realization	0.25	15.38	5.97	4.21
	0.25	15.38	6.08	4.68
	0.25	15.38	6.21	5.28

From the simulated data in each realization were calculated the P_2O_5 grades for 25,524 blocks of 10m x 10m x 10m in the X (east), Y (north), and Z (vertical) directions. The value of each block is the arithmetical average of the previously simulated points within these blocks.

The grid used for the simulation (2m x 2m x 5m) ensured that the distribution of the blocks presented theoretical variance to the variance of blocks (2,54% ²), considering the ergodic fluctuations, which simply means that the number of simulated points within each block is considered sufficient (for the study 50 simulated points were considered). Table 2 shows statistics for realizations with lowest, highest and the variance closest to the median values for the P_2O_5 in the support block.

Table 2. Statistics for realizations in 10m x 10m x 10m support blocks.

	Minimum (% P_2O_5)	Maximum (% P_2O_5)	Mean (% P_2O_5)	Variance (%² P_2O_5)
Realization	1.46	13.77	5.99	2.16
	0.95	13.94	6.07	2.56
	0.39	14.30	6.21	3.10

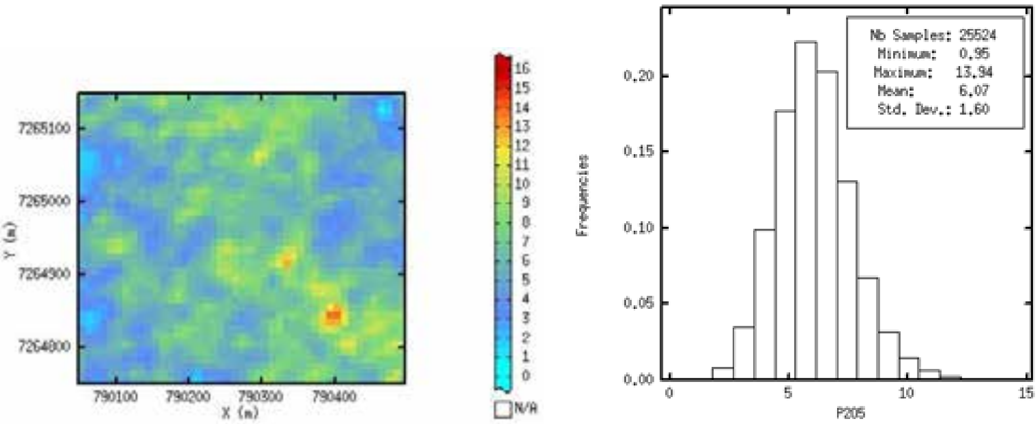

Figure 9. Map and histogram for the realization with nearest variance to the median values in blocks support.

Figure 10 shows the standard deviation for several simulated scenarios. The number of scenarios in which the standard deviation becomes stable is considered satisfactory to assess the range of the uncertainty space. Therefore, it is possible to verify that from 60 scenarios this aim is already reached.

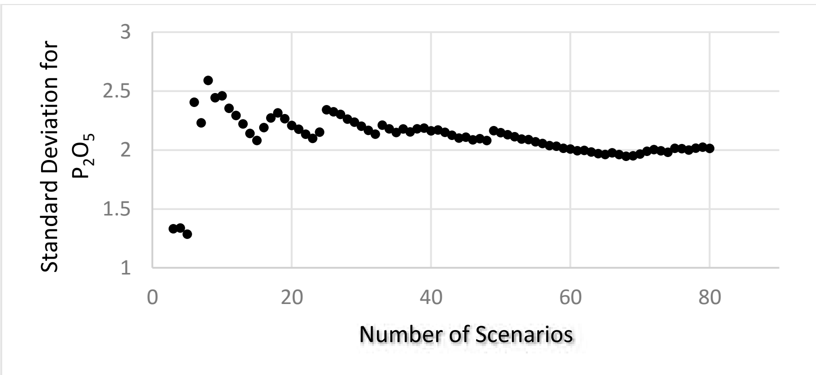

Figure 10. Number of scenarios versus standard deviation of P_2O_5 contents.

Additional Data Obtained From Simulation

Among the 80 simulation scenarios only one was chosen as the reference for the creation of virtual dataset. The reference scenario was the 48th considering it has the closest variance to the median variance of the data, however all scenarios are equiprobable to represent the mineral deposit. Using the realization in points support (2m x 2m x 5m), an interpolation by the nearest neighbor method with a search radius of 0.001 meters at nodes with spacing 6m x 6m x 5m at east, north and vertical direction, respectively. The values interpolated represent the values of samples from a sampling campaign. This procedure created a new dataset with 148,856 samples spaced by of 6m x 6m x 5m.

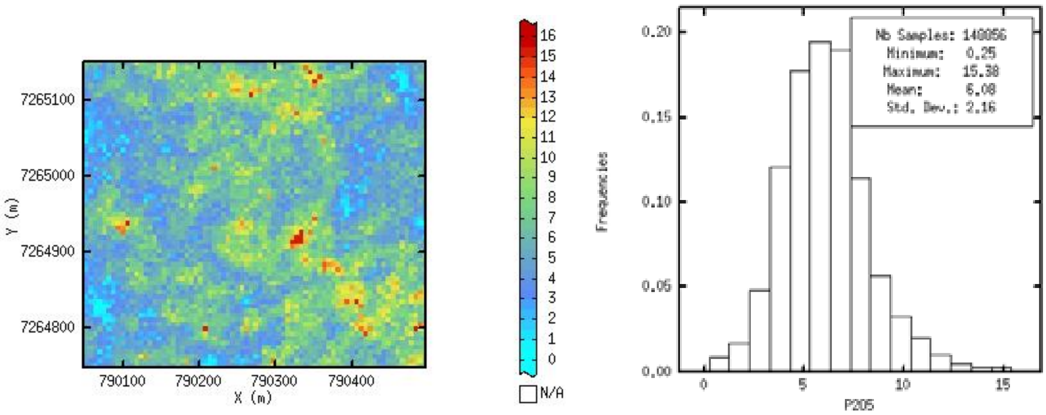

Figure 11. Map and histogram for the new dataset.

These data were normalized, and the variograms were adjusted due to a change in the variance. A new simulation was performed with these data to calculate the new error. From the simulated levels in each realization, the P_2O_5 grade was calculated for 34,200 blocks of 10m x 10m x 10m in the X, Y, and Z directions. The value of each block is the arithmetical average of the points within the blocks.

The result of the simulation provides for each block a distribution probability based on the obtained realizations. The error of the estimate is calculated after the results treatment, by compiling all realizations for each block and their calculation formula is shown in Equation (2), where the Q95 is the quantile 95, Q5 is the quantile 5 and Etype is the average of all scenarios. Equation 2 supposes a distribution of simulated values symmetric around E-type (as illustrated in Figure 12).

$$Error = \frac{Q95 - Q5}{2 \times Etype} \tag{2}$$

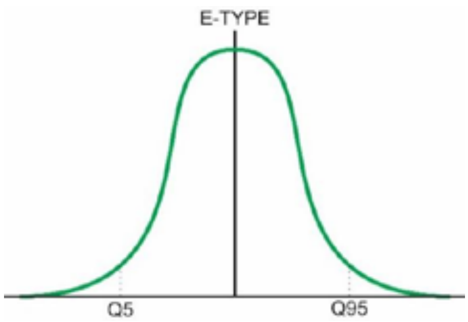

Figure 12. Representation of quantile and E-type realizations.

Based on the information from the mine considered for this case study, an error of 10% about the estimated of each block value can be accepted. With the 6 m x 6 m grid was obtained an average error of 8% for all simulated blocks, which means that when kriging is performed with the data in mesh 6 m x 6 m, the error of the estimated content is 8% higher or lower. Note that this error is an approximation of the maximum error about the estimate of each block.

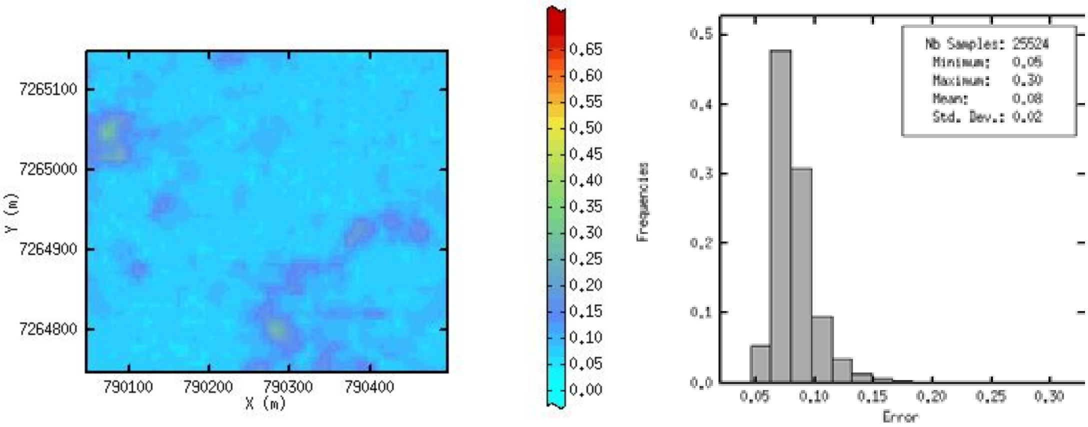

Figure 13. Error map and the histogram to the grid 6m x 6m x 5m.

Using the same methodology, were tested grids with spacing of 10m x 10m, 14m x 14m, 18m x 18m and 22 m x 22m. The results can be seen in Figures 14 to 17.

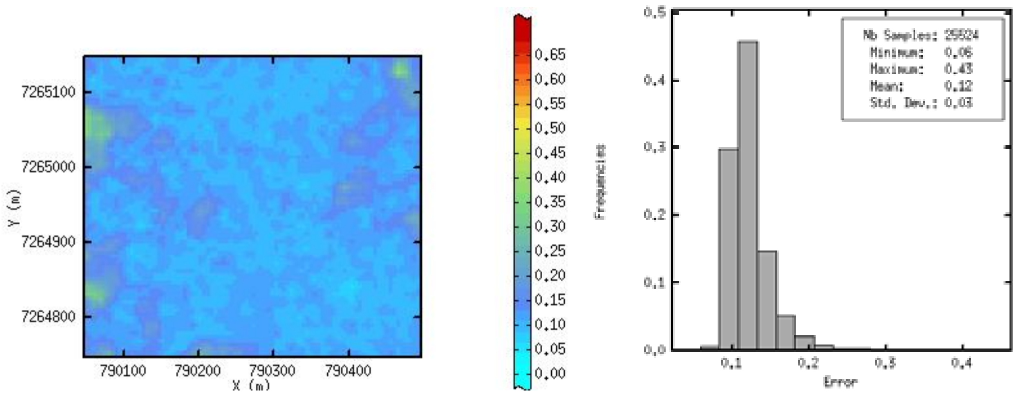

Figure 14. Error map and the histogram to the grid 10m x 10m x 5m.

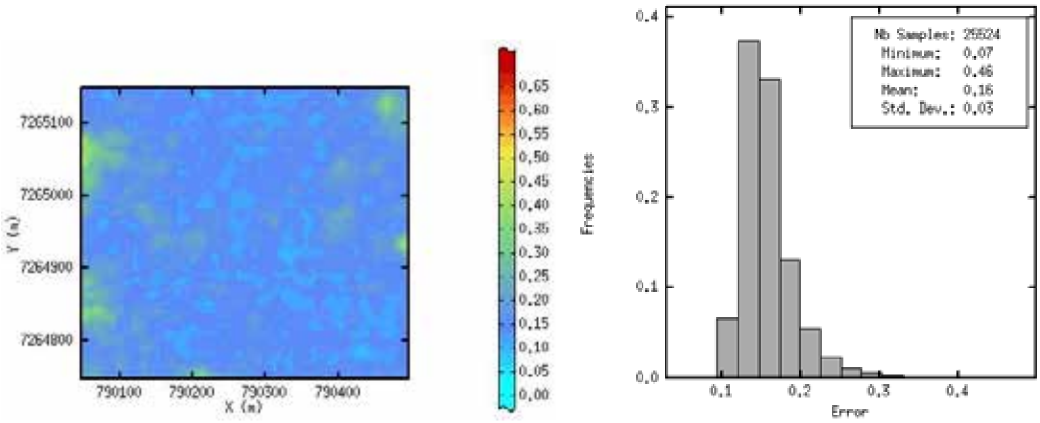

Figure 15. Error map and the histogram to the grid 14m x 14m x 5m.

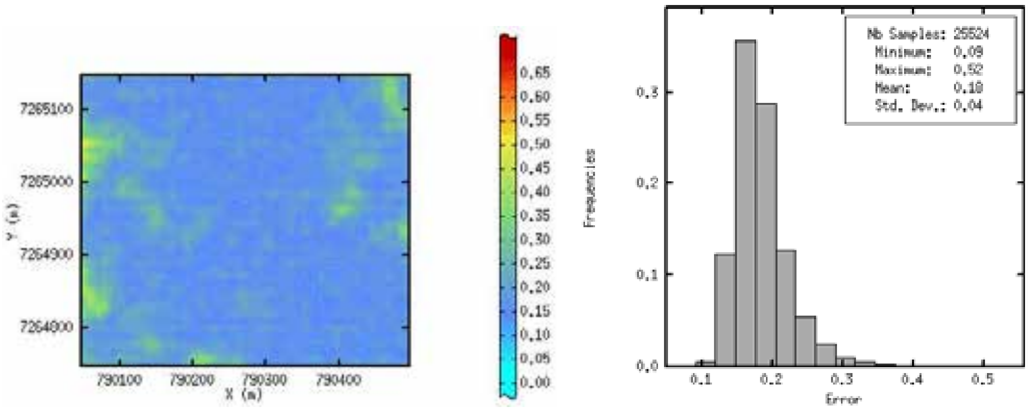

Figure 16. Error map and the histogram to the grid 18m x 18m x 5m.

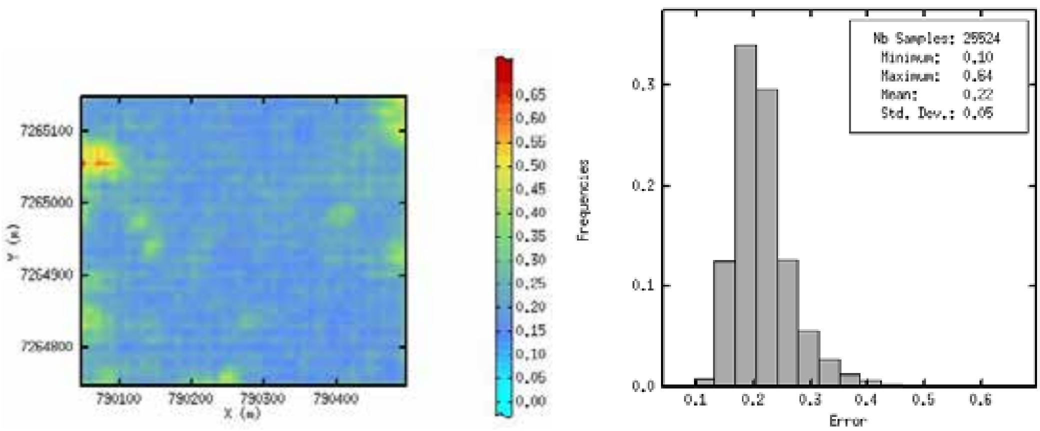

Figure 17. Error map and the histogram to the grid 22m x 22m x 5m.

Figure 18 shows the error obtained for the different grids tested. As can be seen the error increases acoording the sampling density is sparser. With this graph it is possible to select the "optimal" drill hole spacing considering the acceptable error.

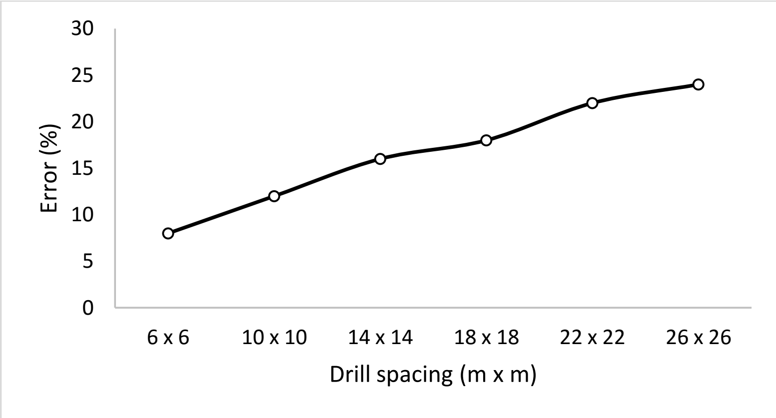

Figure 18. Average error versus drill spacing.

CONCLUSION

The simulation conducted in this study generated 80 equally probable scenarios for the variable P_2O_5, which reproduced the statistics and spatial continuity of the initial data. Considering all the scenarios are equally possible representations of the deposit, one of these scenarios was used as a source of information for testing different sampling spacing. Other scenarios can be chosen as a reference for application of the methodology, where the error on the estimate considered in this study, should be similar for different scenarios used as a source of information.

The results showed that a grid of approximately 8m x 8m could be used to produce the acceptable error for the study area (10%). It should be noted that for regions with different variability of P_2O_5 grades the result may be different. Usually the operation uses a spacing of 25m x 25m. According to the results, this spacing is not sufficient to ensure the minimum acceptable error, which means there will be more uncertainty during the planning of the mine, and the block sent to the process plant will have associated error.

The uncertainty magnitude of the estimates depend on the local sampling variability. Clearly, the use of additional information decreases uncertainty and error because the most information available will result in a more precise estimative. Otherwise, more samples mean more money and time spent. The determination of acceptable error and a proper sampling grid is fundamental to mine planning optimization.

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SIMULATION AND ANALYSIS OF FRAGMENTATION IN ROCK BLASTING AT THE HERVAL QUARRY - BARREIROS – PE

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SIMULATION AND ANALYSIS OF FRAGMENTATION IN ROCK BLASTING AT THE HERVAL QUARRY - BARREIROS – PE

ABSTRACT

The sequence of activities in quarry mining production process involves, basically, the following unit operations: blasting, drilling, loading, transport and crushing, which requires a strict control and continuous monitoring. The rock blasting is an operation that aims to turn the rock into several smaller pieces so they can be transported and crushed by available equipment. One way to improve the production process in quarries, with respect to the rock blasting, is to conduct a more detailed structural analysis of the rock mass, so that can be done an estimative of the mechanical and structural behavior of orebody. For this, it makes use of algorithms that allows analyze the degree of fragmentation by changing the geometrical and geomechanical parameters. Such algorithms results from implementation of the equations proposed by Kuznetsov (1973), Cunningham (1983), Lilly (1986), Rosin-Rammler (1933) and Tidman (1991), and are used, basically, for studying the interaction between explosive and rock. Hence, the probability of success in the blasting operation increases significantly, once it is possible to simulate several scenarios for fragmentation that intends to be achieved. In this context, looking for optimizing the blasting operation at the Herval Quarry that is located in the Barreiros town - PE, this study aims to compare the classical methodology for the fragmentation of rocks, which uses the analytical models, with the method of fragmentation analysis by image, and then evaluate the standard of effectiveness of both methods. The study was done through field data provided by Herval Quarry which were analyzed by Kuz-Ram methodology and by Split-Desktop software for evaluation of fragmentation through images. It can be concluded that the Kuz-Ram model coupled with the Split-Desktop software is a very effective tool for prediction of fragmentation characteristics of blasting, that leads us getting the optimal fire plan for the rock mass.

KEYWORDS

Kuz-Ram equations, Blasting technics; Quarry mining; Rock fragmentation

INTRODUCTION

Various authors have studied models explosive-rock interaction, which provide a useful description of the detonation process as a "background" process for modeling the fragmentation of the rock by explosives. Among the major works include: Kuznetsov (1973), Cunningham (1983), Lilly (1986), Sarma (1994), and Djordjevic (1999).

The rock masses have geomechanical characteristics that depend on the geology of the rocks themselves form and also some geotechnical parameters that are intrinsic of each mass and its tectonic history (Hudson & Cosgrove, 1997). Rarely rock masses have a homogeneous structure and, as a result of the complex history geodynamics (expressed by the regional stress field that were subject) have a network of discontinuities that will have significant impacts on their geomechanical behavior (Gama, 1995). Therefore, there is a need to previously determine the discontinuities in the rock mass and deforming stage reached in order to reduce rock blasting costs. An effective method in this process and will be object of study in this project is the evaluation of discontinuities, analyzing the deformation markers in rock masses, structural and petrographically for subsequent application of these discontinuities in the fire plan through the mathematical model Kuz- Ram.

The Kuz-Ram fragmentation model

In this model, the properties of the rocks, the properties of the explosives and the geometric variables of fire plan are combined using five equations that make up the model of Kuz-Ram fragmentation (Lilly, 1998):

• Breaking theory (Kuznetsov, 1973): the amount of breakage that occurs with a known quantity of an explosive energy that can be estimated using the equation Kuznetsov.

$$X_{50} = A\,x \left(\frac{V_0}{Q_e}\right)^{0,8} x\,(Q_e)^{1/6}\,x \left(\frac{E}{115}\right)^{-19/30} \quad (1)$$

where X_{50} is the average particle size (cm), A is the rock factor, V_0 is the volume of rock disassembled by hole (m^3), Q_e is the mass of the explosive used (kg) and represents the relative energy mass (RWS) of the explosive compared to ANFO.

• Theory size particle distribution (Rosin-Rammler): size distribution of the fragmented rock particles can be determined from the average size, the breakage model is known.

$$P = 100\,x\left[1 - e^{-0,693\,x\,\left(X/X_{50}\right)^{n}}\right] \quad (2)$$

where P is the percentage passing, X is the mesh size sieve, and n is the uniformity index.

• Explosive Detonation Theory (Tidman): the amount of energy released by the explosive is calculated using Tidman's equation.

$$E = \left(\frac{VOD_e}{VOD_n}\right)^2 x\,RWS \quad (3)$$

where E is the effective power by relative mass of an explosive; VOD_e is the speed of effective explosive detonation (measured in the field); VOD_n is the nominal velocity of detonation of the explosive (m/s) and RWS is the energy per mass relative to ANFO (%).

• Correlation of fire plan parameters (Cunningham, 1987): there is a correlation between the various configurations of the fire plans and model of rock fragmentation.

$$n = \left[2,2 - 14\,x\left(\frac{B}{D}\right)\right] x \left[\frac{\left(1 + S/B\right)}{2}\right]^{0,5} x \left[\left(1 - \frac{W}{B}\right) x \left(\frac{L}{H}\right)\right] \quad (4)$$

where B is the burden (m); S is the spacing (m); D is the hole diameter (mm); W is the drilling standard deviation (m); L is the total length (m), and H is the height of the bank (m). When using two explosives in the hole (bottom load and column load), the equation is modified to:

$$n = \left[2,2 - 14\,x\left(\frac{B}{D}\right)\right] x \left[\frac{\left(1 + S/B\right)}{2}\right]^{0,5} x \left\{\left(1 - \frac{W}{B}\right) x \left[abs\,x\,\frac{(BCL - CCL)}{L} + 0,1\right]^{0,1} x \left(\frac{L}{H}\right)\right\} \quad (5)$$

where BCL is the length of the bottom charge (m); CCL is the length of the column load (m); abs is the absolute value concerning (BCL - CCL)/L.

• Correlation of rock types (Lilly, 1986): the properties and characteristics of the rock mass interfere with the result of the fragmentation of detonation. The classification geomechanical Lilly and subsequently modified by Cunningham is used in Kuz-Ram fragmentation model.

$$A = 0,06\,x\,(RMD + RDI + HF) \quad (6)$$

where A is the rock factor, and the values of RMD are obtained from the geomechanical classification of Lilly, and other parameters are obtained with the help of equations:

$$RDI = 25d - 50 \quad (7)$$

$$HF = E/3\,,\,se\ E < 50\ GPa \quad (8)$$

$$HF = UCS/5\,,\,se\ E > 50\ GPa \quad (9)$$

where d is the density of the rock, E is Young's modulus (GPa), and UCS to the uniaxial compressive strength of the rock (MPa).

Fragmentation analysis by image processing

For the analysis of rock fragmentation in blasting was used the Split-Desktop® Version 3.1 program and pictures got in the field. This program is a resource used in image processing to calculate the size distribution of rock fragments through digital image analysis. Digital images can be acquired using digital camera, or from the cell material in the square (original surface or cutting excavation), transport truck and conveyor belt.

METHODOLOGY

The methodology consisted basically in three stages. The first consisted in the recognition of the study area, which is the quarry Herval, located in the city of Barreiros - PE, as well as the collection of technical data to determine the parameters necessary for geomechanical classification of rock mass; the second step consisted in the simulation of fragmentation through the Kuz-Ram model to obtain the particle size distribution curve; and at the third step was carried out the analysis of image processing for fragmentation using the Split-Desktop software.

Collection of technical data

The activities mentioned in the quarry for data collection were: recognition of mining fronts, fronts of measurement to determine the points of existing fractures, orientation of attitudes (sense and diving) fractures and detailed survey of the fire plan.

The quarry Herval has two mining fronts and their exploitation is done in benches, each mining front has three floors, subparallel and horizontal. After collecting the bench data provided by Quarry Herval, it was made tracking a rock blasting, which was held on May 8, 2015.

To determine the RMR was necessary to identify the discontinuities presents on the face of the bench and measure the distances between them, as shown in figure 1, the presence of water in the discontinuities, the degree of weathering of the discontinuities, and the characteristics of surface roughness. All information were measured in the field during the visit to Quarry Herval, as shown in table 1.

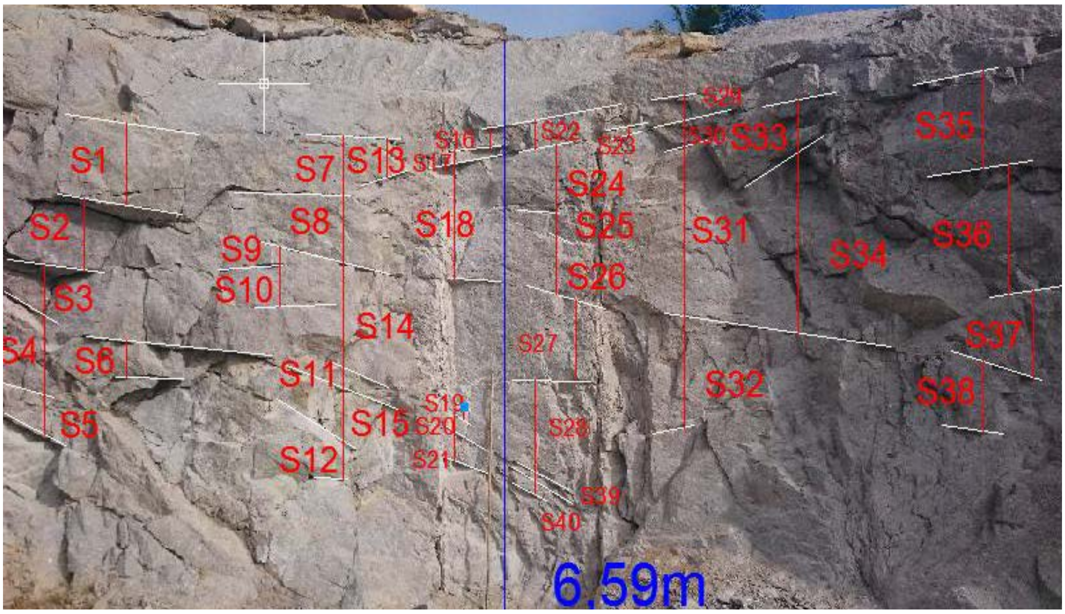

Figure 1 – Bench face showing the discontinuities and the distances between them

Table 1 – Rock mass classification of the quarry Herval

PARAMETERS	WEIGHTS
Uniaxial compressive strength	12
RQD (%)	20
Spacing between discontinuities	15
Condition of discontinuities	20
Water presence in discontinuities	15
WEIGHT TOTAL	**82**

Fragmentation simulation by Kuz-Ram model

The Kuz-Ram model is often used for fragmentation prediction because even the latest models are based on the conventional model of Kuz-Ram. Other models that can be used for this purpose are the knowns Extended Kuz-Ram or KCO Model; Modified Kuz-Ram or CZM and Two component Model of blast fragmentation or TCM. The Kuz-Ram model is based on empirical equations that predict the average fragment size from the load ratio, mass explosive per hole, mass relative energy of the explosive and blastabilidade index. This model requires information about the rock mass. In the equations given there are factors that concern about the geomechanical and geological characteristics of the rock. For this, the classification was made based on geomechanical classification proposed by Bieniawski (1989).

For the simulation of blasting rocks in the quarry Herval through the Kuz-Ram model, a spreadsheet was developed using Excel software to assist in the calculations of the equations proposed by the model. The simulation results are shown below in tables 2 to 4 which show the fire plan parameters, data from the Kuz-Ram model and rock fragmentation results of the Rosin-Rammler model to blasting.

Table 2 – Monitoring the blasting at the quarry Herval

Identification		**Calculation of loads**		
Quarry: Herval		Column load (m)	**0,00**	
Bench:		Bottom load (m)	**5,50**	
Date blasting: 19/06/2015		Full load (m)	**5,50**	
		Load ratio (Kg/m³)	**0,80**	
Hole diameter (pol)	**3,0**	**Explosive consumed (kg)**		
Burden (m)	**1,6**	Emulsion pumped	**0,00**	
Spacing (m)	**3,2**	Cartridge emulsion 2$^{1/4}$"x24"	**1650,00**	
Inclination (degrees)	**15**	**Total**	**1650,00**	
Stem (m)	**1,0**	**Explosive material**		
Meters drilled	**403,00**	**Explosive**	**Amounts**	
Total number of holes	**62**			
Average lenght of holes	**6,50**	Cartridge emulsion	1650,00	
Blistering fator	**1,5**			
Average height	**6,28**	Binding shock tube (17/25/42) ms	61	
Volume per hole (in situ)	**33,28**			
Total volume (in situ)	**2063,36**			
Total volume (swelling)	3095,0	Fuse	2	
Explosive load	**Diameter**	**Linear rate of loading (Kg/m)**		

Pumped emulsion	**3,00**	**4,69**	
Cartridge emulsion	**2,25**	**4,84**	

Table 3 – Kuz-Ram model data from rock blasting

	Description	Classification	Índex
RMD	Description of the rock mass	Friable	10
		Fractured	JF
		Massive	50
JF	Fractured massive	JPS + JPA	
JPS	Spacing of discontinuities (m)	< 0,10 m	10
		0,10 a MS	20
		MS - DP	50
MS	Oversize primary crushing (m)		
DP	Drilling loop parameters (m)		
JPA	Direction and dip with respect to free face	Horizontal	10
		Dipped out the free face	20
		Direction perpendicular to the free face	30
		Plunged into the free face	40
RDI	Influence density (g/cm³)	RDI = 25d - 50	**18,75**
HF	If E < 50 GPa	HF = E/3	**0**
	If E > 50 GPa	HF = UCS/5	**35,33**
E	Young's Modulus (GPa)		**64**
USC	Uniaxial compressive strength (MPa)		**176,63**
	Rock factor (A)	**Tidman equation: Energy Explosive (Er)**	
RMD	50	**VOD_e**	4100
RDI	18,75	**VODn**	5000
HF	35,33	**RWS**	86
A	**6,2448**	**Er**	**57,8264**
	Kuznetsov equation		
V_0	Rock blasting volume		2063,36
Qe	Explosive mass (Kg)		1650,44
K	Load ratio (Kg/m³)		0,80
X_{50}	Average size of the particle (cm)		**39,76**
	Cunningham Uniformity Index (n)		
B	Burden (m)	1,6	
S	Spacing (m)	3,2	
D	Hole diameter (mm)	76,2	
W	Standard deviation drilling (m)	0,00	
L	Total length of the load (m)	5,50	
H	Bench height (m)	6,50	
N		**1,975**	

Table 4 – Data of rock fragmentation in the blasting

	Rosin-Rammler equation (P)	
	Rocks Fragmentation	
	Sieve (cm)	**Passing (%)**
X	1	0,048008388
	2	0,188636898
	3	0,419711810
	4	0,739674470
	5	1,147023734
	6	1,640186276
	7	2,217466315
	8	2,877023389
	9	3,616862958
	10	4,434833923
	20	16,33615872
	30	32,78765843
	40	50,40733475
	50	66,37139288
	60	79,03345191
	70	87,97634093
	80	93,65580155
	90	96,91917615
	100	98,62275220
	110	99,43310042
	120	99,78510142
	130	99,92496372
	140	99,97586262
	150	99,99284590
	160	99,99804600
	170	99,99950810
	180	99,99988590
	190	99,99997560
	200	99,99999520

The figure 2 shows the theoretical particle size distribution of the blasting of quarry Herval obtained by the Rosin-Rammler model.

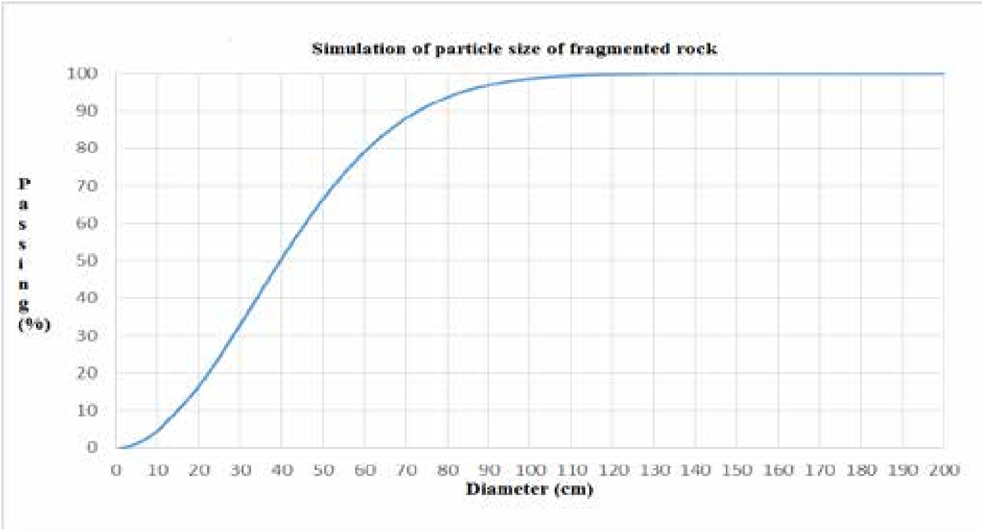

Figure 2 – Particle size curve simulation of rock blasting

Fragmentation analysis by image processing

For analysis of the fragmentation of the blasting of the quarry Herval by image processing, it was used the Split-Desktop® Version 3.1 program and photos of the blasting acquired in field stage. Split-Desktop program is a resource used in image processing to calculate the size distribution of rock fragments through digital image analysis in grayscale. Digital images can be acquired using digital camera.

After image acquisition, analysis of fragmentation from the blasting is carried out in five stages: opening of the images by the program; delineation of images; manual edition delineation to minimize errors; determination of images scale; analysis of the particle size of each image and blasting.

Were captured 5 images of the blasting, which were delineated by the Split-Desktop, making it possible to obtain their particle size distribution. After processing the photos of blasting it was combined the graphs through the own Split-Desktop and generated into a single graph for analysis of fragmentation. Figure 3 shows an example of the images captured for analysis of fragmentation along with their delineation by software, and figure 4 shows the size distribution curve obtained by combination of all graphs of blasting rocks analysis by image processing.

Figure 3 – Image captured for analysis and their delineation by Split-Desktop program

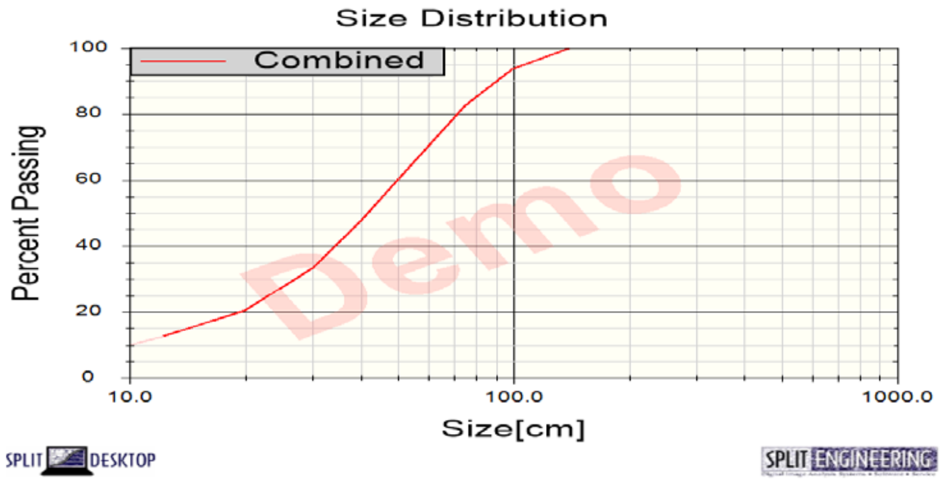

Figure 4 – General graph of particle size analysis of rock blasting for image processing

RESULTS AND DISCUSSIONS

This topic will be compared fragmentations obtained by simulating the Kuz-Ram model and through image processing by Split-Desktop software. To compare the size distribution curves of the simulations and results of imaging were analyzed three basic points of the curves:

- P_{50} - The diameter size through which 50% (fifty percent of the particles);
- P_{80} - The diameter size through which 80% (eighty percent of the particles);
- P_{100} - The maximum diameter through which 100% (one hundred percent of the particles).

The evaluation of these curves showed that the Kuz-Ram model features a higher percentage of fines. In the fraction above 10 cm, the analysis processing of image and model Kuz-Ram are equal. This can be demonstrated by plotting the two curves in the same simulation screen, as shown in figure 5.

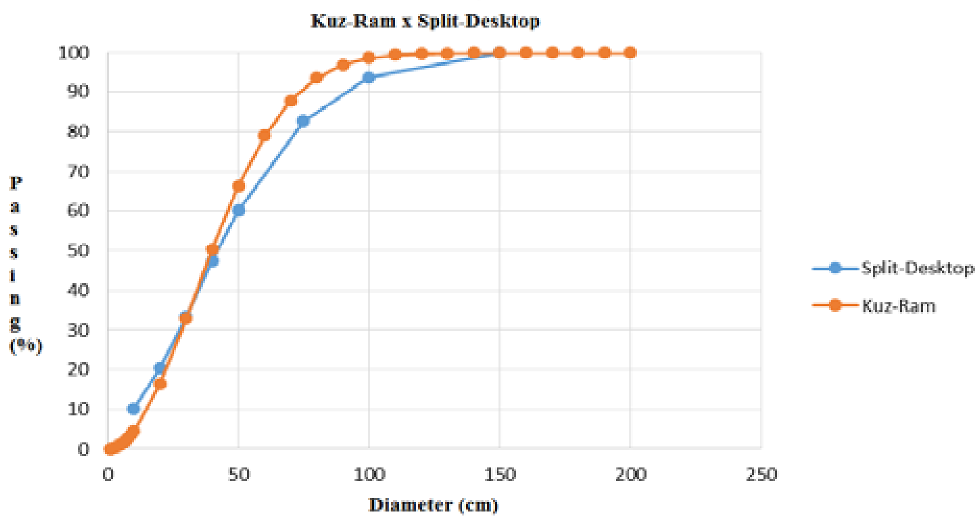

Figure 5 – Kuz-Ram model x Split-Desktop software to rocks blasting

With regard to the P_{80}, it may be noted that there is a small difference. This parameter obtained the value of approximately 60 cm for the simulation and the value of about 72.50 cm for size analysis by image. The discrepancy with the value by image analysis is approximately 17.24%.

In relation to P_{50}, there is a small discrepancy in relation to the simulated curve obtained by imaging of about 5.67% (five point sixty-seven percent), having a diameter of 42.15 centimeters to image analysis and 39,76 cm for the simulation through Kuz-Ram.

It may be noted that the slope in curves appear to be the same, since it is the same fire plan. Another common characteristic between the two curves is that they are in different size ranges. But a last observable characteristic curve of figure 5 is that it shows results for sizes smaller than 10 centimeters, unlike the simulated curve provides sizes in the range of 1 to 10 centimeters.

To the optimization of the blasting was adjusted the particle size so that P_{80}, that is, the particle diameter at which 80% pass (percent) of the material had value approximately less than or equal to 50 cm, which corresponds to the crusher feeding. Thus, have been adjusted the mesh of the fire plan (3 x 1.5) and the stem (50 cm), increasing the number of holes of 62 to 71 therefore increasing the cost of drilling and explosive but reduced costs with comminution (crushing and secondary blasting).

CONCLUSIONS

The role of rock blasting is to fragments as the proper size for crushing, transportation and all subsequent steps, therefore the mathematical modeling for rock blasting simulation is of paramount importance for the planning of operating costs, satisfactory efficiency and efficiency in mining operations and processing unit.

After performing of the particle size analysis by digital image processing with the Split-Desktop program, it was observed that the rock mass characteristics are essential to the best performance of the blasting operation. A more detailed study of mass, beside reduce transportation, handling and crushing costs, reduce the cycle time by optimizing these operations and increasing production.

The analysis by image processing played a fundamental role to assess the relationship between the blasting carried out and mathematical modeling of the rock mass, because it was possible to compare the result with the particle size distribution generated by simulation through the Kuz-Ram model, with the practical result of disassemble done.

It can be seen from simulation to optimization of blasting of quarry Herval that the increase of mesh perforation contributes to disassemble with large particle sizes, although reducing the amount of explosive in bench (loading ratio). Other operational inconvenience are: increased wear of crushing and loading equipments, higher maintenance time, and higher fuel consumption.

It can be said that the Kuz-Ram fragmentation model assisted by simulation shows efficiency in predicting the rock fragmentation using explosives. Furthermore, the model has limitations, such as:

• Overestimation of the amount of fines;
• The model doesn't predicts the outcome of the particle size of rock fragmentation for different types of mesh, and it is known that the type of mesh applied influence the fragmentation blasting;
• Another important factor that must be considered is that there is no prediction or modeling to the way the boot sequence, the maximum load expected nor influence the use of different time delay, both to disassemble controlled electronic fuze as to not electric.

Despite the limitations of both the model and the simulator, this tool proves important in predicting blasting aiming to obtain a suitable drilling mesh in order to maintain a normal operating level and the desired degree of fragmentation to disassemble.

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STOCHASTIC SIMULATION AS A TOOL FOR OPTIMIZING THE ALLOCATION OF MINING TRUCKS

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STOCHASTIC SIMULATION AS A TOOL FOR OPTIMIZING THE ALLOCATION OF MINING TRUCKS

ABSTRACT

Traditional methods correlate the ideal number of trucks to the ratio between haulage cycle time and loading cycle time. In these cases, it is common to use correction factors to approach the theoretical model to the actual condition of the mine. However, in a real situation, the many imperfections on each step of the production cycle result in delays such as trucks having to wait in queues, slow moving trucks getting in the way of faster trucks, and idleness of loader/excavators. These delays are a direct result of the variations on unitary operations and the number of equipment available, but they are very difficult to predict by deterministic modeling. Moreover, frequent changes on these cycle times make any modeling based on the assumption of a static scenario with little practical use on real time optimization. Dynamic allocation algorithms are common optimization tools along with mining management systems, and their details are industrial secrets. These algorithms use many different techniques and are commonly based on linear programming and network flows. They are usually executed in two steps: measurement of input parameters using data collected by the mine management system itself and execution of the optimization algorithm. This process can be run cyclically, resulting in a fast redistribution of the fleet in case the input parameters change. The simulation methodology developed in this work allows for the stochastic prediction of the productivity curve with the use of intensive computational calculation, eliminating the need for adjustment factors since it is possible to predetermine the interactions of the many truck allocation scenarios and the effects of their imperfections on cycle times. With an operation of many loaders/excavators, it is possible to allocate the trucks on the fronts incrementally, where it will become more productive. Therefore, it was then possible to predict a scenario with up to 10 excavators and up to 200 trucks. The methodology can be scaled up, allocating the trucks based on the incremental gain it would bring on each excavator/loader.

KEYWORDS

optimization, trucks allocation, dispatch, stochastic, simulation, open-pit mining, mine productivity

INTRODUCTION

For decades, since the introduction of computers to allocate mining trucks for maximum productivity, the main limiting factors have been the low precision of tracking systems and actual computational capacity. As optimization models grow in complexity, different approaches were proposed and can be generally categorized as follows (Rodrigues, 2006).

Linear and Dynamic Programming Methods

Linear programming methods are generally based on Dantzig's (1987) simplex models. The goal is to evaluate the best route a truck can take to minimize wait times.

Although fundamentally similar, dynamic programming differs by the capacity to do such evaluations on a continuous or nearly continuous manner, meaning that if something changes on one of the cycles, the algorithm can then quickly adapt to the new scenario. In theory, there is no necessity for a truck to be limited to a single excavator. The trucks can be treated as a resource pool and the mine management system can dispatch each trip for the best route available each time.

Genetic Algorithms

Still not widespread in the mining industry, the techniques behind genetic algorithms are well known to be capable of good results in optimization of haul trucks allocation (Burt, 2008).

The main parameters for the chromosome cost function are queue times, the total production, and ore quality. Choosing the most adaptable individuals and mutating generation after generation is a very computationally intensive task, and for most situations, real time results require the whole model to start from scratch in small time intervals, making it very difficult to ensure good results in all situations.

Heuristic Based Methods

These methods rank the priorities (Rodrigues & Pinto, 2012) based on definitions that are usually external to the algorithm and use this ranking to generate scenarios, choosing between them based on what is more important to the operation (Lizotte & Bonates, 1986). Variables are usually quality, rhythm of the operation, queue times, and cycle times.

The Stochastic Approach

By modeling the mine operation using a deterministic model, optimization is more straightforward and the results are perfectly reproducible. On the other hand, these models only mirror a theoretical situation and do not reproduce accurately the real life operation, requiring multipliers to calibrate the model to approximate real life scenarios.

A stochastic model considers the uncertainty as part of the operation and models it accordingly, building a probabilistic simulation in which the variables are random, but the end result is consistent.

SIMULATING THE MINE

The stochastic simulation approach proposes the creation of a general model of the mine operation based on the main subdivisions of the trucks' cycle times, or unitary operations, and its corresponding statistical variances.

Haul trucks' cycle times will generally break down into:

- Moving while empty;
- Queue time at loader/excavator;
- Spot time at loader/excavator;
- Loading time;
- Moving while full;
- Queue time at dump site;
- Spot time at dump site; and
- Dumping time.

Also, movements while the trucks are empty or full can be further broken down into their average speed and average distance.

The picture below shows a simplified scheme of a haulage cycle.

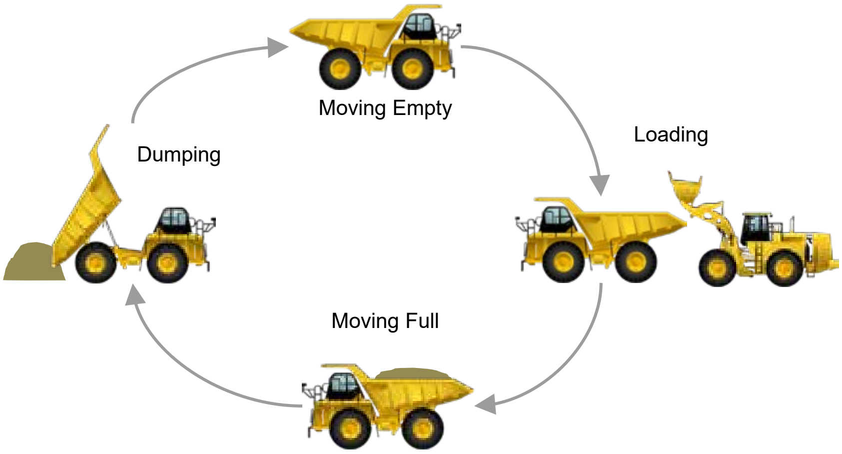


Figure 1 – Basic unitary operations adding to a complete cycle time.

Assuming a cycle time being the result of the combined cycle subdivisions, we have that:

$$CT = ME + QL + SL + LT + MF + QD + SD + DT$$

Where, CT is cycle time, ME is time moving while empty, QL is time while on queue at the loader, SL is spot time at the loader (counting from the time the truck leaves the head of the queue until it is properly positioned to begin loading), LT is loading time, MF is time moving while full, QD is time while on queue at dump site, SD is spot time at dumping site, and DT is dumping time.

This equality can also be described as:

$$CT = \frac{AHD}{AES} + QL + SL + LT + \frac{AHD}{AFS} + QD + SD + DT$$

Where, AHD is average haulage distance, AES is average speed while empty, and AFS is average speed while full.

Queuing as a consequence of cycle imperfections

While part of the haulage cycle, queues are not an actual part of the necessary work. They are simply a consequence of the imperfections of the cycle time.

Assuming a hypothetical cycle, shown in Figure 2, there are four unitary operations, each with a five minute time, adding to a twenty minute cycle time. If those times where perfectly constant, the ideal number of trucks on this cycle could be described by:

$$NT = \frac{CT}{LT}$$

Where, NT is the number of trucks.

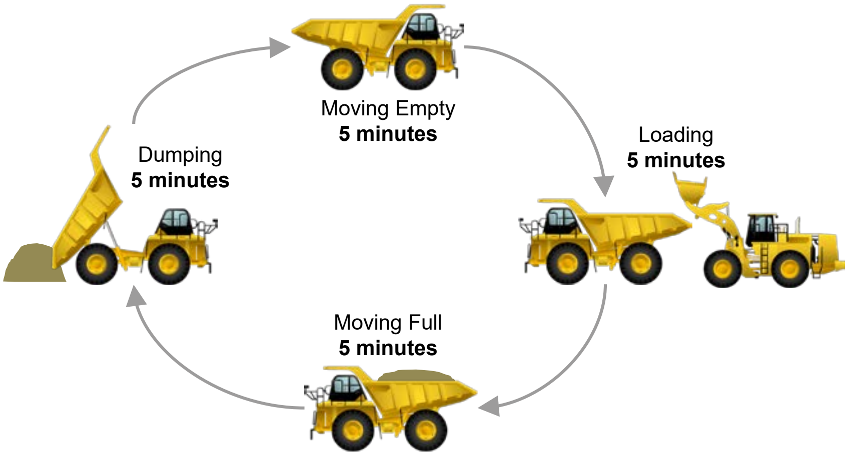


Figure 2 – Cycle time with hypothetical 5 minute unitary operations

While true in a hypothetical scenario, the simplified equation to calculate the number of trucks is no longer true when the many different variables that come into play in a real life operation are factored in. Each of the unitary operations is subject to variability that will result in a truck's cycle being impacted by the cycle of the other trucks. These imperfections will result in both queues and idleness.

When working with a stochastic simulation model, it is possible to completely assume that queues are a consequence of unitary operations variances rather than an actual part of the cycle that has to be accounted for beforehand.

Mine Management Systems as Tool for Reliability

Being able to clearly calculate the averages and standard deviations of each unitary operation is the key to successfully creating a reliable model of the mine.

An inherent requirement of any stochastic modeling is the data being adherent to a known statistical probability distribution. For all samples studied on Gerdau case study regarding unitary operations in mining, the Gaussian distribution has shown adequate fit. The single most important factor is the quality of data acquired from the current mine management system, which monitors all cycle times, trucks speeds, and average distances.

When modeling, it is important to define beforehand how outliers will be handled. For this case study, exceptional situations that impacted cycle times beyond what could be considered a normal work day were excluded from the data, as they add no significant information for real time decisions on a regular work day, but could affect the variance in ways that do not faithfully represent the real life operational conditions. So, to ensure a reliable model, it is important to have proper data acquisition with a reliable mine management system, but also, some degree of human judgment should be applied, whether by handpicking the out of ordinary or by choosing to apply a statistical technique to filter out these outliers.

Logarithmic regression allowing for a simpler maximization algorithm

Different from most approaches that aim to reduce queue times (Ercelebi & Bascetin, 2009; Van Tol & AbouRizk, 2006; Ta, et al., 2013), the proposed method is targeted toward choosing the best excavator/loader to allocate each truck in order to maximize their total production.

In order to achieve this goal, for each different possible cycle, twenty scenarios ranging from one truck to twenty trucks in the cycle were simulated, and the total mine production was calculated in each of these scenarios. The simulation had a different random number generated for each unitary operation on each step of the simulation. The random numbers were adjusted to the typical probability distribution and added to the average value of each operation. The result is that each step of the cycle had an independent value, statistically within the expected values seen in a real life operation.

This allows for the creation of the simulated production curve for each excavator/loader. One example is shown in Table 1:

Table 1 – Simulated production versus number of trucks.

Excavator 1				Excavator 2			
Trucks	Production (day)	UT Trucks	UT Excavator	Trucks	Production (day)	UT Trucks	UT Excavator
1	2.097,70	81,71%	17,47%	1	1.433,15	81,18%	11,98%
2	4.259,28	80,44%	35,83%	2	2.862,64	79,07%	24,14%
3	5.842,39	76,50%	49,38%	3	4.016,08	73,98%	32,93%
4	7.638,44	74,13%	63,09%	4	5.510,75	74,98%	45,76%
5	8.880,82	69,07%	76,23%	5	6.578,74	71,49%	54,27%
6	9.932,09	62,98%	81,03%	6	7.427,04	68,41%	61,68%
7	10.108,71	55,59%	83,70%	7	8.633,88	67,30%	72,17%
8	10.391,45	49,41%	86,03%	8	8.887,88	61,00%	74,82%
9	10.421,28	43,92%	86,54%	9	9.706,58	58,63%	81,21%
10	10.523,05	40,06%	88,31%	10	9.781,32	52,38%	81,55%
11	10.623,79	36,53%	88,70%	11	10.274,96	49,67%	86,07%
12	10.941,08	33,54%	90,03%	12	10.634,74	46,74%	86,90%
13	10.921,50	31,86%	91,53%	13	10.414,73	42,80%	86,43%
14	11.145,77	29,40%	92,32%	14	10.594,04	39,81%	88,47%
15	11.271,00	16,20%	93,64%	15	10.790,79	38,07%	89,09%
16	11.349,99	25,89%	93,74%	16	11.024,06	36,28%	91,46%
17	11.477,06	24,53%	96,11%	17	11.249,82	34,38%	92,66%
18	11.498,10	23,33%	95,75%	18	11.098,16	31,68%	92,10%
19	11.639,91	22,12%	97,49%	19	11.283,76	31,02%	94,20%
20	11.552,17	20,65%	95,70%	20	11.396,05	29,57%	95,04%

Being a stochastic simulation, every time it is run, a different result will emerge. However the results are consistent with a logarithm curve, represented by a function of the form:

$$y = a + b \times \ln(x)$$

Where, coefficients a and b can be calculated using widely known algorithms such as the least squares fitting method (Weisstein, 2002):

$$b = \frac{n\sum_{i=1}^{n}(y_i \ln x_i) - \sum_{i=1}^{n} y_i \sum_{i=1}^{n} \ln x_i}{n\sum_{i=1}^{n}(\ln x_i)^2 - \left(\sum_{i=1}^{n} \ln x_i\right)^2}$$

$$a = \frac{\sum_{i=1}^{n} y_i - b\sum_{i=1}^{n}(\ln x_i)}{n}$$

By fitting the data, it is possible to smooth out the naturally occurring dispersion and clearly define the rate in which each incremental truck adds up to the total production, as shown in Figure 3.

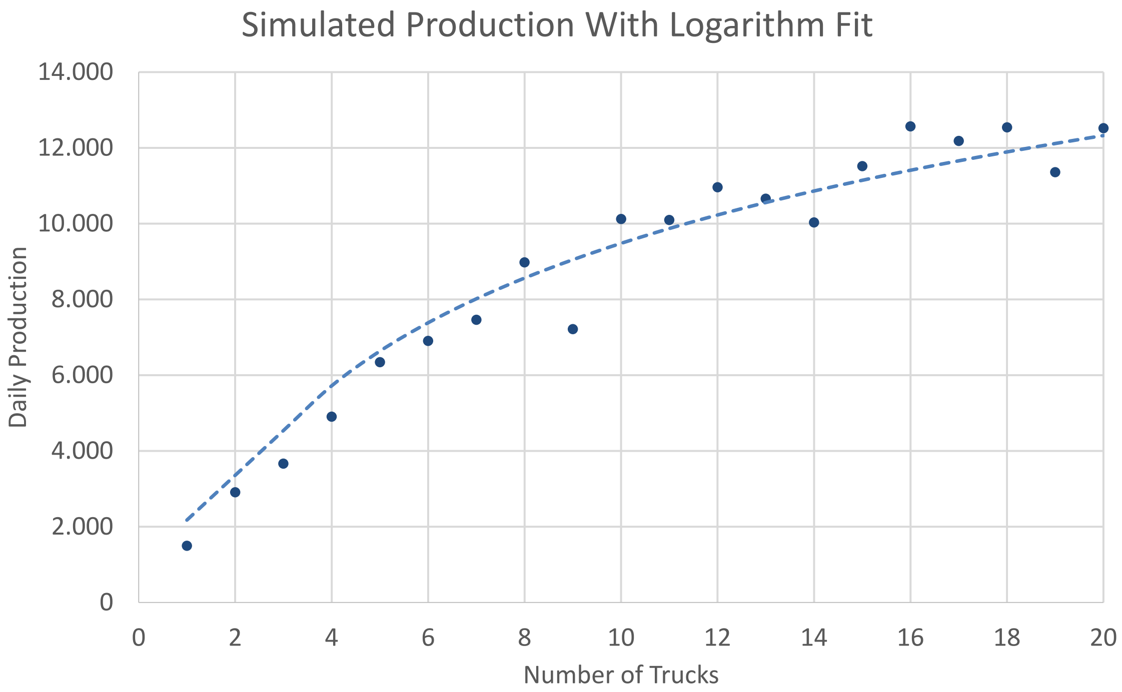


Figure 3 – Logarithm fit by least squares method

Each truck available is then tested to see which excavator or loader will bring in the most production if the truck is added to its cycle. This process, shown in Table 1, is repeated until all trucks are allocated or a stop criterion is met.

Table 2 – Decision table for trucks' allocation

Truck Number	Incremental Production Excavator 1	Excavator 2	Excavator 3	Excavator 4	Chosen Excavator
1	909,04	825,58	793,90	793,86	1
2	705,10	825,58	793,90	793,86	2
3	705,10	674,55	793,90	793,86	3
4	705,10	674,55	648,66	793,86	4
5	705,10	674,55	648,66	615,77	1
6	576,11	674,55	648,66	615,77	2
7	576,11	570,32	648,66	615,77	3
8	576,11	570,32	548,44	615,77	4
9	576,11	570,32	548,44	503,12	1
10	487,10	570,32	548,44	503,12	2
11	487,10	494,04	548,44	503,12	3
12	487,10	494,04	475,08	503,12	4
13	487,10	494,04	475,08	425,38	2

Simulation of Gerdau Várzea do Lopes Operation

Gerdau's Várzea do Lopes mine is an open pit, iron ore operation located in Minas Gerais, Brazil. Excavation is achieved by excavators, and haulage is completed using trucks.

The current operation consists of eight excavators and 50 trucks, moving hematite, itabirite, and waste over average haulage distances ranging from 1km to 4km. Cycle times are measured using Sodep Minetrack system, and the data gathered is validated automatically when parameters of geolocation and measured times fall within expected values ranges. Whenever automatic validation fails, the cycle is validated manually by a human operator.

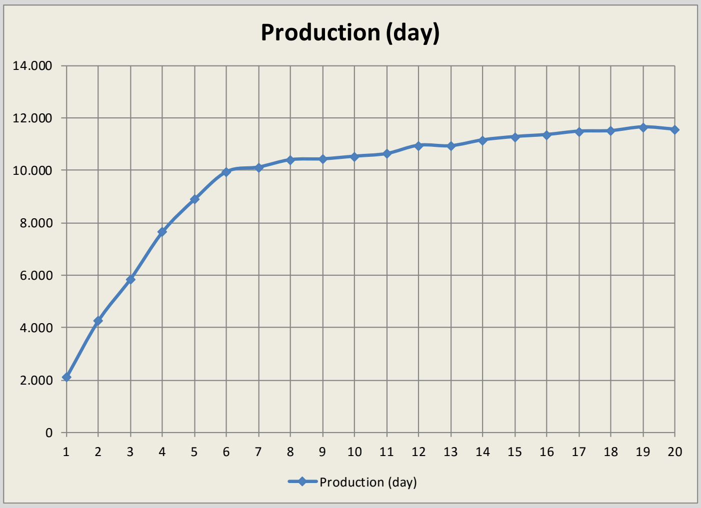
Six basic rules apply to this particular model:

I. Trucks can’t overtake others;
II. Speed limit is 40 km/h;
III. Truck queues will follow the FIFO (first in, first out) logic;
IV. On each run, a whole day of operation will be simulated;
V. Trucks will stop three times a day at regular intervals, during mid-shift, for 1 hour meals; and
VI. Trucks will stop three times a day for shift changes at regular intervals.

Results are shown in Figure 4:

SIMULATION RESULTS

Excavator 1			
Trucks	Production (day)	UT (Trucks)	UT (Excav)
0	0	0,00%	0,00%
1	2.098	81,71%	17,47%
2	4.259	80,44%	35,83%
3	5.842	76,50%	49,38%
4	7.638	74,13%	63,09%
5	8.881	69,07%	76,23%
6	9.932	62,98%	81,03%
7	10.109	55,59%	83,70%
8	10.391	49,41%	86,03%
9	10.421	43,92%	86,54%
10	10.523	40,06%	88,31%
11	10.624	36,53%	88,70%
12	10.941	33,54%	90,03%
13	10.921	31,86%	91,53%
14	11.146	29,40%	92,32%
15	11.271	16,20%	93,64%
16	11.350	25,89%	93,74%
17	11.477	24,53%	96,11%
18	11.498	23,33%	95,75%
19	11.640	22,12%	97,49%
20	11.552	20,65%	95,70%



		ROM	Waste
Daily Production (With 45000 t Target) (t)		45.662	18.672
Number of Trucks Allocated	Excavator 1	5	0
	Excavator 2	5	0
	Excavator 3	5	0
	Excavator 4	4	0
	Excavator 5	5	0
	Excavator 6	5	0
	Excavator 7	0	11
	Excavator 8	0	10
	Excavator 9	0	0
	Excavator 10	0	0

		ROM	Waste
Maximized Production (t)		53.302	15.108
Number of Trucks Allocated	Excavator 1	6	0
	Excavator 2	7	0
	Excavator 3	6	0
	Excavator 4	5	0
	Excavator 5	6	0
	Excavator 6	7	0
	Excavator 7	0	7
	Excavator 8	0	6
	Excavator 9	0	0
	Excavator 10	0	0

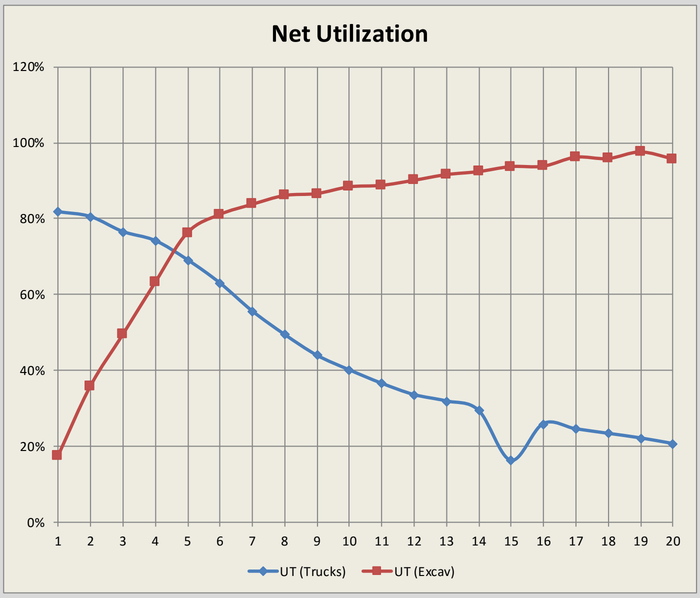


Figure 4 – Results of simulation for Várzea do Lopes mine

CONCLUSION

While there are many different possible approaches to the optimization of mining trucks allocation, most methods focus on keeping the trucks and excavators/loaders as busy as possible rather than understanding ahead of time how each new allocation will impact the total production

Stochastic simulation, although computer intensive, has simpler implementation and allows for the prediction of the cycle behavior as well as the total production with each different scenario of trucks' allocation. It can successfully emulate the optimization results generally expected from a deterministic model, with the added benefit that it doesn't rely on frequent calibrations to approximate the theoretical model to the actual operation

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THE CHALLENGES OF THE MANAGEMENT OF THE SURFACE MINING EQUIPMENT

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THE CHALLENGES OF THE MANAGEMENT OF THE SURFACE MINING EQUIPMENT

ABSTRACT

This study describes a survey of the drilling, loading and hauling equipment of a surface mine, using as an example a mine located in the North of Brazil. We discuss here actions that can be taken to increase the index on current conditions of the mine work. The study analysis the productivity of each equipment looking for the day of the week and shift. Finally, it is concluded that the mine management acting through different concurrently actions, however small ones, it is possible enhance safety and improve the mining process, reaching productivity gains above 20%.

KEYWORDS

Surface Mining, Mining equipment, Equipment Management

INTRODUCTION

The mining operation is focused on ore production more safe, efficient and profitable as possible. This focus was crucial to that in recent years the mining companies to invest in improving the level of automation, and this fact is changing the way mines are operated and managed (Dey, 2005; Hall et. al., 2000; and Burger, 2006). One of the biggest mining challenges remains the improvement of the utilization index of operation equipment.

The equipment utilization can be expressed by the ratio between productive and non-productive hours during a given period (Lees, 2003). This relationship is associated with a number of internal and external factors of the mining operation, such as the maintenance strategy, company managerial and administrative policies, and its effects on the mining working conditions. In the mining the unit operations of drilling, loading and hauling are the production bottleneck of many mines. Automation system can provide control information that can be used to analysis the production and make management choices (Jifei et al., 1997; Automation Resources, 2010).

This paper presents a survey of the utilization and production index of drilling, loading and hauling of a surface mining located in the North of Brazil and discuss actions that can be taken to increase the index on current conditions of the mine work. The study analysis the productivity of each equipment looking for the day of the week and shift is innovative because analysis productivity coefficients, the days of rotation between shifts and the week, allowing that this methodology to be used in any mine.

CASE STUDY

This study presents a survey of the equipment availability and production of drillers, loaders and trucks. The mine is located at Midwest of Brazil of large size with two surface mines. The mine has four types of drills and two types of trucks. The data used are from an actual mine, but was used a multiplier factor to maintain the confidentiality of the original data.

Twenty-two months (January to December of 2014 and January to October of 2015) were analyzed totaling over 7 million meters drilled and 67 million tons of ROM (multiplier factor applied). The following shows the data relating to mining.

Figure 1 and Figure 2 shows the meters drilled per driller and the tons transported in the mine per truck type. They show considerable stability of mining production during the period studied, showing that every month are ideal for analysis.

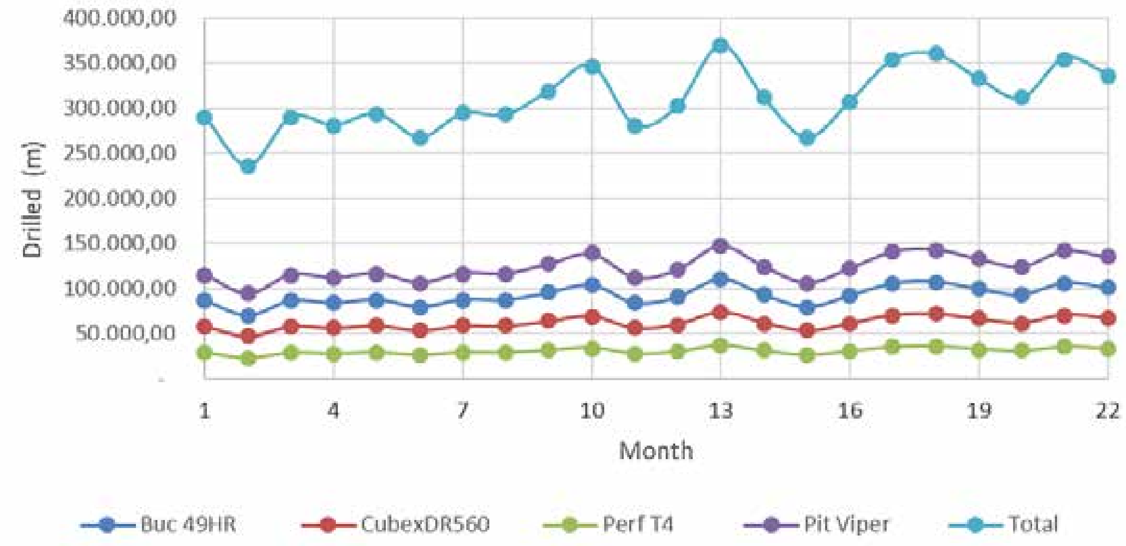

Figure 1 – Meters drilled per driller and total

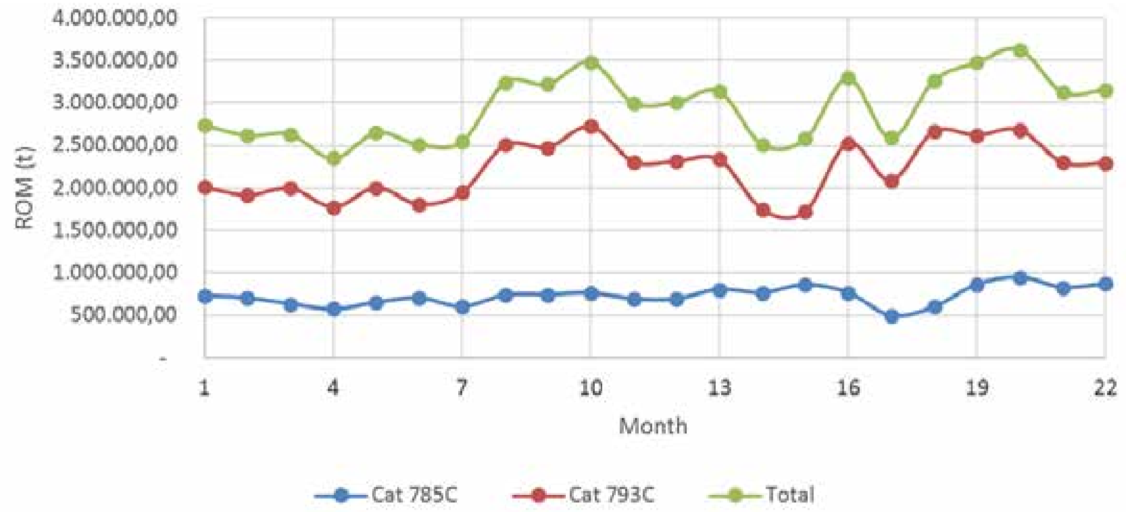

Figure 2 – ROM per type of truck and total

The mine works with four groups (A, B, C and D) that perform daily carvery. In a cycle of 28 days each group works in a 7-1-4-1-3-3-3-1-4-1 schedule (7 days in Morning, 1 day Off, 4 days in Afternoon, 1 day Off, 3 days in Night, 3 days Off, 3 days in Afternoon, 1 day Off, 4 days in Night and 1 day Off).

Table 1- Shift schedule

Day	Shiftt 1 Morning	Shift 2 Afternoon	Shift 3 Night
1	A	B	C
2	A	D	C
3	A	D	B
4	A	D	B
5	A	D	B
6	A	C	B
7	A	C	D
8	B	C	D
9	B	A	D
10	B	A	C
11	B	A	C
12	B	A	C
13	B	D	C
14	B	D	A
15	C	D	A
16	C	B	A
17	C	B	D
18	C	B	D
19	C	B	D
20	C	A	D
21	C	A	B
22	D	A	B
23	D	C	B
24	D	C	A
25	D	C	A
26	D	C	A
27	D	B	A
28	D	B	C

The meters drilled and ROM were analyzed considering the shift schedule day, group, shift and day of the week.

RESULTS

The results are divide in:

- Production per shift: ROM and drilled;
- Production per group: ROM and drilled;
- Production per shift schedule day: ROM and drilled;
- Production per weekday: ROM and drilled;
- Production and physical availability.

Production per shift

Table 2, Table 3, Table 4 and Table 5 show the production (transported material and meters drilled) per shift. The Shift 3 Night works 6 hours, different from Shift 1 Morning and 2 Afternoon that work 8 hours. As aspect, the relative production of transported material was worst in the Shift 3 Night. The driller production was less affected, showing in 2014 a relative hours production around 100% for the tree shifts.

Table 2- ROM Shift production - 2014

Shift	average ROM per shift (t)	relative shift production	average ROM per hour (t)	relative hours production
Shiftt 1 Morning	17184,68	111%	2148,08	**102,2%**
Shift 3 Night	12142,77	79%	2023,80	**96,3%**
Shift 2 Afternoon	17071,57	110%	2133,95	**101,5%**
Total	15467,07		2101,94	

Table 3- ROM Shift production - 2015

Shift	average ROM per shift (t)	relative shift production	average ROM per hour (t)	relative hours production
Shiftt 1 Morning	16845,04	112%	2105,63	**102,9%**
Shift 3 Night	11694,97	78%	1949,16	**95,3%**
Shift 2 Afternoon	16652,19	111%	2081,52	**101,8%**
Total	15069,74		2045,44	

Table 4- Drilled Shift production - 2014

Shift	average drilled per shift (m)	relative shift production	average drilled per hour (m)	relative hours production
Shiftt 1 Morning	880,15	110%	110,02	**101,0%**
Shift 3 Night	653,93	82%	108,99	**100,0%**
Shift 2 Afternoon	862,70	108%	107,84	**99,0%**
Total	798,93		108,95	

Table 5- Drilled Shift production - 2015

Shift	averagedrilled per shift (m)	relative shift production	average drilled per hour (m)	relative hours production
Shiftt 1 Morning	995,42	109%	124,43	**100,1%**
Shift 3 Night	725,14	79%	120,86	**97,2%**
Shift 2 Afternoon	1021,40	112%	127,67	**102,7%**
Total	913,87		124,32	

Production per group

Table 6, Table 7, Table 8 and Table 9 show the production (transported material and meters drilled) per group and shift. The drilled production shows a systemic problem at Group C in 2014 and 2015.

Table 6- ROM Group production - 2015

Group	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Group A	94%	99%	101%	**98%**
Group B	102%	104%	110%	**106%**
Group C	104%	99%	94%	**99%**
Group D	100%	98%	95%	**98%**

Table 7- ROM Group production - 2014

Group	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Group A	97%	103%	101%	**101%**
Group B	95%	106%	102%	**100%**
Group C	101%	95%	94%	**96%**
Group D	107%	96%	103%	**103%**

Table 8- Drilled Group production - 2015

Group	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Group A	98%	102%	95%	**98%**
Group B	108%	102%	112%	**108%**
Group C	89%	89%	93%	**90%**
Group D	104%	108%	100%	**104%**

Table 9- Drilled Group production - 2014

Group	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Group A	93%	93%	92%	**93%**
Group B	99%	96%	105%	**100%**
Group C	85%	94%	90%	**89%**
Group D	124%	117%	113%	**118%**

Production per shift schedule day

The production considering the shift schedule day showed no relevance influence in the production. Cremonese and Tomi (2012) showed that the shift schedule day affect the production, in a schedule where the worker needs to change the shift without a day off. In the case study schedule there are days off between the change of shifts showing that this schedule can get a more stable production.

Production per Weekday

Table 10, Table 11, Table 12 and Table 13 show the production (transported material and meters drilled) per day of week. The transported and drilled production show a systemic difference between the weekends and the middle of the week.

Table 10- ROM production per weekday- 2015

Weekday	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Sunday	112%	107%	110%	**110%**
Monday	100%	101%	101%	**101%**
Tuesday	96%	98%	88%	**93%**
Wednesday	93%	95%	98%	**95%**
Thursday	97%	104%	94%	**98%**
Friday	92%	93%	97%	**94%**
Saturday	111%	103%	111%	**109%**

Table 11- ROM production per weekday- 2014

Weekday	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Sunday	112%	98%	116%	**110%**
Monday	106%	105%	104%	**105%**
Tuesday	95%	102%	89%	**95%**
Wednesday	101%	102%	97%	**99%**
Thursday	91%	97%	95%	**94%**
Friday	92%	99%	91%	**93%**
Saturday	103%	98%	107%	**103%**

Table 12- Drilled production per weekday- 2015

Week day	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Sunday	114%	102%	120%	**113%**
Monday	99%	103%	99%	**100%**
Tuesday	95%	103%	91%	**96%**
Wednesday	96%	94%	92%	**94%**
Thursday	94%	97%	98%	**96%**
Friday	97%	97%	96%	**97%**
Saturday	105%	102%	104%	**104%**

Table 13- Drilled production per weekday- 2014

Week day	Shiftt 1 Morning	Shift 3 Night	Shift 2 Afternoon	Total
Sunday	106%	107%	100%	**104%**
Monday	90%	100%	88%	**92%**
Tuesday	94%	91%	94%	**93%**
Wednesday	96%	96%	104%	**99%**
Thursday	98%	97%	93%	**96%**
Friday	101%	110%	106%	**105%**
Saturday	114%	100%	114%	**110%**

Figure 3 shows the systemic influence in the Sunday and Tuesday production.

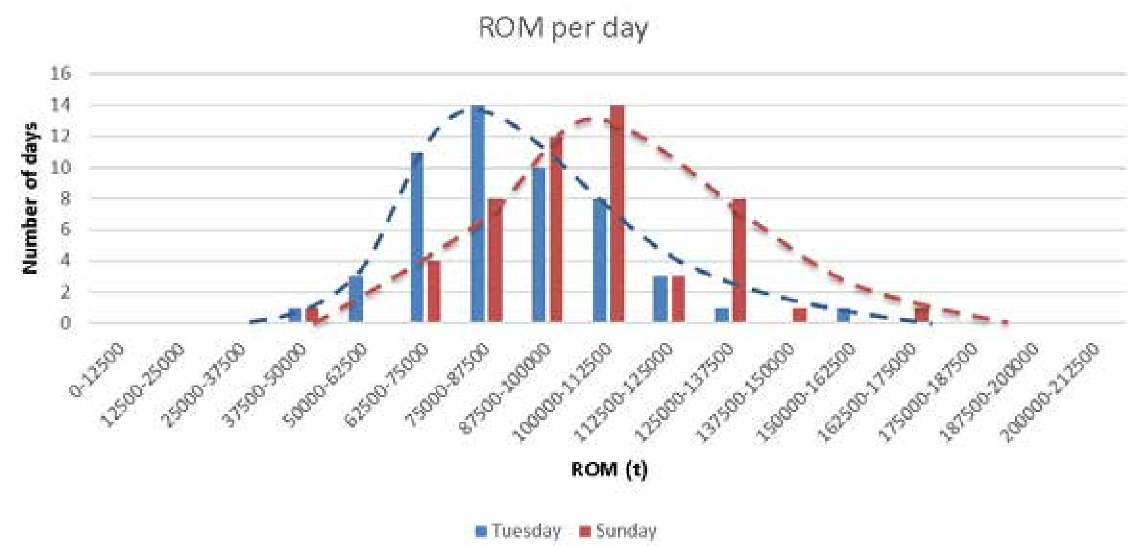

Figure 3 – ROM per day of week

Production and physical availability

Table 14 shows the production and physical availability. It shows an influence in the production, but not the only factor that affects the production.

Table 14- Production and physical availability

Weekday	physical availability	relative physical availability	Production/relatie physical availability
Sunday	64%	104%	**106%**
Monday	62%	101%	**100%**
Tuesday	61%	98%	**95%**
Wednesday	60%	98%	**98%**
Thursday	61%	98%	**100%**
Friday	61%	98%	**96%**
Saturday	64%	104%	**105%**

DISCUSSION

The variation between the middle of week and weekend in transported production is of 18% and in meters drilled is of 20%. The systemic difference of drilled production by group was of 19% and 27% in 2014 and 2015. If a continuous analysis was done at that time, some measures could be made to the critical points and increase the production. One of the measures could be the evaluation of the causes that were decreasing the production of one group. Another one would be the evaluation of work in the weekend to analyze if the equipment are being wrong operated and breaking it in the middle of the week, generating a decrease in the physical availability.

CONCLUSION

To meet the challenges of improving the mining process control, it is necessary that the mine management act through various concomitantly measure. With the control and analysis if the production it is possible to act in determined group or shift that is production less than expected or breaking the equipment.

As examples in this case study, the Group C could be manage to achieve the same production as the other ones.

The mine management acting through different concurrently actions, however small ones, it is possible enhance safety and improve the mining process, reaching productivity gains above 20%. Some of the action are the homogenization of the mine production per weekdays as a result of actions in physical availability and group management.

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THE LOADING AND HAULING SIMULATION OF ORE OPERATION OF A BAUXITE MINE IN WESTERN PARÁ, BRAZIL

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THE LOADING AND HAULING SIMULATION OF ORE OPERATION OF A BAUXITE MINE IN WESTERN PARÁ, BRAZIL.

ABSTRACT

This assignment focuses on mining simulation of the loading and hauling ore operation of a bauxite mine in Juruti, city of Western Pará, Brazil. In surface mining operations, loading and mainly hauling are among the largest of the operating costs. In order to reduce this cost, it is necessary to increase productivity and efficiency of these operations. The higher productivity is possible through to determine the best configuration to operate this strip mine by simulation. Variables such as material features, loading time, average haul distance among others are analysed in the simulation, considering circumstantial and economic limitations, for instance available machinery. Through the mine simulation, it is possible to establish the mine configuration that provides the highest productivity by evaluating results of cycle time, from the loading at the mine strip to the ore dumping on the crusher. This paper propounds to define the optimal number of dump trucks in terms of productivity. This assay also identifies the hauling cycle as the most influential parameter for this ore exploitation as it determines the biggest mine obstacle regarding to productivity. The variables that master these ore operations are evaluated using simulation outcomes in comparison to day-by-day mine reality by means of operation time, productivity and numbers of trucks that end its cycle entirely. The optimal truck fleet presents real productivity gain, which is confirmed through comparative tests in field of loading and hauling operations that prove the simulation efficiency. It is possible to observe a gain up to 18,56 tonnes of exploited bauxite per hour, which means 445,44 tonnes of extra ore production per day.

Keywords: strip Mining, Bauxite, productivity, comparative tests, mine simulation.

INTRODUCTION

Initially, Initially, The rock breakage or ground excavation carry out rock material decompression. Later, loading and hauling operations utilize shovels and trucks, these operations consider the shovel load a truck tray entirely up to the dumping at the hopper crusher, including the way back of the truck at the loading place (Ricardo e Catalani, 2007). The hauling operation is the most expensive operation in a open pit mine, it represents about 50 to 60% of operational costs (Hartman, 1992).

The most utilized operational research technic is simulation, mainly because of the capacity increase of computer processer and software for the last decades (Martins, 2013). Even though Brazil mining is slightly introducing the simulation and its benefits, national mines already use this tool for all decision levels in a mine argues Rodrigues (2006).

The stochastic simulation provides analyses of a mine dynamic as loading and hauling ore operation (TALPAC, 2010). This Article presents a simulation carried out by Talpac® programme, which applies the stochastic simulation in order to productivity estimation among other factors which influences on the ideal tuck fleet size to the mine configuration. The mine simulation system allows configuration changes, in other words, it is possible evaluate the same mine under different parameters, which provides distinct operational scenarios to choose the best possible configuration of the mine or the most adequate to the mine. Therefore, this article objectives to determine the most adequate truck fleet size, which permits the best productivity and efficiency, considering same limitations such as the existent mine machinery in the bauxite mine in Juruti, the town which contains the mine area, western of Pará.

THE BAUXITE MINE

Juruti mine site activities are as follows: vegetation suppression; overburden removal; mechanical rock breakage; hauling and loading ore operation; mineral process, comminution and concentration as examples; tailing deposition and reforestation. Figure (1) shows this mine cycle, with only a different: the rock breakage performed by tractors (D11) instead of dragline.

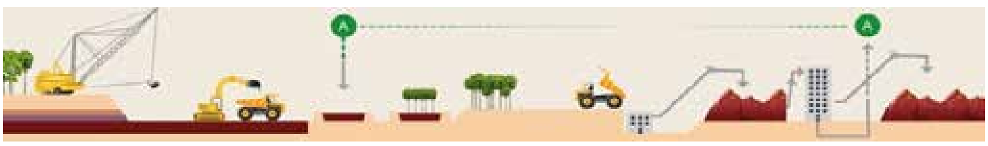


Figure (1) typical mining operation, bauxite strip mine in Amazonia. (MRN, 2011).

In the Juruti mine, the applied loading uses backhoe hydraulic excavators (backhoe shovels), which load the truck tray when the arm shovel is exactly at the back of the truck and they are both at the same ground level, as Figure (2) illustrates. This loading configuration permits high selectivity rate, low shovel cycle times and with swing angles of 90 degrees, generating good flexibility and offer more security during the raining time. It is interesting to highlight that Juruti mine is in the Brazilian rain forest, Amazonia.

Figure (2) truck-shovel loading with swing angle of 90 degrees.

After completely fulfil of the truck tray with ore, the truck goes to the dump area. However, several times a day some trucks need to go to the road scale in order to measure the amount of ore to know how much ore it is taking to the crusher, which is a kind of predictable event. On the other hand, some unavoidable situation may delay the hauling operation, which means to break the truck cycle unpredictably, influencing truck fleet cycle at some periods of the day. For instance, shovel refuelling that is done at any time as long as it is infect necessary in the mining face by the refuelling and lube truck or even truck refuelling, which have an exclusive period per shift to refuelling at the petrol station. In the dump area, the procedures are as follows: the trucks stop by Indian row, if there are trucks waiting to dump. The maximum of two trucks can dump at the two district sides of the hopper crusher, at the same time as Figure (3) Illustrates. The average dumping time is extremely favourable to the truck cycle time, which has the value of 50 seconds. In normal day of operations, after dumping the truck might go back to the same mining strip, actually, it depends on the dispatch system that can order this truck to go to another mining strip, which is performed by radio communication. In order to carry out the mine simulation, there was a change, which is to maintain the same truck fleet size keep going to the same mining strip instead of swapping its destination.

METHODOLOGY

It is essential to define several critical variable in order to simulate loading and hauling ore operations; they must be applied as input of the Talpac® software. Most of collected data came from the field analyses. After set up those critical variables, this software compiles and generates results from algorithms and stochastic simulation. Considering the effects of excess or deficit of the number of trucks, along with how it influences on the productivity, is possible to evaluate the different scenarios of loading and hauling operation and select the most suitable operational configuration.

The objective of this case study is to define the most adequate truck fleet size for the two distinct mining strips in the same work face. Since the dimensioning of the truck fleet size directly affects mine productivity.

DATA COLLECTION

The data gathering and allocation of the pertinent variables of this bauxite mine, it was essential to develop the mine simulation. These important variables are material characteristics, operation time per shift, specific parameters of shovels, configuration of transport path. They are extremely important to estimate the productivity and the truck cycle time.

Material features

The most relevant feature of the ore are the swelling factor and the fill factor regarding loading and hauling. This characteristic is peculiar for each material, which is the bauxite in this case. Every single material has its own especifity. For example, coal presents a standard value 0.9 of bucket fill factor and range about 60% in swelling factor (Taheri 2014). According to Maia (2013), which analysed the swell factor of the bauxite at the same mine, this ore has the average value of 13% in the mining faces as Table 1 presents it. In order to set up a value that well represents the reality of this material, it was preferable to use the local analyses for the swell factor instead of to define a standard value for this kind of ore.

Table (1) – The analyses of ore characteristics.

BAUXITE	TON/M³
Density (ρ)	1,7395338
Swell factor	13%
Loose density (when the swell factor is applied)	1,5133944
Fill Factor of the loader – dimensionless Unit	**0,87**

Source: swell factor (Maia 2013).

Therefore, the fill factor of the shovel bucket presents the value of 0,87%, which was taking into comparison to the reference literature, which identify this ore as a material that has a medium dig conditions, table (2) provides information about it.

Table (2) – Fill factor and dig conditions

BUCKET FILL FACTOR	
Rock-soil mixtures	1,1
Easy digging material (sand, small gravel)	1
Medium digging material (coal, light clay, wet soil, soft ores)	**0,9**
Medium-hard digging material (iron ore, copper ore e etc…)	**0,85**
Hard digging material (blocky iron ore, sandstone, heavy clay)	0,8

Source – fill factor (HARTMAN e MUTMANSKY, 2002).

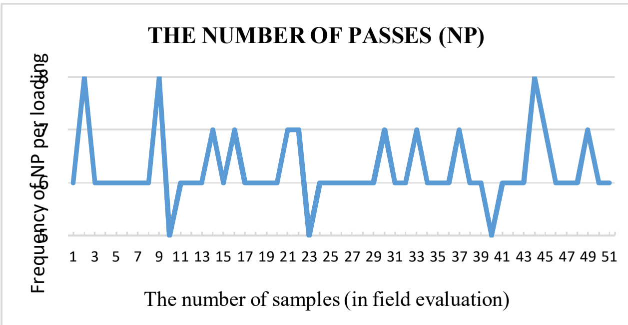
Shift

The standard shifts weekly was specify in the programme. The next step was to identify the lost shifts per year and the operational delays to allocate them. The loading and hauling ore operations work 24 hours a day, there are 2 shifts, 12 hours each for 7 days a week and 52 weeks a year in Juruti mine. It results in 8.760 effective hours be able to produce 5.821 tonnes per annum.

Shovel variables

The simulation must be reliable, which means the input of the programmes have to be as similar as possible to the real configuration of the mine. Therefore, the shovel was set as a backhoe hydraulic shovel, which is the same as it is utilizing to the ore operations. This shovel has the nominal capacity of 3,21 cubic meters or 5,58 tonnes of ore with the payload of 6,3 tonnes of bauxite per heaped bucket. The physical availability was established at the maximum value, because at the second stage of this article, in which some field test occurred according to the simulation results, which means the shovels and truck operated with total availability during the field tests and for the same representative time simulation. In other words, they operate under ideal conditions; there is no adverse situations during the simulation. Another considerable factor related to the shovel is the shovel cycle time is the movement of to dig the material, to carry it, and dump it into the truck tray and goes back to the same initial point of digging material. As long as the shovel cycle time is defined, it is possible to calculate the number of passes, which is how many shovel cycles are necessary to entirely fulfil the truck tray.

The number of passes is important to the loading operation, because it directly influences on the loading time (HUSTRULID e KUCHTA, 1995). Six was the number of passes determined by using the collected data in field of ore operation mine because of its frequency, as it is exposed in the graph (1).



Graph (1) - The frequency of the pass numbers

The ore operations using the swing angle of 90 degrees, as explained before, which indicates an angle factor of 1,00 without units. Table (3) presents this relation between the swing angle and the swing factor.

Table (3) - Determination of swing angle of the pass.

ANGLE OF SWING (DEGREES)	45	60	75	**90**	120	150	180
SWING FACTOR (S)	1.20	1.10	1.05	**1.00**	0.91	0.84	0.77

Source - (HARTMAN e MUTMANSKY, 2002).

The shovel variable configuration is extremely influent to the results generated by the programme in terms of queue time, cycle time and consequently estimated productivity. Regarding shovel cycle time, the recording data was collected from the field by observation and utilizing chronometers. However, the collected date shows a relevant variance. Thus, it was used a histogram in order to reduce this variance and to find out more reliable values. This histogram came from the data presented in tables (4) and (5).

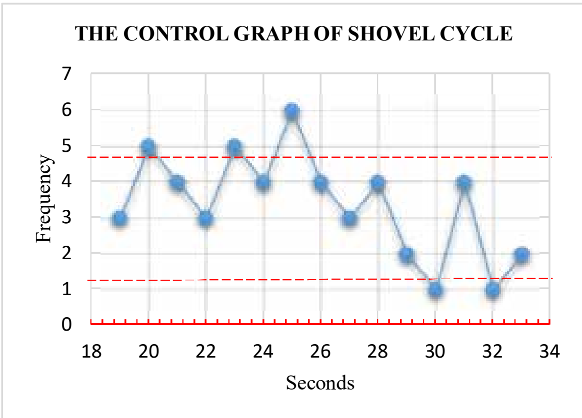
Table (4) – statistic data to build the histogram.

STATISTICAL RANGE	
FREQUENCY	PASSE CYCLE
7	00:00:14
AVERAGE	
FREQUENCY	PASSE CYCLE
3,40	00:00:25
STANDARD DEVIATION	
FREQUENCY	PASSE CYCLE
2,23	00:00:04

Table (5) – calculated data to define the histogram.

CONTROL GRAPH	
FREQUENCY	PASSE CYCLE
5,1	00:00:28
3,4	**00:00:26**
1,7	00:00:22

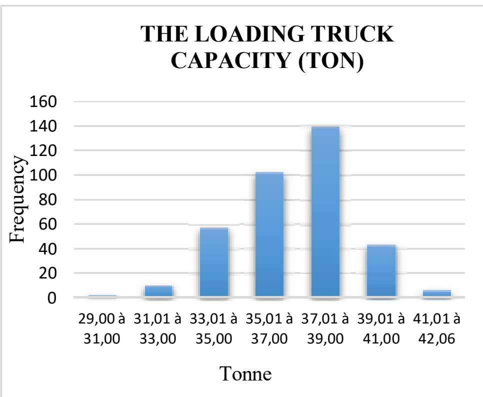
Table 4 utilizes data collected in field in order to verify how long the shovel takes to complete a cycle, whereas table (5) using these statistic data to define the highest, lowest and the central limit. 26 seconds was the value that better represents the shovel cycle time for the bauxite loading according to the graph (2).



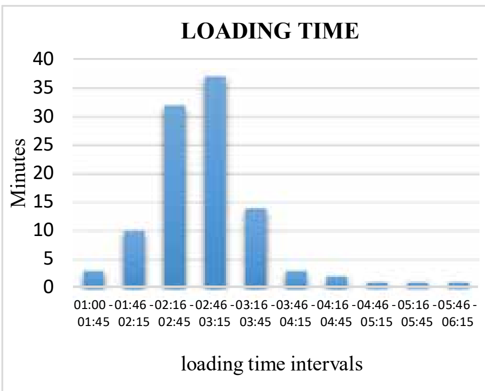
Graph (2) – shovel cycle histogram

The variable of the road truck

The stochastic simulation considers all trucks are all equal one to another and it is reliable. They have the nominal capacity of 37 tones. The graph (3) provides information about the actual carried weight and it shows that an average carried weight per truck cycle has the most reliable value of 37,8 tonnes. Whereas, 2 minutes and 36 seconds is the value that better represents the reality of the loading time by evaluating Graph (4).



Graph (3) - Actual carried weight per truck cycle.



Graph (4) - Truck loading time

Haul path

Average distance of transportation is the most influent variable of haul path. The simulation reveal haul cycle as the most influent factor of all. the simulation and the comparative field tests are related to two different strips of the same mining face. The estimation of the average distance of transportation coming from the mining planning data at the mining strips that is called Strip A and Strip B by convention. Strip A is 6.300 meters distant from the hopper crusher, while strip B is 5.950 meters from the hopper crusher. Figure (4) shows the transport route that the trucks run to build up their transport cycles, the haul road goes to strip A is yellow; whereas the haul road goes to strip B is green.

However, this variable does not is interesting to this mine, because Juruti mine is an area of flat land and does not have a single ramp in the ore path, moreover, this factor does not affect the results very easily, this only happens when great inclination are presented that is common to bench mines and not to strip mine. Thus, it is can be unvalued, considering to the typical terrain of Amazonia region and small value does not affect the simulation results.

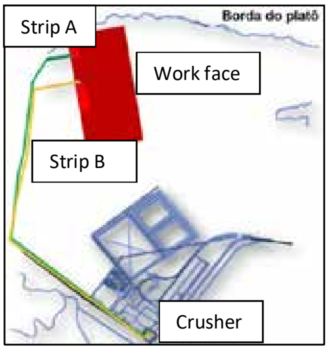


Figure (4) - haul road from the crusher to the mining face (strip A and strip B).

Simulation

It was possible successful ran the simulation of the loading and hauling ore operations because of the collected data of the all required variables, which was explained in this article previously. Actually, many simulations had been done in order to obtain the estimated results that is in next topic, simulation results.

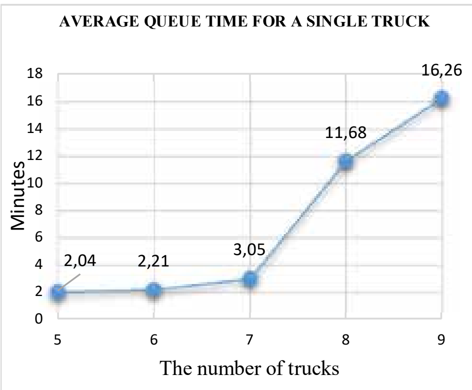
SIMULATION RESULTS

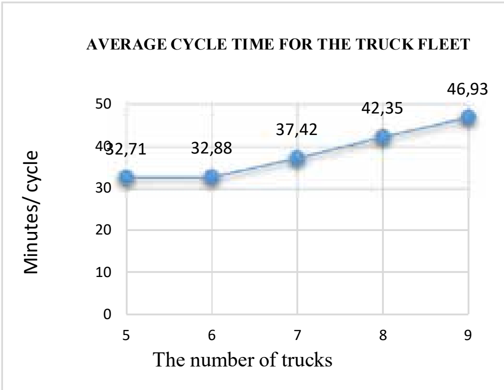
The direct effect of the queue time is the total time that the trucks take to leave and return to their original position, which is the cycle time. it is able to adopt any position as the initial one, since it is an cyclic operation, thus, the initial point is also the final point for the simulation. The first shovel pass of the truck loading was the first step, the second one was the truck hauling, the third step was dumping at the crusher and then the way back of the truck to the same mining strip.

The exposed results in this section refer to the two distinct strips of the same mining face. It is possible to identify as strip A and strip B. Initially, the next plotted results are all about strip A.

The ideal fleet size to the strip A is 7, which is possible to verify in the Graphs (5) and (6). The average cycle time, naturally, arises as the elevation of the number of trucks, the increase of the truck number take approximately 5 minutes to each extra truck, which is inserted, varying from 6 to 7 trucks and it take almost the same time from 7 to 8 trucks. However, the queue time minimally rise from 6 to 7 trucks, whereas the queue time substantially increases from 7 to 8 inserted truck into the operation cycle, which turn the operation with 8 trucks infeasible for loading and hauling ore. Because of this,

An interesting observation is about queue time is that it just vary a little from the beginning (5 truck operation) up to 7 trucks. On the other hand, the queue time dramatically arises when to turn the fleet size of 7 into 8 trucks, over three times more waiting time for loading and hauling truck. It is possible to calculate

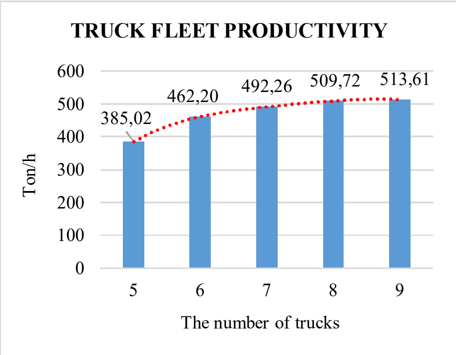


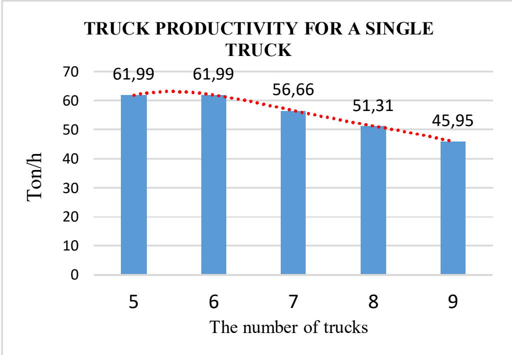


the productivity of each fleet size configuration from the cycle times. the fleet size productivity depends on the number of trucks directly, as Graph (7) shows.

Graph 5 – Average queue time of a single truck

Graph 6 – Truck cycle time (strip A).





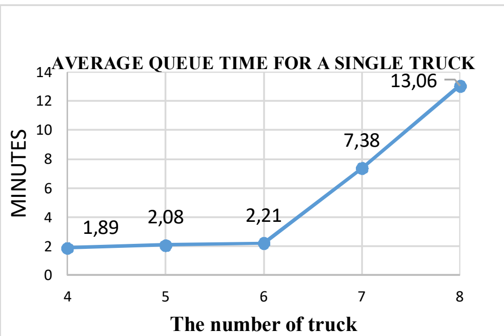
Graph 7 – Truck productivity (strip A).

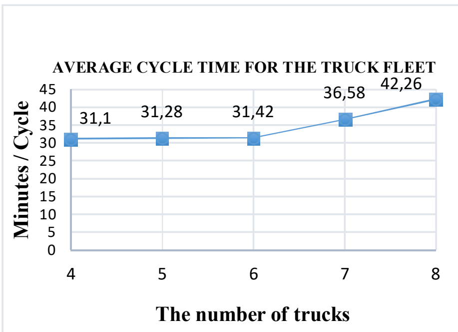
Graph (8) - Productivity variation (Strip A)

This Graph (7) exposes the fleet size productivity, which estimates the loading and hauling ore quantity per hour considering the number of trucks. Even though the productivity variation is superior from 5 to 6 trucks (increasing 77,18 tone per hour), it still vary positively from 6 to 7 trucks considerably with an extra positive value of 30,06 tonnes per hour, which does not means a great elevation of a queue time that only arise 50 seconds from 6 to 7 trucks. In this situation, the average queue time turn 2,21 into 3,05 minutes. Whilst, when the fleet size turn 8 into 9 trucks the productivity minimally increase and operates with excessive cycle time. The truck productivity only focuses on a single truck that compose a fleet size, as it is possible to verify in graph (8).

As the increase of truck numbers there is a reduction of the individual productivity, they have an inverse relation. This truck number elevates the queue time; Graph (8) considers the fleet size of 5 trucks as initial point for the productivity of a single truck, which means zero of productivity variation. Hence, there is a reduction of the individual productivity as the increase of operating truck numbers; they have an inverse relation. The truck queue time at the shovel naturally elevates because of the increase truck numbers, since a single truck takes longer to be loaded. Considering the raised truck queue time at the shovel, which influences on the truck cycle time. This entire process creates a smaller truck productivity per working hour. On the other hand, there is a significant productivity arise of the truck fleet as Graph (7) shows it.

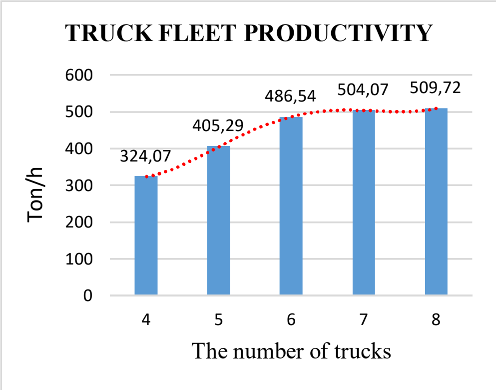
Another simulation have been done regarding the same working face, but in other strip that is strip B (named for convention). The increase queue time is approximately constant, which is under a minute for the truck fleet with; 4 trucks, 5 trucks and up to 6 trucks composing the truck fleet. However, this constant increase behaviour changes by inserting the seventh truck, the average queue time dramatically arises, becoming 3 (three) times over the average queue time by using 6 trucks in the truck fleet. This excessive increase of queue time at the shovel generates a substantial increase of the average cycle time pf the truck, which proves that the hauling and load ore operation is infeasible with 7 trucks operating in this strip. The graphs (11) and (12) provide information about truck fleet productivity. Graph (11) exposes the quantitative estimative of hauled and loaded ore for each truck fleet productivity, which considers a truck fleet as a whole. in terms of productivity variation, there is a constant growth up to 6 trucks in the fleet. When the fleet is composed by 7 trucks the real gain is not significant, with the value of extra 17,53 tonnes per hour. In this situation (7 trucks operating), the truck cycle time reaches 3 (three) times over then it was with 6 trucks, which is from 2,21 becoming 7,38 minutes. Thus, 7 trucks operating in this trip present an excessive queue time, consequently, the cycle time is very elevated and produce only 17,53 tonnes as extra weight per hour.

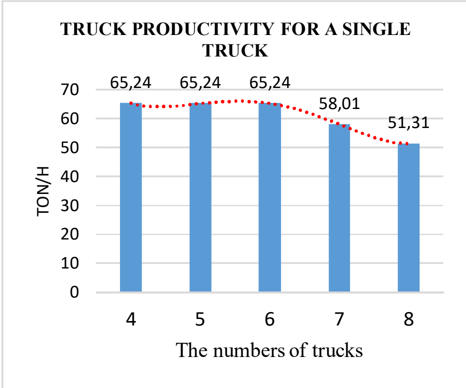




herefore, the ideal truck fleet to operate in this Strip B is determined as 6 trucks, taking into accounting productivity, queue time and cycle time.

Graph (09) - Average queue time a single truck (strip B)　　　Graph (10) - cycle time (strip B)





Graph (11) Truck fleet productivity (strip B)　　　Graph (12) single truck productivity (strip B)

The simulation indicates the maximum productivity occurs by using 6 fixed trucks as the ideal truck fleet size for the strip B. Graph (11) shows this fleet size has high productivity in comparison to immediately inferior fleet size (with 4 and 5 trucks). However, from the utilization of 7 trucks the productivity gain is inexpressive that offers unnecessary risks such as tire burst and machinery depreciation. More than 6 truck fleet size means an excessive queue time, which turn the loading and hauling operations into a much more expensive activity, since trucks spend much more fuel just because of a little more tonnes per hour. In other words, the use of 7 trucks or more would not be cost effective, particularly in terms of risk management and fuel cost.

COMPARATIVE TESTS

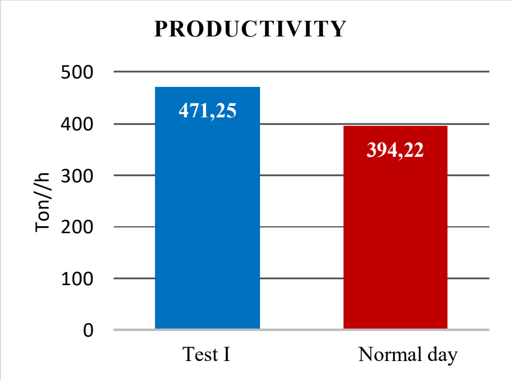
The comparative tests comprise of three field experiments. That is, the hauling and loading ore operation happened three times according to the simulation results, it would present the best configuration in terms of productivity. Hence, the results was putted into practice, which is to fix the number of trucks operating in the mining face, being 6 trucks in strip A and 7 trucks operating in the trip B, without insert or remove trucks for these ore operations.

These tests are comparative respecting productivity, it because they can be compared with the loading and hauling ore operation during a normal day, when the number of truck vary many times per shift, this variance occur because the ideal queue time is not identified yet. Therefore, the important of the tests is to validate the simulation and determine the ideal number of trucks. Thus, this work of simulation and result validation in field determine the ideal queue time, which provides a best possible productivity. The evaluation of the tests and normal days results is recorded by the dispatched mine programme, which can verify productivity and number of trips, this last one is, a complete cycle that a truck does considering the truck arrival and dumping at the crusher.

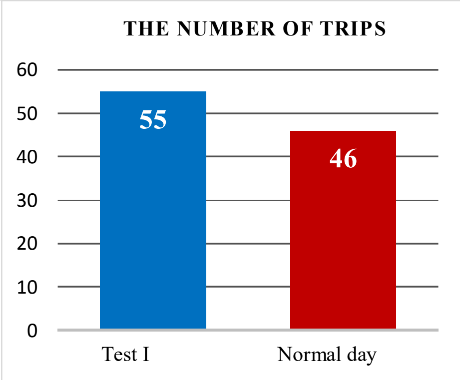
Unfortunately, the dispatch does not shows exactly the it is limited by the consideration of the truck capacity is always 35,52 tonnes per trip (haulage cycle). Although the productivity value presented by the mine dispatch does not represent the real quantity of haulage ore with fidelity, the value supply by the dispatch programme are sufficient to compare them, since both of them are evaluated by the same method. The (I) first and (II) the second test happened in strip A, whereas (III) the third one in strip B, remembering that all of them are in the same mining face, which is known as mining face Y.

Test I

The test I was carry out in 28th of august of 2015 and the comparative normal day was 30th of august of 2015. From 8:30am to 12:50pm for 4,2 hours.



Graph (13) – Productivity comparison – Test I



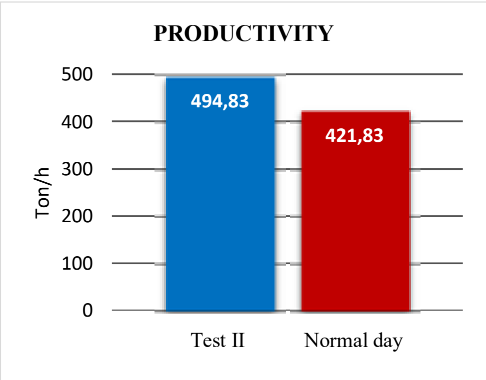
Graph (14) – The complete cycle times – Test I

There were seven trucks operating in test I, where the productivity was superior during the same time, that is 4,2 hours. The test truck fleet size configuration produce 77,03 more tonnes than the normal day operation for 4,2 hours, in other words, it is possible to produce an extra amount of 18,56 tonnes per hour by using fixed number of seven trucks for whole operation as Graph (13) shows it.

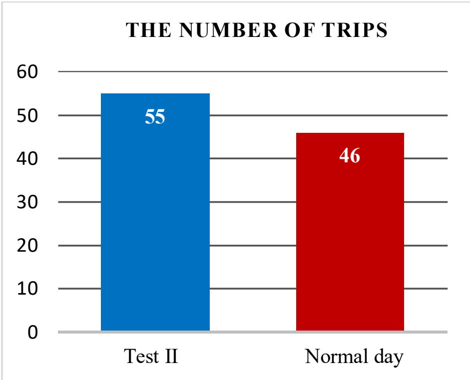
The dispatch results confirm that the test I operation is superior regarding productivity also by the number of trips or complete cycles. Test I complete 55 trips from the crusher to the strip A, whilst the normal day operation executed only 46 complete cycles, that is, Test I achieve 9 more trips than the normal day of operation, as it is possible to notice in graph (14).

Test II

Test II was carried out in 28th of august 2015 in strip A, whereas the normal comparative day was 30 of august 2015. The test started 2pm and finished 8pm, and it is relevant to highlight that the Juruti bauxite mine operates with two turns (day and night) of 12 hours. The Test II took 4,67 hours, while the comparative day took 4,88 hours. This difference happened because of dispatch system limitations, which it essentially dependent of the person that execute and contact all the machinery operators. This difference means a slight vantage that is 20 extra minutes to the comparative day. Although comparative day has this benefit, test II is more productive than its comparative day, as Graph (15) presents.



Graph (15) – Productive comparison – Test II



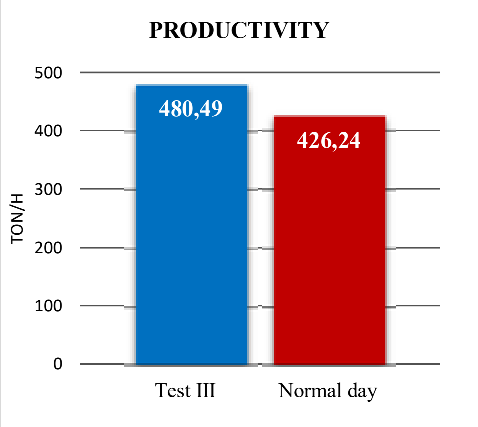
Graph (16) – The complete cycle times – Test II

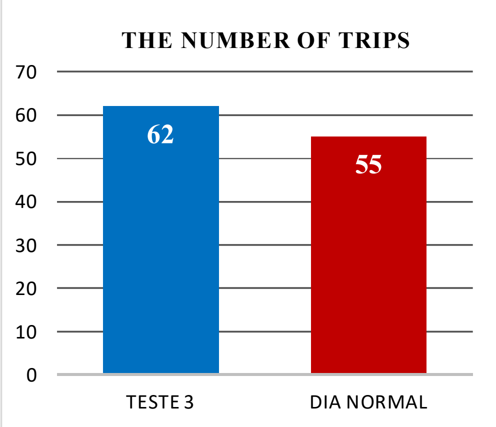
Test II set seven for whole operation in strip A, where the productivity was superior, even if it took less operation time according to the dispatch report. This test produced exactly 73 tonnes, in other words, its productivity was 14,96 tonnes per hour. Test II also presents a higher productivity than the ore operation in a comparative normal day in terms of trip numbers or complete cycle time of the trucks. Test II make seven extra trips possible in comparison with the normal day of loading and hauling operation, even though this test has an advantage regarding time operation, which is able to observe in Graph (16).

TEST III

Test II presents the ore operation in accordance to simulation results for strip B, it occurred in 31st of August 2015, and its comparative normal day operated in 1st of September 2015.

Both of them occur from 12:45pm to 5:20pm, which is 4,58 hours.Test III put 6 trucks as the constant truck fleet size in strip B for whole operation time according to simulate propounded. This test produced 54,25 more tonnes than the comparative operation day, which means 11,84 extra tonnes per hour in terms of productivity, as it is possible to see in graph (17).





Graph (17) – Productive comparison – Test III

Graph (18) – The complete cycle times – Test II

Another way to evaluate these productivity is according to the number of trips or the complete truck cycle time, this test is more productive by this evaluation method in comparison to the normal day for ore operations. Test III reach 7 trips over then the normal operation reaches as it is shown in graph (18).

CONCLUSION

The field test proves the great benefits obtained from the simulation in terms of truck fleet efficiency and mine productivity. The mine productivity increase up to 19,8%, with only a slight arise in the queue time, which is less than a minute. It means the extra amount of 21,1 bauxite tonnes per hour or 475,2 tonnes of ore in the mine ROM. All of this was possible by steady the number of truck per strip, which was 7 road trucks as the ideal number operating with an hydraulic back shovel in strip a, whereas the ideal number of trucks was 6 for the strip B.

This article recommend to utilize simulation for the loading and hauling mine operations whenever it is possible in order to determining of the way to operate in terms of productivity, since the dynamic of mine change all the time, especially regarding average transport distance (ATD) and queue time. The mine operational criteria should be the smallest queue time, which is responsible for better queue cycle performance that allows higher productivity rate. Working on this method it is possible to prioritize the reduction of idle time (waiting time) for loaders, which positively affects even the frequency and necessity loader repositioning, that takes long time for loader repositioning from a strip to another, this is important to consider in order to meet the production requirements related to ore quality. Therefore, the smallest queue time means also the better operation rate.

This study recommends investing in the project in simulation both for mine planning and operation, for the sake of improving queue time, and ATD reduction by finding other possible paths, which connect mine to the mineral process plant.

Considering that the mine does not operates under ideal conditions as simulation and field tests showed, this article suggest monitoring loading and hauling operations in respect of improve mine productivity. These mine adverse conditions should be taken into consideration by simulation for normal days of operation. However, it is not possible, because real time information is required. Thus, the recommendation is to invest in Global Positioning System (GPS), which would make this monitoring possible. The mine dispatch should be able to collect this real time data about loaders and truck. Data such as, where they are and what they are doing, which creates enough information to optimize the everyday operations.

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THE LOSS OF PRODUCTIVITY AND COSTS ASSOCIATED WITH THE PREMATURE SCRAPPING OF MINING TRUCKS TIRES

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ABSTRACT

The search for increasing productivity and reduction operational cost are the major challenges for companies in an increasingly competitive and demanding market. Currently companies with high costs tend to lose ground, products and/or services to competitors. In the mining industry is no different, the costs ratings become a routine in which the market scenario requires quick and efficient answers. In mining operation, specially loading and haulage activities, three major costs comprise the consumption of diesel and tires, and personnel. The reduction in operating costs goes through one of these pillars and in this context the improvement of processes or operation becomes paramount. In this paper we present a study of tires, which become unserviceable due to road hazard injury (e.g., a cut, snag, bruise, impact damage or puncture). The study comprises the data assessment of premature tire replacement rate due operational failures in the years 2014 and 2015. In this study, the loss of productivity caused by clashes of truck tires with sharp blocks of rock in the haulage and access road and in the loading and unloading places were compiled. The results show the loss of production due tires which become premature unserviceable and consequent stops for corrective maintenance of tires accounted 10% for the year 2014 and 7% for the first semester of 2015 with a cost amounted of US$ 416,000 in 2014 and US$ 270,981 in the first semester of 2015 (1US$ = 3.943 R$). Additionally, in this paper we present a feasibility study of adding, to the mining fleet, one loader with the aim of keep the roads in good condition in order to achieve sustainable productivity improvements, reduce premature loss of tires and minimize associated costs.

KEYWORDS

Haul road tires, premature unserviceable, tire’s performance, cost reduction.

INTRODUCTION

Haul road truck tires are one of the largest expenses for most mines. Tires have historically accounted for 25 to 30% of the operating costs in an open pit mine (Cutler, 2002; Tannant & Regensburg, 2001). This means that improving their lifespan cannot only save millions of dollar, but it can also help minimize unexpected downtime caused by damaged truck tires. According to (Vista Training, 2013) the average lifespan of a Caterpillar 797 haul truck tire is about 9,000 hours of operation. Nevertheless, a variety of hazards can cut this life span by half or more. According to Carter (2007), nearly three quarters of all tire failures are caused by cuts and impacts. The common threats to haul truck tires and the main characteristics of them are presented in Table 1 (Carter, 2007; Vista Training, 2013).

Table 1 – Main threats to haul road trucks and their characteristics

Threats to haul road truck tires	Characteristics
Sidewall cuts	One of the leading causes of tire failure that often happens when trucks get too close to berms and high bank faces, and rocks and other hazards can slash tire sidewalls.
Running in ruts	Running a tire on a rutted road can cut the sidewall and put stress on the carcass when entering and leaving the rut. Once in the rut, a tire will wear unevenly due to surface variations.
Under inflation	Underinflated tires experience significant sidewall deflection, especially when trucks are travelling under load. The result is higher tread wear, stress in tread and plies, weakened bonding and increased heat build-up.
Excessive speed on rough haul roads	Washboard profile of sections of haul road can give the truck's operator a roller-coaster ride, especially if it runs over them at high speed. These conditions are also very hard on the sidewalls of the truck's tires. Too much bouncing may even cause tires to momentarily leave the road surface, further amplifying sidewall flexing. Damage from excessive flexing is not immediately noticeable, but is cumulative and can be hard to spot in its early stages. Excessive speed in corners also has a similar effect.
Spilled material on the haul road	Running over any material spilled onto the haul road can make big threats to truck tires.
Bumper blocks	The edge of a concrete bumper block at a mine's waste rock dump can be a severe tire hazard. Backing over spilled material or sitting on top of it when dumping a load can place excessive stress on tires.
Windrows	Windrows created by normal grader road maintenance can cause sidewall flexing when haul trucks pass over them.
Dry steering	This term refers to the practice of turning the haul truck's front wheels while it is sitting still, which causes excessive forces on the sidewalls of the tires.

One of the most obvious factors of tire care is proper haul road management. Spillage from truck bodies, for instance, is a prime cause of tire damage and lack of proper road maintenance can dramatically affect the operating cost of a haulage fleet and cuts and impacts cause premature tire fails. Additionally, even with a clean surface, repeated use of the same path on a haul road lane can lead to increased tire wear. Truck units consistently using the same path in the respective haul lanes, the concentrated load will eventually wear ruts or furrows. Running a tire on a rutted road can cut the sidewall and put stress on the carcass when entering and leaving the rut. Once in the rut, a tire will wear unevenly due to surface variations (Carter, 2007).

Therefore, mine management must be aware of changing haul road conditions and contribute to tire life by making decisions that favor improved care of these critical components – such as cycling a truck with high tire temperature to a less demanding haul profile. In addition, the mine planning and engineering group must consider the design of haul roads, ramps, super elevation of curves, road base and construction materials, drainage and other factors that can have a positive or negative effect on tire life.

In the mine object of this study, the factor that hinders the proper tire performance as well as its wear and its premature end of life are cuts and punctures due to the existence of sharp rocks and rebar on the roads and load and dump places.

PDCA tools were adopted for the development of this study to evaluate the possibility of using a front loader Komatsu WA470 for road, and dump and load places clearance in order to, consequently, reduce the premature fail of tires. Figure 1 shows the PDCA adopted for this study.

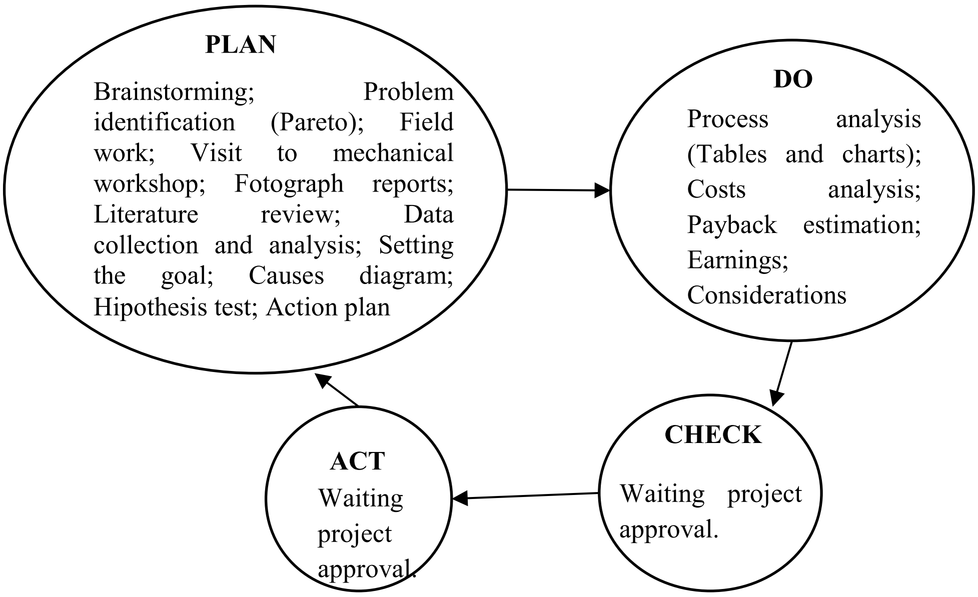

Figure 1 – PDCA adopted in the study

PROBLEM DIAGNOSIS

From Figures 2 and 3 it can be seen that greatest loss in tire scrap is due to operational failures relating to cuts and punctures in the tread and the sidewall, both in 2014 and in 2015 (to July)

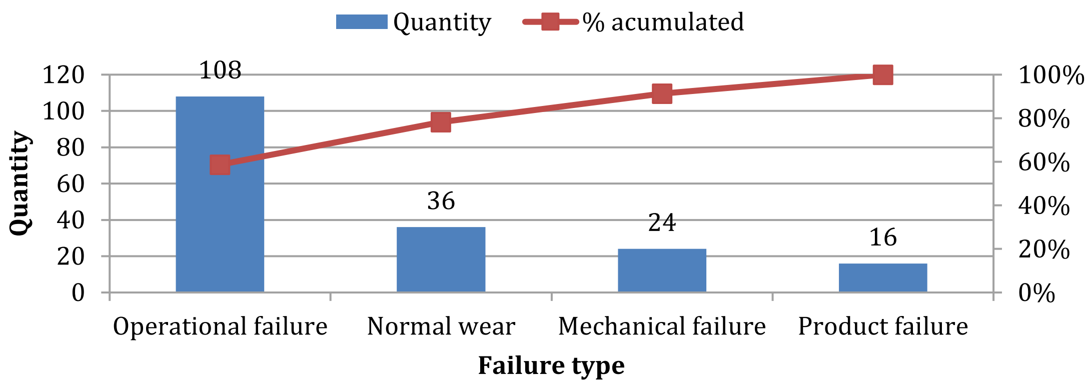

Figure 2 – Main causes of truck tires fails in 2014

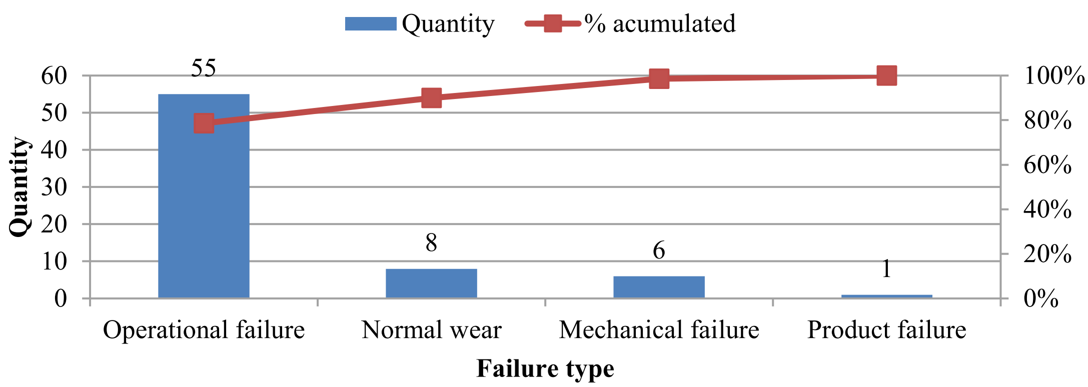

Figure 3 – Main causes of truck tires fails in 2015 (until July)

Off-road tires are designed to be used in non-paved ways, usually built of land with rough surfaces, where stones, wood or metal pieces are found. These obstacles are considered responsibility of the staff (Caterpillar Global Mining, 2007). The Mine Infrastructure Department frequently withdraw boulders from the slopes and close to the windrows, improving the equipment access conditions (Figure 4).

Figure 4 – Part of the road where windrow corners and roads were cleared.

Nevertheless, it is possible to see blocks and granular materials near the windrows at certain access points (Figure 5), with significant consequences (defects) in tires (Figure 6).

Figure 5 – Blocks moved to the foot of the side windrow access.

Figure 6 – Truck tire presenting high incidence of side cut

Data Assessment

Data collected in the mine were evaluated to quantify the costs and losses in cash and ore production, respectively, due to the existence of sharp rocks and rebar on the roads and load and dump places. Figures 7 and 8 show the quantitative assessment of tires, by fleet of trucks, which have been replaced or sent to reform and their repair costs in 2014. Figures 9 and 10 shows the tire's routine until July of 2015. Figure 11 show the loss (in Brazilian Real), due to the premature fail of tires in 2014 and Figure 12 show the loss (in Brazilian Real) due to premature fail of tires until July of 2015. Figure 13 summarize the costs and losses for the years 2014 and 2015 (until July). Table 2 shows the number of tires corresponding to the amount of money. Table 3 shows the amount of new tires needed in the years 2014 and 2015 (till July) due to premature fail of tires.

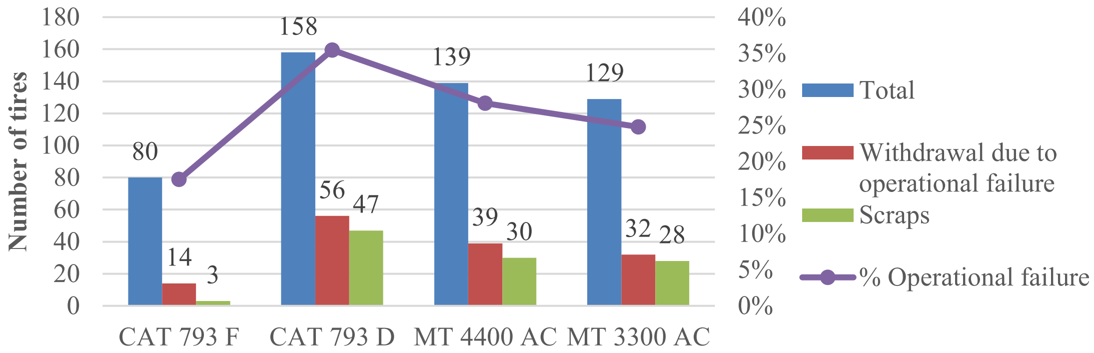

Figure 7 – Assessment of tires in 2014

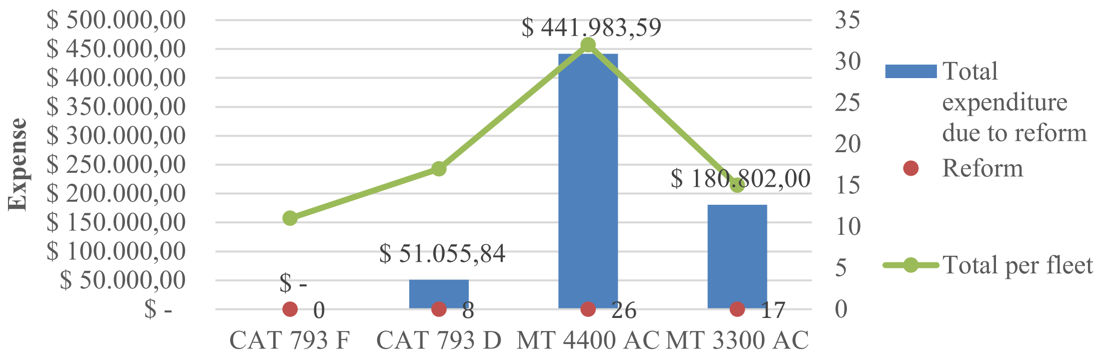

Figure 8 – Tire reforms expenditure in 2014

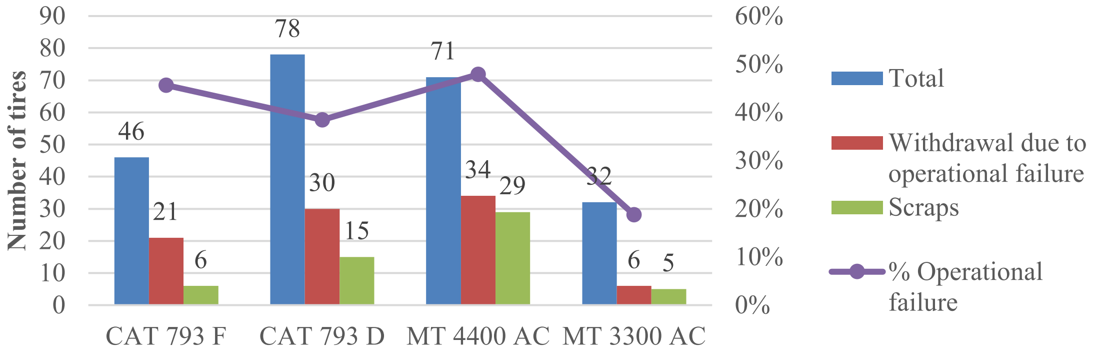

Figure 9 – Assessment of tires in 2015 (until July)

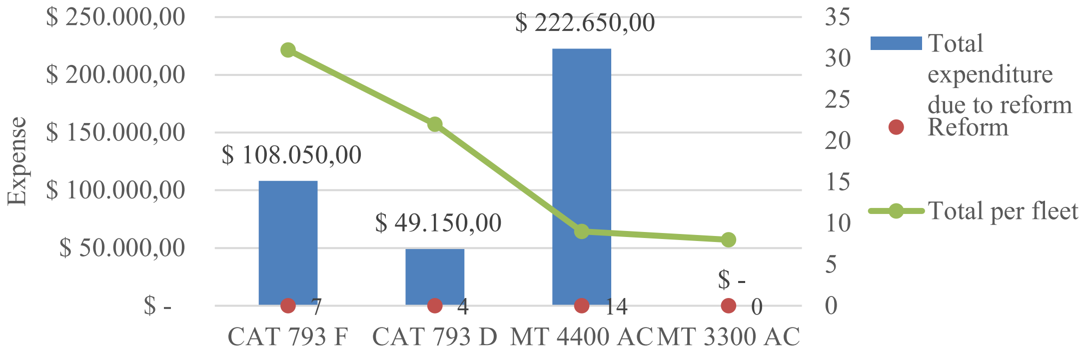

Figure 10 – Tire reforms expenditure in 2015 (until July)

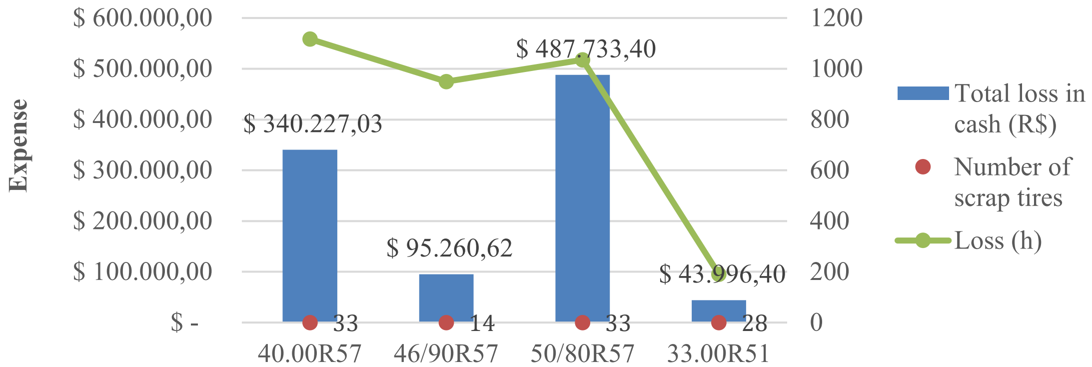

Figure 11 – Loss of cash and time due to scrap of tires in 2014

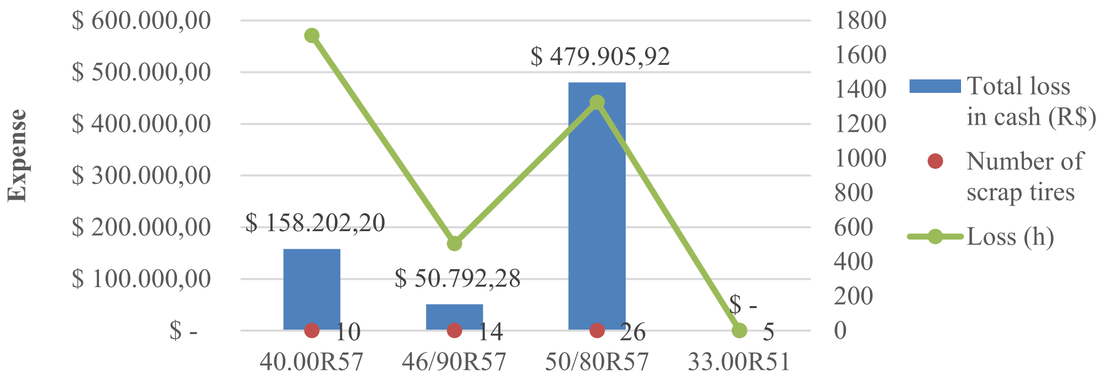

Figure 12 – Loss of cash due to scrap of tires in 2015 until July.

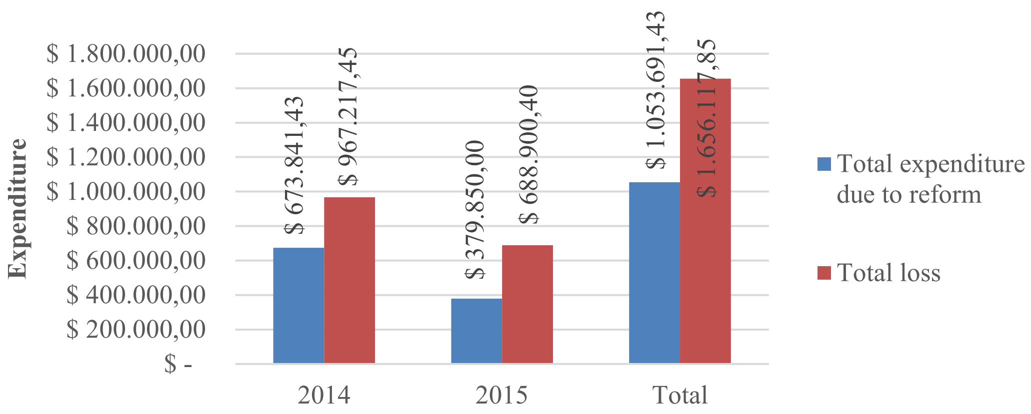

Figure 13 – Loss of cash in 2014-2015.

Table 2 – Number of tires corresponding to the amount of money in 2014 and 2015

Tire model	New tire value (R$)	Quantity (2014)	Quantity (until 2015 July)
40.00R57	83,737,62	20	13
46/90R57	58.270,77	28	18
50/80R57	101.482,69	16	11
33.00R51	58.746,97	28	18

Table 3 – Number of new tires used in 2014 and 2015.

Tire brand	Dimension	Quantity (2014)	Quantity (until 2015 July)
Michellin	40.00R57	36	26
Michellin	46/90R57	4	12
Michellin	50/80R57	118	43
Michellin	33.00R51	28	6

SOLUTION CONSIDERED

We conducted a study of the economic feasibility to add to the equipment fleet one loader brand Komatsu WA470 type in order to clear the access, windrows corners, cargo squares and unloading of existing blocks thus preserving the cuts tires and potential holes. This equipment was considered since it was available in the company, but in another sector. Table 4 shows the obtained data. The purchase price shown has the value of the last purchase made, dating from the year 2012.

Table 4 – Obtained data for the considered equipment

Model	Purchase value (R$)	Physical availability (%)	Utilization (%)	Operational cost per year (R$)
WA470-6	907.903,68	79	51	83.924,47

For the feasibility study considered only the costs of hours of maintenance, lubrication, tires, wear materials and fuel equipment by the number of hours worked per year. The annual operating cost corresponds to 5% of the total in 2014 and 8% of the total in 2015, as depicted in Figure 13.

It was also quantified production losses due to downtime for corrective maintenance of tires. Table 5 show, for each fleet type, the tons of ore left to be produced in the number of downtime for corrective maintenance in 2014 and 2015.

Table 5 – Losses in ore tonnages due to corrective maintenance stoppages in 2014

Tire brand	Loss of final ore product (t) in 2014.	Loss of final ore product (t) in 2015.
CAT 793D	11.090,7	17098,3
CAT793F	1.561,8	1499,3
Terex MT3300AC	4793,3	2255,8
Terex MT3300AC	9782,6	9931,4
Total	27.228,4	29.949,6

It may be observed that the amount of ore, which has not produced in the year 2015 (until July) due to stops in corrective maintenance of tires, it is 13% higher than in 2014.

Furthermore, in Table 6 it can be observed the monthly production in 2014 and the average of the year. It is noticed a loss of about 1% of the final product due to production stops when it has occurrences of threats to the tires. The same amount can be observed for the year 2015.

Table 61 – Monthly production in 2014

Month	Production 2014 (k ton)	Production 2015 (k ton)
Jan	6699,8	6443,1
Feb	6876,8	5119,8
Mar	7141,3	6143,4
Apr	7634,0	5893,7
May	8662,05	5609,58
Jun	7452,4	6888,2
Jul	7452,4	6936,1
Ago	8081,5	-
Sep	8516,3	-
Oct	8049,85	-
Nov	6456,4	-
Dec	6788,3	-
Average	7468,1	6147,7

CONCLUSION

This study shows the feasibility of maintaining a front loader in the mine in order to maintain roads and load and dump places free blocks. When comparing the costs of premature loss of tires more spending on renovations and repairs, it is observed that this equipment costs 5% of the sum of the total amount spent in 2014 and 8% of the sum of the total amount spent in 2015 (until July). It should also be accounted for by the loss of production stoppages in corrective maintenance of truck tires due to cutting and collision with blocks at the corners of windrows, access and squares of the mine. The year 2015 (until July) has already exceeded by 18% the 2014 year of production loss due to stop by corrective maintenance of tires, which shows an increase in the rate of collisions and cut the tires.

It follows also that the withdrawal by wheel loader, blocks or existing sharps in the markets, and access windrows corners, in addition to increasing the mine safety level, reduce the risk of tire bursts. Therefore, it is recommended based on facts and numbers to use Front loader for cleaning roads and load and dump areas. In the analysis of tire costs for large haul trucks a number of problems exist relating to the quality of available data. Since a mine has a limited number of roads of variable quality, any model of cost variation with road roughness or other geometric parameters will not be particularly robust. Other limitations exist with regard to damage attributable to loading or dumping areas as opposed to the road itself; up to 70% of tire damage may occur in loading or dumping areas.

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